Preliminary Economic Assessment White Pine North Michigan, U.S.A

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Preliminary Economic Assessment – White Pine North

Revision #

Michigan, U.S.A

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1. SUMMARY

1.1 Introduction

G Mining Services Inc. (“GMSI”) was retained by Highland Copper Company Inc. (“Highland”) or the (“Company”) to produce a Preliminary Economic Assessment (the “PEA” or “Study”) for the White Pine North Project located in the western Upper Peninsula of Michigan, USA and to prepare a technical report (“the Report”) in accordance with Canadian National Instrument 43-101 (“NI 43-101”) *Standards of Disclosure for Mineral Projects* to support the results of the PEA and the Mineral Resource Estimate (“MRE”) as disclosed in Highland’s press release entitled “*Highland Copper announces positive PEA results and mineral resource estimate for the White Pine North Copper Project in Michigan*” dated September 23, 2019.

1.2 Property Description and Ownership

The White Pine North Copper Project is located in the historical copper range district of the Upper Peninsula of Michigan, approximately 7.5 kilometers (“km”) south of Lake Superior in Ontonagon County. The Project covers approximately 4,500 hectares (“ha”) (11,000 acres) of surface rights and approximately 11,990 ha (29,615 acres) of mineral rights.

In May 2014, Highland completed the interim closing of the acquisition of the White Pine North Project from Copper Range Company ("CRC"), a subsidiary of First Quantum Minerals Ltd. The final closing of the acquisition is subject to several conditions including releasing CRC from certain environmental obligations associated with the remediation and closure plan of the historical White Pine Mine site and replacing the related environmental bond for an amount expected to be approximately US$1.7 M. The deadline to complete the acquisition of the White Pine North Project from CRC has been extended to January 31, 2020. A large portion of the Mineral Resources are located on the CRC property. There can be no assurance that the Company will be able to complete the acquisition of the White Pine North Project.

In April 2015, Highland entered into an agreement with Great Lake Resources, LLC to lease certain mineral rights covering an area of approximately 1,816 acres within the white Pine North Project area. The mineral lease is for 20 years, with a option for an additional 5 years.

1.3 Accessibility, Climate, Local Resources, Infrastructure and Physiography

The area is accessible via Michigan State Highway 64, which runs north-south 0.5 km west of the Project footprint. The Project is close to several communities, including White Pine, Ontonagon, Bergland, Wakefield and Ironwood.

The unincorporated town site of White Pine lies immediately across M-64, 0.6 km to the southwest of the mine site and had a population of 474 persons in the 2010 census. The town was built during the construction of the present White Pine Mine in 1952 to service employees of the mine.

The Michigan Upper Peninsula has well developed infrastructure, with paved road, optic fiber, natural gas, power grid and rail assets. The Project area is accessible by a Canadian National ("CN") rail spur, which leads to the Morengo junction in Ashland County, Wisconsin. The other nearest rail spur is in Ontonagon County, owned by Escanaba and Lake Superior Railroad, which leads southwest to Escanaba and connects to the CN rail grid. Both rail spurs would need to be refurbished if to be used commercially by a new mining operation.

CRC decided to sell all existing facilities upon closure in 1995 and several parties bought various buildings and parcels of land of what was called "White Pine Industrial Park". The processing plant and smelter were dismantled, but the power plant, refinery and other buildings kept and sold. Some of these buildings could be repurposed and used for a new mining operation.

The water intake is located off the mouth of the Big Iron River in Silver City and it was constructed by mining a tunnel under Lake Superior. The tunnel is 110 feet ("ft") below the pump house and 80 ft below the crib, 2,600 ft from pump house to crib. The current water withdrawal limit, based on pumping capacity, is 26 million gallons per day.

1.4 History

The discovery by CRC in the 1930s that lower grade zones of chalcocite mineralization extended over a very large area, coupled with increasingly sophisticated metallurgical techniques for treating fine-grained sulfide mineralization, led to development of the White Pine Mine and subsequent discovery of the Copperwood deposit farther west.

Construction began in March 1952 of the White Pine Mine and on March 31, 1953, the first ore was hauled to surface via the portal. The mill was completed in 1954 and the first pour of copper in the smelter was on

January 13, 1955. Inmet Mining Corporation (formerly Metall) announced in July 1995 that CRC would suspend all conventional mining and milling operations at the White Pine Mine on September 30th.

1.5 Geological Setting

The White Pine copper deposit is located in the Western Upper Peninsula of Michigan (U.S.A) on the south side of Lake Superior. The copper mineralization in the area of the former White Pine Mine occurs in the bottom 6 m (20 ft) of the Nonesuch Formation at the contact with the Copper Harbor Conglomerate. The shale and siltstone in the lower part of the Nonesuch Formation are divided into two mineralized shale units, the lower “Parting Shale” and the upper “Upper Shale”. The mineralized units are laterally persistent over tens of kilometers. The Parting shale has an average thickness of 2.2 m for the entire of the deposit, and the Upper shale has a thickness of around 3.0 m.

1.6 Mineralization

Copper mineralization at the White Pine deposit occurs in two modes; as very fine-grained sulfide (chalcocite) and as native copper. Sulfide mineralization is estimated to account for 85-90% of the copper in the deposit, but both modes of copper are intimately associated throughout the deposit. Sulphides occur as fine-grained lamellae in laminites and partings in interbedded sandstone and shale, very-fine grained disseminations and discrete clots in siltstone, and in veinlets and veins. Native copper mineralization occurs as sheet copper and mineralized sandstone. Sheet copper forms along thrust surfaces and are bedding parallel as well as cross-cutting stratigraphy.

1.7 Deposit Types

The mineralization of the White Pine North Project is classified as a reduced facies stratiform sediment-hosted copper deposit and is often compared to the Kuperschiefer-type in Germany and Poland.

1.8 Exploration

All exploration works completed on the White Pine North Project prior to 2014 were performed by the previous owner, CRC, who is now a wholly-owned subsidiary of First Quantum Minerals Ltd. Highland explored the property and conducted diamond drilling between 2014 and 2015 to complete a MRE, and additional metallurgical and geotechnical test work was also undertaken.

1.9 Drilling

Before the White Pine Mine was closed in 1997, all drilling activities undertaken on the property were performed by previous owners. In 1907, Calumet and Hecla Mining Co. began an extensive drilling program that discovered locally high grades of native copper. CRC conducted a continuous drilling program at the White Pine Mine from 1929 until the early 1970s. There was a hiatus in drilling until the commencement of a drilling program in 1994 – 1995. The 1994 – 1995 drilling program was conducted to provide a historical estimate supporting a feasibility study to build a new smelter at the White Pine Mine. Limited data are available from historical drilling, which totals 244,453 m.

Highland carried out two phases of drilling at the White Pine North Project in 2014 and 2015, with the aim of completing a current resource estimate for the Project as well as obtaining information for mine planning purposes. Drilling conducted by Highland in 2014 and 2015 totals 30,462 m.

1.10 Data Verification

Highland provided GMSI data files for the White Pine North Project, in date of March 2015. The drilling database was reviewed, and only minor errors were detected and corrected.

A site visit was conducted by Réjean Sirois, P.Eng. of GMSI to validate drill logs, assay certificates, sample intervals, downhole survey information and field checks to validate drill collars. No major discrepancies were found, and it is GMSI's opinion that the drill hole database is acceptable for use in calculating Mineral Resources.

1.11 Mineral Processing and Metallurgical Testing

Copper mining has been conducted at the White Pine Mine since 1952 and has produced over 2 Mt of copper until the mine's closure in 1994. In 1993, the mill treated 4.5 Mt of ore at a grade of 1.17% Cu. The average recovery and grades were 88% and 30% respectively, and the total energy consumption of the mill was approximately 31 kWh/t. Silver recovery was reported to be in the order of 90%.

The two principal copper minerals are chalcocite (Cu_2S), accounting for 80-85% of the total copper, and native copper ("Cu"), accounting approximately 10% of the copper. Minor sulfide minerals in the ore zone consist of covellite ("CuS"), Bornite ("Cu_5FeS_4") and chalcopyrite ("$CuFeS_2$"). Major constituents of the mineralized zone are sandstone, shale, siltstone and limestone.

In 2014, Highland initiated a preliminary metallurgical testing program at COREM laboratories. Testing look to validate and improve the historical performances producing a final concentrate. Flotation testing focussed on samples from the Parting Shale (“PS”) formation. The key process design criteria are summarized in Table 1.1.

Table 1.1: Key Process Design Criteria

Parameter	Units	Value
Plant Throughput	tpd	15,000
Head Grade - LoM	% Cu	1.0
Head Grade – Silver (Ag)	g/t	10
Plant Availability	%	91.3
Crushing Work Index (CWi)	kWh/t	12.2
Bond Ball Mill Work Index	kWh/t	14.2
Plant Operating Time	hr	8,000
Grind Size (P_{80})	µm	106
Rougher Conditioning Time - Laboratory	min	5
Rougher Residence Time - Laboratory	min	30
Scavenger Residence Time - Laboratory	min	10
Cleaner 1 Residence Time - Laboratory	min	6
Regrind Mill Product Size (P_{80})	µm	20
Target Concentrate Grade	% Cu	31.5
Target Overall Recovery	%	88

1.12 <u>Mineral Resources Estimate</u>

GMSI has prepared a Mineral Resource Estimate (“MRE”) for the White Pine North Project based on data generated up to March 2015. The main objective of this assessment was to produce a Mineral Resource for the White Pine North sector. The MRE was prepared under the supervision of Mr. Réjean Sirois, P.Eng. GMSI, Vice President Geology and Resources, an independent “Qualified Person” (“QP”) as defined in the NI 43-101 and has an effective date of August 30th, 2019.

The 3D geological modelling performed for the resource estimate was produced by GMSI based on the drill hole database and historical information pertaining to the three mineralized columns (the Parting Shale

(“PS”), Upper Shale (“US”) and Full Column (“FC”). A minimum true thickness of 2 m was applied during modelling of these columns.

One composite was generated per drill hole, per column of varying thickness. Statistics were calculated on the resulting copper and silver grade, and it was judged unnecessary at this stage to apply capping values as no major outliers were observed.

A homogeneous 2.70 g/cm^3 density value was used for all rock types in the block model.

Grade variography was generated in preparation for the estimation of copper grades using the Ordinary Kriging (“OK”) interpolation method. The variography was undertaken on the composites for each of the three columns (PS, FC and US). No variography was undertaken on silver composites.

Two interpolation techniques were selected for the White Pine North Project MRE. The OK method was used for copper grade interpolation and the Inverse Distance Squared (“ID^2”) for silver grades. A percentage-style block model was created using the wireframes of the mineralised columns and was used during grade interpolation. A three-pass estimation strategy was adopted, using progressively larger search ellipses and relaxed estimation parameters for later passes.

The block model was validated visually on a global and local scale, and statistical checks were made between the block model grades and composite grades (swath plots, descriptive statistics). The block model was found to be a good representation of the composites.

Blocks were classified in accordance with the Canadian Institute of Mining, Metallurgy and Petroleum (“CIM”) guidelines for Mineral Resources and Reserves. Classification was primarily based on estimation pass, with a manual coding step to ensure a coherent classification. No measured was declared at White Pine North due to the lack of QAQC and supporting information from the historical data. The deposit comprises of mainly Indicated and Inferred Mineral Resources.

A 300 m buffer zone (or boundary pillar) was applied around existing workings and excluded from the Mineral Resource. Only blocks within mineral leases where Highland has a greater than or equal to 25% ownership were classified as Mineral Resources.

The cut-off for the declaration of the Mineral Resource is 0.9% Cu, and was calculated using a copper price of US$3.00/lb., and a silver price of US$16/oz. A metallurgical recovery of 88% for copper and 73.4% for silver was assumed, with a payable rate of 96.5% for copper and 89.3% for silver. A flat NSR royalty rate of $0.05/lb. Cu payable was applied, which incorporates two royalties on the Project (Osisko Gold Royalty and Great Lakes Royalty). No mining dilution or loss was applied during the calculation in the cut-off grade.

Table 1.2 reports Mineral Resources for the White Pine North deposit. All parameters used in the calculations are also presented in the table's notes.

Table 1.2: Mineral Resource for the Parting Shale Column – White Pine North Deposit
0.9% Cu Cut-off Grade – August 30, 2019

Resource Category	Tonnage (Mt)	Copper Grade (%)	Silver Grade (g/t)	Copper Contained (M lbs)	Silver Contained (M oz)
Indicated	133.4	1.07	14.9	3,154	63.8
Inferred	97.2	1.03	8.7	2,210	27.2

Notes on Mineral Resources:

1) *Mineral Resources are reported using a copper price of US$ 3.00/lb. and a silver price of US$ 16/oz*
2) *A payable rate of 96.5% for copper and 89.3% for silver was assumed.*
3) *Metallurgical recoveries of 88% for copper and 76% of silver were assumed.*
4) *A cut-off grade of 0.9% Cu was used based on an underground "room and pillar" mining scenario*
5) *Operating costs are based on a processing plant located at the White Pine site.*
6) *A flat NSR royalty rate of $0.05/lb. Cu payable was applied, which incorporates two royalties on the project (Osisko Gold Royalty and Great Lakes Royalty)*
7) *The PS Column was modelled using a minimum true thickness of 2 m*
8) *No mining dilution or mining loss was considered for the Mineral Resources*
9) *Mineralized rock bulk density is assumed at 2.7 g/cc*
10) *Classification of Mineral Resources conforms to CIM definitions*
11) *The qualified person for the estimate is Mr. Réjean Sirois, P.Eng., Vice President - Geology and Resource for GMSI. The estimate has an effective date of August 30, 2019*
12) *Mineral Resources that are not Mineral Reserves do not have demonstrated economic viability. The estimate of Mineral Resources may be materially affected by environmental, permitting, legal, title, taxation, sociopolitical, marketing, or other relevant issues.*
13) *Parting Shale: Interval defined from the base of the Lower Transition to the top of the Tiger units*
14) *The quantity and grade of reported Inferred Resources in this estimation are uncertain in nature and there has been insufficient exploration to define these Inferred Resources as Indicated or Measured Mineral Resources.*

1.13 Mineral Reserves Estimate

Given that this report is a PEA, there is no Mineral Reserve Estimate for the White Pine North Project.

1.14 Mining

1.14.1 Mining Method

The proposed mining method for the White Pine North Project is conventional drill and blast room-and-pillar given the relatively sub-horizontal thin orebody. The method consists of the extraction of a series of entries and cross cuts in the ore leaving pillars in place to support the back. The entries, cross cuts and pillars are sized using a geotechnical analysis of the rock, and experience from the old White Pine mine with similar ground conditions.

1.14.2 Mine Access

The mine will comprise three sectors; the Eastern, Center and Western parts. The mine will be accessed via a new covered box-cut to establish a portal at the mine entrance from the surface, located at the western side of the deposit. The pre-production period requires 41,512 m of development to establish the main entry panel requiring four to six drifts according to the ventilation requirements. All drifts are set at a width of 6.1 m, and their height varies from a minimum of 3.5 m to a maximum of 6.1 m.

1.14.3 Rock Mechanics and Geotechnical Design Criteria

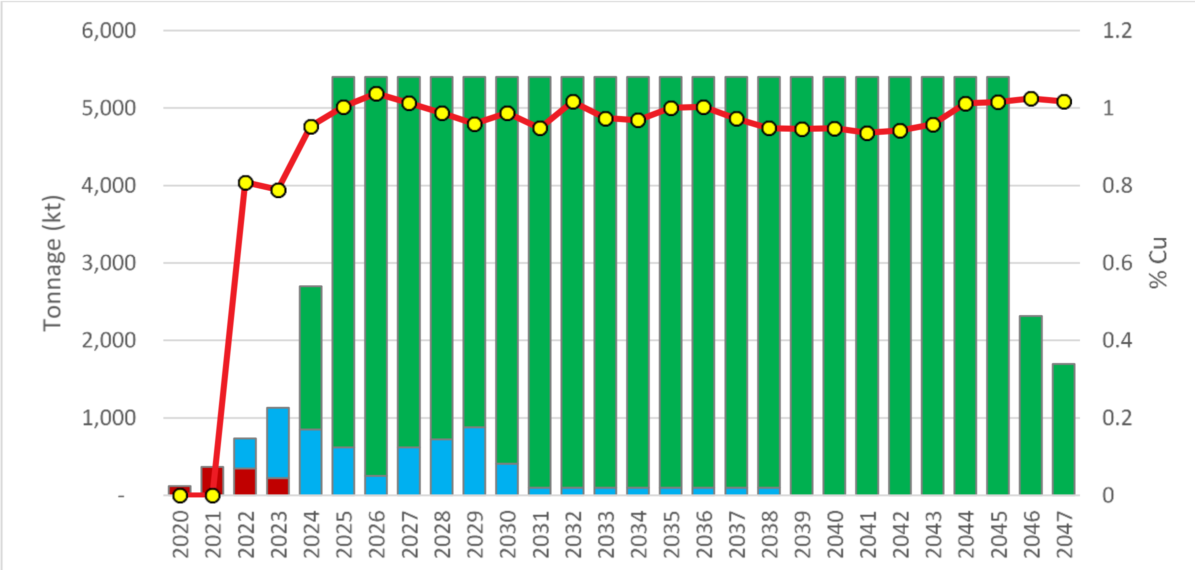
The old White Pine Mine was in operation from 1955 to 1995 as a room and pillar operation. Conditions in the mine are reported as variable, depending on the proximity to major structures and the syncline axis. For the most part, back conditions were observed to be good where the back was formed in sandstone. The ground support planned consists a 1.8 m rebar bolts on a 1.2 m x 1.2 m pattern.

At present, a 300 m pillar has been retained with the former White Pine Mine. This pillar dimensioning will require further investigation and sizing.

1.14.4 Mine Design and Production Schedule

The production schedule is based on mining a fixed target of 5.4 M/yr. To achieve this annual production, seven to fourteen production panels must be in production simultaneously. The number of required panels depends on the tonnage from the development as well as the height of the rooms of each panel. The mining of the room will be done using a two-pass approach. In the first pass, larger pillars are left in place. The mining recovery of the first pass is estimated at 40%. Once the first pass is completed, the size of the pillars is reduced via a second pass to increase the average mining recovery to approximately 57%.

Figure 1.1: Mine Production Schedule



1.14.5 <u>Mine Operations</u>

To access the stope four access drifts are excavated at the entrance of the stope. One of these drifts is used for fresh air ventilation and the second for exhaust air ventilation. One drift will be used for the stope conveyor and the last one for circulation. From these accesses, the panel operation begins with the drilling and blast method. To achieve and maintain an adequate level of production, the panel must contain at least 12 rooms (headings) in operation simultaneously.

The mining cycle includes drilling, blasting, ore mucking, ore transportation to a rock breaker and the stope conveyor, scaling and finally ground support. The mining of the room will be done in two-pass approach.

In the room-and-pillar mining method the mining cycle begins with the drilling of the working face. To perform face drilling, a low-profile hydraulic-electric jumbo with 2 booms is planned. The drilling technique will use a burn cut to allow drilling a length of 4.25 m with an effective break length of 4.0 m.

Blasting crews will load the rounds with explosives and initiate blasts at the end of each shift. Explosives will consist of a mixture of ANFO and emulsion where there is presence of water. A decoupled explosive charge is recommended to presplit the back of the room.

The third mining activity is to muck the blasted ore from the face and to transport it with a low-profile 10t LHD. The performance of the LHD is a function of the dip of the stope and the distance between the face heading and the rock breaker. The LHD performance will vary from 3.9 km/h at 17% (loaded) to 8.9 km/h at -17% (unloaded). To reduce the haulage distance, the unloading point will be moved regularly to be normally less than 250 m from the working face.

The stope dumping point is a system composed of a grizzly, a rock breaker and loading points to the conveyor system. The ore will be transferred on the 42 in. stope conveyor that will be extended depending on the progress of the stope.

The next step in the mining cycle is to scale the back and wall of the excavation. To do this a smaller low-profile LHD equipped with a scaling arm is used. A low-profile rock bolter is used to install the roof and wall support. In the room excavation 1.8 m rebar bolts are required according to a 1.2 m x 1.2 m pattern currently in this Study no wire mesh is planned.

1.14.6 Mine Services

The ventilation system will consist of a push system whereby two 1250 HP parallel main fans will be installed at surface providing approximately 225 m^3/s each at 6.0 kPa. The two main fans will be installed and provide heated air through a 5 m ventilation raise and air will be distributed throughout the mine using ventilation regulators, auxiliary fans, doors and bulkheads.

Water is required underground for the drilling and controlling of dust. It must also be available for firefighting. Water will be distributed underground by a 4 in steel pipe schedule 40 in the main access drift and 2 in light wall steel pipe in the stopes.

A high voltage cable (13.8 kV) will be installed in the conveyor drift access. This high voltage cable will connect to a substation in each production panel which will drop the voltage to 600 V for the electrical needs of the operation.

1.15 Recovery Methods

The process plant design for the White Pine North Project is based on a metallurgical flowsheet designed to produce copper concentrate. The process plant has been designed for a nominal throughput of 15,000 tpd. The overall flowsheet includes the following steps:

- Crushed ore reclaim;
- Grinding and classification;
- Rougher flotation;
- Rougher concentrate regrinding;
- Cleaner flotation, using three stages of cleaning with flotation cells and columns;
- Concentrate thickening and filtration;
- Tailings pumping and disposal in the common Tailings Disposal Facility (“TDF”).

1.16 Project Infrastructure

The White Pine North Project requires several infrastructure elements to support the mining and processing operations. The infrastructure planned for the Project includes the following:

- Public access road from Michigan Highway 64;
- Site access roads;
- Power generation plant (30MW);
- Site electrical distribution;
- Gatehouse;
- Communications network;
- Lake Superior water intake tie-ins;
- Potable water treatment plant tie-ins;
- Sewage treatment tie-ins;
- Covered box-cut for mine access;
- Ore stockpile pad;
- Truck shop, wash bay, warehouse and offices;
- Fuel storage;
- Mill offices and metallurgical laboratory;
- Administration office and assay laboratory;
- Concentrate transload facility;

- TDF;
- Effluent Treatment Plant.
- Event pond ditches for surface water management at mill site.

1.17 Market Studies and Contracts

The metal prices selected for the economic evaluation in this Report use a constant long-term copper price of US$ 3.10/lb and silver price of US$ 16.00/oz over the life-of-mine (“LoM”).

The copper concentrate produced from White Pine will require downstream smelting and refining to produce marketable copper and silver metal. Concentrate transportation charges will be a function of the final destination. The concentrate will be loaded into heavy-duty dump trailers with a cover and transported by truck to rail trainload facility located in Park Falls, Wisconsin. The truck configuration consists of 5 axles and will transport approximately 20 t per shipment. Park Falls is a preferred trainload location as it is currently served by the Canadian National (“CN”) railroad and is approximately 100 miles from the mine site.

A summary of the copper concentrate marketing assumptions is summarized in Table 1.3.

Table 1.3: Concentrate Marketing Assumptions

Copper Concentrate Marketing Assumptions	
Copper Payable Rate	96.5% payment of Cu in concentrate >22% Cu and <32% Cu subject to a 1% minimum deduction
Silver Payable Rate	90% payment of Ag subject to 30 g/dmt minimum deduction
Copper Treatment & Refining Charge (TC/RC)	TC = US$ 70/dmt of concentrate, RC = $0.070/lb of Cu
Silver Refining Charge	RC = US$ 0.50/oz of Ag

1.18 Environmental Studies and Permitting

The former White Pine Mine ceased operation in 1995 and has been the subject of an extensive remediation program outlined in judicial Consent Decree and Remedial Action Plan agreements between CRC, Michigan’s Attorney General and the Michigan Department of Environment, Great Lakes, and Energy. The entire surface area overlying the underground mine along with the associated surface component area and tailings impoundments are listed as a “facility” under Part 201, Environmental Remediation, of Michigan’s Public Act 451 of 1994 as Amended, the Natural Resource and Environmental Protection Act.

Pending final closing of the acquisition of the Project, the Company began mineral exploration and baseline environmental surveys under an access agreement with CRC and under a mining lease with Great Lakes. Historical environmental data for the former White Pine Mine site operated by CRC was reviewed and compared with the Company’s initial project plans and Michigan’s Part 632 regulatory requirements. CRC had compiled extensive information on surface water, ground water and near-surface soils at the project site. Biological monitoring data in the Project area was mostly limited to very brief descriptions, e.g. the Remedial Investigation Report of 1999, or the more thorough description of the 1978 Baker Report that is now over 40 years old. Data from limited nearby stream monitoring completed by the State of Michigan in 1999 and earlier is also available.

Upon completion of the final closing of the acquisition of the mineral and surface rights from CRC, the Company will assume all environmental liabilities related to the Consent Decree and on-going environmental obligations.

1.19 Capital and Operating Costs

The capital expenditure (“CAPEX”) for Project construction, including concentrator, mine equipment, support infrastructure, pre-production activities and other direct and indirect costs is estimated to be US$512.5 M. The total initial Project capital includes a contingency of US$90.8 M, which is 21.5% of the total CAPEX before contingency, and excludes pre-production revenue of US$55.7 M. Net of pre-production revenue, the initial CAPEX is estimated at US$456.7 M as presented in Table 1.4. The initial Project CAPEX is spent over a period of four years starting in January 2020 and ending in December 2023. The first two years are mostly for box cut and mine underground development work.

Table 1.4: Initial Capital Expenditure Summary

Initial CAPEX	k US$
000 - General	212
100 - Infrastructure	31,214
200 - Power & Electrical	64,051
300 - Water & TSF Mgmt.	15,078
400 - Mobile Equipment	44,453
500 - Mine Infrastructure	39,183
600 - Process Plant	47,359
700 - Construction Indirects	25,739
800 - General Services & Owner's Costs	18,193
900 - Pre-Production, Commissioning	136,190
Sub-Total Before Contingency	**421,671**
Contingency (21.5%)	**90,788**
Total Incl. Contingency	**512,460**
Less: Pre-Production Revenue	**(55,715)**
Total Incl. Contingency & Pre-Prod. Revenue	**456,745**

Sustaining capital expenditures during operations are required for additional mine equipment purchases and replacements, mine development work, tailings storage expansion and general plant sustaining capital allowances. The LoM sustaining CAPEX is estimated at US$459.3 M with the breakdown presented in Table 1.5.

Table 1.5: Sustaining Capital Expenditure Summary

Sustaining CAPEX	LoM ($M)	$/t ore	$/lb Cu Payable
Tailings Disposal Facility Expansion	17.56	0.15	0.01
Mine Equipment Purchases	121.49	1.01	0.06
Mine Development Expenditures	291.72	2.43	0.13
Other Plant	28.53	0.24	0.01
Total Sustaining CAPEX	**459.29**	**3.82**	**0.21**

Operating expenditures ("OPEX") include mining, processing, G&A services, concentrate transportation and concentrate treatment and refining charges. The concentrate transportation, treatment and refining charges are deducted from gross revenues to calculate the Net Smelter Return ("NSR"). The NSR for the Project during operations is estimated at US$6,444 M excluding US$55.7 M of NSR generated during pre-production and treated as pre-production revenue. The average NSR over the LoM is US$2.92/lb of payable copper. Detailed operating cost budgets have been estimated from first principles based on detailed wage scales, consumable prices, fuel prices and productivity. The operating costs are detailed in Section 21 of this Report. The average OPEX over the LoM is US$25.67/t of ore or US$1.40/lb of payable copper with mining representing 66% of the total OPEX, or US$ 16.96/t of ore. A summary of operating cash flow and operating costs is presented in Table 1.6.

Table 1.6: Operating Cost Summary

Operating Cash Flow	LoM (US$M)	US$/t ore	US$/lb Cu Payable
Cu Revenue	6,615	55.07	3.000
Ag Credits	496	4.13	0.225
Revenue	**7,111**	**59.20**	**3.225**
Less: Concentrate Transportation Costs	260	2.17	0.12
Less: Treatment & Refining Charges	407	3.39	0.18
Net Smelter Return	**6,444**	**53.65**	**2.92**
Royalties	113	0.94	0.05
Mining Costs	2,038	16.96	0.92
Processing Costs	740	6.16	0.34
G&A Costs	193	1.60	0.09
Total OPEX	**3,084**	**25.67**	**1.40**
Operating Cash Flow	**3,359**	**27.97**	**1.52**

Note: Ore tonnage and payable copper unit costs during operations period only

1.20 Economic Analysis

The undiscounted after-tax cash flow is estimated at US$1,907 M for the White Pine North Project. The pre-tax net present value at 8% ("$NPV_{8\%}$") is estimated at US$557 M with an 19.2% internal rate of return ("IRR") and 4.5 y payback period. Similarly, the after-tax $NPV_{8\%}$ is estimated at US$416 M with an 16.8% IRR and 5.2 y payback period.

The annual cash flow is summarized in Figure 1.2 and a cash flow waterfall for the White Pine North Project is presented in Figure 1.3.

Figure 1.2: After-Tax Annual Project Cash Flow

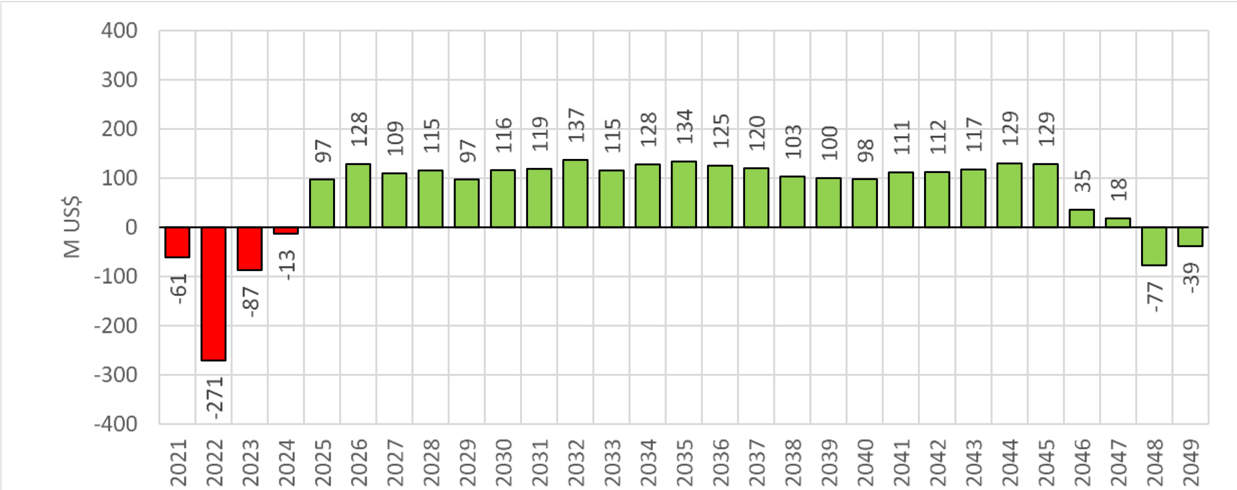
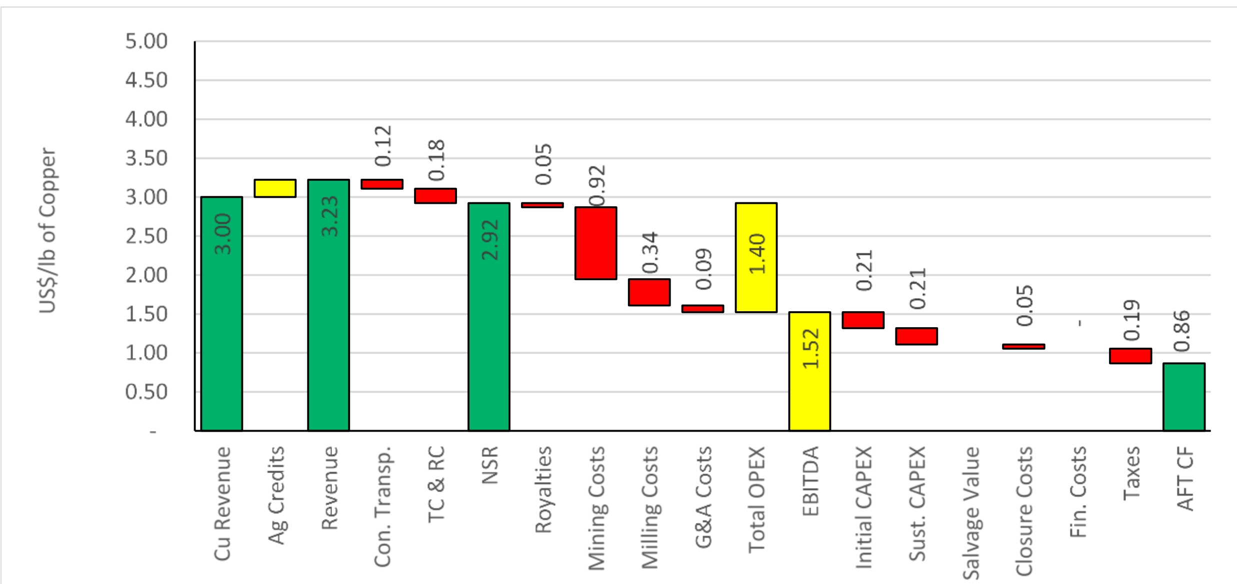


Figure 1.3: After-Tax Project Cash Flow Waterfall



1.21 Other Relevant Data and Information

The reader is cautioned that this PEA is preliminary in nature as it includes Inferred Mineral Resources that are too speculative geologically to have the economic considerations applied to them that would enable them to be categorized as mineral reserves, and there is no certainty that the PEA will be realized.

1.22 Risks and Opportunities

The risks and opportunities identification and assessment process are iterative and have been applied throughout the PEA Study phase. The following risks and opportunities are summarized in Table 1.7.

Table 1.7: Project Risks and Opportunities

Project Risks	Project Opportunities
Permit acquisition or delays	Combination of White Pine Project with Copperwood whereby the latter can provide ore earlier and at higher grade
Ability to attract experienced professionals	Reduction in pillar with former White Pine mine
Declining metal prices	Mining with a continuous miner
Faults creating offsets in the mineralization	Shaft to accelerate access to the White Pine North ore body
	Metallurgical recovery improvements from flotation process or heap leaching of copper ore and SX-EW option
	Underground tailings disposal

1.23 Recommendations

Based on the positive results of the PEA, GMSI recommends that the White Pine North Project move forward to the next phase which would include the following:

- Infill resource drilling at White Pine North deposit (eastern sector) to upgrade current Inferred Mineral Resources to Indicated category in order to support a Pre-Feasibility Study;
- Review project as integrated complex with the Copperwood deposit feeding ore to a centralized processing facility located at White Pine. Investigate ore transport options between Copperwood and White Pine such as trucking to rail, conveying to rail or simply trucking;
- Metallurgical optimization testwork and leaching evaluation;
- Initiate geotechnical investigations to further dimension the required pillar with the former White Pine mine;
- Trade-off study of sinking a shaft versus a portal and ramp decline to access the ore body;
- Trade-off study of grid power connection versus on-site natural gas power generation.

2. INTRODUCTION

In May 2014 Highland completed the interim closing of the acquisition of the White Pine North Project from Copper Range Company ("CRC"), a subsidiary of First Quantum Minerals Ltd. The final closing of the acquisition is subject to several conditions including releasing CRC from certain environmental obligations associated with the remediation and closure plan of the historical White Pine mine site and replacing the related environmental bond for an amount expected to be approximately US$1.7 M.

The deadline to complete the acquisition of the White Pine North Project from CRC has been extended to January 31, 2020. A large portion of the Mineral Resources are located on the CRC property. There can be no assurance that the Company will be able to complete the acquisition of the White Pine North Project.

In April 2015, Highland entered into an agreement with Great Lakes Resources, LLC to lease certain mineral rights covering an area of approximately 1,816 acres within the White Pine North project area. The mineral lease is for 20 years, with an option for an additional 5 years.

2.1 Scope of Work

GMSI was retained by Highland to prepare a Technical Report in accordance with Canadian Instrument 43-101 (“NI 43-101”) *Standards of Disclosure for Mineral Projects* for the White Pine North Project located in the western sector of the Upper Peninsula of Michigan, USA.

This Report supports the results of the Preliminary Economic Assessment (“PEA”) and Mineral Resource Estimate (“MRE”) as disclosed in Highland’s press release entitled “Highland Copper announces positive PEA results and mineral resource estimate for the White Pine North Copper Project in Michigan” dated September 23, 2019.

The reader is advised that a PEA is preliminary in nature and is intended to provide only an initial, high-level review of the Project potential and design options. The PEA mine plan and economic model include numerous assumptions and the use of Inferred resources. Inferred resources are too speculative geologically to have the economic considerations applied to them that would enable them to be categorized as Mineral reserves and to be used in an economic analysis except as allowed for in PEA studies. There is no guarantee that Inferred resources can be converted to Indicated or Measured Resources, and as such, there is no guarantee the Project economics described herein will be achieved.

This Report has several cut-off dates for information:

- The effective date of the Current Mineral Resource is August 30, 2019;
- The effective date of this Report is September 22, 2019.

The PEA is focused on the extraction and processing of potentially economic mineralization from the White Pine North deposit which lies to the north of the former White Pine Mine.

The PEA scope includes the following main aspects:

- Mineral resource drilling and mineral resource estimation;
- Geotechnical assessment and updated mine design criteria;
- Mine engineering, including mine design and production schedule;
- Metallurgical testing confirming historical metallurgical performances of the former White Pine Mine;
- Simplified metallurgical flow sheet;
- Power supply options evaluation;
- Infrastructure requirements;
- Tailings disposal evaluation using historical tailings impoundment footprint;
- Estimation of operating expenditures (OPEX) and capital expenditures (CAPEX) for the Project;
- Economic analysis.

2.2 Sources of Information and Data

Some of the information and data contained in this Report was obtained from Highland; sources included the previously published NI 43-101 technical report prepared by Rod Johnson & Associates Inc. in February 2014 and references cited in this report. . However, this report did not include a mineral resource estimate.

GMSI has sourced information from historical reports and appropriate reference documents as cited in the text and summarized in Section 27 of this Report. GMSI has relied upon other experts in the fields of mineral tenure, surface rights, permitting and environment as outlined in Section 3.

2.3 Qualifications and Experience

GMSI was responsible for the overall PEA. A summary of the Qualified Persons ("QPs") responsible for each section of the Report is detailed in Table 2.1.

Table 2.1: Summary of Qualified Persons

	Qualified Person	Company	Report Sections
1	Louis-Pierre Gignac, M.A.Sc., P. Eng.	GMSI	1, 2, 3, 4, 5, 6, 19, 21.4, 22, 25.5-25.6, 27
2	Réjean Sirois, P. Eng.	GMSI	7, 8, 9, 10, 11, 12, 14, 23, 25.1.1, 26
3	Carl Michaud, P. Eng.	GMSI	15, 16, 25.1.2
4	Paul Murphy, P. Eng.	GMSI	13, 17, 18, 20, 21.1 to 21.3, 24, 25.3-25.4

2.4 Site Visits

Mr. Réjean Sirois visited the site from January 13th to January 16th, 2014 to review information and to confirm drill logs, assay certificates, sample intervals, downhole survey information and field checks to validate drill collars. He returned on October 16th to October 18th, 2019, visited the core shack, reviewed drill hole collars and samples, and visited the site aboveground.

Mr. Louis-Pierre Gignac visited the White Pine site from September 15th to September 20th, 2014 when an underground tour of the former White Pine Mine and various planning meetings regarding site infrastructure were held. The meetings were attended by Highland personnel, including Mr. Carlos H. Bertoni, Vice President, Exploration.

Mr. Carl Michaud has not visited the White Pine site.

Mr. Paul Murphy has not visited the White Pine site.

2.5 Units of Measure, Abbreviations and Nomenclature

Unless otherwise indicated, this Technical Report uses Canadian English spelling, USA dollar currency and System International (metric) units. Coordinates in this Technical Report are presented in metric units metres or kilometres using the Universal Transverse Mercator ("UTM") projection (UTM Zone 16, NAD83 datum). Elevations are reported as metres above mean sea level.

A list of the main abbreviations and terms used throughout this Report is presented in Table 2.2**.**

Table 2.2: List of Main Abbreviations

Abbreviations	Full Description
amsl	Above Mean Sea Level
Actlab	Activation Laboratories Ltd.
AX	AX Size Core; Core Diameter 3.01 cm
G	Billion
Ga	Billion years
BCM	Bank Cubic Meter
BSZ	Basic Shear Zone / Basal Gouge Zone
BX	BX Size Core; Core Diameter 4.20 cm
CAPEX	Capital Expenditures
CBS	Copper Bearing Sequence
cm	Centimetre
CN	Canadian National
CFM	Cubic foot per minute
CoV	Coefficient of variation
CPG	Certified Professional Geologist
Chesborough	A.M. Chesborough
CIM	Canadian Institute of Mining Metallurgy and Petroleum
CRI	Copperwood Resources Inc. (formerly known as Orvana Resources US Corp.)
CRM	Control Reference Material
CSA	Canadian Securities Administrators
CSF	Confinement Strength Factor
Cu	Copper
°	Degrees (Azimuth or Dip)
°C	Degrees Celsius
Dmt	Dry metric tonne
E	East
EIA	Environmental Impact Assessment
Eng	Engineering
ERP	Enterprise Resource Planning
FS	Feasibility Study
ft	Feet
FEMA	Federal Emergency Management Agency
Fe-O	Iron Oxide
G&A	General & Administration
GMSI	G Mining Services Inc.
Golder	Golder Associates Ltd.
GLGT	Great Lake Gas Transmission

Abbreviations	Full Description
g	Grams
g/t	Grams per Tonne
ha	Hectares
HDPE	High Density Polyethylene
Highland	Highland Copper Company Inc.
HQ	HQ Size Core; Core Diameter 6.35 cm
ICP OES	Inductively Coupled Plasma Optical Emission Spectrometry
IDB	Influent Design Basis
IEC	International Electrotechnical Commission
IRR	Internal Rate of Return
IRS	Internal Revenue Service
ISO	International Organization for Standardization
KLA	Keweenaw Minerals, LLC
Kg	Kilogram
k/t	Kilogram per tonne
km	Kilometre
kV	Kilovolt
LAN	Local Area Network
LCCS	Low Cost Country Sourcing
LiDAR	Light Detection and Ranging
l	Litre
LHD	Load Haul Dump
LCBS	Lower Copper Bearing Sequence
LLC	Limited Liability Company
LoM	Life of Mine
Lyco	Lycopodium Limited
METCON	Metcon Research
m	Metre
m/d	Metres per day
masl	Metres above sea level
MDEQ	Michigan Department of Environmental Quality
MDNR	Michigan Department of Natural Resources
MDOT	Michigan Department of Transportation
MST	Nonferrous Metallic Minerals Extraction Severance Tax
µm	Micron
mm	Millimetre
Mt	Million Tonnes
Mtpa	Million tonnes per annum

Abbreviations	Full Description
MACRS	Modified Accelerated Cost Recovery System
GEOID03	National Geodetic Survey Geoid 03
N	North
NAD83	North American Datum 1983
NAVD88	North American Vertical Datum 1988
NI 43-101	National Instrument 43-101
NI 43-101CP	National Instrument 43-101 Companion Policy
NI 43-101F1	National Instrument 43-101 Form 1
NNG	Northern Natural Gas
NPV	Net Present Value
NQ	NQ Size Core; Core Diameter 4.80 cm
NREPA	Natural Resources and Environment Protection Act
NSR	Net Smelter Return
NCNST	North Country National Scenic Trail
NREPA	Natural Resources Environmental Protection Act
OK	Ordinary Kriging
OPEX	Operating Expenditures
PMWSP	Porcupine Mountains Wilderness State Park
Osisko	Osisko Gold Royalties Ltd.
Orvana	Orvana Minerals Corp.
lb	Pound(s)
%	Percent
PE	Professional Engineer
Project	Copperwood Project
QA/QC	Quality Assurance/Quality Control
QP	Qualified Person
REI	Resource Exploration Inc
R&P	Room and Pillar
Ag	Silver
S	South
Sage	Sage Minerals Inc.
SG	Specific Gravity
SGS	SGS Lakefield
SGCN	Michigan Species of Greatest Conservation Need
km^2	square kilometre
TC/RC	Treatment Charge and Refining Costs
TDF	Tailings Dam Facility
TSF	Tailings Storage Facility

Abbreviations	Full Description
TDM	Tailings & Water Disposal Management
3D	Three Dimensional
t	Tonnes
tpa	Tonnes per annum
tpd	Tonnes per day
tpy	Tonnes per year
UCBS	Upper Copper Bearing Sequence
US$	United States Dollars
USA	United States of America
USGPM	US Gallon per minute
USG	US Gallon
USFWS	U.S. Fish and Wildlife Service
USMR	United States Metals Refining Company
UTM	Universal Transverse Mercator
WBS	Work Breakdown Schedule
WC	Working Capital
WTP	Water Treatment Plant
WWTP	Waste Water Treatment Plant
W	West
wt.%	Weight Percent
yr	Year

3. RELIANCE ON OTHER EXPERTS

This Report has been prepared by GMSI for Highland .The information, conclusions, opinions, and estimates contained herein are based on:

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

Certain sections of the Report rely on reports and statements from legal and technical experts who are not Qualified Persons ("QP") as defined by National Instruments 43-101 ("NI 43-101"). The QPs responsible for preparation of this Report have reviewed the information and conclusions provided and determined that they conform to industry standards, are professionally sound and are acceptable for use in this Report.

The following companies and consultants have been retained by Highland to prepare some aspects of this Report. Their involvements are listed below:

- GMSI has relied upon information provided by Highland including legal opinions concerning certain mineral rights prepared by Kendricks, Bordeau, Adamini, Greenlee & Keefe, P.C., a Michigan law firm, and a commitment for title insurance issued by First American Title Insurance Company for the surface rights;
- Concept Consulting LLC conducted a rail transportation study for the White Pine North Project. GMSI relied on this report for concentrate transportation costs and the selection of a transload facility location;
- GMSI has relied on input from KPMG regarding the taxation model and calculations used to estimate after-tax cash flows in the economic model;
- GMSI has relied on geotechnical input from a review of historical pillar dimensioning of the former White Pine mine conducted by Itasca Consulting;

This Report is intended to be used by Highland as a Technical Report with Canadian Securities Regulatory Authorities pursuant to provincial securities legislation. Except for the purposes contemplated under provincial securities laws, any other use of this Report by any third party is at the party's sole risk.

Permission is given to use portions of this Report to prepare advertisement, press releases and publicity material, provided such advertisement, press releases and publicity material does not impose any additional obligations or create liability for GMSI.

4. PROPERTY DESCRIPTION AND LOCATION

4.1 Location

The White Pine Project is located in the Upper Peninsula of the State of Michigan, USA, approximately 7.5 kilometers (“km”) south of Lake Superior in Ontonagon County at 46º 45’ 42” N latitude and 89º 33’ 52’ W latitude (UTM coordinates 5181816N, 304,170E). The county seat is Ontonagon, 25 km northeast of the Project.

The White Pine Project covers approximately 4,500 hectares (11,000 acres) of surface rights and approximately 11,990 hectares (29,615 acres) of mineral rights. Surface and mineral rights are located in portions of Township 51N Range 42W, Township 51N Range 41W, Township 50N Range 42W, and Township 50N Range 41W in the Township of Carp Lake, Ontonagon County, Michigan as shown on Figure 4.1. Third party properties are also shown on Figure 4.2. Those areas overlie mined-out portions of the former White Pine Mine, the mine portal, a refinery and a power plant, none of which are being acquired by the Company.

Figure 4.1: White Pine Location Map

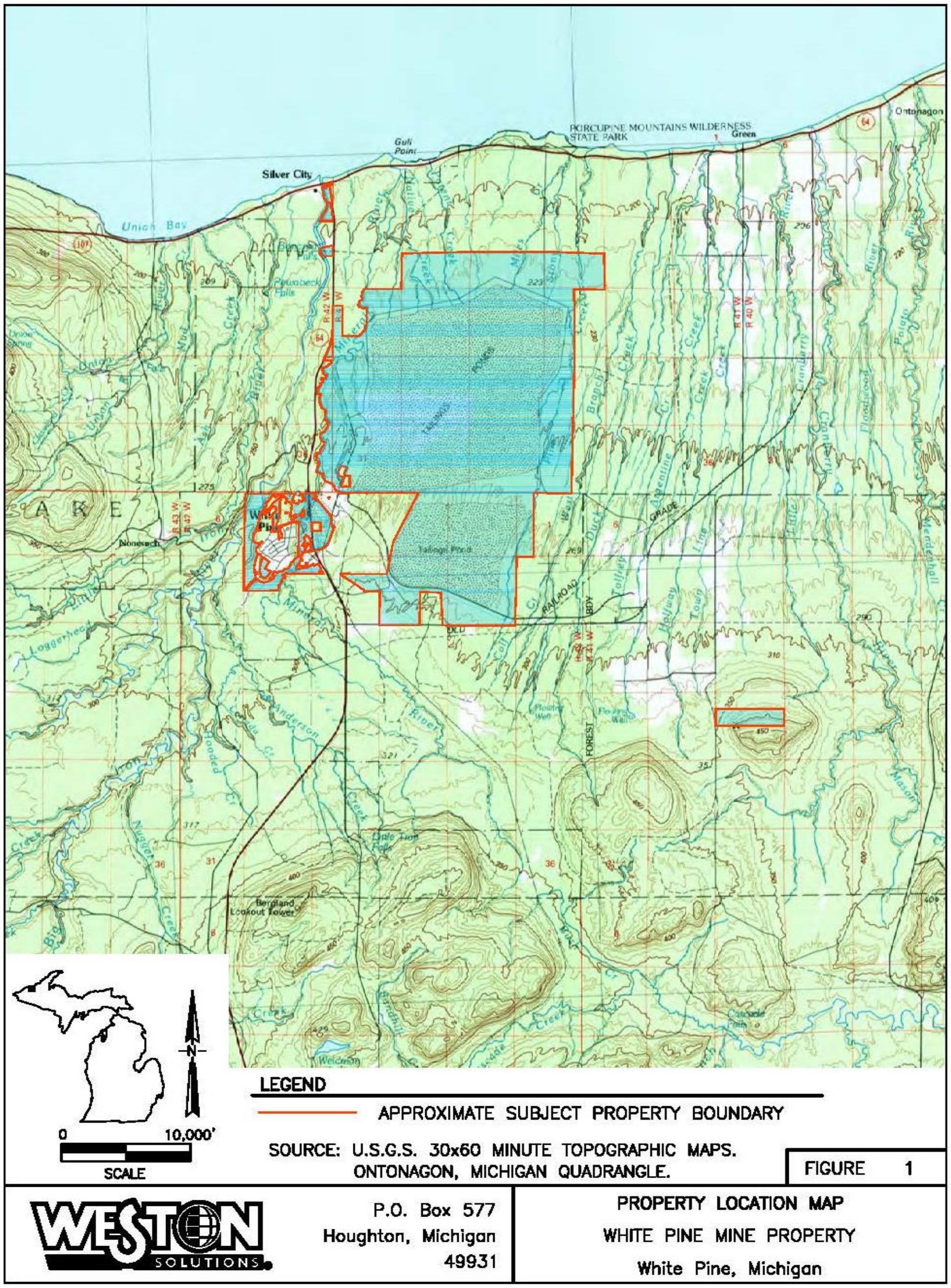
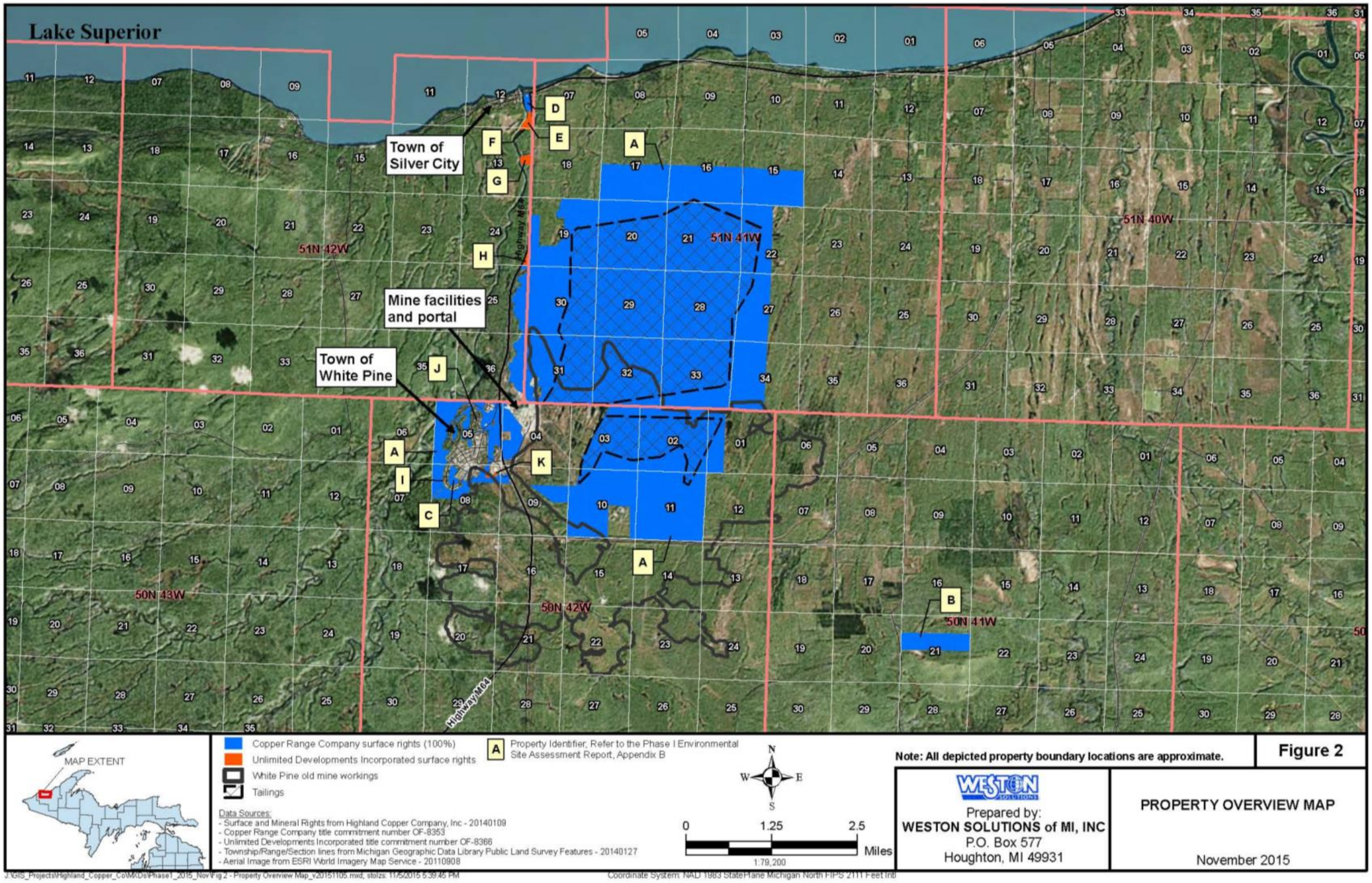


Figure 4.2: White Pine Property Outline (in red) showing Copper Range Surface Ownership



4.2 Michigan Property Rights

Ownership of mineral resources in Michigan was originally granted to the persons who owned the surface. These property owners had both "surface rights" and "mineral rights". This complete private ownership is known as a "fee simple estate". Mineral rights may be severed from the surface estate and held by separate parties. Where severed from the surface rights, the mineral rights become subject to Michigan's *Marketable Record Title Act* of 1945, as amended.

Surface and mineral rights in Michigan are located and described with reference to a grid established by the federal government as part of the Public Lands Survey System. Townships are squares of 36 km^2 comprising 6 x 6 arrays of 36 sections, named according to distance and direction from a principal meridian and baseline. Sections are one-mile square, and can be divided into quarters, labeled NE, NW, SE, and SW. Each quarter can also be split into halves or quarters, which are labeled according to the side or corner of the quarter section they encompass (e.g., NE 1/4 of the NW 1/4).

The township and range grid in the White Pine area was established in 1851. Curvature of the earth and survey errors both result in variations in the sizes of the townships and sections. Section boundaries are usually marked in the field by small survey monuments.

4.3 Highland Copper Interest in the White Pine Project

4.3.1 Mineral and Surface Rights Acquisition

On March 5, 2014, the Company entered into an asset purchase agreement setting out the terms and conditions under which a wholly-owned subsidiary of Highland will acquire all of the rights, title and interest of Copper Range Company ("CRC"), a subsidiary of First Quantum Minerals Ltd., in the White Pine Project.

On May 8, 2014, the Company and CRC completed an interim closing of the acquisition of the White Pine Project. The final closing of the acquisition is subject to a number of conditions including (i) replacing a US$2.85 million financial assurance bond associated with the remediation and closure plan of the historical White Pine mine site in a manner that is acceptable to all parties involved, including the applicable governmental authorities (it is expected that the bond to be posted will initially be for an amount of approximately US$1.7 million) (ii) releasing CRC from its environmental obligations with the Michigan Department of Environment, Great Lakes, and Energy ("EGLE") (previously referred to as the Department of Environmental Quality) under a Consent Decree (see Section 4.6); and (iii) obtain a consent from a third party to the assignment by CRC to Highland Copper of certain contractual obligations. The deadline to complete the final closing has been extended several times. The current deadline is January 31, 2020.

At closing, the Company will undertake to pay to CRC an amount equal to US$0.005 (one half of one cent) per pound for the first 1 billion pounds of proven and probable reserves of copper and US$0.0025 (one quarter of one cent) for each additional pound of proven and probable reserves of copper as determined in a feasibility study. The payment can be made in cash or in common shares of the Company, at the option of CRC.

The Company has access to perform exploration and other activities associated with the development of White Pine under an access agreement with CRC. The Company reimburses CRC quarterly for all direct maintenance and monitoring costs required to be done on the property.

The properties being acquired from CRC comprise: (i) areas of mineral rights 100% owned by CRC; (ii) areas where CRC holds a fee simple interest in both the surface rights and mineral rights; and (iii) four areas where CRC holds partial (75%) mineral interests. Michigan law provides that, where multiple parties own the mineral rights in a parcel of property, any owner holding at least 75% of the mineral rights may obtain a court decree allowing that owner to explore and develop the minerals under that parcel. As part of the interim closing, a commitment for title insurance on the fee simple interests was issued and the Company received a title opinion to confirm the ownership by CRC.

4.3.2 Mineral Lease Agreement

In April 2015, the Company entered into an agreement to lease certain mineral rights from Great Lakes Resources LLC ("Great Lakes") located in White Pine. The leased mineral rights cover an area of approximately 1,816 acres. No survey has been conducted. The mineral rights are located within portions of Sections 20, 21, 28 and 33 of Town 1 North, Range 41 West and portions of Section 36 of Town 51 North, Range 41 West. A title opinion on the leased mineral rights has been prepared at the Company's request by Ronald Greenlee of Kendricks, Bordeau, Keefe, Seavoy & Larsen, P.C.

Of an initial closing payment of US$800,000, a balance of US$165,000 at June 30, 2019 remains to be paid to Great Lakes in equal quarterly payments of US$27,500. The mineral lease with Great Lakes is for 20 years, with an option for an additional five years. Annual lease payments are $25,000 for the first five years, $30,000 for the sixth and seventh years and $1,000,000 thereafter. Beginning on the eight (8^{th}) anniversary, all annual rentals paid by the Company will be treated as advance royalty payments and will be a credit in favor of the Company against the future production royalty to be paid. Upon commencement of production, the Company will have to pay a sliding scale royalty on copper and silver production from the leased mineral rights with a base royalty of 2% for copper and 2.5% for silver. The Company has an option to repurchase 50% of the royalties.

4.4 Title Over the Mineral Resource Area

The Mineral Resources reported in this Report is covered by mineral rights held by CRC (to be transferred to the Company at the final closing) and Great Lakes leased mineral rights located in portions of Township 51N Range 41W in the Township of Carp Lake, Ontonagon County, Michigan. The Company has obtained a title opinion from Ronald Greenlee of Kendricks, Bordeau, Keefe, Seavoy & Larsen, P.C. confirming that CRC and Great Lakes owns the mineral rights over the area covering the Mineral Resources.

4.5 Permits

Most of Michigan's environmental regulations are referred to as "Part(s)" and are contained in the Natural Resources and Environmental Protection Act, 1994 PA 451, as amended ("NREPA"). The Oil, Gas, and Minerals Division of the Michigan Department of EGLE administers Part 625, Mineral Wells, of the NREPA. This statute and the promulgated rules govern aspects of well location, drilling, operation, plugging, and restoration for solution mining wells, brine production wells, certain types of disposal wells, and test wells associated with mineral exploration and extraction. In addition, test wells must meet the requirements of other Parts of the NREPA to prevent damage to water, air, soil, wetlands, and other environmental values.

Nonferrous minerals such as copper and silver are regulated by Part 632, Nonferrous Metallic Mining. Part 632 provides a regulatory framework for construction, operation, and reclamation of mining operations required for the safe and environmentally sustainable extraction of these metallic minerals. The Oil, Gas, and Minerals Division of the Michigan Department of EGLE is responsible for the implementation of Part 632.

Activities which impact "regulated wetlands" and/or "inland lakes or streams" may require a joint permit from the MDEQ under Part 303 (Wetland) and/or Part 301 (Inland Lakes and Streams) of NREPA.

The nonferrous metallic mining industry is also regulated by other environmental statutes and divisions within the EGLE such as Air Quality Division and Water Resources Division.

4.6 Environmental Liabilities

The historical mining, ore processing and smelting operations at the former White Pine mine property (the "Mine Property") until 1995 resulted in releases of hazardous substances on and beneath the former Mine Property and is a known site of environmental contamination. The Mine Property include a portion of the area identified as the "White Pine Mine Facility" in a consent decree between CRC, Michigan's Attorney General, and EGLE, dated October 1997, as amended (the "Consent Decree"). Pursuant to the Consent

Decree, CRC has been addressing the identified environmental impacts at the Mine Property by undertaking certain environmental response activities. These include, soil relocation, source control, capping and re-vegetation, groundwater monitoring, and storm water management with effluent discharges as permitted under a National Pollutant Discharge Elimination System (“NPDES”) permit, and land use restrictions in the form of recorded declarations of restrictive covenants.

The environmental response activities are being implemented through response activity plans (“IRAPs”) and a remedial action plan (“RAP”) approved by the MDEQ on October 13, 2005. Extensive remedial actions have occurred to address identified impacts and recognized environmental conditions. To a large extent, the environmental response activities have been completed. However, on-going responsibilities and liabilities remain.

Upon completion of the final closing of the acquisition of the mineral and surface rights from CRC, the Company will assume all environmental liabilities related to the Consent Decree and on-going environmental obligations. These on-going responsibilities and obligations include:

- Consent Decree
    - Completion of all required environmental response activities.
    - Quarterly written progress reports.
    - Maintenance of a financial assurance mechanism to cover the anticipated costs of future environmental response activities.
    - Completion and EGLE approval of an underground mine closure plan.
    - Comprehensive general liability insurance coverage with limits of $15 million.
- IRAPs and RAP
    - Compliance with recorded declarations of restrictive covenants.
    - Quarterly engineered barrier inspection and maintenance.
    - Runoff monitoring from uncapped slag pile areas that do not drain to the NPDES system.
    - Bedell Pond constructed passive wetland maintenance.
    - Portal Creek biological monitoring to determine recovery status.
    - On-site response action repository inspection.
    - Permanent marker maintenance.
    - Execution of a post-closure agreement between Copper Range and the MDEQ.
- National Pollutant Discharge Elimination System (“NPDES”) Permit
    - Routine monitoring of effluents from the NPDES System and discharge compliance. The NPDES permit issued to CRC was renewed in 2016.

  - NPDES permit renewal application submitted to EGLE in April 2019, current permit expires October 1, 2019.
- Underground Mine Closure Plan
  - Maintenance of a “fresh water cap”.
  - Removal of all contaminants in the underground mine
  - Flooding of the underground mine.
  - Post-flooding sampling and analysis program.
- Groundwater Monitoring
  - Routine monitoring and data evaluation to assess plume movement and natural attenuation of metals (primarily barium, lithium, manganese, and strontium).
  - Monitoring associated with the on-site response action repository.
  - Monitoring continues until no risk is demonstrated.
- Re-vegetation of the tailings basins
  - Achieve minimum 70% effective vegetation cover and monitor for 5 years after achieving effective cover.
  - Achieve diversity of 5+ species in at least 3 of 5 years after ceasing augmented management.
  - North #2 Pond has met objectives, North #1 and North #2 Cyclone Sands Area have not and are surveyed annually.
- Dam Safety (to prevent failure and tailings release)
  - Routine monitoring (operational inspections) and maintenance.
  - Formal inspection of North No. 1 every 5 years (low hazard potential).
  - Formal inspection of North No. 2 every 4 years (significant hazard potential).
  - Maintenance of an up-to-date emergency preparedness response plan.

5. ACCESSIBILITY, CLIMATE, LOCAL RESOURCES, INFRASTRUCTURE AND PHYSIOGRAPHY

5.1 Accessibility

The White Pine North Project is in the Upper Peninsula of the State of Michigan, USA, approximately 7.5 kilometres (“km”) south of Lake Superior in Ontonagon County at 46° 45’ 42” N latitude and 89° 33’ 52’ W longitude (UTM coordinates 518,816N, 304,170E). The county seat is Ontonagon, 25 km northeast of the Project.

The area is accessible via Michigan State Highway 64, which runs north-south 0.5 km west of the Project footprint. The nearest airports serviced by commercial flights are Ironwood, Hancock and Marquette.

5.2 Climate

The Project area has a humid continental climate strongly influenced by the proximity to Lake Superior, characterized by weather patterns usually known as “lake effect”, which affects temperature, precipitation and cloud cover. This lake effect phenomenon exists because the Lake Superior water warms and cools more slowly than the surrounding air. This proximity also results in increased precipitation because large amounts of moisture are available for air masses as they travel across the lake. With the predominant wind direction from the west, the lake effect is exacerbated in the winter as cold air blows across the warmer lake, acquiring moisture and dumping heavy snowfall as it meets the colder land mass.

The average annual rainfall in Ontonagon County is 86 centimetres (“cm”) (33.9 inches (“in.”) with snowfall of 459 cm (177 in.) at White Pine. There are on average 185 sunny days and 142 days with precipitation. The average July high temperature is 26°C (78.8°F) and the average January low temperature is -16°C (3.7°F).

5.3 Local Resources

The Project is close to several communities, including White Pine, Ontonagon, Bergland, Wakefield and Ironwood, all suffering from declining economic activity in the region and population loss, particularly young people. These communities offer ample real estate opportunities for the influx of mine workers. The unincorporated town site of White Pine lies immediately across M-64, 0.6 km to the southwest of the mine site and had a population of 474 persons in the 2010 census. The town was built during the construction of the present White Pine Mine in 1952 to service employees of the mine. White Pine underwent an expansion during 1968-1969. The town site provides access to a restaurant and motel complex and a small mall, containing the post office and a bank. The major population centers for the region are Houghton, located about 111 km to the northeast with a population of 7,708 in 2010 and Marquette, located about 201 km to the east with a population of 21,355 in 2010.

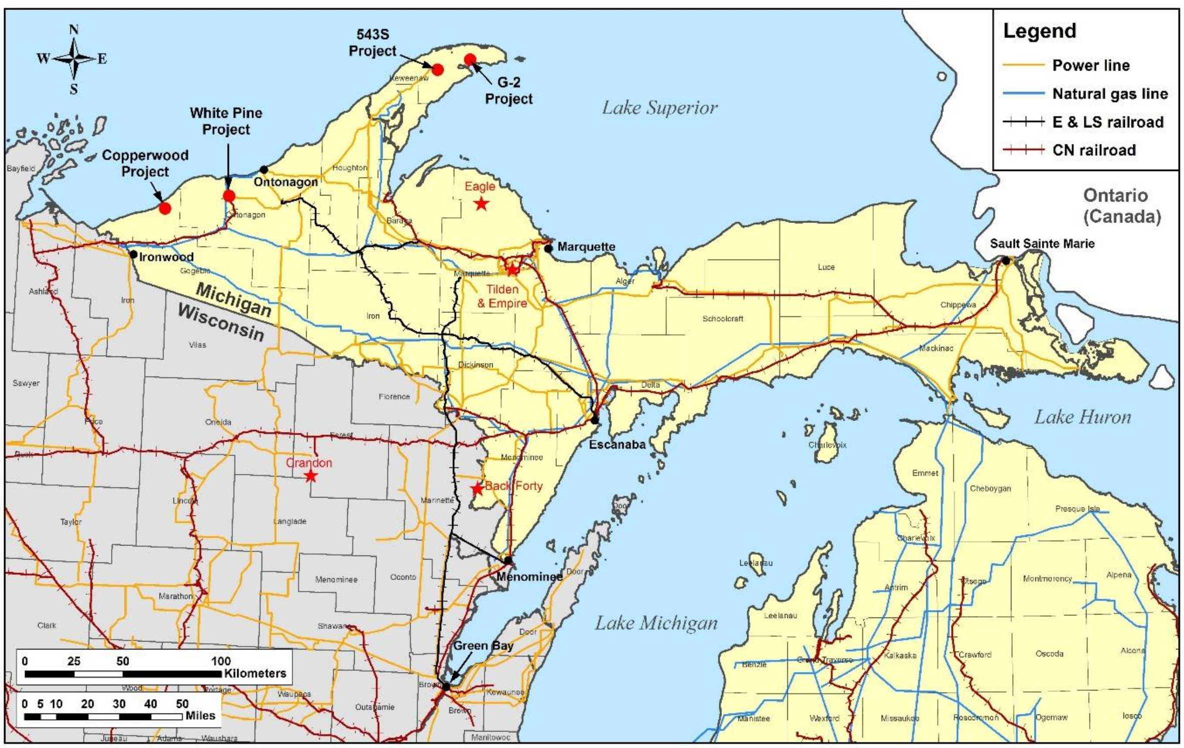
The schooling system in the region is good, with an outstanding engineering university at Houghton (Michigan Tech), a community college at Ironwood and the University of Northern Michigan in Marquette.

Even with a relatively low level of economic activity, there are several service groups available in the region, including engineering, environmental, analytical, transportation, etc. In the Marquette area, there are several service companies that cater to the needs of both Cliffs and Eagle mining operations.

5.4 Infrastructure

The Michigan Upper Peninsula has well developed infrastructure, with paved road, optic fiber, natural gas, power grid and rail assets available (Figure 5.1). There are also lacustrine ports at Marquette and Ontonagon (to be rehabilitated), which could receive and ship bulk goods.

Figure 5.1: Map of Available Infrastructure in the Upper Peninsula of Michigan



5.4.1 Roads and Railroads

Michigan State Highway 64, running north-south along the Project area, is linked to both the Michigan and federal highway systems nationwide.

The Project area is accessible by a Canadian National ("CN") rail spur, which leads to the Morengo junction in Ashland County, Wisconsin (Figure 5.2). The other nearest rail spur is in Ontonagon County, owned by Escanaba and Lake Superior Railroad, which leads southwest to Escanaba and connects to the CN rail grid. Both rail spurs would need to be refurbished if to be used commercially by a new mining operation.

5.4.2 Services Buildings and Ancillary Facilities

Copper Range Copper ("CRC") decided to sell all existing facilities upon closure in 1995 and several parties bought various buildings and parcels of land of what was called "White Pine Industrial Park" (Figure 5.3). The processing plant and smelter were dismantled, but the power plant, refinery and other buildings kept and sold. Some of these buildings could be repurposed and used for a new mining operation.

The White Pine copper refinery is a modern facility that was part of a fully integrated copper producing operation that included a smelter. The refinery treated Hudson Bay anodes from Flin Flon, Canada, until it was sold again in 2011. It has a design capacity of 80,000 tonnes per year ("tpy") and consists of an electrolytic copper refinery using Mt. ISA stainless steel technology, including a modern EMEW electro winning plant commissioned in 2008 with rated capacity of 1,500 tpy, an AISCO anode preparation machine, and a MESCO cathode stripping and Sumitomo anode scrap washing machines.

The White Pine Mine underground facilities and its surface footprint were sold to SubTerra, a subsidiary of a Canadian group that intends to use the available underground openings to grow marijuana, when it becomes legal in Michigan.

Figure 5.2: Aerial of White Pine "Industrial Park" Facilities as of 2014

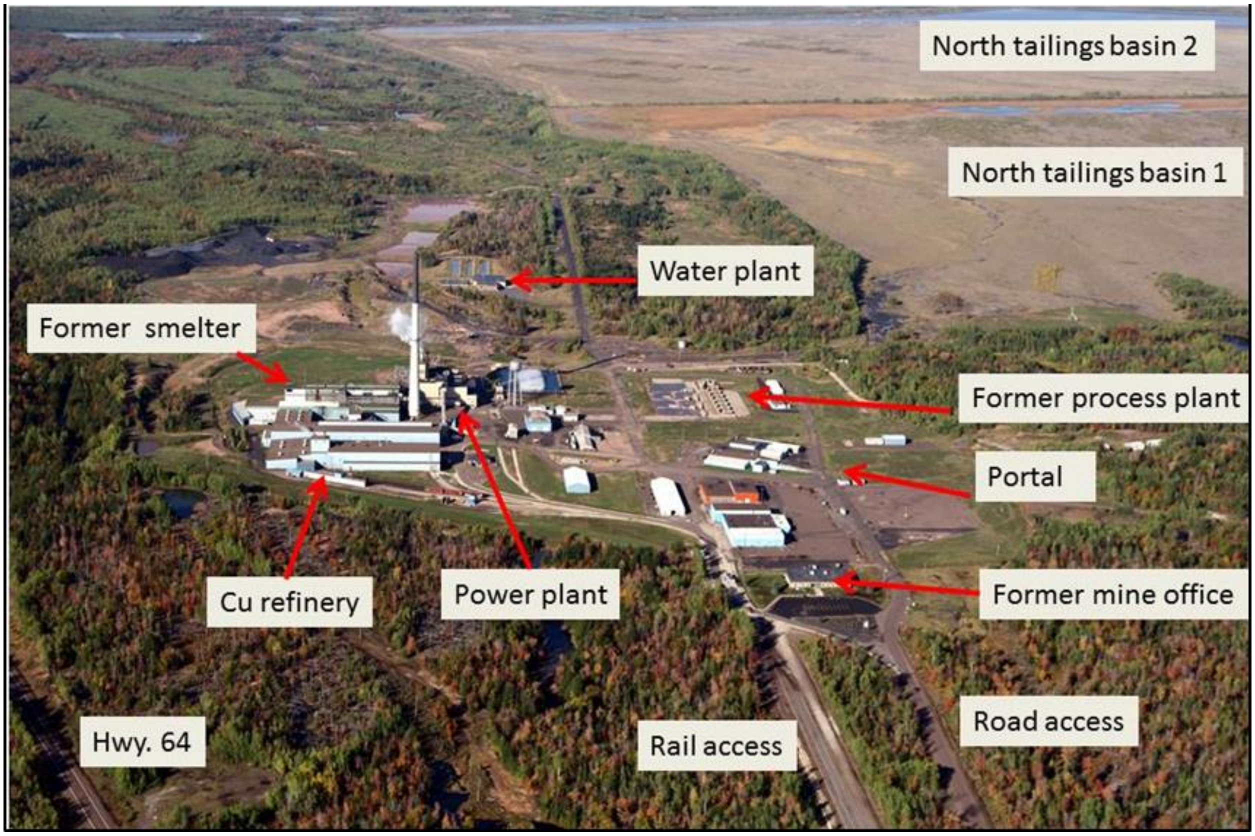
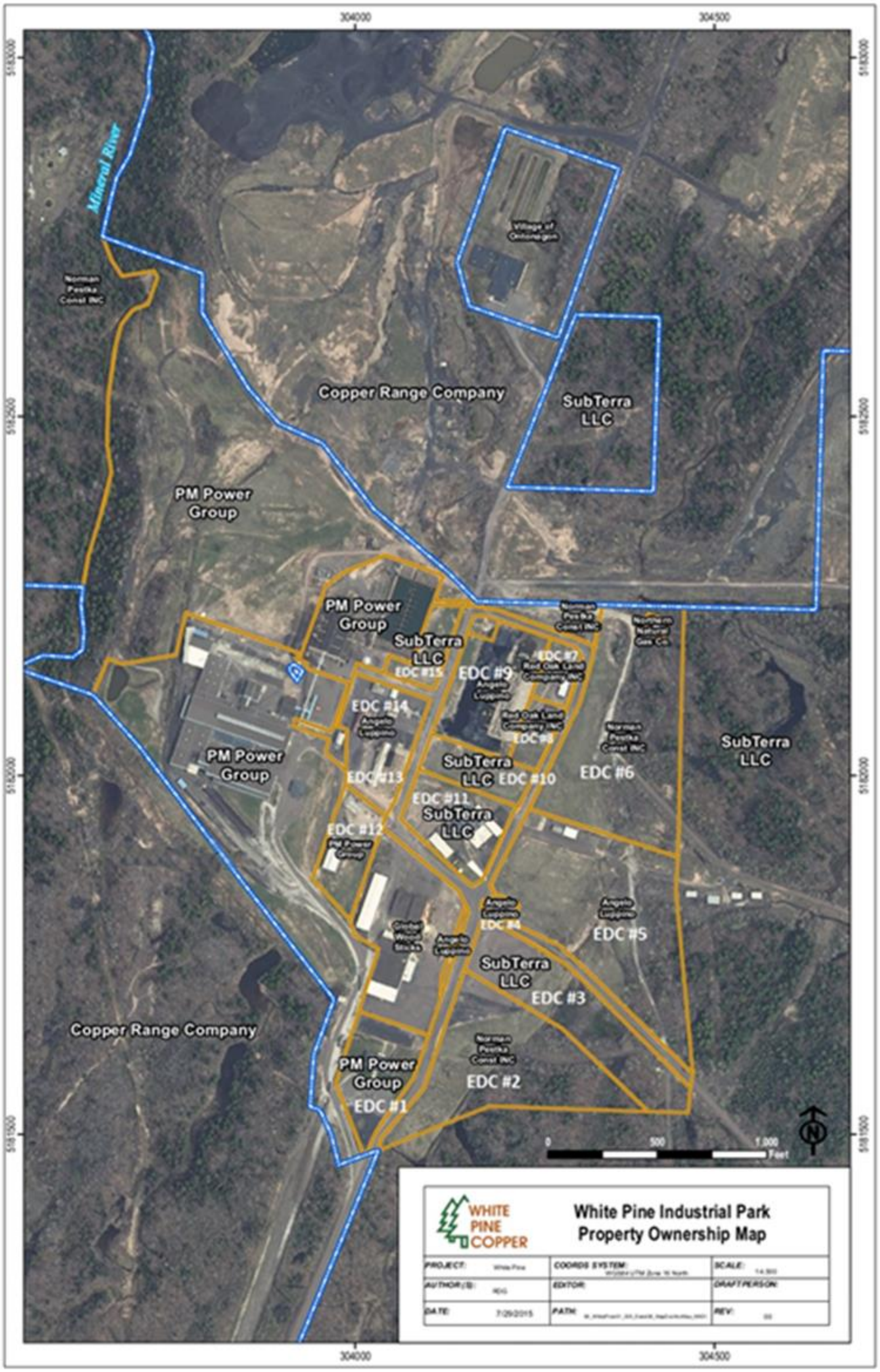


Figure 5.3: Layout of White Pine "Industrial Park" Ownership



5.4.3 Power Supply and Distribution

The White Pine Project site is at the boundary of two utility franchises: Xcel Energy to the west and UPPCO to the east and neither are currently able to supply power at the envisaged loads. Excel´s grid at 138 kV capacity stops in Ironwood and ATC´s at Ontonagon. The site is currently served by a 69 kV line coming from the east. ATC has medium-term plans to link its 138 kV grid with Xcel´s by constructing a new line, which would run by the vicinity of White Pine.

The Project is served by two natural gas pipeline systems, which could be tapped to produce thermal power. The Northern Natural Gas pipeline reaches White Pine, and the Great Lakes pipeline runs about 20 km to the south. The Northern Natural Gas pipeline services the power plant at White Pine (30 MW nominal capacity), which used coal when the mine was in production, and until recently was part of the grid as a “peaking” plant called to service in periods of high demand. This plant could theoretically be used by a mining operation if its refurbishment and operation are economic.

In 2015, the government of Michigan announced an agreement between utilities and the largest power consumer in the Upper Peninsula, Cliffs Natural Resources, by which Cliffs would purchase most of its power from the coal-fed Presque Isle plant in Marquette until the facility's retirement in 2020. A replacement natural gas-fired cogeneration power plant would be built, owned and operated by Invenergy on Cliffs' property in Marquette County and make abundant power available for other industries in the UP.

5.4.4 Water

CRC installed a robust water supply system for all its needs based on water intake from Lake Superior. With the closure of the White Pine Mine, the Village of Ontonagon acquired, in 2001, the company´s raw water intake, pumping equipment, and supply pipes. The Village also acquired the potable water distribution system for the community of White Pine. A 36 in. concrete raw water supply pipe from Silver City was repurposed for potable water distribution to serve Silver City and Ontonagon. A 42 in. steel raw water supply pipe provides water from Lake Superior to the White Pine mine site.

The water intake is located off the mouth of the Big Iron River in Silver City and it was constructed by mining a tunnel under Lake Superior. The current water withdrawal limit, based on pumping capacity, is 26 million gallons per day (“MGPD”), verified by the Michigan Department of Environmental Quality – Drinking Water Division staff. The pumping station is located just west of the intersection of M-64 and M-107 on the south side of the highway - five pumps are on site, but only two of the original pumps are still used (600 hp, 7,000 gpm). The other two original pumps were removed and replaced with three smaller pumps

(2,000 gpm). Power to the pumps is a three phase, 600 A service, 480 V load, backed-up by a 500 kW diesel generator.

The 42 in. raw water supply pipe is located adjacent to highway M-64 between Silver City and White Pine, climbing about 80 m from the lake. The pipe crosses the highway once and has a length of 7.4 km from the pump house in Silver City to the White Pine site. There is a water tower by the former mine plant with a capacity of 200,000 gallons. Raw water is distributed at various locations by the mine site and pipes would require reconnections to existing or abandoned service lines and pumps.

The Village of Ontonagon owns a water treatment plant a few hundred meters north of the former mine site, with a capacity of 2.1 MGPD of potable water, from where water is redistributed to local communities. The use of raw and treated water for a new mining operation would require a service contract with the Village of Ontonagon.

5.4.5 Supply

The Project is in the United States mid-west region and close to major heavy equipment, material and service suppliers.

5.5 Physiography

The elevation at the old White Pine mine portal is 265 m.a.s.l. The Project area is in the Lake Superior Lake Plain regional ecosystem, consisting of a landscape formed by water-reworked moraine and glacial till, drained by numerous small streams and wetlands, slopping gently north toward Lake Superior. On a larger scale, the region is within the southern boundary of the boreal forest (taiga) ecosystem which, locally, contains a variation noted as northern forest. This variation is characterized by coniferous forests, consisting mainly of pine, spruce, and larch mixed with areas of northern hardwoods, paper birch, and aspen. Nearly all of the virgin white pine in the area was logged off in the late 19th century and the area is now covered by second and third growth forest. The system is noted for the abundance of water but poor topsoil due to repeated glaciation.

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6. HISTORY

Native copper mining in the Lake Superior region dates back at least 5,000 years, evidenced by numerous ancient pits found along the length of the Keweenaw Peninsula. These ancient pits contain masses of native copper in various stages of removal together with crude stone tools that were used in mining the copper. The Lake Superior area source of this material is established by the unique presence of silver alloyed with the copper.

The first recorded mining operation started in 1771 with an attempt to recover copper from what turned out to be a large glacial erratic. A sustained copper industry in the Peninsula began in 1830 when Douglas Houghton, Michigan's first State Geologist, visited the region. His report in 1841 led to combined State and Federal topographic and geologic surveys in 1844 and in 1845 the Cliff Mine, the first underground mine on the native copper lodes, was opened. The Calumet and Hecla Mine, the largest by far in the district, was opened in 1864. Most of the native copper production came from a 20 km long belt between the towns of Houghton and Calumet and was mined both from amygdaloidal zones in the tops of basalt lava flows and from interbedded conglomerates. During its productive period, from 1845 to 1977, the Keweenaw district produced 11.5 billion pounds of copper from over 300 Mt of ore. All this production was from underground mines.

Concurrent with the development of the native copper mines, exposures of high-grade silver and copper as chalcocite in siltstone units of the Nonesuch Formation were discovered and mined west of the town of Ontonagon in the western Upper Peninsula. The discovery by the Copper Range Company ("CRC") in the 1930s that lower grade zones of chalcocite mineralization extended over a very large area, coupled with increasingly sophisticated metallurgical techniques for treating fine-grained sulfide mineralization, led to development of the White Pine Mine and subsequent discovery of the Copperwood deposit farther west. The White Pine Mine was in production from 1953 through 1995 with only a two-year interruption in 1984 - 1985. By the time it closed, over 2.04 billion kilograms (4.5 billion pounds) of copper had been produced from the Mine.

6.1 Prior and Current Ownership

In 1865, Frank Cadotte discovered the copper-bearing Nonesuch shale in an outcrop in the bed of the Little Iron River and the Nonesuch Mine was opened 3.2 kilometres west of the White Pine Mine. The copper-bearing shale formation was given the name Nonesuch because "no other deposit like it" existed in the Michigan copper district.

In 1879, Thomas Hooper, a Cornish mining captain, started the original White Pine Mine on the bank of the Mineral River. The property was named for the giant white pine trees in the area. Mining concentrated on the fine-grained native copper in the sandy portion of the conglomerate underlying the shale. The copper sulfide mineralization was known, but no economic method existed for its recovery. Lack of capital forced the closure of the original project in 1881.

In 1907, under the direction of Tom Wilcox, the Calumet and Hecla Mining Company ("C&H") purchased the properties and conducted diamond drilling from the original Nonesuch Mine eastward to the area that became the White Pine Mine. Thomas Hooper's original #1 shaft was deepened, and sandstone with greater than 10% native copper was discovered.

In 1912, an additional shaft was sunk by C&H, but it was discovered that a large fault had displaced the ore horizon to what was considered an unreasonable depth at that time. Two additional shafts were sunk to the east and production increased. In 1915, a railroad spur and 1,000-short ton-per-day capacity ball mill were constructed. Most of the smelting was done in Houghton.

From 1915 to 1920, C&H produced 18 million pounds of native copper and 260,000 oz of silver. In late 1920, the mine was closed because of a recession and depressed copper prices.

In 1929, William Schacht, acting as agent for CRC, attended a sheriff's auction for back taxes in Ontonagon and acquired the White Pine properties. It took another 23 years of research to determine an economic method to recover the copper sulfide that existed in the Nonesuch Shale.

In 1950, the outbreak of the Korean War forced the US to consider increasing domestic sources of copper. The federal government requested that CRC consider the completion of its plans to exploit the deposit at White Pine. Financing consisted of a loan of US $68M from the federal government under the Defense Production Act and US $13 M from CRC.

In March 1952, construction of the White Pine Mine began and on March 31, 1953, the first ore was hauled to surface via the portal. The mill was completed in 1954 and the first pour of copper in the smelter was on January 13, 1955.

In 1965, the one-billionth pound of copper was poured in the smelter. An expansion project in this year added an additional mine shaft, an additional mill section, and a second furnace in the smelter.

In 1975, Amax Inc. and CRC agreed to a merger. The US Justice Department followed the next month with an antitrust suit to block the merger and additionally to require Amax to divest itself of its 20% ownership of

CRC. The federal district court in New York later ruled in favor of the Justice Department and the merger failed.

In 1977, the Louisiana Land & Exploration Company ("LL&E") purchased the White Pine Mine and CRC became a wholly owned subsidiary of LL&E.

In 1982, LL&E closed the White Pine Mine and put the mine up for sale after continuing losses due to low copper prices and escalating production costs.

In 1984, Echo Bay Mines, Ltd. ("Echo Bay"), a Canadian company, purchased the LL&E interest in the Round Mountain (Nevada) gold mine. The deal required Echo Bay to acquire ownership of the White Pine Mine and all of its legacy environmental concerns. Echo Bay immediately began a plan for permanent closure of the White Pine Mine.

In 1985, Echo Bay agreed to sell CRC and the White Pine Mine assets to a management group and the mine employees. CRC was reorganized as a Delaware corporation owned 70% by an employee stock ownership plan and 30% by a management group called Northern Copper Corporation, headed by Russell Wood, a former Vice President of mining for LL&E.

In 1987, Plans were underway to take CRC public through an initial public offering ("IPO") to obtain a listing and initiate public trading in stock of the company. An unexpected, large, instantaneous ground failure occurred during the summer of 1987. The failed area occurred in an area of second-pass mining of mineralized Lower Sandstone. The system failed simultaneously and explosively in workings that ranged from 460 to 550 metres ("m") (1,500 to 1,800 ft) in depth. Final subsidence on surface ranged from one third to greater than one meter. The ground failure in the southwest part of the mine threatened production and the IPO was postponed indefinitely.

In 1989, Metall Mining Corporation ("Metall") and CRC announced an agreement for Metall to acquire CRC.

In 1993, CRC announced the initiation of studies to determine the viability of solution mining in the White Pine Mine. The MDEQ had issued a permit for this Study.

In 1995, Inmet Mining Corporation (formerly Metall) announced in July 1995 that CRC would suspend all conventional mining and milling operations at the White Pine Mine on September 30th. The smelter had been idled in February due to environmental concerns. The solution mining pilot program continued, as did operation of the refinery.

In 1996, CRC announced approval of the permit for a solution mining operation. Opposition from environmental groups and regional Native American tribes became so intense that the project was put on hold.

In 1997, CRC announced that it was dropping all plans for solution mining operations within the White Pine Mine and that all operations would cease. Plans for removal of all underground assets were begun. Plans for flooding the mine and negotiations with the State of Michigan for the final environmental agreement were undertaken. Reclamation plans for the tailings disposal sites began.

In 1998, the White Pine refinery was sold to HudBay Minerals and in 2011, HudBay Minerals ceased operations at the White Pine refinery and sold it to Traxys North America LLC.

In 2013, First Quantum Minerals, Ltd. took over Inmet Mining Corporation, the parent company of CRC, and acquired indirect ownership of what was left of the White Pine mine and the surrounding surface and mineral rights.

In 2014, Highland announced that its wholly owned subsidiary, Upper Peninsula Copper Holdings Inc., had completed the interim closing of the acquisition of the White Pine Project from CRC, a subsidiary of First Quantum Minerals Ltd. The final closing of the acquisition is still pending, subject to releasing CRC for financial assurance and assuming environmental obligations.

In April 2015, Highland entered into a 20-year lease agreement over certain mineral rights located within the White Pine North project area. The leased mineral rights cover an area of approximately 1,816 acres.

6.2 Exploration History and Historical Drilling

Although the copper mineralization crops out along the Mineral River, there are very few exposures of the Nonesuch Shale in the area. Drilling has been necessary to identify the distribution of copper mineralization. CRC conducted a continuous drilling program at the White Pine Mine until the early 1970s. There was a hiatus in drilling until the commencement of a drilling program in 1994-1995. The 1994 drilling program was conducted to provide a resource estimate supporting a feasibility study to build a new smelter at the White Pine site.

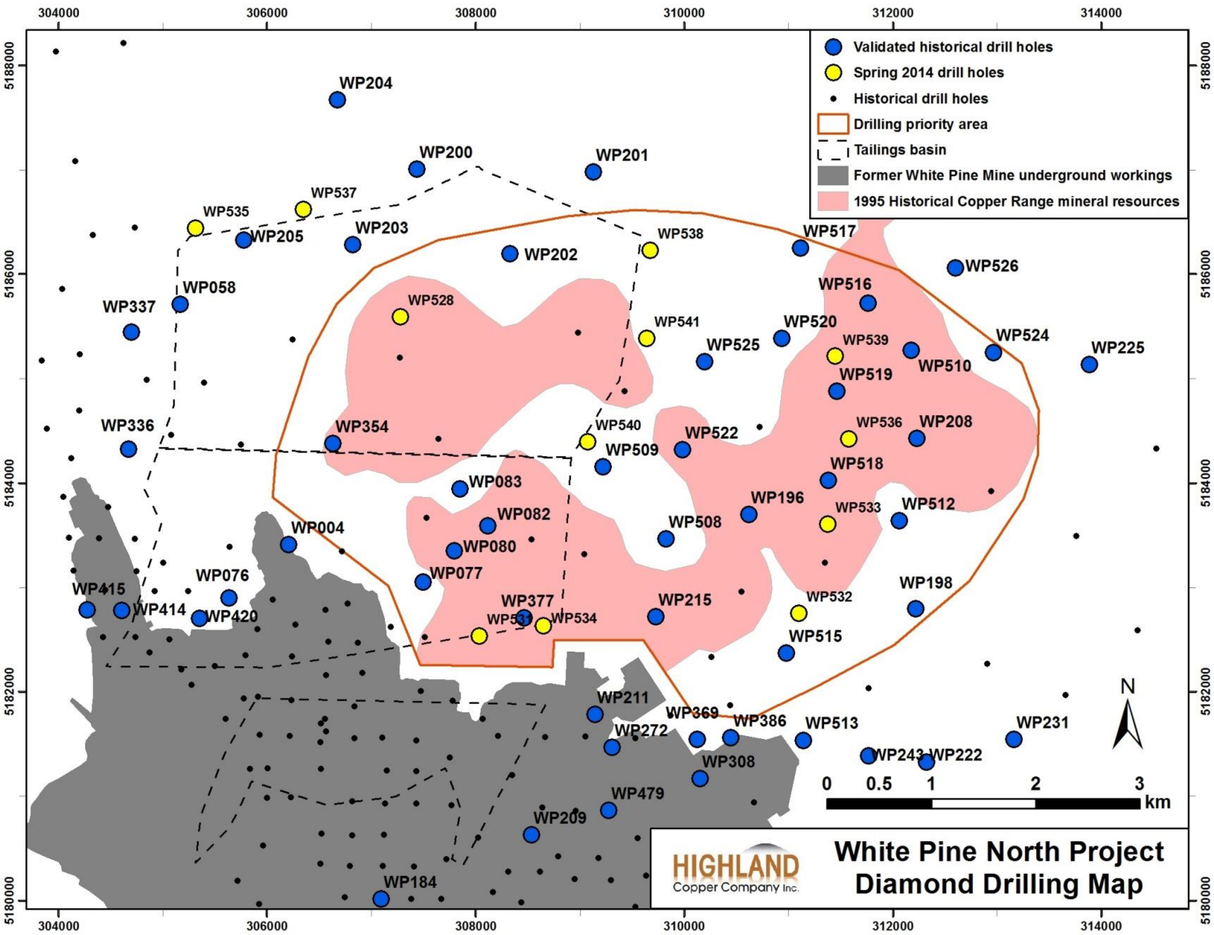
Most of the early drilling was BQ core. The glacial overburden was cased to bedrock. In the pre-1970s drilling, the core from above the ore zone was laid on the ground and logged. The core from the fringe or Top of Mineralization through the Lower Transition units was stored in five-foot-long spruce boxes and the core was transferred to the lab at the White Pine Mine. The core was logged and the beds in the mine stratigraphy were identified.

The 1994-1995 drilling program was conducted using NQ core. The glacial overburden was cased to bedrock. The core was logged in detail and the beds in the mine stratigraphy were identified.

Early diamond drill holes were abandoned without cementing. Later drill holes were cemented through the overburden. The drill holes from the 1994 - 1995 drilling program were cemented from the bottom of the hole to the surface.

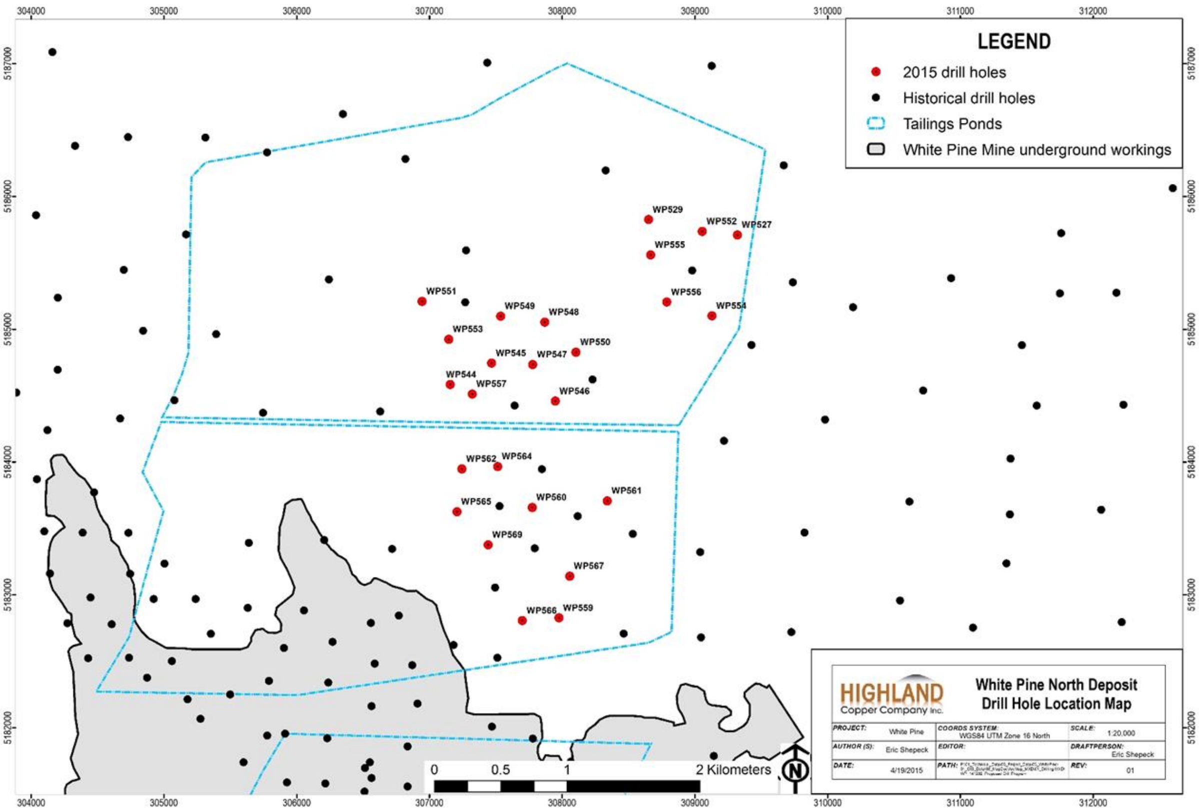
In January 2014, Highland initiated an analytical program to validate historical assay results from 51 diamond drill holes completed by CRC in the White Pine North deposit. Thirty-six of these holes were drilled between 1958 and 1980 with both BQ and AQ core, while the other 15 holes were drilled in 1994 and 1995 with NQ core. Highland's validation program used a ¼ cut of the original whole core from 883 historic sample intervals. This resampling duplicated the exact interval previously sampled and assayed in the historical programs. The remaining ¼ of the original core was retained as reference material. The validation analytical technique used both a screen metallic assay method and a 2.5 g digestion ICP assay method to determine total copper and results from both methods were in good agreement. The location of the validated historical drill holes is shown on Figure 6.1.

**Figure 6.1: Historical Diamond Drill Holes Location Map
Highland's Validation Sampling 2014 Drilling Program**



During January and February 2015, Highland completed 27 diamond drill holes totaling 19,152 m over an area of about 8 km^2 at White Pine North (Figure 6.2). The program used HQ core size and recoveries averaged over 99%. Highland designed this 2015 winter drilling program to (i) infill the historical drill grid to prepare an estimate of mineral resource and (ii) obtain information to guide mine planning. The program was successful and the results from this second phase infill drilling were consistent with results from Highland's 2014 drilling program and confirmed copper-silver mineralization from adjacent historical drill holes completed by CRC. Highland also completed seven wedges to obtain approximately 200 kg of mineralized samples for metallurgical testing.

Figure 6.2: Location Map of Diamond Drill Holes Bored by Highland 2015 Winter Drilling Program



Several reflection seismic surveys were conducted by CRC at the White Pine area, designed to investigate caved areas of the mine or to identify the location and faulted offsets along thrust and strike slip faults. One survey was carried out in 1975, two surveys were conducted in the fall of 1994 and another in the winter of 1995.

6.3 Historical Mineral Resources Estimate

Just prior to the mine closure, CRC extended exploration infill drilling to the north and northeast of the mine limits and in 1995 its chief geologist did a resource estimate (Johnson, 2014). The White Pine geologic model was built by defining the surfaces and thicknesses of individual beds within the ore interval based on 526 surface diamond drill holes. Isopachs were plotted for each individual bed within the ore interval and interrogated for geologic integrity and honoring of data. Following interrogation, copper and silver grades were composited (accumulated) over individual mining configuration intervals. Isogrades were plotted for bed and mining configuration intervals and interrogated for geologic integrity and honoring of data.

The cut-off copper grade was determined by considering the production costs from the Northeast Mine (a mining area of the White Pine Mine). In June 1995, Northeast Mine production cost of one-pound equivalent cathode was US $1.28 at an average grade of 19.2 pounds of copper per short ton. This compared favorably with studies indicating a future cost of US $1.30/lb. Hence, at a copper price of US $1.30, the break-even grade (and cut-off grade) was approximately 19 lb of copper per ton. This calculation assumed a mill recovery of 87.5% and a payable copper content in the concentrate of 96.5%.

Individual mining blocks were defined, limited either by the cut-off grade of 19 lb of copper per short ton (in situ), by adjacent blocks of different mining configuration or by the arbitrary north limit of the North Mine (latitude 50,000 N, White Pine Mine coordinates). The extraction rate used to calculate the historical estimate was 57%. This extraction rate provided a mine-wide estimate of extraction considering first pass, second pass, and ground left in pillars and barriers. The grades for each mining configuration were diluted based on past mining experience.

The official estimate at the time of closure was calculated for a minimum 2.9 m mining height (Table 6.1). [1]"Proven Reserves" were defined by CRC as those areas containing drill holes on a spacing of 305 m and meeting or exceeding the cut-off grade. This definition was validated by historical comparison of mill grade versus geology estimated grade. The geology estimated grade had predicted mill grade within 3% in the period January 1, 1990, to January 1, 1993. In 1993, CRC began milling "secondaries" (slag), and difficulties in estimating the grade of the slag and copper recovery from the slag introduced error into the reconciliation of mine grade with mill grade. "Probable Reserves" were defined by CRC as those areas which contained drill holes at a spacing between 305 and 914 m and met or exceeded the cut-off grade.

[1] This historical estimate does not use the categories set out in the CIM Definition Standards on Mineral Resources and Mineral Reserves and mandated by Canadian National Instrument 43-101("NI 43-101). The terms "proven and probable reserves" are historical terms used by CRC, not comparable to the CIM defined Probable Mineral Reserve and Proven Mineral Reserve and should be compared to a potential mineral deposit requiring further exploration drilling to define an initial resource. A qualified person ("QP") has not done enough work to classify this historical estimate as a current mineral resource and the historical estimate is not being treated as current mineral resources and should not be relied upon. The use in this section of the term 'reserves' does not mean to imply that the White Pine Project has reserves as defined in the current CIM Standards.

Table 6.1: White Pine Mine Historical Resource Estimate (Johnson, 1995)

Area	Class	Owner	Minable Tons	Mining Height (feet)	Dilution (percent)	Mining Grade (pounds/ton)	Contained Copper (pounds)
Central portion of the mine							
FC-17S	**proven**	**CRC**	**6,048,000**	**13.2**	**3.0**	**27.4**	**165,938,000**
Eastern portion of the mine							
MFC-1S	proven	CRC	6,202,000	9.5	3.0	19.3	119,885,000
FC-12S	proven	CRC	3,971,000	10.9	3.0	21.3	84,432,000
FC-13S	proven	CRC	994,000	12.9	3.0	21.0	20,864,000
FC-14S	proven	CRC	1,292,000	10.9	3.0	19.6	25,301,000
FC-15S	proven	CRC	3,741,000	9.5	3.0	21.1	78,924,000
FC-16S	proven	CRC	1,676,000	9.5	3.0	21.1	35,342,000
Subtotal			**17,876,000**			**20.4**	**364,748,000**
Northeast portion of the mine							
FC-8E	probable	CRC	1,925,000	13.9	3.0	19.9	38,388,000
FC-9E	probable	CRC	6,631,000	14.5	3.0	19.8	131,397,000
FC-10E	probable	CRC	214,000	13.6	3.0	19.9	4,261,000
FC-11E	probable	CRC	791,000	14.0	3.0	22.1	17,463,000
USH-2E	probable	CRC	1,412,000	9.5	7.0	18.6	26,208,000
USH-3E	probable	CRC	1,186,000	9.5	7.0	18.7	22,190,000
Subtotal			**12,159,000**			**19.7**	**239,907,000**
Northeast, East and Central Mines							
Total			**36,083,000**			**21.4**	**770,593,000**
North mine							
FC-1N	probable	CRC	11,476,000	17.5	3.0	20.7	237,431,000
FC-2N	probable	CRC	7,970,000	15.2	3.0	21.3	169,691,000
FC-3N	probable	CRC	10,122,000	14.0	3.0	19.6	198,607,000
FC-4N	probable	CRC	13,161,000	15.6	3.0	20.1	264,114,000
PSH-1N	probable	CRC	4,219,000	9.9	3.0	19.9	83,807,000
PSH-2N	probable	CRC	50,000	9.5	3.0	20.9	1,044,000
PSH-3N	probable	CRC	4,286,000	9.5	3.0	19.3	82,558,000
PSH-4N	probable	CRC	28,745,000	10.5	3.0	20.8	597,226,000
USH-1N	probable	CRC	2,566,000	9.5	7.0	19.5	50,097,000
Subtotal			**82,595,000**			**20.4**	**1,684,575,000**
Total: Proven and probable			118,678,000			20.7	2,455,168,000

6.4 Previous Economic Evaluation

Between 1993 and 1995, CRC conducted a feasibility study of both solution mining of exploited areas in the mine and the development of the White Pine North deposit.

7. GEOLOGICAL SETTING AND MINERALIZATION

7.1 Regional Geology

The White Pine copper deposit is located in the Western Upper Peninsula of Michigan (U.S.A) on the south side of Lake Superior (Figure 7.1) Regionally, White Pine lies on the south flank of the Midcontinent Rift System, a 2,500 kilometers ("km") long structure of Precambrian age. The Project is located east of the town of White Pine at the east end of the Iron River syncline (Figure 7.2). The Nonesuch Formation, the host of the mineralization, is part of a Keweenawan-aged (~ 1.1 Ga.) continental rift-fill sequence (Figure 7.3). At the base of these rocks are the Portage Lakes Volcanics, which are composed of olivine tholeiite lava flows. The basalt volcanic rocks are overlain by the Porcupine Volcanics, which are composed of intermediate to felsic volcanic rocks. The Porcupine Volcanics are in turn overlain by the Copper Harbor Conglomerate, an alluvial fan deposit. In the area of the White Pine Project the Copper Harbor Conglomerate is composed of red (oxidized) lithic sandstone with subordinate amounts of conglomeratic sandstone. Overlying the Copper Harbor Conglomerate is the Nonesuch Formation, composed of gray to black to reddish-brown thinly interbedded siltstone, mudstone, and minor shale and sandstone. The base of the Nonesuch Formation interfingers with the top of the Copper Harbor Conglomerate. Overlying the Nonesuch Formation is the Freda Formation, composed of red to reddish-brown fluvial sandstone. The Portage Lake Volcanics are renowned for their native copper lodes (Native Cu Belt) and chalcocite occurrences (Chalcocite Belt) and are bounded to the south-east by the major Keweenaw thrust fault. This thrust fault put in contact the younger post-rift Jacobsville Sandstone with the older syn-rift volcanic units (Porcupine and Portage Lake Volcanics).

Figure 7.1: Regional Geology – Upper Peninsula of Michigan
https://www.highlandcopper.com/white-pine-north-project.

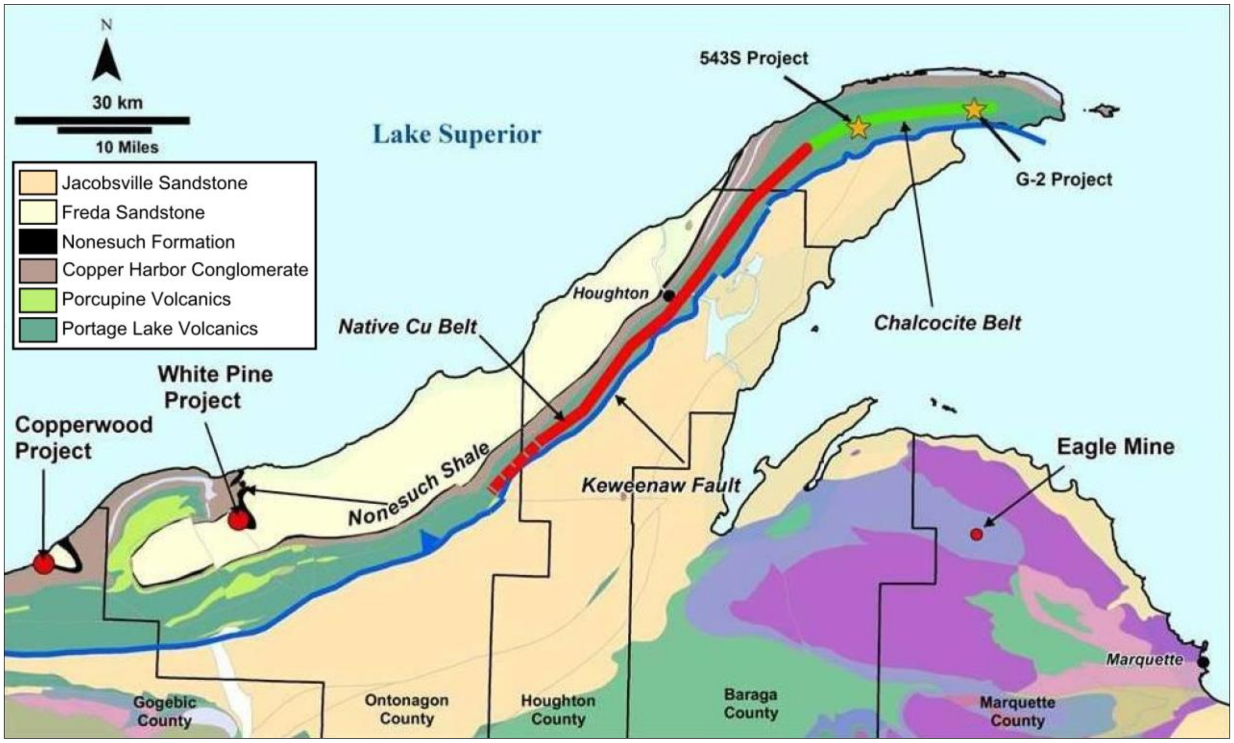


Figure 7.2: Geologic Map of the White Pine North Project Area
https://www.highlandcopper.com/white-pine-north-project.

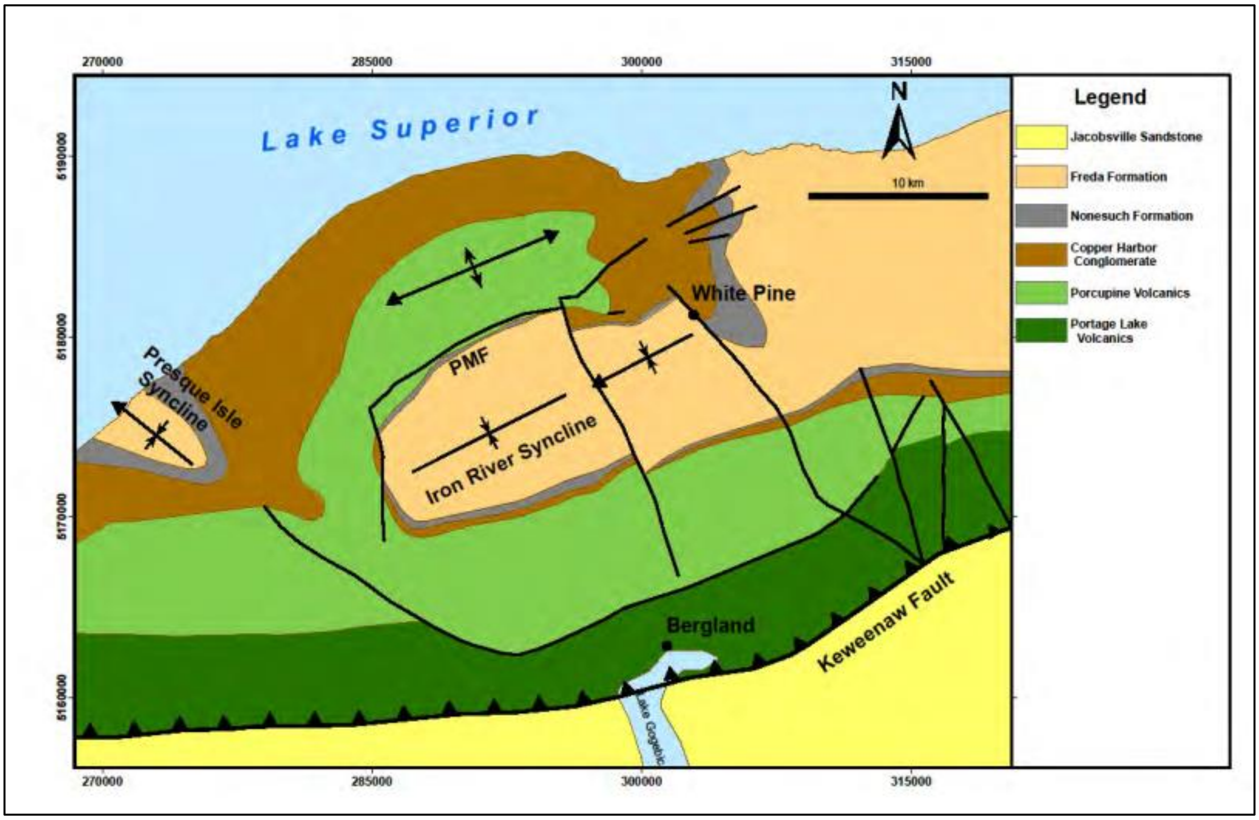
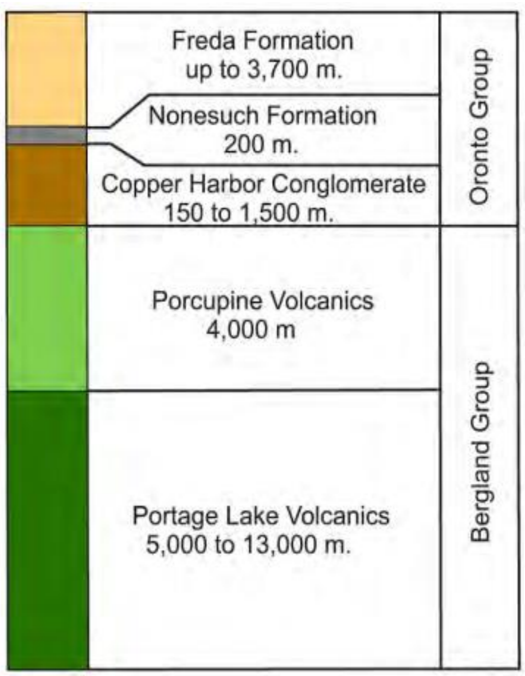


Figure 7.3: Generalized Stratigraphic Column in the White Pine Area. Redrawn from Daniels (1982) and Cannon and Nicholson (1992).



7.2 White Pine Stratigraphy

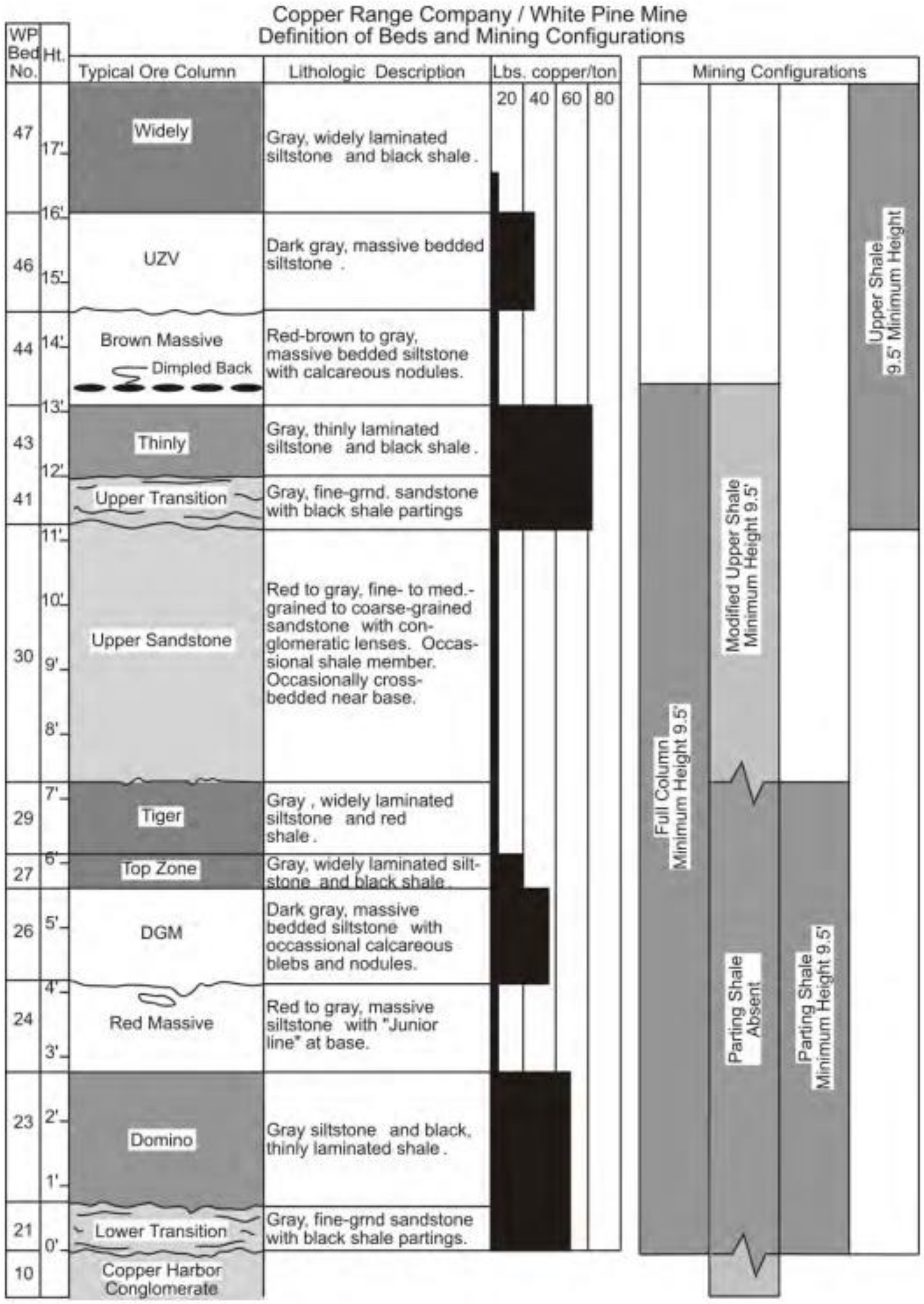
The copper mineralization in the area of the former White Pine Mine occurs in the bottom 6 m (20 ft) of the Nonesuch Formation at the contact with the Copper Harbor Conglomerate (Figure 7.3 and Figure 7.4). Beds within the lower 21 m ("m") (70 ft) of the Nonesuch Formation are laterally persistent and can be correlated across the mine. The shale and siltstone in the lower part of the Nonesuch Formation are divided into two shale units, the lower "Parting Shale" and the upper "Upper Shale", separated by the Upper Sandstone. Both shale units are present throughout the north part of the mine, but the Parting Shale pinches-out in the Southwest mine. Following are descriptions of the recognized beds that compose the top of the Copper Harbor Conglomerate and the lower portion of the Nonesuch Shale.

- **Copper Harbor Conglomerate (Lower Sandstone)** Brownish pink to medium gray, coarse- to very fine-grained, calcareous lithic sandstone. The top of the Copper Harbor Conglomerate ("CHC") formed the floor for most mining configurations at the historical White Pine Mine. The nomenclature used to distinguish between the CHC and the Lower Transition causes some confusion. The mine nomenclature defines the base of the Lower Transition and hence the base of the Nonesuch Shale at the first shale parting. However, in a regional context the Nonesuch Formation and, hence, the Lower Transition begin at the transition from oxidized to reduced sedimentary rocks. In the mine, particularly south of the White Pine fault, trapped hydrocarbons cause the sandstones to be locally reduced without containing shale partings. Strictly, the reduced sandstones are part of the Nonesuch Formation not the CHC or Lower Sandstone as used in mine parlance. Nonetheless, this terminology was useful for communicating with and guiding miners at the former mine.
- **Lower Transition [absent – 2.9 m (9.6 ft.), 0.5 m (1.6 ft.) avg.]** Interbedded red brown to gray, coarse- to very fine-grained, massive, planar-bedded to micro-trough cross bedded calcareous, hematitic, lithic sandstone and medium gray to gray black siltstone and shale partings.
- **Domino [absent – 1.1 m (3.6 ft.), 0.3 m (0.6 ft.) avg.]** Gray green to black, well laminated shale at the base grading upward to chloritic/micaceous, crudely laminated siltstone.
- **Junior Line [0.03 m (0.1 ft.) avg.]** Light gray, fine-grained limestone. Mud cracks are common in the Junior Line. The Junior Line is laterally persistent and can be identified as far away as Houghton. It was a useful marker within the mine and is used as a reference to help miners keep the floor at the correct height.
- **Red Massive [absent – 1.3 m (4.3 ft.), 0.4 m (1.4 ft.) avg.]** Gray brown to gray green, massive, well-indurated siltstone with faint shale partings throughout, hematitic at base becoming increasingly chloritic upward, commonly contains slumped/distorted bedding giving a swirled appearance.

- **Dark Gray Massive (DGM) [absent – 1.1 m (3.5 ft.), 0.3 m (1.0 ft.) avg.]** Dark gray green, massive, well-indurated massive siltstone, with calcareous nodules near the base, grading upward to crudely-laminated, chloritic/micaceous siltstone with faint shale partings.
- **Top Zone [absent – 2.1 m (6.8 ft.), 0.2 m (0.6 ft.) avg.]** Interbedded gray green, very fine-grained chloritic/micaceous sandstone and micaceous siltstone with gray black, truncated and distorted shale laminae containing load casts and flame structures.
- **Tiger [absent – 1.2 m (4.1 ft.), 0.2 m (0.8 ft.) avg.]** Green gray, very fine-grained micaceous sandstone grading upward to red brown, ferruginous sandy siltstone, siltstone and mudstone containing slumped beds, mud chip clasts, and load casts.
- **Upper Sandstone [absent – 3.8 m (12.5 ft.), 1.1 m (3.7 ft.) avg.]** Brown gray, coarse- to very fine-grained, moderate to well sorted, planar and cross-trough bedded, calcareous, lithic sandstone. Interbedded, gray green, sandy siltstone and siltstone beds.
- **Upper Transition [absent – 1.3 m (4.4 ft.), 0.2 m (0.5 ft.) avg.]** Interbedded green gray, medium-to very fine-grained calcareous, chloritic sandstone and gray black siltstone and shale partings.
- **Thinly [absent – 1.7 m (5.7 ft.), 0.4 m (1.4 ft.) avg.]** Gray black, thinly laminated shale and siltstone.
- **Brown Massive [0.2 – 2.5 m (0.5 – 8. ft.), 0.7 m (2.2 ft.) avg.]** Gray brown, massive appearing, well indurated, calcareous, chloritic and micaceous, very fine-grained sandstone, siltstone, and shale. Near the base of the Brown Massive are flattened elliptical to amoeboid calcareous nodules. The casts of these nodules form the “dimpled back” in the mining parlance.
- **Upper Zone of Values (UZV) [0.3 – 2 m (0.9 – 6.5 ft.), 0.8 m (2.6 ft.) avg.]** Laminated green black shale and dark gray green, calcareous siltstone.
- **Widely [0.03 – 4.0 m (0.1 – 16.2 ft.), 0.9 m (3.0 ft.) avg.]** Interbedded (widely laminated) gray black, chloritic/micaceous sandy siltstone and gray black shale. Very fine-grained pyrite throughout.
- **Red and Gray [0.03 – 4.5 m (0.1 – 14.8 ft.), 1.3 m (4.4 ft.) avg.]** Interbedded olive gray, planar bedded, chloritic/micaceous, sandy siltstone and siltstone with shale.
- **Tiebel Sandstone** Interbedded medium-gray to gray-green, medium- to very-fine grained, moderate- to well-sorted calcareous sandstone and chloritic-micaceous siltstone and shale. Massive to horizontally stratified and micro-trough cross-bedded sandstone and siltstone with mudstone drapes, shale partings, rip-up clasts, graded beds, fining-upward sequences, and soft-sediment deformation features.

- **Stripey** Lenticular to planar bedded, medium gray to gray-green, calcareous, very fine-grained sandstone and chloritic/micaceous siltstone and shale with mudstone drapes, partings, and load casts. Fining upward to sandy-siltstone and siltstone-shale couplets (< 1 cm. thick).
- **Marker** Crudely-laminated to well-laminated, light-gray calcareous siltstone and black to dark gray-green, pyritic shale (laminae < 1 mm. thick). Siltstone laminae are commonly truncated or discontinuous with numerous load features, giving the unit a blebby appearance.

Figure 7.4: Nomenclature for Mineralized Strata and Conventional Mining Configurations at the Former White Pine Mine. Modified from Ensign et.al. (1968).



7.3 White Pine Project Structural Geology

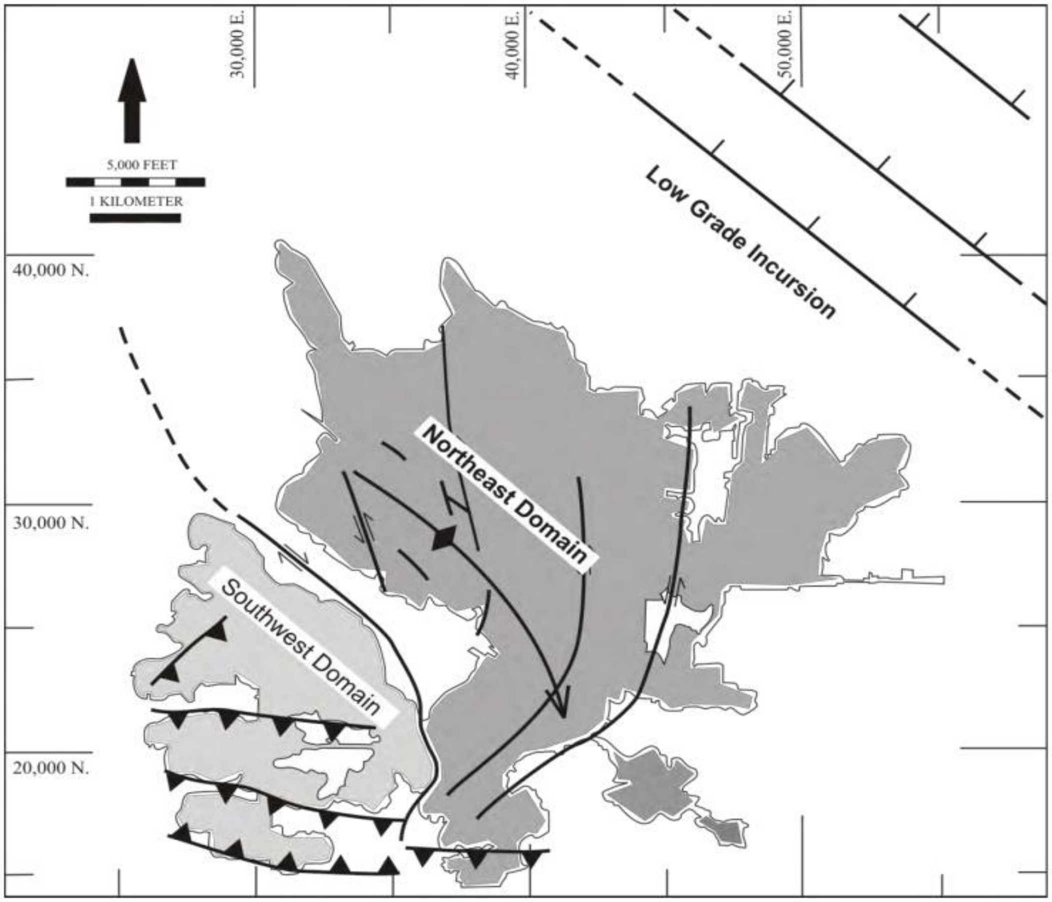
The White Pine Mine area is located at the east end of the Iron River syncline between the Keweenaw fault and the eastern extension of the Porcupine Mountain fault (Figure 7.2) -- two major north-dipping moderate to steep reverse faults. The major structural features of the area geology are the White Pine fault, thrust faults in the Southwest Mine, strike-slip faults in the North Mine, and a shallow east-southeast plunging anticline immediately north of the White Pine fault.

The Middle Proterozoic rocks of the Mid-Continent rift system (including those of the White Pine Mine area) have been subjected to at least two periods of deformation. An early period of extension contemporaneous with Keweenawan-aged rifting and a later period of compression associated with the development of the Keweenaw fault and Lake Superior syncline.

The rocks of the White Pine Mine area show the effects of the earliest period of deformation (extension) in soft-sediment deformation features, growth faults, and possibly the development of steep normal faults associated with listric faults. The later stage of deformation can be identified by folds and strike-slip and thrust faults.

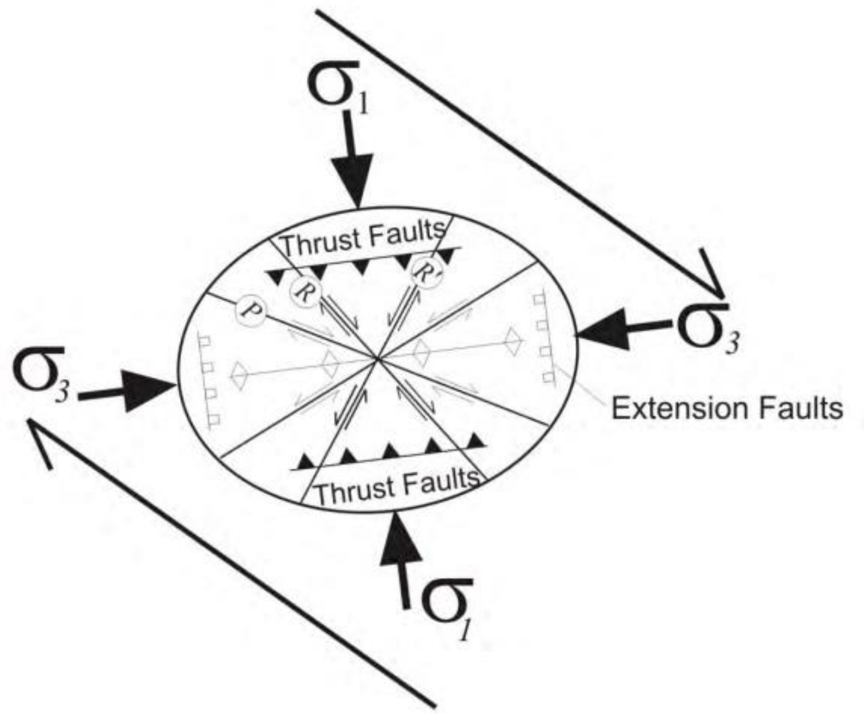
- **Domains:** The White Pine Mine area can be divided into two major structural domains, the Northeast and Southwest (Figure 7.5). The domains are separated by the White Pine fault, the major structural feature of the deposit. The Southwest Domain is distinguished from the Northeast Domain by the presence of north and south dipping thrust faults (Figure 7.5). The Northeast Domain contains few thrust faults but does contain strike-slip faults that can be followed for thousands of feet. Both domains contain abundant strike-slip faults.
- **Folds:** The White Pine Mine area contains a wide range of magnitudes of folds, from major folds associated with right lateral strike-slip faults to drag folds associated with thrust faults. The largest fold in the former mine is an asymmetric, open, shallow east-southeast plunging anticline immediately north of the White Pine fault, heretofore referred to as the White Pine anticline. The White Pine anticline forms the physical centerline of the former mine. The arcuate shape of the White Pine anticline is due to progressive simple shear (Figure 7.6). The fold formed and, as simple shear proceeded, the strike-slip motion along R faults produced the right-lateral deformation (southward bend) of the fold. Associated with the thrust faults in the Southwest Domain are drag faults and en-echelon plunging folds. Folds can also be identified above thrust tips.

Figure 7.5: Map Showing the Structural Domains and Major Structures of the White Pine North Project. Also Shown are the Interpreted Growth Faults and "Low Grade Incursion".



- **Wrench Faults:** Strike-slip and tension faults are found throughout the White Pine Mine area. The geometric relationship of strike-slip and tension faults is characteristic of wrench/strike-slip fault systems (Figure 7.6). In the Northeast Domain, left-lateral faults (R') develop the largest amount of horizontal displacement, e.g. the Pine Creek fault. The right-lateral (R) strike-slip White Pine fault separates the Northeast and Southwest Domains and forms a restraining bend within the mine. Tension faults are much less common throughout the mine area but host spectacular specimens of carbonate ± galena, sphalerite, nickel arsenides, native copper, and native silver. The Southwest Domain, particularly adjacent to the White Pine fault, is bisected by an anastomosing network of wrench faults. The shortening direction in wrench-fault systems is defined by the acute bisector of R and R'. In the Northeast Domain the shortening direction changes from a NNW-SSE direction in the northwest to near N-S in the northwest and rotates to a SSW-NNE direction to the south of the Northeast Domain. The dogleg in the White Pine fault forms a restraining bend and results in the formation of thrust faults and folds.

Figure 7.6: Strain Ellipse Showing the Geometric Relationship of Folds R, R' and Thrust Faults in the White Pine North Project.



- **Thrust Faults:** Thrust faults are found in the Southwest Domain and to a lesser degree in the Northwest Domain. The thrust faults in the Southwest Domain are both north and south dipping and strike from west-northwest to east-west and dip 30°. The strike of thrust faults form at right angles to the direction of shortening.
- **Growth Faults:** Growth faults are interpreted from rapid changes in thickness of beds. The White Pine fault is interpreted as a reactivated growth fault. A "low grade incursion," a northwest-southeast trending zone of low-grade mineralization, was identified in the North Mine during the 1994 - 1995 drilling program. To the northeast of the low grade incursion the Upper Sandstone thickens abruptly.
- **Joints and Fractures:** Most joints in the mine are classified as shear joints with fewer extension joints. Shear joints share a similar geometry to those of wrench faults with joints parallel to R' most abundant and those parallel to R second most abundant (Figure 7.6). Extension joints form normal to the principal-shortening axis. On close examination, the joints within the mine share the same geometric relationship as the wrench faults with the addition of joints developed orthogonal to the shortening direction.
- **Structural Discussion:** All brittle deformation features of the White Pine Mine are compatible with right-lateral simple shear (Figure 7.5) resulting from regional N-S directed shortening and are, therefore, synchronous. The regional N-S directed shortening was deflected about the

Porcupine Mountains and resulted in a zone of dextral transpression in the area of the White Pine deposit and sinistral transpression in the area of the Western Syncline.

7.4 Mineralization

Copper mineralization at the White Pine deposit occurs in two modes -- as very fine-grained sulfide (chalcocite) and as native copper. Sulfide mineralization is estimated to account for 85-90% of the copper in the deposit, but both modes of copper are intimately associated throughout the deposit. The copper mineralization at White Pine is unusually consistent. All drill holes within the deposit intercepted mineralized strata. Within the deposit, the grades of the mineralization are usually above cut-off grade over normal mining configurations. Most of the beds in the mineralized horizon are continuous over the entire deposit. The beds comprising the Parting Shale pinch out in the southwest part of the historic mine. The variation of the thickness of mineralized beds is also low from drill hole to drill hole.

Sulfide Mineralization: The dominant copper mineral in the White Pine deposit is chalcocite (Cu_2S). It occurs as fine-grained lamellae in laminites and partings in interbedded sandstone and shale, very-fine grained disseminations and discrete clots in siltstone, and in veinlets and veins. The top of the copper mineralization is identified as the Top of Mineralization ("TOM") Line or "fringe," a narrow transition zone between cupriferous and pyritic zones. The fringe is typically very narrow (a few inches) and is identified by the sequence: chalcocite, digenite, bornite, chalcopyrite, and pyrite. Immediately above the cupriferous zone is a narrow zone containing disseminated greenockite, galena, and wurtzite. The yellow color of greenockite is easily spotted in drill core. The TOM Line cross-cuts stratigraphy. In the shallow areas of the mine to the west near the portal, the TOM Line is typically 9.5 m (30 ft) above the Lower Sand while to the east the TOM Line descends through the otherwise normally mineralized beds.

Native Copper: Native copper mineralization occurs throughout the deposit. The most significant occurrences are as sheet copper and mineralized sandstone. Sheet copper forms along thrust surfaces in the southwest mine. The sheet copper in thrust surfaces is bedding parallel as well as cross-cutting stratigraphy. Sheets can reach spectacular size. It was observed that some sheets could be traced through entire pillars. Mineralized sandstone occurs in the uppermost part of the Copper Harbor Conglomerate and is invariably associated with trapped hydrocarbons. The greatest amounts of mineralized sandstone were found in areas adjacent to the White Pine fault.

Mixed Sulfide and Native Copper Mineralization: Native copper and chalcocite are found throughout the deposit. Native copper is found in close relationship to copper sulfide in sandy lenses and pods (load casts) in the Lower Transition. Native copper in the Lower Transition is more common in channels incised into the top of the CHC. Both chalcocite and native copper mineralization are ubiquitous features of the mineralization of the Dark Gray Massive bed as well; chalcocite occurs as very-fine grained disseminations; and native copper, as discrete blebs.

Structural Relationship: Structure imposes a significant control on the distribution and grade of mineralization. Higher-grade ore is spatially associated with the White Pine fault and thrust and strike-slip faults in the Southwest mine. Part of the increase in grade is due to the presence of mineralized sandstone and/or sheet copper. In addition, chalcocite mineralization is also enhanced as wider lamellae and cross-cutting veins and veinlets in the laminites.

Formation Water: The formation water encountered in the CHC is an alkaline brine (Table 7.1) with a chloride and TDS content approximately twice that of seawater. These compositions are thought to represent an approximate original composition of the depositional lake water and ore bearing fluid. Further support for alkaline brines existing during Nonesuch times is the abundance of carbonate throughout the CHC and Nonesuch Formation.

Hydrocarbons: The White Pine Mine is famous for its hydrocarbon seeps. In many areas near the White Pine fault, hydrocarbons seep out of the back, drip, and form puddles of "oil" on the floor. The most prolific seeps were noted in the northwest portion of the mine near and beneath the North Number One tailings dam.

7.5 Hydrology

Water flow into the historical White Pine Mine was through the rock formations, drill holes, caved areas of the mine, and along strike-slip faults. During the 1994 – 1995 drilling all the diamond drill holes flowed to surface and the water flowing from the casings was saline. Packer tests conducted on drill hole 508 confirmed that hydrostatic head was greater than the lithostatic head. Packer tests of underground drill holes across the southernmost thrust fault also indicated that hydrostatic head was greater than lithostatic head. Following closure of the mine, fresh water was pumped into the mine to slow down the rate at which saline formation waters (Table 7.1) would fill the mine. The surface of the water level in the mine is maintained lower than the level of water in Lake Superior by pumping.

Table 7.1: Chemical Analyses of Deep Mine Water from the Historical White Pine Mine*

	MS01[1]	MS02[1]	T4-2-1[2]	T4-2-2[2]	T4-2-3[2]
Field pH	6.6	6.7	NA	NA	NA
Lab pH	6.6	6.5	5.9	5.7	5.9
TDS (mg/l)	195,000	133,000	289,000	296,000	284,000
Density (g/ml)	NA	NA	1.1935	1.1921	1.1931
HCO3 (mg/l)	22	10	12	12	15
CO3 (mg/l)	0	0	0	0	0
Cl (mg/l)	132,000	96,000	170,000	145,000	160,000
SO4 (mg/l)	68	2	470	430	400
Br (mg/l)	1,500	1,080	1,820	1,790	2,120
F (mg/l)	0	0	NA	NA	NA
Ca (mg/l)	44,500	35,300	72,100	61,400	71,300
Mg (mg/l)	1,112	400	935	930	940
Na (mg/l)	12,100	9,700	15,500	18,600	15,800
K (mg/l)	204	<50	125	125	125
$NO_3 + NO_2$ (mg/l)	8.7	0.07 BL	NA	NA	NA
SiO_2 (mg/l)	4	<10	NA	NA	NA
Ag (mg/l)	<0.1	<0.5	NA	NA	NA
Al (mg/l)	0.9	<3	NA	NA	NA
As (mg/l)	0.03	0.02	NA	NA	NA
Ba (mg/l)	14.1	70	NA	NA	NA
Cd (mg/l)	<0.05	<0.3	NA	NA	NA
Cr (mg/l)	<0.1	<0.5	NA	NA	NA
Cu (mg/l)	<0.1	0.5	NA	NA	NA
Fe (mg/l)	0.2	<1	NA	NA	NA
Hg (mg/l)	<0.0002	<0.0002	NA	NA	NA
Mn (mg/l)	25.9	22.5	NA	NA	NA
Pb (mg/l)	<0.2	<1	NA	NA	NA
Se (mg/l)	<0.02	<0.02	NA	NA	NA
Sr (mg/l)	940	770	NA	NA	NA
Zn (mg/l)	<0.1	<0.5	NA	NA	NA

[1] Seeps

[2] Flow from underground diamond drill holes.

Note: Samples are from Seeps (MS) and Flow from Underground Diamond Drill Holes through the Southernmost Thrust Fault (T4) in the Mine (Johnson et.al., 1995).

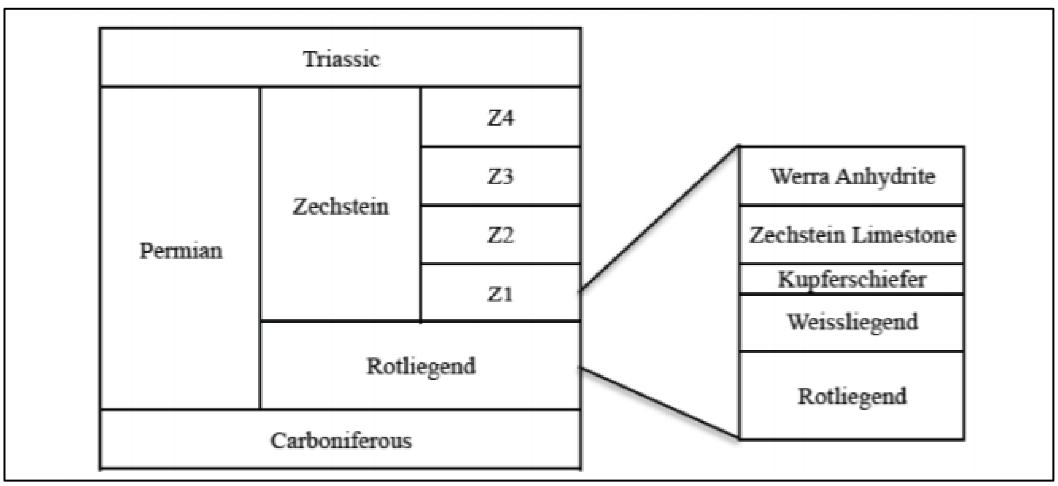
8. DEPOSIT TYPES

The mineralization of the White Pine North Project is classified as a reduced facies stratiform sediment-hosted copper deposit. Another deposit of this type is the Kuperschiefer deposit in Germany.

8.1 Kuperschiefer Copper-Silver Mineralization Model

The Kupferschiefer mineralization is a classic example of a sediment-hosted stratiform copper deposit. The shale is an Upper Permian black, organic-rich, fine-grained, and finely laminated (clayey) marl unit of marine origin. The Kupferschiefer is part of the Middle to Late Permian Zechstein group, which is composed of multiple depositional cycles, each beginning with a marine transgression and ending with the restriction of the basin (Ziegler, 1990). The Kupferschiefer is recognized as the basal unit of the first Zechstein cycle, at the transition between the underlying Weissliegend and Rotliegend sandstones and the overlying Zechstein limestone and Werra Anhydrite (Figure 8.1).

Figure 8.1: Stratigraphy of Permian Sediments in Germany. Modified from Asael (2009)

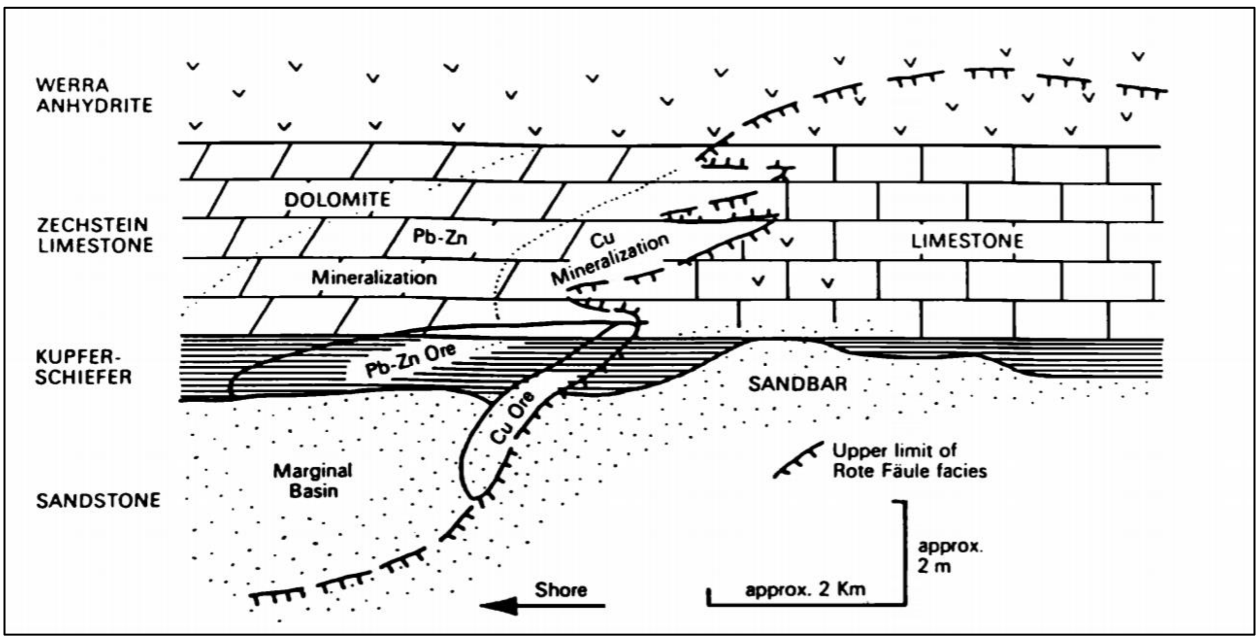


** Note. Z1-4 represents cycles within the Zechstein deposit.*

The Kupferschiefer copper-silver sulphide deposits occur across the contact between the Upper Permain Zechstein (Werra carbonates, dolomite, anhydrite, and saline rocks) restricted marine sequence and the Lower Permian Rotliegende (red sandstone) continental volcanic and clastic sequence. The Kupferschiefer (copper-bearing black shale) ore series can be split into two types of deposits, a reduced zone composed of dark-grey, organic rich and metal sulphide containing sediments and an oxidized zone of red-stained organic matter-depleted and iron oxide-bearing sediments, known as the Rote Fäule. The transition zone from oxidized to reduced rocks occurs both vertically and horizontally and is characterized by sparsely disseminated remnant copper sulphides within hematite-bearing sediments, replacements of copper sulphides by iron oxides and covellite, and oxide pseudomorphs after framboidal pyrite. These textural features and copper sulphide replacements after pyrite in the reduced sediments imply that the main oxide/sulphide mineralization postdated formation of an early diagenetic pyrite. The hematite rich sediments locally contain enrichments of gold and platinum group elements. The Kupferschiefer mineralization resulted from upward and laterally flowing fluids which oxidized originally pyritiferous organic matter-rich sediments to form hematitic areas (Rote Fäule) and which emplaced base and noble metals into reduced sediments .

The Rote Fäule distribution and the mineral zoning in relation to the Zechstein lithologies, as shown schematically in Figure 8.2, are useful as exploration guides to favourable areas for both Cu-Ag and new Au-Pt-Pd Kupferschiefer-type deposits.

Figure 8.2: Schematic Cross-section Showing the Position of Rote Fäule in Relation to Lithological Types and Mineralization (Vaughan et Al. - 1989)



Copper mineralization in the Kupferschiefer sensu stricto consists of chalcocite, digenite, covellite, bornite, and chalcopyrite associated with copper-arsenic ore consisting of tennantite and enargite. Galena and sphalerite commonly occur in the distal areas and only rarely in the Cu high-grade zones. Gold and silver occur as electrum in bornite and digenite . Pyrite occurs in the Kupferschiefer and in the low-grade, mineralized zone distant from the high-grade copper mineralization. Silver and gold occur disseminated in copper sulfides as native metals and as exsolution in the form of electrum. Hematite occurs in the Rotliegend sediments and as a primary mineral in the Rote Fäule zone. Other minerals in the Kupferschiefer include marcasite, clausthalite, barite, and rutile, and also kerogen, hematite, calcite, quartz, clay minerals, and the detrital relicts of titanite, zircon, and apatite.

8.2 White Pine North Mineralization Model

The Project chalcocite mineralization is usually attributed to the flow of copper rich brines through pyrite-bearing shale. The source of the copper in the brines is either attributed to the Copper Harbor Conglomerate red beds and the underlying mafic volcanic rocks or to a distant felsic intrusive body.

Shortly before the mine closing, geologic staff proposed the following model (Johnson et.al. 1995) which satisfies the observations made at the White Pine Mine:

- Accumulation of reduced laminites adjacent to oxidized permeable strata;
- Presence of oxidized brines;
- Source rock for copper (tholeiites and/or derived sediments);
- Regional burial;
- Flow out of the basin due to compaction;
- Elevation of pore fluid pressure due to regional shortening and reduction of permeability;
- Compression;
- Fault development;
- Flow towards faults;
- Hydrocarbons trapped in anticlines;
- Precipitation of copper sulfides as fluids react with pyritic strata;
- Precipitation of native copper as fluids react with trapped hydrocarbons.

Exploration for additional White Pine style mineralization would concentrate on areas that contain accumulations of reduced sedimentary rocks near major structures and are stratigraphically lower than the TOM (“Top of Mineralization”) line.

9. EXPLORATION

9.1 Historical Exploration (Pre-2014)

All exploration work completed on the White Pine North Project prior to 2014 were performed by the previous owner, Copper Range Company ("CRC"), who is now a wholly-owned subsidiary of First Quantum Minerals Ltd. CRC conducted a regional exploration program in the 1960s and 1970s called the "Trace Drilling Program" designed to identify White Pine style and scale mineralization from White Pine northeastward towards Houghton. This program consisted of drilling vertical holes on approximately one mile centers to depths of between 150 and 518 meters (500 and 1,700 ft) along the base of the Nonesuch Shale. No economic mineralization was intercepted during this drilling program and it was believed this indicated no White Pine scale deposits existed northeastward of White Pine.

A summary of historical exploration activities conducted on the Project is presented in Section 6**Error! Reference source not found.** of this Technical Report. The following sections focus primarily on the exploration programs implemented by Highland Copper Company Inc. ("Highland") between 2014 and 2015.

9.2 2014-2015 Exploration Program

During 2014 and 2015, Highland carried out a drilling program comprising of 42 HQ-diameter and an additional 18 wedges for a total of 30,462 m of core. The drilling provided 1,714 samples for copper and silver assaying and 635 kilograms ("kg") of mineralized samples taken for metallurgical testing. The 2014-2015 drill program was designed to upgrade the historical mineral resource area at the northern section of the deposit, in-fill the historical drill grid, obtain metallurgical samples and carry out geotechnical studies to refine the mining plan. Six holes were surveyed with televiewer technology for an improved understanding of the rock's in situ geotechnical characteristics.

9.2.1 Sampling Methods and Quality

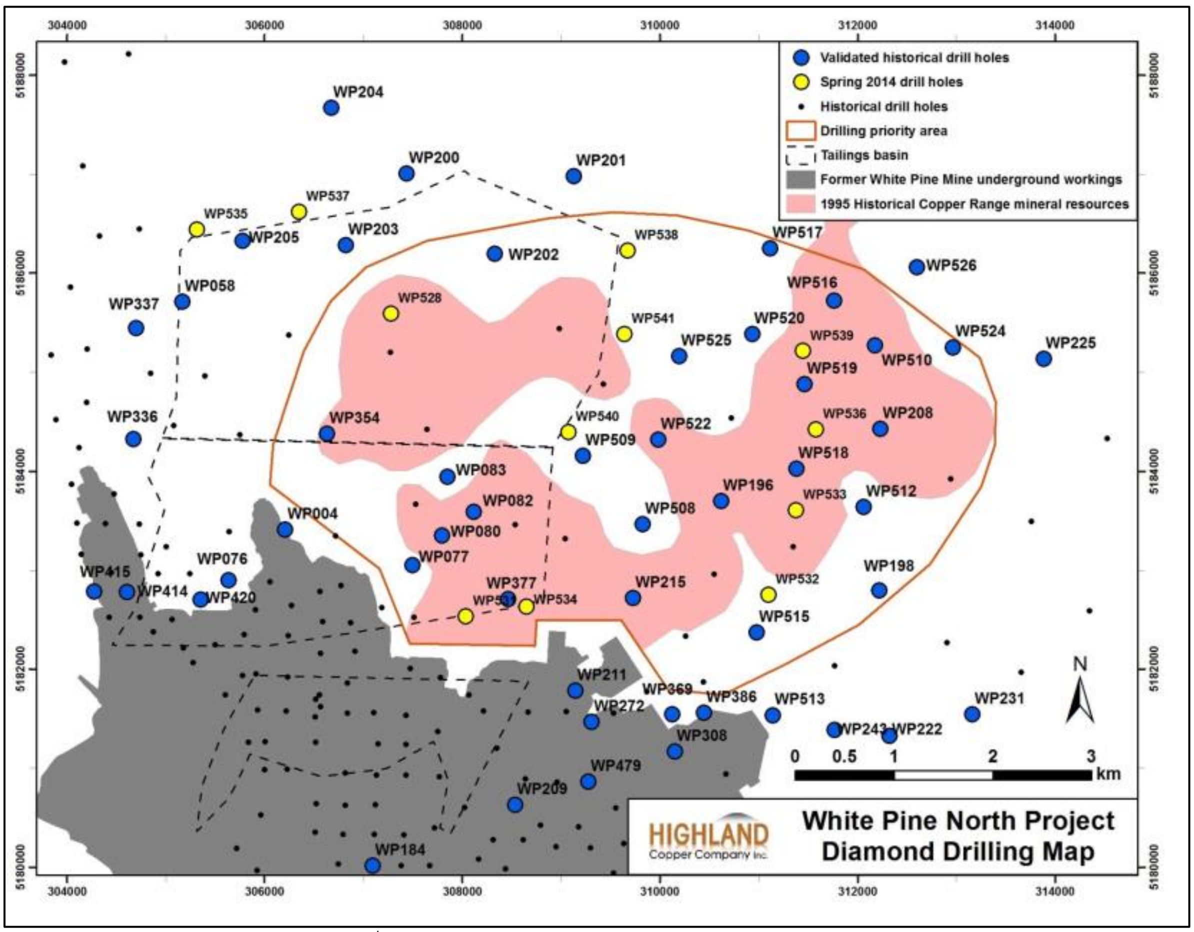
Activation Laboratories in Thunder Bay, Ontario, Canada (IOS 17025 accreditation), assayed all samples from Highland's exploration programs, using an ICP method tailored for the project samples, followed by a metallic procedure for samples containing at least 0.1% Cu. Highland applied industry standard QA/QC protocols to all steps of the drilling program.

9.2.1.1 White Pine North: Validation of Historical Drilling Assays

In January 2014, Highland initiated an analytical program to validate historical assay results from 51 diamond drill holes completed by CRC in the White Pine North deposit. Thirty-six of these holes were drilled between 1958 and 1980 with both BQ and AQ core, while the other 15 holes were drilled in 1994 and 1995 with NQ core.

Highland's validation program used a ¼ cut of the original whole core from 883 historic sample intervals. This resampling duplicated the exact interval previously sampled and assayed in the historical programs. The remaining ¼ of the original core was retained as reference material. The validation analytical technique used both a screen metallic assay method and 2.5 g digestion ICP assay method to determine total copper and results from both methods were in good agreement. The location of the validated historical drill holes is shown on Figure 9.1**Error! Reference source not found.**.

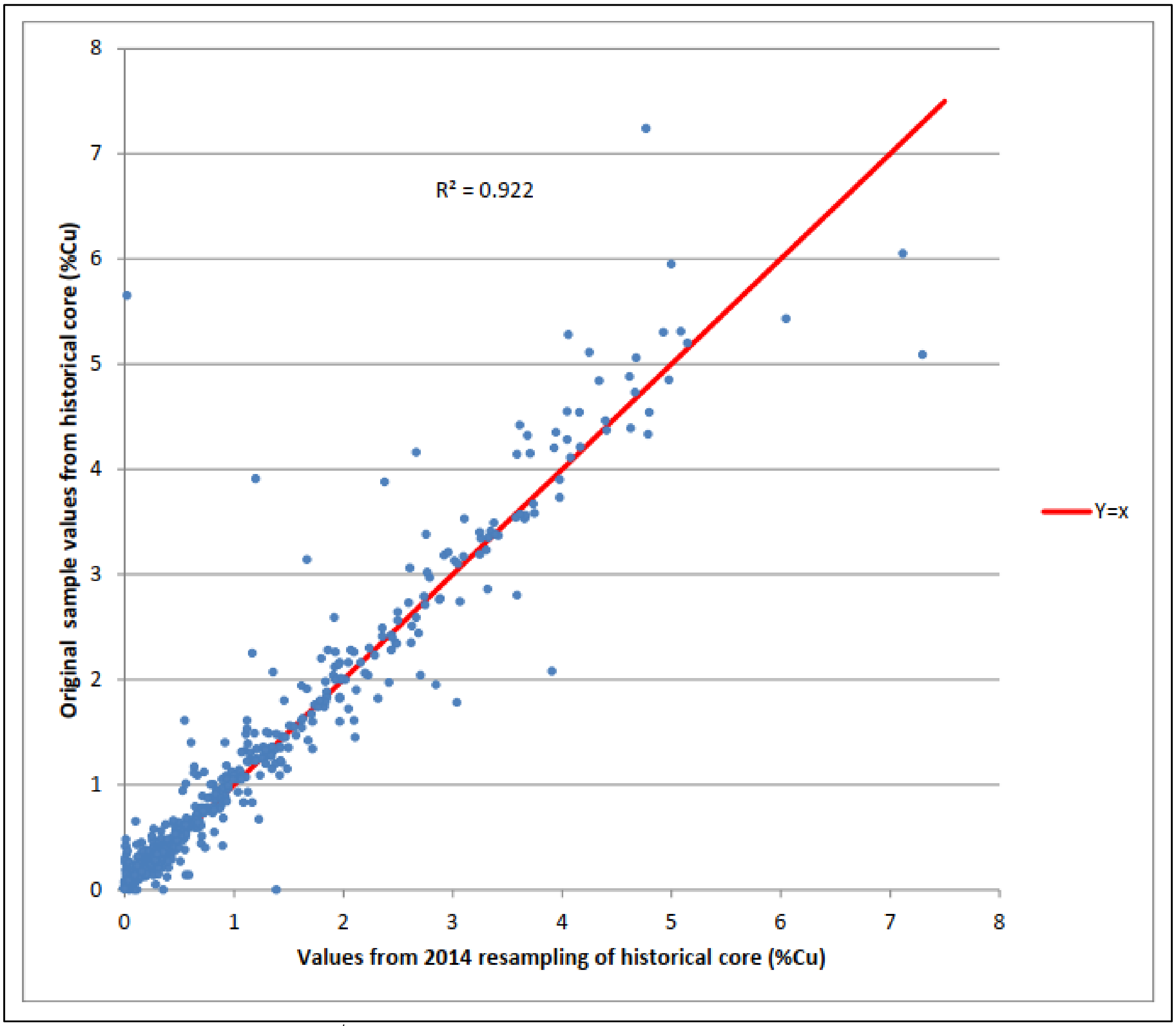
Figure 9.1: Location Map of Holes Completed During 2014 Drilling Program and Historical Diamond Drill Holes Used in Highland's Validation Sampling Program



From Highland announcement, July 3rd, 2014.

The results from this validation program are shown graphically on Figure 9.2. Highland considers the correlation between the historical and validation assays to be excellent, showing no bias between the two groups of assays. Highland planned to use the sample values from the original program for the current Mineral Resource estimate at White Pine North Project.

Figure 9.2: X-Y Plot Comparing Analytical Results from Historical and Validation Sampling of Historical Drill Core from the White Pine North Project (883 Samples)



From Highland announcement, July 3rd, 2014.

9.2.1.2 Quality Assurance / Quality Control ("QA/QC")

The Company maintains a rigorous QA/QC program with respect to the preparation, shipping, analysis and checking of all samples and data from the properties. Quality control for drill programs at the Company's projects covers the complete chain of custody of samples, including verification of drill hole locations, core handling procedures and analytical-related work, including duplicate sampling, check analyses at other laboratories and the insertion of standard and blank materials. The QA/QC program also includes data verification procedures. Activation Laboratories in Thunder Bay, Ontario, Canada (IOS 17025 accreditation) assayed all samples from the 2015 winter drilling program using an ICP method tailored for the project samples. This is discussed further in Section 11.

9.3 Airborne Geophysical Studies

There are no known surface geophysical exploration programs for the Project.

9.4 Geochemical Surveys

No surface geochemical exploration programs have been conducted on the Project since the mine was closed.

10. DRILLING

10.1 Drilling History

Before the White Pine Mine was closed in 1997, all drilling activities undertaken on the property were performed by previous owners. In 1907, Calumet and Hecla Mining Co. began an extensive drilling program that discovered locally high grades of native copper. The Copper Range Company ("CRC") conducted a continuous drilling program at the White Pine Mine from 1929 until the early 1970s. There was a hiatus in drilling until the commencement of a drilling program in 1994 – 1995. The 1994 – 1995 drilling program was conducted to provide a historical estimate supporting a feasibility study to build a new smelter at the White Pine Mine. Limited data are available from historical drilling (i.e. drill holes surveys, QA/QAC programs, sampling methods etc.) The historical drilling programs are discussed in Section 6.

10.2 2014-2015 Drilling Program

Highland Copper Company Inc. ("Highland") carried out two phases of drilling at the White Pine North Project ("The Project") in 2014 and 2015, with the aim of completing a current resource estimate for the Project as well as obtaining information for mine planning.

Between March and August 2014, Highland completed fourteen (14) diamond drill holes totaling 10,481 metres ("m") using HQ core size at the Project. Nine of the 14 holes were drilled vertically, and core recoveries averaged over 99%. Highland's objective for its 2014 winter drilling program was both to in-fill the historical drill grid and to expand the historical mineral resource area.

During January and March 2015 Highland completed an additional 28 diamond drill holes totaling 19,981 m over an area of about 8 square kilometres at White Pine North. The program used HQ core size and again recoveries averaged over 99%. Six holes were inclined to obtain structural data for geotechnical studies. Highland designed its 2015 winter drilling program primarily to infill the historical drill grid to prepare an estimate of mineral resource and obtain information to guide mine planning.

The results of the first and second phases of infill drilling are consistent with one another and the results from previous Copper Range Company ("CRC") drill programs and confirmed copper-silver mineralization from adjacent historical drill holes completed by the previous operator.

Highland also completed a total of 18 wedges in Phases 1 and 2 to obtain approximately 635 kilograms ("kg") of mineralized samples for metallurgical testing.

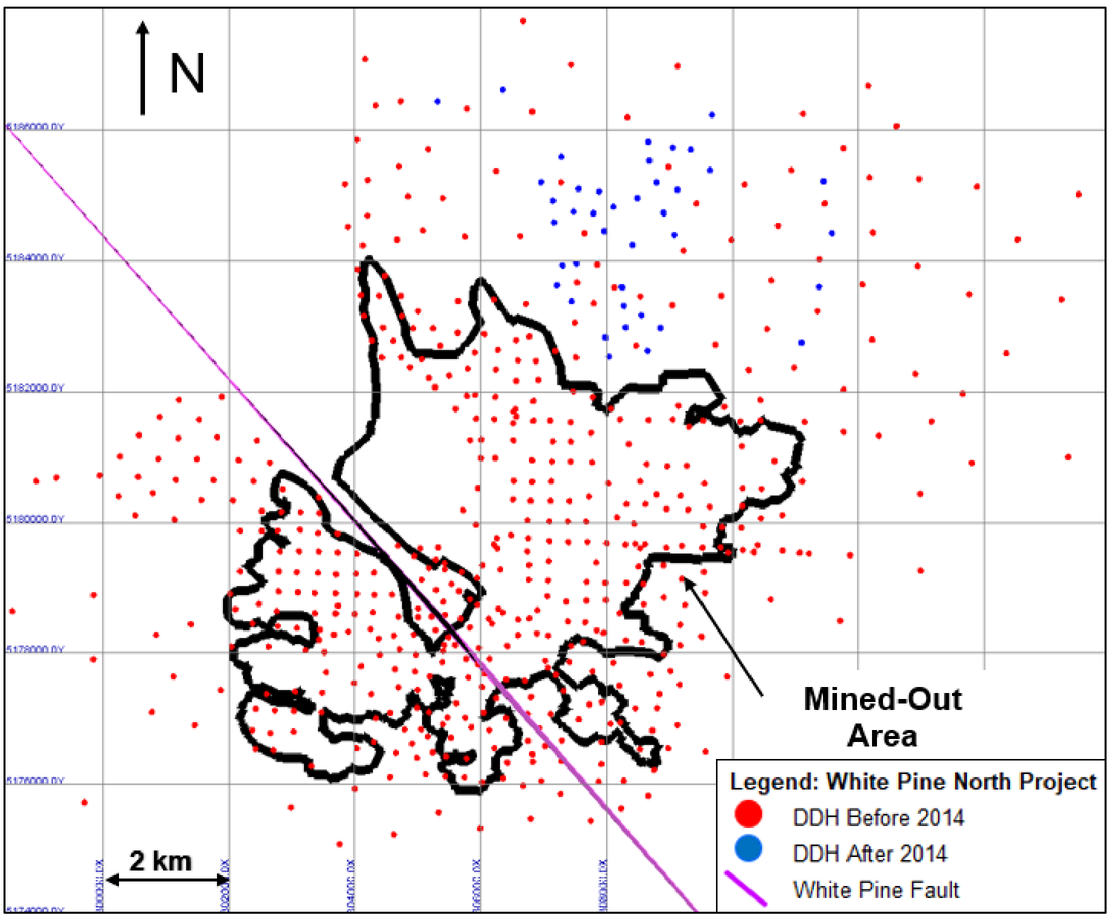
Table 26.1 summarizes the historical drilling program and completed drill holes by Highland.

Table 10.1: Drilling Programs by Company and Exploration Campaign

Company	Period	Core Size	Drill Hole Count	Length (m)	% of Total Drilling
Copper Range Company	1956 to 1998	AQ, BQ, NQ	526	244,453	89%
Highland Copper Company	2014-2015	HQ	42	30,462	11%
All Programs	**1956 to 2015**	**AQ, BQ, NQ & HQ**	**568**	**274,914**	**100%**

Figure 10.1 shows the location of the legacy drill holes. Historical collars are illustrated in red (1956-1998) while the holes drilled by Highland are shown in blue (2014-2015).

Figure 10.1: Plan View of the Historical and Highland Drilling Programs



11. SAMPLE PREPARATION, ANALYSES AND SECURITY

This section is based on information provided by Highland Copper Company Inc. ("Highland"). Sample preparation, analyses and security for the White Pine North Project (the "Project") prior to 2014 are described in the Section 6.

11.1 Sample Preparation

11.1.1 Drill Core Sampling (2014-2015)

For the diamond drill core samples, the sampling intervals were determined by Highland geologists depending on lithological contacts and the presence of mineralization. The sampled intervals are not be longer than 0.5 metre ("m") and they do not cross geologic contacts. In addition, the geologist inserts control standards ("Certified Reference Materials" or "CRMs"), blank material and drill core duplicates following the Quality Control ("QC") manual guidelines described in Section 11.3.

The sampling method implemented at White Pine North Project is straightforward. After drill core logging is completed, the core intervals to be assayed are identified in the core box. The core is sawn and the left ½ of the sample placed in a bag by the core cutter. The right ½ will be retained for reference and returned to the core box. Half of the sample tag is removed from the box and placed in the bag at the start of sampling an interval. When the sample is completed, the bag is sealed. All samples are assigned a unique sample number. The sample number does not include any reference to drill hole number or meterage for security reasons.

G Mining Services Inc. ("GMSI") validated the exploration methodology and sampling procedures used by Highland as part of an independent verification program. The Qualified Person ("QP") concluded that the drill core handling, logging and sampling protocols are at conventional industry standard and conform to generally accepted best practices. It is the opinion of GMSI that the samples quality is good and that the samples are representative.

11.1.2 Laboratory Sample Preparation

The mass of each sample was recorded prior to crushing. The entire sample up to 7 kilograms ("kg") was crushed to 80% passing 2 millimetres ("mm") with the jaw. A split of 250 grams ("g") sample was then pulverized to 95% passing 140 mesh (105 µm). All remaining pulps were saved and returned to Highland for storage.

11.1.3 Sample Analysis and Geochemistry

All Highland's drill core sample preparation (drying, crushing and pulverising) and assaying was handled exclusively by Activation Laboratories Ltd. ("Actlabs") in Thunder Bay, Ontario, Canada. Actlabs is an independent geochemical laboratory and implements a quality system compliant with the International Standard Organization ("ISO") 9001 Model for Quality Assurance and ISO/IEC 17025 General Requirements for the Competence of Testing and Calibration Laboratories. Actlabs is also accredited with CAN-P-1579 Requirements for the accreditation of mineral analysis testing laboratories.

The information below was taken entirely and/or summarised from the Actlabs Schedule of Services Brochure 2019 available on their website. http://www.actlabs.com/files/Actlabs_-_Schedule_of_Services_-_Canada_-_2019-07-22.pdf

All 2014-2015 drill core samples were analysed for Ag and Cu with 4-Acid ICP-OES (method code 8) and for 36 elements (Ag, Al, As, Ba, Be, Bi, Ca, Cd, Co, Cr, Cu, Fe, Ga, Hg, K, Li, Mg, Mn, Mo, Na, Ni, P, Pb, S, Sb, Sc, Sr, Te, Ti, Tl, U, V, W, Y, Zn, & Zr) including Ag and Cu with ICP Total Digestion (method code 1F2). The 4-Acid ICP-OES analysis is the higher ranked analysis for silver and copper and to be used for silver and copper. The lower detection limits for the 4-Acid ICP-OES are 0.001 % for copper and 3 g/t for silver.

11.2 Density

In-house bulk density was determined per mineralized horizon by measuring specific gravity by the water immersion method on whole core. Samples are dried in a drying oven at 60°C until they are completely free of moisture (4-16 hours). The scale is checked that it is on a level surface and that it is calibrated. The scale is then zeroed with the tray apparatus so that it will not have to be subtracted out later. The core sample is weighed for the dry mass. The water temperature at each measurement is recorded to more accurately determine water density.

Quarter (¼) core was sent to Actlabs for bulk density determination using the wax immersion method following the American Society for Testing and Materials ("ASTM") Designation C914-09. In-house samples were dried in a drying room at 60° C for 4 to 16 hours. Halved (½) core is weighed for a dry weight. A calibrated Radwag balance is used for these measurements. This certified scale has a 0.01 g accuracy and is calibrated and re-certified every year. Each sample is carefully wax coated with care to remove all trapped air from the wax. The cumulative weight of the waxed pieces is measured. The cumulative suspended weight is determined by placing all individually waxed pieces into a submerged wire basket. Specific gravity determined by the wax-immersion method has to be multiplied by the density of water to yield density. The temperature in the laboratory is 23°C, ± 1°C, which results in a water density of 0.997 g/cc.

Although Highland has performed Density measurements In-Situ, GMSI decided to apply a homogeneous 2.70 g/cc density value for all rock types in the White Pine North block model. Table 11.1 summarizes the values of densities used in the Mineral Resource estimation ("MRE") by GMSI.

Table 11.1: Specific Gravity Averages Used in the Resource Estimation

Lithology	Specific Gravity (g/cc)
Air	0.00
Overburden	2.20
Parting Shale (PS)	2.70
Full Column (FC)	2.70
Upper Shale (US)	2.70
Waste	2.70

11.3 Quality Control ("QC")

In addition to the Actlabs internal Quality Control ("QC") protocol, Highland implemented a rigorous QA/QC program for its drill core sampling completed in 2014-2015. As part of the QA/QC procedure, Highland inserted blank materials, control standards ("Certified Reference materials", or "CRMs"), core sampling stage duplicates and preparation stage duplicates.

Highland QA/QC samples included in the 2014-2015 drilling program are outlined in Table 11.2.

Table 11.2 : List of QA/QC Samples– 2014 & 2015 Drilling Campaigns for Cu % and Ag g/t

QAQC Sample Type	No of Samples	% of Sampling	Frequency of Insertion
Certified Coarse Blank - BL-10	17	6%	Approx. 1/20
Certified Coarse Blank - OPTA	72	27%	
Certified Coarse Blank – WPB-HC	5	2%	
Certified Blank Material Total[1]	**94**	**6%**	
CRM - OREAS 162	28	11%	Approx. 1/20
CRM - OREAS 95	27	10%	
CRM - OREAS 97	27	10%	
CRM – CDN-ME-1205	18	7%	
CRM – CDN-ME-13	44	17%	
CRM – CDN-ME-19	28	11%	
Certified Reference Material Total[2]	**171**	**10%**	
Sampling Stage Core Duplicate	15	0.9%	Approx. 1/20
Crushing stage duplicate	120	7%	4/100

** Note. the following CRMs were excluded from this list since they have less than three samples inserted in the QA/QC program (CDN-CM-17; CDN-ME-11 and OREAS 98).*

A geologist regularly inserted two CRM's, three coarse blanks, and one core duplicate for each drill hole. CRMs with a high-grade, medium-grade, and low-grade values (% Cu) were inserted in high, medium and low mineralized intervals respectively. Coarse blanks were inserted between high-grade intervals. A quarter (¼) core from the same assay interval was taken for a coarse duplicate.

For 2014-2015 drilling campaign, a total of 171 standards, 94 blanks, 15 drill core duplicates and 120 preparation duplicates were submitted to the laboratory for quality assurance purposes, which together comprise 24% of all drill core samples assayed (1,701) during that period.

11.3.1 Blanks and Assessment of Contamination

11.3.2 Copper and Silver

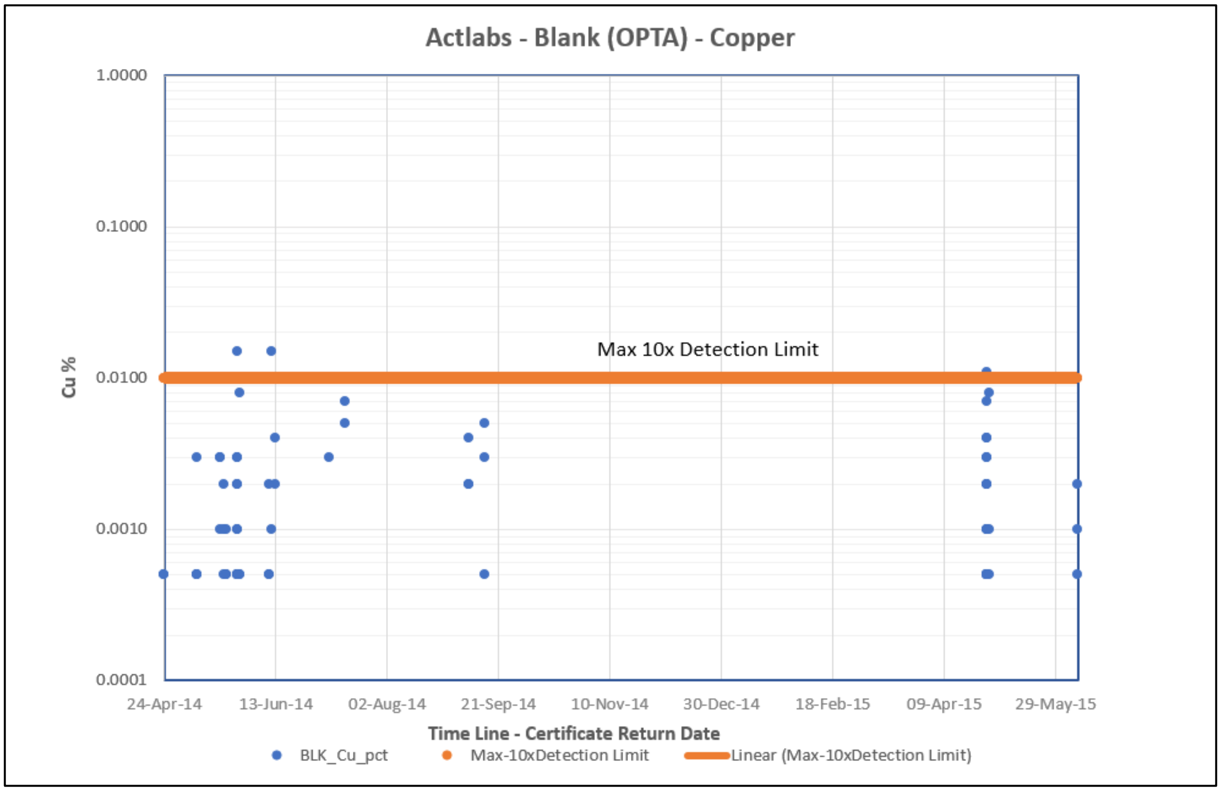
To monitor contamination during sample preparation and assaying, Highland inserted three types of blank materials ("OPTA", "BLK-10" and "WPB-HC"). In total of 94 blanks were analyzed as part of the sample stream between 2014-2015 (Table 11.1). From the 94 blanks, 96% of them returned less than 0.01% Cu which is 10 times the detection limit (10 x DL) of the 4-Acid Total Digestion ICP analytical method. From the three blanks with analytical value greater than 0.01% Cu (100 ppm), no analytical values were greater than 1% Cu (10,000 ppm). 100% of the coarse blank silver assay values were under the detection limit 3 g/t Ag. Descriptive statistics of coarse blanks demonstrates also no contamination for copper and silver (Table 11.3).

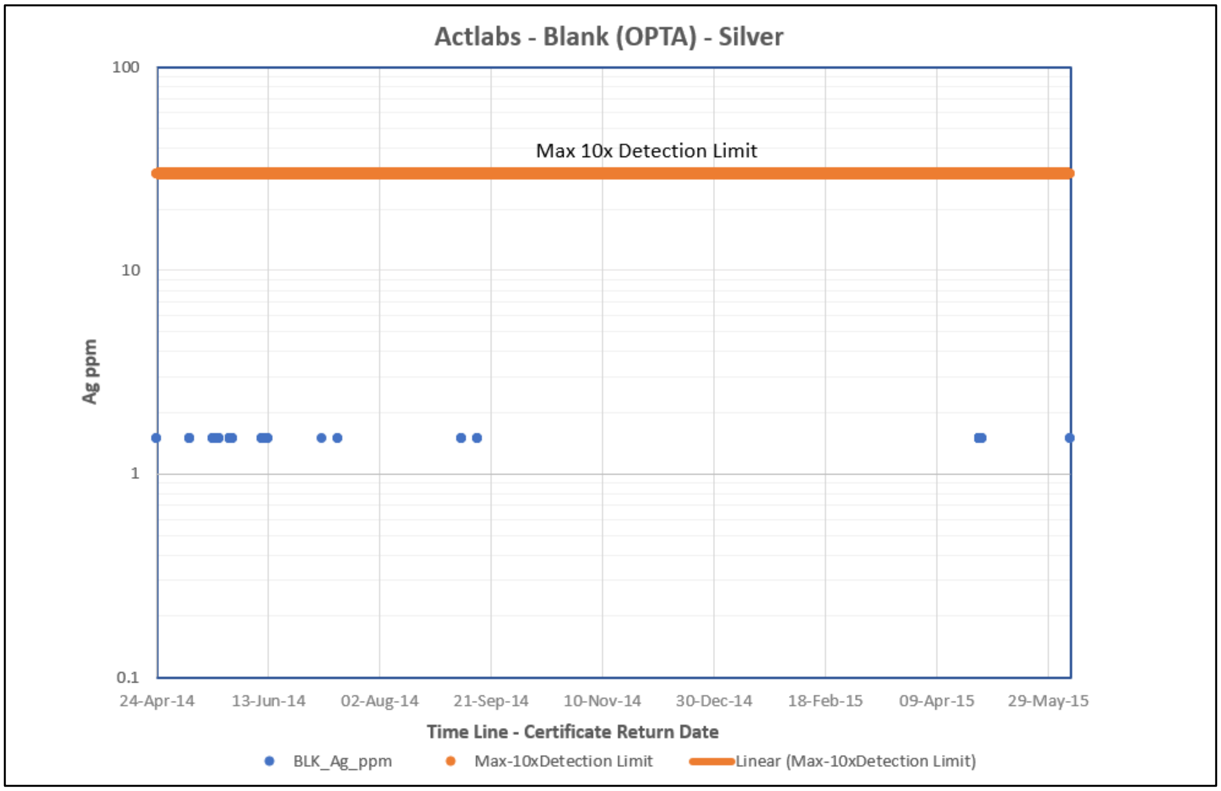
Table 11.3: Descriptive Statistic of Blank Material Assaying Results for Copper (% Cu)

HCC Blank Material (OPTA)	Cu %	Ag g/t
Mean	0.0025	1.5
Standard Error	0.0004	0.0
Median	0.001	1.5
Mode	0.0005	1.5
Standard Deviation	0.003	0.0
Sample Variance	9.9E-06	0.0
Minimum	0.0005	1.5
Maximum	0.015	1.5
Count	72	72
Confidence Level (95.0%)	0.000739	0.0

The Figure 11.1 shows the analytical results for copper and silver observed by the OPTA certified blank material over time. The majority of copper blanks (96%) is falling below the 10 x DL threshold. All silver blanks values are below the detection limit of 3 ppm.

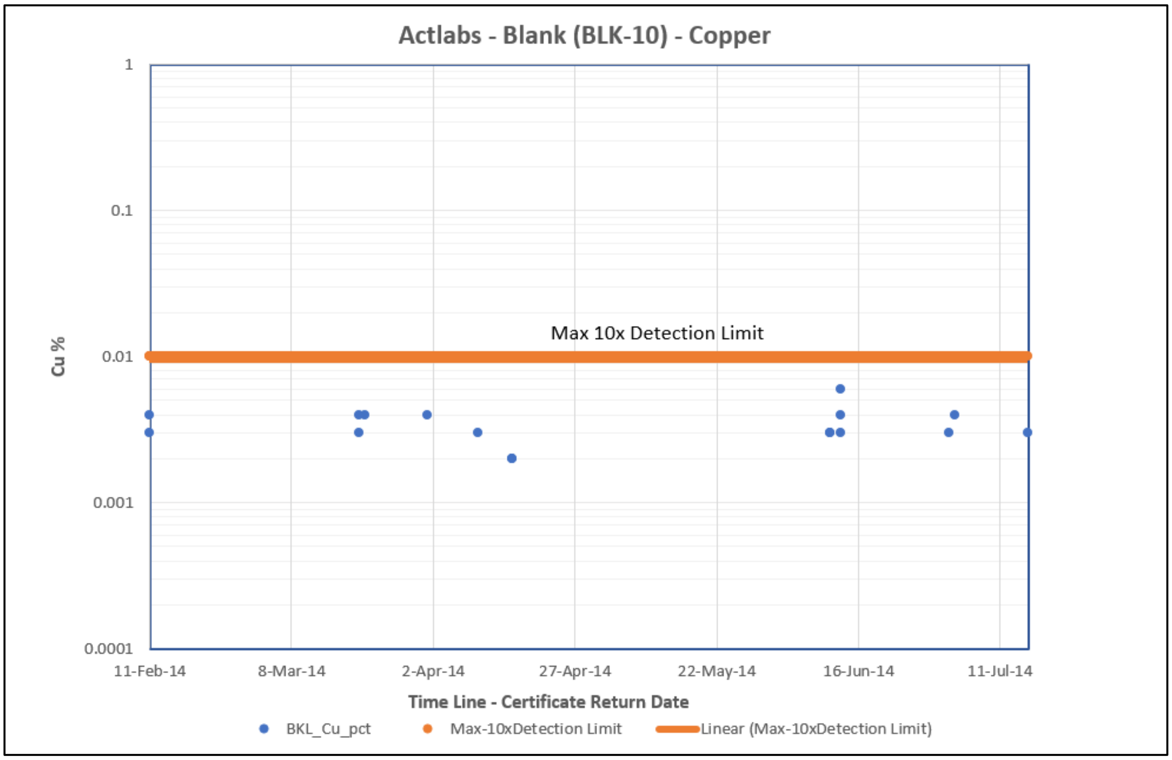
Figure 11.1: Blank Material (OPTA) Time Plots with Analytical Results for Copper (top) and Silver (bottom)





The Figure 11.2 and Figure 11.3 show the analytical results for copper and silver observed by the BLK-10 certified blank material over time. In both charts, 100% of copper and silver blanks submitted were under the acceptable limits, and it is assumed that no significant contamination occurred during the sample preparation, delivery, and laboratory analysis.

Figure 11.2: Blank Material (BLK-10) Time Plots with Analytical Results for Copper (top) and Silver (bottom)



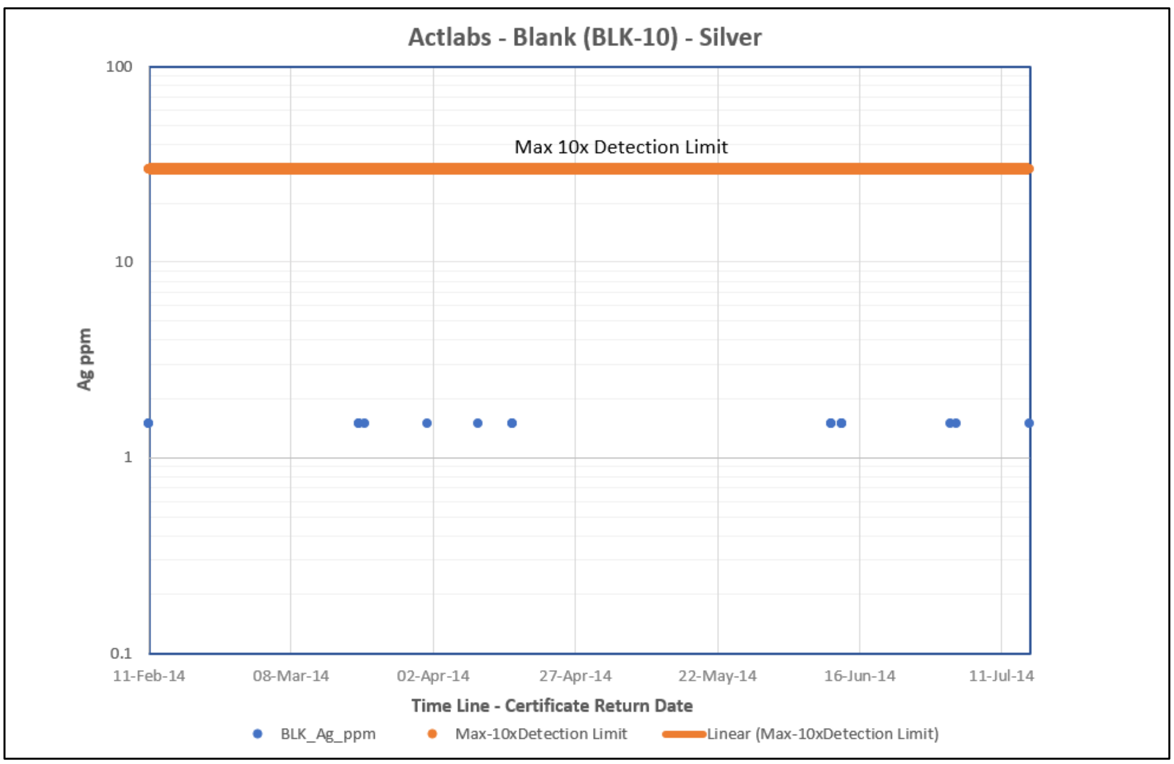
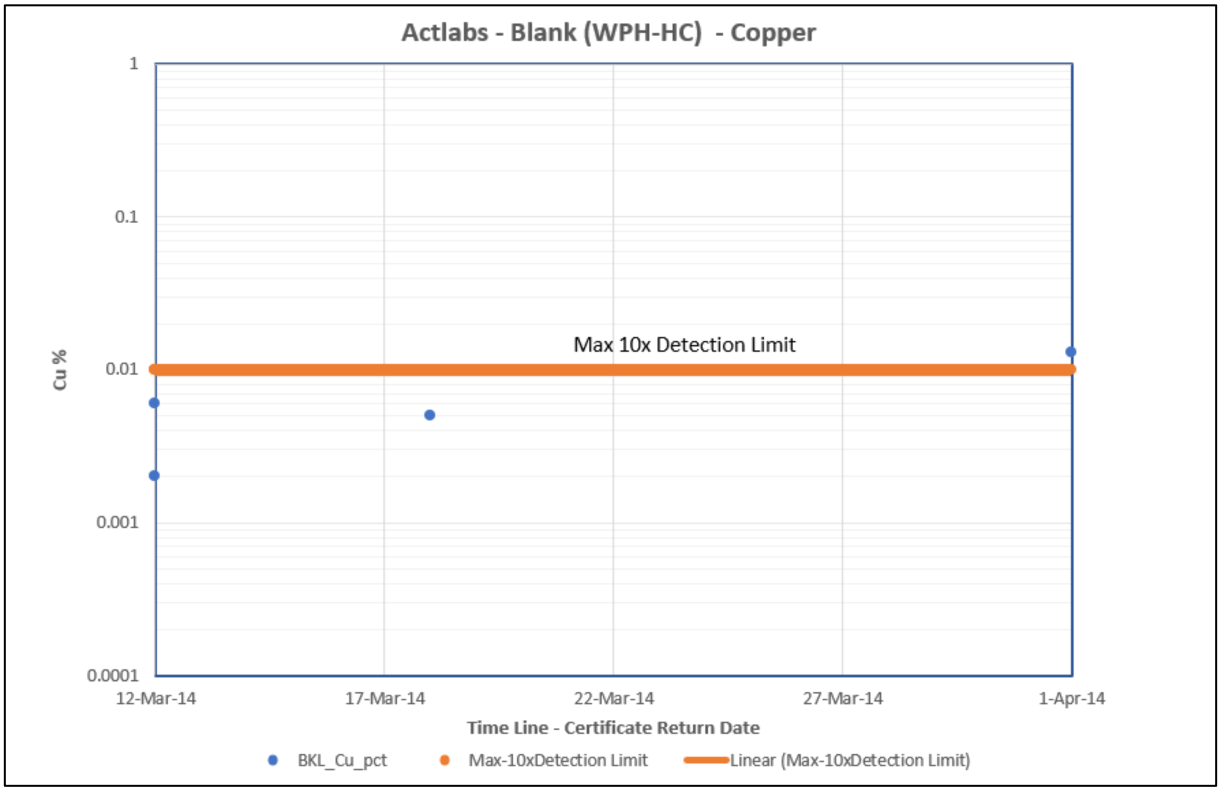
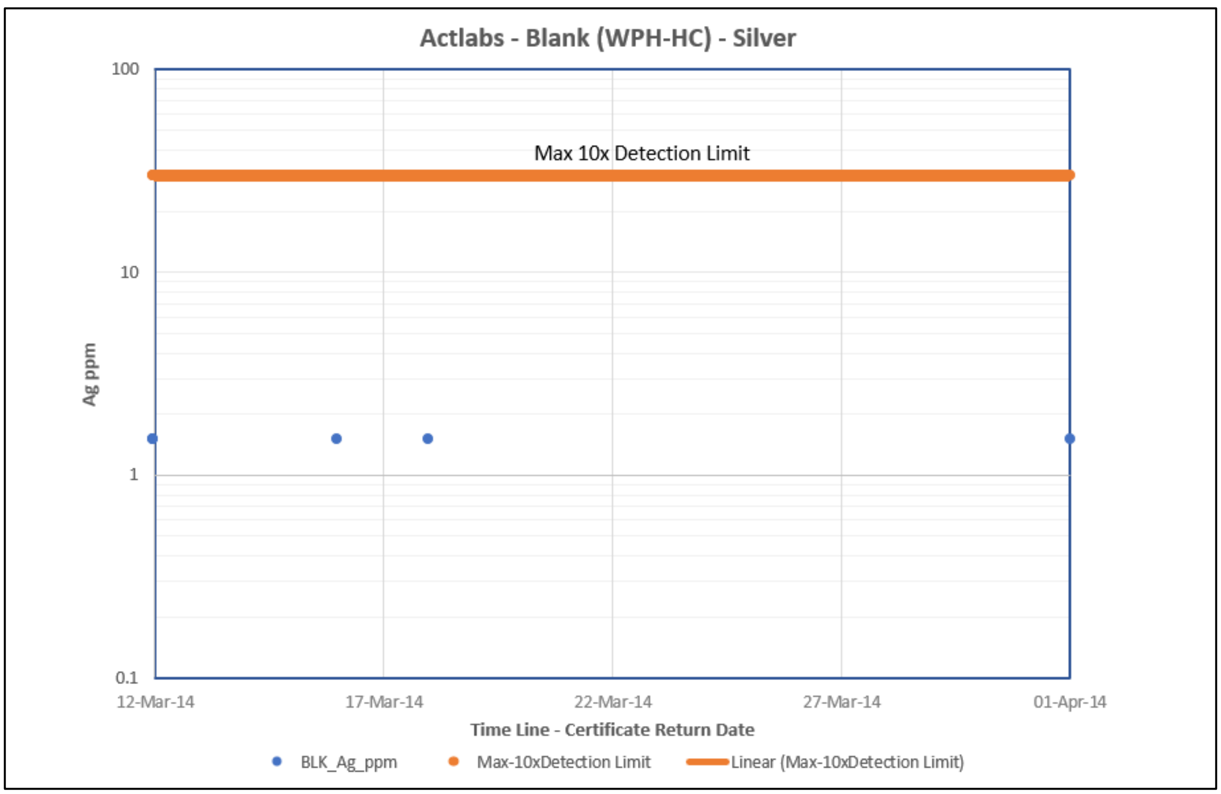


Figure 11.3: Blank Material (WPH-HC) Time Plots with Analytical Results for Copper (top) and Silver (bottom)





11.3.3 Duplicate Sample Performance

The duplicate samples included in 2014-2015 drilling program consist of sampling stage core duplicates and crushing stage duplicates. The drill core duplicates were sampled and inserted by the geologists on site. The crushing stage duplicates were collected in the preparation laboratory after jaw crushing. Core duplicates were inserted at a 0.9% rate and crush duplicates at a 7.0% rate.

The core duplicates performance is considered to be acceptable reflecting good overall precision and negligible sampling and analytical error for drill core samples.

Three copper core duplicates out of 15 core duplicates have a mean pair relative difference greater than 20% and possibly highlight variability characteristics of the ore deposit (Figure 11.4). Three silver core duplicates also have a mean pair relative difference greater than 20% and one of the silver duplicates coincident with one of the three deviating copper core duplicates (Figure 11.4).

The crush duplicates performance is considered to be acceptable reflecting good overall laboratory precision and negligible preparation and analytical error. 93% copper crush duplicates (111 duplicate samples of 120 in total) have a mean pair relative difference less than 20% while one silver crush duplicate is marginally over 20%. Again, all the majority of crush duplicate silver values for the original sample compares to the duplicate sample are between the acceptable limits ± 20% (Figure 11.5).

According to the statistical analysis, *t-Test: Paired Two Samples for Mean*, done on the original and duplicate samples values, no bias was detected by GMSI during the QA/QC validation. The average copper grade from the selected original values was 1% lower from the duplicate sample values. The average silver grade from the selected original values was 1% lower from the duplicate sample values. The T-Test statistical results for copper and silver preparation duplicates versus original samples are tabulated in Table 11.4 and Table 11.5.

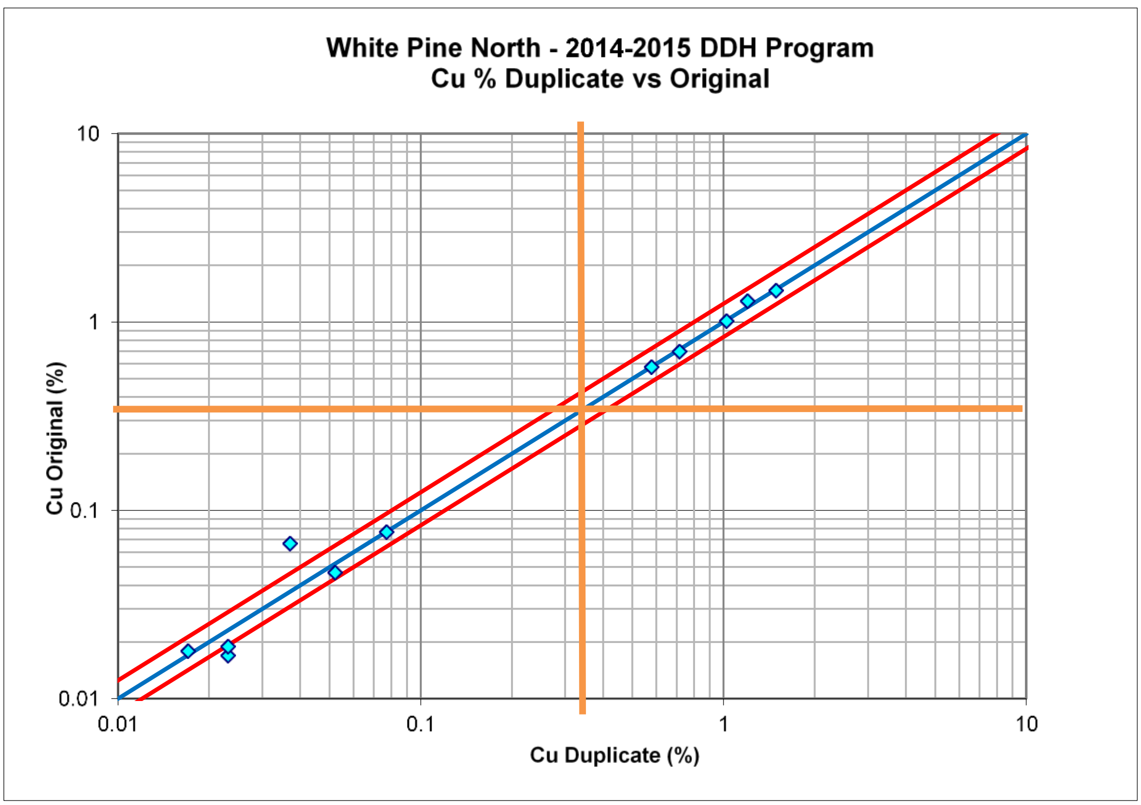
Table 11.4: T-Test: Paired Two Sample for Copper Original vs. Duplicate Sample Means

Statistical Analysis "T-Test"	Cu % Original Sample	Cu % Duplicate Sample
Mean	0.724	0.719
Variance	1.51	1.48
Observations	120	120
Pearson Correlation	0.9997	
t Critical two-tail	1.98	

Table 11.5: T-Test: Paired Two Sample for Silver Original vs. Duplicate Sample Means

Statistical Analysis "T-Test"	Ag g/t Original Sample	Ag g/t Duplicate Sample
Mean	11.37	11.21
Variance	1963.37	1880.44
Observations	120	120
Pearson Correlation	0.9994	
t Critical two-tail	1.98	

Figure 11.4: Drill Core Duplicate Performance for Cu (top) and Ag (bottom)



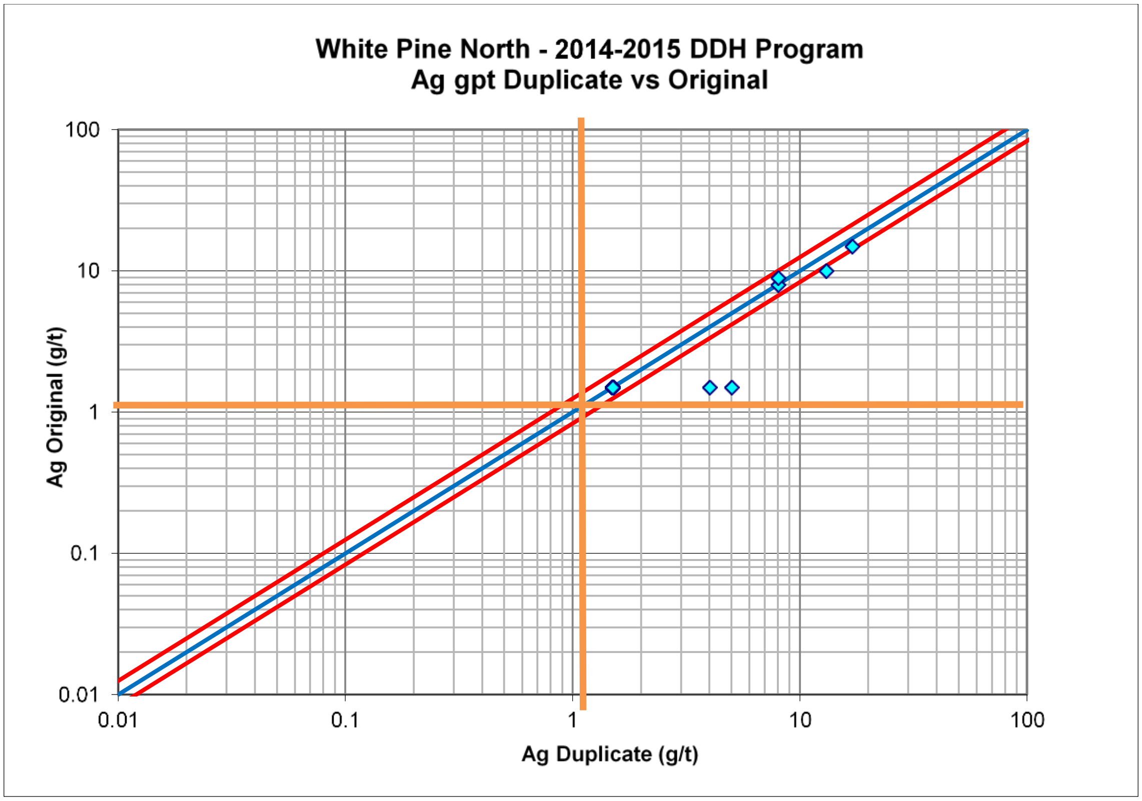
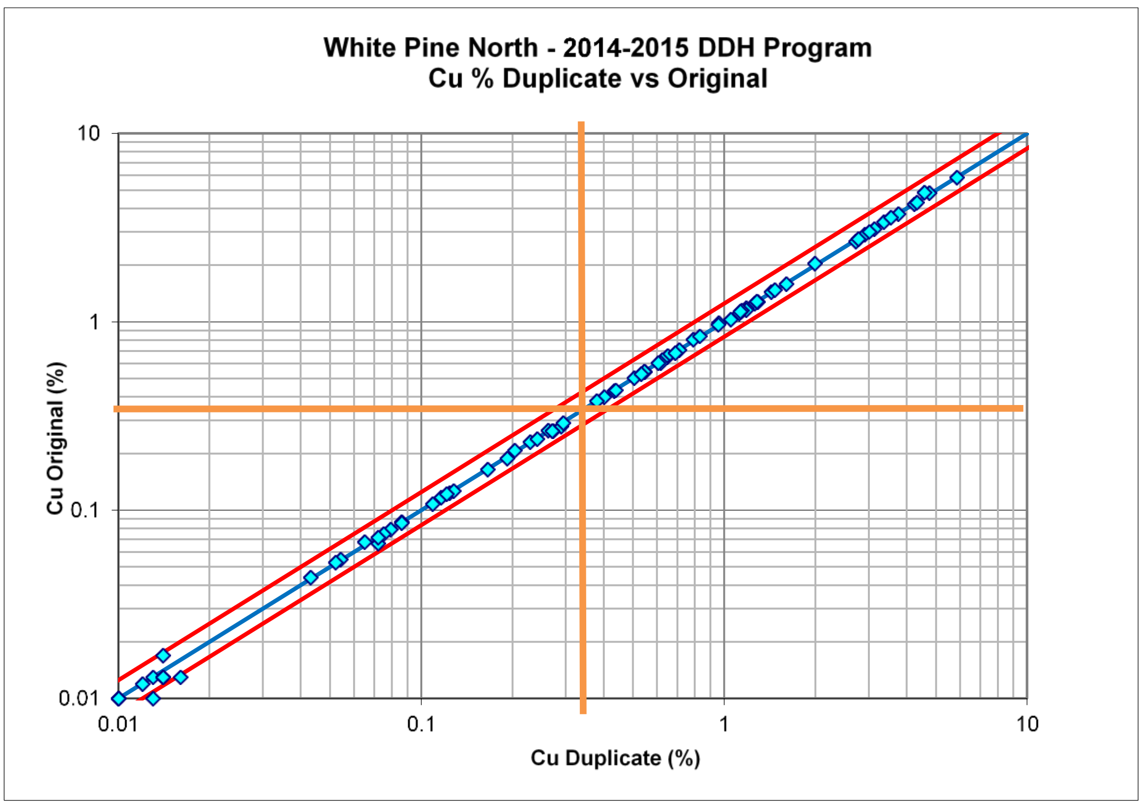
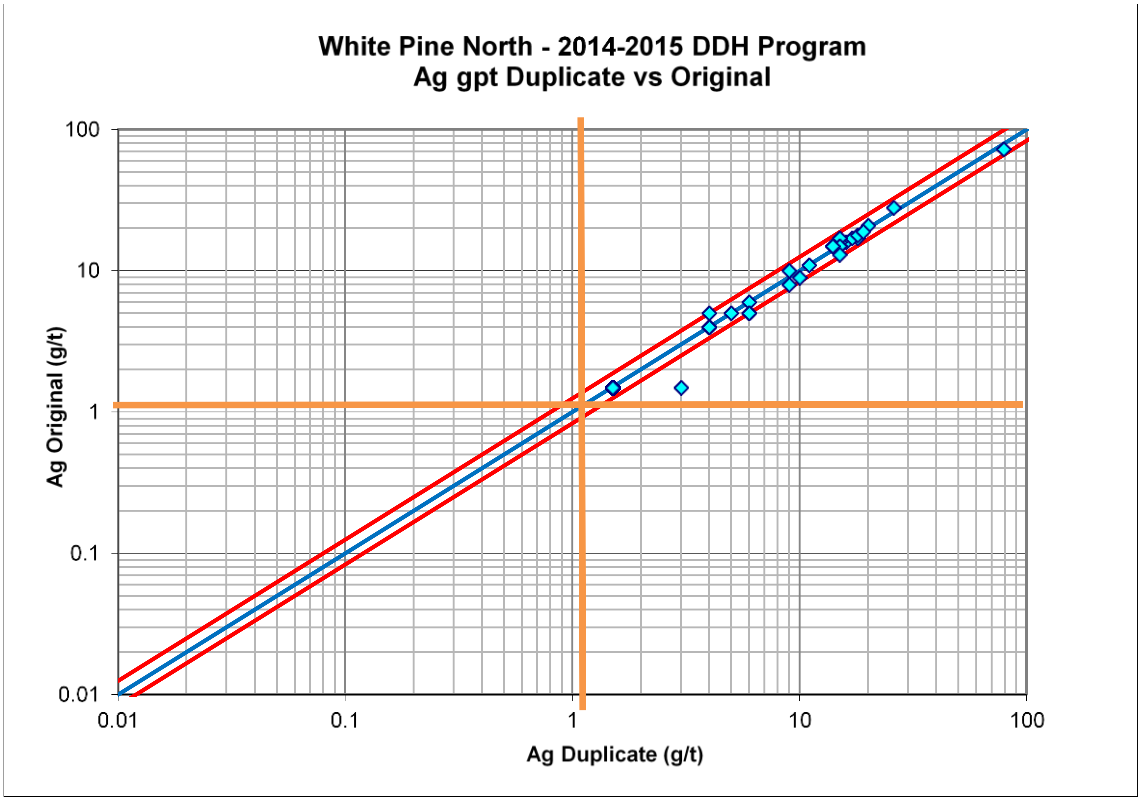


Figure 11.5: Preparation Duplicate Performance for Cu (top) and Ag (bottom)





11.3.4 <u>Performance of Certified Reference Material (CRMs or Standard)</u>

Highland's protocol is to insert two Certified Reference Materials ("CRMs" or "standard") every set of samples from one drill hole (approximately 1 in 20 samples). The site geologist alternated between a low-grade standard, middle (ore) grade standard, and a high-grade standard. CRMs were submitted with core samples for assay as control standards to identify any possible problems with specific sample batches or long-term biases in the overall dataset.

In total, six CRMs were used by Highland to monitor the consistency and accuracy of a laboratory. Three of six CRMs were manufactured by Ore Research & Exploration Pty Ltd ("OREAS"), in Australia. The other three CRMs were produced by CDN Resource Laboratories Ltd. ("CDN Labs"), in Canada. Both OREAS and CDN standards are certified in accordance with International Standards Organization ("ISO") recommendations. The Performance Gates applied for the White Pine North Project are available on the ORE Research & Exploration Pty Ltd. and CDN Resource Laboratories Ltd. website respectively (https://www.ore.com.au/oreas-reports/ and http://www.cdnlabs.com/Cu-Au-standards.htm).

Table 11.6 and Table 11.7 summarizes the CRMs of copper ("Cu") and silver ("Ag") content used for the White Pine North project and the recommended values defined by either ± 3 standard deviations (3σ) or ±

5% acceptable limits. For copper , the three standard deviation limits are used to assess results by assay methods, but the ± 5% range are used to assess geochemical results (i.e. 1F2 method – 4-acid – ICP finish).

Table 11.6: Recommended CRMs Cu (%) Values – White Pine North Drilling Program (2014-2015)

CRMs Code	Laboratory Supplier	Expected Cu Value (%)	Standard Deviation (σ)	Performance Gates			
				3σ		5%	
				Low	High	Low	High
CDN-ME-13	CDN Laboratories Inc.	2.69	0.10	2.39	2.99	2.56	2.82
CDN-ME-19	CDN Laboratories Inc.	0.474	0.009	0.447	0.501	0.450	0.498
CDN-ME-1205	CDN Laboratories Inc.	0.218	0.006	0.200	0.236	0.207	0.229
OREAS 95	Ore Research & Exploration Pty Ltd	2.59	0.01	2.39	2.79	2.46	2.72
OREAS 97	Ore Research & Exploration Pty Ltd	6.31	0.03	5.28	7.33	5.99	6.62
OREAS 162	Ore Research & Exploration Pty Ltd	0.772	0.007	0.694	0.849	0.733	0.810

For Silver (Ag), the range of values in the table below are used to asses QC failures (Table 11.7).

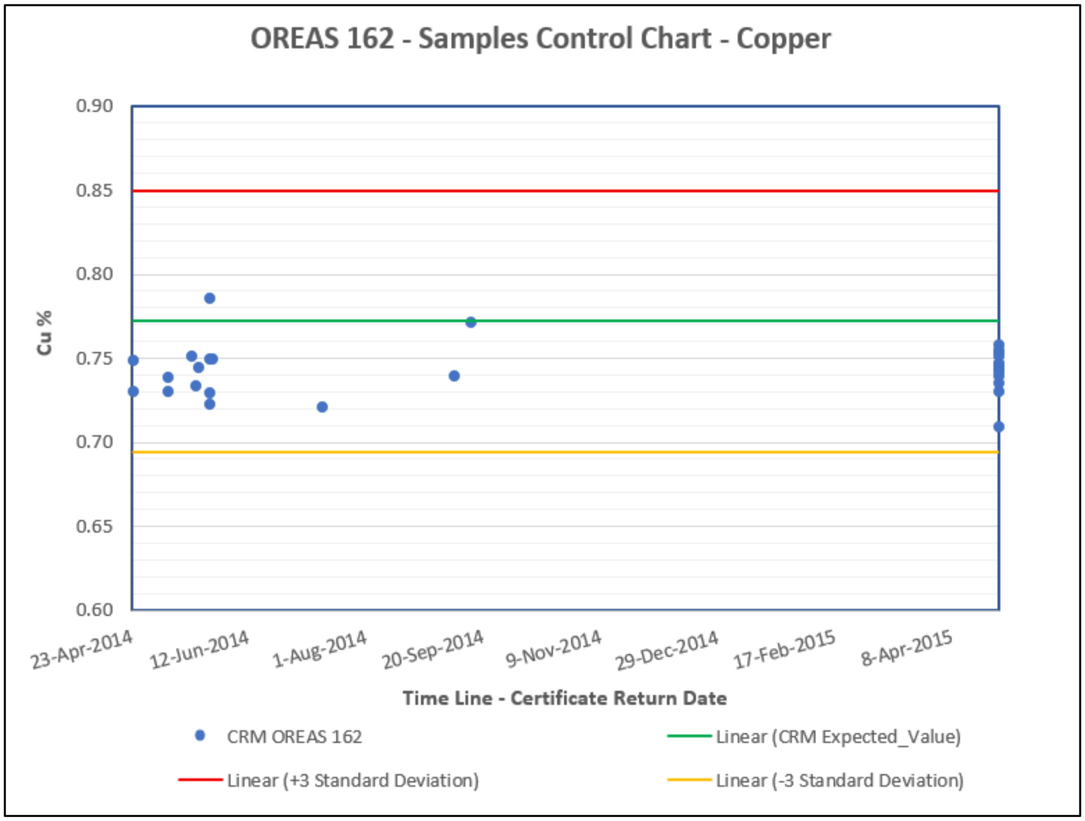
Table 11.7: Recommended CRMs Ag (gpt) Values – White Pine North Drilling Program (2014-2015)

CRMs Code	Laboratory Supplier	Expected Ag Value (g/t)	Standard Deviation (σ)	Performance Gates			
				3σ		5%	
				Low	High	Low	High
CDN-ME-13	CDN Laboratories Inc.	76.5	3.4	66.3	86.7	57.375	80.325
CDN-ME-19	CDN Laboratories Inc.	103	3.5	92.5	113.5	77.25	108.15
CDN-ME-1205	CDN Laboratories Inc.	25.6	1.2	22	29.2	19.2	26.88
OREAS 95	Ore Research & Exploration Pty Ltd	7.70	0.06	6.69	8.70	7.31	8.08
OREAS 97	Ore Research & Exploration Pty Ltd	19.6	0.2	15.7	23.6	18.7	20.6
OREAS 162	Ore Research & Exploration Pty Ltd	3.5	0.6	1.6	5.4	3.3	3.7

A total of 171 standards was submitted to Actlabs for analytical assaying. Figure 11.6, Figure 11.7, Figure 11.8, Figure 11.9, Figure 11.10 and Figure 11.11 illustrate the assaying results of the six reference materials used by Highland with a rate of 10.1% throughout 2014 and 2015 .

The overall CRMs performance is within acceptable industry parameters and indicate no significant lab bias. All 171 CRMs have analytical values less than ±3 standard deviations (± 3σ) from the certified value for copper and seven of these have an analytical value greater than ±3 standard deviations (± 3σ) from the certified value for silver. One of the silver standards fail only marginally with an analytical value of 66 g/t Ag. The lower acceptance limit for the standard is 66.3 g/t Ag so the standard was considered to pass the QA/QC.

Figure 11.6: Performance of Control Reference Material OREAS 162 for Cu (top) and Ag (bottom)



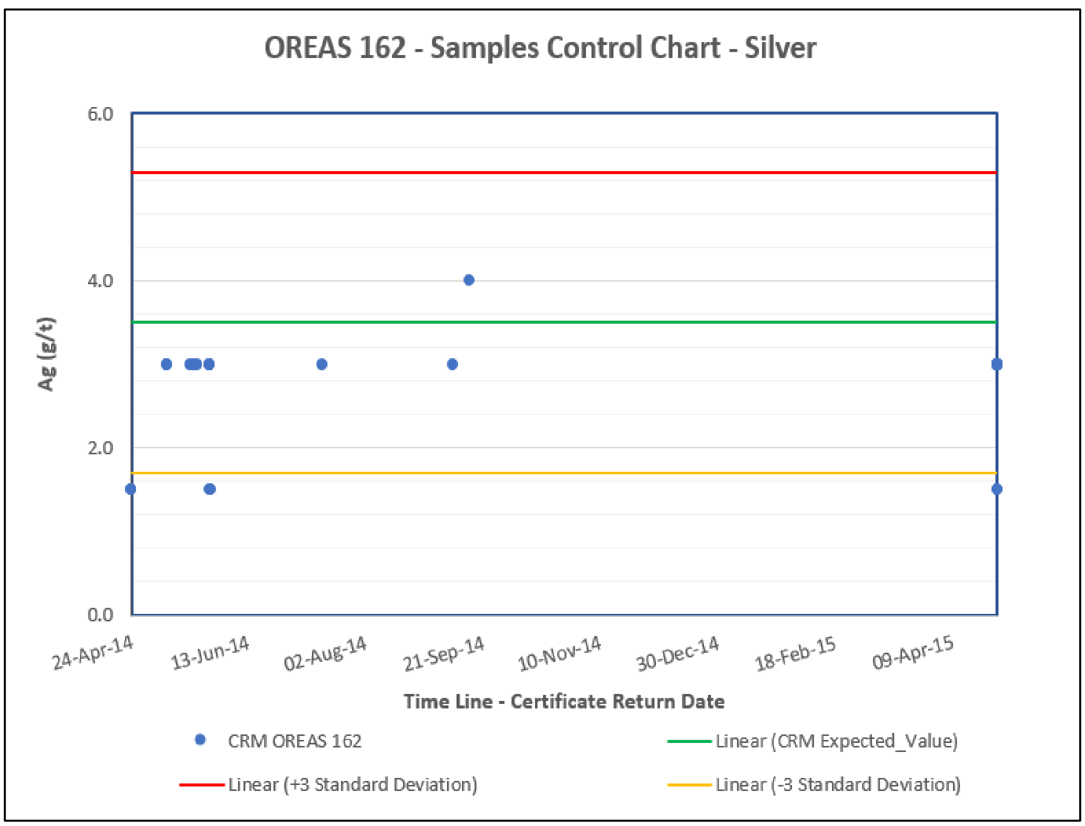
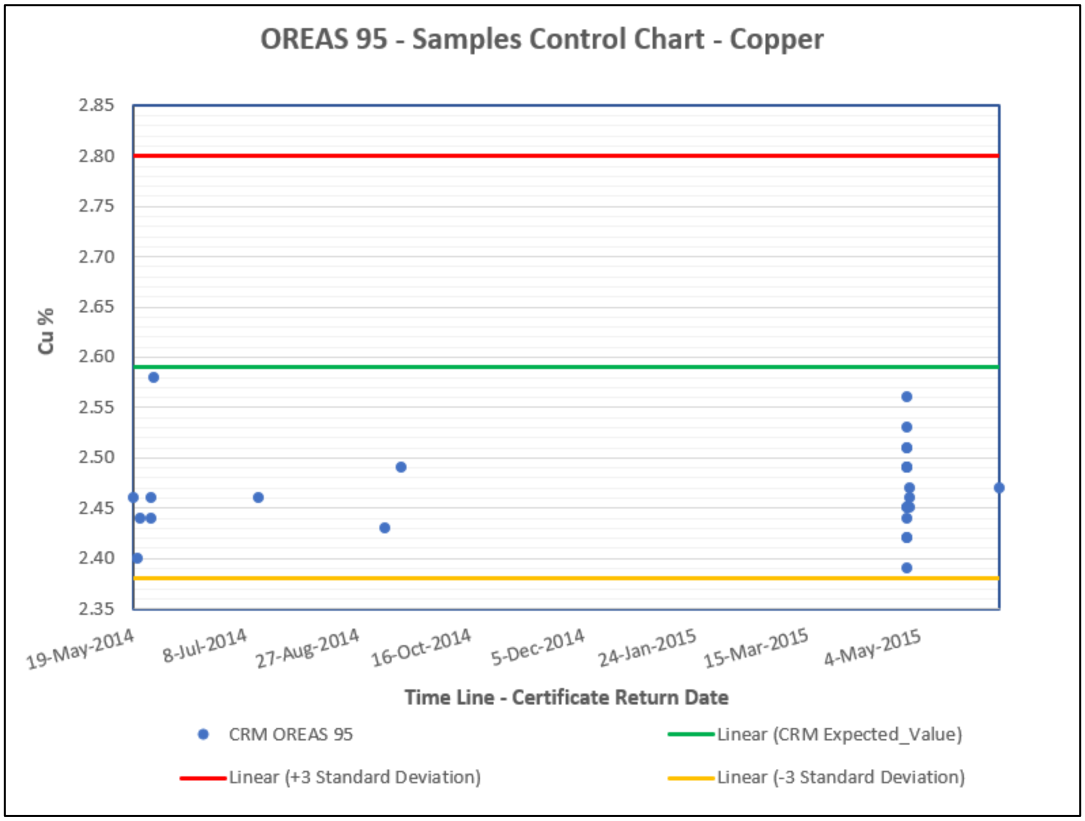


Figure 11.7:Performance of Control Reference Material OREAS 95 for Cu (top) and Ag (bottom)



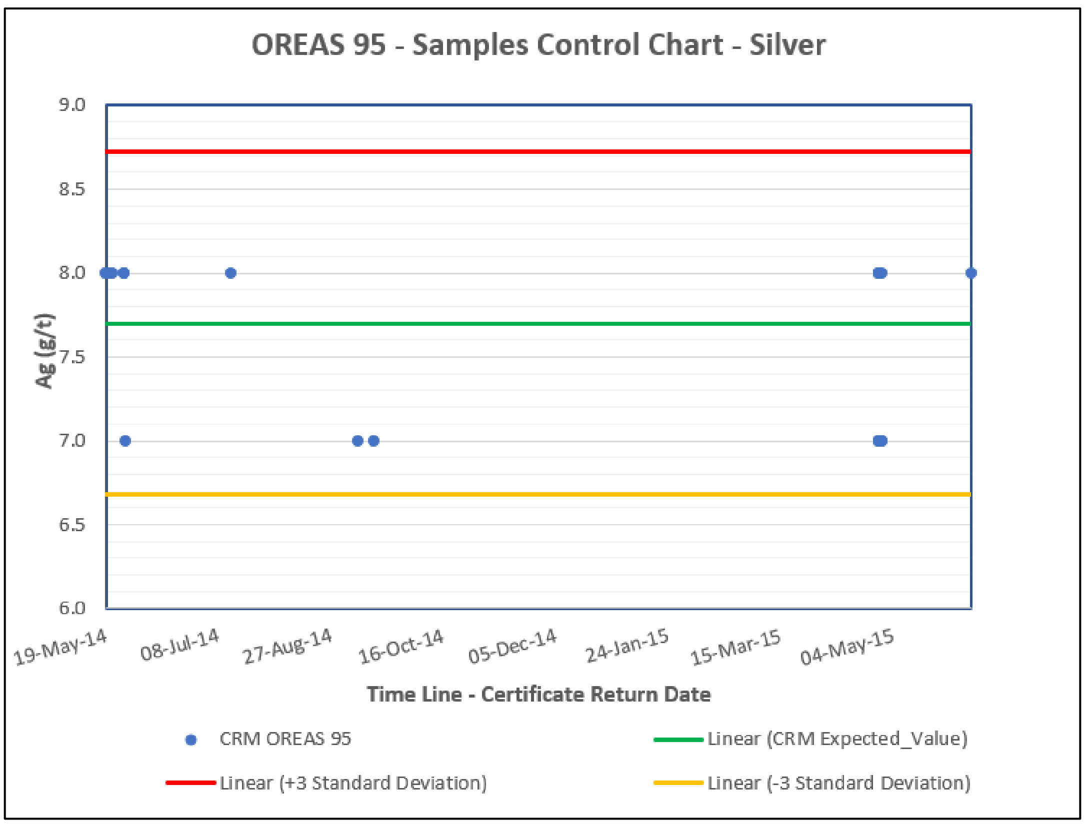
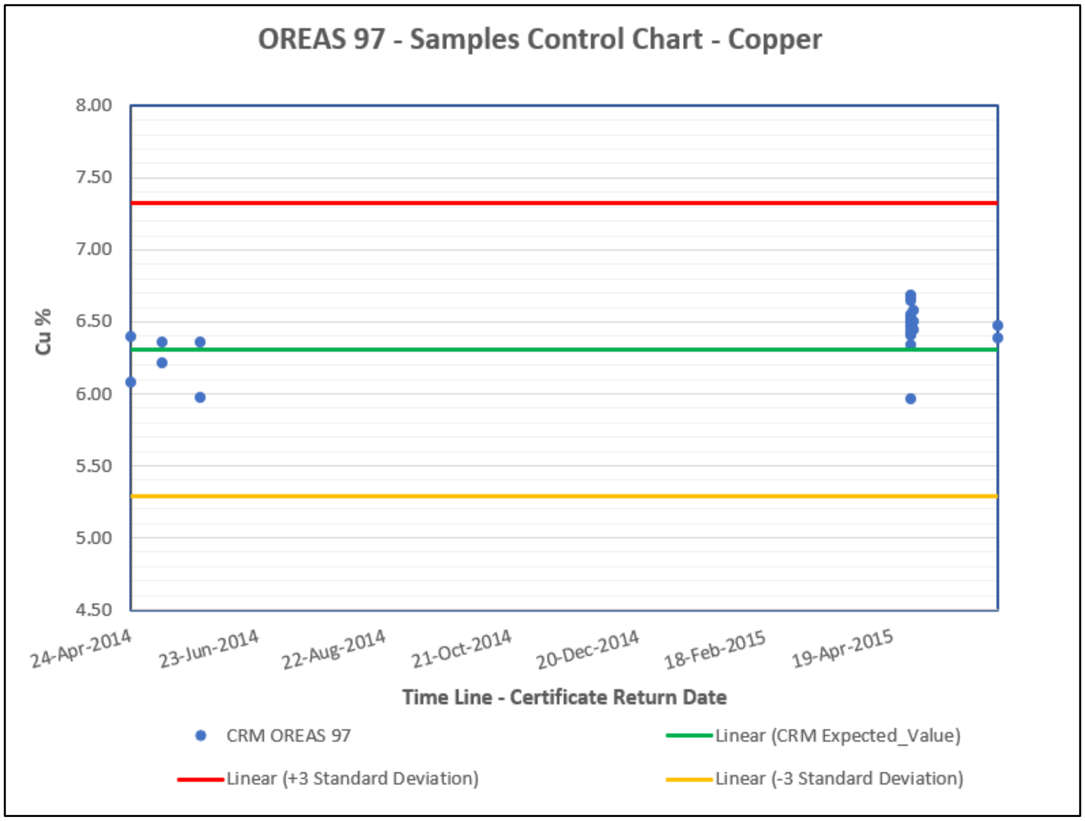


Figure 11.8: Performance of Control Reference Material OREAS 97 for Cu (top) and Ag (bottom)



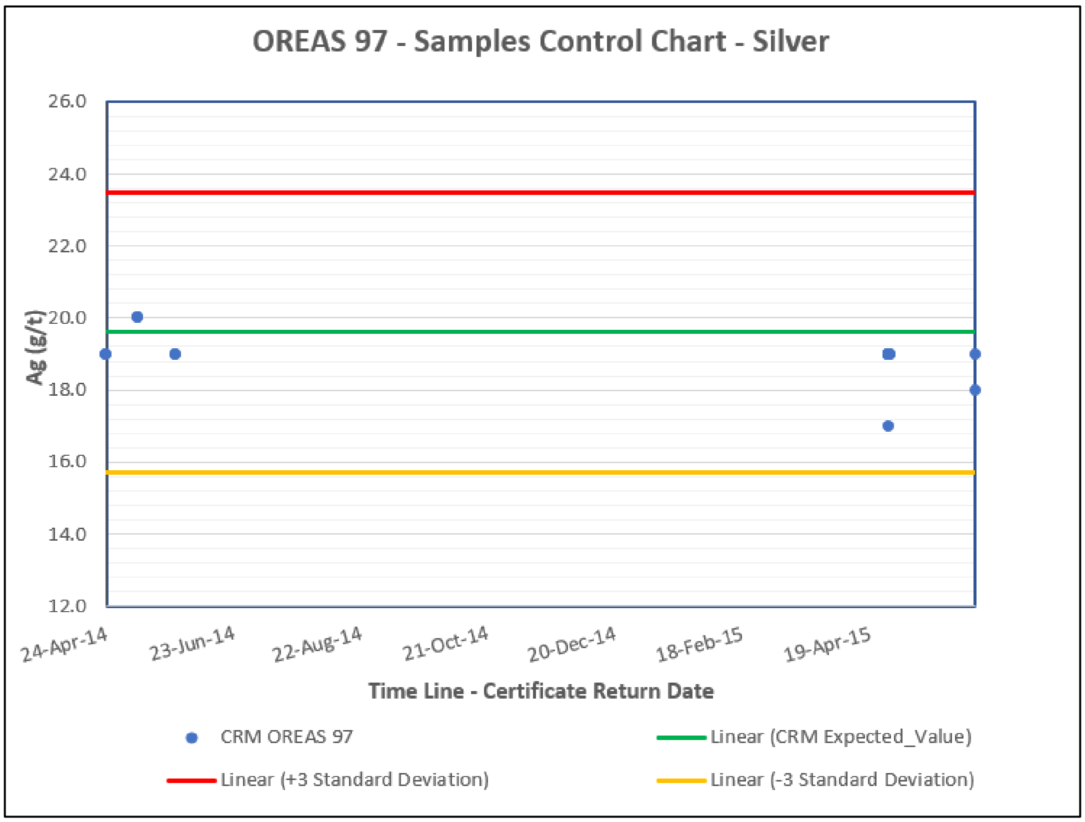
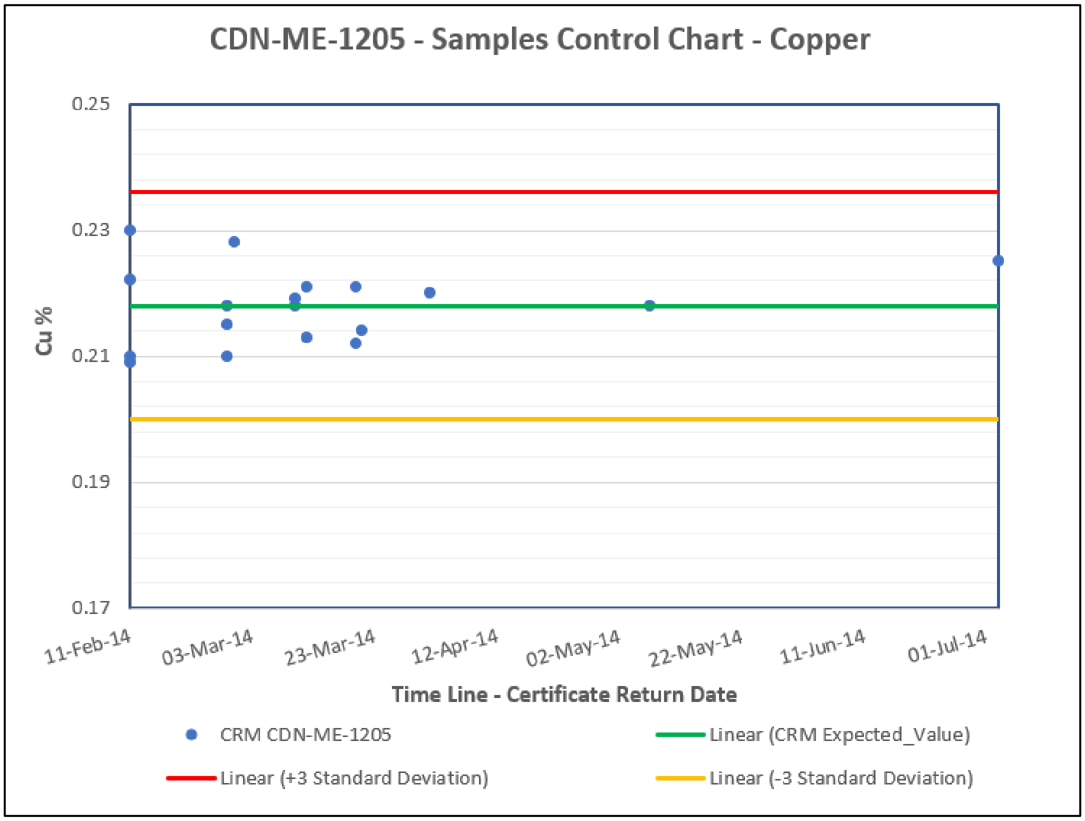


Figure 11.9: Performance of Control Reference Material CDN-ME-1205 for Cu (top) and Ag (bottom)



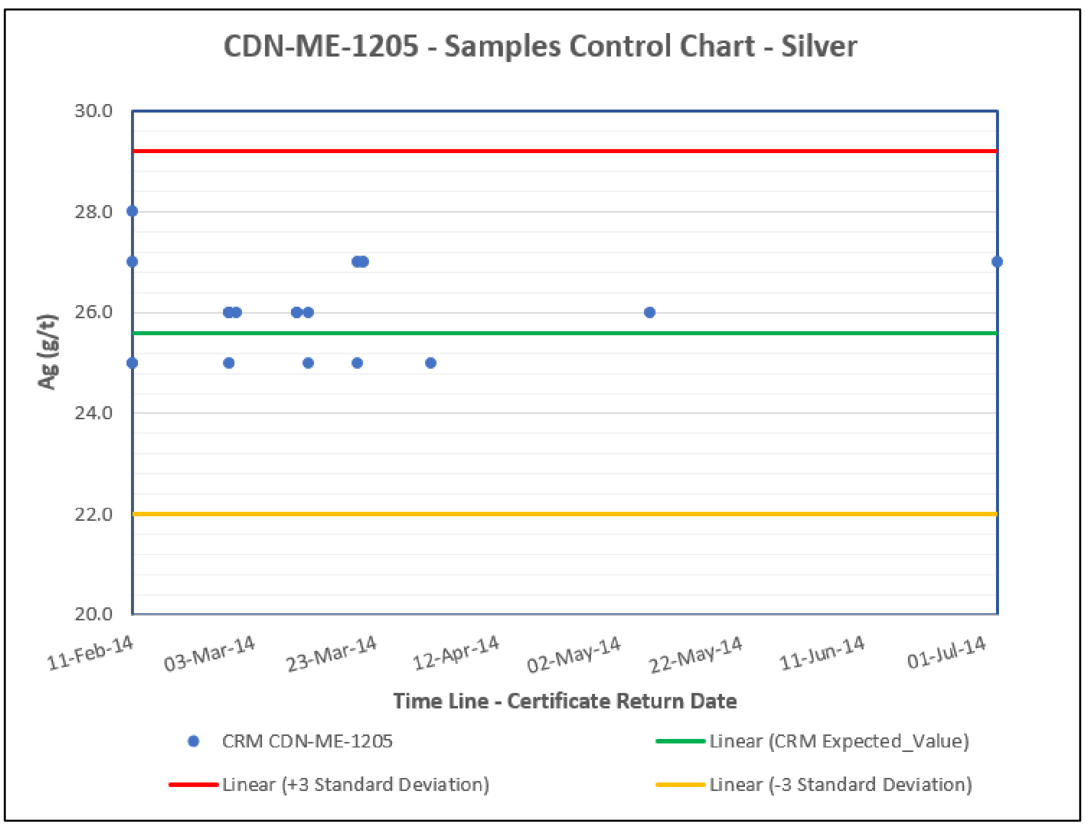
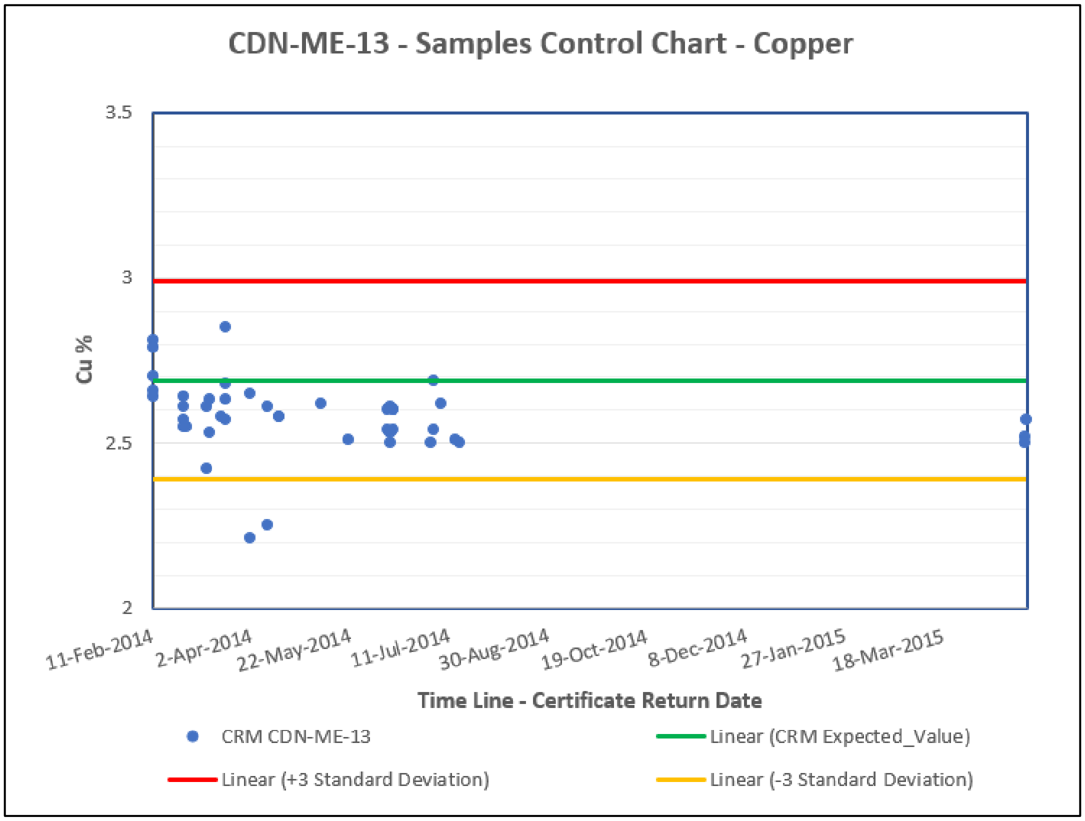


Figure 11.10: Performance of Control Reference Material CDN-ME-13 for Cu (top) and Ag (bottom)



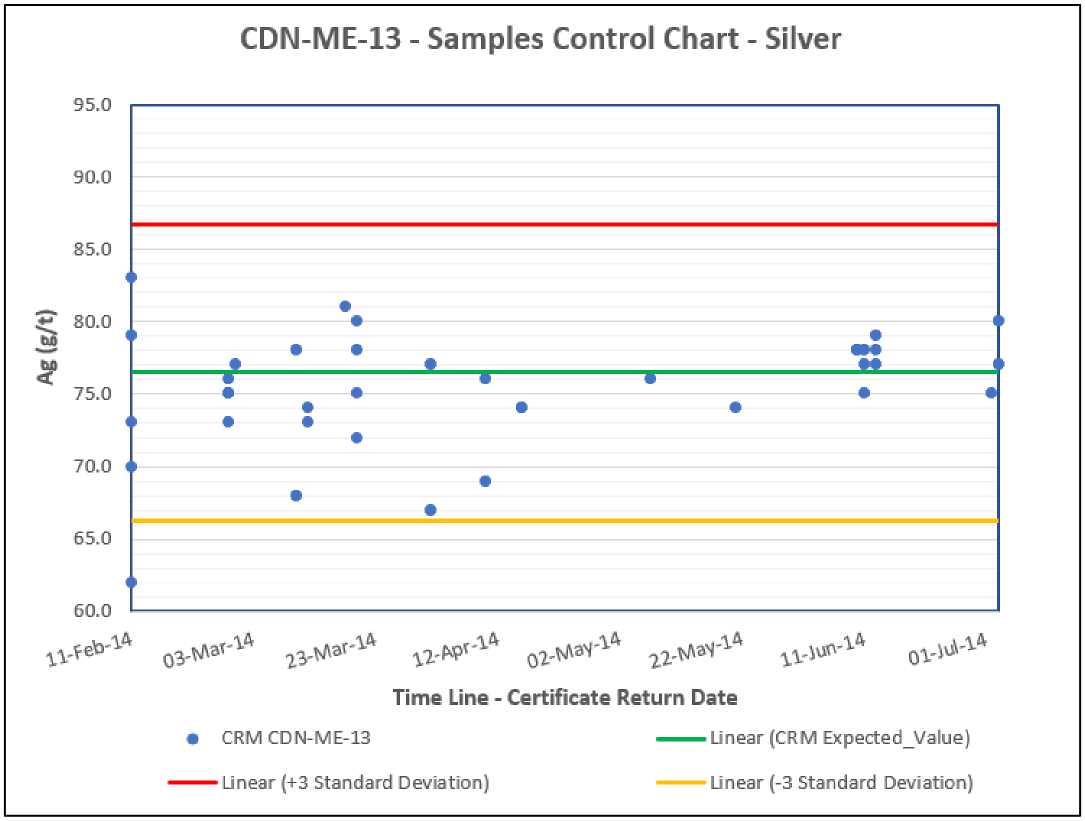
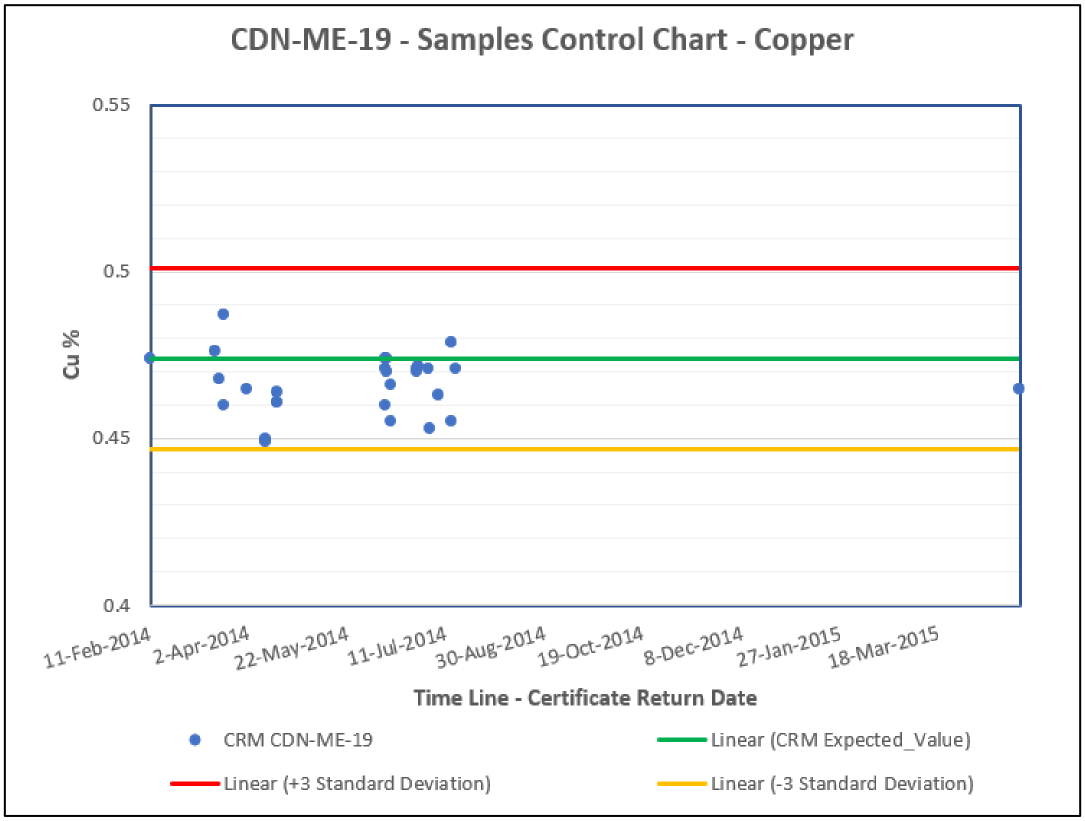
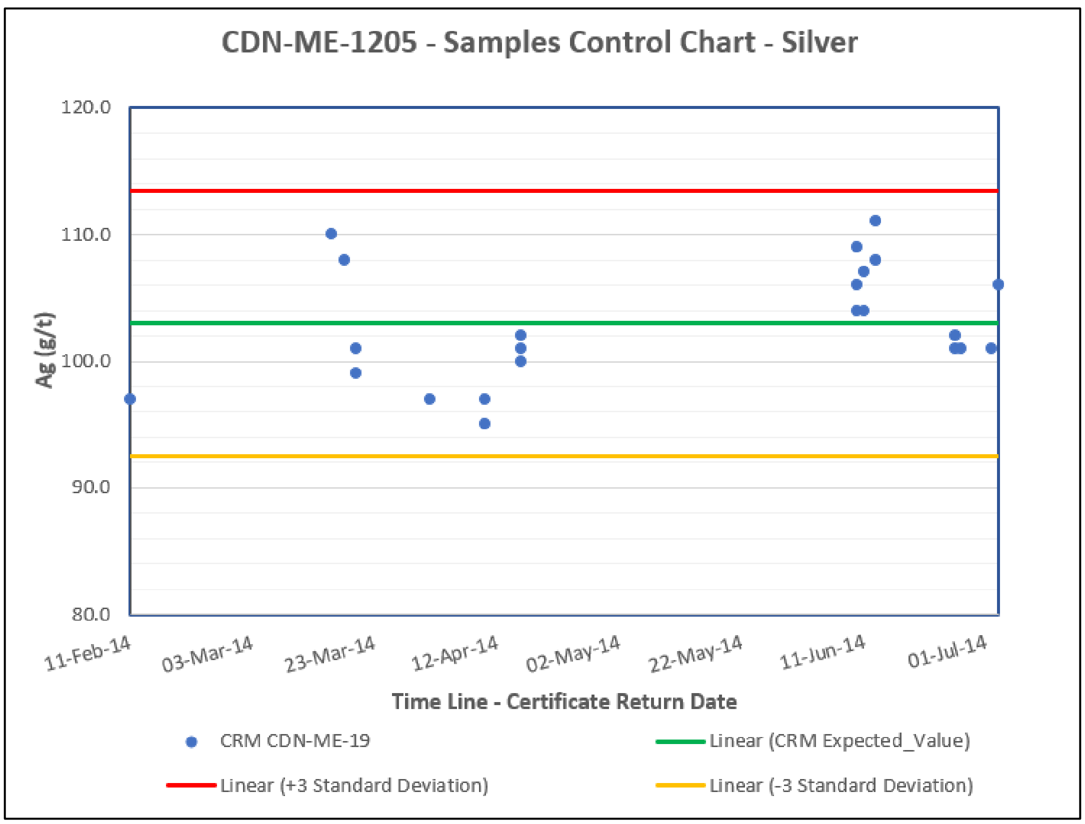


Figure 11.11:Performance of Control Reference Material CDN-ME-19 for Cu (top) and Ag (bottom)





11.4 Security

Highland maintained sample chain of custody protocols on every step of sample handling, from the drilling site to the delivery of assay results to the database manager.

11.5 GMSI Conclusions

GMSI is of the opinion that the sample preparation, analysis and QA/QC protocol used by Highland for the White Pine North Project meet accepted industry standards for the 2014-2015 drilling program. The performance of inserted blank materials and standards indicate that the sample preparation and the lab accuracy have been of good quality. Drill core sample and preparation duplicate results were acceptable for use in the current MRE.

12. DATA VERIFICATION

12.1 Database

Highland Copper Company Inc. ("Highland") provided G Mining Services Inc. ("GMSI") data files for the White Pine North Project, in date of March 2015. The information consisted of drill hole data in the form of CSV files, the White Pine fault surface trace and a Mine Workings polyline dataset. The drill hole files received were transformed in metric coordinates by Highland prior to delivery and consisted of the following tables and fields:

- Collar information: Hole ID, coordinates of collar and length of hole;
- Down-hole survey: Hole ID, down-hole depth, dip, azimuth;
- Lithology information: Hole ID, Sample ID, sample interval (From and To), rock type, bed code and ore zone;
- Assay: Hole ID, Sample ID, sample interval (From and To), length, Copper (%) and Silver (ppm).

GMSI imported the files into a MS Access database using the Geovia GEMS™ software, after converting depth and length values from feet to meters. The database was reviewed, and only minor errors were detected and corrected (mostly too short length in collar file). GMSI assumes the translation from imperial to metric coordinates was properly done and that the database is matching the original. Further field investigation is required to confirm the location of drill holes in UTM coordinates.

12.2 Drillhole Database Content

The database includes historic diamond drill holes collected between 1956 and 1995, as well as drilling by Highland up to 2015. The content of the database is summarized in Table 12.1.

Table 12.1: Content of Drill Hole Database Available for the Resource Evaluation

Hole ID	Number of Holes	Min. Length (m)	Max. Length (M)	Average Length (m)	Total Length (km)	Number of Assays	Dip Angle
WP001 to WP571	571	32.00	1,154.31	484.00	274.91	15,743	From -48° To -90°

Note: Three holes have been abandoned by Highland and have therefore been excluded from the number of holes (571) tabulated in the table (abandoned holes: WP563, WP570 and WP571).

A total of 568 diamond drill holes with assay information were available for grade estimation, and a further 567 drill holes contained lithology information which was used to build the geological model for each Ore Column. The database was reviewed and corrected if necessary, prior to final formatting for resource evaluation. The following activities were performed during database validation:

- Validate total hole lengths and final sample depth data;
- Verify for overlapping and missing intervals;
- Check drill hole survey data for out of range or suspect down-hole deviations;
- Visual check of spatial distribution of drill holes and trenches;
- Validate lithology codes.

12.3 GMSI Data Verification

During January 2014, GMSI reviewed the historical database, focussing on drilling undertaken in the north-eastern part of the White Pine Deposit.

Thorough checks of the historical information were done by examination of the drill hole log books that Highland recovered from the previous owner. The information was validated by comparison between log books and the digital database that GMSI received from Highland. Overall, the digital database was found to be in good condition and the information contained within is judged to be adequate for a resource estimate. It must be noted, however, that drill hole collar locations (in local mine grid) cannot be validated other than by field investigation.

The Qualified Person (“QP”) visually inspected the core logs which consisted of detailed information for each sample interval, including “From-To” intervals in feet, copper and silver grades in lbs/short ton and oz/short ton respectively, lithology/bed information and down-hole survey details. Except for the latter, most of the information was easily recovered for each hole in the Priority Zones Area north-east of the mined-out White Pine Mine (Figure 12.1). These 41 drill holes in question are listed in Table 12.2, by Priority Sector. Holes highlighted in red are holes from the 1994-1995 drilling campaign, logged in different books. All noted errors were transmitted to Kelly Azevedo, database manager.

Figure 12.1: Drill Holes in the Vicinity of White Pine Mined-Out Area

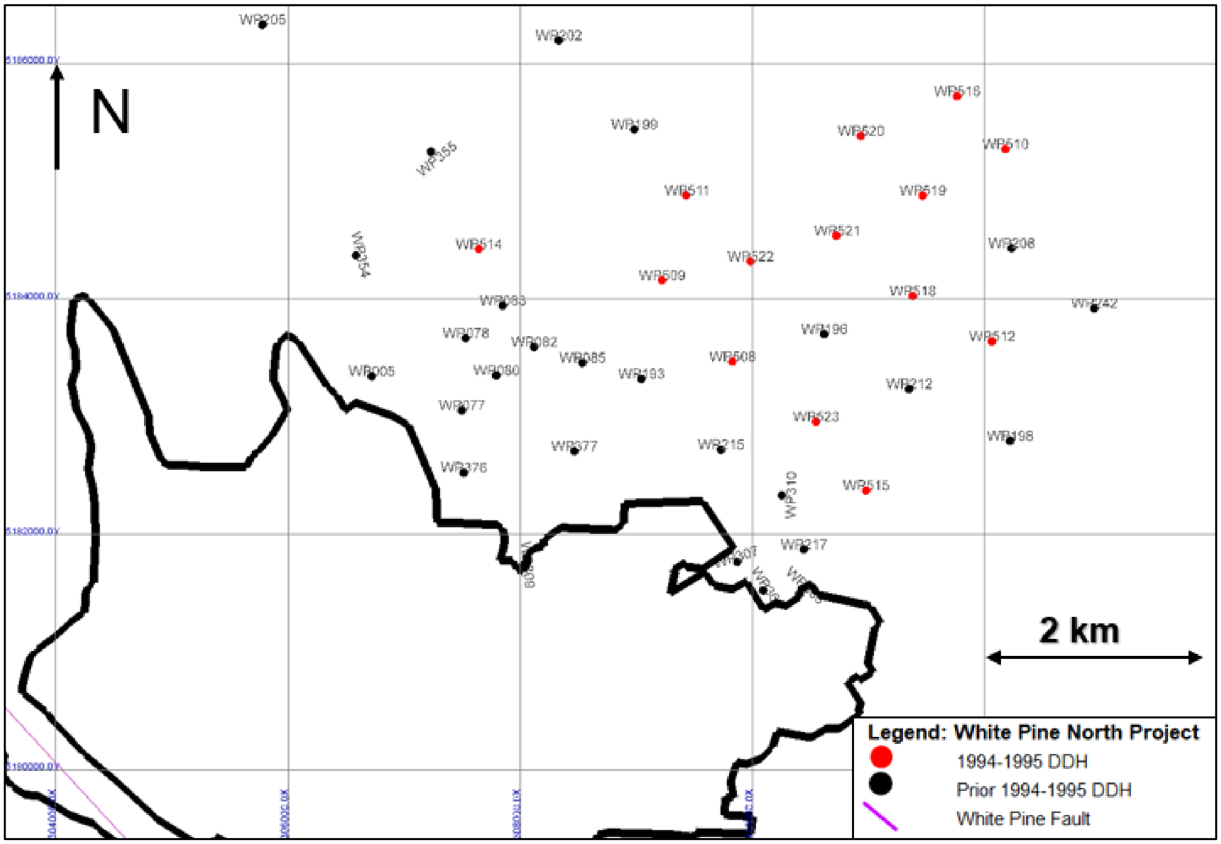


Table 12.2: Drill Holes in the Vicinity of White Pine Mined-Out Area

Historical Drilling Programs	DDH Hole ID						
DDH 1956-1980	WP_005	WP_077	WP_078	WP_080	WP_082	WP_083	WP_085
	WP_193	WP_196	WP_198	WP_199	WP_202	WP_205	WP_208
	WP_212	WP_215	WP_217	WP_242	WP_307	WP_309	WP_310
	WP_354	WP_355	WP_369	WP_376	WP_377	WP_386	
DDH 1994-1995	WP_508	WP_509	WP_510	WP_511	WP_512	WP_514	WP_515
	WP_516	WP_518	WP_519	WP_520	WP_521	WP_522	WP_523

12.3.1 Sample Interval Checks

A simple check of each sample interval was done by visual comparison of the original geological dataset (detailed core logs book) to the digital database. “From” and “To” in feet were used for comparison purpose and verifications were made to assure that the translation to metric system was done properly. Of the 38 drill holes checked, only three errors were found out of 967 intervals, making it for about 0.3% error in this category. It must be noted also that no information was found on drill holes WP_519, WP_522 and WP_523.

The errors found are listed below:

- **WP_508:** interval 2,814**.90** ft – 2,816.16 ft has been manually changed in the log book to 2,814**.50** ft – 2,816.16 ft. The digital database displays the original interval (2,814**.90** ft – 2,816.16 ft). This error is of minor importance given that the erroneous interval is above the Full Column (boundary between WID and UZV) and thus outside of grade interpolation range.
- **WP_510:** interval 3,189**.73** ft – 3,191.09 ft is a typographical error in the digital database. It should read 3,189**.23** ft – 3,191.09 ft. This error is of minor importance since it delineates a boundary between two samples of the same bed (Domino) with similar grade (2.35% vs. 2.28%)
- **WP_511:** two identical pages on lithology are in the log book, with only a change in the two topmost intervals. It is not known which one is the good information, as the original was kept in the books and not deleted. This potential error is of minor importance since the two intervals in question are above the Full Column (Widely (B47) and UZV (B46)) and thus outside of grade interpolation range.

12.3.2 Lithology Checks

Concurrently with the interval checks, a lithology validation was also performed systematically on all intervals found in log books. Of the 39 drill holes investigated, only one error was found out of 986 samples (0.1% error). No information was found for two holes (WP_522 and WP_523).

The single error found is in drill hole **WP_369:** a manual modification made in the original paper log book changed the first occurrence of L SAND (B10, Copper Harbor Conglomerate) to L TRAN (B21, Lower Transition). The digital database displays the original Copper Harbor Conglomerate (B10) data. Visual check with rock core should be made to assess this potential error, because copper grade suggest that the interval should be left as it is in the digital database, *i.e.* as Copper Harbor Conglomerate (see Table 12.3).

Table 12.3: Possible Error in Data Transcription

HOLE-ID	From (m)	To (m)	From (ft)	To (ft)	Length (m)	Code	Zone	Cu (%)
WP_369	773.69	773.80	2,538.36	2,538.70	0.10	21	LT	0.49
WP_369	773.80	773.95	2,538.70	2,539.22	0.16	21	LT	0.51
***WP_369**	**774.78**	**774.90**	**2,541.92**	**2,542.32**	**0.12**	**10**	**CHC**	**0.17**
WP_369	775.56	775.67	2,544.50	2,544.84	0.10	10	CHC	0.16

**Note: Potentially erroneous intervals*

12.3.3 Assay Checks

At the same time as sample interval and lithology checks, a visual inspection was also done on copper and silver assays. Only the transcriptions were assessed for most of the holes, since only 9 copies of laboratory certificate were available at the time of database review, with 8 of them pertaining to the area of interest: WP_508 to WP_512 and WP_514 to WP_516. All of the certificates match the digital database. Of the 35 drill holes visually inspected, only one error was found out of 908 samples (0.1% error). The following six drill holes did not contain any sample data to verify: WP_518 to WP_523.

The error found is in drill hole **WP_386**: a copper assay was mistyped in the digital database at the interval 2,616.50 ft – 2,616.92 ft (797.509 m – 797.6372 m). The copper grade found in the original log book states a grade of 1.20% Cu, whereas a value of 0.12% Cu is recorded for the same interval in the digital database. A 1.20% Cu for a sample of Upper Sandstone is considered high, especially since it is surrounded by low-grade material, so the 0.12% Cu value was retained. It should be checked more thoroughly if that mistyping was not made in the paper log books and later corrected in the database (check with laboratory certificates).

In addition to visual checks carried out on original documents during the site visit, GMSI performed data verification of assay certificates in August 2019. Approximately 50% of the assays that included only drill holes from 2014 and 2015 drilling programs (1,701 assays), was checked against the original laboratory certificates for possible typographical errors, wrong sample numbers or duplicates. No error was found during the verification.

12.3.4 Down-Hole Survey Checks

Down-hole survey verification has been less conclusive than other validations given that few drill holes in the northern sector had their survey logged in the historical log books. This is especially true for the 1994-1995 campaign where no information was available at the time of the visit; down-hole surveys are supposedly stored in a warehouse. Out of the 41 drill holes in the Priority Zones area, only seven had down-hole survey information logged in the historical log books: WP_307, WP_309, WP_310, WP_354, WP_355, WP_369 and WP_386. Out of 134 deviation intervals (each with depth, azimuth and dip information), only one error was found (about 0.2% error). The first deviation information of drill hole WP_309 reads S70°E (or N110°) in the log books, but it is recorded as N250° in the digital file. It may be an error in the original log, given that the following deviations follow the N250° trend.

Erroneous casing readings near the top of the drill hole were also identified in WP534 and WP543, and were subsequently removed

When importing the survey data into GEMS, GMSI noted that some inclined drill holes contained a vertical survey reading at the start and end of the drill hole, which created unusual deviations in 3D. These readings were retained for vertical drill holes where no surveys were available, however they were removed for the inclined drill holes.

12.4 Historical Documentation

A brief exploration of historical documents available in Highland's Calumet office was carried out. The objective was to find any piece of document useful for the continuation of the resource evaluation. Several maps were judged valuable and handed over to Highland personnel for digitizing. Those included some maps of fairly good printing quality with fine details on the structural geology of the White Pine Mine (as well as the North-Mine sector). Location of faults in this area will be critical to mine development and ore displacement.

12.5 Conclusions

GMSI assumes that all the steps leading to the final database were completed following the industry best practices to properly fulfill a Preliminary Economic Assessment for the White Pine North project. Highland retains full responsibility for the quality of the database.

In addition and based upon the evaluation of the QA/QC program undertaken by Highland, it is GMSI's opinion that the results are acceptable for use in the current Mineral Resource estimate.

13. METALLURGICAL TESTING

13.1 Historical Data

During the first 16 years of White Pine's production history, the belief was that the ore should be grinded to -45 microns in order to obtain a concentrate grading up to 35% Cu with an 85% recovery:

"Ever since work began on White Pine 16 years ago, it has been recognized that the grinding was a major problem. For a long time, it was believed that it was necessary to grind all the ore to minus 325 mesh to get a 35% concentrate with an 85% recovery". (R.H. Ramsey, 1953)

The White Pine flotation concentrate was reported to be free from any penalties:

Because the concentrate was largely copper sulphide and silicates of various kinds, it was necessary to add pyrite and lime rock to make the concentrate produce a more satisfactory slag for efficient smelting.

As sent to the reverberatory furnace., the concentrate would assay about:

	(%)
Moisture	*20*
Copper	*35*
Al_2O_3	*8*
Sulphur	*7*
Iron	*5*
CaO	*2*
MgO	*3*
SiO_2	*28*
Total	***88% dry***

Testwork then indicated that, at a 65 mesh grind, about half the copper could be recovered at once in a cleanable concentrate. The other half was contained in a floatable middling. It became apparent that the flowsheet ought to provide for rougher flotation at a coarse grind and for regrinding a part of this rougher concentrate to 325 mesh. If that were done, it ought to be possible to recover most of the remaining copper in a satisfactory concentrate.

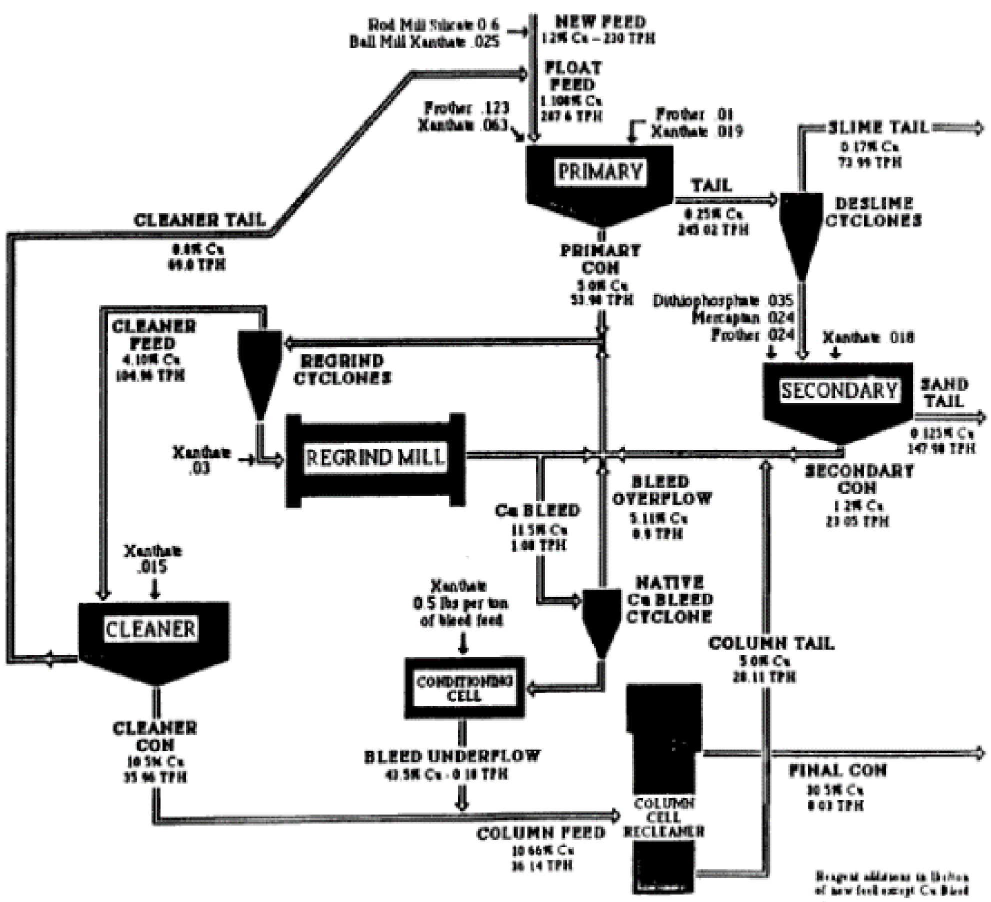
Desliming of the rougher tailing was also considered as a key part of the flotation improvement

Following the rougher flotation step, the tailing went through a cyclone with the sand portion going to a second rougher float and eventually to regrind. The slimes were discarded. This step was highly important in that it got rid of about 37% of the feed as a barren tail; by removing the slimes it sped up subsequent flotation and allowed the use of a smaller amounts of a strong collector that would otherwise be rapidly absorbed by the slime.

Reagents used are: lime, fuel oil to cut the action of some bitumen found in the ore; Minerec B and Xanthates for collectors, and pine oil and an alcohol for frothers."

A more recent report was prepared in 1992 and captured the processing flowsheet as shown in Figure 13.1 (US Environmental Protection Agency, 1992).

Figure 13.1: White Pine Historical Flowsheet



The main features of this subsequent processing flowsheet were:

- Flotation feed at P95=152 microns;
- Flotation is accomplished in four stages;
- The de-sliming of the rougher tailing;
- The regrind of the scavenger (middling) concentrate;
- A final concentrate grading around 30% Cu at an 87-89% recovery.

Used reagents and their dosage were as reported in Table 13.1:

Table 13.1: White Pine Historical Reagent Types and Consumption Rates

1991 Annual Reagent Consumption and 1992 Application Rates at Copper Range Company's White Pine Mine

Reagent	1991 Annual Consumption (tons/year)	1992 Application Rate (lbs/ton)
Xanthate	987,865	0.1821
Test Collectors	87,812	0.0160
n-Dodecyl Mercaptan	158,592	0.0273
Flocculants	62,248	0.0061
Defoamers	7,614	0.0153

13.2 Solution Mining

In the early 1990's, CRC proposed to use in situ leaching as a supplemental mining method to recover the ore remaining or to be left at White Pine from conventional mining, targeting an annual production of 60 million pounds of copper cathodes. Extensive laboratory and pilot-scale testing for solution mining operation assessment was performed in 1994 by CRC

Available data indicated that excursions of leaching solution to surrounding formations were unlikely. The direction of ground water movement and natural neutralizing capacity of the surrounding formations favored containment of the leach solution. The poor quality of ground water in the mining horizon would also be documented and monitored a part of the planned studies. Given the groundwater gradient, natural neutralizing capacity of surrounding rock, and existing poor water quality, CRC believes that it was unlikely that in situ leach mining at White Pine will result in degradation of current or potential potable water supplies.

A ferric sulfate leach on White Pine ore was closely investigated as part of a geological master theses in 1988. The thesis primarily looked at a “bio-leach” in which bacterial cultures would perform the oxidation from ferrous to ferric iron. The tests included the bio-leaching of 16 small columns over a period of 112 days. Each ore type was leached in a separate column. Some of the conclusions summarized in this thesis are listed because of their relevance to the recent investigations.

- The iron content was found to increase in the leaching solutions as the tests progressed. The dissolution was directly related to the degree of iron oxide mineralization (hematite) in the ore;
- Iron was kept in solution at pH of 1.8 or lower. Columns cemented at a pH of 2.2 due to iron mineral precipitation;
- The particle size was identified as extremely important for the copper recovery. Extraction rates of 65% were realized in 112 days on ore samples of less than 0.185 inch in size;
- Acid consumption was directly related to copper extraction and the amount of calcite gangue mineralization.

The results of subsequent laboratory tests led to the conclusion that leaching of the White Pine ore body was feasible. The major concern was that the production of ferric iron through bio-leaching was slow and questionable in the chloride rich mine water.

Throughout the history of the White Pine Mine, leach mining has been considered several times as a potential method to recover copper from the White Pine ore. This proposed concept was modified and adapted to the recent advances in the hydro-metallurgical technology.

13.3 Historical Copper Production

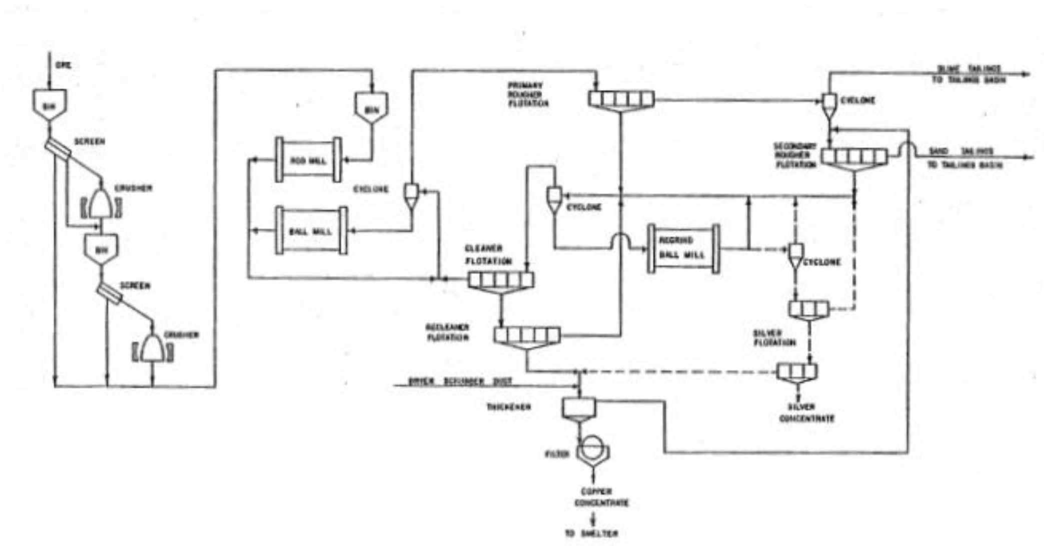
Copper mining was conducted at the White Pine Mine since 1952 and produced over 2 Mt of copper until the mine closure in 1994. In 1993, the mill treated 4.5 Mt of ore at a grade of 1.17% Cu. The average concentrate recovery and grades were 88% and 30% respectively, and the total energy consumption of the mill was approximately 31 kWh/t. Silver recovery was reported to be in the order of 90%

The general processing scheme used for White Pine copper production consisted of conventional crushing/rod and ball milling followed by staged roughing/regrind and cleaning flotation circuit, subjected to various modifications/improvements throughout the mine operation period. The last reported version of the process flowsheet is outlined in Figure 13.2 with the following highlighted features:

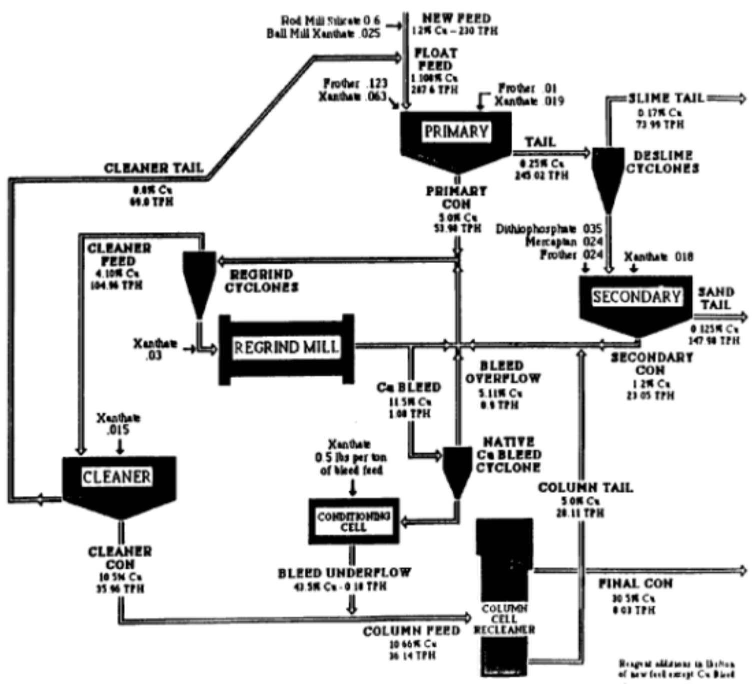
- Simple comminution circuit suitable for control of fines generation (slimes) control through rod milling;
- De-sliming/regrinding;
- Gravity Circuit (via cycloning) for Native Cu capture and Silver recovery;
- Possibility to operate the circuit for two separate concentrates (Cu, Ag) production when relevant.

Figure 13.2 Historical Processing Scheme (a & b)

(a)



(b)



13.4 <u>Recent Metallurgical Testing</u>

In 2014, Highland Copper Company ("HCC") initiated a preliminary metallurgical testing program at COREM laboratories. The objective was to validate and improve the historical performances producing a final concentrate grading of approximately 30% Cu at an average 88% recovery. Flotation testing focussed on samples from the Parting Shale ("PS") formation.

13.4.1 Metallurgical Sampling

This first testing phase used the first batch of samples from the White Pine North Mine deposit drilling. The samples locations are reported in Figure 13.3. The samples/composites inventory are listed in Table 13.2.

Figure 13.3: Met Samples Used for PS Mineralization Testing

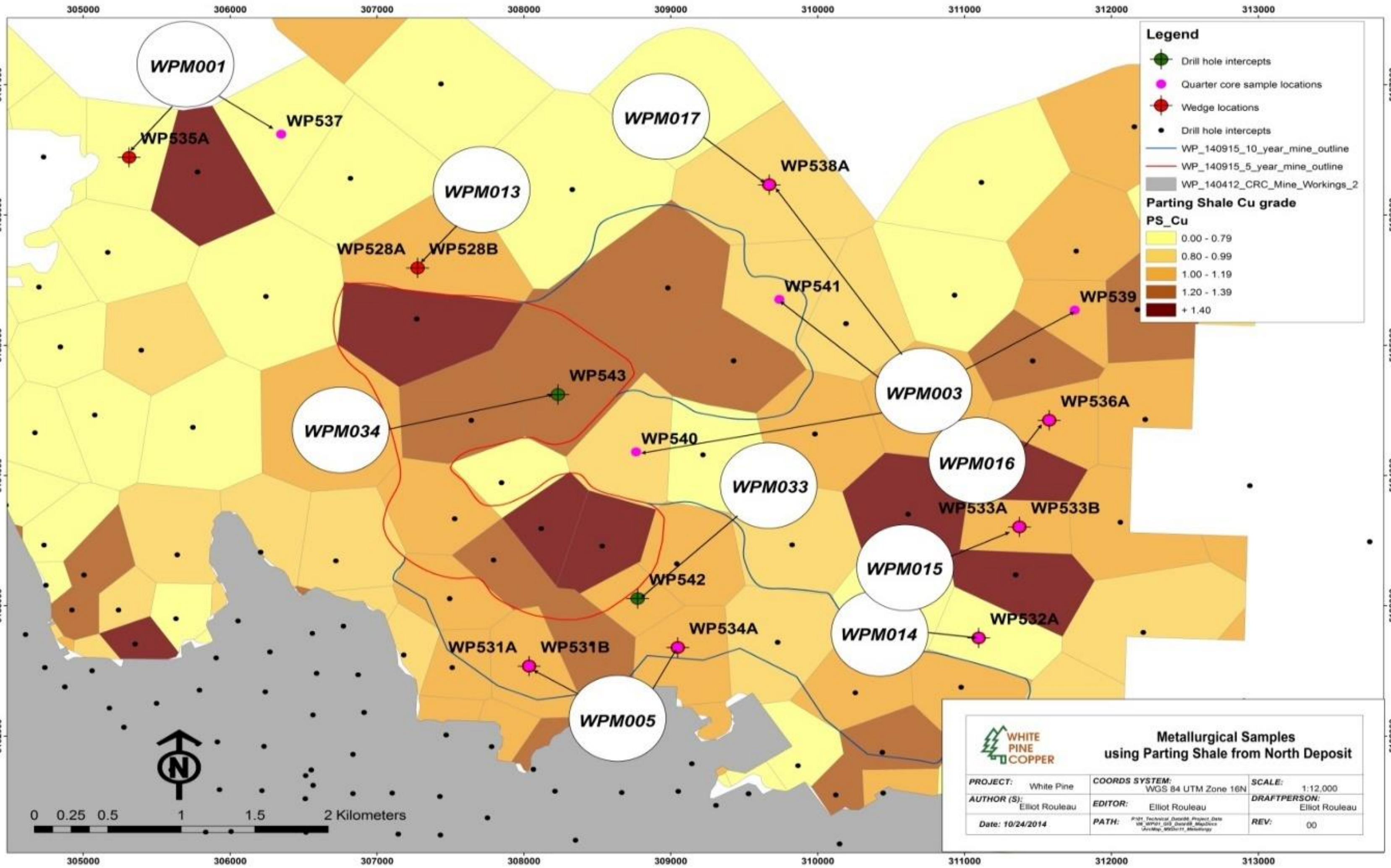


Table 13.2 Met Samples Used for White Pine North Different Mineralization Testing

Sample No.	Complete Description	HoleID	From	Configuration	% Cu	Length
WPM001	Parting Shale Outlier	WP537	quarter core	Parting Shale	0.62	3.52
		WP535A	wedge	Parting Shale	0.66	3.87
WPM002	Full Column for Ball Mill Work Index	WP532	quarter core	Full Column	0.57	4.36
		WP536	quarter core	Full Column	0.75	5.1
WPM003	Parting Shale for Rod Mill Work Index	WP538	quarter core	Parting Shale	0.79	2.80
		WP539	quarter core	Parting Shale	1.13	3.58
		WP540	quarter core	Parting Shale	0.83	2.61
		WP541	quarter core	Parting Shale	0.9	2.40
WPM004	Full Column Rod Mill Work Index	WP531A	wedge	Full Column	1.07	3.79
WPM005	Parting Shale for Ball Mill Work Index	WP531	Quarter	Parting Shale	1.06	2.20
		WP534	Quarter	Parting Shale	1.08	3.05
WPM006	Modified Parting Shale for flotation	WP534A	wedge	Modified Parting Shale	0.99	2.8
WPM007	Modified Parting Shale for flotation	WP531B	wedge	Modified Parting Shale	1.22	1.75
WPM008	UPSA for Rod Mill Work Index	WP537	Quarter	UPSA		2.61
		WP538	Quarter	UPSA		2.01
		WP539	Quarter	UPSA		2.15
		WP540	Quarter	UPSA		1.51
WPM009	UPSA for Ball Mill Work index	WP533	Quarter	UPSA		1.92
		WP534	Quarter	UPSA		2.39
		WP541	Quarter	UPSA		2.03
WPM010	UPSA for diluting (50 grams)	WP531	Quarter	UPSA		1.28
WPM011	Full Column for SMC	WP533A	Wedge	Full Column	0.87	4.24
WPM012	Modified Parting Shale for flotation	WP528A	Wedge	Modified Parting Shale	1.37	2.28
WPM013	Parting Shale for Flotation	WP528B	Wedge	Parting Shale	1.12	2.92
WPM014	Parting Shale for Flotation	WP532A	Wedge	Parting Shale	0.6	1.96
WPM015	Parting Shale for Flotation	WP533B	Wedge	Parting Shale	1.22	2.011
WPM016	Parting Shale for Flotation	WP536A	Wedge	Parting Shale	0.97	2.735
WPM017	Parting Shale for Flotation	WP538A	Wedge	Parting Shale	0.79	2.94
		WP528A	Wedge	Upper Sandstone	0.014	1.85
WPM018	UPSA for SMC Testing	WP528B	Wedge	Upper Sandstone	0.014	1.82
		WP532A	Wedge	Upper Shale	1.05	1.935
WPM019	Upper Shale for SMC Testing	WP536A	Wedge	Upper Shale	0.88	2.215
WPM020	Upper Shale for Flotation	WP528A	Wedge	Upper Shale	1.01	1.9
WPM021	Upper Shale for Flotation	WP528B	Wedge	Upper Shale	1.01	1.9
WPM022	Upper Shale for Flotation	WP531B	Wedge	Upper Shale	1.18	1.8
WPM023	Upper Shale for Flotation	WP533B	Wedge	Upper Shale	1.04	2.4
WPM024	Upper Shale for Flotation	WP534A	Wedge	Upper Shale	1.3	3.1
WPM025	Upper Shale for Flotation	WP535A	Wedge	Upper Shale	1.11	2.7
WPM026	Upper Shale for Flotation	WP538A	Wedge	Upper Shale	0.68	2.4
WPM027	LWSA for dilution	WP534A	Wedge	Lower Sandstone	0.001	0.5
WPM028	WIDE for dilution	WP528A	Wedge	WIDE	0.014	0.19
WPM029	UPSA for Crushability	WP534A	Wedge	UPSA	0	2.4
WPM030	Parting Shale for Crushability	WPU005	Bullk	Parting Shale	~1	2.46
WPM031	Parting Shale for SMC	WPU006	Bulk	Parting Shale	~1	2.12
WPM032	Parting Shale “Pillar Bench Marking” .	WPU014	Bulk	Parting Shale	~1	2.46
WPM033	Parting Shale from deposit	WP542	Half Core	Parting Shale	1.04	3.26
WPM034	Parting Shale from deposit	WP543	Half Core	Parting Shale	1.33	2.96

This campaign was almost entirely performed in Q1 2015 and a second batch of metallurgical samples in the PS was generated and submitted to COREM to be stored for PFS testing. The second sampling campaign captured the first 5-10 years of the potential mining plan.

13.4.2 Comminution Testing

Basic Bond rod and ball mill work index testing was performed at COREM on different lithological ore samples and results are in Table 13.3 and Table 13.4. No significant difference can be observed in terms of ore grindability hardness between the samples, and the ore is generally classified as a hard ore, based on the JKMRC database.

Table 13.3 Bond Ball Mill Work Index

Sample	Litho/ Configuration	Closing Sieve (µm/mesh)	BMWI (kWh/t)	Classification
WPM002	Full Column (FC)	106/150	14.5	Hard
WPM005	Parting Shale (PS)	106/150	13.9	Medium
WPM009	Upper Shale Sandstone (UPSA)	106/150	14.1	Hard
		Average	14.2	Hard

Table 13.4 Bond Rod Mill Work Index

Sample	Litho/ Configuration	Closing Sieve (mm/mesh)	RMWI (kWh/t)	Classification
WPM003	Parting Shale (PS)	1.18/14	15.9	Hard
WPM004	Full Column (FC)	1.18/14	14.8	Hard
WPM008	Upper Shale Sandstone (UPSA)	1.18/14	14.2	Hard
		Average	15	Hard

Sag Milling Comminution ("SMC") tests were performed by JKTech on samples from Full Column (WPM011), Upper Shale Sandstone (WPM018), and (WPM019), Old mine Pillar PS (WPM031) and reported in Table 13.5. Results suggested moderate hardness convenient for a SAG milling operation with no foreseen comminution circuit design issues.

Table 13.5: SMC Testing Results

Sample Designation	*A*b*				t_{10} @ 1 kWh/t			
	Value	Category	Rank	%	Value	Category	Rank	%
WPM011	33.0	hard	975	21.9	26.1	hard	1030	23.1
WPM018	37.6	moderately hard	1434	32.2	28.4	moderately hard	1496	33.6
WPM019	34.3	hard	1105	24.8	26.4	hard	1073	24.1
WMU006 (WPM031)	38.2	moderately hard	1496	33.6	28.4	moderately hard	1495	33.6

Crushing work index ("CWi") tests were conducted at FLSmidth laboratories on samples from Upper Shale Sandstone (WPM029) and old mine Pillar PS (WPU005) for preliminary hardness assessment and results are reported in Table 13.6.

Table 13.6: CWi Testing Results

Sample ID	Number of Samples Tested	Relative Density	Crusher Work Index		Classification
			kWh/short t	kWh/metric t	
WPU005	20	2.72	11.1	12.2	Soft
T-1701 WPM 0029	10	2.68	8.5	9.4	Very Soft

13.4.3 Mineralogy

The two principal copper minerals are chalcocite (Cu_2S), accounting for 80-85% of the total copper, and native copper (Cu), accounting approximately 10% of the copper. Minor sulfide minerals in the ore zone consist of covellite (CuS), Bornite (Cu_5FeS_4) and chalcopyrite ($CuFeS_2$). The ore contains approximately 10 g |Au/t. Major constituents of the mineralized zone are sandstone, shale, siltstone and limestone with the components order of magnitude in the studied PS metallurgical samples reported in Table 13.7. Roughly, gangue minerals are as following:

- 58-60% SiO_2;
- 13-14% $Al2O_3$;
- 5-7% Fe;
- 1-3% CaO;
- 3-4% Mgo;

Table 13.7: Metallurgical Samples Composition

Samples	SiO2	Al2O3	MgO	CaO	K2O	TiO2	MnO	P2O5	Co	**Cr**	Cu	Fe	Ni	Pb	**S**	**Zn**	Ag
	(%)	(%)	(%)	(%)	(%)	(%)	(%)	(%)	(%)	**(%)**	(%)	(%)	(%)	(%)	**(%)**	**(%)**	ppm
WPM013-14-16-17	57.02	13.74	3.71	2.14	2.27	1.10	0.10	0.22	0.02	**0.05**	0.95	6.36	0.03	0.02	**0.30**	**0.02**	14.27
PS outlier	62.66	12.86	3.41	1.76	2.06	1.06	0.12	0.22	0.02	0.05	0.80	5.88	0.09	0.02	0.28	0.02	16.84
WPM015	56.44	14.30	3.96	1.76	2.42	1.14	0.11	0.24	0.02	0.05	1.22	6.62	0.03	0.02	0.32	0.02	11.49
WPM013	57.64	13.41	3.78	2.33	2.31	0.99	0.14	0.21	0.02	0.05	1.10	6.30	0.03	0.02	0.23	0.02	38.33
WPM020-21-22-24	57.16	13.58	3.18	2.78	2.45	0.99	0.10	0.19	0.02	0.04	1.05	5.88	0.03	0.02	0.36	0.02	
WPM032	56.64	13.82	4.02	1.82	2.60	1.04	0.11	0.21	0.02	0.04	1.01	6.68	0.02	0.02	0.25	0.02	

A liberation study was performed on samples ground at P80=118 microns. Three size fractions (+75 µm, -75 +38 µm and -38 µm) of two flotation feed samples (PS Outlier, WPM533P) were mounted in polished sections and studied under Mineral Liberation Analyser ("MLA"). Samples were studied to better understand their composition and their copper, silver and iron distributions. Mineral composition of the two samples is given in Table 13.8 and Table 13.9. Chalcocite is the most abundant copper; native copper is in lower concentration and chalcopyrite is mostly undetected in any sample. In these samples, the silicates, mainly represented by the quartz and the feldspars, constitute the main minerals. As indicated in Table 13.10 and Table 13.11, copper seems to be equally concentrated in the size fractions at a level close to 1%.

Table 13.8: Mineral Composition of PS Outlier Sample

PS Outlier - Wt%				
Size fraction	+75 µm	-75 +38 µm	-38 µm	Total
Albite	24	28	27	24
Quartz	38	34	27	37
Mixed Feldspars	12	10	11	12
Chlorite	10.3	12	14	11
Orthoclase	7.3	6.8	7.2	7.3
Muscovite	2.3	2.3	2.7	2.3
Amphibole(s)	1.5	0.9	1.5	1.4
Plagioclase	0.4	0.3	0.7	0.4
Chalcocite	0.7	1.0	1.0	0.7
Titanite	1.2	2.0	1.9	1.3
Other silicates	0.4	0.8	2.6	0.5
Calcite	0.9	1.1	0.9	0.9
Ilmenorutile	0.3	0.4	0.6	0.3
Apatite	0.3	0.4	0.4	0.3
Iron hydroxides	0.1	0.1	0.2	0.1
Native_Copper	0.1	0.1	0.0	0.1
Ilmenite	0.1	0.1	0.3	0.1
Iron oxides	0.1	0.2	0.2	0.1
Bornite	0.0	0.1	0.1	0.0
Zircon	<0.1	<0.1	<0.1	<0.1
Native silver	n.d.	n.d.	n.d.	n.d.
Galena	n.d.	n.d.	n.d.	n.d.
Gold	n.d.	n.d.	n.d.	n.d.
Chalcopyrite	n.d.	n.d.	n.d.	n.d.
Greenockite	n.d.	n.d.	n.d.	n.d.
Pyrite	n.d.	n.d.	n.d.	n.d.
Pyrrhotite	n.d.	n.d.	n.d.	n.d.
Total	100.0	100.0	100.0	100.0

Table 13.9: Mineral Composition of WPM533P Sample

WPM533P - Wt%				
Size fraction	+75 µm	-75 +38 µm	-38 µm	Total
Albite	27	29	27	27
Quartz	22	27	28	22
Mixed Feldspars	22	16	12	22
Chlorite	9.8	11	12	10
Orthoclase	7.7	7.4	7.7	7.7
Muscovite	3.7	2.7	3.2	3.6
Amphibole(s)	2.0	1.5	1.3	1.9
Plagioclase	1.4	0.8	0.9	1.3
Chalcocite	0.9	1.0	1.4	0.9
Titanite	0.9	1.2	1.7	0.9
Other silicates	0.7	1.0	1.6	0.7
Calcite	0.4	0.7	0.6	0.4
Ilmenorutile	0.3	0.4	0.5	0.3
Apatite	0.2	0.3	0.4	0.2
Iron hydroxides	0.2	0.1	0.2	0.2
Native_Copper	0.2	0.3	0.1	0.2
Ilmenite	0.1	0.1	0.3	0.1
Iron oxides	0.1	0.1	0.4	0.1
Bornite	0.1	0.1	0.2	0.1
Zircon	<0.1	<0.1	<0.1	<0.1
Native silver	n.d.	n.d.	n.d.	n.d.
Galena	n.d.	n.d.	n.d.	n.d.
Gold	n.d.	n.d.	n.d.	n.d.
Chalcopyrite	n.d.	n.d.	n.d.	n.d.
Greenockite	n.d.	n.d.	n.d.	n.d.
Pyrite	n.d.	n.d.	n.d.	n.d.
Pyrrhotite	n.d.	n.d.	n.d.	n.d.
Total	100.0	100.0	100.0	100.0

Table 13.10: Calculated Assay of PS Outlier Sample

PS Outlier - Wt%				
Size fraction	+75 µm	-75 +38 µm	-38 µm	Total
O	48.9	48.4	47.7	48.9
Si	33.8	32.5	30.6	33.6
Al	6.5	6.9	7.6	6.6
Na	2.5	2.8	2.8	2.6
Ca	2.0	2.0	2.1	2.0
Mg	1.8	1.9	2.4	1.8
K	1.3	1.2	1.3	1.2
Fe	1.3	1.6	2.6	1.4
Cu	0.6	0.9	0.9	0.7
Ti	0.4	0.5	0.6	0.4
S	0.2	0.2	0.3	0.2
H	0.2	0.2	0.2	0.2
Ta	0.2	0.2	0.3	0.2
Nb	0.1	0.1	0.2	0.1
C	0.1	0.1	0.1	0.1
F	0.0	0.1	0.1	0.0
P	0.1	0.1	0.1	0.1
RE	0.0	0.1	0.1	0.1
Zr	0.0	0.0	0.0	0.0
Total	99.8	99.8	99.7	99.8

Table 13.11: Calculated Assay of WPM533P

WPM533P - Wt%				
Size fraction	+75 µm	-75 +38 µm	-38 µm	Total
O	47.8	47.8	47.4	47.8
Si	31.2	31.6	31.1	31.2
Al	8.3	7.7	7.4	8.2
Na	3.3	3.1	2.8	3.3
Ca	2.5	2.2	2.0	2.5
Mg	1.8	1.9	2.0	1.8
K	1.5	1.3	1.4	1.4
Fe	1.4	1.6	2.2	1.4
Cu	1.0	1.2	1.3	1.0
Ti	0.3	0.4	0.5	0.3
S	0.2	0.2	0.4	0.2
H	0.2	0.2	0.2	0.2
Ta	0.1	0.2	0.3	0.1
Nb	0.1	0.1	0.1	0.1
C	0.0	0.1	0.1	0.1
F	0.1	0.0	0.1	0.1
P	0.0	0.0	0.1	0.0
RE	0.0	0.0	0.1	0.0
Zr	0.0	0.0	0.0	0.0
Total	99.6	99.6	99.4	99.6

Typical copper and iron distributions in different size fractions are shown in Table 13.10. Chalcocite is the dominant source of copper contributing to 86% and 76% of the copper in the two samples. It should be noted that chalcocite is evenly distributed into the three size fractions. The second source of copper is the native copper that could count for as much as 27% of the total copper. It could be concluded that native copper is better liberated than chalcocite and almost completely recovered in the two finest size fractions. Chalcopyrite doesn't count as a source of copper, excepted for a very minor contribution in the sample PS Outlier. Table 13.12 presents the iron distribution in the samples. In “a” the hematite only counts for less than 10% of the total iron present in these samples. The two main sources of iron are observed in “b” which are the chlorite retaining up to 70% of the total iron and the other silicates that retain approximately 15% of it. In total, the iron locked in the silicate composition represents 85% of the total iron present in the samples.

Table 13.12: Iron Distribution in the Three Samples

a. Iron and iron oxide content

Element	PS outlier - Wt%	WPM533P - Wt%	Test11 Tail - Wt%
Fe	1.37	1.42	1.80
Fe2O3	1.96	2.03	2.57
Measured hematite	0.13	0.10	0.26

b. Iron oxide mineral distribution

Mineral	PS outlier - Fe (%)	WPM533P - Fe (%)	Test11 Tail - Fe (%)
Chlorite	69.77	63.16	60.38
Other silicates	13.04	16.62	17.84
Iron oxides	6.7	5.1	9.9
Iron hydroxides	4.1	9.2	4.7
Ilmenite	2.9	2.7	3.9
Ilmenorutile	1.9	1.6	2.1
Titanite	1.3	0.9	1.1
Bornite	0.2	0.5	<0.1
Pyrite	<0.1	<0.1	<0.1
Chalcopyrite	<0.1	<0.1	<0.1
Pyrrhotite	<0.1	0.2	<0.1
Total	100	100	100

The chalcocite liberation and association are summarized in Table 13.13, Table 13.14 and Table 13.15 it indicates that liberation is extremely low, even in the -38 µm size fraction where it reaches only 20% in the head samples and 0% in the tail sample of a rougher flotation tailing sample. Most of the chalcocite (close to 75%) is locked in ternary (or more) particles, suggesting it is still associated to two minerals or more in a single particle.

Table 13.13: Chalcocite Liberation in PS Outlier

PS Outlier - Wt%				
Size fraction	+75 µm	-75 +38 µm	-38 µm	Total
Free	4.1	13.3	18.7	5.0
In binary particle	14.9	50.2	62.7	17.7
In ternary+ particle	81.0	36.6	18.6	77.3
Main binary associations				
Quartz	6.0	9.3	0.0	6.0
Albite	4.2	14.0	0.0	4.6
Mixed Feldspars	3.2	6.6	0.0	3.3
Chlorite	0.2	10.5	0.0	0.7
Orthoclase	0.0	0.0	0.0	0.0
Other silicates	0.0	9.7	62.7	2.0

Table 13.14: Chalcocite Liberation in WPM533P

WPM533P - Wt%				
Size fraction	+75 µm	-75 +38 µm	-38 µm	Total
Free	2.3	7.8	51.9	5.5
In binary particle	19.5	35.2	30.7	20.6
In ternary+ particle	78.2	57.0	17.4	73.8
Main binary associations				
Quartz	4.9	6.8	10.1	5.3
Albite	7.6	11.4	5.8	7.6
Mixed Feldspars	1.9	2.9	0.7	1.9
Chlorite	0.5	3.2	2.7	0.7
Orthoclase	0.4	1.0	1.7	0.5
Other silicates	0.5	4.3	2.1	0.7

Table 13.15: Chalcocite Liberation in Test 11 Tail

Test11 Tail - Wt%				
Size fraction	+75 µm	-75 +38 µm	-38 µm	Total
Free	0.0	0.0	0.0	0.0
In binary particle	56.2	100.0	0.0	57.5
In ternary+ particle	43.8	0.0	0.0	42.5
Main binary associations				
Quartz	13.4	0.0	0.0	13.0
Albite	42.6	0.0	0.0	41.3
Mixed Feldspars	0.0	0.0	0.0	0.0
Chlorite	0.0	0.0	0.0	0.0
Orthoclase	0.0	100.0	0.0	2.9
Other silicates	0.0	0.0	0.0	0.0

As shown in Table 13.16, Table 13.17 and Table 13.18, liberation of native copper is extremely low, only reaching 14.6, 5.0 and 0.2 % into the PS Outlier, WPM533P and Test 11 Tail respectively. In the three samples, the native copper is mostly locked in ternary particles with albite, quartz and chlorite.

Table 13.16: Native Copper Liberation in PS Outlier

PS Outlier - Wt%				
Size fraction	+75 µm	-75 +38 µm	-38 µm	Total
Free	13.4	11.3	40.4	14.6
Binary	23.5	39.7	31.0	24.7
Ternary	63.1	49.1	28.5	60.7
Main binary associations				
Albite	7.9	8.4	5.1	7.8
Quartz	8.5	6.9	2.2	8.1
Muscovite	1.1	1.1	1.6	1.2
Mixed Feldspars	3.2	6.3	3.2	3.3
Chlorite	0.6	10.5	2.8	1.2

Table 13.17: Native Copper Liberation in WPM533P

WPM533P - Wt%				
Size fraction	+75 µm	-75 +38 µm	-38 µm	Total
Free	4.1	13.3	18.7	5.0
Binary	14.9	50.2	62.7	17.7
Ternary	81.0	36.6	18.6	77.3
Main binary associations				
Albite	4.2	14.0	0.0	4.6
Quartz	6.0	9.3	0.0	6.0
Muscovite	0.5	0.0	0.0	0.4
Mixed Feldspars	3.2	6.6	0.0	3.3
Chlorite	0.2	10.5	0.0	0.7

Table 13.18: Native Liberation in Test 11 Tail

Test11 Tail - Wt%				
Size fraction	+75 µm	-75 +38 µm	-38 µm	Total
Free	0.3	0.3	0.0	0.2
Binary	19.6	25.9	46.7	28.2
Ternary	80.2	73.8	53.3	71.6
Main binary associations				
Albite	8.2	10.3	1.9	8.1
Quartz	3.6	6.6	20.2	8.4
Muscovite	0.8	0.8	0.0	0.6
Mixed Feldspars	3.9	4.8	8.1	5.1
Chlorite	1.2	1.2	13.2	3.5

Liberation of silver minerals was investigated in one sample. Only 5 native silver grains were observed in PS Outlier. These grains were associated with quartz and feldspars in ternary particles. This tendency was also shared with some chalcocite grains. Table 13.19 presents these associations where only 43% of the native silver grains are "visible" at the edges of particles.

Table 13.19: Native Silver Liberation in PS Outlier

Mineral	Native silver
Quartz	35.15
Albite	1.3
Chlorite	0
Titanite	0
Mixed Feldspars	16.38
Orthoclase	0
Chalcocite	3.85
Calcite	0
Muscovite	0
Other silicates	0
Bornite	0
Amphibole(s)	0
Plagioclase	0
Free Surface	43.32

Table 13.20: Copper Distribution in PS Outlier

PS Outlier - Wt%				
Size fraction	+75 µm	-75 +38 µm	-38 µm	Total
Chalcocite	86.3	89.8	87.4	86.5
Native_Copper	11.3	5.8	3.6	10.7
Bornite	2.3	3.9	8.9	2.7
Chalcopyrite	0.1	0.5	0.1	0.1
Total	100.0	100.0	100.0	100.0

Table 13.21: Copper Distribution in WPM533P

WPM533P - Wt%				
Size fraction	+75 µm	-75 +38 µm	-38 µm	Total
Chalcocite	76.6	68.5	81.9	76.6
Native_Copper	19.8	27.2	8.4	19.4
Bornite	3.6	4.3	9.6	4.0
Chalcopyrite	0.0	0.0	0.1	0.0
Total	100.0	100.0	100.0	100.0

Liberation classes are defined as (cumulative from 100% free of the native copper and the chalcocite. Table 13.20 and Table 13.21 show the liberation classes (cumulative from 100% free) of the native copper and the chalcocite Size distribution of the native copper and the chalcocite (expressed as p-value), shows that ~20 % of the mineral is smaller than 22 µm for native copper and 8 µm for chalcocite, suggesting that fine regrind less than 20 microns is recommended for +30% Cu concentrate grade target with acceptable Cu recovery.

13.4.4 Flotation Testing

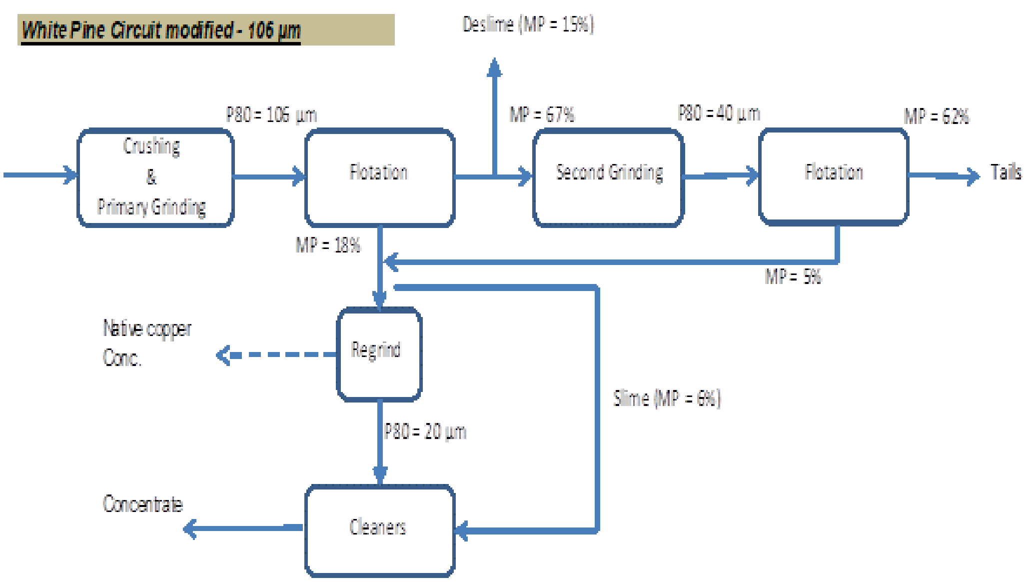
Table 13.22: Flotation Testing

Sample	Head Grade		Primary	Regrind	Open circuit Performances						Expected Closed Circuit	
	Cu	Ag	P_{80}			MP	Copper		Silver		Copper	
	%	g/t	µm			%	% cu	% Rec	g/t	% Rec	% cu	% Rec
WPM-001	0.77	17.5	45.7		Ro conc	35.0	2.1	93.8	48.2	96.3		
				17.0	Final conc	2.1	31.4	87.5	749.6	91.5	30	91
WPM-001	0.80	16.8	36.3		Ro conc	35.0	2.2	92.1	46.1	96.1		
				15.9	Final conc	2.5	28.4	85.8	617.8	91.6	27	89
WPM-013	1.10	38.3	49.0		Ro conc	42.9	2.4	93.2	86.7	97.0		
				18.1	Final conc	3.4	28.3	86.0	984.1	86.3	27	90
WPC-M13-14-16-17	0.95	14.3	70.0		Ro conc	41.8	2.1	92.1	33.7	95.9		
				26.0	Final conc	2.9	27.9	85.1	452.1	91.5	27	89
WPC-M13-14-16-17	0.91	16.0	54.0		Ro conc	42.3	2.1	93.7	36.4	92.8		
				16.5	Final conc	3.0	26.6	86.5	472.1	87.5	26	90
WPC-M13-14-16-17	0.92		40.1		Ro conc	39.4	2.2	93.4	35.5	92.0		
		15.2		15.5	Final conc	2.4	32.3	84.9	535.5	85.3	31	89

*** Assuming 50% of the differences between Ro & Final open recovery will be added to the final close circuit recovery without losing the grade (common industrial correlations) at the cost of 1% Grade loss.*

It was concluded that further optimization focusing on reagent consumption was needed.

Figure 13.4: White Pine Circuit – 106 um



- Cu Recovery : 88%;
- Copper Conc. Grade : 30%;
- Flotation reagent cost excluding lime & flocculants : <1.5 $/t .

Figure 13.5: White Pine Circuit Modified – 104 µm

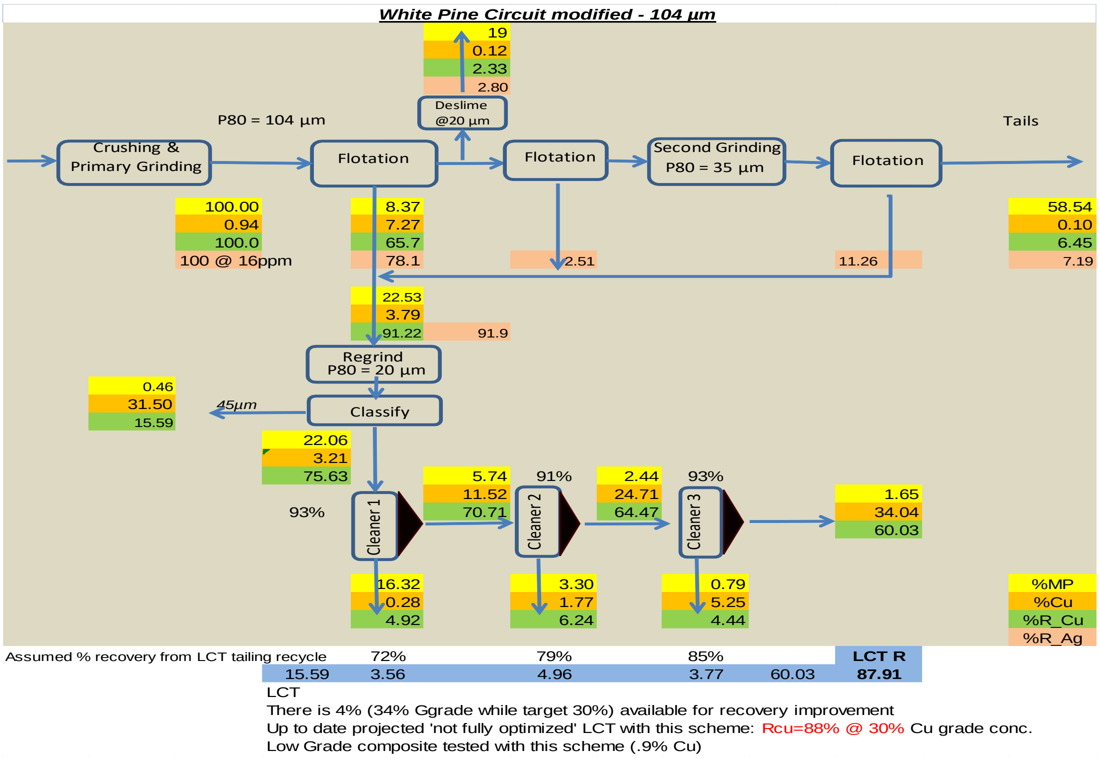


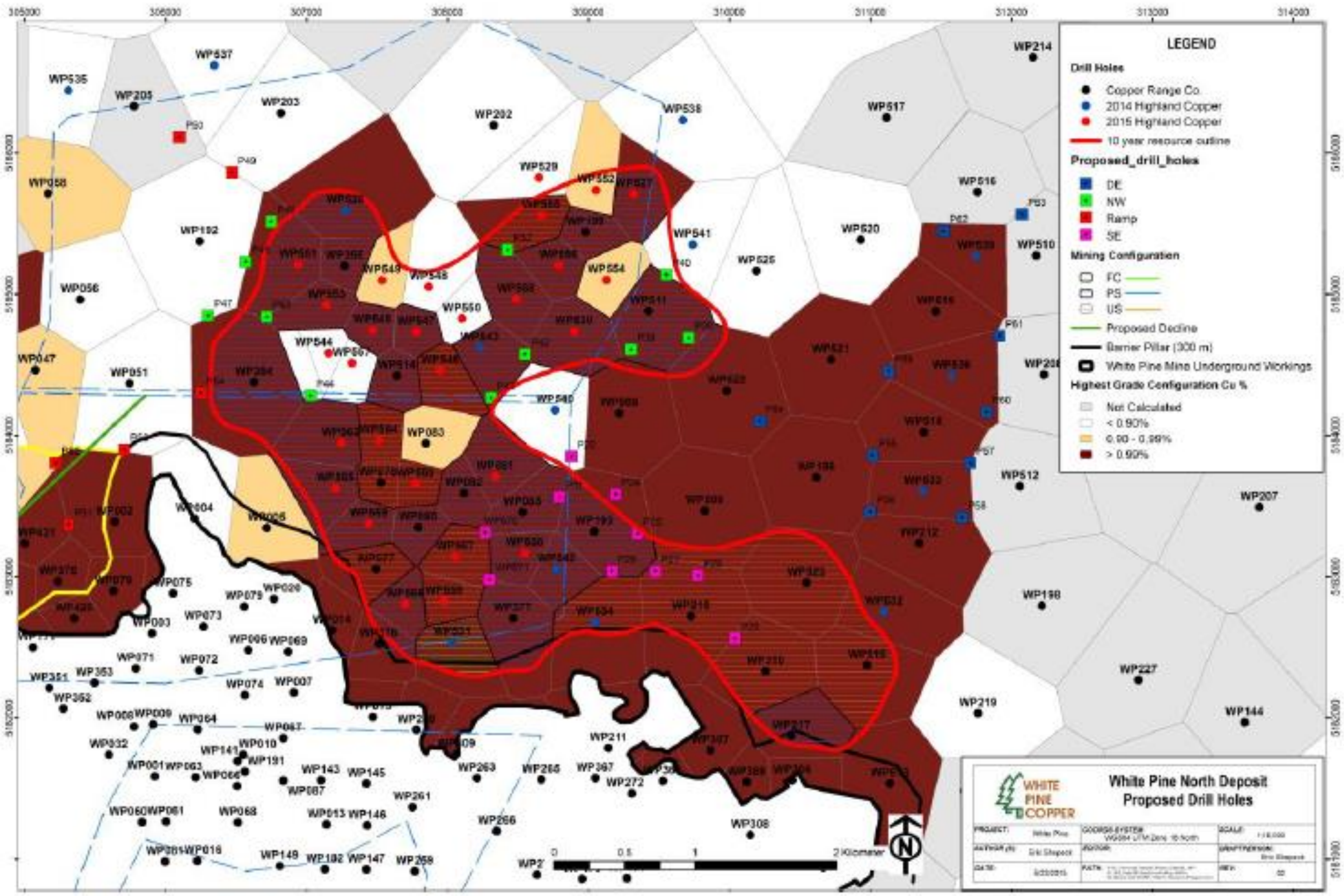
Table 13.23: Reagents Cost 104 µm

	Cost Base		WP51-FOT10			Cytec	Open	Closed
	CW2012	CW2015	CW2015			2015	Circuit	
	$/Kg			g/t	$/t	$/kg	$/t	
KAX	2.4	2.62	2.62	160.0	0.42	4.41	0.71	0.64
MIBC	6.39	6.98	6.98	55.0	0.38	3.96	0.22	0.22
D-250	2.15	2.35	2.35	110.0	0.26	2.35	0.26	0.23
NDM	5.4	5.90	5.90	28.0	0.17	5.90	0.17	0.15
A-249	2.76	3.02	3.02	44.0	0.13	3.3	0.15	0.13
NaHS	1.93	2.11	2.11	0.0	0.00	1.65	0.00	0.00
SS	1.27	1.39	1.39	260.0	0.36	0.62	0.16	0.15
CMC	1.1	1.20	1.20		0.00	3.3	0.00	0.00
Lime								
	Projected WP				1.72	US $/MT	1.65	1.51

13.5 Pending Testing Programs

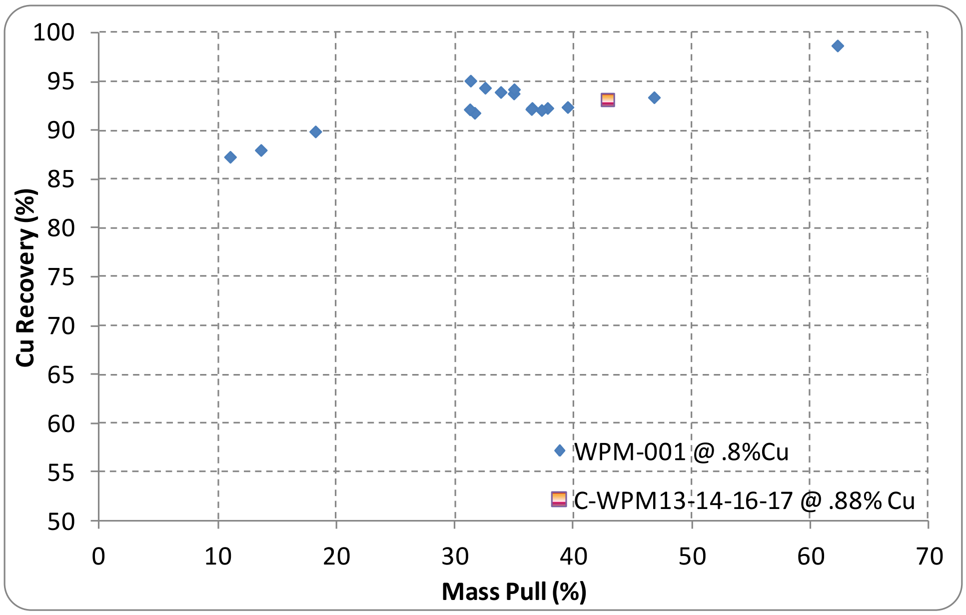
- New drill core samples More representative of the first 5-10 years (depending on processing rate of WPN);
- Around 500 kg of PS material;
- Will be used in confirmatory and variability testing;
- Scheme revisiting for adequation with the integrated project;
- Continue optimisation towards FS (using the balance of samples);
- Extract some engineering parameters (tailings, water, Conc., etc.) (pending).

Figure 13.6: White Pine North Deposit Proposed Drill Holes



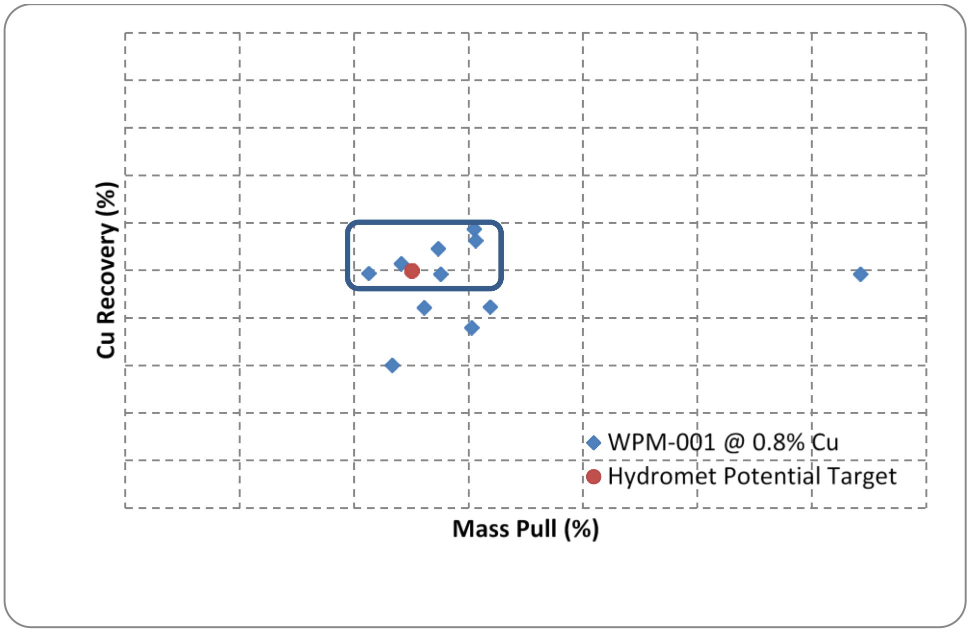
Preliminary tests using PS mineralization samples and looking for rougher flotation conditions and reagents screening indicated that rougher mass pull of 35% with up to 95% Cu recovery could be achievable (Figure 13.7), with the run-of-mine ("ROM") primarily ground to P80 = 50-60 microns.

Figure 13.7: Rougher Mass Pull –Recovery Curves



Tests results (Figure 13.8) show an interesting possibility of making hydrometallurgy suitable rougher concentrate of less than 5% mass pull with around 75% recovery in the first 3-4 flotation minutes. In that case, the remaining rougher concentrate completing the 35% target mass pull could be further reground and cleaned to make saleable concentrate or be added to the hydromet feed making the overall mass pull less than 8% and the overall copper recovery higher than 90%. This option will be studied as an opportunity for further testing/assessment at the Feasibility Study stage.

Figure 13.8: Rougher Mass Pull / Recovery Curves Opportunity



Preliminary open tests show that an FL2 flotation circuit could deliver interesting metallurgical performances. As indicated in Table 13.24, saleable concentrate grading up to 30% Cu could be obtained while recovering up to 90% of the copper in a LCT based projection. The selected reagent recipe was proven very suitable for silver recovery as well, indicating recoveries greater than 90%.

Table 13.24: Preliminary FL2 Open Circuit Outcomes

Sample	Head Grade		Primary	Regrind	Open circuit Performances					
	Cu	Ag	P80			MP	Copper		Silver	
	%	g/t	µm			%	% cu	% Rec	g/t	% Rec
WPM-001	0.77	17.5	45.7		Ro conc	35.0	2.1	93.8	48.2	96.3
				17.0	Final conc	2.1	31.4	87.5	749.6	91.5
WPM-001	0.80	16.8	36.3		Ro conc	35.0	2.2	92.1	46.1	96.1
				15.9	Final conc	2.5	28.4	85.8	617.8	91.6
WPM-013	1.10	38.3	49.0		Ro conc	42.9	2.4	93.2	86.7	97.0
				18.1	Final conc	3.4	28.3	86.0	984.1	86.3
WPC-M13-14-16-17	0.95	14.3	70.0		Ro conc	41.8	2.1	92.1	33.7	95.9
				26.0	Final conc	2.9	27.9	85.1	452.1	91.5
WPC-M13-14-16-17	0.91	16.0	54.0		Ro conc	42.3	2.1	93.7	36.4	92.8
				16.5	Final conc	3.0	26.6	86.5	472.1	87.5
WPC-M13-14-16-17	0.92		40.1		Ro conc	39.4	2.2	93.4	35.5	92.0
		15.2		15.5	Final conc	2.4	32.3	84.9	535.5	85.3
WPC-M13-14-16-17	0.91	16.0	105.0	20.0	slime	21.3	0.1	3.2		
				40.8	Ro tails	54.4	0.09	5.1		
					Ro conc	24.3	3.4	91.6		
				20.2	Clnr 2 conc	2.3	29.4	75.3		

The last test reported was repeated by further optimization and through extracting native copper from the cleaner feed to avoid losses in cleaner tailings. Results should be improved since the rougher mass pull is the lowest among all other tests. Lower mass pull will help avoid slime build-up and its negative effects.

Looking for a mid-size capacity (15-20 ktpd) processing plant, the comminution circuit optimization will be needed. Partial comminution tests (Table 13.25 and Table 13.26) suggested that the ore could be classified as soft to moderately hard (PS_SMC: Ab= 38.2).

Table 13.25: Comminution Testing Results

Sample	Litho/ Config.	Test	Value
WPM003	P. Shale	RMWI	15.9 kWh/t
WPM004	Full Column	RMWI	14.8 kWh/t
WPM002	Full Column	BMWI	14.5 kWh/t
WPM005	P. Shale	BMWI	13.9 kWh/t
WPM008	UPSA	RMWI	14.2 kWh/t
WPM009	UPSA	BMWI	14.1 kWh/t
WPM011	Full Column	SMC	Tableau bas
WPM018	Full Column	SMC	Tableau bas
WPM019	Upper Shale	SMC	Tableau bas
WPM029	UPSA	CWI	9.4 kwh/t
WPM030	P. Shale	CWI	12.2 kwh/t
WPM031	P. Shale	SMC	Tableau bas

Table 13.26: Comminution Testing Results- SMC

										particle size (mm)								
										14.5	14.5	14.5	28.9	28.9	28.9	57.8	57.8	57.8
										t10								
ID	DWi	DWi	M_{ia}	M_{ih}	M_{ic}	A	b	sg	ta	10	20	30	10	20	30	10	20	30
	kWh/m^3	%	kWh/t	kWh/t	kWh/t					kWh/t	kWh/t	kWh/t	kWh/t	kWh/t	kWh/t	kWh/t	kWh/t	kWh/t
WMU006 (WPM031)	7.20	70	20.1	15.1	7.8	60.7	0.63	2.76	0.36	0.36	0.76	1.21	0.27	0.57	0.92	0.21	0.43	0.69
WPM011	8.39	81	22.9	17.7	9.1	67.4	0.49	2.75	0.31	0.42	0.89	1.42	0.32	0.67	1.07	0.24	0.51	0.81
WPM018	7.32	71	20.6	15.5	8.0	63.7	0.59	2.73	0.35	0.37	0.78	1.24	0.28	0.59	0.94	0.21	0.45	0.71
WPM019	8.04	78	22.1	17.0	8.8	62.3	0.55	2.75	0.32	0.40	0.85	1.36	0.30	0.64	1.03	0.23	0.49	0.78

Table 13.27: Copper Distribution in WPM533P

WPM533P - Wt%				
Size fraction	+75 µm	-75 +38 µm	-38 µm	Total
Chalcocite	76.6	68.5	81.9	76.6
Native_Copper	19.8	27.2	8.4	19.4
Bornite	3.6	4.3	9.6	4.0
Chalcopyrite	0.0	0.0	0.1	0.0
Total	100.0	100.0	100.0	100.0

Liberation classes are defined as cumulative; 100% free of the native copper and the chalcocite. Table 13.20 and Table 13.21 show the liberation classes (cumulative from 100% free) of the native copper and the chalcocite. Size distribution of the native copper and chalcocite (expressed as p-value), shows that ~20 % of the mineral is smaller than 22 µm for native copper and 8 µm for chalcocite, suggesting that fine regrind less than 20 microns is recommended for +30% Cu concentrate grade target with acceptable Cu recovery.

13.5.1 Flotation Testing

Results from the open circuit bench scale flotation tests are presented in Table 13.22.

Table 13.28: Flotation Testing

Sample	Head Grade		Primary	Regrind	Open circuit Performances						Expected Closed Circuit	
	Cu	Ag	P_{80}			MP	Copper		Silver		Copper	
	%	g/t	µm			%	% cu	% Rec	g/t	% Rec	% cu	% Rec
WPM-001	0.77	17.5	45.7		Ro conc	35.0	2.1	93.8	48.2	96.3		
				17.0	Final conc	2.1	31.4	87.5	749.6	91.5	30	91
WPM-001	0.80	16.8	36.3		Ro conc	35.0	2.2	92.1	46.1	96.1		
				15.9	Final conc	2.5	28.4	85.8	617.8	91.6	27	89
WPM-013	1.10	38.3	49.0		Ro conc	42.9	2.4	93.2	86.7	97.0		
				18.1	Final conc	3.4	28.3	86.0	984.1	86.3	27	90
WPC-M13-14-16-17	0.95	14.3	70.0		Ro conc	41.8	2.1	92.1	33.7	95.9		
				26.0	Final conc	2.9	27.9	85.1	452.1	91.5	27	89
WPC-M13-14-16-17	0.91	16.0	54.0		Ro conc	42.3	2.1	93.7	36.4	92.8		
				16.5	Final conc	3.0	26.6	86.5	472.1	87.5	26	90
WPC-M13-14-16-17	0.92		40.1		Ro conc	39.4	2.2	93.4	35.5	92.0		
		15.2		15.5	Final conc	2.4	32.3	84.9	535.5	85.3	31	89

*** Assuming 50% of the differences between Ro & Final open recovery will be added to the final close circuit recovery without losing the grade (common industrial correlations) at the cost of 1% Grade loss.*

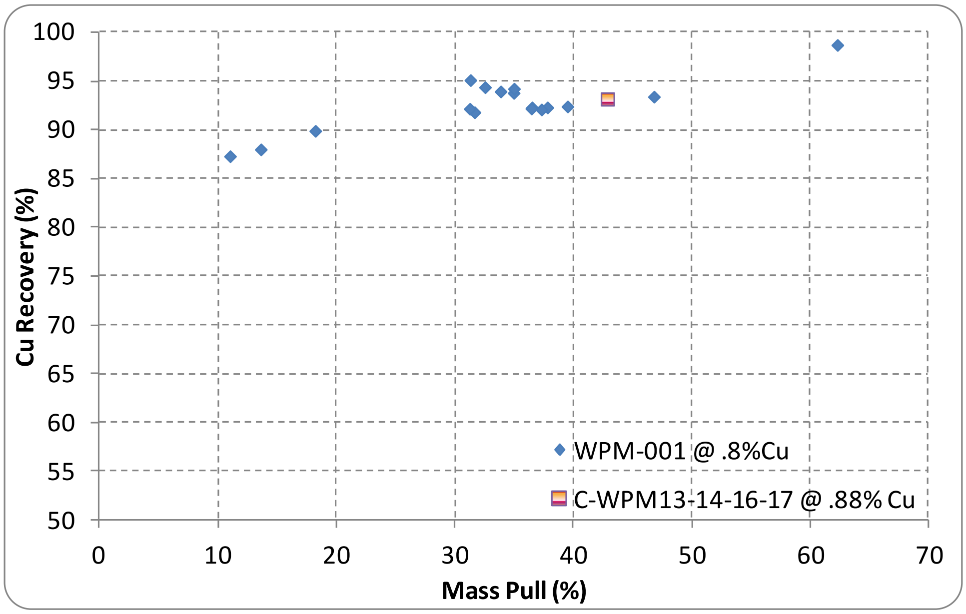
It was concluded that further investigations focusing on reagents consumption were needed.

13.6 Pending Testing Programs

- New drill core samples more representative of the first 5-10 years (depending on processing rate of WPN);
- Approximately 500 kg of PS material;
- Will be used in confirmatory and variability testing;
- Scheme revisiting for adequation with the integrated project;
- Continue optimization towards FS (using the balance of samples);
- Extract some engineering parameters (tailings, water, conc., etc.); (pending).

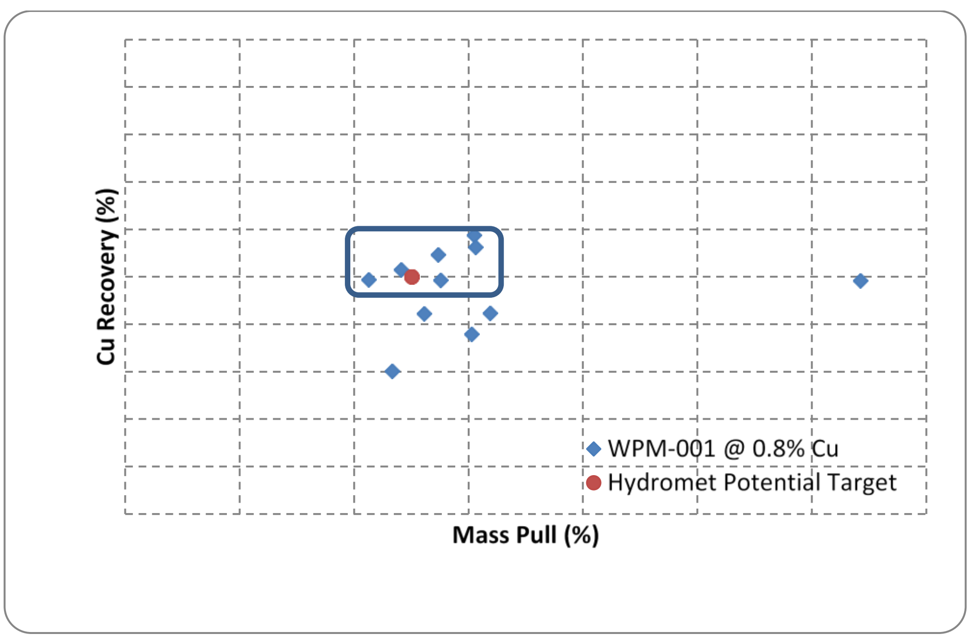
Preliminary tests using PS mineralization samples and looking for rougher flotation conditions and reagents screening indicated that rougher mass pull of 35% with up to 95% Cu recovery could be achievable (Figure 13.7), with the run-of-mine ("ROM") primarily ground to P80 = 50-60 microns.

Figure 13.9: Rougher Mass Pull –Recovery Curves



Tests results (Figure 13.8), shows an interesting opportunity of making hydrometallurgy suitable rougher concentrate of less than 5% mass pull with around 75% recovery in the first 3-4 flotation minutes. In that case the remaining rougher concentrate completing the 35% target mass pull could be further reground and cleaned to make saleable concentrate or to be added to the hydromet feed making the overall mass pull less than 8% and the overall copper recovery higher than 90%. The complete feature of this opportunity will be looked at as an opportunity for further testing/assessment in the Feasibility Study.

Figure 13.10: Rougher Mass Pull / Recovery Curves opportunity



Preliminary open tests show that an FL2 flotation circuit could deliver interesting metallurgical performances. As indicated in Table 13.24 saleable concentrate grading up to 30% Cu could be obtained while recovering up to 90% of the copper in a LCT based projection. The selected reagent list was proven very suitable for silver recovery too indicating recoveries greater than 90%.

Table 13.29: Preliminary FL2 Open Circuit Outcomes

Sample	Head Grade		Primary	Regrind	Open circuit Performances					
	Cu	Ag	P_{80}			MP	Copper		Silver	
	%	g/t	µm			%	% cu	% Rec	g/t	% Rec
WPM-001	0.77	17.5	45.7		Ro conc	35.0	2.1	93.8	48.2	96.3
				17.0	Final conc	2.1	31.4	87.5	749.6	91.5
WPM-001	0.80	16.8	36.3		Ro conc	35.0	2.2	92.1	46.1	96.1
				15.9	Final conc	2.5	28.4	85.8	617.8	91.6
WPM-013	1.10	38.3	49.0		Ro conc	42.9	2.4	93.2	86.7	97.0
				18.1	Final conc	3.4	28.3	86.0	984.1	86.3
WPC-M13-14-16-17	0.95	14.3	70.0		Ro conc	41.8	2.1	92.1	33.7	95.9
				26.0	Final conc	2.9	27.9	85.1	452.1	91.5
WPC-M13-14-16-17	0.91	16.0	54.0		Ro conc	42.3	2.1	93.7	36.4	92.8
				16.5	Final conc	3.0	26.6	86.5	472.1	87.5
WPC-M13-14-16-17	0.92		40.1		Ro conc	39.4	2.2	93.4	35.5	92.0
		15.2		15.5	Final conc	2.4	32.3	84.9	535.5	85.3
WPC-M13-14-16-17	0.91	16.0	105.0	20.0	slime	21.3	0.1	3.2		
				40.8	Ro tails	54.4	0.09	5.1		
					Ro conc	24.3	3.4	91.6		
				20.2	Clnr 2 conc	2.3	29.4	75.3		

The last test reported was repeated by further optimization and through extracting of native copper from the cleaner feed to avoid losses in cleaner tailings. Results should be improved since the rougher mass pull is the lowest among all other tests. Lower mass pull will help to avoid slime build up and their negative effect.

Looking for a mid-size capacity (15-20 ktpd) processing plant, the comminution circuit optimization will be needed. Partial comminution tests (Table 13.25 and Table 13.26) suggested that the ore could be classified as soft to moderately hard (PS_SMC: Ab= 38.2).

Table 13.30: Comminution Testing Results

Sample	Litho/ Config.	Test	Value
WPM003	P. Shale	RMWI	15.9 kWh/t
WPM004	Full Column	RMWI	14.8 kWh/t
WPM002	Full Column	BMWI	14.5 kWh/t
WPM005	P. Shale	BMWI	13.9 kWh/t
WPM008	UPSA	RMWI	14.2 kWh/t
WPM009	UPSA	BMWI	14.1 kWh/t
WPM011	Full Column	SMC	Tableau bas
WPM018	Full Column	SMC	Tableau bas
WPM019	Upper Shale	SMC	Tableau bas
WPM029	UPSA	CWI	9.4 kwh/t
WPM030	P. Shale	CWI	12.2 kwh/t
WPM031	P. Shale	SMC	Tableau bas

Table 13.31: Comminution Testing Results- SMC

										particle size (mm)								
										14.5	14.5	14.5	28.9	28.9	28.9	57.8	57.8	57.8
										t10								
ID	DWi	DWi	M_{ia}	M_{ih}	M_{ic}	A	b	sg	ta	10	20	30	10	20	30	10	20	30
	kWh/m³	%	kWh/t	kWh/t	kWh/t					kWh/t	kWh/t	kWh/t	kWh/t	kWh/t	kWh/t	kWh/t	kWh/t	kWh/t
WMU006 (WPM031)	7.20	70	20.1	15.1	7.8	60.7	0.63	2.76	0.36	0.36	0.76	1.21	0.27	0.57	0.92	0.21	0.43	0.69
WPM011	8.39	81	22.9	17.7	9.1	67.4	0.49	2.75	0.31	0.42	0.89	1.42	0.32	0.67	1.07	0.24	0.51	0.81
WPM018	7.32	71	20.6	15.5	8.0	63.7	0.59	2.73	0.35	0.37	0.78	1.24	0.28	0.59	0.94	0.21	0.45	0.71
WPM019	8.04	78	22.1	17.0	8.8	62.3	0.55	2.75	0.32	0.40	0.85	1.36	0.30	0.64	1.03	0.23	0.49	0.78

Looking for a mid-size capacity (15-20 ktpd) processing plant, the comminution circuit optimization will be needed. Partial comminution tests (Table 13.25 and Table 13.26) suggested that the ore could be classified as soft to moderately hard (PS_SMC: Ab= 38.2).

As the development of White Pine is moving forward, these 2013/14 tests will need to be confirmed with new samples and testworks.

14. MINERAL RESOURCE ESTIMATES

G Mining Services Inc. (“GMSI”) has prepared a preliminary Mineral Resource estimate (“MRE”) for the White Pine North Project (Figure 14.1) based on data generated up to March 2015. The main objective of this assessment is to produce a Mineral Resource for the White Pine North sector. Resource estimation methodologies, results and validations are presented in this section of the Technical Report.

In the opinion of GMSI, the MRE reported herein is a reasonable representation of the global Mineral Resources found in the North-East Sector at the current level of sampling.

The MRE was prepared under the supervision of Mr. Réjean Sirois, Eng. GMSI, Vice President Geology and Resources, an independent “Qualified Person” (“QP”) as defined in in National Instruments NI 43-101 (“NI 43-101) *Canadian Standards of Disclosure* . Geovia GEMS™ and Leapfrog Geo™ software were used to facilitate the Resource estimation process.

The Mineral Resource Estimate includes Inferred Mineral Resources that are normally considered too speculative geologically to have economic considerations applied to them, which would enable them to be categorized as Mineral Reserves. There is also no certainty that these Inferred Mineral Resources will be converted to the Indicated and Measured categories through further drilling, or into Mineral Reserves, once economic considerations are applied.

14.1 Data

The database used in this Technical Report consists of diamond drilling sampling data intersecting the mineralized stratigraphic horizons. GMSI received the drill hole database in the form of CSV files from Highland Copper.

The current Resource estimate is derived exclusively from the database described Section 12. GMSI reviewed the database and is satisfied with the integrity of the drilling database and that it can be used for Resource estimation. Some minor errors were found in the survey table, but these were corrected before Resource modelling and are discussed in Section 12

14.1.1 Drill Hole Spacing

The surface drill hole grid spacing is around 300 meters (“m”) in the mined-out area (the historical White Pine Mine), and roughly 700-800 m in the North-East Sector (Figure 14.1). The drill spacing and distribution is judged adequate to develop a reasonable model of the mineralization distribution, and to quantify its volume and quality with an acceptable level of confidence. Figure 14.2 illustrates a 3D view of drill hole spacing for the White Pine North Project.

Figure 14.1: Plan View of Drill Hole Collars – White Pine North Project

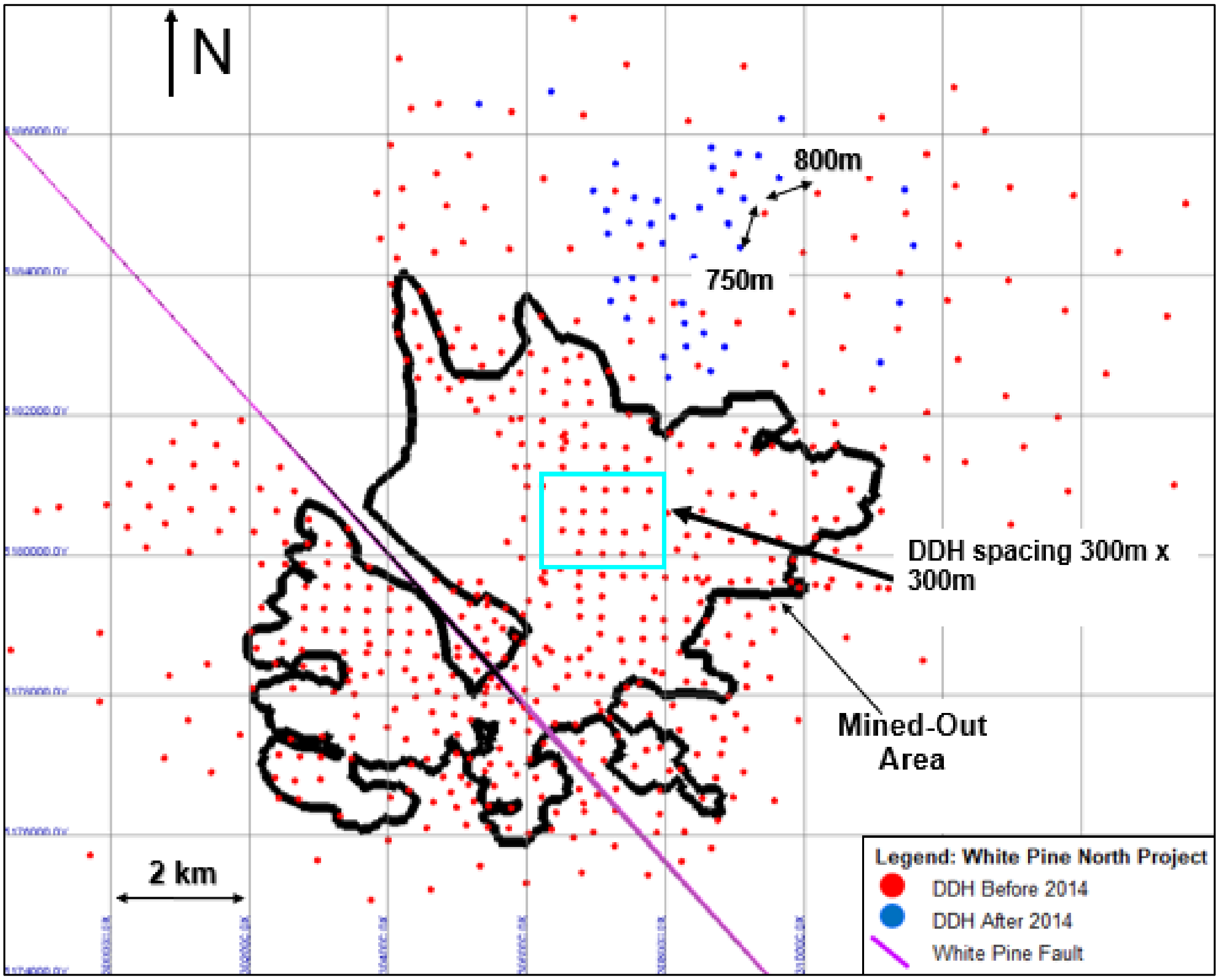
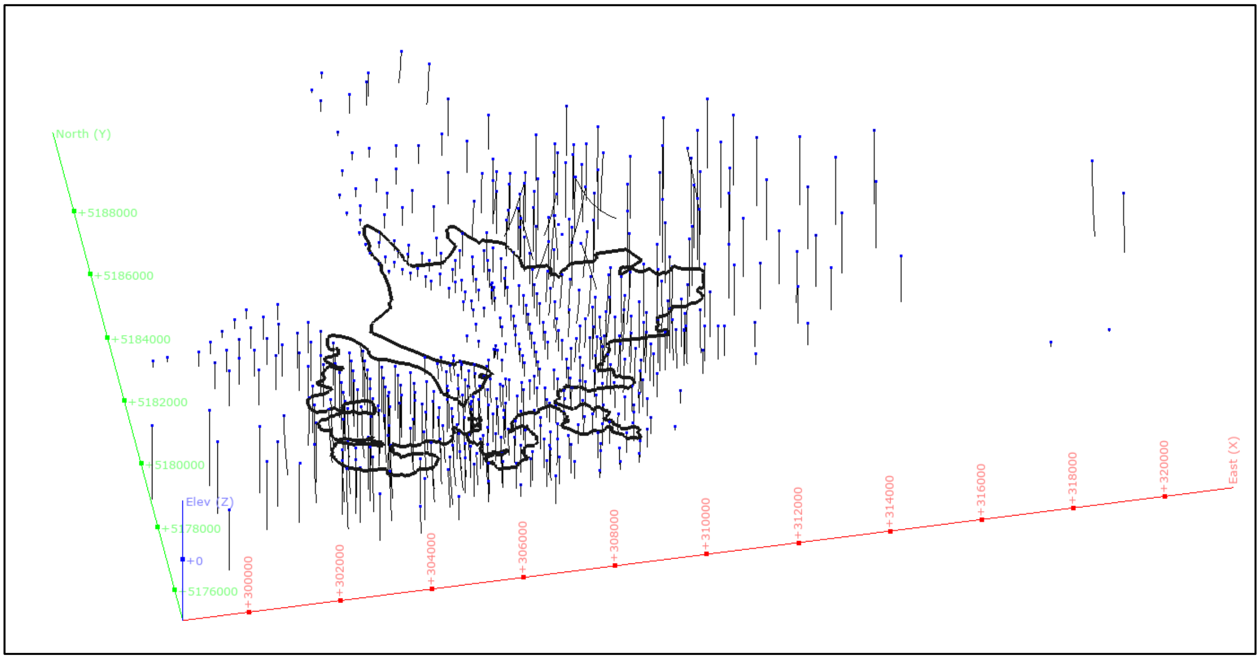


Figure 14.2: 3D View of Drill Holes and Mined-Out Area, View Towards North-East



14.2 Modelling Approach

The 3D geological modelling performed for the Resource estimate was produced by GMSI based on the drill hole database and historical information about the mineralized columns. The modelling of the mineralized zones was carried out by using the 3D geological modelling software Leapfrog Geo™ v.4.5 (“Leapfrog”). The solids were then transferred into Geovia GEMS™ v.6.8.2.2 (“GEMS”) software for block modeling.

14.2.1 Mineralization Column Modelling

The modelling of the copper mineralization horizons was based on the footwall and hanging wall of the three selected columns to model, namely the Parting Shale (“PS”), the Full Column (“FC”) and the Upper Shale (“US”) (Figure 14.3). These intervals will be referred as “Geological Intervals” (PS-GEO, FC-GEO and US-GEO).

In some areas, the total true thickness of the PS, the FC and/or US is less than the minimum height required for underground mining, a minimum true thickness of 2.0 m was applied to all intervals of each column and stored separately. Those intervals will be referred as the “Mining Intervals” (PS-MINING, FC-MINING and US-MINING) and are identical to the Geological Intervals where the minimum height of 2.0 m is reached. The hanging wall contact remained unchanged, and only the footwall contact was adjusted lower to arrive at a minimum true thickness of 2.0 m. The true thickness of each intercept was calculated mathematically using the dip and dip direction of the stratigraphy, and the angle of the drilling.

In both GEO and MINING cases, the hanging wall of the PS is the base of the Upper Sandstone (B30) or the top of the Tiger (B29), whereas the hanging wall of the FC was set at the base of the Brown Massive (B44) or the top of the Thinly (B43). The latter was done to reflect historical mining reaching up to the dimpled back (calcareous nodules in the Brown Massive). As for the US, the hanging wall was set at the top of the Widely unit (B47). In most cases, the footwalls of the PS and the FC were set to the base of the Lower Transition (B21) but extended down to the top of the Copper Harbor Formation (B10) where the height of the column was less than 2.0 m (PS-MINING and FC-MINING). In regard to the US, the footwall was set to the base of the Upper Transition formation (B41).

Figure 14.3: Mineralized Columns at the White Pine Deposit

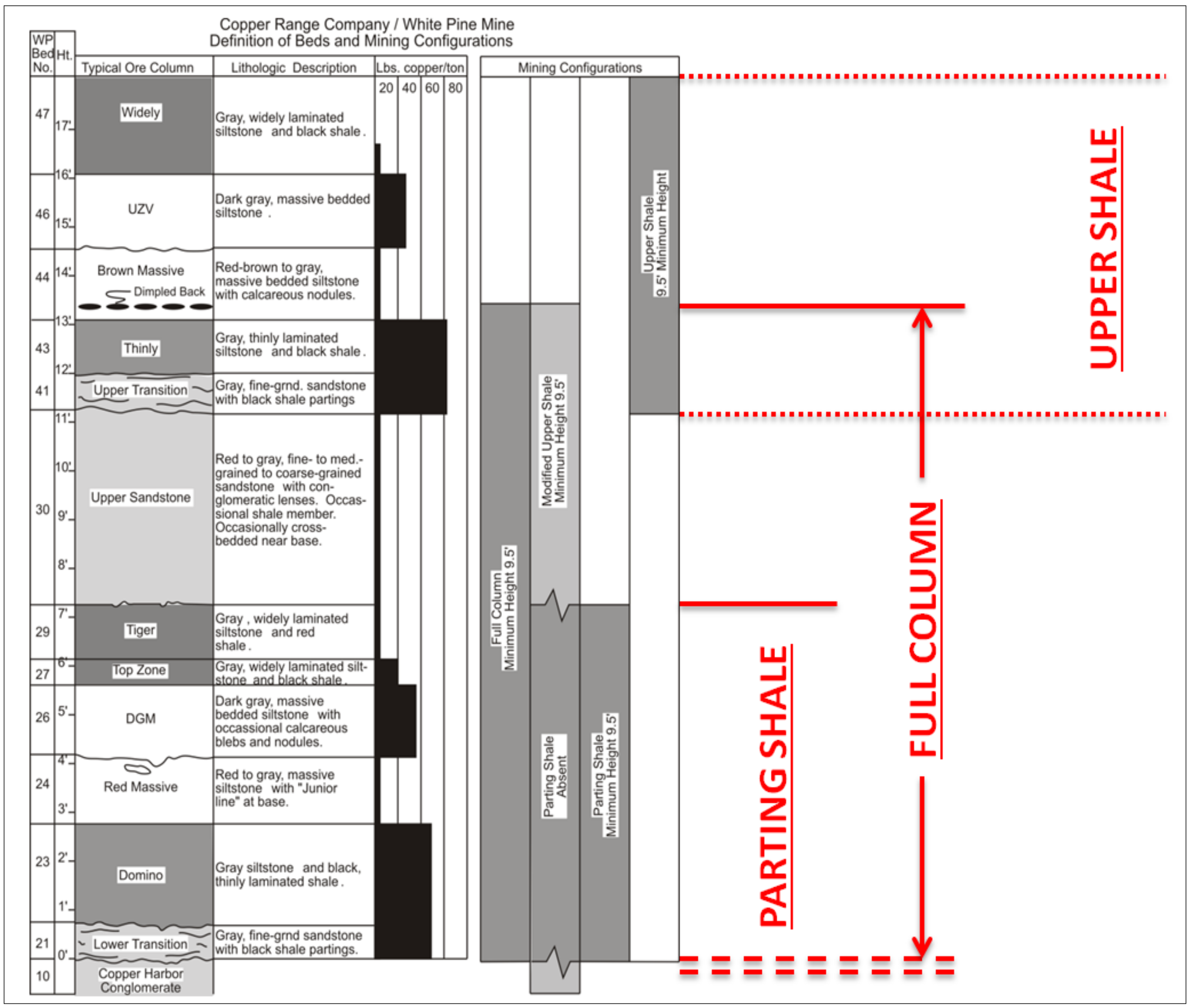
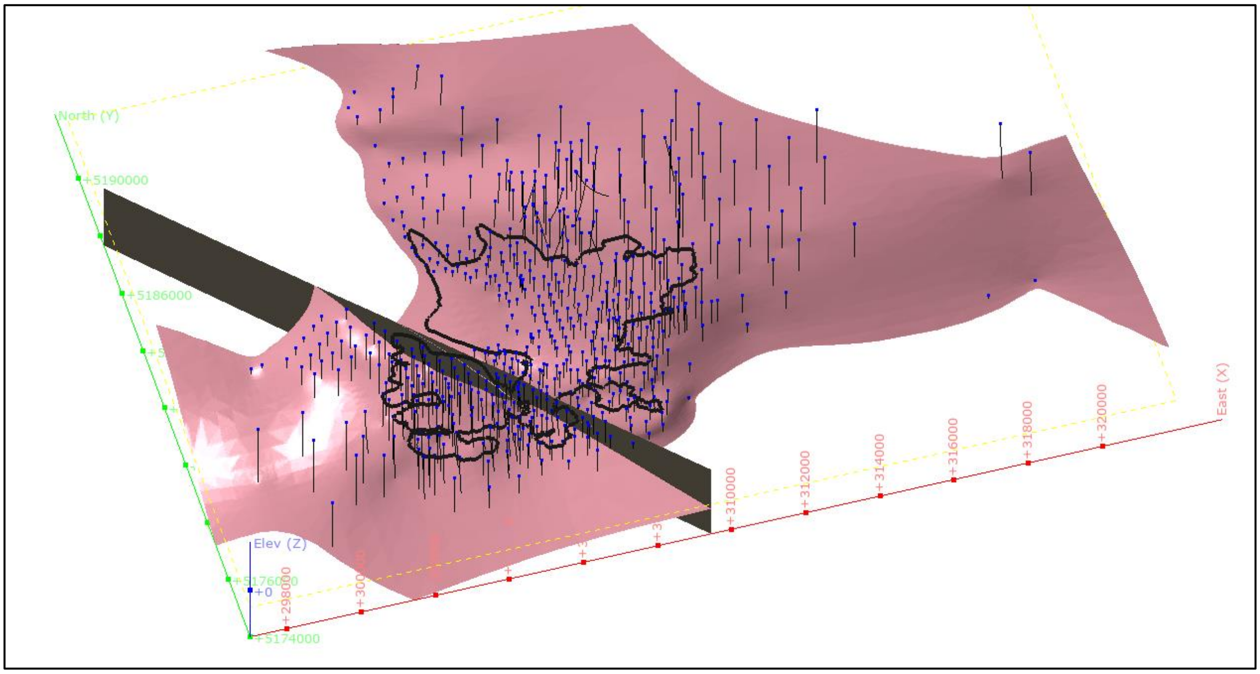


Figure 14.4 shows a view of the PS, modelled in Leapfrog GEO™.

Figure 14.4: Parting Shale and White Pine Fault – Leapfrog 3D Model



14.2.2 Additional Datasets

Other than drill holes, only the historical mine workings outline was directly imported in the GEMS database. GMSI also received the White Pine fault trace, which was extrapolated in Leapfrog and imported into GEMS. The White Pine fault is shown in Figure 14.4 as the dark brown vertical surface.

14.3 Statistical Analysis

14.3.1 Statistics of Original Assays

Statistical analyses were conducted using the assays available in the drilling database (including the mined-out area). Summary of the statistical analysis for all beds up to B47 inclusively are presented in Table 14.1 and Table 14.2, for copper and silver respectively. The silver dataset was trimmed to ease visualization of grade distribution (high coefficient of correlation before trimming), where only grades greater than 0.1 g Ag/t were kept to produce the tabulations. Highlighted beds (in yellow) are those containing the bulk of copper mineralization.

Table 14.1: Summary Statistics by Bed – All Drill Holes (Copper)

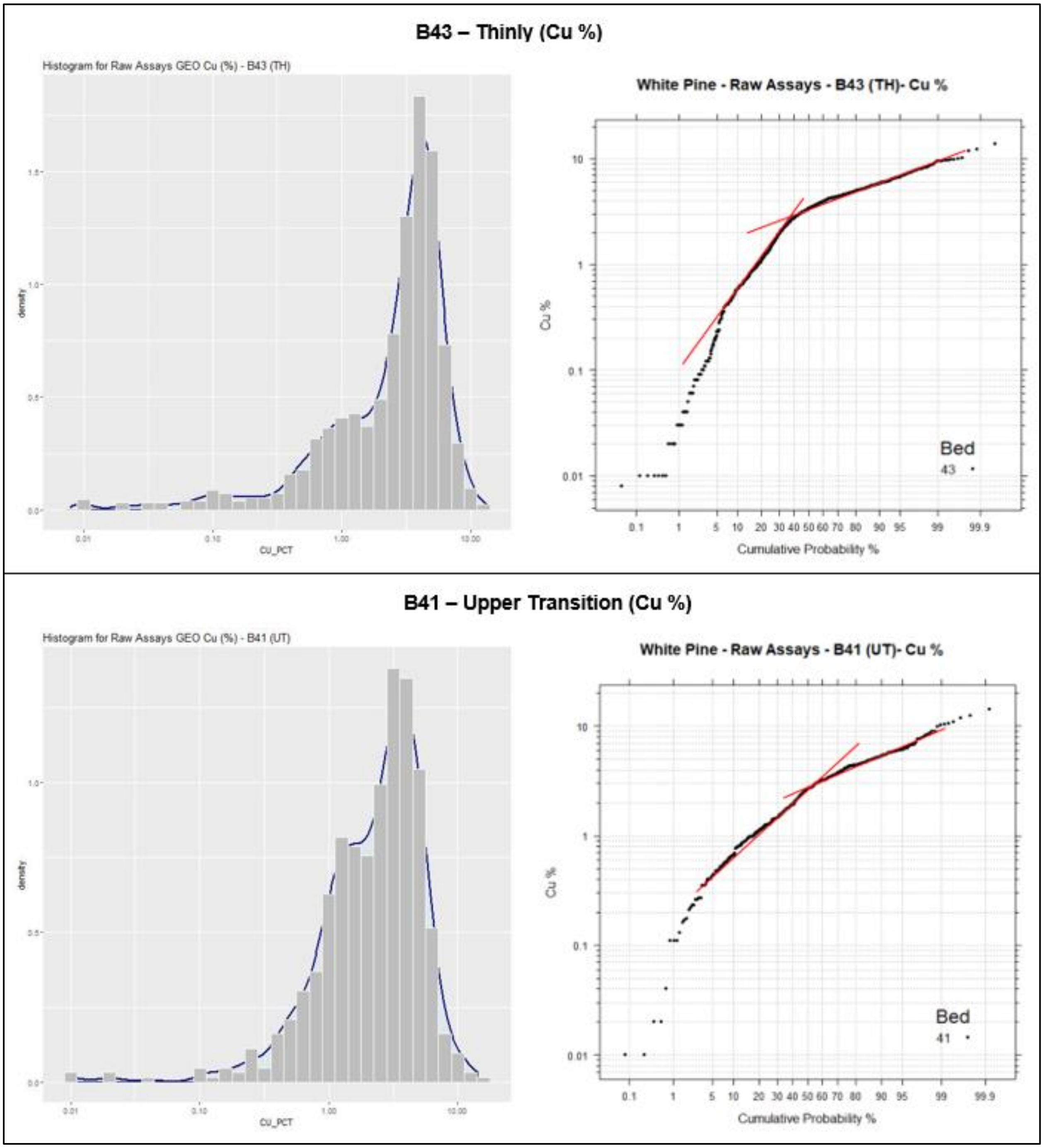
Beds	Number of Assays	Average Thickness m	Grade			Standard Deviation	CoV
			Min.	Max.	Mean		
			Cu %	Cu %	Cu %	Cu %	
B47 - Widely	1,716	1.00	0.00	2.85	0.20	0.35	1.79
B46 - UZV	1,565	0.85	0.00	4.32	0.63	0.65	1.04
B44 - Brown Massive	1,297	0.68	0.00	4.17	0.14	0.24	1.80
B43 - Thinly	1,253	0.40	0.00	13.78	3.26	2.11	0.65
B41 - Upper Transition	628	0.18	0.00	14.25	2.86	2.04	0.71
B30 - Upper Sandstone	1,645	1.45	0.00	4.71	0.20	0.25	1.25
B29 - Tiger	463	0.46	0.00	2.56	0.23	0.22	0.98
B27 - Top Zone	366	0.42	0.00	3.20	0.76	0.48	0.63
B26 - Dark Gray Massive	570	0.46	0.00	5.49	2.05	1.16	0.57
B24 - Red Massive	795	0.50	0.00	2.44	0.24	0.21	0.87
B23 - Domino	623	0.21	0.03	10.47	2.38	1.85	0.78
B21 - Lower Transition	1,148	0.66	0.03	14.92	1.30	1.38	1.07
B10 - Copper Harbor Congl.	2,427	1.66	0.00	13.45	0.28	0.96	3.39

Table 14.2: Summary Statistics by Bed – All Drill Holes (Silver)

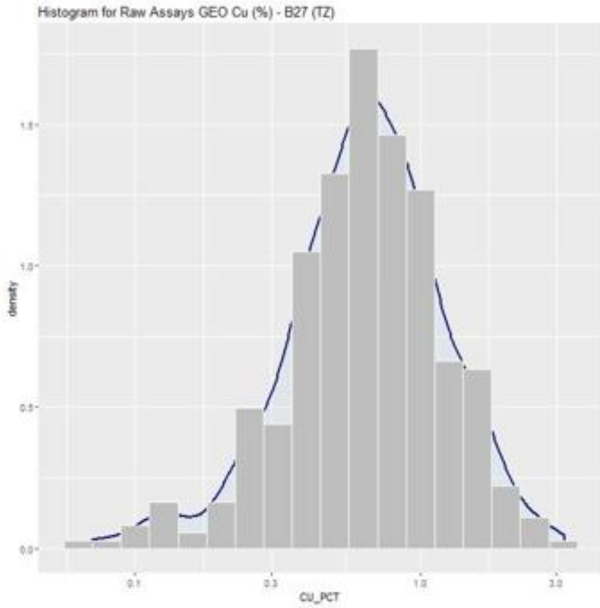
Beds	Dataset	Number of Assays	Grade Min.	Grade Max.	Grade Mean	Standard Deviation	CoV
			g Ag/t	g Ag/t	g Ag/t	g Ag/t	
B47 - Widely	No trimming	1,716	0.10	64.40	0.78	3.05	3.91
	Only >0.1 g Ag/t	283	0.30	64.40	4.22	6.50	1.54
B46 - UZV	No trimming	1,565	0.10	63.80	3.59	6.00	1.67
	Only >0.1 g Ag/t	768	0.30	63.80	7.20	6.90	0.96
B44 - Brown Massive	No trimming	1,297	0.10	64.40	0.67	3.48	5.21
	Only >0.1 g Ag/t	184	0.30	64.40	4.11	8.48	2.07
B43 - Thinly	No trimming	1,253	0.10	276.30	14.67	23.79	1.62
	Only >0.1 g Ag/t	1,099	0.30	276.30	16.71	24.73	1.48
B41 - Upper Transition	No trimming	628	0.10	240.00	10.86	16.42	1.51
	Only >0.1 g Ag/t	539	0.30	240.00	12.64	17.08	1.35
B30 - Upper Sandstone	No trimming	1,645	0.10	33.90	0.53	1.83	3.45
	Only >0.1 g Ag/t	335	0.30	33.90	2.22	3.60	1.62
B29 - Tiger	No trimming	463	0.10	29.50	0.62	2.29	3.69
	Only >0.1 g Ag/t	90	0.30	29.50	2.79	4.63	1.66
B27 - Top Zone	No trimming	366	0.10	72.70	2.32	5.30	2.29
	Only >0.1 g Ag/t	184	0.30	72.70	4.51	6.81	1.51
B26 - Dark Gray Massive	No trimming	570	0.10	144.00	11.80	15.46	1.31
	Only >0.1 g Ag/t	455	0.30	144.00	14.76	16.01	1.08
B24 - Red Massive	No trimming	795	0.10	241.30	1.27	9.65	7.62
	Only >0.1 g Ag/t	138	0.30	241.30	6.82	22.41	3.29
B23 - Domino	No trimming	623	0.10	1,327.70	68.47	126.26	1.84
	Only >0.1 g Ag/t	525	0.30	1,327.70	81.23	133.74	1.65
B21 - Lower Transition	No trimming	1,148	0.10	1,460.10	20.84	79.55	3.82
	Only >0.1 g Ag/t	767	0.30	1,460.10	31.14	95.69	3.07
B10 - Copper Harbor Congl.	No trimming	2,427	0.10	117.90	1.60	7.15	4.48
	Only >0.1 g Ag/t	448	0.30	117.90	8.20	14.96	1.82

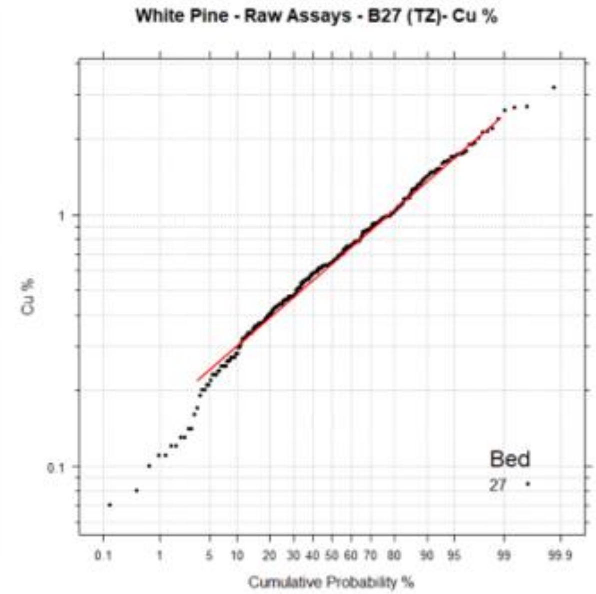
Histogram distributions superposed by normal density curves and probability plot curves of copper and silver grades are presented in Figure 14.5 and Figure 14.6, only for the following selected intervals, from top to bottom: Thinly (B43), Upper Transition (B41), Top Zone (B27), Dark Gray Massive (B26), Domino (B23) and Lower Transition (B21). Since few extreme values are present in the dataset, and high-grades are generally very small intervals found in the mined-out areas, it was judged unnecessary to apply a high-grade capping to raw assays at this stage of the MRE. Statistics will be revisited after compositing to determine if any capping is required.

Figure 14.5: Raw Assays Histograms and Probability Plots of Selected Beds – Copper (Cu %)

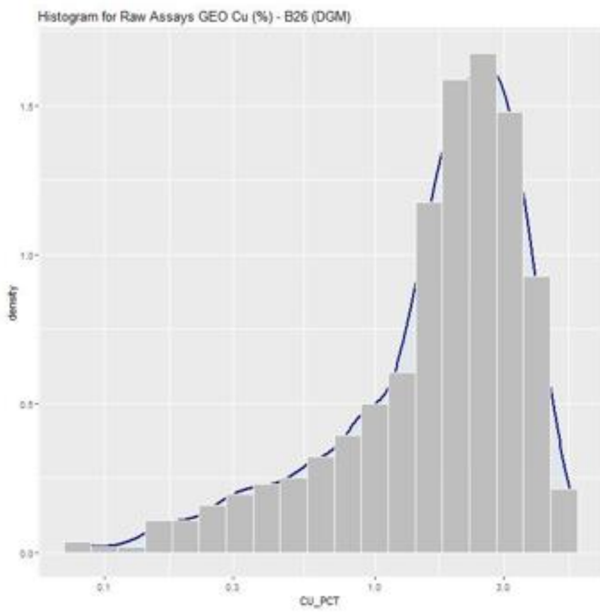


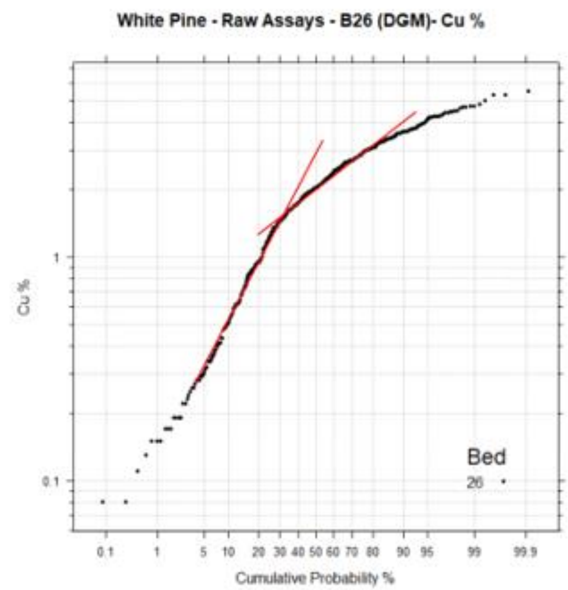
B27 – Top Zone (Cu %)



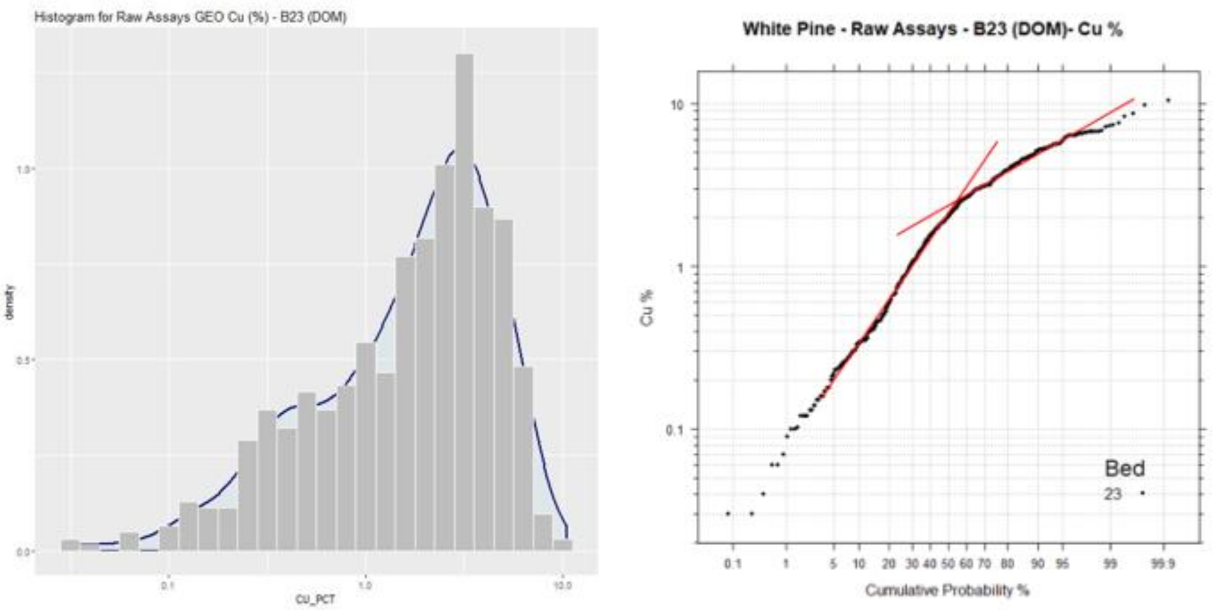


B26 – Dark Gray Massive (Cu %)





B23 – Domino (Cu %)



B21 – Lower Transition (Cu %)

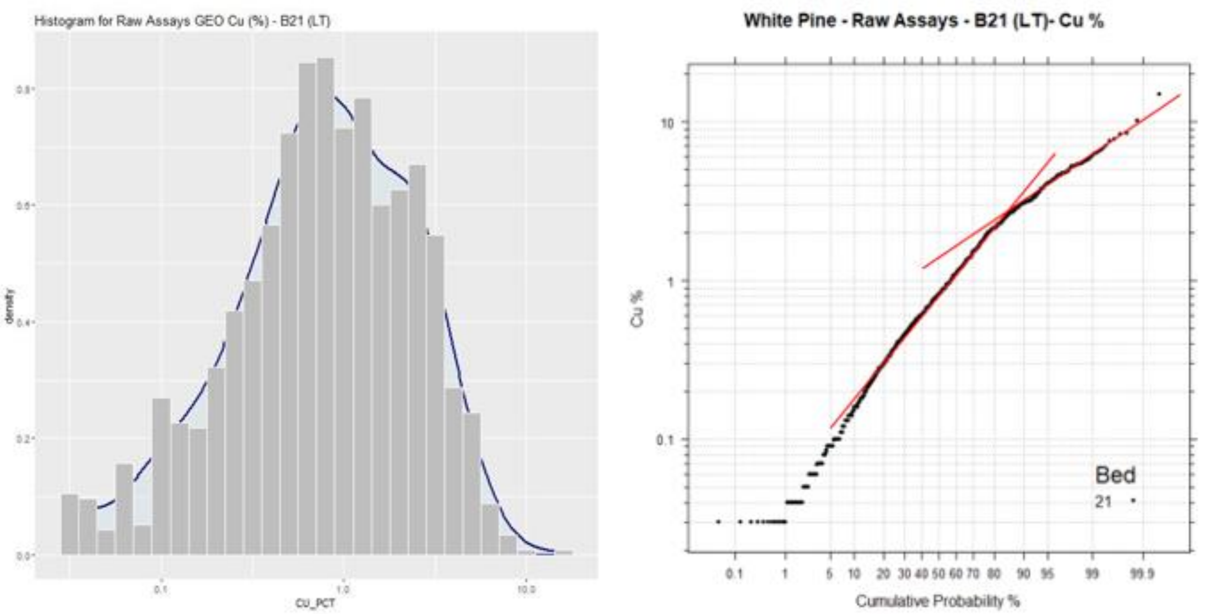
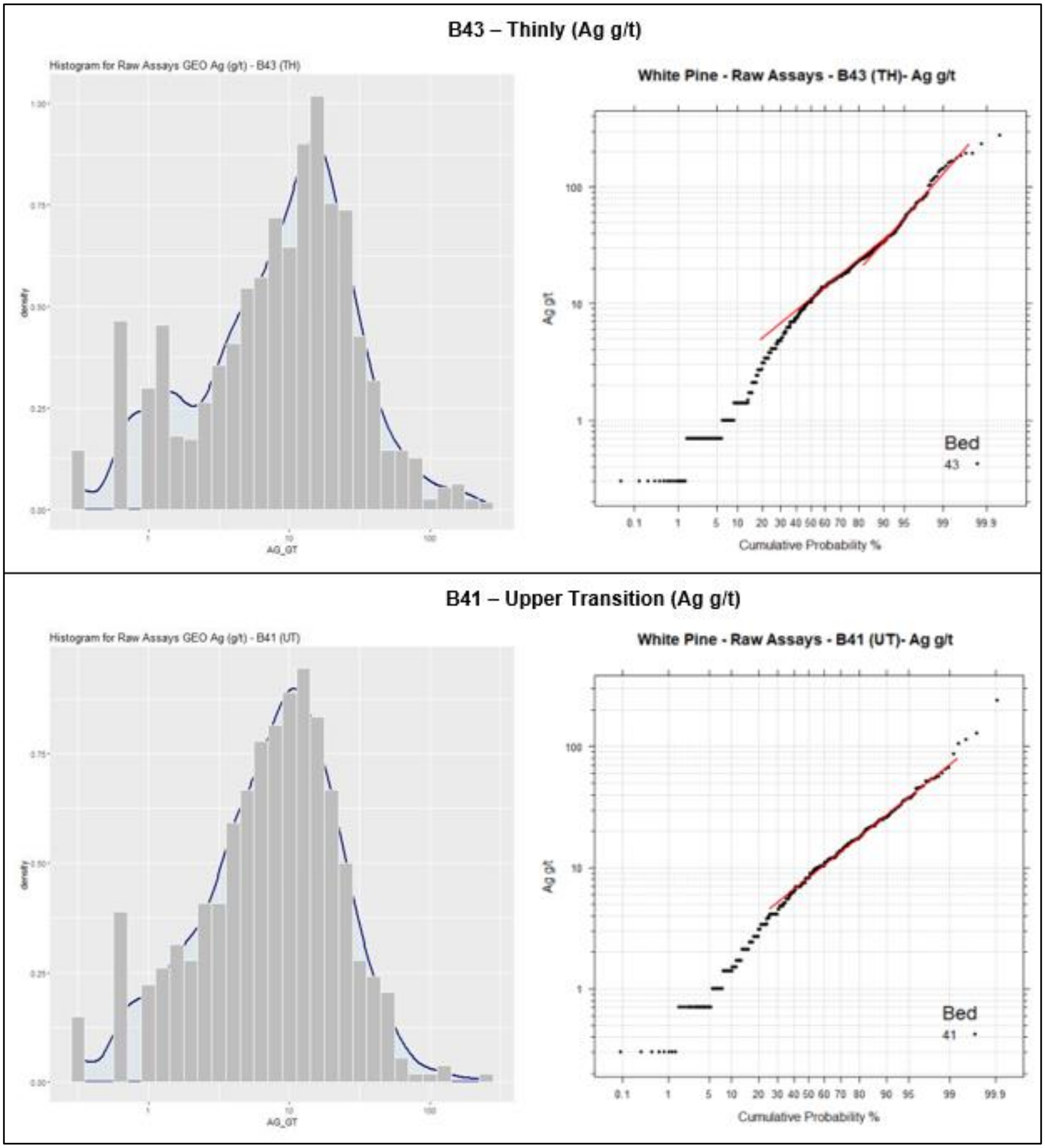
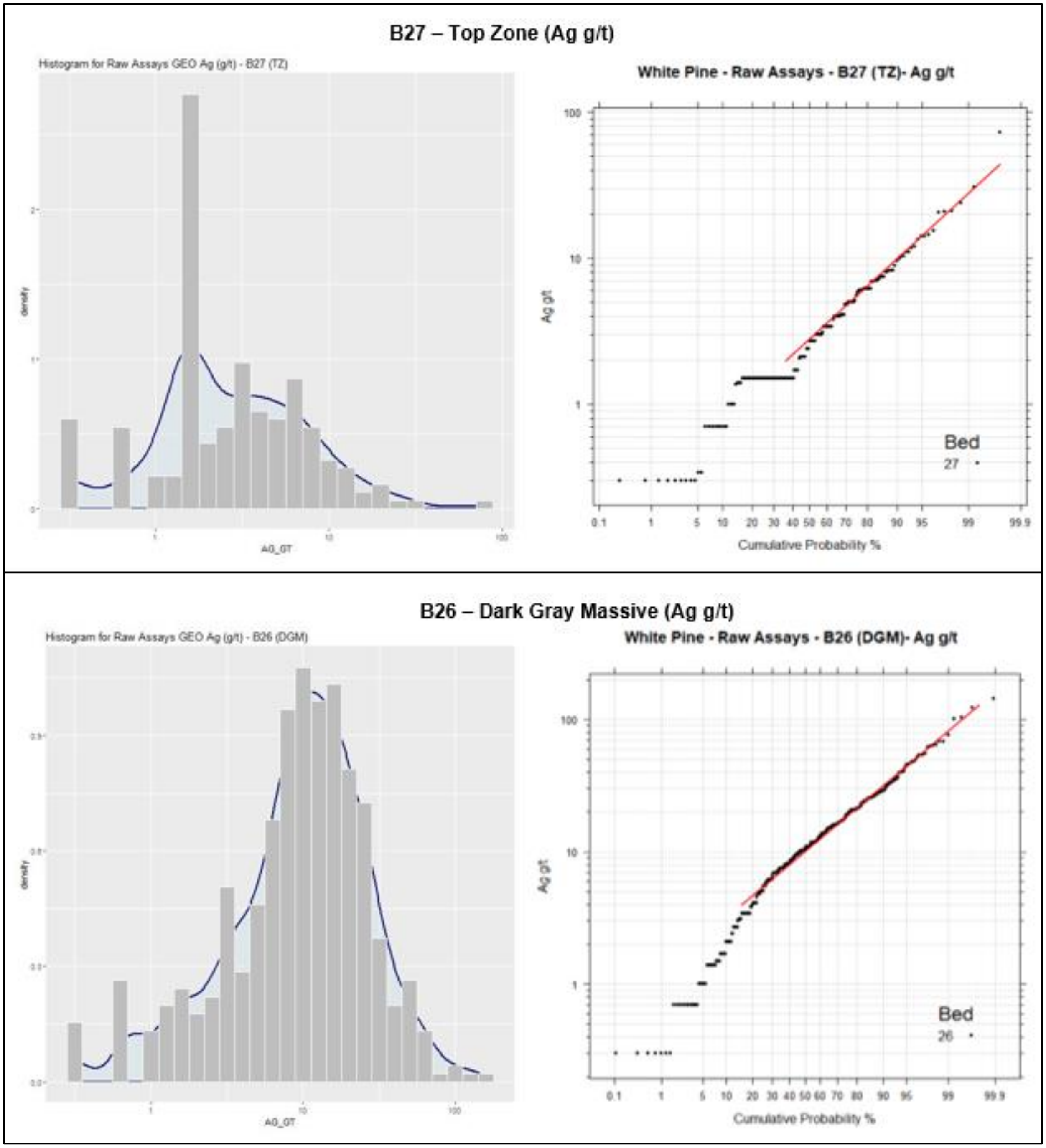
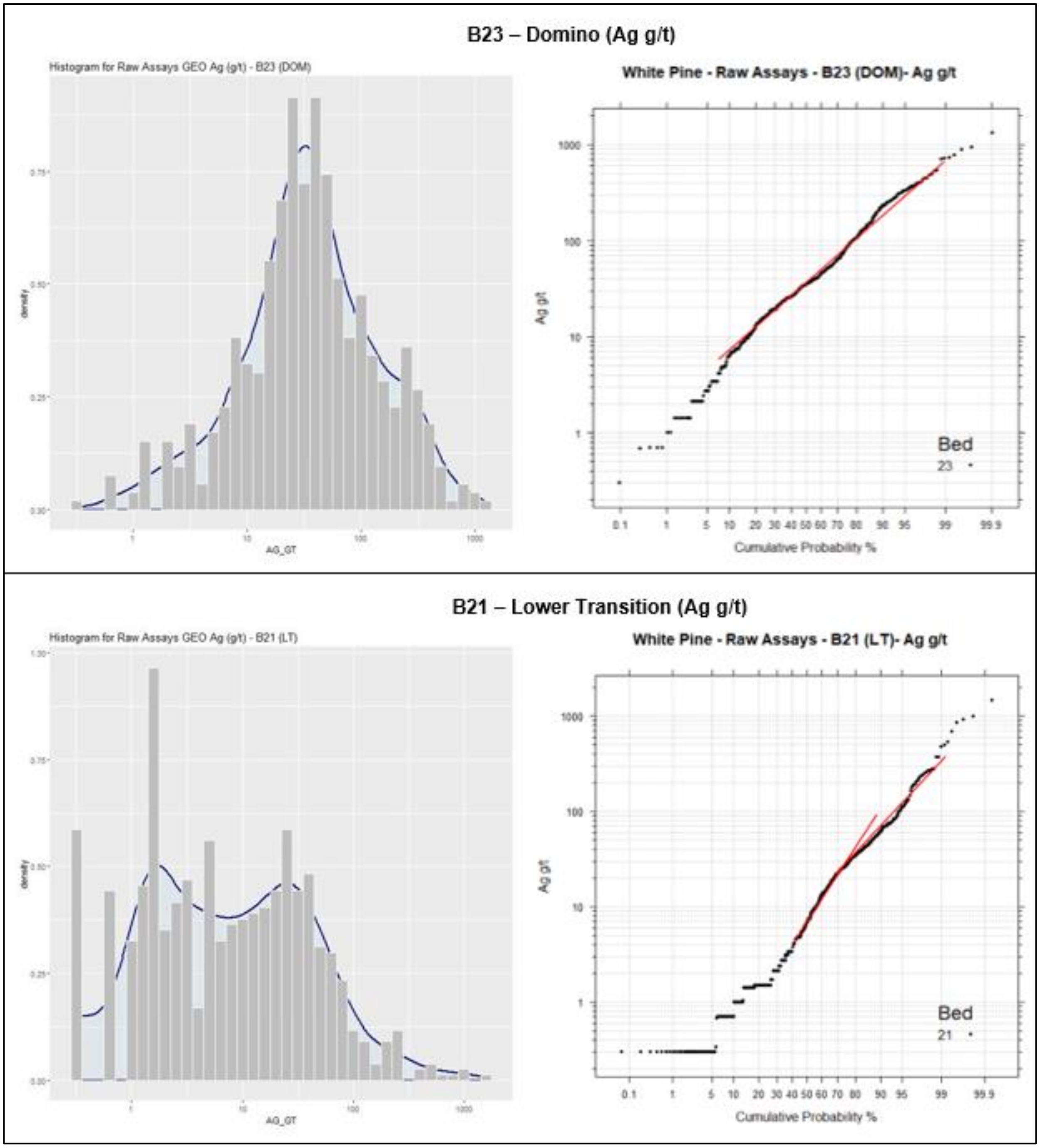


Figure 14.6: Raw Assays Histograms and Probability Plots of Selected Beds – Silver (Ag g/t)




B27 – Top Zone (Ag g/t)
Histogram for Raw Assays GEO Ag (g/t) - B27 (TZ)
density
AG_GT
White Pine - Raw Assays - B27 (TZ)- Ag g/t
Ag g/t
Cumulative Probability %
Bed
27
B26 – Dark Gray Massive (Ag g/t)
Histogram for Raw Assays GEO Ag (g/t) - B26 (DGM)
density
AG_GT
White Pine - Raw Assays - B26 (DGM)- Ag g/t
Ag g/t
Cumulative Probability %
Bed
26



14.4 Statistics of Mined-Out Area

Statistics were calculated separately for the area encompassing the mined-out sector and are presented in Table 14.3. Copper grades are almost always higher in the mined-out sector compared to the complete drilling database**.** However, the poorly mineralized beds display similar values: Copper Harbor Conglomerate (B10), Lower Transition (B21), Red Massive (B24), Dark Gray Massive (B26), Tiger (B29), Upper Sandstone (B30), Brown Massive (B44), UZV (B46) and Widely (B47) show little change between the mined and unmined areas. Beds with the largest copper grade difference are the Domino, Upper Transition and Thinly. The average thickness of the all beds are mostly the same in the mined-out area. Table 14.4 present silver statistics for the same area. When considering silver values greater than 0.1 gpt, we observe that the major differences are in the middle to the upper portion of the FC, where B24 to B44 show higher silver grades in the mined-out area. The Thinly bed (B43) is the only exception since the silver grade in mined-out area is 9% lower than in the entire deposit. In the PS, the Lower Transition and Domino beds (B21 and B23) have lower silver grades (15% and 38% respectively).

Table 14.3: Summary Statistics by Bed – Mined-Out Area (Copper). Highlighted Units - Most Productive at the Historic White Pine Mine

Beds	Number of Assays	Average Thickness m	Grade			Standard Deviation	CoV
			Min.	Max.	Mean		
			Cu %	Cu %	Cu %	Cu %	
B47 - Widely	837	0.90	0.00	2.85	0.20	0.37	1.89
B46 - UZV	695	0.75	0.00	4.32	0.68	0.69	1.00
B44 - Brown Massive	595	0.66	0.00	2.68	0.14	0.18	1.34
B43 - Thinly	623	0.46	0.00	12.19	3.53	1.96	0.56
B41 - Upper Transition	271	0.12	0.00	14.25	3.37	2.09	0.62
B30 - Upper Sandstone	630	1.16	0.00	3.00	0.23	0.25	1.12
B29 - Tiger	209	0.38	0.00	1.30	0.26	0.22	0.86
B27 - Top Zone	123	0.27	0.00	3.20	0.92	0.54	0.59
B26 - Dark Gray Massive	248	0.43	0.00	5.49	1.98	1.21	0.61
B24 - Red Massive	377	0.50	0.00	2.36	0.24	0.21	0.86
B23 - Domino	348	0.31	0.07	10.47	2.91	1.88	0.65
B21 - Lower Transition	523	0.59	0.03	14.92	1.40	1.50	1.07
B10 - Copper Harbor Congl.	1,210	1.59	0.00	13.45	0.39	1.23	3.18

Table 14.4: Summary Statistics by Bed – Mined-Out Area (Silver)

Beds	Dataset	Number of Assays	Grade Min. g Ag/t	Grade Max. g Ag/t	Grade Mean g Ag/t	Standard Deviation g Ag/t	CoV
B47 - Widely	No trimming	837	0.10	64.40	0.69	3.41	4.95
	Only >0.1 g Ag/t	76	0.30	64.40	6.59	9.53	1.45
B46 - UZV	No trimming	695	0.10	63.80	3.54	6.21	1.75
	Only >0.1 g Ag/t	299	0.30	63.80	8.09	7.29	0.90
B44 - Brown Massive	No trimming	595	0.10	38.40	0.43	2.57	5.94
	Only >0.1 g Ag/t	33	0.30	38.40	6.09	9.34	1.53
B43 - Thinly	No trimming	623	0.10	183.40	13.67	17.67	1.29
	Only >0.1 g Ag/t	558	0.30	183.40	15.25	18.02	1.18
B41 - Upper Transition	No trimming	271	0.10	240.00	11.85	18.97	1.60
	Only >0.1 g Ag/t	241	0.30	240.00	13.32	19.63	1.47
B30 - Upper Sandstone	No trimming	630	0.10	33.90	0.43	2.28	5.28
	Only >0.1 g Ag/t	44	0.30	33.90	4.84	7.38	1.52
B29 - Tiger	No trimming	209	0.10	29.50	0.51	2.75	5.41
	Only >0.1 g Ag/t	12	0.30	29.50	7.22	9.55	1.32
B27 - Top Zone	No trimming	123	0.10	72.70	2.39	7.30	3.06
	Only >0.1 g Ag/t	43	0.30	72.70	6.64	11.24	1.69
B26 - Dark Gray Massive	No trimming	248	0.10	144.00	11.38	16.75	1.47
	Only >0.1 g Ag/t	184	0.30	144.00	15.30	17.85	1.17
B24 - Red Massive	No trimming	377	0.10	78.80	0.81	4.74	5.82
	Only >0.1 g Ag/t	28	0.7	78.8	9.71	14.96	1.54
B23 - Domino	No trimming	348	0.10	368.20	44.64	59.52	1.33
	Only >0.1 g Ag/t	309	0.7	368.20	50.26	60.89	1.21
B21 - Lower Transition	No trimming	523	0.10	854.30	16.60	50.25	3.03
	Only >0.1 g Ag/t	326	0.30	854.30	26.57	61.57	2.32
B10 - Copper Harbor Congl.	No trimming	1,210	0.10	72.70	1.81	7.43	4.10
	Only >0.1 g Ag/t	112	0.30	72.70	18.60	16.97	0.91

14.4.1 Statistics of North-East Sector

Drill holes of the north-east sector (see Figure 14.1) were extracted and descriptive statistics are presented in Table 14.5 and Table 14.6, for copper and silver respectively. Mean copper grades of these 153 drill holes are generally lower than those in the mined-out area, with the exception of a +8% difference for the Thinly (B43) (4.07% Cu compared to 3.53% Cu), and a minor difference observed in the Red Massive (B24) as well in the Dark Gray Massive (B26). The higher grade in these two beds is shadowed by thinner intervals. Conversely, a higher thickness in the Upper Transition (B47) makes up for a lower copper-grade compared to the mined-out area. The Top Zone (B27) and the Upper Sandstone (B30) show increases of 104% and 77% respectively in average bed thicknesses, while hosting low copper grades. While the Top Zone (B27) has a mean copper grade of 0.66% (28% decrease from mined-out area), the Upper Sandstone (B30) has an average grade of 0.18% Cu over an average thickness of 2.05 m. The Domino bed (B23) shows a decrease in both copper grade and bed thickness but has a significant increase of 251% in silver grade. Some other beds also show increases in average silver grade: Lower Transition (B21) and Thinly (B43) for 62% and 49% increase respectively.

Table 14.5: Summary Statistics by Bed – North-East Area (Copper)

Beds	Number of Assays	Average Thickness m	Grade			Standard Deviation	CoV
			Min.	Max.	Mean		
			Cu %	Cu %	Cu %	Cu %	
B47 - Widely	337	1.06	0.00	1.98	0.12	0.21	1.83
B46 - UZV	492	1.22	0.00	3.02	0.68	0.60	0.89
B44 - Brown Massive	304	0.67	0.00	4.17	0.15	0.33	2.21
B43 - Thinly	176	0.12	0.01	9.95	4.07	2.05	0.50
B41 - Upper Transition	175	0.20	0.02	7.91	2.81	1.75	0.62
B30 - Upper Sandstone	668	2.05	0.00	4.71	0.18	0.27	1.44
B29 - Tiger	196	0.56	0.00	1.24	0.19	0.16	0.82
B27 - Top Zone	211	0.55	0.00	2.69	0.66	0.38	0.58
B26 - Dark Gray Massive	234	0.50	0.08	5.28	2.08	1.02	0.49
B24 - Red Massive	225	0.41	0.00	1.84	0.26	0.19	0.74
B23 - Domino	158	0.09	0.06	6.52	1.55	1.42	0.92
B21 - Lower Transition	370	0.71	0.03	7.56	1.23	1.13	0.92
B10 - Copper Harbor Congl.	531	1.59	0.00	3.75	0.11	0.24	2.24

Table 14.6: Summary Statistics by Bed – North-East Area (Silver)

Beds	Dataset	Number of Assays	Grade			Standard Deviation	CoV
			Min.	Max.	Mean		
			g Ag/t	g Ag/t	g Ag/t	g Ag/t	
B47 - Widely	No trimming	337	0.10	17.50	0.90	1.79	1.99
	Only >0.1 g Ag/t	148	0.30	17.50	1.92	2.34	1.22
B46 - UZV	No trimming	492	0.10	35.30	4.86	6.34	1.30
	Only >0.1 g Ag/t	319	0.30	35.30	7.44	6.56	0.88
B44 - Brown Massive	No trimming	304	0.10	20.20	0.89	1.84	2.07
	Only >0.1 g Ag/t	131	0.30	20.20	1.94	2.44	1.26
B43 - Thinly	No trimming	176	0.10	142.00	21.60	26.45	1.22
	Only >0.1 g Ag/t	167	0.70	142.00	22.76	26.67	1.17
B41 - Upper Transition	No trimming	175	0.10	127.50	11.54	14.16	1.23
	Only >0.1 g Ag/t	157	0.70	127.50	12.86	14.38	1.12
B30 - Upper Sandstone	No trimming	668	0.10	15.80	0.58	1.04	1.78
	Only >0.1 g Ag/t	226	0.30	15.80	1.53	1.35	0.88
B29 - Tiger	No trimming	196	0.10	5.00	0.86	0.82	0.95
	Only >0.1 g Ag/t	114	0.30	5.00	1.41	0.66	0.47
B27 - Top Zone	No trimming	211	0.10	30.50	2.31	3.89	1.69
	Only >0.1 g Ag/t	130	0.30	30.50	3.69	4.44	1.20
B26 - Dark Gray Massive	No trimming	234	0.10	101.71	13.25	14.00	1.06
	Only >0.1 g Ag/t	207	0.30	101.71	14.97	14.01	0.94
B24 - Red Massive	No trimming	225	0.10	241.30	2.10	16.37	7.79
	Only >0.1 g Ag/t	96	0.30	241.30	4.79	24.88	5.20
B23 - Domino	No trimming	158	0.10	1327.70	147.23	210.04	1.43
	Only >0.1 g Ag/t	132	0.30	1327.70	176.21	218.48	1.24
B21 - Lower Transition	No trimming	370	0.10	1460.10	32.12	114.82	3.57
	Only >0.1 g Ag/t	276	0.30	1460.10	43.03	131.22	3.05
B10 - Copper Harbor Congl.	No trimming	531	0.10	117.90	1.48	7.15	4.84
	Only >0.1 g Ag/t	275	0.30	117.90	2.76	9.77	3.54

14.4.2 Compositing

Raw data were composited into each mineralized column domain of unequal length, for both Geological and Mining Intervals, leading to six sets of composites: PS-GEO, PS-MINING, FC-GEO, FC-MINING, US-GEO and US-MINING. One composite was generated per drill hole and per column. Each composite was coded using the pertaining column and interval type code.

Statistical checks were undertaken to ensure that the composites were an accurate representation of the raw assays (i.e. length-weighted statistics of assays should be similar to composites for each unit).

14.4.3 Statistics of the Composites (entire White Pine deposit)

Statistical analysis for each column and interval type was undertaken to describe the characteristics of copper and silver grades in the entire White Pine Deposit (as undertaken for the assays). The summary of the statistics for all composites is presented in Table 14.7 and Table 14.8 for copper and silver respectively.

Table 14.7: Summary Statistics of Composites – Entire White Pine deposit (Copper)

Mineralized Column		Number of Composites	Average Thickness m	Grade			Standard Deviation %	CoV
				Min. %	Max. %	Mean %		
Copper	PS - GEO	485	2.18	0.05	14.92	1.07	0.83	0.76
	PS - MINING	485	2.41	0.03	2.92	0.98	0.50	0.51
	FC - GEO	567	3.69	0.05	9.80	1.04	0.87	0.68
	FC - MINING	568	3.85	0.04	3.06	1.00	0.47	0.44
	US - GEO	561	3.02	0.00	2.07	0.80	0.32	0.38
	US - MINING	561	3.09	0.00	2.07	0.79	0.31	0.38

Table 14.8: Summary Statistics of Composites – Entire White Pine deposit (Silver)

Mineralized Column		Number of Composites	Average Thickness m	Grade			Standard Deviation g Ag/t	CoV
				Min. g Ag/t	Max. g Ag/t	Mean g Ag/t		
Silver	PS - GEO	485	2.18	0.1	239.3	10.48	14.74	1.37
	PS - MINING	485	2.41	0.03	75.91	9.57	9.24	0.97
	FC - GEO	567	3.69	0.1	85.87	7.20	6.48	0.85
	FC - MINING	568	3.85	0.06	35.26	6.92	4.72	0.70
	US - GEO	561	3.02	0.1	20.4	3.53	2.51	0.69
	US - MINING	561	3.09	0.09	20.4	3.45	2.51	0.70

Histogram distribution superposed by normal density curves and probability plot curves for copper and silver composites for the PS and FC are illustrated in Figure 14.7, Figure 14.8, Figure 14.9 and Figure 14.10 respectively. Since very few extreme values were found, it was judged unnecessary at this stage to applying capping values.

Figure 14.7: Parting Shale Composite Histogram and Probability Plot – Copper

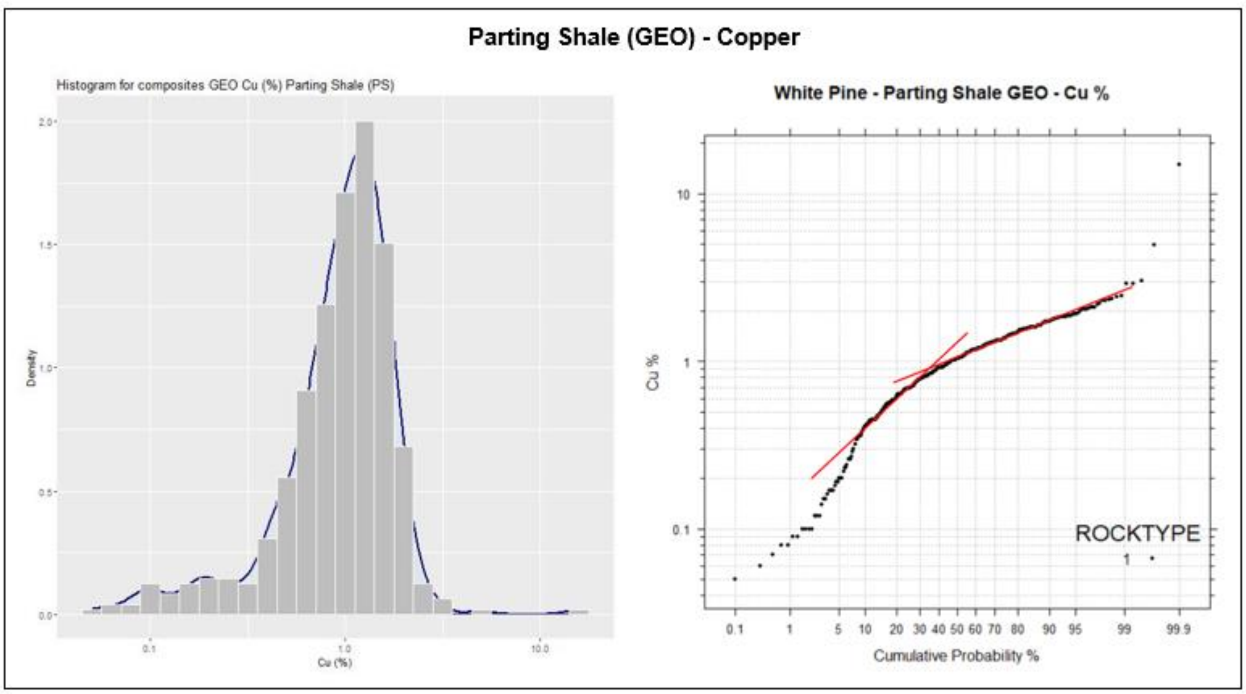


Figure 14.8: Full Column Composite Histogram and Probability Plot – Copper

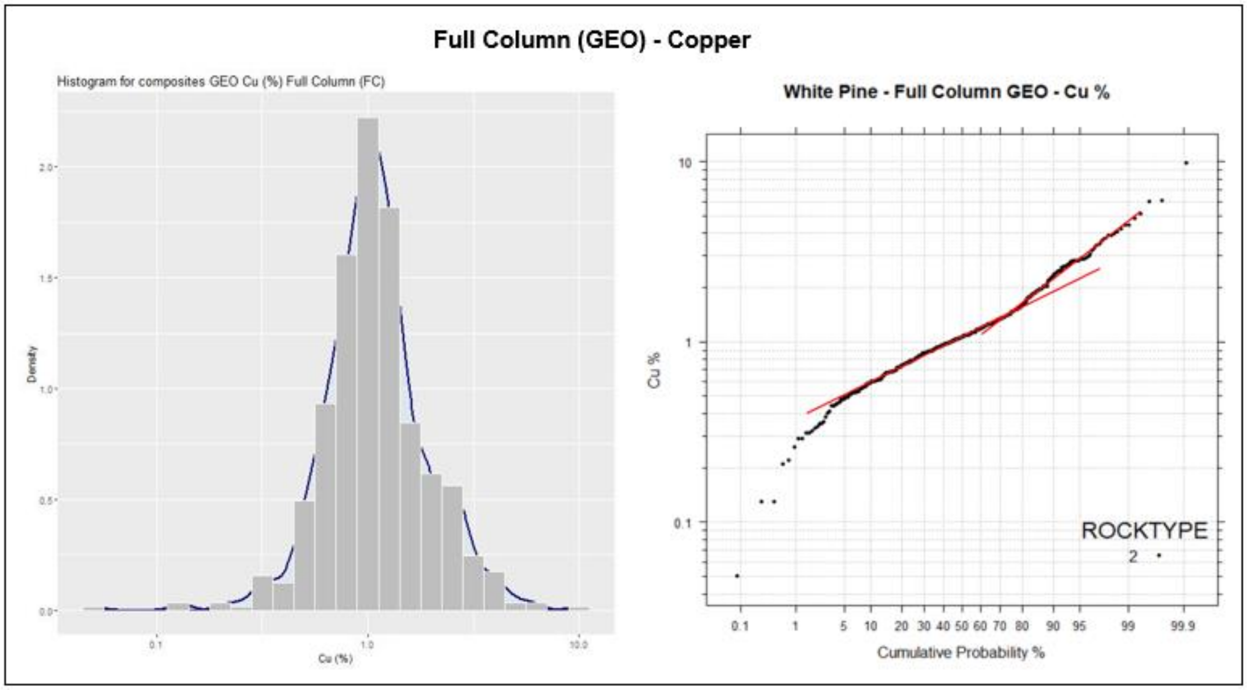


Figure 14.9: Parting Shale Composite Histogram and Probability Plot – Silver

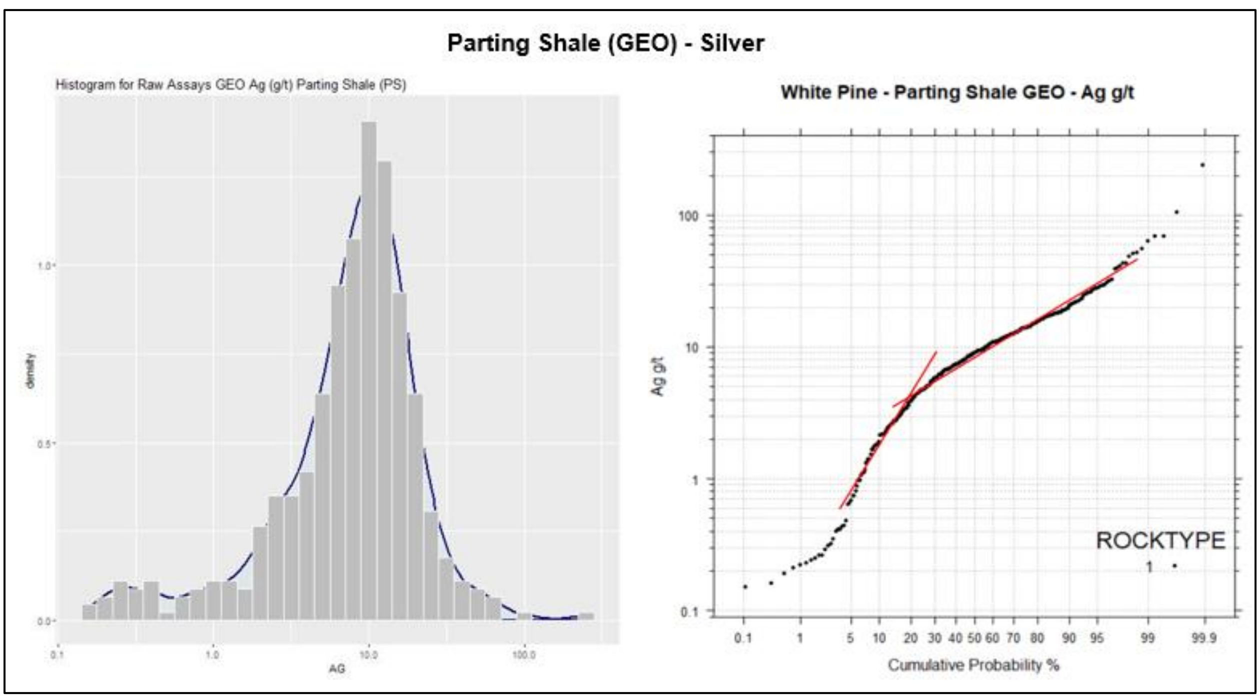
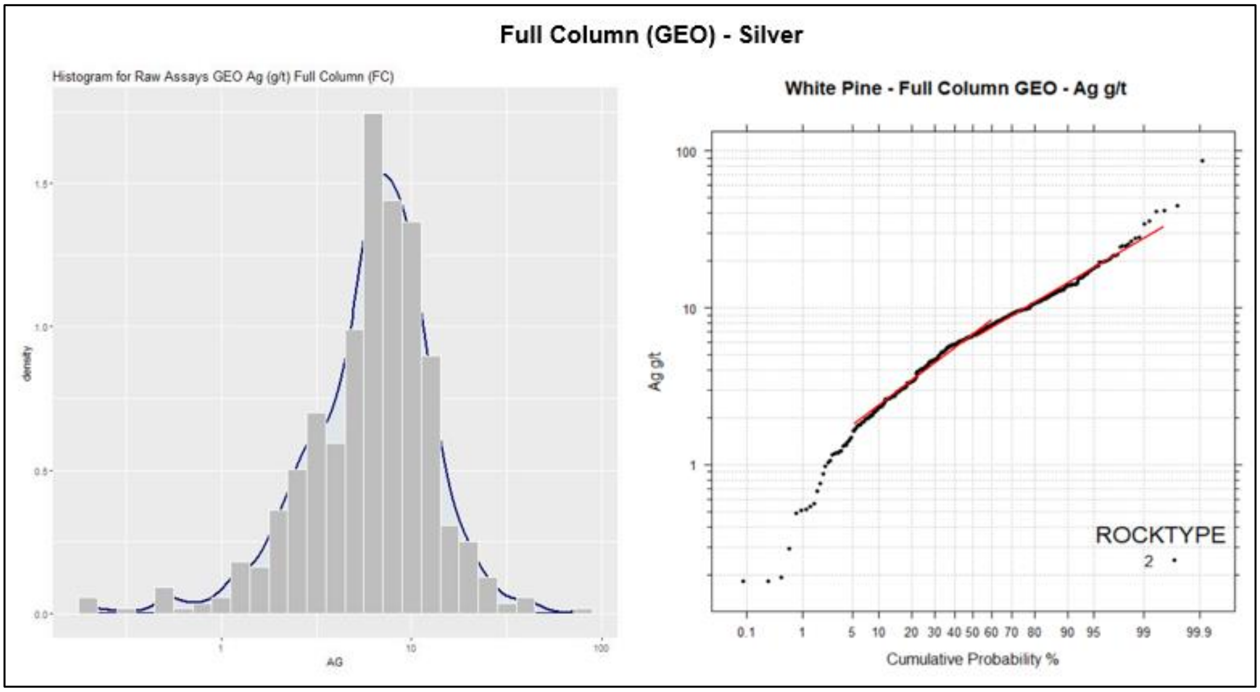


Figure 14.10: Full Column Composite Histogram and Probability Plot – Silver



14.4.4 <u>Statistics of the Composites (Mined-Out Area)</u>

Composites of the mined-out area were extracted, and statistics are presented in Table 14.9 and Table 14.10, for copper and silver respectively. Almost all mean copper and silver grades are higher when compared to the entire drill hole database, with 12% to 41% higher copper grades and 5% to 18% higher silver grades. Only one exception is observed regarding a lower silver grade of 2% within the PS MIN composites. In both datasets of composites (GEO and MIN), average thicknesses are fairly smaller (5% to 9% thinner) in the mined-out area.

Table 14.9: Summary Statistics of Composites – Mined-Out Area (Copper)

Mineralized Column		Number of Composites	Average Thickness m	Grade			Standard Deviation %	CoV
				Min. %	Max. %	Mean %		
Copper	PS - GEO	227	2.06	0.06	14.92	1.25	1.06	0.85
	PS - MINING	227	2.29	0.04	2.92	1.10	0.53	0.48
	FC - GEO	260	3.36	0.13	5.99	1.47	0.77	0.52
	FC - MINING	261	3.51	0.04	3.06	1.28	0.44	0.35

Table 14.10: Summary Statistics of Composites – Mined-Out Area (Silver)

Mineralized Column		Number of Composites	Average Thickness m	Grade			Standard Deviation g Ag/t	CoV
				Min. g Ag/t	Max. g Ag/t	Mean g Ag/t		
Silver	PS - GEO	227	2.06	0.10	239.3	11.00	17.04	1.55
	PS - MINING	227	2.29	0.04	50.57	9.39	7.5	0.80
	FC - GEO	260	3.36	0.10	85.87	8.46	7.04	0.83
	FC - MINING	261	3.51	0.06	27.65	7.45	4.34	0.58

14.4.5 Statistics of the Composites (North-East Sector)

Mean copper grades in the north-east sector are lower than those of the mined-out area, especially for the FC (Table 14.11**Error! Reference source not found.**), where grades decrease by 47% and 28% for the GEO and MINING configurations, in that order. The average thickness of the composites in the MINING configuration represents an increase greater than 30% and 14% for the FC and PS respectively.

As for silver grade (Table 14.12**Error! Reference source not found.**), the PS configurations (both GEO and MINING) have higher silver grades in the north-east sector. When considering the FC, silver grades decrease significantly when compared to the PS.

Table 14.11: Summary Statistics of Composites – North-East Area (Copper)

Mineralized Column		Number of Composites	Average Thickness (m)	Grade			Standard Deviation (%)	CoV
				Min. (%)	Max. (%)	Mean (%)		
Copper	PS - GEO	152	2.47	0.15	2.44	0.98	0.36	0.37
	PS - MINING	152	2.60	0.05	2.32	0.93	0.35	0.38
	FC - GEO	153	4.89	0.26	1.87	0.78	0.26	0.33
	FC - MINING	153	4.92	0.23	1.42	0.77	0.24	0.32

Table 14.12: Summary Statistics of Composites – North-East Area (Silver).

Mineralized Column		Number of Composites	Average Thickness	Grade			Standard Deviation	CoV
			m	Min.	Max.	Mean	g Ag/t	
				g Ag/t	g Ag/t	g Ag/t		
Silver	PS - GEO	152	2.47	0.10	104.14	13.46	14.31	1.06
	PS - MINING	152	2.60	0.07	75.91	12.53	12.03	0.96
	FC - GEO	153	4.89	0.10	35.25	7.44	5.54	0.75
	FC - MINING	153	4.92	0.10	35.26	7.39	5.54	0.75

14.5 Bulk Density Data

A homogeneous 2.70 g/cm^3 density value was used for all rock types in the block model.

14.6 Variography

Grade variography was generated in preparation for the estimation of copper grades using the Ordinary Kriging ("OK") interpolation method. The variography was undertaken on the composites for each column (PS, FC and US). Geovia GEMS™ was used to perform the variographic analysis.

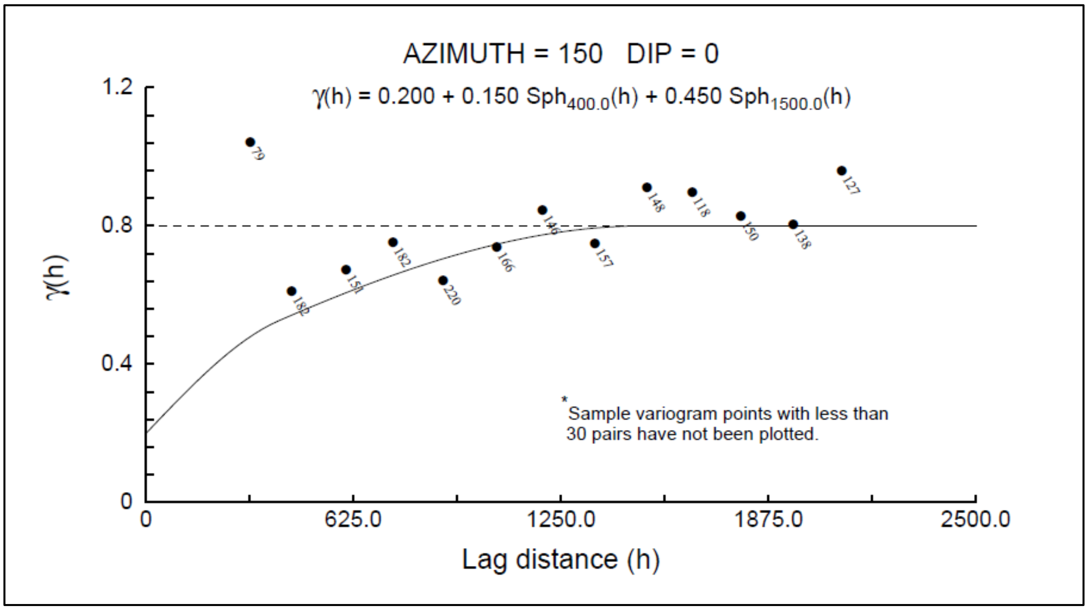
Due to the shallowly dipping nature of the mineralized beds, variograms were unfolded to horizontal and modelled in 2D. A series of variograms was generated from the composites of each column every 30 degrees azimuth on the horizontal plane. The spread angle was set to 30 degrees, with a bandwidth of 500 m. A lag distance of 150 m was applied. All composites (mined and unmined areas) were selected to produce the variograms (PS, FC, US). The manually-fitted variogram models included a nugget effect and two spherical structures. The variography study highlighted a weakly anisotropic distribution of copper towards the south-east in the PS, and a low nugget effect on copper and silver grades. The results of the models for copper are tabulated in Table 14.13.

Table 14.13: Variogram Models for the Copper and Silver Composites of Mineralized Column

Element	Interval Codes	Variogram Type	Nugget	Ranges of Influence (m)								Rotation		
				1st Structure				2nd Structure				Z	Y	Z
				X	Y	Z	Sill	X	Y	Z	Sill			
Cu	PS	Spherical	0.20	400	300	125	0.15	1500	1000	250	0.45	-60	0	0
	FC		0.10	500	350	125	0.20	2000	1500	250	0.25	-60	0	0
	US		0.15	500	400	125	0.45	2000	1000	250	0.2	-60	0	0

Figure 14.11 shows an example of a relative semi-variogram for Cu% for the principal direction, orientated towards 150 degrees azimuth.

Figure 14.11: Variogram Model Cu% for the Parting Shale (PS) Column



14.7 Block Modeling

Three block models were constructed for the White Pine North Deposit, one for each of the columns: PS_2019, FC_2019 and US_2019. All three block models have the same basic parameters, as displayed in Table 14.14. A block size of 25 m x 25 m x 5 m was chosen by GMSI. Blocks were assigned a rock type based on the Leapfrog model discussed in Section 14.2 : 100 for the FC, 200 for the PS and 300 for the US.

Since block height is set at 5 m, and that columns have mean heights between 2.18 and 3.02 m (see Table 14.7), a percent attribute was used in the grade interpolation process. Each block was assigned a percentage related to the overlapping between the block and the column wireframes. This percent attribute is only used when reporting global Resources. A small block size was chosen to ensure the accuracy of the percentage attribute.

Table 14.14: Block Model Parameters

<table>
<tr><th colspan="2">Description</th><th>Number of Blocks</th><th>Block Size (m)</th><th colspan="2">Dimension (m)</th><th>Rotation</th><th colspan="2">Origin (UTM)</th></tr>
<tr><td rowspan="3">PS, FC and US Model</td><td>Colum</td><td>560</td><td>25</td><td>Width</td><td>14,000</td><td rowspan="3">0</td><td>East</td><td>303,000</td></tr>
<tr><td>Row</td><td>400</td><td>25</td><td>Length</td><td>10,000</td><td>North</td><td>5,180,000</td></tr>
<tr><td>Level</td><td>300</td><td>5</td><td>Height</td><td>1,500</td><td>Elevation</td><td>350</td></tr>
</table>

A series of attributes were added for all six block models and are presented in Table 14.15. These are incorporated into the block model to capture the various attributes needed during the block modelling development.

Table 14.15: Block Model Attributes

Model Name	Description
RockType2m	Domain coding (diluted to 2 m)
Density	Specific gravity (2.7 g/cm^3)
Percent2m	Percent block attribute (diluted to 2 m)
CU_2M_OK	Copper grades (in percent, diluted to 2 m)
AG_2M_ID2	Silver grades (in g Ag/t, diluted to 2 m)
AVG_DIST	Average distance for sample used
DSTCLOSET	Actual distance to closest point
CATEG	Resource classification (2= Indicated and 3 = Inferred)
PASS	Interpolation pass
TRUETHICKNESS_DIL	Thickness of the Mineralized Column (diluted to 2 m)
DIP	Dip of stratigraphy
IN_LEASE	Within Lease boundaries

14.8 Grade Estimation Methodology

Two interpolation techniques were selected for the White Pine North Project MRE. The OK method was used for copper grade interpolation and the Inverse Distance Squared (“ID^2”) for silver grades.

A percentage block model was used during grade interpolation for copper and silver. All blocks overlapping any mineralized column wireframe are calculated but are weighted with the percentage of volume they occupy inside that wireframe when calculating volumes of material.

The sample search approach used for the estimate of the block model is summarized below and is identical for each block model (also in Table 14.16):

- **First Copper Pass:** A minimum of 5 and maximum of 16 composites within the search ellipse ranges.
- **Second Copper Pass:** A minimum of 3 and maximum of 16 composites within the search ellipse ranges. Only blocks which were not estimated during the first pass could be estimated during the second pass.
- **Third Copper Pass:** A minimum of 1 and maximum of 16 composites within the search ellipse ranges. Only blocks which were not estimated during the first and second pass could be estimated during the third pass.
- **First Silver Pass:** A minimum of 5 and maximum of 16 composites within the search ellipse ranges.
- **Second Silver Pass:** A minimum of 3 and maximum of 16 composites within the search ellipse ranges. Only blocks which were not estimated during the first pass could be estimated during the second pass.
- **Third Silver Pass:** A minimum of 1 and maximum of 16 composites within the search ellipse ranges. Only blocks which were not estimated during the first and second pass could be estimated during the third pass.

Since each hole had a maximum of one composite, no limit of sample per hole was necessary.

It was not judged necessary to apply restrictions on the search ellipse distance on composites of higher grade (high-grade restraining).

The various parameters of interpolation and search ellipses utilized in the Resources estimation of the block model are respectively tabulated in Table 14.16 and Table 14.17. Figure 14.12 illustrates the interpolation passes within the White Pine North deposit area and Figure 14.13 shows a plan view of the Pass 1 search ellipse used in grade interpolation.

Table 14.16: Interpolation Profile Parameters

Composites	Metal	Pass	Composites			Rock Code Target
			Min	Max	Max per Hole	
PS-MINING FC-MINING and US- MINING	Copper	1	5	16	N/A	100 (FC) or 200 (PS) or 300 (US)
		2	3	16	N/A	100 (FC) or 200 (PS) or 300 (US)
		3	1	16	N/A	100 (FC) or 200 (PS) or 300 (US)
	Silver	1	5	16	N/A	100 (FC) or 200 (PS) or 300 (US)
		2	3	16	N/A	100 (FC) or 200 (PS) or 300 (US)
		3	1	6	N/A	100 (FC) or 200 (PS) or 300 (US)

Table 14.17: Sample Search Ellipsoid Settings

Interval Code	Element	Pass	Ellipse Profile Name	Anisotropy Range (m)			Rotation		
				X	Y	Z	Z	X	Z
PS	CU, AG	1	PS_1	750	500	250	-60	0	0
		2	PS_2	1000	750	350			
		3	PS_3	2000	1500	450			
FC		1	FC_1	750	500	250	20	0	0
		2	FC_2	1000	750	350			
		3	FC_3	2000	1500	450			
US		1	US_1	750	500	250	30	0	0
		2	US_2	1000	750	350			
		3	US_3	2000	1500	450			

Figure 14.12: Interpolation Passes - White Pine North Deposit

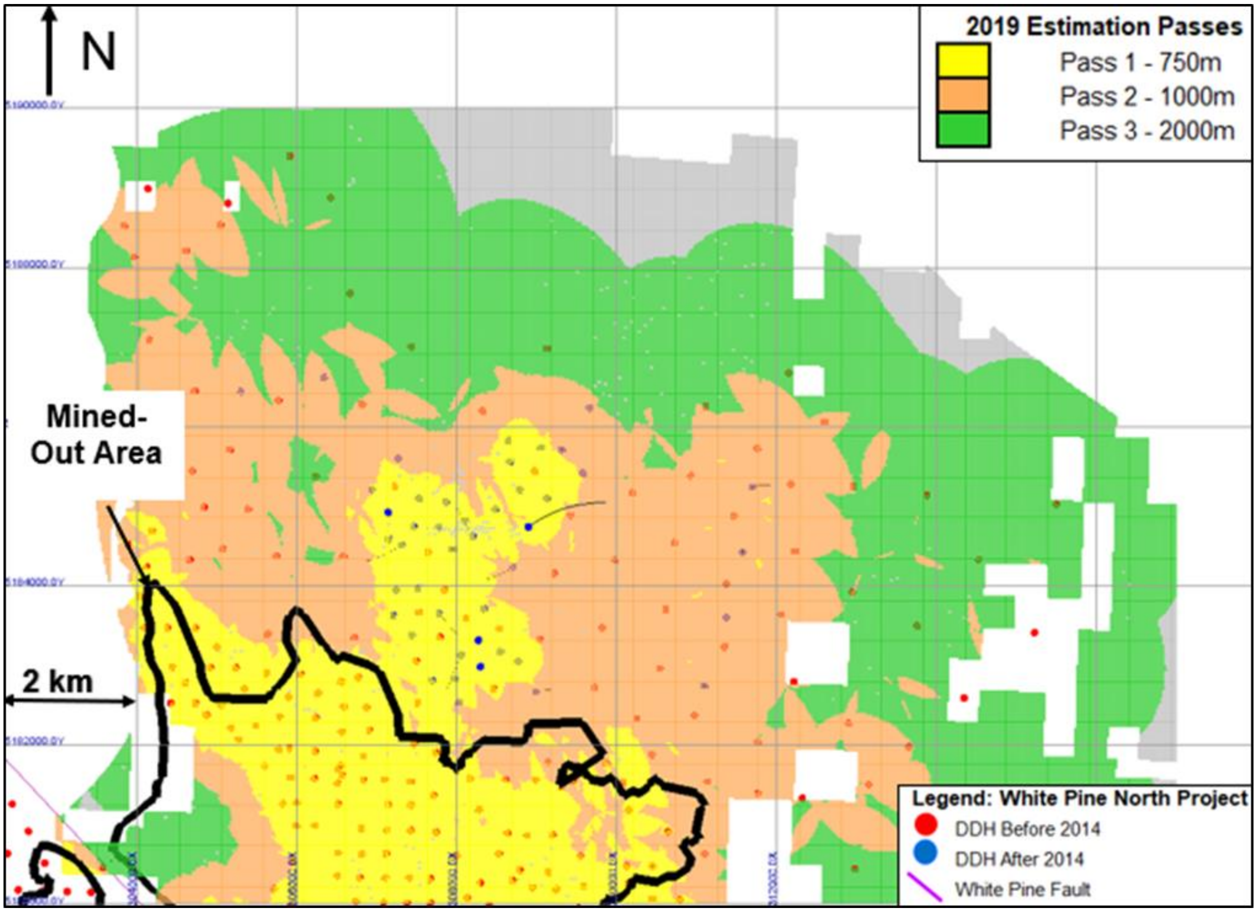
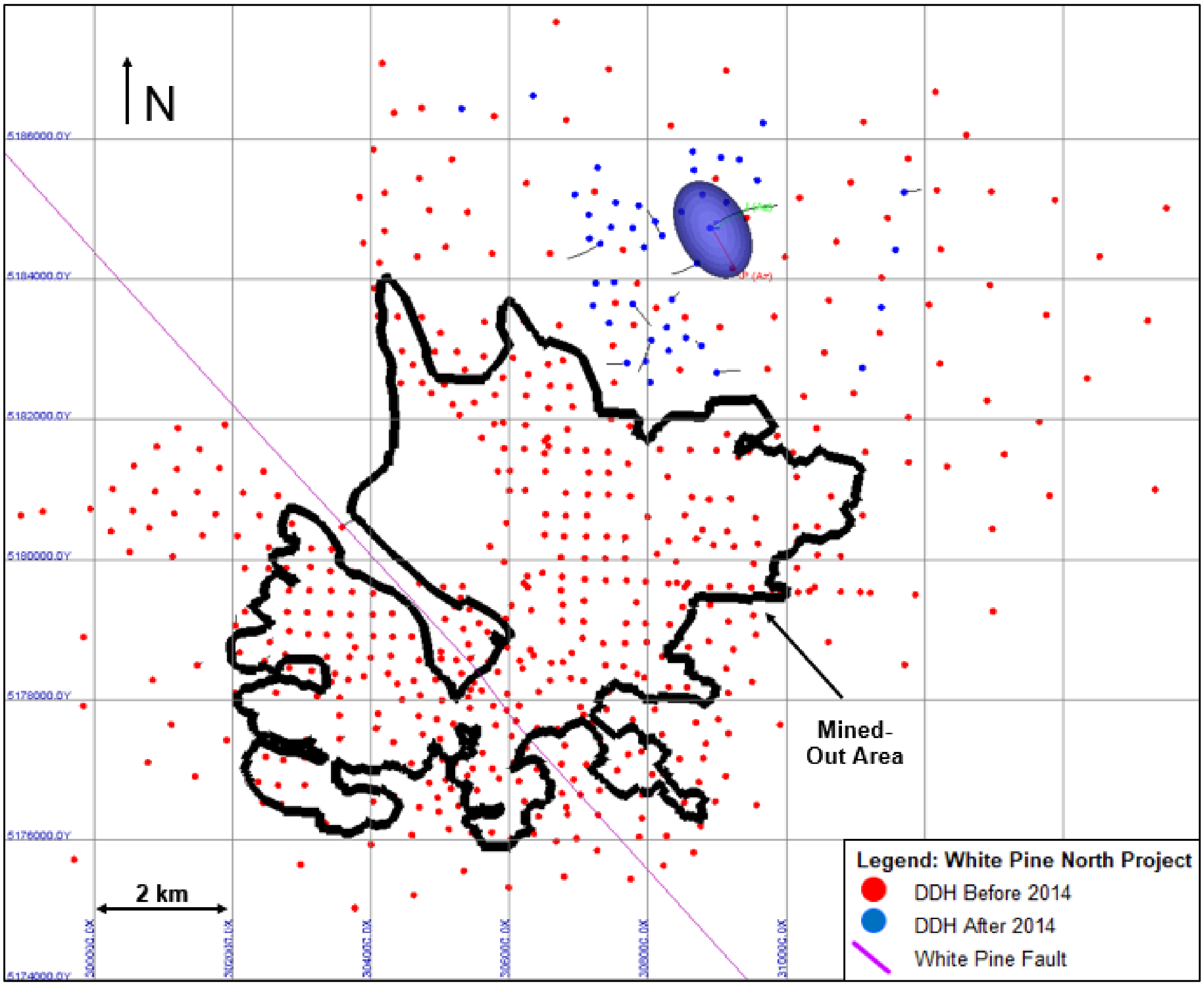


Figure 14.13: Plan View of Pass 1 Search Ellipsoid



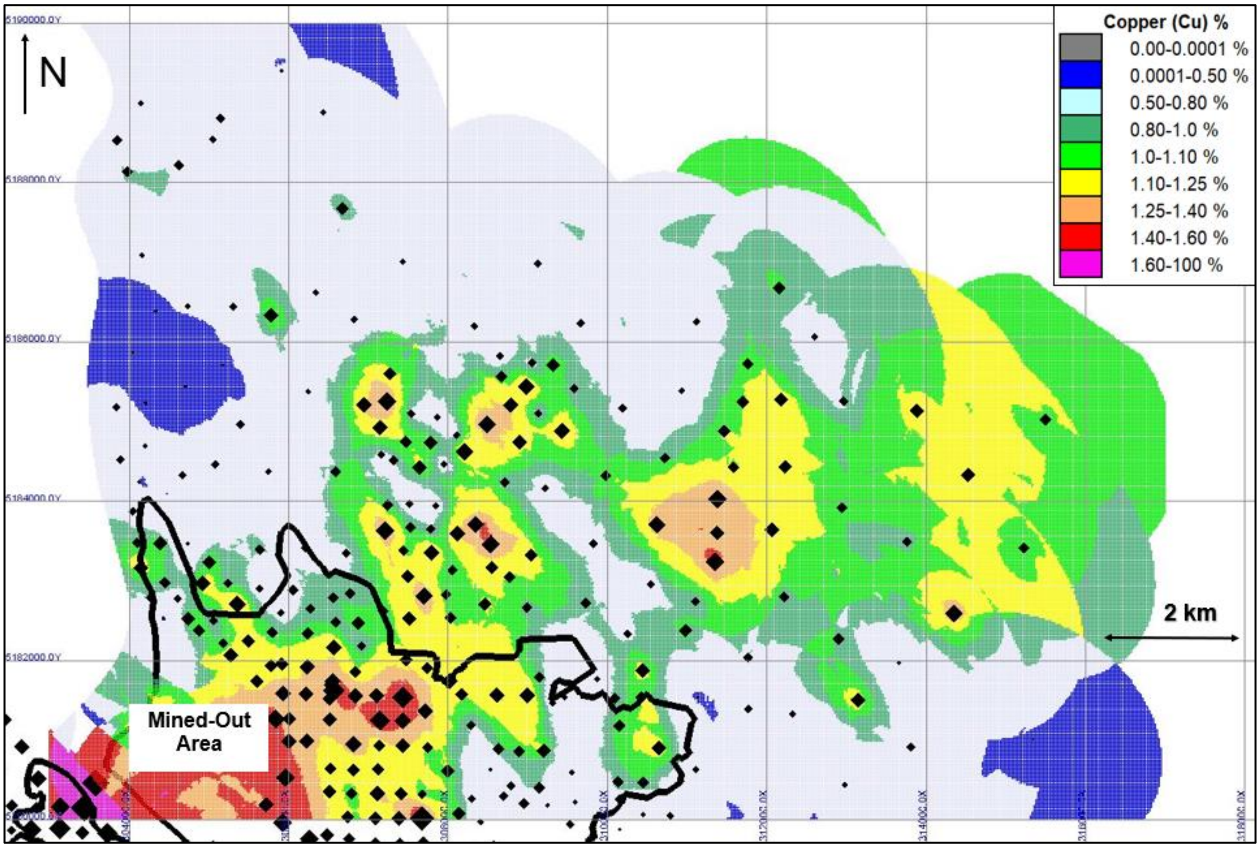
14.9 Grade Estimation Validation

Validation was thoroughly completed on the GMSI block models. The validation process included the following revision steps: 1) visual checks of the resulting grade models; and 2) statistical reviews of the models against the composite data.

14.9.1 Composites versus Interpolated Blocks

A statistical and visual review comparing the average block grades against the average composite grade for each block model was undertaken as a validation tool for the copper grades in each of the interpolation run of the block model. This method of average grade comparison between the estimated results and the composite data sources can indicate a possible distortion in the grade distribution. GMSI is of the opinion that there are no major irregularities between the populations of the composites and the interpolated grade results. Figure 14.14 shows an example of copper composite grades (black diamonds, varying size) versus copper block grades (in color).

Figure 14.14: Parting Shale Block Model and Composites by Copper Grade



14.9.2 <u>Statistical Validation</u>

A statistical comparison between composites used in the interpolation and block grades was performed to evaluate if samples used in the estimation are well represented in the block model. Global statistics were calculated for the zones of mineralisation (PS, FC and US), defined by all blocks and composites between 303000mE - 317000mE, and 5180000mN - 5190000mN (within the block model). Declustering of composites is necessary due to the variable sample spacing, therefore weightings were calculated for each composite and applied during the compilation of descriptive statistics.

Table 14.18 and Table 14.19 compare the weighted mean block and the declustered mean of composite grades for copper and silver considering Pass 1 and 2 for the unmined portion of the deposit.

Table 14.18: Comparative Statistics for Cu % Between Composites and Blocks Grouped by Column (Pass 1 and 2 Only).

Domain	Rock Code	No. of Composites	CU (%) Composites							No. of Blocks	CU (%) Blocks - Passes 1 & 2 Only						
			Min	Max	Declustered Mean	Median	Standard Deviation	Var	CoV		Min	Max	Weighted Mean	Median	Standard Deviation	Var	CoV
FC	100	153	0.23	1.42	0.74	0.76	0.24	0.06	0.32	185,595	0.31	1.45	0.75	0.77	0.18	0.03	0.24
PS	200	152	0.05	2.32	0.91	0.92	0.35	0.12	0.38	149,171	0.11	2.22	0.91	0.90	0.24	0.06	0.27
US	300	150	0.01	1.15	0.57	0.63	0.26	0.07	0.43	162,701	0.02	1.14	0.59	0.60	0.19	0.04	0.33

Table 14.19: Comparative Statistics for Ag (g/t) Between Composites and Blocks Grouped by Column (Pass 1 and 2 Only).

Domain	Rock Code	No. of Composites	AG (g/t) Composites							No. of Blocks	AG (g/t) Blocks - Pass 1 & 2 Only						
			Min	Max	Declustered Mean	Median	Standard Deviation	Var	CoV		Min	Max	Weighted Mean	Median	Standard Deviation	Var	CoV
FC	100	153	0.1	35.26	6.98	6.83	5.54	30.64	0.75	185,595	0.12	35.17	7.4	7.05	3.68	13.54	0.50
PS	200	152	0.07	75.91	11.94	10.82	12.03	144.61	0.96	149,171	0.07	75.67	11.81	11.12	8.32	69.23	0.69
US	300	150.00	0.09	7.89	3.16	3.49	1.73	3.01	0.51	162,701	0.10	7.88	3.37	3.49	1.18	1.40	0.35

The comparison is considered by GMSI as good match between the block model estimated grades and the declustered composites. The difference in mean grades between the composites and blocks is minimal for both copper and silver.

14.9.3 Local Statistical Validation - Swath Plots

The swath plot method is considered a local validation, which works as a visual mean to compare estimated block grades against composite grades within a 3D moving window. It is used to identify possible bias in the interpolation (i.e. over/under estimation of grades)

Swath plots were generated for all composites of the three columns (PS, FC and US) at increments of 300 m (Easting) for both Cu% and Ag g/t and only for blocks estimated by Pass 1 and 2 in the unmined portion of the deposit. Peaks and lows in estimated grades should generally follow peaks and lows in composite (or point) grades in well informed areas of the block model, whereas less informed areas can occasionally show some discrepancies between the grades.

Figure 14.15 illustrates an example swath plot of copper grades for the PS mineralized zone by Easting within the unmined area and, also considering only Pass 1 and 2 as described in Section 14.8. In general, the block model reflects very well the trends shown by the composite copper grades.

Figure 14.15: Swath Plot of Cu % for the Parting Shale (PS) by Easting (Pass 1 and 2 Only – Unmined Area)

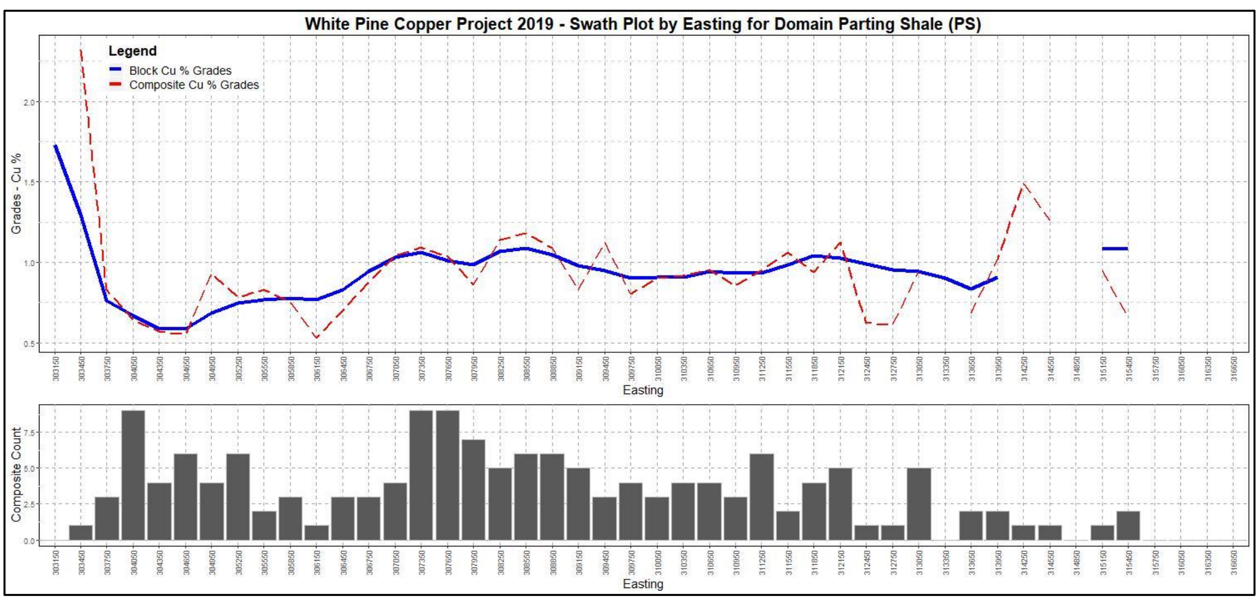
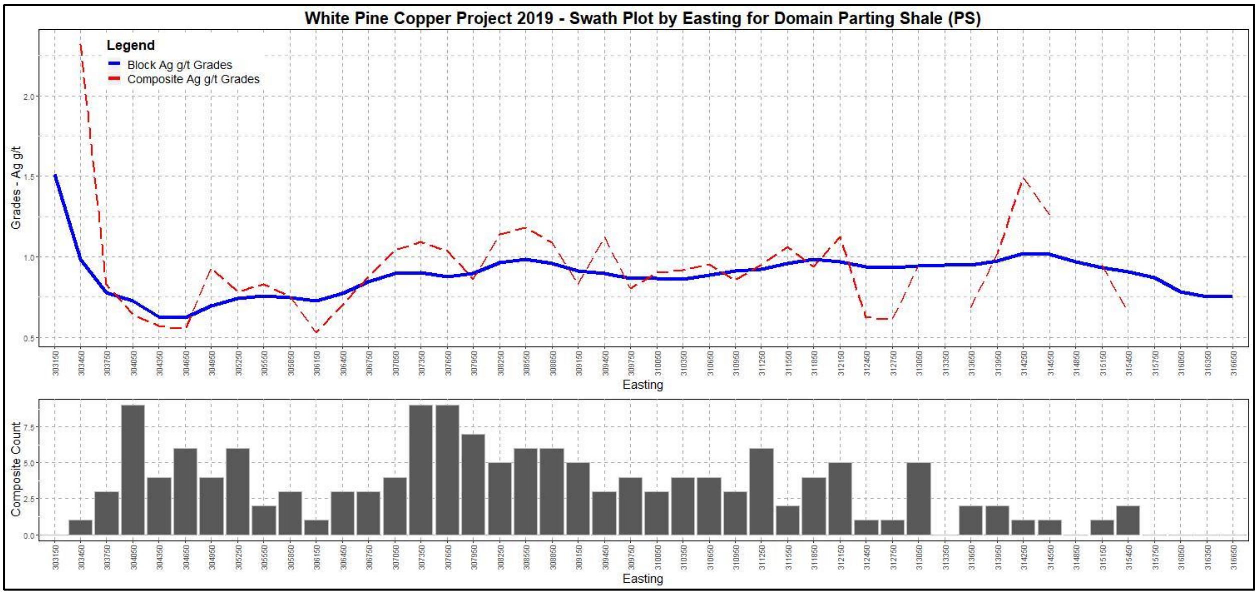


Figure 14.16 illustrates an example swath plot of silver grades for the PS mineralized zone by Easting within the unmined area and, also considering only Pass 1 and 2 as described in Section 14.8. The silver grades of interpolated blocks are generally lower than the composite grades. Overall, the local statistical validation shown illustrated by the easting (X-direction) swath plots did not identify any bias regarding the Resource estimate.

Figure 14.16: Swath Plot of Ag gpt for the Parting Shale (PS) by Easting (Pass 1 and 2 Only – Unmined Area).



14.10 Classification and Resource Reporting

The resource classification definitions used for this report are those published by the Canadian Institute of Mining, Metallurgy and Petroleum ("CIM"). The "*CIM Definition Standards on Mineral Resources and Mineral Reserves*", prepared by the CIM Standing Committee on Reserve Definitions and adopted by the CIM council on May 10, 2014, provides standards for the classification of Mineral Resources and Mineral Reserve estimates into various categories. The category to which a Resource or Reserve estimate is assigned depends on the level of confidence in the geological information available on the mineral deposit, the quality and quantity of data available, the level of detail of the technical and economic information which has been generated about the deposit and the interpretation of that data and information. Under CIM Definition Standards:

A *"Measured Mineral Resource"* is that part of a Mineral Resource for which quantity, grade or quality, densities, shape, physical characteristics are so well established that they can be estimated with confidence sufficient to allow the appropriate 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.

An *"Indicated Mineral Resource"* is that part of a Mineral Resource for which quantity, grade or quality, densities, shape and physical characteristics can be estimated with a level of confidence sufficient 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 detailed and reliable exploration, sampling and testing and is sufficient to confirm 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.

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

In addition, the classification of interpolated blocks is undertaken by considering the following criteria:

- Quality and reliability of drilling and sampling data;
- Distance between sample points (drilling density);
- Confidence in the geological interpretation;
- Continuity of the geologic structures and the continuity of the grade within these structures;
- Statistics of the data population;
- Quality of assay data.

The Resources were classified according to the above mentioned criteria which also directed the choice of the search parameters for each interpolation pass during the block estimation.

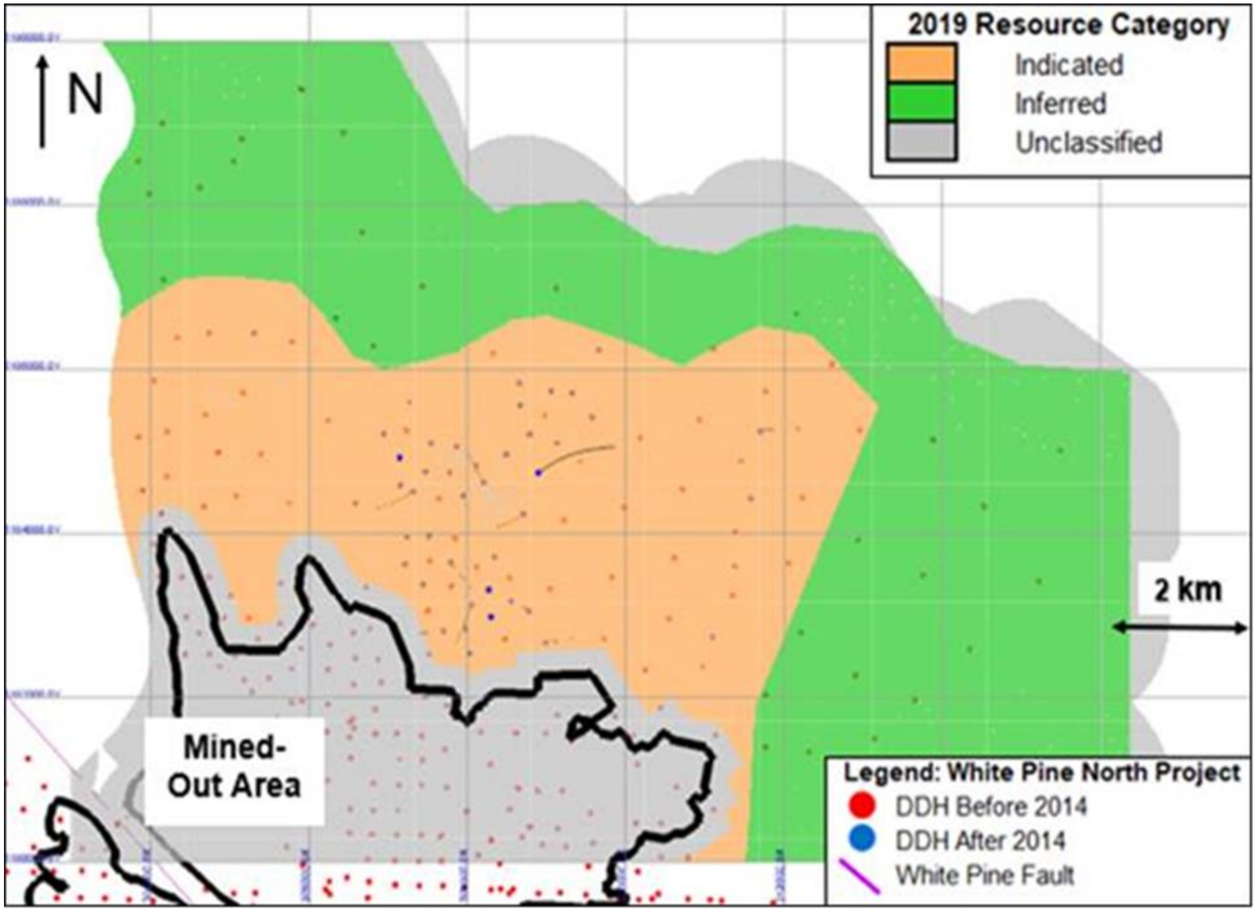
- No Measured Resources are reported for this Mineral Resource;
- Indicated Resources correspond to the blocks which were estimated in the first and second pass copper estimation pass;
- Inferred Resources are the blocks estimated from the third copper estimation pass.

The Mineral Resource classification was subsequently refined manually in plan view to create a coherent classification.

Figure 14.17 shows how the Resource categories are distributed in the deposit. Indicated and Inferred Resources are spatially limited to the north-east of the mined-out area, where lower density drilling occurs.

A 300 m buffer zone (or boundary pillar) was applied around existing workings. Any blocks within this buffer zone were unclassified. Lastly, only blocks within mineral leases where Highland has a greater than or equal to 25% ownership were classified as Mineral Resources.

Figure 14.17: Resource Categories in White Pine North Deposit



14.10.1 Discussion on Block Model Validation

Globally, the White Pine North block model is a good representation of composite copper and silver grades used in the estimation. Global statistical validations show no significant over/under-estimation of copper and silver grades. Local statistical validations illustrate good local correlation between the interpolated blocks compare to the composite for copper and silver grades, and no overestimation of grades was observed during the validation of estimated grades for the White Pine North Project.

14.11 Underground Constrained Resources

To establish a Mineral Resource estimate, an underground Room and Pillar ("R&P") mining scenario is judged to be the most adapted to the geometry and dip of the PS, as well as to the tonnage of the deposit. To assess reasonable prospects of economic extraction by underground mining, GMSI considered several parameters such as concentrate prices, process recoveries, operating costs and mining costs to evaluate and calculate a copper cut-off grade. All blocks below this cut-off grade were removed from the constrained Mineral Resources. As mentioned, a minimum mining height of 2.0 m was used to model the PS column. No mining recovery or dilution was applied.

After consideration of the mining parameters, it was deemed that only the PS is potentially economic, therefore the FC and US will not be reported in this Mineral Resource.

14.11.1 Underground Cut-off Calculation Parameters

The following conceptual mining parameters were considered:

- A flat NSR royalty rate of $0.05/lb. Cu payable was applied, which incorporates two royalties on the project (Osisko Gold Royalty and Great Lakes Royalty);
- No mining loss and no mining dilution was considered at this stage for the Mineral Resources;
- Mineral Resources are reported using a copper price of US$3.00/lb. and a silver price of US$16/oz;
- Metallurgical recovery of 88% for copper and 73.4% for silver;
- A payable rate of 96.5% for copper and 89.3% for silver was assumed;
- A cut-off grade of 0.9% Cu was used to report the Mineral Resources;
- Operating costs are based on a processing plant located at the White Pine site.

14.11.2 Underground Mineral Resource Estimate

The White Pine North Deposit Underground Indicated Mineral Resources are reported at 133.4 million tonnes grading an average 1.07% Cu and 14.9 g/t Ag containing 3.15 billion pounds of copper and 63.8 million ounces of silver using a lower cut-off grade of 0.9% Cu for the Parting Shale column. Inferred Mineral Resources are reported at 97.2 Mt grading an average 1.03% Cu and 8.7 g/t Ag containing 2.21 billion pounds of copper and 27.2 Moz of Ag using a cut-off grade of 0.9% Cu.

Table 14.20 reports Mineral Resources for an underground R&P mining scenario for the White Pine North Deposit by Resource categories. All parameters used in the calculations are presented in the table's notes.

Table 14.20 Mineral Resources for the Parting Shale Column - White Pine North Deposit 0.9% Cu Cut-off Grade - August 30, 2019

Resource Category	Tonnage (M tonne)	Copper Grade (%)	Silver Grade (g/t)	Copper Contained (M lbs.)	Silver Contained (M oz)
Indicated	133.4	1.07	14.9	3,154	63.8
Inferred	97.2	1.03	8.7	2,210	27.2

Notes on Mineral Resources:

15) Mineral Resources are reported using a copper price of US$ 3.00/lb. and a silver price of US$ 16/oz.
16) A payable rate of 96.5% for copper and 89.3% for silver was assumed.
17) Metallurgical recoveries of 88% for copper and 76% of silver were assumed.
18) A cut-off grade of 0.9% Cu was used based on an underground "room and pillar" mining scenario.
19) Operating costs are based on a processing plant located at the White Pine site.
20) A flat NSR royalty rate of $0.05/lb. Cu payable was applied, which incorporates two royalties on the project (Osisko Gold Royalty and Great Lakes Royalty).
21) The Parting Shale column was modelled using a minimum true thickness of 2 m.
22) No mining dilution or mining loss was considered for the Mineral Resources.
23) Mineralized rock bulk density is assumed at 2.7 g/cc.
24) Classification of Mineral Resources conforms to CIM definitions.
25) The qualified person for the estimate is Mr. Réjean Sirois, P.Eng., Vice President - Geology and Resource for GMSI. The estimate has an effective date of August 30, 2019.
26) Mineral Resources that are not Mineral Reserves do not have demonstrated economic viability. The estimate of Mineral Resources may be materially affected by environmental, permitting, legal, title, taxation, sociopolitical, marketing, or other relevant issues.
27) Parting Shale: Interval defined from the base of the Lower Transition unit to the top of the Tiger unit.
28) The quantity and grade of reported Inferred Resources in this estimation are uncertain in nature and there has been insufficient exploration to define these Inferred Resources as Indicated or Measured Mineral Resources.

14.11.3 Mineral Resource Sensitivity

Table 14.21 and Table 14.22 summarize the sensitivity of the constrained underground Mineral Resources of the PS column for a series of selected cut-offs. The sensitivity analysis uses cut-off grades between 0.8% and 1.25% Cu.

Table 14.21: Parting Shale Constrained Mineral Resource Sensitivity – Indicated

Cut-off Grade (% Cu)	White Pine North Deposit - Indicated				
	Tonnage (Mt)	Grade Cu (%)	Copper Contained (Mlbs)	Grade Ag (g/t)	Silver Contained (Moz)
1.25%	12.0	1.32	349	19.9	7.7
1.0%	90.2	1.13	2,251	16.1	46.8
0.9%	133.4	1.07	3,154	14.9	63.8
0.8%	177.5	1.02	3,982	13.7	78.0

Table 14.22: Parting Shale Constrained Mineral Resource Sensitivity – Inferred

Cut-off Grade (% Cu)	White Pine North Deposit - Inferred				
	Tonnage (Mt)	Grade Cu (%)	Copper Contained (Mlbs)	Grade Ag (g/t)	Silver Contained (Moz)
1.25%	-	-	-	-	-
1.0%	58.0	1.08	1,386	8.9	16.6
0.9%	97.2	1.03	2,210	8.7	27.2
0.8%	157.2	0.96	3,326	9.4	47.6

15. MINERAL RESERVE ESTIMATES

This report is a Preliminary Economic Assessment ("PEA"), there is no Mineral Reserve Estimate stated on the White Pine North Project as per National Instrument NI 43-101 Canadian Standards of Disclosure for Mineral Projects regulations.

16. MINING METHODS

16.1 Introduction

The proposed mining method for the White Pine North Project is conventional drill and blast room-and-pillar given the relatively sub-horizontal thin orebody,

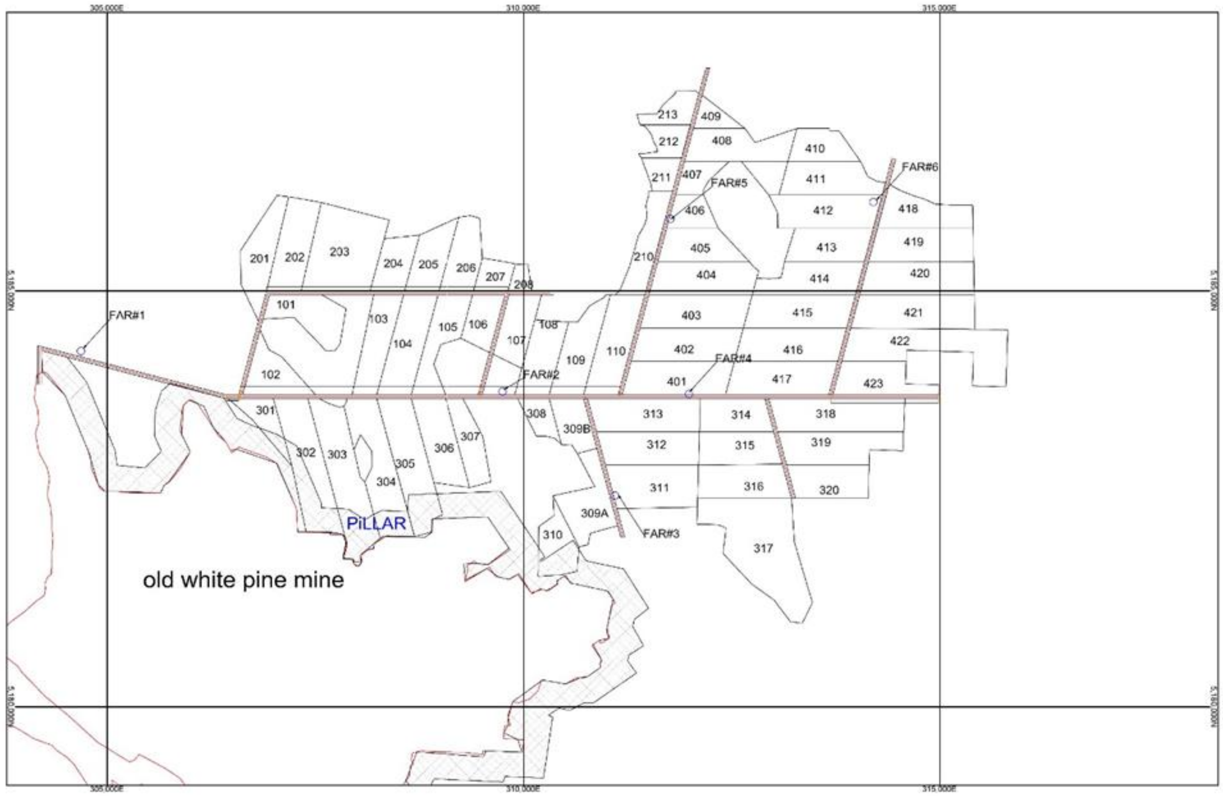
The method consists of the extraction of a series of entries and cross cuts in the ore leaving pillars in place to support the back. The entries, cross cuts and pillars are sized using a geotechnical analysis of the rock, and experience from the old White Pine Mine with similar ground conditions. At this point in the Study, no new data study of rock mechanics has been carried out. However, a later phase will require a rock mechanics analysis.

The Project's mining equipment consists of a low-profile two-boom electric-hydraulic jumbo for drilling. A one-boom electric-hydraulic low-profile bolter is considered for the installation of ground support. A load haul dump ("LHD") unit with a 6 yd^3 capacity is planned for ore removal from the face and haulage of the broken ore to a rock breaker-loading point. A rock breaker will reduce the size of larger particles in the blasted ore, which will be placed on a belt conveyor and transported to the surface crushed ore storage bins from which the mill is fed.

Main accesses and haulage of ore from certain distant working areas are developed using 30 t underground mining trucks to transport the ore to the rock breaker or to the surface stockpile. A mix of ANFO and emulsion explosives are used for blasting to reduce the excavation overbreak. The rooms are mined with a two-pass approach. This approach is recommended for better control, better productivity and to reduce the ground support costs. The first pass will allow a mining recovery of 40%, then part of the pillars is recovered to reach an average mining recovery of 57%.

The mine is comprised of three sectors; the East, Center and West sections. Since access to the mine is from the West portion of the mine, this sector will be mined first. The West is subdivided into 22 extraction panels, the Center is subdivided into 23 extraction panels, and the East is subdivided into 21 extraction panels (see Figure 16.1).The mining direction will generally follow the dip of the orebody, however, in some areas the dip is too steep to follow. In the areas where the dip is too steep, the mining will be performed at an angle to the dip direction. The mining direction will have to consider the direction of the field of stress; a wrong mining direction can cause unplanned dilution.

Figure 16.1: Mine Configuration

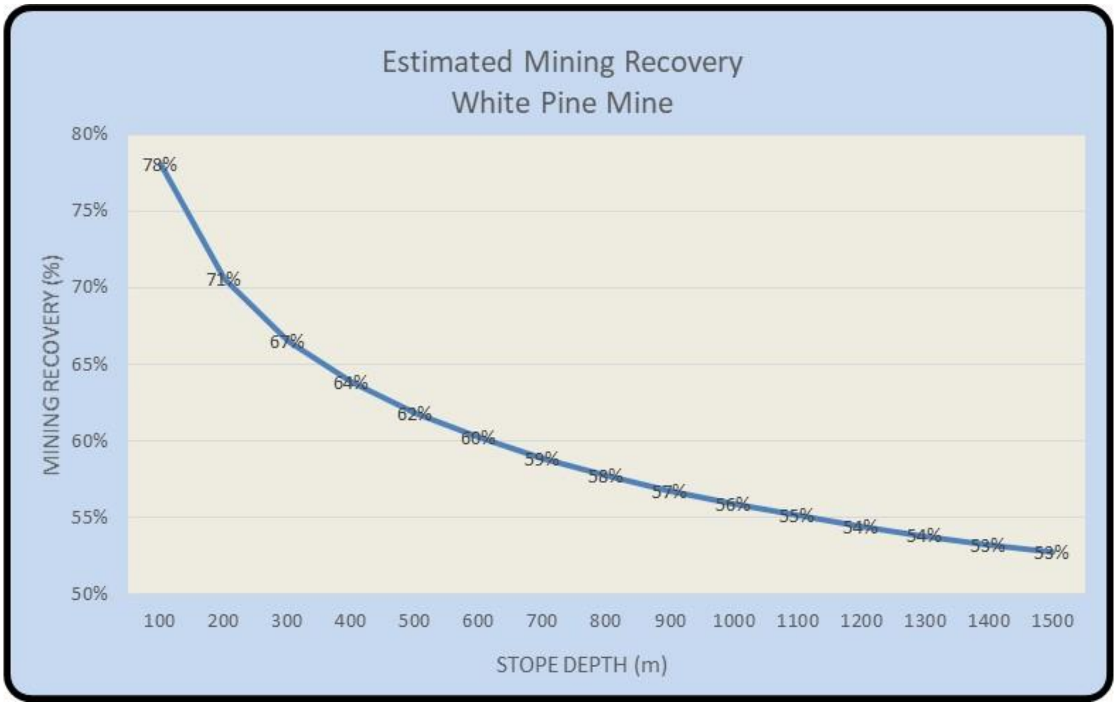


16.2 Geotechnical Considerations

Geotechnical investigation has not been conducted on the underground operations at White Pine North since the closure of the former White Pine Mine. The previous geotechnical work carried out during the operation of the old White Pine Mine was analyzed and used to produce this Preliminary Economic Assessment ("PEA"). A back analysis of the old White Pine was performed by Itasca at the beginning of the Project. The old White Pine Mine was in operation from 1955 to 1995 as a room and pillar operation. Conditions in the mine were reported as variable, depending on the proximity to major structures and the syncline axis. For the most part, back conditions were observed to be good where the back was formed in sandstone. In general, back stability issues were a problem in an area of faulting that was exacerbated by high horizontal stresses. Hence the importance of considering the orientation of the stress fields for the future advancement of excavations. The back analysis determined that the rate of mineral recovery was as follows:

$$\text{Mining Recovery\%} = 1.5219 \times \text{depth (m)}^{-0.145}$$

Figure 16.2: Mining Recovery vs. Stope Depth



As very little analysis has been done at this time, a 300 meter ("m") pillar has been retained with the former White Pine Mine. Moving forward with the Project, the pillar size will have to be analyzed.

16.2.1 <u>Selection of Mining Method</u>

Based on geotechnical information, White Pine Mine history and mineralization geometry, an underground room-and-pillar method is selected for the White Pine North deposit. This mining method allows for both good ore selectivity and productivity. However, a series of pillars are left in place to provide roof stability. The mining design was based on a mining rate of approximatively 5.4 Mt/yr. The underground access and infrastructure development were designed to support the mining method and size based on mining equipment and production rate requirements. Historically the old White Pine Mine has reached this mining rate. In addition, many assumptions are based on historical data from the old White Pine Mine.

16.2.2 <u>Cut off Grade Estimation</u>

The cut-off content was estimated at the start of the Project taking into account the economic parameters as well as the distribution of tonnage and grade. There is very little high grade in the White Pine North deposit. Therefore, a low cut-off grade makes it possible to significantly increase the tonnage to mine but may have a negative impact on the Project's internal rate of return ("IRR"). To reach this, the cut-off grade was increased to ensure the initial investment is taken into account as well as the impact on the IRR. The initial cut-off grade is shown in Table 16.1, with the cut of grade calculations. The estimated cut-off grade is 0.77% Cu. However, it was increased to 0.9% Cu to raise the Project's profitability.

Table 16.1: Cut-off Grade Calculation

Metal Prices	Units	Value
Copper	$/lb	3.00
Silver	$/oz	16.00
In-situ Grade	**Units**	**Value**
Mining Dilution	%	8.0%
In-situ Cu Grade	%	0.77
In-situ Ag Grade	g/t	2.50
Diluted Cu grade	%	0.71
Diluted Ag grade	g/t	2.31
Process Recovery	**Units**	**Value**
Copper	%	88.0%
Silver	%	76.0%
Metal	**Units**	**Value**
Contained Cu Metal	lbs	15.7
Contained Ag Metal	ozs	0.1
Recovered Cu Metal	lbs	13.8
Recovered Ag Metal	ozs	0.1
Concentrate Grade	**Units**	**Value**
Concentration Ratio		48.8
Concentrate Moisture		9.0%
Concentrate Cu Grade	%	30.50
Concentrate Ag Grade	g/t	85.83
Operating Cost	**Units**	**Value**
Processing	$/t ore	6.16
G&A	$/t ore	1.60
Tailings	$/t ore	0.00
UG Mining Cost	$/t ore	16.96
Ore Transport	$/t ore	0.00
Royalty	$/t ore	0.94
Sustaining	$/t ore	7.55
Tax	$/t ore	3.46
Total	**$/t ore**	**36.67**
Margin	$/t ore	**0.00**

Payable Metal	Units	Value
Payable Rates		
Cu	%	96.50
Ag	%	90.00
Minimum Deductions		
Cu	%	1.00
Ag	g/t	30.00
Payable Metal		
Cu	lb/t con.	649
Ag	oz./t con.	2
TC/RC	**Units**	**Value**
Refining Rates		
Cu	$/lb	0.07
Ag	$/oz	0.50
Refining Charges		
Cu	$/t con.	70.00
Ag	$/t con.	0.00
Transportation Costs		
Total Transportation Costs	$/t con.	68.16
Insurance		
Total Insurance Rate	%	0.10

Net Smelter Return	Units
Concentrate Value	
Cu Payable Metal Value	$/t con.
Ag Payable Metal Value	$/t con.
Cu Refining Charges	$/t con.
Ag Refining Charges	$/t con.
Transportation Costs	$/t con.
Insurance Costs	$/t con.
NSR	$/t con.
Ore Value	**Units**
Cu Payable Metal Value	$/t ore
Ag Payable Metal Value	$/t ore
Cu Refining Charges	$/t ore
Ag Refining Charges	$/t ore
Transportation Costs	$/t ore
Insurance Costs	$/t ore
NSR	$/t ore
Ag Credit Net of Refining	$/t ore

16.2.3 Potentially Mineable Portion of the Mineral Resources

The Potentially Mineable Portion of the Mineral Resource comprises 122 Mt at a copper grade of 0.98% Cu and 11.8 g/t Ag and containing 2,67 billion pounds of copper and 46 Moz Ag.

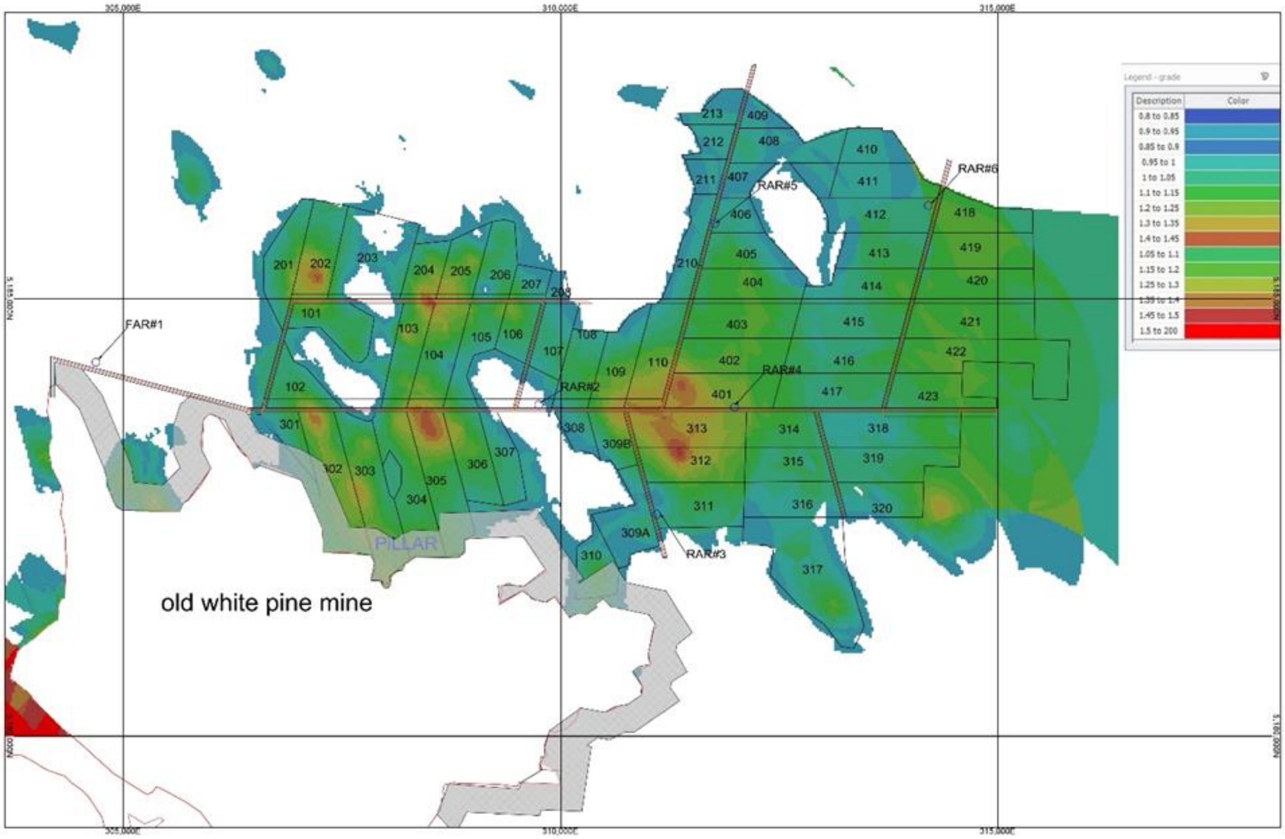
Table 16.2: Potentially Mineable Portion of the Mineral Resource Summary

Description	Tonnes	Cu grade	Ag Grade	Cu Content	Ag Content
	t (X 1000)	(%Cu)	(g/t)	(X 1000) lbs	oz (X 1000)
Development	6,444	0.70	7.74	100,918	1,603
Room and Pillar	114,961	0.99	12.03	2,553,276	44,465
Total	**121,405**	**0.98**	**11.80**	**2,654,194**	**46,068**

Note: Figures have been rounded and totals may be affected by small rounding errors. The potentially extractable tonnages in the Mineral Resource Estimate utilized in this PEA contain both Indicated and Inferred Mineral Resources. The reader is cautioned that Inferred Mineral Resources are considered too speculative geologically to have economic considerations applied to them that would enable them to be categorized as Mineral Reserves, and there is no certainty that value from such Mineral Resources will be realized either in whole or in part.

Copper and silver are the only recovered payable elements; the cut-off grade is defined as copper percent per tonne ("%/t"). The cut-off grades provided in Table 16.1 were calculated using input parameters, such as commodity price, exchange rate, processing parameters and marketing costs. The mining recovery included in this Potentially Mineable Resources is based on the mining recovery formula in Section 16.2. The dilution used in the calculation of Potentially Mineable Resources is based on the historical data from the White Pine Mine, which was 8% dilution on average.

Figure 16.3: Grade Distribution



16.2.4 <u>Main Access Drift</u>

The mine will be accessed via an open box-cut to establish a portal at the mine entrance from surface. Only two drifts are excavated from the portal for the first 35 m deep. Four drifts are subsequently excavated up to the first ventilation raise located at a depth of 189 m. From this ventilation raise to the beginning of the West section of the mine, 6 parallel drifts will be excavated to allow a high ventilation flow rate to the mine. Once in the West section, only four drifts will be required to access the different mining panels. These four drifts are divided into a fresh air intake drift, an ore conveyor drift, a hauling drift and a return air drift. Two more drifts will be added in the six drifts section for fresh air.

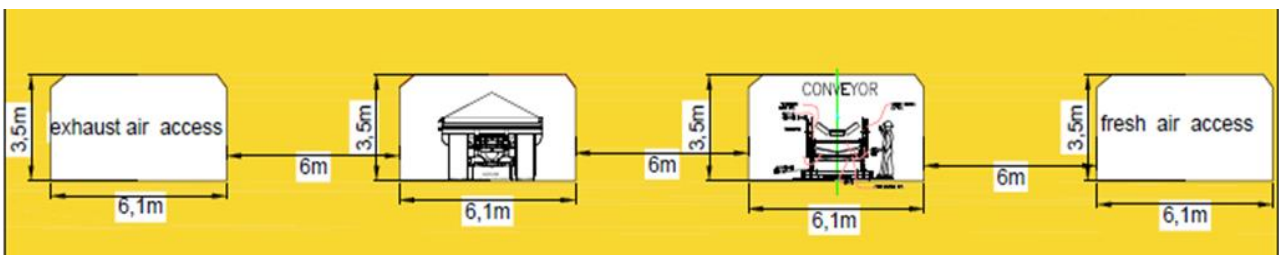
The mine consists of three mining sectors: East, Central and West. In these three sections, the mine development is designed with four parallel drifts per main access including: a fresh air intake drift, an ore conveyor drift, a hauling drift and a return air drift.

The main access drifts will be excavated in the waste from the box-cut to the western section of the mine, this waste will be transported to the surface. Once in the West section, all the sterile will be stored in a closed underground excavation. All drifts are set at a width of 6.1 m, and their height varies from a minimum of 3.5 m to a maximum of 6.1 m.

The main entrance drift back will follow the Full Column geology to allow a better height and resource recovery. The floor however will be flat for equipment purposes. The height in the intersections of the two conveyor drifts is set at a minimum of 6 m to allow the installation of a transfer point between the two conveyors. If a drift intersects a conveyor drift, the height of this section of conveyor drift will also be 6 m to allow for the installation of a steel overpass system. A series of barrier pillars between the main access drift and the stope will remain in place until mining has ended in this mining area. These barrier pillars are designed to be recovered at the end of the mine.

The first drift will be excavated from the portal to the West sector. After, this drift will be excavated through from the West sector and cross the entire mineralized zone from west to east. From this drift, secondary drift direction north-south or south-north will be excavated to allow to for the exploitation of the furthest zone.

Figure 16.4: Stope Entry Configuration



16.2.5 <u>Stope Entry</u>

To access the mining production panel, four stope entry drifts will be excavated. The first stope entry drift will be used for fresh air intake, the second one for hauling and traveling, the third for the stope conveyor and the last one as an exhaust drift. The width of the drift will be 6.1 m and its height will be the same as the production panel.

16.2.6 <u>Intake Ventilation Raise</u>

In addition to the drift, a series of 6 raises will be excavated to allow efficient ventilation of the mine. A fresh air raise will be located at the West sector of the mine. This 5 m diameter raise will be excavated with a raise-boring from the surface with an emergency egress excavated in the center of the West sector of the mine and be raise-bored 5 m in diameter and 148 m long. The raise will provide fresh air for the production period.

16.2.7 Exhaust Ventilation Raise

Five exhaust air raises will be needed to ventilate the mine. The first exhaust air raise will be located at the end of the West section on the mine, the five other exhaust raises are distributed to facilitate mine ventilation according to the life-of-mine ("LoM"). All exhaust ventilation raises will be raise-bored 5 m in diameter.

16.2.8 Stope Design

The orebody was divided into 67 main panels. Due to the orebody's dip in different areas, the access point had to be designed to mine in the best direction to reduce the slope on mining equipment; therefore panels 2 and 3 are separated but mined from the same drift. The stope directions will need to be reassessed to pay special attention to the stress fields. A 300 m horizontal pillar with the old White Pine Mine was maintained. This last pillar could be revaluated in the future to be either reduced or mined.

Table 16.3: Mine Design Summary

LOM Physicals	
Ore Tonnes Development ('000t)	6,444
Cu Grade %	0.75
Ag Grade (g/t)	7.74
Stope Production ('000t)	114,961
Cu Grade %	0.99
Ag Grade (g/t)	12.03
Total Underground Production ('000t)	121,405
Cu Grade %	0.98
Ag Grade (g/t)	11.8
Waste Tonnes ('000t)	1,060
Waste	
Main Drift	90,739
Conveyor Drift	29,034
Ventilation Raise 5 m (m)	4,906

16.3 Mine Operations

16.3.1 Stoping

To access the stope, four access drifts are excavated at the entrance of the stope. One of these drifts is used for fresh air ventilation and the second for exhaust air ventilation. One drift will be used for the stope conveyor and the last one for circulation. From these accesses, the panel operation begins with the drilling and blast method. To achieve and maintain an adequate level of production, the panel must contain at least 12 rooms (headings) in operation simultaneously. If the panel contains less rooms, the mining cycle may be delayed, and productivity will decrease. The mining cycle includes drilling, blasting, ore mucking, ore transportation to a rock breaker and the stope conveyor, scaling and finally ground support. The mining of the room will be done with the two pass approach. In the first pass, larger pillars are left in place; the mining recovery of the first pass is 40%. Once the first is completed, the size of the pillars is reduced to an average mining recovery of 57%. The configuration of the production panels in the first and second passes are presented in Figure 16.5 and Figure 16.6 respectively.

Figure 16.5: Room and Pillar Stope Configuration Phase 1

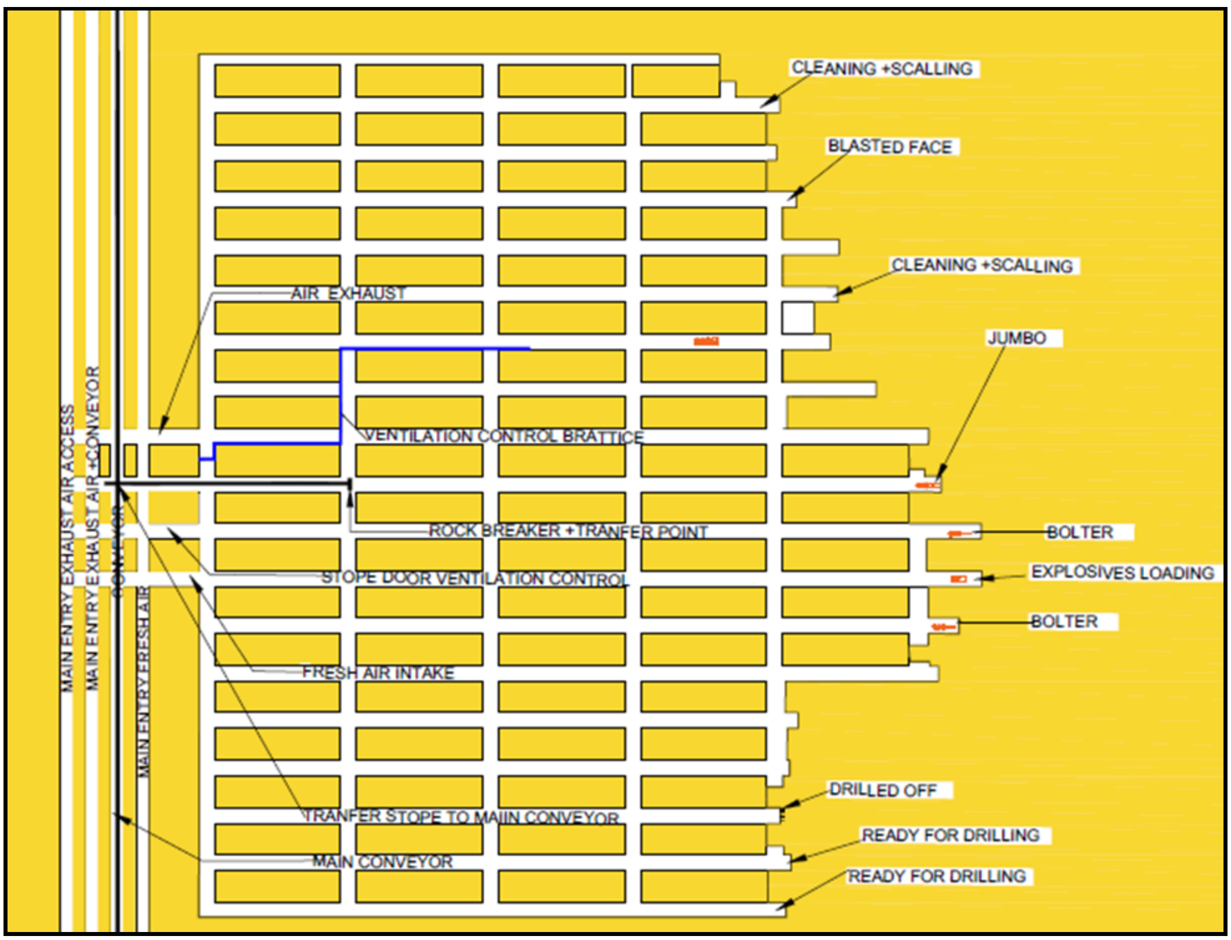
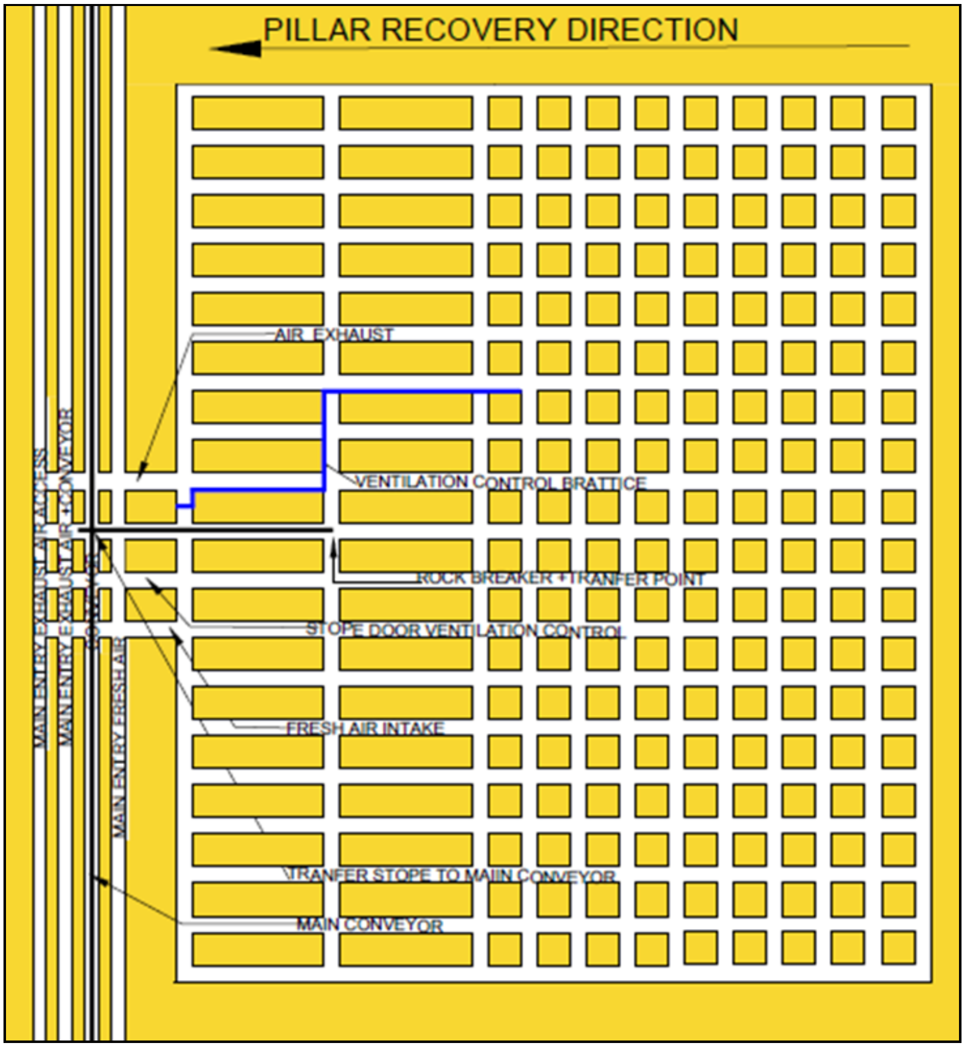


Figure 16.6: Room and Pillar Stope Configuration Phase 2 Pillar Recovery



In the room-and-pillar mining method the mining cycle begins with the drilling of the working face. To perform face drilling, a low-profile hydraulic-electric jumbo with 2 booms is planned. The drilling technique will use a burn cut to allow drilling a length of 4.25 m with an effective break length of 4.0 m. The drilling diameter is 51 millimetres ("mm"), however this dimension can be adjusted according to blasting results. The drilling penetration rate is evaluated at 1.85 m/s and the average drilling time per round is evaluated at 2.5 hrs/round to 3.5 hrs/round, depending on the room height. The drilling penetration rate was determined based on the historical data. Figure 16.7 shows the drilling pattern for the development drift and Figure 16.8 shows the drilling pattern for the production room.

Figure 16.7: Drilling Pattern for the Development Drift

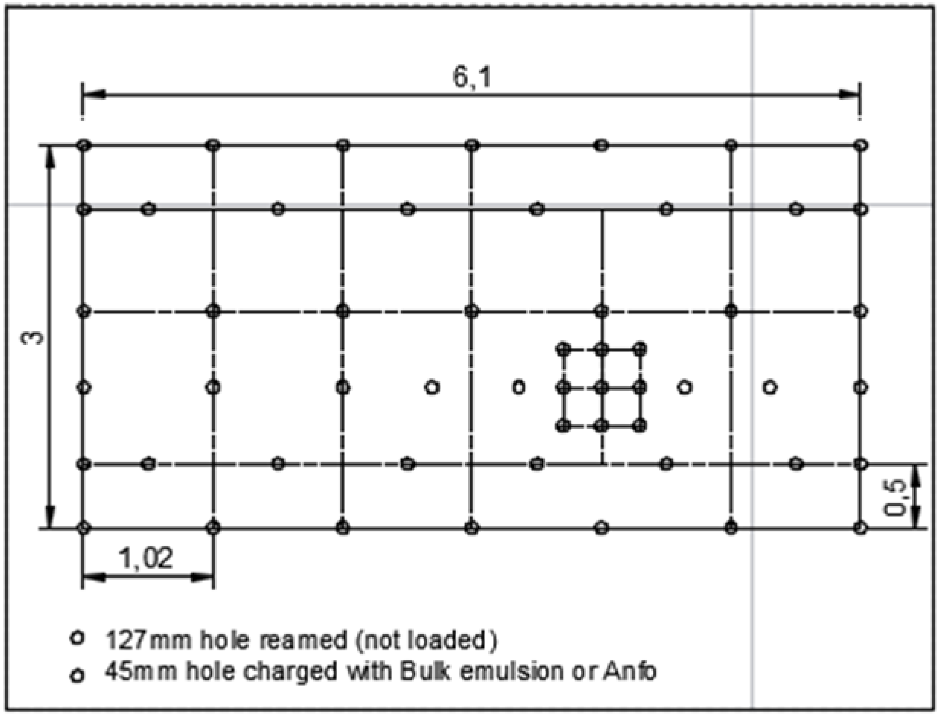
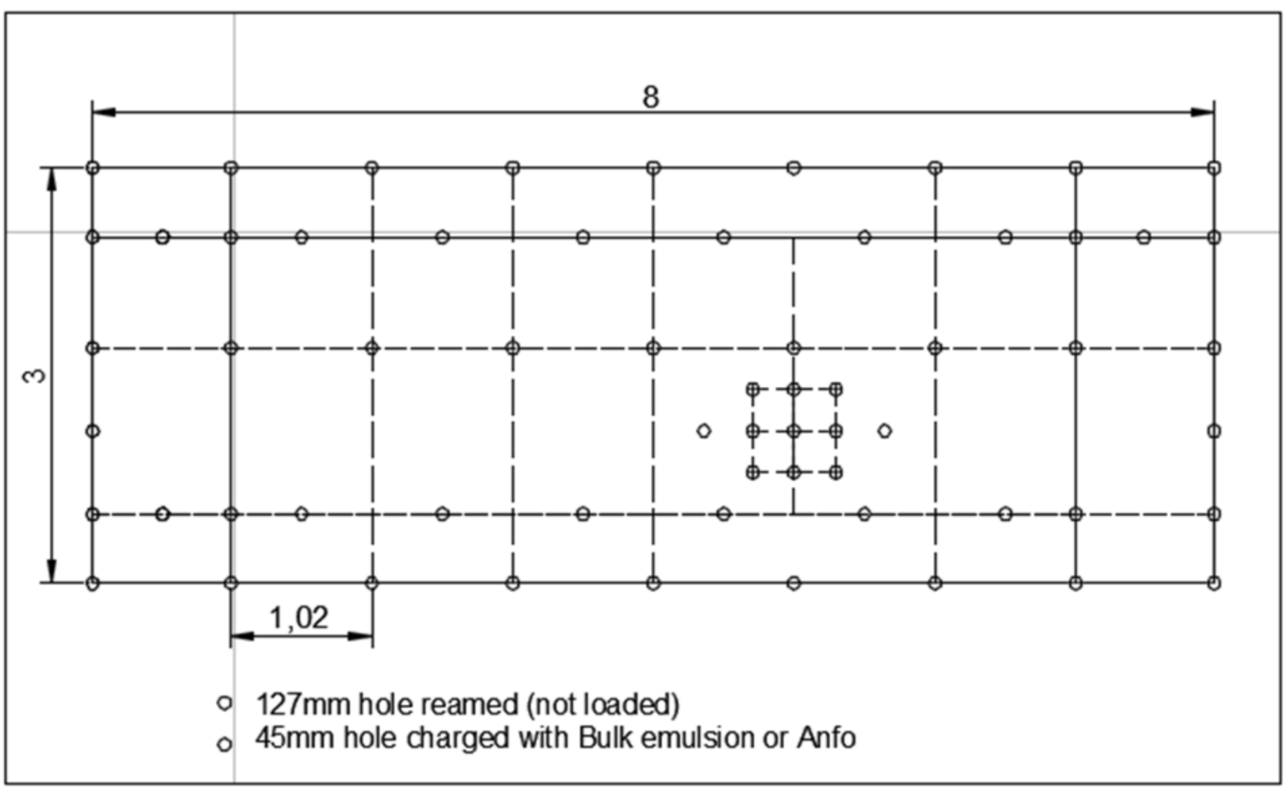


Figure 16.8: Drilling Pattern for the Production Room



Once the area mining is completed, the maximum amount of ore will be recovered from the barrier and drift

Blasting crews will load the rounds with explosives and initiate blasts at the end of each shift. Explosives will consist of a mixture of ANFO and emulsion where there is presence of water. A decoupled explosive charge is recommended to presplit the back of the room. Control of drilling and blasting is very important for the White Pine North project. The perimeter control of the drilling should allow to reduce the dilution to a minimum. Historically the dilution was 8% for the White Pine Mine. We used this same dilution percentage for this Report.

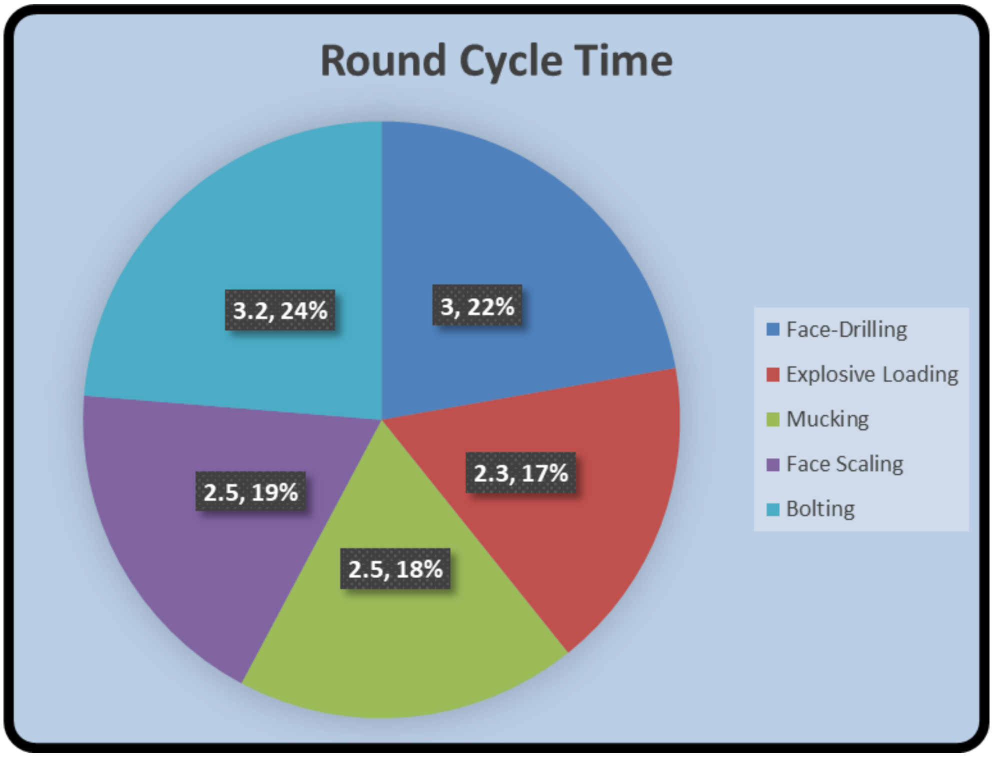
The blasting of the loaded round will be performed at the end of every shift. A period of 2 hours is planned between shifts to vent blasting fumes from the mine. The main access and ventilation raises will be monitored with gas detectors.

The third mining activity is to muck the blasted ore from the face and to transport it with a low-profile 10t LHD. The performance of the LHD is a function of the dip of the stope and the distance between the face heading and the rock breaker. The LHD performance will vary from 3.9 kilometres per hour (“km/h”) at 17% (loaded) to 8.9 km/h at -17% (unloaded). To reduce the haulage distance, the unloading point will be moved regularly to be normally less than 250 m from the working face. however, a case-by-case evaluation was made for each of the planned rock breaker moves, to justify economically this displacement. For the economic evaluation of the project, the average hauling distance was calculated for each of the planned rock breaker positions. For operating cost calculations, a capacity of 9.12 t per bucket is used which considers the fill factor and the loading equipment.

The next step in the mining cycle is to scale the back and wall of the excavation by using a smaller low-profile LHD equipped with a scaling arm. the LHD's arm repeatedly rubs the roof and wall of the drift to remove the loose rock. This method was used at the old White Pine Mine and is very effective in sedimentary (stratified) rock.

A low-profile rock bolter is used to install the roof and wall support. In the room excavation 1.8 m rebar bolts are required according to a 1.2 m x 1.2 m pattern currently in this Study; no wire mesh is planned. However, in the more problematic areas, a wire mesh may possibly be installed. The drilling performance of the bolter is estimated at 2 m/s. The total round cycle time is estimated at an average of 13.5 hrs/round.

Figure16.9: Round Cycle Time



Once the area mining is completed, the maximum amount of ore will be recovered from the barrier and drift pillars as illustrated in Figure 16.10 and Figure 16.11.

Figure 16.10: Drilling Pattern Production Room

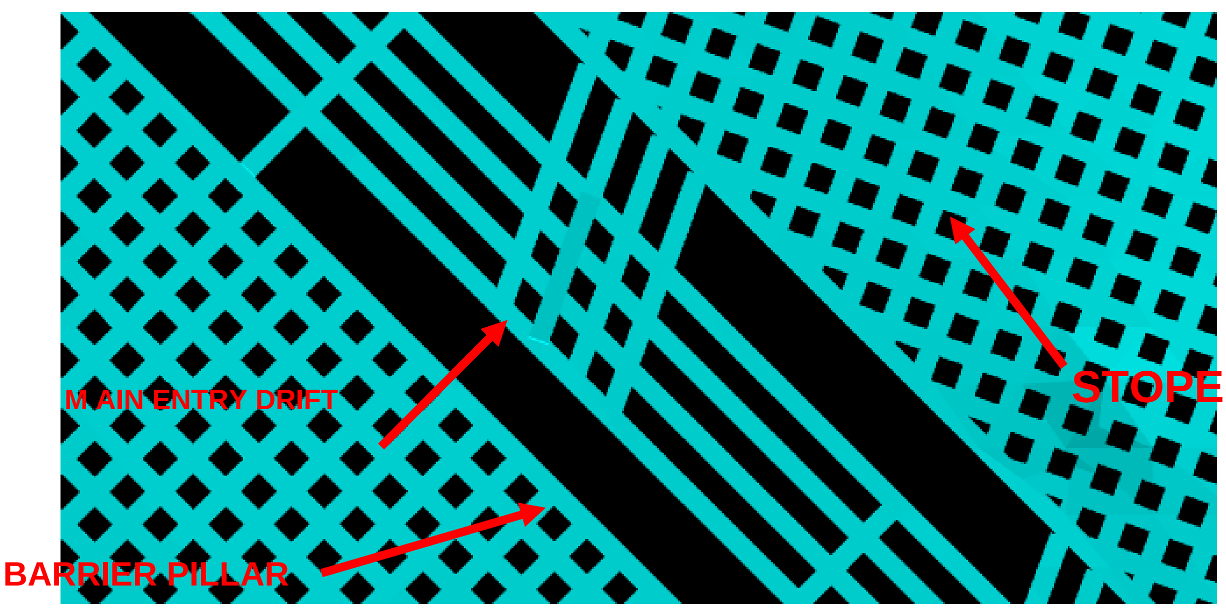
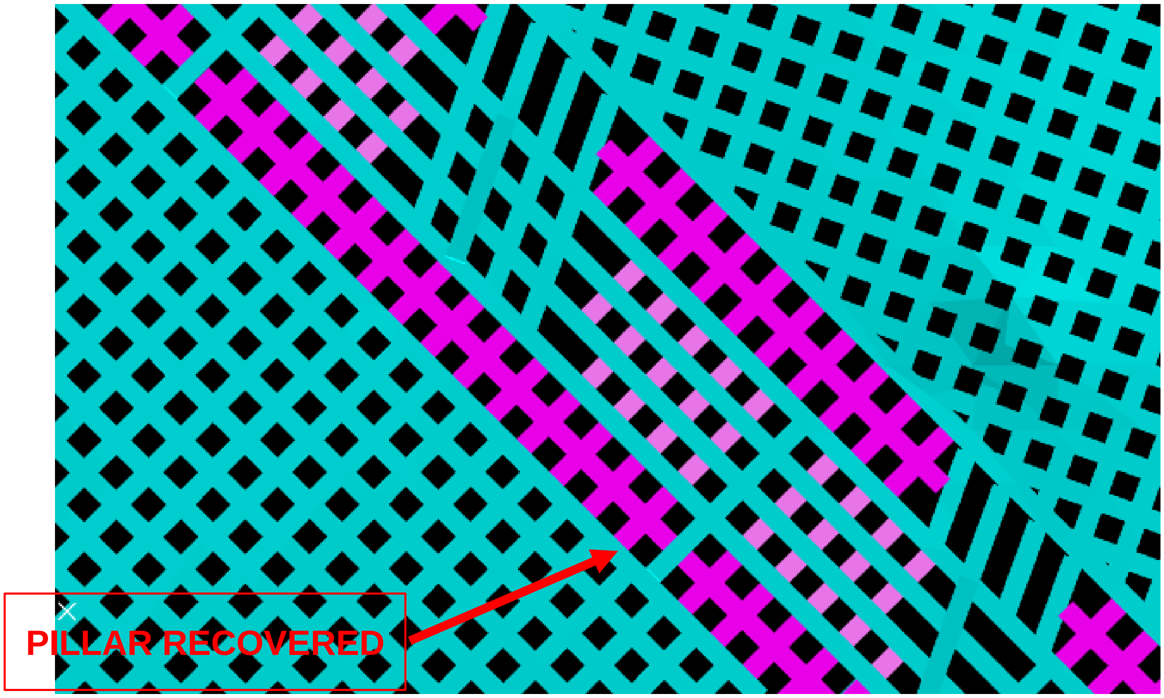


Figure 16.11: Drift, Room and Pillar after Recovery



16.3.2 Mining Parameters

The basic operational assumptions are summarized below:

- Minimum mining height 2.1 m (limited by the equipment);
- Maximum mining height 7.0 m;
- Average mining height 3 m;
- First pass recovery 40%
- Second pass mining recovery% = 5219 x depth (m) - 0.145;
- Cut-off grade 0.9% Cu;
- Annual production - 5.4 Mt;
- Entry drift (main access) 6.1 m;
- Mining room width 8 m;
- Old mine safety pillar 300 m;
- Fresh air raises 5 m;

- Exhaust air raise 5 m;
- Conveyor maximum optimal distance to the face heading 250 m;
- Minimum of 12 rooms per operating panel.

16.3.3 Ore Handling System

The broken ore from the development headings will be mucked by a 10 t low-profile LHD to temporary re-muck bays located up to 200 m from the face, and then hauled by 30 t low-profile trucks to the surface or to a rock breaker loading point. The broken ore from the stope will be mucked by a low-profile LHD to a stope dumping point. The stope dumping point is a system composed of a grizzly, a rock breaker and loading points to the conveyor system. This system will be installed in every production panel and can be moved when the faces are too far apart. The parallel bar grizzly with 200 mm openings prevent oversize material from entering the conveyor system. The hydraulic rock breaker will be used to break oversized material on the grizzlies. The hydraulic rock breaker will be remotely controlled from the surface by an operator. The present study presumes that one operator can operate 4 rock breakers from surface. The ore will be transferred on the stope conveyor. The 42 in wide belt stope conveyor, comprised of a 500 HP motor can be extended depending on the progress of the stope. It is currently planned to advance these conveyors every 250 m according to the progression of the stope. The broken ore is then transferred to the principal conveyor located in the main drift conveyor.

16.3.4 Mining Equipment

Table 16.4 shows the equipment requirements to support the planned 15,000 tpd nominal production rate.

Table 16.4: Mine Equipment Requirements

Mobile Equipment	
Low-Profile 2 Booms Jumbo Drill	15
Low Profile 1 Boom Electric-Hydraulic Bolter	17
Low Profile LHD 10 Mt	17
Low Profile LHD 8 Mt	3
Scaler	5
Development Truck	7
Lube Trucks	2
Flat Bed Trucks	2
Scissor Lift	13
Grader	1
Tractor -Underground	32
ATV -Underground	20
Cable Bolt Drill Stope Mate Drill	1
Ore Handling System	
Loading Point+ Rock Breaker	13
Main Conveyor & Chute (650 m)	1
Main Conveyor & Chute (2,900 m)	3
Main Conveyor & Chute(1,300 m)	4
Main Conveyor & Chute (XXX)	6
Stope Conveyor 500 m – 500 HP	30
Electric-Sumps-Pumps	14
Orca Series Station	7
3" Versa-Matic Pump	10
Production Panel Auxiliary Fan	22
15 MBTU Pre-Production Propane-Heater	1
Preproduction Fan	1
Main Ventilation Fan	1
Main Ventilation Propane-Heater	1
Other	
Shotcrete Machine	1
Communication System	1

16.4 Development Schedule

Development will be divided into two periods: a pre-production development period (from the beginning to the Q3 2023) and a production period (from the Q3 2023 to the end).

16.4.1 Pre-production Objectives

- Achieve early production from the west part of the mine;
- Provide access for equipment;
- Provide ventilation and emergency egress;
- Establish ore handling systems;
- Install first mining services (power distribution, IT communications system, dewatering system, compressed air and water supply);
- Develop sufficient production panels to support the mine production rate.

16.4.2 Production

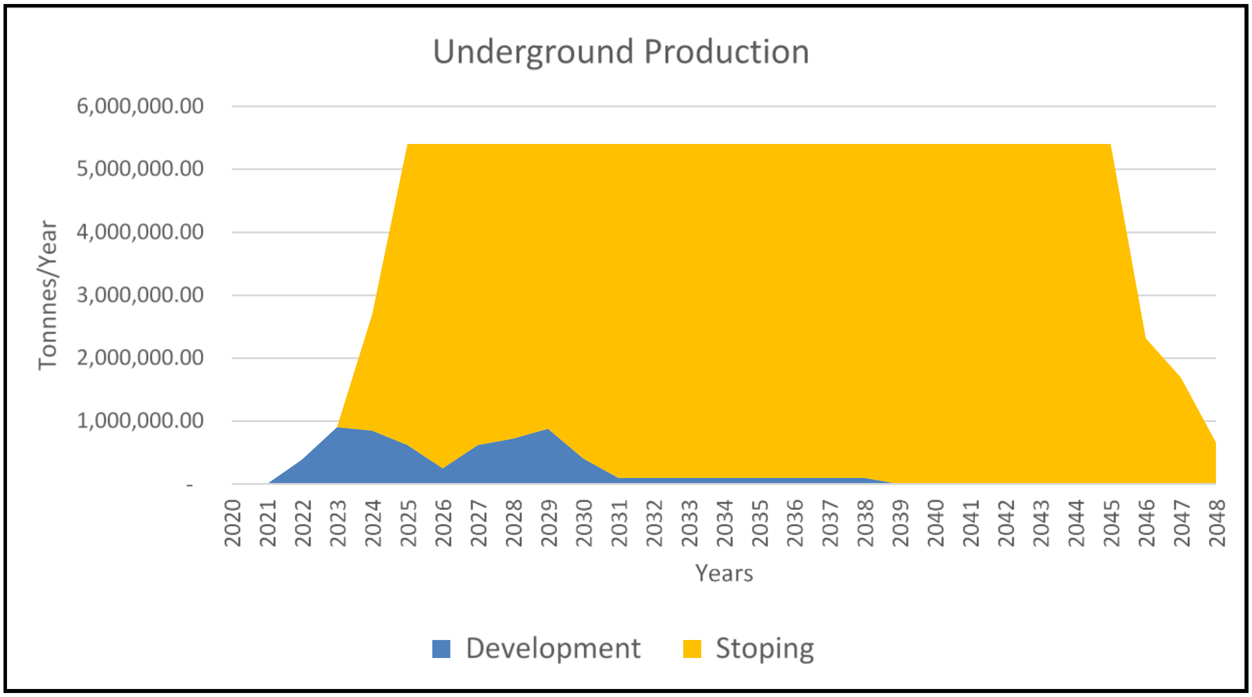
It was assumed that pre-production and production drift development will be excavated by the Owner's mining department. The owner approach is preferred to reduce development costs, mining contractors typically do not have low profile equipment. Once the portal is built, development of the main access drifts will get underway. The production of the 2 main access drifts from the portal will be 5 m/d. Once the main access drift divides into 4 drifts, production will increase to 10 m/d. As soon as a new heading is available, a new team will be added to reach a maximum of 3 development teams. As soon as the first stope is developed, production can commence. Development will continue at a reduced pace once production begins. Elaboration of the vertical and inclined ventilation raises will be performed by the contractor’s raise-boring crew. Raise development was used in the elaboration of the mine schedule. It was assumed that a raise-boring crew can drive the raise at an advance rate of 90 m/mo. It was estimated that all pre-production development will be completed in Q4 2023.

16.5 Production Schedule

The production schedule is based on mining a fixed target of 5.4 Mt/yr. To achieve this annual production, seven to fourteen production panels must be in production simultaneously. The number of required panels depends on the tonnage from the development, as well as the height of the rooms of each panel.

In Q1 2024, the first stope will begin to reach a production rate of 7,500 t/d on 2024. Before January 2024 the difference between the daily underground production and the daily mill production will come from the surface stockpile accumulated during the pre-production period. In the pre-production period, the priority is to start the production from the West sector of the mine.

Figure 16.12: Production Schedule



The stoping productivity varies for each stope depending on the mining height. For the minimum stope height of 2.1 m the production rate is estimated at 981 tpd and can reach up to 1,367 tpd for a 3.9 m high stope. The limit of stope production is the productivity of the jumbo drill.

Table 16.5: Productivity per Mining Panel

Panel Height	Panel Productivity tpd
2.1	981
2.3	1040
2.5	1,095
2.7	1137
2.9	1184
3.1	1218
3.3	1,262
3.5	1302
3.7	1330
3.9	1367

Table 16.6: Mine Production Schedule Summary

			Pre-Production				Production								
Development Mining		**Total**	**2020**	**2021**	**2022**	**2023**	**2024**	**2025**	**2026**	**2027**	**2028**	**2029-3033**	**2029-3034**	**2029-3035**	**2029-3036**
Tonnage	kt	6,444	-	-	389	905	850	621	252	620	721	1,587	499	-	-
Cu Head Grade	% Cu	0.70	-	-	0.01	0.79	0.78	0.79	0.82	0.82	0.68	0.72	0.57	-	-
Ag Head Grade	g/t	7.74	-	-	7.67	8.61	9.15	8.94	8.07	9.18	6.92	6.35	5.91	-	-
Cu Contained Metal	kt	99,324	-	-	69	15,752	14,640	10,853	4,569	11,177	10,819	25,172	6,273	-	-
Ag Contained Metal	k oz	1,603	-	-	96	251	250	178	65	183	160	324	95	-	-
Production Mining															
Tonnage	kt	114,961	-	-	-	-	1,850	4,779	5,148	4,780	4,679	25,413	26,501	27,000	14,811
Cu Head Grade	% Cu	0.99	-	-	-	-	1.03	1.03	1.05	1.04	1.03	0.99	0.99	0.95	1.02
Ag Head Grade	g/t	12.03	-	-	-	-	15.60	15.28	14.53	15.20	16.71	14.91	11.78	8.66	8.81
Cu Contained Metal	kt	2,512,951	-	-	-	-	42,081	108,643	118,947	109,446	106,720	556,280	576,192	562,990	331,651
Ag Contained Metal	k oz	44,465	-	-	-	-	928	2,348	2,406	2,335	2,514	12,183	10,040	7,518	4,193
Total Mining															
Tonnage	kt	121,405	-	-	389	905	2,700	5,400	5,400	5,400	5,400	27,000	27,000	27,000	14,811
Cu Head Grade	% Cu	0.98	-	-	0.01	0.79	0.95	1.00	1.04	1.01	0.99	0.98	0.98	0.95	1.02
Ag Head Grade	g/t	11.80	-	-	7.67	8.61	13.57	14.55	14.23	14.50	15.41	14.41	11.68	8.66	8.81
Cu Contained Metal	kt	2,612,275	-	-	69	15,752	56,721	119,496	123,517	120,623	117,539	581,452	582,465	562,990	331,651
Ag Contained Metal	k oz	46,068	-	-	96	251	1,178	2,526	2,471	2,518	2,675	12,507	10,135	7,518	4,193

Table 16.7: Operating Shift Assumptions

OPERATING PARAMETERS	
Days in period	365
Shifts per day	2
Hours per shift	10
Total hours/year	7300
Total days lost/ year	5
Total days operated/year	360
Scheduled Hours/year	7200
Equivalent scheduled shifts	720
Shift Composition (minutes)	
Travelling to work place	30
Workplace inspection	15
Equipment inspection/set-up	15
Lunch (+ travel to and back)	45
Supervision	15
Operation delays	30
Travelling to surface	30
Change	0
Total time loss (minutes/shift)	180
Total time loss (hours/year)	2160
Jumbos availability	85%
Jumbos available hour	6120
Utilization %	65%
JUMBO operating hour	3960

16.6 Manpower and Working Schedule

Labor levels are estimated based on the production schedule and equipment requirements to reach a production level of 5.4 Mt/yr. To achieve the level of productivities used in this study, the workforce must be a mix of skilled labor with an experienced management team. The mine work schedule is based on working two shifts per day, seven days per week, 360 days per year. A rotation schedule of 7 days in and 7 days out has been selected for mine operation requirements, with rotation days and nights.

Several mine services will however be on a 5-2 schedule of 5 or 7 days in and 7 days out on day shifts only. Table 16.8 represents the different schedules for the underground mining operation

Table 16.8: Production Working Schedule

Grade	Job Title	Schedule	Hours Worked
Supervision			
Staff	Mine Manager	5-2	2,080
Staff	Mine Ops. Superintendent	5-2	2,080
Staff	Mine Secretary	5-2	2,080
Staff	Mine Captain	5-2	2,080
Staff	Mine-Ops-Foreman	7-7 on 2 Shifts	1,825
Staff	Mine-Ops-Trainer	7-7 on 1 Shift	1,825
Mine Operations			
Hourly Class 1	Jumbo Operator + Bolter +LHD Operator + Truck Operator	7-7 on 2 Shifts	1,825
Hourly Class 1	Grader Operator	5-2	2,080
Hourly Class 1	Loading Point Operator	7-7 on 2 Shifts	1,825
Hourly Class 1	U/G Construction Maintenance + Material Handling + Ventilation Crew	7-7 on 1 Shift	1,825
Hourly	Conveyor Service Man	7-7 on 1 Shift	1,825
Hourly	Labour, Fryman, Drill Bits Sharpener	7-7 on 1 Shift	1,825
Technical Services			
Staff	Chief Mine Engineer + Chief Geologist	5-2	2,080
Staff	Engineer and Geologist	5-2	2,080
Staff	Mine technician and Senior Surveyor	5-2	2,080
Staff	Surveyor + Geology Technician + Geotech Technician	7-7 on 1 Shift	1,825
Mechanical - Electrical Services			
Staff	Mechanical and Electrical Superintendent	5-2	2080
Staff	General Foreman + Planner + Engineers	5-2	2080
Staff	Supervisor	7-7 on 2 Shifts	1825
Hourly Class 1	Fixed and Mobile Mechanics	7-7 on 2 Shifts	1825
Hourly Class 1	Electricians	7-7 on 2 Shifts	1825
Hourly Class 1	Technician and Electronics Technician	7-7 on 2 Shifts	1825

No allowance has been made for absenteeism, sickness, snow days, or dumped shifts. Holidays and vacation expenses are covered in the fringe benefit allowance.

Table 16.9: Mine Manpower Requirements Production

Department	Manpower Year 2029
Mine Supervision	41
Mine Operation	320
Mine Services	85
Technical Services	39
Mechanical Services	98
Electrical Services	58
Total	641

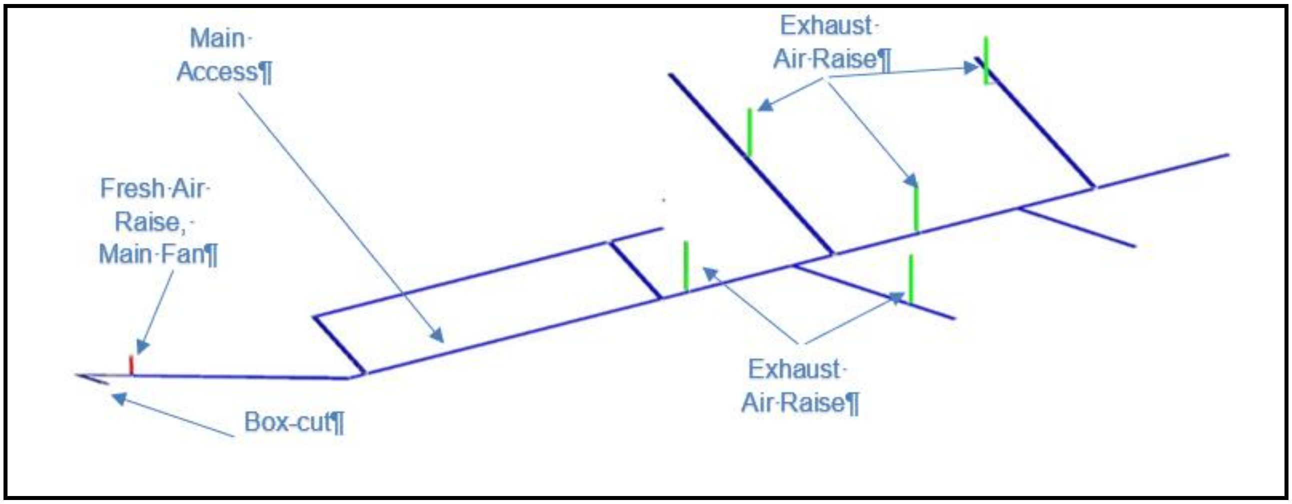
16.7 Mine Services

16.7.1 Ventilation

During the pre-production period, air requirements will be supplied through two 300 HP 1.4 m diameter parallel Van axial fans at surface. The two fans will be installed on a metallic stand and connected with a vent tube directed to the portal. The two fans in parallel will generate approximately 55 m^3/s each at 2.5 kPa of water gauge. These two fans will be used until the main fan intake is commissioned. The fresh air will circulate in two of the main drifts, and the exhaust air will be returned to the surface in the two other drifts.

The ventilation system will consist of a push system whereby two 1250 HP parallel main fans will be installed at surface providing approximately 225 m^3/s each at 6.0 kPa. The two main fans will be installed and provide heated air through a 5 m ventilation raise and air will be distributed throughout the mine using ventilation regulators, auxiliary fans, doors and bulkheads. Also included are five 5 m diameter exhaust ventilation raises distributed in the operating mine. Emergency egress is to be installed in the fresh air raise. A 125 cfm/hp factor was used to estimate ventilation requirements if the equipment was not MSHA approved.

Figure 16.13: Ventilation Layout during Production Period



16.7.2 Water Supply

Water is required underground for drilling and controlling dust and must also be available for fire protection. Water will be distributed underground by a 4 in steel pipe schedule 40 in the main access drift and 2 in light wall steel pipe in the stopes. This pipe size will provide adequate quantity and pressure to meet dust control and fire protection requirements.

Table 16.10: Equipment Water Consumption

Underground Water	Consumption (l/min)	Use (eff. time)
Washing Working Faces	15	5%
Jumbo Drilling	40	65%
Bolters	45	65%
Cable Bolters	45	15%
Shotcrete Machines	45	35%
Diamond Drilling	60	0%
Raise Boring Machines	65	25%
Feeder-Breakers	0	85%
Wetting Muck Piles	5	85%
Dust Suppression	25	50%

16.7.3 Power

Major electrical power consumption in the mine will be required for the following equipment:

- Main and auxiliary ventilation fans;
- Main conveyor system;
- Stope conveyor system and rock breaker-loading points;
- Jumbo and bolter equipment;
- Mine dewatering pumps.

A high voltage cable (13.8 kV) will be installed in the conveyor drift access. This high voltage cable will connect to a substation in each production panel which will drop the voltage to 600 V for the operation's electrical requirements.

16.7.4 Dewatering

Water in the mine will emanate from the underground water inflow and mining operations (total of 6,300 l/min). The dewatering system will pump commonly called ''dirty water''. This water will be cleaned and sent to sedimentation ponds at the surface preventing mining operations from cleaning sumps underground. Pumping stations have been designed to operate 50% of the time, allowing at least double the maximum required capacity. The White Pine dewatering system consists of seven permanent pumping stations (Figure 16.14). The main pumping station is PU1, pumping all underground water towards the surface; it receives water from PU2, PU3. Pump PU2 receives water from mining panels around and from the pump PU4. Pump PU4 receives water from mining panels around and from pumps PU5 and PU6. Pump PU6 receives water from mining panels around and from pump PU7. Pump PU5 and PU7 are pumping only the water coming from the mining panel around.

Figure 16.14: Pump Localization

16.7.5 <u>Compressed Air</u>

Compressed air supply will be provided by electric compressors installed temporarily for the pre-production period. For the production period compressed air supply will be provided by 1,200 cfm electric compressors. The compressed air piping network will be installed along the main access consisting of an 8 in diameter steel pipe. A smaller 4 in line will be installed in the production panel in the main room. Compressed air will provide power to a small pump for dewatering development work. Handheld drills will also provide an emergency supply of air to the refuge station.

16.7.6 <u>Fuel Storage and Distribution</u>

The haulage trucks and all auxiliary vehicles will be fuelled at surface fuel stations. Two fuel/lube cassette trucks will be used to distribute the fuel underground to the LHD, jumbo, bolter and scissor lift equipment.

16.7.7 <u>Communications</u>

The mine's communication system will consist of a LTE communication system. Telephones will be located at key infrastructure locations such as the refuge and lunchrooms. Key personnel (mobile mechanics, crew leaders, and shift bosses) and mobile equipment operators (LHD, truck, grader and utility vehicle operators) will be supplied with an underground radio connected to the LTE network. This system also makes it possible to transmit the necessary data for the teleoperation of certain equipment.

16.7.8 Explosives Storage and Handling

During the pre-production and the start of the production, the explosives will be stored at the surface in permanent magazines. The accessories (detonators) will be stored in a separate magazine at the surface. Once panel rooms become available, an underground explosive and detonator magazine will be prepared. The Study provides for two underground explosives. One at the West sector of the mine and the other to at the East. Explosives will be transported from the surface magazine to the underground magazine by flat bed service trucks. ANFO will be used as the major explosive for mine development and production. Packaged emulsion will be used as a primer, lifter holes and pre-split blasting.

16.7.9 Personnel and Underground Material Transportation

Supplies and personnel will access the underground via the main access drift. A series of farm tractors modified for the underground will be used to shuttle men from surface to the underground. Supervisors, engineers, geologists will use diesel-powered ATVs for transportation underground. Mechanical and electricians will use maintenance farm tractors. A flat bed with a service boom will be used to move supplies from the surface to the underground active panel. Service LHDs with forks will be used for material transportation.

16.7.10 Underground Construction and Mine Maintenance

Several crews will be assigned to mine construction and maintenance. Teams will be assigned to maintain ventilation fans, mine brattice and other installations to allow for a good ventilation of the work areas. Another team will be assigned to the maintenance and installation of the conveyors. This team will install the main conveyor, stope conveyor, extend the stope conveyor, move them as needed and provide for their maintenance. Another team would be used to do the remaining underground construction, which includes the shotcrete wall construction and any other construction work. Another team will be used to transport underground material with flatbed trucks and fuel with fuel-lube truck.

16.7.11 Equipment Maintenance

All major mechanical maintenance will be performed in an underground maintenance workshop. The shop has not been positioned yet as it is planned to use old excavation to do the maintenance

16.8 Safety Measures

16.8.1 Industrial Hygiene

All employees will perform a health test: audiogram, breath, etc.; to allow the Company to follow their conditions during their tenure at the mine and apply adequate accident prevention programs.

16.8.2 Emergency Exits

Emergency exits underground will consist of the portal ramp, fresh air ventilation raises and manways. The underground alarm system will have a radio alert signal to all the workforce simultaneously when Mercaptan stench gas is introduced in the ventilation system to alert employees they need to reach for safety. Pursuant to *Regulation 57.4363*, underground workers need to be drilled every 12 months on emergency exit underground requirements. Pursuant to *Regulation 57,4361*, mine evacuation drills shall be held every 6 months for each shift. All exercises and instruction records will be kept at least one year.

16.8.3 Refuge Stations

Refuge stations are positioned in a way that an employee will need 30 min of less to access the refuge from the moment he leaves his workplace. At White Pine North, both moving and permanent refuge stations will be installed to be airtight and fire resistant. Two permanent and six moving refuges are planned for the White Pine North LOM. Each refuge station will be equipped with the following:

- Telephone or radio to surface, independent of mine power supply;
- Compressed air, water lines and water supply;
- Emergency lightning;
- Hand tools and sealing material;
- Plan of underground work showing all exits and the ventilation plans.

16.8.4 Fire Protection

Underground mobile vehicles and conveyor belts will be equipped with automatic fire suppression systems in accordance with regulations.

Fire extinguishers will be provided and maintained in accordance with regulations and best practices at the electrical installations, pump stations, conveyors, service garages and wherever a fire hazard exists. Every vehicle will carry at least one fire extinguisher of adequate size and proper type.

A mine stench gas warning system will be installed at the ventilation and compressed air system to alert underground workers in the event of an emergency.

16.8.5 Mine Rescue

Fully trained and equipped mine rescue teams will be established in accordance with regulations. A mine rescue room will be provided in the administration building. Mine rescue equipment and a foam generator will be located on site. The mine rescue teams will be trained for surface and underground emergencies. An Emergency Response Plan will be developed, kept up to date, and followed in the event of an emergency.

16.8.6 Emergency Stench System

A mine stench gas warning system will be installed at the ventilation (temporary and permanent system) and compressed air system to alert underground workers in the event of an emergency.

16.8.7 Dust Control

Broken ore will be wet down after blasting and mucking.

17. RECOVERY METHODS

17.1 Process Design

The process plant design for the White Pine Project ("The Project") is based on a simplified metallurgical flowsheet designed to produce copper concentrate. The flowsheet is based on well proven unit operations in the industry, as well as in the White Pine Column Cell Conversion memorandum from Ronald M. Woody, White Pine Mill Superintendent dated 1991.

The key criteria for equipment selection are suitability for duty, reliability and ease of maintenance. The plant layout provides ease of access to all equipment for operating and maintenance requirements whilst maintaining a layout that will facilitate construction progress in multiple areas concurrently.

The key project design criteria for the plant are:

- Nominal throughput of 15,000 tonnes per day ("tpd") ore;
- Process plant availability of 91.3% through the use of standby equipment in critical areas and reliable power supply;
- Adequate automated plant control to minimize the need for continuous operator interface and allow manual override and control when required.

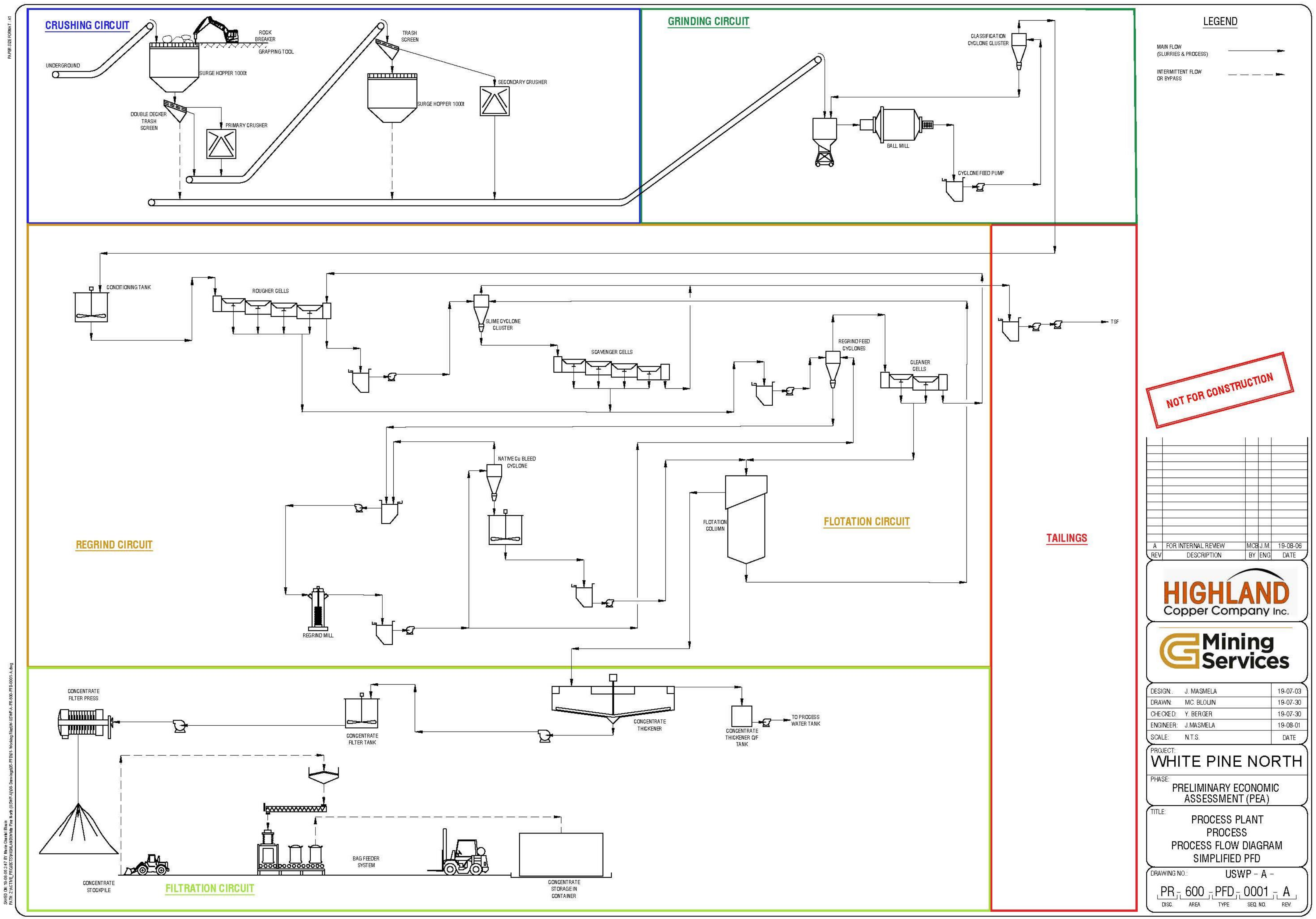
17.1.1 Selected Process Flowsheet

The process plant has been designed for a throughput of 15,000 tpd (dry). The overall flowsheet includes the following steps:

- Crushing, grinding and classification;
- Rougher flotation;
- Rougher concentrate regrinding;
- Cleaner flotation, using two stages of cleaning with flotation cells and columns;
- Concentrate thickening and filtration;
- Tailings pumping and disposal in the common Tailings Disposal Facility ("TDF");

Figure 17.1 presents a simplified flow diagram depicting the major unit operations incorporated on the selected process flowsheet.

Figure 17.1: Overall Process Flow Diagram



17.1.2 Key Process Design Criteria

The key process design criteria were listed mainly in the Column Cell conversion report and formed the basis of the preliminary process design criteria and mechanical equipment list. Additional metallurgical test work shall confirm the number of flotation stages as well as the use of flotation cells or columns. In addition, flotation residence time, flowsheet configuration and reagents may need adjustments according to the following metallurgical test results.

Table 17.1: Key Process Design Criteria

Parameter	Units	Value
Plant Throughput	tpd	15,000
Head Grade - LOM	% Cu	1.0
Head Grade – Silver (Ag)	g/t	10
Plant Availability	%	91.3
Crushing Work Index (CWi)	kWh/t	12.2
Bond Ball Mill Work Index	kWh/t	14.2
Plant Operating Time	hr	8,000
Grind Size (P_{80})	µm	106
Rougher Conditioning Time - Laboratory	min	5
Rougher Residence Time - Laboratory	min	30
Scavenger Residence Time - Laboratory	min	10
Cleaner 1 Residence Time - Laboratory	min	6
Regrind Mill Product Size (P_{80})	µm	20
Target Concentrate Grade	% Cu	31.5
Target Overall Recovery	%	88

17.2 General Process Description

The process plant has been designed for a through put of 15,000 tpd (dry). The overall flowsheet includes the following steps:

- Crushed ore and conveying;
- Grinding and classification;
- Rougher flotation;

- Rougher concentrate regrind;
- Cleaner flotation, using three stages of cleaning;
- Concentrate thickening and filtration;
- Tailings disposal.

17.3 Crushed Ore Reclaim

Ore from the underground mine will be crushed down to 150 mm in order to be conveyed to a transfer conveyor equipped with a weightometer. Ore is received on surface into two 1,500 live ton coarse ore bins. Initial processing plant will include a Crushing Circuit to reduce ore size from 150 millimeters ("mm") to P80 = 16 mm. To achieve these results, there is a series of scalping screens and crushers to process ore. The ore is first screened over double deck scalping screens. Screen oversize is fed to a standard crusher operating at about 30 mm closed-side setting. Scalping screen undersize is sent directly to the fine ore bin while the intermediate screen product is combined with the standard crusher discharge and conveyed a 1,500 live ton feed bin. Shorthead feed is processed through screening and crushing line including double deck screens and a shorthead crusher which are set at a 6 mm closed-side setting and operate in open circuit. Crushing plant product is nominally minus 19 mm with an 80% passing size of 16 mm.

17.4 Grinding and Classification Circuit

The grinding circuit will receive ore at a nominal top size of 19 mm with an 80% passing size of 16 mm. The circuit will consist of a Ball mill in closed circuit with a cyclone cluster. The grinding line is fed from a dedicated 1,500 live ton fine ore bin and consists of one Ball mill. The Ball mill will be a 5.80 m diameter x 9.86 m EGL overflow mill, with a 5,500 kW fixed speed motor. The mill will operate with between 30 and 35% ball charge. The cyclone feed pumps will deliver slurry to the cyclone cluster where will be classified. Cyclone underflow will be directed to the ball mill, while cyclone overflow will gravitate back to the flotation conditioning tank. Product from the ball mill will discharge over the cyclone feed pump box. again. Product size is set to feed the flotation circuit at 80% passing size of 106 microns.

Two vertical spindle sump pumps will service the grinding and classification area. The concrete floor under the mill area will slope to the sumps to facilitate cleanup. Grinding media for the mills will be introduced by use of a dedicated kibble.

17.5 Rougher Flotation

Flotation feed will pass through the trash screen designed to remove foreign material prior to flotation. Trash will report to the trash bin which will be periodically emptied. Screen undersize will gravitate to the rougher conditioner tank. A sampler will be installed on the screen underflow line to take a sample to the On-stream Analyzer ("OSA") for metallurgical, process control and particle size measurement purposes.

Frother and other flotation reagents will be added into the rougher conditioner tank. Process water can be added if required to dilute the feed to the appropriate slurry density.

The rougher flotation cells will consist of eight 100 m^3 forced air tank cells in series. Rougher concentrate will flow to the regrind cyclone feed hopper. A sampler will be installed on the rougher concentrate discharge line to take a sample to the OSA for process control purposes.

The rougher tailings will gravitate to the Roughers flotation tails pump box and a sampler will be installed to take a sample to the OSA for metallurgical and process control purposes. The roughers tails will be pumped to the Slime Cyclone Cluster; the underflow will proceed to eight 100 m^3 forced air Scavengers tank cells in series. The Scavenger concentrate will also flow to the regrind cyclone feed hopper, to be combined with the Roughers concentrate. A sampler will be installed on the Scavengers concentrate discharge line to take a sample to the OSA for process control purposes; the Scavengers tailings will gravitate to the flotation tails pump box and a sampler will be installed to take a sample to the OSA for metallurgical and process control purposes.

A distribution system to dose reagents along the Rougher and Scavengers flotation cells train will be provided so that stage collector and frother can be added if required.

The flotation building overhead crane will be used for all maintenance lifting functions within the flotation area. A vertical spindle sump pump will service this area for spillage cleanup.

17.6 Regrind

Rougher and Scavenger concentrate will report to the regrind cyclone feed pump box. The slurry will be pumped to the regrind cyclone cluster by the regrind cyclone feed pumps. The cyclone underflow will gravitate to the regrind mill where water and lime (if required) will be added to achieve the milling density and desired operating pH. The regrind mill will be a vertical mill and grinding will be achieved via attrition and abrasion of the particles in contact with steel media. Mill discharge will gravitate back to the regrind cyclone feed hopper for classification in the regrind cyclones.

Regrind cyclone overflow will gravitate to the cleaner conditioner tank. A sampler will be installed on the cyclone overflow line to take a sample to the OSA for process control and particle size measurement purposes.

Media will be introduced via the regrind media hopper. A vertical spindle sump pump will service this area for spillage cleanup.

17.7 Cleaner Flotation

Final arrangement regarding recirculation of cleaning streams will be made according to additional testwork program. The final arrangement includes recirculation of the first cleaner scavenger concentrate to the regrinding/first cleaner circuit and tailings to the rougher last cells.

Regrind cyclone overflow will proceed to the cleaner conditioning tank, where reagents will be added to this tank. The facility to add process water to dilute the slurry to the desired density will also be provided.

The first cleaner flotation cells will consist of six 20 m^3 trough cells in series. The first cleaner concentrate will be pumped to the second cleaner flotation columns, a sampler will be installed on the discharge line of the pump to take a sample to the OSA for process control purposes. The first cleaner tailings will be pumped back to the rougher flotation circuit.

The Cleaner flotation Columns will consist of six 240 m^3 columns in series. Frother will be added to the feed box. Column concentrate will be collected in a pump box and be pumped to the concentrate thickener. Flotation Columns tailings will gravitate to a pump box from where the material is pumped to the Slime Cyclone Cluster. A sampler will be installed on this stream to take a sample to the OSA for metallurgical and process control purposes.

Two vertical spindle sump pumps will service the cleaner flotation area for spillage clean-up.

17.8 Concentrate Thickening and Filtration

Final concentrate will be pumped to the high rate concentrate thickener, along with filtrate return from the filtration area. Flocculant stock solution will be further diluted to 0.25% w/w with process water in an in-line mixer prior to addition to the concentrate thickener. Thickener overflow will gravitate to the process water tank for re-use.

Concentrate thickener underflow, at approximately 60% solids w/w, will be pumped to the agitated concentrate filter feed tank by one operating, with one standby, concentrate thickener underflow pump. This tank will provide 12 hours of surge capacity between the thickener and filter.

Thickened concentrate will be pumped batch wise to the concentrate filter press using one operating, and one standby, filter feed pumps. The filter will remove water from the concentrate to meet the target moisture of approximately 9% w/w using a series of pressing and air blowing steps. After the desired filtration time of approximately 12 minutes, the filter press will open and discharge concentrate directly to the floor of the concentrate shed. Following discharge of concentrate, the filter cloth will be washed prior to the next cycle using raw water. Some 9.9 m^3/h filtrate from the concentrate filter will be returned to the concentrate thickener by gravity. Filter cloth wash will be drained into the filter area sump pump.

A front-end loader ("FEL") will be used to remove the concentrate from beneath the filter press and transfer it to the adjacent 542 t concentrate storage areas. Concentrates will be loaded into the loadout hopper by the FEL when required. Concentrate from the load-out hopper will be transferred to the concentrate trucks via a 900 mm wide concentrate feeder and 750 mm wide truck loading conveyor. The truck loading conveyor will be equipped with a weightometer.

Two vertical spindle sump pumps will be provided in the thickener and filtration area to return spillage to the concentrate thickener.

17.9 Tailings Handling

Slimes overflow and scavenger tailings will be combined in a mixing box from where a final sampler will take a sample to the OSA for metallurgical and process control purposes. The mixing box discharge will combine with a number of intermittent reagent sump pump streams in the flotation tailings pump box. Flotation tailings will be pumped to the TDF.

A vertical spindle sump pump will be provided to return spillage to the flotation tailings pump box.

17.10 Raw Water, Potable Water and Process Water

Raw water make-up will be supplied to the raw water tank.

Raw water will be used for the following duties:

- Filter cloth wash via the raw water pumps;

- Reagent make-up via the raw water pumps;
- Cooling water, via the raw water pumps.

The decant water will be filtered and used for:

- Low pressure gland water, using the low-pressure gland water pumps;
- OSA.

The quality of filtered water used for GSW and OSA needs to be confirmed by suppliers during detail engineering.

Potable water will be supplied to the potable water tank where a ring main system will be installed to provide potable water to the safety showers and drinking fountains around the plant.

Concentrate thickener overflow and TDF decant water will be sent to the process water tank for re-use in the process plant. Raw water will be used as make-up as required. Anti-scalant will be added to the process water tank as required.

Process water will be used for the following duties:

- Filter manifold wash via the manifold wash water pumps;
- General process uses in the grinding, flotation and thickener areas via the process water pump.

17.11 Reagents

17.11.1 Frother

Frother will be delivered in bulk and stored in the reagent building until required. Glycol Frother will be dosed at a rate of new feed to the following locations:

- Primary Float Feed 0.037 kg/t of new feed
- Primary Float Mid 0.004 kg/t of new feed
- Secondary Float Head 0.008 kg/t of new feed
- Secondary Float Mid 0.0016 kg/t of new feed

- Column Cell Sparger Water 0.003 kg/t of new feed

Multiple diaphragm style dosing pumps will deliver the reagent to the required locations within the flotation circuit. A dedicated air diaphragm sump pump will be provided for spillage control.

17.11.2 Isobutyl Xanthate ("SIBX")

SIBX will be delivered in pellet form in bulk bags within boxes and stored in the reagent building. Raw water will be added to the agitated SIBX mixing tank. Bags will be lifted into the SIBX bag breaker, located on top of the tank, using the SIBX lifting frame and hoist. The solid reagent will fall into the tank and be dissolved in water to achieve the required dosing concentration. SIBX solution will be transferred to the SIBX storage tank using the SIBX transfer pump. Both the mixing and storage tanks will be ventilated using the SIBX tank fan to remove carbon disulphide gas.

SIBX will be delivered to the flotation circuit using the SIBX circulating pump and a ring main system. Actuated control valves will provide the required SIBX flowrates at a number of locations around the flotation circuit. SIBX will be dosed at a rate of new feed to the following locations:

- Ball Mill Feed 0.029 kg/t of new feed
- Primary Float Feed 0.037 kg/t of new feed
- Primary Float Mid 0.009 kg/t of new feed
- Secondary Float Mid 0.009 kg/t of new feed
- Cleaner Float Mid 0.0012 kg/t of new feed
- Regrind Mill Feed 0.014 kg/t of new feed
- Cu Bleed Conditioner 0.0008 kg/t of new feed

The SIBX mixing area will be ventilated using the SIBX area roof fan. A dedicated air diaphragm sump pump will be provided for spillage control.

17.11.3 Sodium Silicate ("SS")

SS will be delivered in bulk boxes and stored in the reagent building. The solid reagent will fall into the tank and be dissolved in raw water to achieve the required dosing concentration. SS solution will be transferred to the SS storage tank using the SS transfer pump. Both the mixing and storage tanks will be ventilated using the SS tank fan.

Diaphragm style dosing pumps will deliver the solution to the required locations of the circuit. A dedicated air diaphragm sump pump will be provided for spillage control.

17.11.4 N-Dodecyl Mercaptan (“NDM”)

NDM will be delivered in bulk boxes and stored in the reagent building until required. NDM will be dosed neat, without dilution. A diaphragm style dosing pump will deliver the reagent to the rougher flotation circuit. Top up of the permanent bulk boxes will be carried out manually as required.

A dedicated air diaphragm sump pump will be provided for spillage control.

17.11.5 Flocculant

Powdered flocculant will be delivered to site in 25 kg bags and stored in the reagent shed. A vendor supplied mixing and dosing system will be installed, which will include flocculant storage hopper, flocculant blower, flocculant wetting head, flocculant mixing tank, and flocculant transfer pump. Powder flocculant will be loaded into the flocculant storage hopper using the flocculant hoist. Dry flocculant will be pneumatically transferred into the wetting head, where it will be contacted with water. Flocculant solution, at 0.25% w/v will be agitated in the flocculant mixing tank for a pre-set period. After a pre-set time, the flocculant will be transferred to the flocculant storage tank using the flocculant transfer pump.

Flocculant will be dosed to the concentrate thickener using variable speed helical rotor style pumps. Flocculant will be further diluted to approximately 0.025% w/v just prior to the addition point.

A dedicated vertical spindle sump pump will be provided in this area.

17.11.6 Hydrated Lime

Hydrated lime will be delivered to site in a tanker and will be pneumatically conveyed from the tanker to the lime storage silo. The hydrated lime will be extracted from the lime storage silo via a rotary valve and screw feeder and discharged into the lime slurry storage tank. Raw water will also be added to the slurry storage tank to achieve the desired lime density.

The lime slurry from the lime storage tank will be distributed throughout the process plant by the lime slurry circulation pump and a ring main, with take-offs distributing lime to the process as required.

A dedicated vertical spindle sump pump will be provided for spillage control.

17.11.7 Anti-scalant

Anti-scalant will be delivered in bulk boxes and stored in the reagent building until required. Permanent bulk boxes will be installed to provide storage capacity local to each dosing point. Anti-scalant will be dosed neat, without dilution. Positive displacement style dosing pumps will deliver the anti-scalant to the process water tank. Top up of the permanent bulk boxes will be carried out manually as required.

17.12 Services and Utilities

17.12.1 On-stream Analysis ("OSA") System

The performance of the flotation circuit will be monitored by a dedicated OSA system, to allow the operator to make air, level or reagent changes based on real time assays. Analysis will include percent solids, copper, iron, and silver assays.

Cumulative shift samples for laboratory analysis will also be collected via the OSA sampling system. The system will have a stand-alone control, calibration and reporting system but will have the capacity to provide assay data to the plant control system if required.

Process streams that will be analyzed are listed as follows:

- Flotation feed;
- First rougher concentrate;
- Rougher concentrate;
- Regrind cyclone overflow;
- Scavenger concentrate;
- First cleaner scavenger tailings;
- Flotation columns concentrate;
- Rougher tailings;
- Flotation tailings.

Samples will be collected using a combination of sample pumps, pressure pipe samplers and linear samplers as required. Samples will be logically combined after analysis and returned to the process using vertical spindle style pumps.

17.12.2 High and Low-pressure Air

High pressure air at 700 kPa (g) will be provided by two high pressure air compressors, operating in a lead-lag configuration. The entire high-pressure air supply will be dried and can be used to satisfy both plant air and instrument air demand. Dried air will be distributed via the main plant air receiver, with an additional receiver in the grinding area.

Rougher flotation air will be supplied by two low pressure blowers. Cleaner flotation air will be supplied by two low pressure blowers.

18. PROJECT INFRASTRUCTURE

18.1 General

This section discusses the required infrastructure to support the mining and processing operations and includes the following areas:

- Public access road (County Road 64);
- Site access roads;
- Power generation plant;
- Site electrical distribution;
- Gatehouse;
- Communications network;
- Lake Superior water intake tie-ins;
- Potable water treatment plant tie-ins;
- Sewage treatment tie-ins;
- Covered box-cut for mine access;
- Ore stockpile pad;
- Truck shop, wash bay, warehouse and offices;
- Fuel storage;
- Mill offices and metallurgical laboratory;
- Administration office and assay laboratory;
- Concentrate transload facility;
- Tailings Disposal Facility (“TDF”);
- Effluent Treatment Plant.
- Event pond ditches for surface water management at mill site.

Figure 18.1 presents the White Pine Project site general arrangement and Figure 18.2 presents a close - up view of the general arrangement of the plant area.

Figure 18.1: White Pine General Arrangement

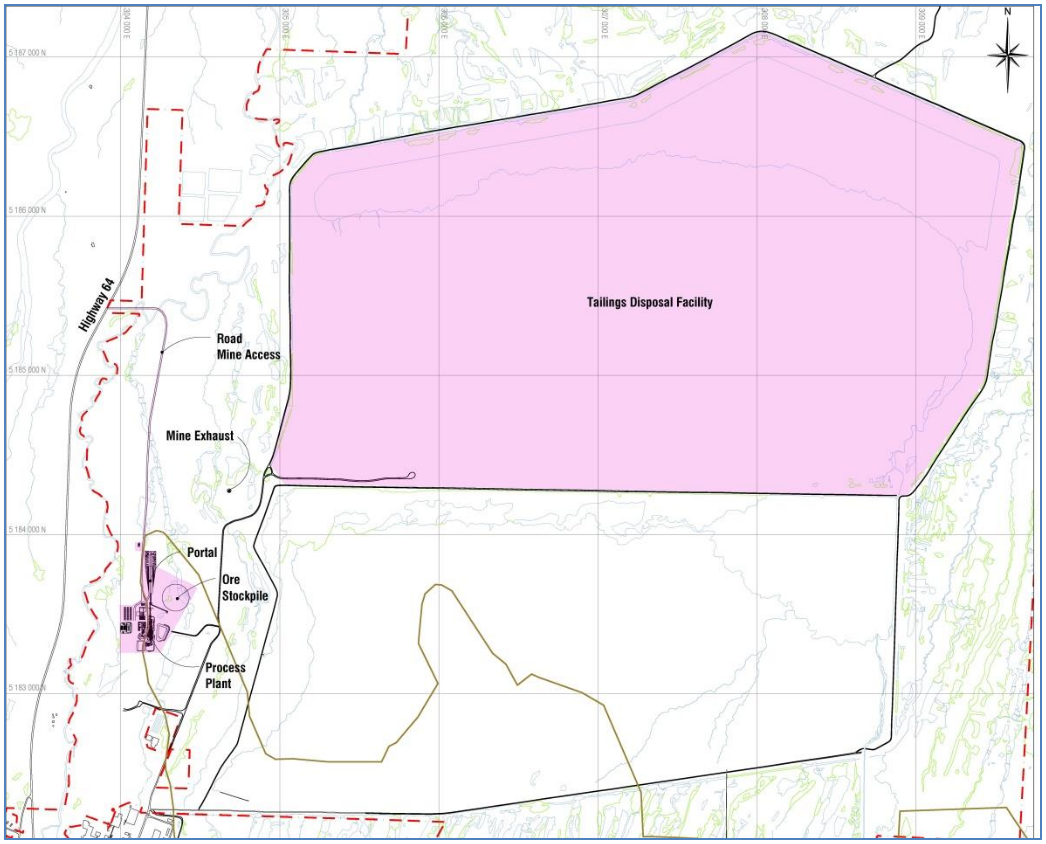


Figure 18.2: White Pine General Arrangement Plant Area - Close-up View

18.2 Public Access Road

The Project is accessed via the existing Michigan Highway 64 ("M-64") located on the West boundary of the site. M-64 connects the site entrance to major roads in the area and will handle all traffic to the site. The site entrance is located approximately 23 kilometer ("km") going south from the intersection of Highway 64/28, Bergland, Michigan. Owned and maintained by the Michigan Department of Transportation ("MDOT"), the road is fully paved. A survey performed by MDOT in 2009 showed that the volume of traffic from US2 and the Gogebic-Ontonagon county line was on average 418 vehicles. Therefore we can assume that White Pine Project traffic should not have a significant negative impact.

18.3 Communications

It is assumed that fiber optic or at least coaxial cables are available close to White Pine. A "backbone" point-to-point ("P2P") radio wave connection using proprietary dish's at emitting and receiving towers will also be put in place.

The proprietary or lease tower will be built at the mine site in order to install the P2P receiving dish and the Long-Term Evolution ("LTE") antennas to cover the area of the property. LTE antennas placed on the tower will be part of a surface/underground Private LTE Network ("PLTEN") to insure communication between workers (within as well as outside of the mine site). PLTEN will also be used to maximize any potential use of the "Internet of Things" ("IoT") by connecting mobile and fixed equipment, computers and telemetries to help in performing live monitoring and data capture.

A traditional Gigabit Wi-Fi connection connected to a Local Area Network ("LAN") will also be installed in the offices, mill, maintenance shop and other specific locations; in order to upgrade to the LTE/5G network once all the personnel and routing equipment capable of handling the increased network capacity are in place.

Cloud based software applications, including Enterprise Resource Planning ("ERP"), are preferable in limiting CAPEX expenses as well as maintenance/support costs related to the equipment's "On Premise" software licenses.

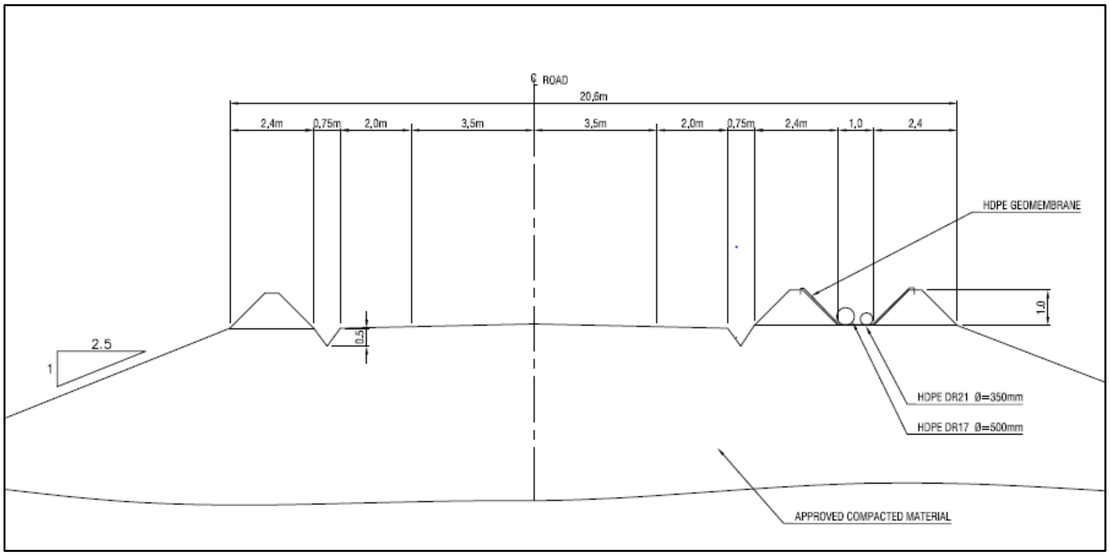
18.4 Site Roads

18.4.1 Main Access Road

The site is largely developed except for the main access planned from public highway Michigan M-64. This new main access road of approximately 2 km is now considered. Other options have been examined; however, these options may require agreement with land owners knowing that these options contain some private sections.

Therefore, the main access road connects the infrastructure area to the public road, M-64. All traffic coming to and leaving the site will use the main access road. The road runs in a primarily South-North direction across the site on the west side of the Tailings Disposal Facility ("TDF"). The reclaim system pipelines run along the north side of the road between the decant barge location and the mill area. The geometry of the road is designed based on a speed of 40 kmph (25 mph) with consideration given to maximum and minimum grades required for heavy trucks travelling on this road. Steel culverts will be used for the road's stream crossings. The full length of the road will use an aggregate surface course placed directly on compacted subgrade. The road has one 3.5 m wide lane and one 2.0 m shoulder in each direction with containment ditches and safety berms outside of both shoulders. The east side of the road has an additional safety berm to contain the reclaim pipeline. The two-berm containment system is lined with an HDPE membrane for potential spill containment. Safety berms have a height of 1.0 m. The resulting total road width is 20.6 m.

Figure 18.3: Main Road Cross-section

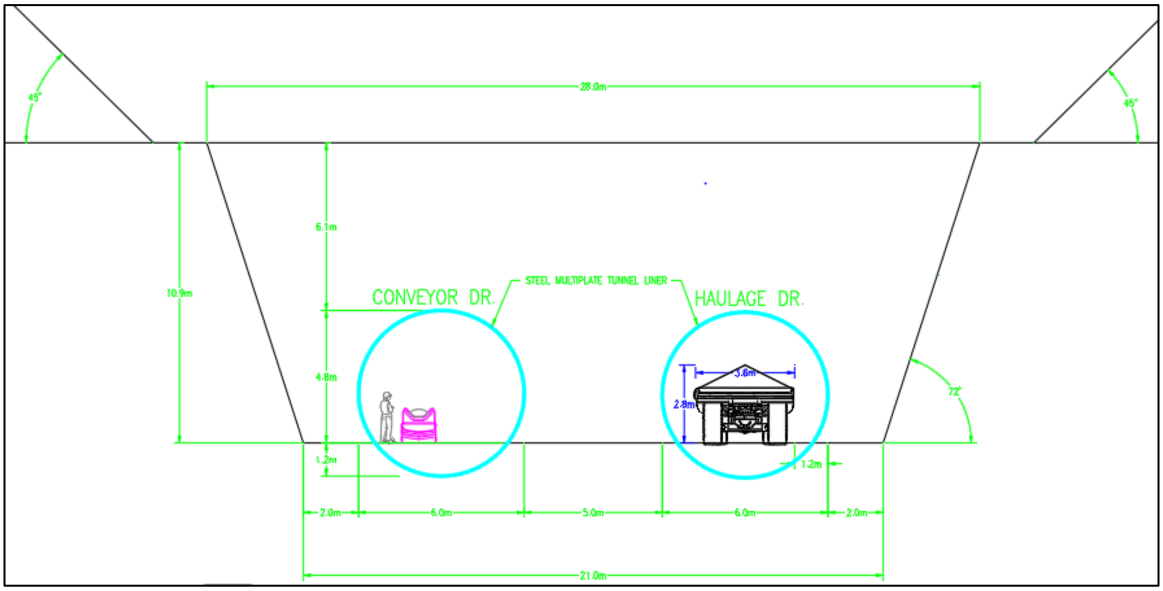


18.5 Box-Cut and Ore Stockpile

18.5.1 Box-Cut

The box cut entrance is located approximately 150 m south of the mill area and provides access to the mine dry, maintenance shop and warehouse from the South, and the ore stockpile to the east. The box cut design will have an approximately 250 m long ramp with a 15% elevation grade that grants access the mine portal and to the underground mine. The box cut will be excavated at a minimum of 15 m into the fresh rock, where tunnel multi-plate liners will be placed, and then backfilled for water management. The box cut uses two separate 6 m diameter fully round steel tunnels. The culvert of the steel tunnel is backfilled to create a driving surface for the mine equipment.

Figure 18.4: Box Cut Entrance



18.5.2 Ore Stockpile Pad

The ore stockpile pad is located 200 m southeast of the top of the box cut ramp. The ore stockpile is designed with a capacity of 500,000 Mt" at a maximum height of 15 m. Over the pre-production period, the ore will be hauled with mining trucks to the stockpile pad. After the end of the initial construction period a stacker will be used to manage the stockpile. Ore will be transferred from the ore stockpile to the mill feed conveyors using a front-end loader and a feeding chute.

The pad is approximately 50,000 m^2 in area and will consist of at least 300 mm of low permeability fill placed on top of the existing ground. The fill will be covered by an HDPE geomembrane. Water that contacts ore on the pad is considered contact water and must be directed to the TDF. The stockpile has a cross-slope that directs all runoff water into lined ditches. The water will eventually drain to a collection point on the NW corner of the stockpile where it will be pumped to the event pond and ultimately to the TDF or the water treatment plant at a later date in the life of the mine.

18.6 Water Management

18.6.1 Sewage Treatment

Sewage treatment will be handled using existing stabilization ponds. The stabilization ponds were used previously in White Pine and are now owned and maintained by the town of Ontonagon. The ponds were yielded by White Pine Mine to Ontonagon at the end of the mining activities.

18.6.2 Water Filtration

Gland water and OSA water will come from filtered reclaim water from the TDF. The water filtration unit will process all the water that is to be used as gland or OSA water.

18.6.3 Water Treatment Plant

The general operating strategy assumes that the makeup water source will be used to supply water to the TDF prior to the start of operations so that there is sufficient water available for the mill. Process water could come from Lake Superior's water intake as well. The capacity of the intake and pumps allows utilizing it as a sole source of water for process and human usage as it was in the past. The actual limit as known is 26 MGD (100 k m^3 per day). This limit is driven by the pumping capacity.

The water balance model inputs include the following:

- Production schedule;
- Climate data;
- Underground mine flow;
- TDF operation approach;
- WTP start date and capacity;
- Potable water treatment;
- Event pond and contact area runoff to the TDF.

18.6.4 Water Treatment Plant Design

The Water Treatment Plant ("WTP") influent water quality by source and the blended influent water quality. For the purpose of this Study, it is assumed that the water treatment design is based on the Copperwood Project in which the capacity has increased as the through-put increased.

18.7 Potable Water – Existing System

The existing water in-take was built for the previous mining operations to supply the town of White Pine. Upon the end of the operation, the potable water system was turned over to the town of Ontonagon. Ontonagon has been operating and maintaining the system since then. It is assumed that Ontonagon will make potable water available for the new operations and the town of White Pine.

18.8 Site Run-off and Spillage Control

A network of ditches is designed on order to discriminate contact water from the non-contact water. The ditches will drain the run-off non-contact into a pond located west of the site and is redirecting the water to the nearest creek.

The ditches that carry the contact water from the ore stockpile, mill area or hauling roads is redirected to the contact event pond that is located east of the site. From the contact water event pond, the contact water is pumped to the Tailings Management Facility ("TMF") and then will recirculate into the process water circuit.

18.9 Fuel

A fuel storage will be built strictly for mining and support mine equipment. The dike tank for diesel is designed to have a capacity of 10,000 litres with pumps and concrete pads, which are located south of the mine entrance.

18.10 Power Supply and Distribution

The previous mining operations were powered by a natural gas-fired power generation plant. The plant was supplying power throughout the operation and the region. The current situation of the power plant was not evaluated however it is known that the natural gas pipeline is still in service up the power plant location.

This Study plans to build a new natural gas-fired power generation plant. The natural gas will come from the existing natural gas pipeline system. It is assumed that the operation costs will remain the same.

The cost estimate includes a 3 mi. long powerline at 13.8 kV and will provide power to all areas. Overall, approximately 30 MW will be required to adequately service the Project.

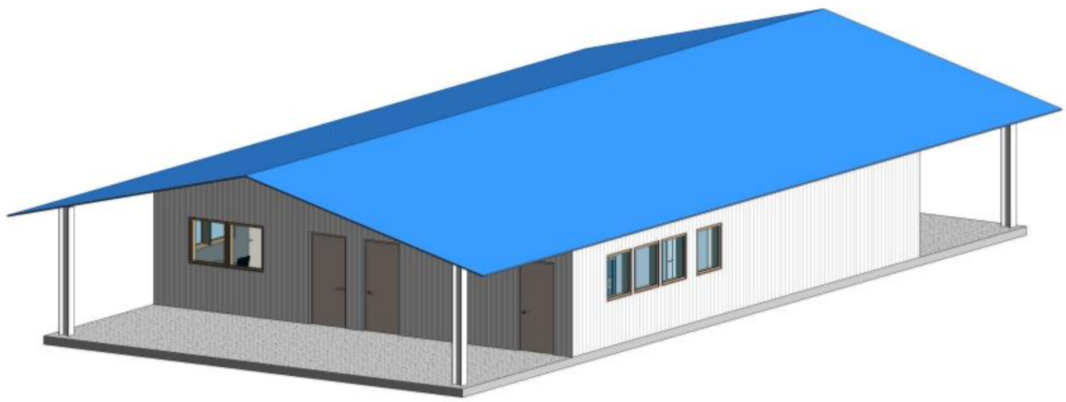
18.11 Fire Protection

Water for emergency fire extinguishing will be stored in an underground tank south of the Gate House. The tank will be located south of the main access road to isolate it from other infrastructure and increase its elevation compared to the process plant. Two tanks will have a 50,000 USG capacity each for a total of 100 000 USG. Fire pumps will provide the proper water flow and pressure as stipulated in the North American codes.

18.12 Security

The site access will be secured by the gatehouse located adjacent to the main access road in the southern portion of the process area. All traffic coming to or leaving the process area will pass through the gate house. The rooms in the gate house will include; visitor registration, security office, induction room, vehicle control room, and bathrooms. The interior area of the gate house will be 150 m with two covered 50 m^2 concrete aprons. Vehicle access will be controlled by a boom gate.

Figure 18.5: Gatehouse



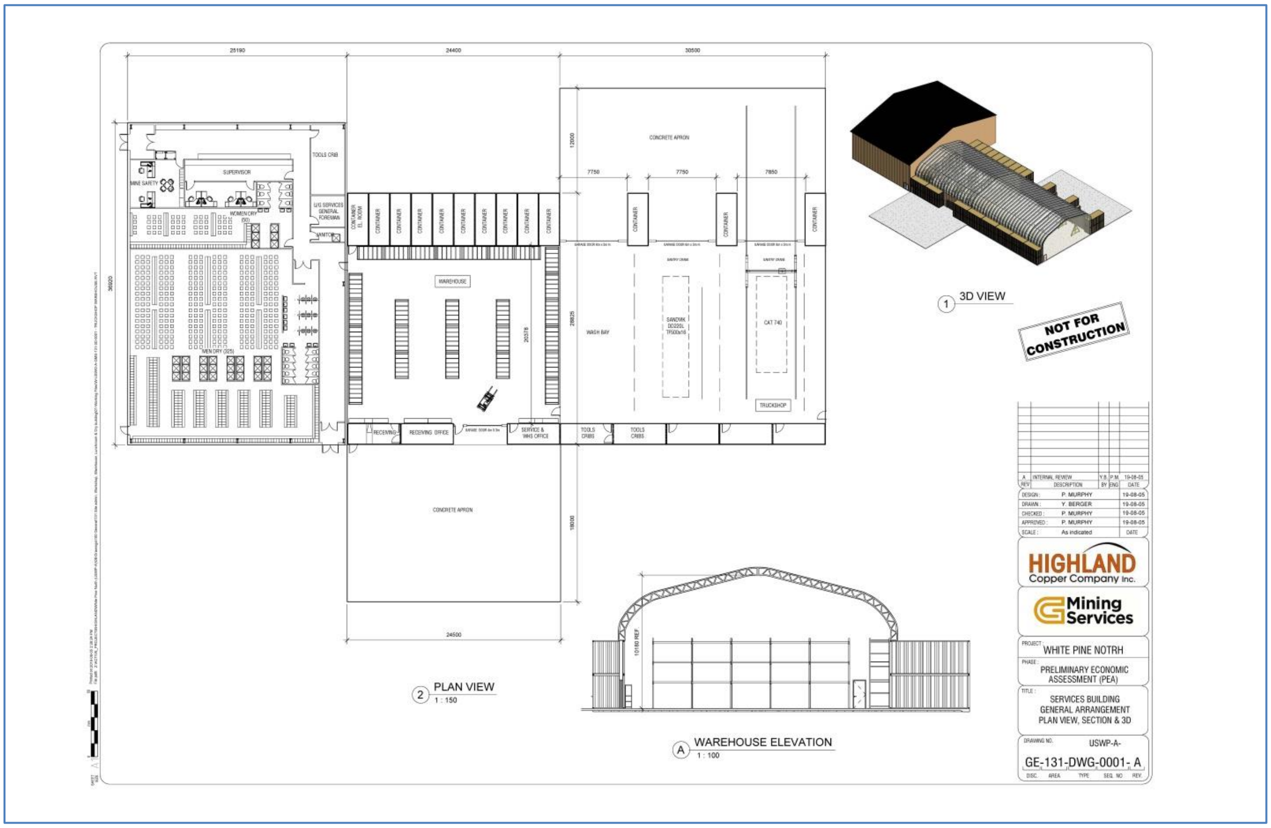
18.13 On-site Buildings

18.13.1 Truck Shop, Warehouse and Related Offices

The truck shop and warehouse are connected to and located in the northwest part of the site. The truck shop and warehouse will share a single insulated tension fabric roof set on top of the containers, divided by a fabric wall. The truck shop will be used primarily for heavy-duty vehicle maintenance. The truck shop will have 3 separate bays each equipped with a 6 m wide x 5 m high roll-up door. Two bays will have their own railed gantry crane with a 15 t capacity which can be moved inside or outside of the building. One bay will be used for washing purposes. Some of the containers supporting the dome will be insulated and converted into offices. Water used to clean vehicles in the truck shop is to be considered as contact water and will be collected and sent to the event pond.

The warehouse will include racking to store spare parts and consumables. The warehouse's interior dimensions will be 20 m x 25 m. The warehouse and truck shop will both have concrete aprons to better handle heavy vehicle traffic.

Figure 18.6: Truck Shop



18.13.2 Mine Dry

The mine dry will be adjacent to the truck shop and warehouse. The dry will serve as locker rooms for the mine workers between shifts and contain the mine rescue equipment, medical offices and a few offices for management personnel. The dry has enough locker space for a total of 375 workers. The men's portion accommodates 325 workers and includes showers, toilets, urinals, lockers and baskets. The women's portion accommodates 50 workers and includes showers, toilets, lockers, and baskets. The dry is a pre-engineered steel-clad building.

18.13.3 Construction Offices

The construction offices are containerized and located just north of the process plant. Six trailers with built-in-place corridors are planned and provide enough space for the Owner's construction management. The construction offices will serve as the office space during the construction phase and for mill office space over the life of the mine.

18.13.4 Off-Site Buildings

The following areas are considered project infrastructure for mining operations, but are located off the Copperwood site:

- Administration building and assay laboratory;
- Transload facility.

18.13.5 Administration Building and Assay Laboratory

The Administration building and the assay laboratory will be located in the Town of White Pine using spaces already built. The actual plan takes into consideration lease spaces. Included in the project costs are major upgrades for plumbing and HVAC as well as architecture renovations and furniture. For the Assay Laboratory, all technical equipment is to be purchased by the Project.

18.13.6 Transload Facility

The transload facility will be located at a rail siding in Park Falls, Wisconsin 200 km from site. The location has been chosen due to the costs and mainly because it provides access of the Canadian National Railway networks, for easy shipment to known economical smelters. The facility accepts concentrate shipments from site via side-dump haul trucks. Haul trucks enter the building, dump the concentrate, and exit the building. Concentrate is loaded into rail cars using a front-end loader. The building is fully enclosed to ensure the control of air quality with sufficient air changes, as per the usual codes. Entrances and exits will have roll-up style doors to regulate airflow through the building. Each haul truck carries a concentrate payload of around 18 Mt, a weight that is limited by the Wisconsin DOT to a maximum gross vehicle weight of 80,000 lb.

Figure 18.7: Transload Building Cross-section

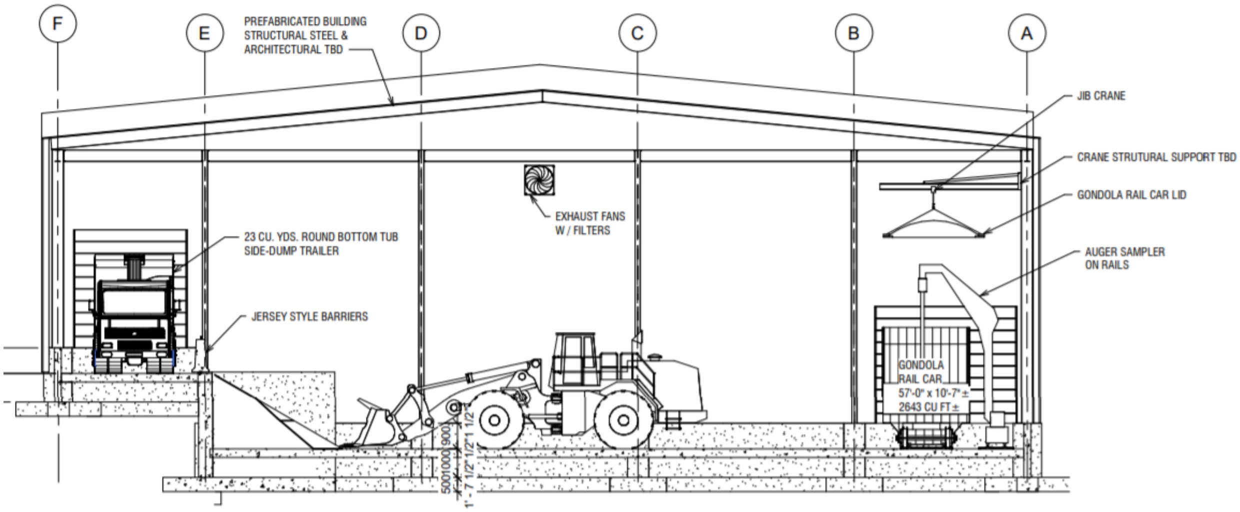
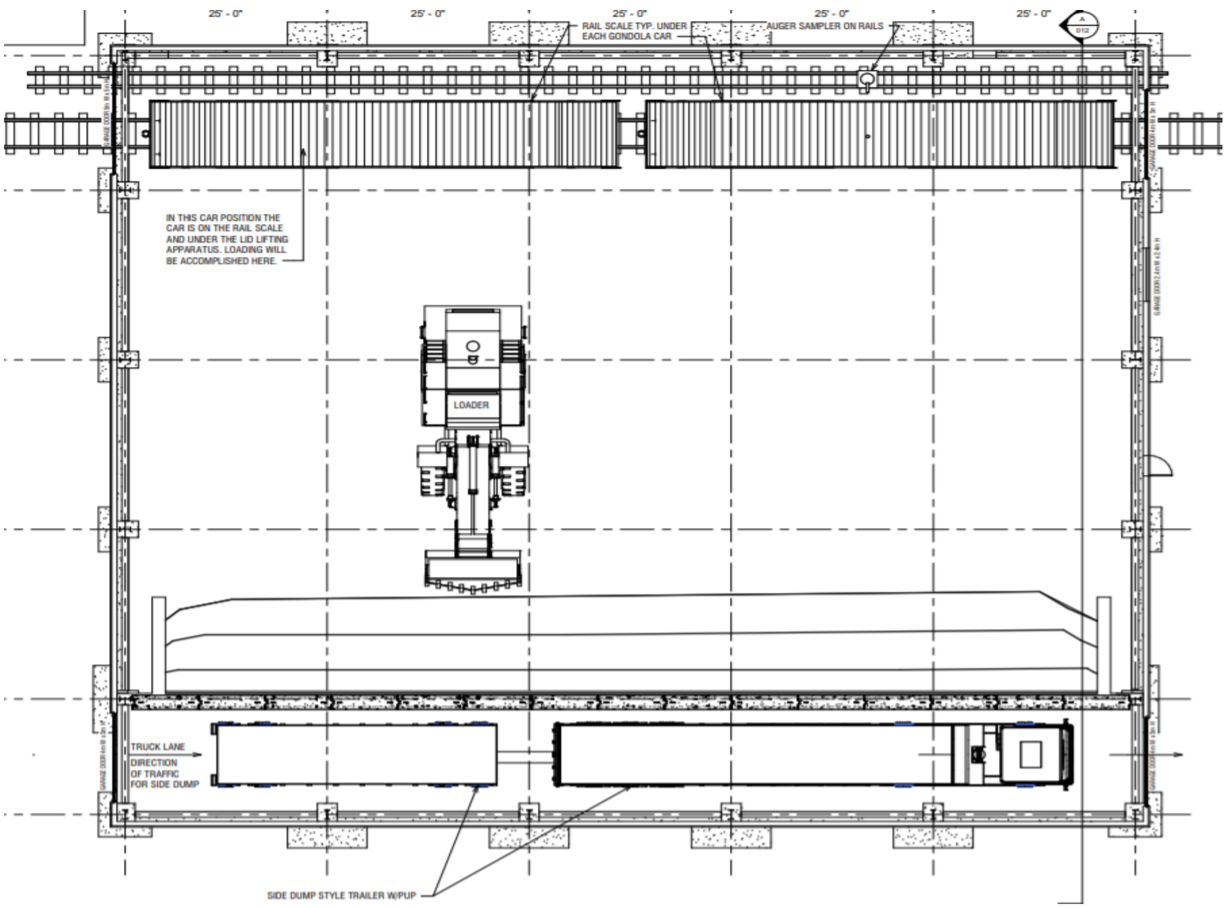


Figure 18.8: Transload Building Plan View



18.14 Tailings Disposal Facility

18.14.1 General Arrangement and Development

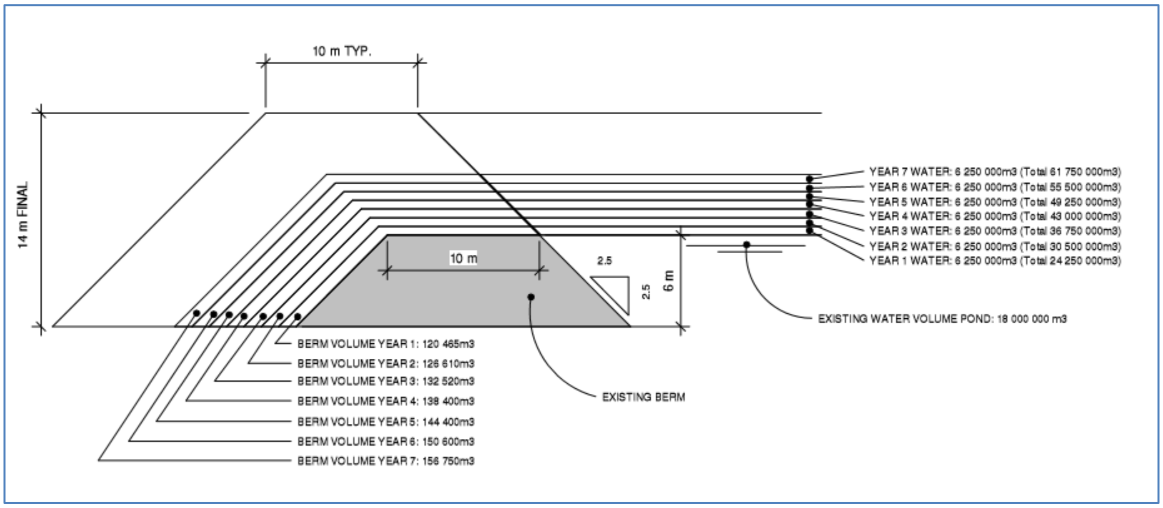
In its current state, North Pond 2 (NP2) has approximately 18 million cubic meters (approximately 22 Mt) of storage without considering any dam raises and without freeboard. Storage of the entire tailings volume along with accommodating a two meter freeboard will require raising the existing NP2 embankment to an elevation between 255 and 256 masl. NP 2 is lower in elevation than NP1 and raises to the NP2 embankment, which will be on the North, East, and West sides. The common side along the south edge of NP2 and north edge of NP1 will remain higher than NP2, even after the NP 2embankments are raised to reach the required capacity at the end of the mine.

The embankment will be constructed sequentially using downstream methods, which means that the upstream toe will remain fixed while the downstream toe will progressively advance downstream as the embankment height increases.

The design criteria for slopes as established in the Golder Associated, November 2015 are as follow:

- 2.25H: 1V - This was the slope proposed in the Bechtel design which resulted in the need for a large internal seepage system (toe and chimney drains) in order to reach a slope stability factor of safety (FOS>1.5).
- 2.5H: 1V - This slope provides an increased factor of safety and reduces the internal seepage system requirements.
- 3H: 1V- This slope further improves the factor of safety and reduces the seepage system requirement. It should be noted that with this external slope, the toe would encroach upon the existing perimeter channel in some locations.

Figure 18.9: Typical Cross-section as the Crest Increases



19. MARKET STUDIES AND CONTRACTS

19.1 Metal Prices

The metal prices selected for the economic evaluation in this Report are presented in Table 19.1. A constant long-term price of US$3.00/lb for copper and US$16.00/oz for silver has been assumed.

Table 19.1: Metal Price Assumptions

Metal Price Scenario	Yr 1+ (2023)
Copper (US$/lb)	3.00
Silver (US$/oz)	16.00

There is no guarantee that copper and silver prices used in this Study will be realized at the time of production and will be subject to normal market price volatility and global market forces of supply and demand. Prices could vary significantly higher or lower with a corresponding impact on Project economics.

The 10-year historical price for copper as presented in Figure 19.1 highlights the variable nature of metal prices with a high of approximately US$4.50/lb seen in 2011 and a low of US$1.30/lb in mid-2008. The 5-year historical price for silver is similarly presented in Figure 19.2 and ranges between US$14/oz and US$20/oz.

Figure 19.1: 10-year Historical Copper Prices

Figure 19.2: 5-year Historical Silver Prices

19.2 Market Studies

19.2.1 Copper Concentrate

The copper concentrate produced from White Pine will require downstream smelting and refining to produce marketable copper and silver metal. Several smelters could receive concentrate with the nearby candidates being the Horne smelter located in Noranda, Quebec or the copper smelter in Sudbury, Ontario. Other alternatives include seaborne export to Asia or Europe. Concentrate transportation charges will be a function of the final destination and will be a combination of trucking, rail and possibly shipping.

The concentrate treatment and refining charges ("TC/RC") vary depending on the state of the economy and the supply and demand dynamics for copper concentrates available for smelting.

Copper payment is based on copper content of the concentrate. For a concentrate less than 32% but above 22% the payable rate is typically 96.5%, subject to a minimum deduction of 1%. Payment of precious metals in copper concentrates varies by region and customer but typically pays 90% if greater than 30 g/dmt with a 30 g minimum deduction. A summary of the copper concentrate marketing assumptions is summarized in Table 19.2.

Table 19.2: Concentrate Marketing Assumptions

Copper Concentrate Marketing Assumptions	
Copper Payable Rate	96.5% payment of Cu in concentrate >22%Cu and <32%Cu subject to a 1% minimum deduction
Silver Payable Rate	90% payment of Ag subject to 30g/dmt minimum deduction
Copper Treatment & Refining Charge (TC/RC)	TC = US$70/dmt of concentrate, RC = $0.070/lb of Cu
Silver Refining Charge	RC = US$0.50/oz of Ag

Penalties may be applied to copper concentrates that have excessive amounts of deleterious elements such as lead, zinc, arsenic, antimony, bismuth, nickel, alumina, fluorine, chlorine, magnesium oxide, and mercury. The White Pine concentrate can be classified as a clean concentrate and no penalties for deleterious elements are foreseen.

19.3 Realization Costs

19.3.1 Concentrate Transportation

The transportation of concentrate was evaluated by Concept Consulting LLC with the current assumption that concentrate would be destined for the Horne smelter in Noranda. However, no contracts are in place at this time and other smelters can be considered.

The concentrate from White Pine will be loaded into heavy-duty dump trailers with a cover and transported to a truck to rail transload facility located in Park Falls, Wisconsin. The truck configuration consists of 5 axles and will transport approximately 20 t per shipment. Park Falls is a preferred transload location as it is currently served by the Canadian National ("CN") railroad and is approximately 159.3 kilometers ("km") (99 mi) from the mine site. The CN is a Class 1 railroad and its network spans three coasts with over 33,800 km (21,000 mi) of track and access to 75% of the North American continent and currently has operating lines in Michigan and Wisconsin. The pertinent operating line is the Ashland Sub Line in Wisconsin which as a maximum rail line load rating of 121,560 t (268,000 lbs) and only operates north up to Park Falls. The CN line from Marengo Junction to White Pine is inactive.

The concentrate transportation costs are estimated at US$68.16/t of concentrate which includes trucking, transload operations, CN rail transportation and gondola lease costs as summarized in Table 19.3.

Table 19.3: Concentrate Transportation Cost (Mine to Horne Smelter)

Concentrate Transportation	Cost (US$/t)
Truck Transportation	19.00
Transload Operations	3.00
CN Rail Transportation	38.00
Gondola Lease Costs	8.16
Total Transport Cost	**68.16**

19.3.2 Insurance

An insurance rate of 0.10% was applied to the provisional value of the concentrate to cover transport from the mine site to the smelter.

19.4 Contracts

There are no mining, concentrating, smelting, refining, transportation, handling, sales and hedging, forward sales contracts, or arrangements for the Project. This situation is typical for a development stage project still several years away from production.

20. ENVIRONMENTAL STUDIES, PERMTTING AND SOCIAL OR COMMUNITY IMPACT

20.1 Introduction

Mining and ore processing occurred at the White Pine location from the late 1800s through 1995. The underground mine workings extend under an area of approximately 16,000 acres and from the surface to a maximum depth of approximately 2,800 ft. The surface components of the mine were located on approximately 400 A and the tailings impoundments occupy approximately 5,500 A.

The White Pine Mine ceased operation in 1995 and has been the subject of an extensive remediation program outlined in judicial Consent Decree ("CD") and Remedial Action Plan ("RAP") agreements between the Copper Range Company ("CRC") and the State of Michigan. The entire surface area overlying the underground mine along with the associated surface components area and tailings impoundments are listed as a "facility" under Part 201, Environmental Remediation, of Michigan's Public Act 451 of 1994 as Amended, the Natural Resource and Environmental Protection Act ("NREPA").

White Pine Copper LLC ("WPC"), a subsidiary of the Highland Copper Company ("Highland"), began mineral exploration and baseline environmental surveys at the White Pine North Project site under the terms of a March 2014 Asset Purchase Agreement ("APA") between CRC and Highland. The APA includes an interim closing agreement allowing Highland access to CRC property and a final closing that was to be completed on or before December 31, 2015. Completion of the final APA closing has not occurred due to financial constraints and the closing date has been extended by mutual agreement between CRC and Highland.

Under terms of the APA, Highland is required to assume CRC's environmental liabilities and ongoing remedial action obligations at the former White Pine Mine as required by the CD and the RAP. This includes establishing a financial assurance mechanism ("FAM") with the State of Michigan to ensure continued operations, monitoring and maintenance requirements of the RAP are met and that predicted water treatment of the current North #2 Tailings Pond discharge will be provided. Negotiations with the State of Michigan for amendment of the CD and establishment of the FAM have continued; however, there has been no activity since July 2018.

There has been very limited WPC activity at the White Pine North Project location since the end of 2015. A mineral resource remains to be confirmed and a feasibility study completed. Baseline environmental surveys require updating and additional field work completed before WPC can proceed with the permitting process under Michigan's NREPA, in particular for a Part 632 Nonferrous Metallic Metal Mining permit. Permitting is discussed in more detail in Sections 20.16 to 20.18.

Historic reports are available that document environmental conditions at the former CRC mine site workings and nearby areas. These include but are not limited to: 1) Environmental Assessment Report – White Pine Copper Division, Baker, August 1978, 2) Environmental Aspects of the Proposed Solution Mining Operation, Shepard Miller Inc., February 1995, and 3) Remedial Investigation Report – White Pine Mine, MFG, Inc., December 1999.

The above historic reports, ongoing CRC monitoring reports required by the RAP and information gathered during WPC's baseline monitoring efforts in 2014 and 2015 will be relied upon to describe site conditions at the White Pine North Project area. Additional discussion of baseline environmental study requirements is provided in Section 20.4.

20.2 Geology and Geomorphology

The White Pine North Project is located in the Ontonagon High Clay Plain physiographic province. The province is characterized as a glacial lakebed plain, mostly flat with local, deeply incised stream channels. The low relief plain is partly swampy and includes gravely-clay soils and bedrock outcrops. Elevations range from about 750 to 1,100 ft above mean sea level ("amsl"), with surface elevations at the former mine site being about 200 to 500 ft feet above Lake Superior (602 ft amsl).

The geologic setting of the province is near the southern edge of the Canadian Shield. The underlying bedrock is of Precambrian age, overlain by unconsolidated and glacio-lacustrine sediments of Pleistocene and recent ages. These units are described below.

Unconsolidated surficial deposits of mainly glacial and lacustrine origin are deposited on top of the Precambrian age bedrock and range in thickness from 0 to 130 ft in the mine vicinity. The majority of the unconsolidated section is Pleistocene clay till. The base of the till commonly contains bedrock boulders in a silty and sandy matrix. Upward in the section the boulders become less common and the matrix becomes clay. The uppermost Pleistocene deposits are lacustrine clay. Exploration borings and monitoring well logs show that most of the Project area has less than 50 ft of unconsolidated sediment. The median unconsolidated thickness is approximately 37 ft. A few areas of unconsolidated sediment thicker than 100 ft occur south and west of the mine site.

The three bedrock formations of primary interest in the White Pine vicinity are the Freda Sandstone, Nonesuch Shale, and Copper Harbor Conglomerate. These bedrock formations are of Precambrian

Keweenawan age and underlie the unconsolidated deposits. The Freda, Nonesuch, and Copper Harbor overlie the older Porcupine Volcanic and Portage Lake Lava Series. The Precambrian bedrock is consolidated and highly cemented.

The Freda Sandstone is usually described as a series of red, massive, fine-grained sandstones and siltstones, with some laminated silty shales. The sandstone contains minor amounts of the heavy minerals ilmenite, leucoxene, and epidote. The thickness of the Freda can reach thousands of feet.

The Nonesuch Shale is a series of shales and siltstones with minor sandstones. It is approximately 600 ft thick in the White Pine area. In contrast to the Freda red beds above and the Copper Harbor red beds below, the Nonesuch is generally grey to black.

The Copper Harbor Conglomerate ("CHC") underlies the Nonesuch Shale, with a 350 to 6,000 ft thickness. The CHC is a fining-upward series of sandstone, felsite-pebble conglomerate, and siltstone. The formation does contain basaltic lava flows of limited extent. The CHC is generally colored red by hematite, except immediately below the Nonesuch Shale where hematite is absent and may appear bleached and carbonaceous.

20.3 Acid Rock Drainage Potential

Acid Base Accounting ("ABA") and synthetic precipitation leaching procedure ("SPLP") tests were performed on samples of White Pine tailings as part of CRC's remedial investigation to evaluate acid generation and leaching potential. Neutralization potential of the tailings was found to be high and acid generation potential was calculated as zero with pyritic sulfur in the samples less than the analytical detection limit (0.01%). The SPLP test results were less definitive due to reasons of analytical detection limits and the SPLP test only being an indicator of short-term mobility of leachable metals. The historic White Pine tailings are permanently disposed of and are interacting with interstitial pore water, precipitation and, to a limited extent, ambient oxygen.

A test program needs to be completed as part of the environmental impact study that includes bulk characterizations and both short-term and long-term tests under static and kinetic conditions as appropriate to simulating planned environmental conditions. Low detection limit analytical methods need to be used, e.g. EPA method 1631 for ultra low-level mercury analysis.

The geochemical testing program completed for Highland's Copperwood Project (formerly Orvana Minerals) can serve as a model and also as an indicator of expected results for the White Pine North Project as the geologic settings for both mines are very similar, i.e. glacial lake bed plain overlying Freda, Nonesuch and CHC bedrock formations with ore mined from the base of the Nonesuch formation.

In September 2014, Highland engaged consultants Theodore Bornhorst LLC and Mark Logsdon, Geochemica (also designed and executed the program for the Copperwood Project) to design a geochemical testing program that would satisfy Michigan's Part 632 requirements for either a combined Copperwood – White Pine North Project with ore processing and tailings disposal at White Pine or a stand-alone White Pine North Project. The plan was to begin the geochemical testing in mid-2015 when project operations were defined. The geochemical test work was not started due to financial constraints.

20.4 Environmental Baseline Study

After entering into the APA agreement with CRC, historical environmental data for the White Pine Mine site from the CRC ownership period was reviewed and compared with Highland's initial project plans and Michigan's Part 632 regulatory requirements. CRC had compiled extensive information on surface water, ground water and near-surface soils at the Project site (note that this data is now over 20 years old). Biological monitoring data in the Project area was mostly limited to very brief descriptions, e.g. the Remedial Investigation Report of 1999, or the more thorough description of the 1978 Baker report that is now over 40 years old. Data from limited nearby stream monitoring completed by the State of Michigan in 1999 and earlier is also available.

Highland identified information gaps and engaged consultants to design updated ground water (Golder Associates) and surface water (GEI Consultants) monitoring studies that would provide baseline hydrology and water quality data necessary to satisfy the Michigan Part 632 permit requirements. The initial monitoring event was completed in October 2014. Subsequent monitoring events were completed in March 2015, May 2015, July 2015 (field data only) and October 2015. At a minimum, one additional monitoring event will be required under each of winter and spring conditions and two monitoring events under summer conditions.

Field wetland delineations were completed in 2014 by consultant King & MacGregor Environmental for an approximately 900 A area west of the North #1 and #2 Tailings Ponds thought likely to be utilized for new development, e.g. box cut/mine portal, ore stockpile, process plant, access roads, etc. Additional field delineation of wetlands will be required adjacent to the north and east embankments of the North #2 Tailings Pond if expansion by downstream construction is chosen to provide increased disposal capacity.

A biological consultant (White Water Associates) was engaged to design a baseline flora and fauna monitoring plan that would satisfy Part 632 permit requirements. The initial study area was planned to include the above-mentioned area west of the North #1 and #2 Tailings Ponds, perimeter areas adjacent to both the North and South Tailings Ponds and stream and pond surveys in select locations both up and down gradient of the former CRC facilities. Initial monitoring began in September 2014 for migratory birds and in December 2014 for winter mammal tracking. Field work planned for the 2015 spring-summer-fall field season included mist netting for bats, terrestrial mammals and plants, pond complex surveys and stream surveys. Other biological survey work required, but not planned for 2015, included birds, reptiles and amphibians. While the 2015 bat survey work was completed, all other field work was suspended by mid-summer or not started due to financial constraints. Completion of the biological monitoring is required to meet Part 632 regulations and could be completed in one spring-summer-fall field season.

20.5 Air Quality

Meteorological conditions at the White Pine North Project are comparable to those at Highland's nearby Copperwood Project. Both are located in lake plain physical settings with topography rising steeply to over 1,000 ft amsl to the south of the Project site. White Pine, at 5 miles ("mi") inland, is a little farther from the Lake Superior shoreline than Copperwood at a little under 2 mi. Following is a description of local conditions from the CRC Remedial Investigation Report:

"The climate of the Michigan Upper Peninsula is typical of most mid-western states in the northern United States. The Mine area experiences a continental to semi-maritime climate, largely dominated by the passage of weather systems from west to east and the modifying influence of Lake Superior. The stabilizing effects of the lake on air temperatures and prevailing winds result in cold winters, cool summers, and consistently high humidity throughout the year. The mean annual temperature for White Pine, Michigan is approximately 40 degrees Fahrenheit, with monthly mean temperatures ranging from minus seven to six degrees Fahrenheit in the winter months and from 70º to 80º F in the summer months. Precipitation is well distributed throughout the year with the non-snow season (May through October) receiving an average of 18.43 in (58% of the total, annual rainfall of 32 in). The average, annual snowfall at White Pine is 177 in. The Mine area averages 142 days with one inch or more of snow on the ground. Peak snow depth typically occurs in February and March."

The meteorological station used for on-site monitoring at the Copperwood Project was dis-assembled and moved to the White Pine North Project location in November 2014 and the monitoring instruments sent for calibration and/or repairs. Return and reassembly of the calibrated instruments was completed in April 2015 and the weather station was placed in service. While initial instrument readings appeared to be accurate, problems with readings stored in the data logger became apparent in May 2015. In particular, temperature and humidity values were not believable, and the barometer readings were not being recorded. Financial constraints have prevented repairing or upgrading the station. While Part 632 regulations require description of site meteorology and predicted variations, they do not specifically require this information to be based on data from an on-site weather station. Current and historical local weather data is available from stations located in Ironwood (35 mi west southwest) and Bergland (12 mi south).

The White Pine North Project is located in an attainment area for all pollutants and there are no active major sources of air pollution in close proximity with the nearest being a natural gas transmission station 26 mi southwest of White Pine and a newly commissioned natural gas engine power plant 45 mi to the east. The White Pine Electric Power generating station in White Pine could potentially restart if the boilers are upgraded to meet current regulations. Air permitting requirements for White Pine North are likely similar to those of the recent Copperwood air permit issued by the State of Michigan. Permitting is discussed in more detail in Sections 20.16 to 20.18.

20.6 Hydrology

The Project site is located approximately five miles from Lake Superior. The surface water system in the vicinity includes the Mineral River, Perch Creek, and a number of smaller perennial and intermittent streams, all of which flow principally south to north in a parallel drainage pattern and empty into Lake Superior. Other surface water features of interest include the tailings pond impoundments which cover over six square miles, flooded borrow pits and depressions from tailings impoundment construction and numerous swampy areas often related to beaver activity. Streams that historically flowed through the areas that became tailings impoundments have been rerouted to streams flowing near the east and west perimeters.

The Mineral River receives the majority of its water from surface runoff, including the North #2 Tailings Pond Outfall into Perch Creek (Current CRC National Pollution Discharge Elimination System (“NPDES”) Permit MI0006114). Additional discussion of this discharge is found in Section 20.7. The outfall is permitted for a maximum discharge of 51.1 million gallons per day (“MGD”) although flow is typically around 9 MGD. Base flow is supported throughout most dry seasons via influent groundwater from the shallow aquifer.

The groundwater system in the vicinity of the Mine property can be divided into two basic hydrostratigraphic units: 1) the shallow aquifer consisting of unconsolidated overburden sediments and the underlying, shallow, fractured bedrock; and 2) the lower bedrock units consisting of low permeability siltstone, sandstone, and shale with relatively few open fractures. Additionally, there is perched groundwater of limited lateral extent located in the former surface facilities area.

The unconsolidated sediments that comprise the upper portion of the shallow aquifer consist of low permeability clay, silt, and sand of glacial and lacustrine (lake-deposited) origin. Recharge to the unconsolidated sediments occurs mostly by direct infiltration of precipitation. Because of the low productivity of these sediments, very few wells in the region derive groundwater solely from the unconsolidated sediments but rely, in part, on the underlying shallow fractured bedrock for adequate water supply. The permeability of the Precambrian bedrock in the vicinity of the Project is low. Groundwater from the upper bedrock is derived almost entirely from secondary fracture permeability formed as a result of glacial loading and unloading. As a result, the hydraulic characteristics of the shallow bedrock are similar, despite a wide range of bedrock lithologies. Fracturing of this type decreases rapidly with depth until permeability of the bedrock is too low to yield groundwater in usable quantities.

Groundwater in the deep unit is hydraulically disconnected from the shallow aquifer except where drill holes and faults may provide localized points of connection. While the drill holes may present a pathway for impacted surface water or shallow groundwater to reach the deeper aquifer, the extent of impacts to both surface water and shallow groundwater identified to date and the potential for transport to and down drill holes is relatively insignificant. In addition, the deeper aquifer is high in TDS and does not represent a usable aquifer.

CRC's 1995 Shepard Miller solution mining report provided a detailed analysis of ground water hydrology in both the near surface shallow aquifer and the deep bedrock aquifer and will provide a very useful reference for ground water hydrology of the White Pine North Project and vicinity. Shepard Miller did not use the commonly accepted ModFlow software to create a ground water model. The Copperwood mine permit groundwater analysis and ModFlow model should also be a useful reference. Highland has not begun any efforts to update and verify reported groundwater hydrology for the White Pine North Project area.

20.7 Water Quality

The water of Lake Superior is of high quality due largely to its immense volume and the general lack of human activity and industrial development in most of its drainage basin. Many of the problems that have plagued the lower Great Lakes, such as algal blooms, are absent in Lake Superior. Lake Superior is an oligotrophic lake (i.e., of low biological activity). Unlike the other Great Lakes, such as Lake Erie, the levels of nutrients (e.g., phosphorous and nitrogen) are very low and, consequently, algal populations are also low. The populations of lake trout and whitefish attest to Lake Superior's high water quality.

Surface water sampling in stream segments and drainage areas of the former White Pine Mine surface facilities was completed as part of CRC's remedial investigation. A limited amount of sampling occurred in off-site streams considered to have had no impacts from historic mining activity. Constituents of concern identified on the basis of analytical results are boron, copper, lithium, manganese, strontium and zinc.

Elevated copper concentrations are common in surface waters of the White Pine Mine area, especially the Mineral River. Some of this can be attributed to naturally occurring Nonesuch formation outcrops in stream beds with significant contributions from recent and historic mining and ore processing operations. Duck Creek to the east, and away from, mine influenced areas also exhibits elevated copper concentrations.

Perch Creek is heavily influenced by the current tailings pond discharge that includes precipitation runoff from the North and South Tailings Ponds and a significant amount of underground mine water that is pumped into the North #1 Pond to maintain a set water level in the mine. The deep bedrock water infiltrating into the former mine workings is very high in dissolved solids (mainly chlorides) and is mixing with a freshwater cap that CRC flooded most of the former mine workings with as part of their site closure activities. Recent modelling by CRC's consultant Tetra Tech predicts that in the year 2034 the blended tailings pond discharge water will reach a dissolved solids concentration that is toxic to aquatic life necessitating start of a treatment program.

After reviewing historical surface water data for the White Pine Mine, GEI Consultants provided the following analysis to Highland: *"In general, the historic data from the site was collected in the late 1990s. This data is not sufficient to be utilized for direct inclusion into a mine permit application under the Part 632 regulations. Sampling was limited during these investigations. Little or no flow data was collected on the streams, analysis of the water samples did not include the full set of parameters required in Part 632, analytical detection limits ("DLs") were too high for many of the analytes (leading to a significant proportion of "non-detect" values), the sampling was not conducted over the 2-year period as required in Part 632, and sampling was not conducted over the White Pine North area."*

GEI concluded that the historic data can be utilized for comparison purposes and that the Copperwood Project water quality data should provide a good analog for data at the White Pine site. Sampling completed by Highland at White Pine in 2014 and 2015 includes analyses for a full suite of parameters at appropriate detections limits as based on the issued Copperwood Part 632 mine permit.

CRC also completed ground water sampling in the vicinity of the White Pine Mine as part of its remedial investigation including a limited number of monitoring wells considered to be outside the influence of mining activities. Constituents of concern identified as a result of the sampling are barium, boron, manganese, lithium and strontium. These elements are naturally present as trace metals in deep bedrock groundwater at the White Pine Mine and throughout the Canadian Shield. It is also known from historical data that ground water quality is poor in local areas outside the influence of the mine, in particular for elevated chloride content.

As with GEI's observation of historical surface water data, the parameter list and analytical detections limits for available ground water data are not sufficient to meet the Part 632 permit requirements. As with GEI's conclusions on surface water quality, the historic ground water data can be used for comparison and the Copperwood ground water data set should also be useful. Highland's sampling of monitoring wells includes the same suite of parameters and detections limits as used for surface water sampling of 2014 and 2015.

20.8 Soil Quality

Glacial geology of the White Pine area was described by consultant Golder Associates: *"Unconsolidated surficial deposits of mainly glacial and lacustrine origin are deposited on top of the Precambrian age bedrock and range in thickness from 0 to 130 feet in the mine vicinity. The majority of the unconsolidated section is Pleistocene clay till. Based on a set of clay till samples from ten exploratory borings. . . . ., below the glacio-lacustrine surficial clay unit, the bulk of the glacial till is classified under the Unified Soil Classification System as a Sandy Lean Clay with minor fractions of gravel. Measurements of vertical hydraulic conductivity during the 1995 remedial investigation (MFG, 2005) yielded an overall value of 3×10^{-8} cm/sec. The base of the till commonly contains bedrock boulders in a silty and sandy matrix. Upward in the section the boulders become less common and the matrix becomes clay. The uppermost Pleistocene deposits are lacustrine clay. At some locations, elongated sandy and gravelly deposits mark former lake shorelines (Hack, 1965). Post-glacial and recent sandy alluvial deposits occur in modern stream beds and near the Lake Superior shore (Shepard Miller, Inc. 1995)."*

Near surface soils in the White Pine area are further described by the US Natural Resources Conservation Service the Forest Service in the publication *Soil Survey of Ontonagon County, Michigan*, *2010* and available as a web-based resource or downloadable manuscript. In the White Pine North Project area, there are four soil units that comprise approximately 60% of survey map units described as loamy to fine-loamy with varying clay contents.

CRC's remedial investigation included both gridded and targeted soil sampling, mostly for metals and organics. Not surprisingly, copper content was found to be elevated above background levels, especially in the former surface facilities areas and in clusters along the Mineral River. Elevated levels of petroleum products were noted at locations in the surface facilities area. Other metals, including arsenic and lead, were found above various Michigan's Part 201 screening levels. Again, the bulk of these were in the surface facilities area. Remediation for these risks identified in CRC's RAP was placement of an engineered barrier (either compacted clay or asphalt) and deed restrictions for industrial property use only.

20.9 Flora

The only historic information available to Highland of flora specific to the White Pine North area is the 1978 Baker report that describes typical western Upper Peninsula tree and other vegetation present and gives a brief listing of species present in the White Pine Mine area. Highland completed a timber stand survey in 2015 for the White Pine North Project area as it was defined at that time. Results confirm the brief summary in the Baker report with a detailed GIS map prepared showing forest community boundaries and areas plotted.

In the early summer of 2015, Highland's biological consultant White Water Associates ("WWA") competed a set of early season meander surveys of understory and ground level vegetation in the Project area west of the North Tailings Ponds. Financial constraints precluded the typical follow up surveying in late summer and no report was produced. The early field work identified a possible state threatened plant species (Sweet Cicely) present in low abundance. Since early growth of this species can be confused with similar non-listed plants in the same family, its identity was to be confirmed in the late summer round when the plants had produced seeds.

Flora surveys in the White Pine North Project area will need to be re-initiated according to Part 632 requirements. The Copperwood Project area, located in a similar lake plain setting can be used for comparison and a single field season of surveys should satisfy the Part 632 requirements.

20.10 Fauna

The availability of fauna information for the White Pine North area is similar to that for flora with brief descriptions in the 1978 Baker report of typical western Upper Peninsula wildlife and species known to inhabit the White Pine Mine area. Baker noted that migratory waterfowl that pass through the area and that resident populations of Blue Herons and Canada Geese are present. Baker also provided an appendix listing all mammal, bird, reptile and amphibian species known to be present in the White Pine area.

Species listed under the Endangered Species Act known to be present in 1978 are still present in the area. Bald Eagles have recovered and are no longer listed as threatened or endangered. Presence of Eastern Gray Wolves was noted as rare in 1978. Wolf presence was noted during winter mammal tracking surveys that WWA competed in 2014-2015. While they are considered as being recovered but still listed as endangered, the US Fish & Wildlife service is in the process of delisting them. In 2015, the presence of Northern Long Eared Bats (recent threatened species listing) was noted in the Project area. Current restrictions are mainly not disturbing hibernation sites (caves and old mines) and no removal of roosting trees during the June to July pup rearing season.

In the early summer of 2015, a subcontractor of WWA completed the field work portion of a bat survey in the Project area west of the North Tailings Ponds. One threatened species and three common species were found to be present. WWA completed one round of small mammal tracking surveys in the summer of 2015 but financial constraints halted this work. Stream surveys for aquatic life were also completed in eight locations but final analyses of field collected macroinvertebrates was not completed. A new biological monitoring program will have to be initiated to meet Part 632 requirements. Field data from 2015 Copperwood Project data set can be used for reference, as with water quality and flora, and a single field season of surveys should satisfy requirements.

20.11 Landscape

The 1978 Baker report and the 1999 CRC Remedial Investigation report provide descriptions of physical features and land use in the White Pine vicinity. Over 75% of the total land area is covered in dense, second growth forest. The US Forest Service (Ottawa National Forest) and State of Michigan (Porcupine Mountain Wilderness State Park ("PMWSP"), Copper Country State Forest) own a combined 41% of land in the western Upper Peninsula of Michigan. Other significant landowners include CRC's holdings in the former White Pine Mine area and those of forest management companies engaged in timber harvesting activities. Topography in Ontonagon county ranges from nearly level lake plains to steep bedrock hills and bluffs. The lowest elevation would be that of Lake Superior at 602 ft amsl to Summit Peak in the PMWSP at 1,950 ft amsl. Section 20.2 provides additional information on the physical setting in the area.

20.12 Socio-Cultural

The area of the former White Pine Mine, and Ontonagon County in general, has been experiencing a steadily declining population since closing of the mine in 1995 and closing of a kraft pulp mill in Ontonagon in 2010. US Census data for Ontonagon County confirms this decline in total population (rounded data): Year 2000 - 7,800 residents, Year 2010 - 6,800 residents, Year 2019 estimate - 5,800 residents. Even more telling from a socio-economic viewpoint is the median age estimate of 57 years. The townsite of White Pine has seen a population decline from 1,100 residents in 1995 when the mine closed to an estimated 450 in 2019, median age is 58. A number of residential dwellings in White Pine are owned by non-residents and used as vacation/recreation homes.

There are limited employment opportunities in Ontonagon County with the largest remaining employers being the Aspirus Ontonagon Hospital and Long-Term Care Facility, various units of the Ontonagon County Government and the two Ontonagon County school districts. A number of working age residents of the county commute into neighboring Gogebic County for employment.

20.13 Archeology

Highland is not aware of any Phase I Archeological Surveys that have been completed in the White Pine Mine area. Published information has established a history of native copper mining from Ontonagon to the Keweenaw Peninsula going back as far a 7,000 years ago. When the US government ratified the 1842 Treaty of LaPointe with the local Ojibwa Tribes, the area was opened to exploration and commercial exploitation of these same copper deposits. In the Treaty of LaPointe, the Ojibwa natives retained rights to hunt, fish, gather and otherwise use natural resources in the ceded territory of the treaty. In legal terms these are referred to as usufructuary rights.

Highland participated in a scoping meeting hosted by the State of Michigan in May of 2015 that included tribal representatives. A report prepared by the Great Lakes Indian Fish and Wildlife Commission was shared with Highland. The title of the report is *Cultural and Economic Importance of Natural Resources Near the White Pine Mine to the Lake Superior Ojibwa*, June 1998. Most of the document is based on activities of the Ojibwa natives after arrival of Europeans in the early 1600's. Activities noted in the White Pine Mine area centered around seasonal encampments at the mouths of the Big Iron and Ontonagon Rivers. There was an established foot trail that headed south from the Big Iron River (5 mi north-northwest of the mine) that was used for access to the Lake Gogebic area (12 mi to the south of the mine). The report does not provide any indications of the exact route of the trail. For reference, the Big Iron River is west of State Highway M-64 and outside of the White Pine North Project area.

At a minimum, Phase I Archeological Surveys will be required in any area of the project that will have direct impacts on the land surface or water resources. Highland has had some success discussing tribal issues with impacts of the Copperwood Project and should continue these efforts for a White Pine Project.

20.14 Environmental Management Plan

Part 632 regulations require development of a mining and environmental protection plan (Rule 203) that must meet very specific operation, monitoring and maintenance requirements. Other associated Project permits will also have requirements for monitoring and management.

Monitoring plans according to varying timelines include:

- Surface Water Quality;
- Surface Water Flow;
- Groundwater Quality;

- Wildlife and Vegetation Surveys;
- Leachate Collection and Leak Detection Systems;
- Liners, Pipelines, Berms, and Embankments.

Documented plans required by Part 632 and associated permits for the Project will include:

- Soil Erosion and Sediment Control Plan;
- Spill Prevention Control and Countermeasures Plan;
- Pollution Incident Prevention Plan;
- Storm Water Pollution Prevention Plan;
- Invasive Species Management Plan;
- Wetland Mitigation Plan;
- Stream Mitigation Plan;
- Sampling Analysis Plan;
- Contact Water Management Plan;
- Quality Assurance and Quality Control Plan for Site Monitoring;
- Fugitive Dust Control Plan;
- Reclamation and Closure Plan;
- Treatment and Containment Plan;
- Integrated Contingency Plan;
- Monitoring Plan for Reactive Material;
- Threatened and Endangered Species Plan.

20.15 Closure, Decommissioning and Reclamation

Part 632 regulations require a financial assurance instrument must be put in place sufficient to cover the cost to administer and to hire a third party to implement the reclamation, remediation and post closure monitoring plan for the mine site. The assurance instrument must be in place prior to starting what Part 632 defines as “mining activity” that includes tree clearing and grubbing. The assurance can be negotiated in staged amounts commensurate with the amount of reclamation effort each stage of project construction will cause.

Release of the financial assurance instrument will be made based on the requirements listed below:

- Reclamation of the mine box-cut and impacted area;
- Demolition and/or removal of the process plant and the related features;
- Demolition and/or removal of ancillary structures;
- Demolition and/or removal of utilities infrastructure;
- Reclamation of the TDF;
- The reclamation of the TDF is expected to proceed over a period of several years. Release of the financial assurance instrument can be proportional to the work completed; to be determined and agreed upon the State of Michigan and Highland.
- Full release of the financial assurance instrument upon documentation of successful completion of the 20-year post closure monitoring plan and agreement by the State of Michigan that all requirements of the Part 632 regulations have been satisfied.

The reclamation plan serves to bring the Project site back to a self-sustaining ecosystem that is close to pre-existing conditions. This includes:

- Reclaimed topography and land use;
- Surface features remaining after reclamation;
- Roads and dikes;
- TDF reclamation;
- Plant site reclamation;
- Disposal of waste materials;
- Closure of the underground mine and access;
- Site revegetation;
- Groundwater and surface water quality monitoring.

20.16 Legal Framework

Michigan's Public Act 451 of 1994, as Amended, commonly known as the NREPA, sets the framework for environmental regulations and permitting in Michigan. This Act is subdivided into many Parts, encompassing all major natural resource and environmental protection topics. Michigan has

Memorandums of Agreement and State Implementation Plans in place with the US Environmental Protection Agency that provide the State with delegated authority for Federal Clean Air Act and Clean Water Act permits except those involving direct impacts to the Great Lakes and commercially navigable rivers and harbors (US Army Corps of Engineers permit required in addition to a permit under NREPA). Only two US states have delegated authority to issue permits for impacts to inland lakes, streams and wetlands (commonly referred to as Section 404 permits).

20.17 Permitting Process

The permitting process begins by conducting a baseline study of the conditions of the Project area. In Michigan's Part 632, this means a 2-year survey of topics relating to water, soil, air, vegetation, wildlife, social, cultural, and historical resources (note that some field work requirements can be satisfied in a single year). The Project plans and related analysis of those plans must be completed to fully understand and discuss the impacts associated with the Project. The application for a permit must be created according to the rules of each permitting program and submitted to the responsible regulatory agency. The regulatory agencies review the documents and request corrections, clarifications, or amplifications, if needed, from the permittee until they deem the application administratively and technically complete. Once this is done, the application goes out for public comment and a public hearing may be held on the application. Any outstanding questions arising from public comments during this period will need to be answered by the applicant and, once this process is completed the regulatory agency can either grant or deny the permit. Permit applicants must demonstrate their proposed project will *"reasonably minimize the actual and potential adverse impacts on natural resources, the environment and public health and safety"*.

20.18 Permits to Obtain

To start construction and begin operation of this Project, several permits and approvals must be obtained and agreed upon between Highland and the regulatory authorities on federal, state and local levels.

The major environmental permits required include:

- Part 632 Non-Ferrous Metallic Mining Permit;
- Part 31 National Pollutant Discharge Elimination System Permit;
- Part 55 Air Permit to Install;
- Part 301 Inland Lakes and Streams Permit;
- Part 303 Wetland Permit;

- Part 315 Dam Safety Permit.

Other minor and local permits and approvals are also required to start construction and mine operation that include:

- Local building and zoning permits;
- Explosives handling permit from the US Bureau of Alcohol, Tobacco, and Firearms;
- Storage tank permits;
- Mine Safety and Health Administration registration.
- Mine Safety and Health Administration registration.

21. CAPITAL AND OPERATING COSTS

21.1 Capital Expenditures

The capital cost estimate was established using a hierarchical work break down structure. Estimates are based on benchmarks and scaling from the Copperwood Feasibility Study. This is a Class 5 estimate prepared in accordance with AACE International's Cost Estimate Classification System. The accuracy range of the capital cost estimate is +/-35%. The base currency of the estimate is US dollars (US$). No escalation was built into the capital cost estimates. The estimates are as of Q3-2019.

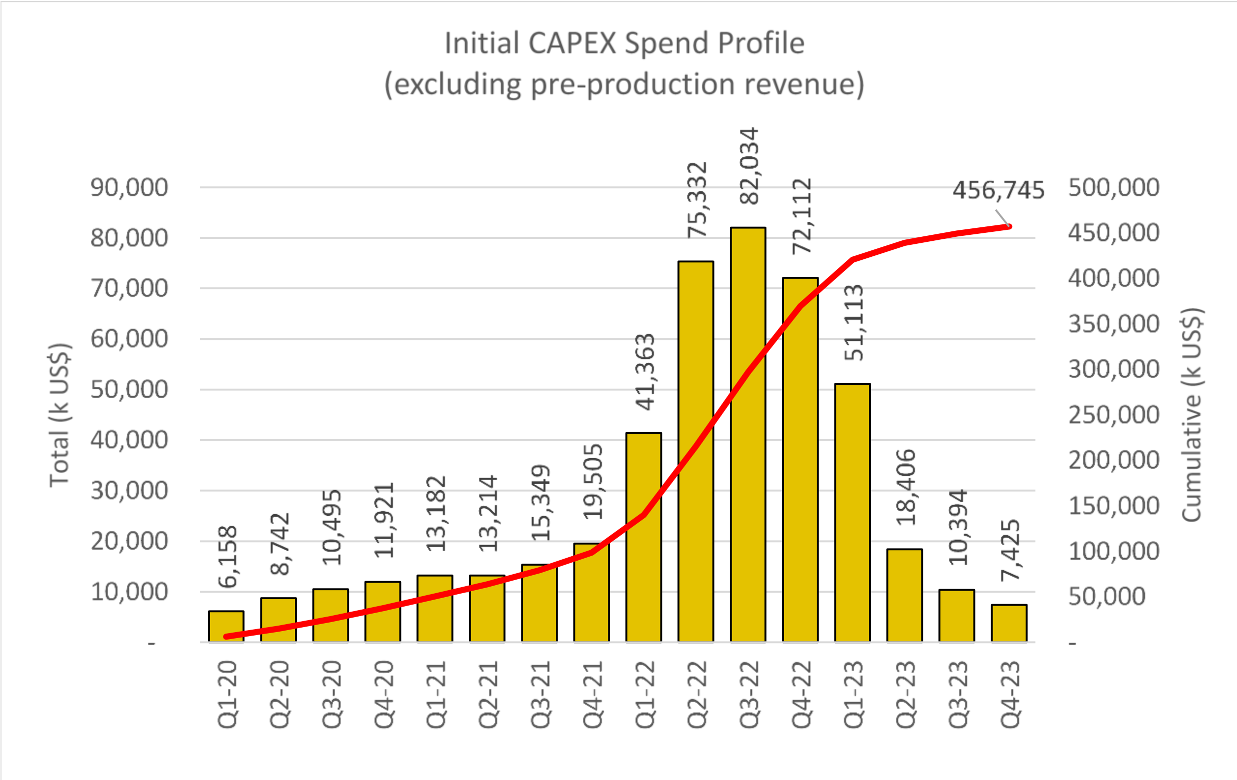
This capital cost is estimated at US $456.7 M net of pre-production revenue as presented in Table 21.1.

Table 21.1: Capital Expenditures Summary

Initial CAPEX	k US$
000 - General	212
100 - Infrastructure	31,214
200 - Power & Electrical	64,051
300 - Water & TSF Mgmt.	15,078
400 - Mobile Equipment	44,453
500 - Mine Infrastructure	39,183
600 - Process Plant	47,359
700 - Construction Indirects	25,739
800 -General Services & Owner's Costs	18,193
900 - Pre-Production, Commissioning	136,190
Sub-Total Before Contingency	**421,671**
Contingency (21.5%)	**90,788**
Total Incl. Contingency	**512,460**
Less: Pre-Production Revenue	**(55,715)**
Total Incl. Contingency & Pre-Prod Revenue	**456,745**

The initial capital expenditures spend schedule over a 4-year period (2020 to 2023) is presented in Figure 21.1. The first two years are mostly for box cut and mine underground development work.

Figure 21.1: Initial Capital Expenditures Spend Schedule



21.1.1 <u>Infrastructures</u>

A capital expenditure ("CAPEX") summary for infrastructures is presented Table 21.2. The detailed description of infrastructures and roads are presented in Section 18.

Table 21.2: Infrastructure CAPEX

Area	k US$
110 – Infrastructure	**2,366**
111 - Main Access Road	904
112 - Site Roads	913
114 - Fencing	5
117 - Employee Parking Lot	544
120 - Workshops / Storage	**1,861**
123 - Plant Workshop & Stores	497
124 - Reagents Storage Building	889
125 - Explosives Plant / Magazine	475
130 - Support Buildings	**9,549**
131 - Workshop, Warehouse, Lunchrooms & Dry Building	5,498
135 - Main Gatehouse	384
138 - Off-Site Facilities - Transload Building & Facilities	3,667
150 - Process Plant Buildings system	**14,298**
151 - Process Plant Main Building	14,2984
160 - Laboratories	**2,944**
161 - Assay, Environmental Laboratory	2,944
170 - Fuel Systems	**196**
173 - Diesel Fuel Storage	196
Grand Total	**31,214**

21.1.2 Power Supply and Communications

A summary of the CAPEX for electrical and communications is presented Table 21.3. They include all equipment and installations for power supply and distribution. The power line and main site substation costs are negotiated with the power rates with the utility company and therefore are not shown in this table. The electrical infrastructures are detailed in Section 18 of this Report.

Table 21.3: Power Supply and Electrical Capital Expenditures

Area	k US$
210 - Main Power Generation	**59,379**
212 -Main Power Generation Station	58,895
217 - Emergency Power Generation (Surface)	484
220 - Process Plant Electrical Rooms	**2,355**
221 - Process Plant E-Room	1,560
225 -Tailings E-Room	245
227 -Other E-Room	550
240 - Site Power Distribution	**599**
241 - Site Powerlines	599
250 – U/G Power Distribution	**1,250**
251 -U/G Sub-Station	750
255 -U/G Distribution	500
270 - U/G Communications Network	**468**
271 - U/G Communications Network	468
Grand Total	**64,051**

21.1.3 Water and Tailings Disposal Management

Details and description of Tailings and Water Disposal Management (“TDM”) installation and systems are provided in Section 18. The Tailings Disposal Facility “(TDF”) is built within the existing Tailings facility called North Pond 2 (NP2). It is assumed as of Golder’s report that with the removal of the superior layer capping the tailings, the dams will provide sufficient volume to cover the 2 to 3 years of production. Capital costs include earthworks, concrete, structure steel, mechanical, electrical and instrumentation equipment and labor.

The surface water management system is constructed to gather all contact water generated on site. It includes the lined ditches, pumping station and pipelines from pumping stations to the event pond. From the event pond, the plan is to ultimately pump the water to the TDF.

The existing Lake Superior water in-take works include works at the pumping station only.

The existing potable water system is sufficient to provide potable water to the Project.

The fire water estimate includes the fire pumps, the distribution network within the processing and mine plant.

A CAPEX summary for water is presented in Table 21.4.

Table 21.4: Tailings & Water Capital Expenditures

Area	k US$
310 - Raw Water Supply & Potable Water	**747**
311 - Process Water	597
316 - Lake Superior In-take	150
320 - Reclaim Water	**1,760**
321 - Reclaim Water System	865
322 - Reclaim Pipeline	704
322 - Gland Water	191
330 - U/G Water Management	**494**
331 - Water Management Surface	494
340 - Tailings Disposal Facility	**5,056**
344 - TDF Basin (Cover Removal)	3,611
346 - TDF Pipeline	1,445
360 - Effluent Water Management	**9,188**
364 - Effluent Treatment Plant	9,188
370 - Fire Water	**756**
371 - Fire Water Distribution	756
380 – Domestic Sewage	**689**
381 - Sewage Treatment System	689
Grand Total	**18,690**

21.1.4 Mobile Equipment

Mine Equipment includes all capital expenditures related to the acquisition of primary mining and support equipment. Equipment CAPEX include the purchasing cost, assembly cost and all safety and optional installs on the equipment. Construction mobile equipment includes purchasing costs for a front-end loader to be used to lift equipment. All other equipment is either included in construction contracts or rented. Rental costs for light vehicles required for the construction commissioning period.

A summary for the capital expenditures for mobile equipment is presented in Table 21.5

Table 21.5: Mobile Equipment Capital Expenditures

Area	US$
410 - U/G Mining Equipment & Maintenance	**43,731**
412 - U/G Mining Equipment	43,731
414 - U/G Support Equipment	
419 - Mining Equipment Capital Spares	
420 - Construction Vehicles and Equipment	**322**
422 - Light Vehicles and Other Equipment	322
430 - Surface Mobile Equipment	**400**
431 - Surface Mobile Equipment	400
Grand Total	**44,453**

21.1.5 Mine Infrastructure

Mine infrastructure CAPEX include the portal excavation, installation of multi-plate culverts, and backfill. Hauling starts outside of the ramp to the ore stockpile. Mine development includes labor, consumables to complete the drifts to reach mining panels.

Other costs are all related to safety, utilities work and infrastructure such as refuge, lunchrooms, ventilation raises, in-take and exhaust and pumping systems.

Mine infrastructure also includes the feeders and underground main conveyor to be installed over the pre-production period.

A summary of the CAPEX for mine infrastructure is presented in Table 21.6.

Table 21.6: Mine Infrastructure Capital Expenditures

Area	k US$
510 - Surface Mine Infrastructure	**3,205**
512 - Haul Road	480
515 - Ore Handling / Reclaim	1,780
517 - Ore Stockpile Pad	945
520 - U/G Mine Infrastructure	**5,455**
522 - Portal (Box-cut)	5,455
530 - Ventilation raise & Escapeways	**1,434**
531 - Collar & Excavation	1,434
550 - U/G Mine Dewatering System	**359**
551 - U/G Mine Dewatering System	359
580 - U/G conveying/crushing system	**28,730**
581 - Feeder breakers and Primary Conveyors	28,730
Grand Total	**39,183**

21.1.6 Process Plant and Related Infrastructures

The initial capital cost estimate for the processing facility is provided in Table 21.7 The estimate includes earthworks, concrete, structural steel, mechanical, piping, electrical / instrumentation and architecture equipment and labor.

Quantities for the earthwork, concrete, structure, piping, electrical, instrumentation and architecture material take-offs were estimated as per similar project, namely Copperwood project. The unit rates for material were estimated by GMSI. The list of mechanical equipment was derived from PFDs.

The estimate covers all costs and construction works related to the processing plant. The process plant building, and other secondary structural steel are included in Area 150. Scope includes the haul ramps to access the feed hopper and finishes at the tailings pumps located after cyanide destruction. All related plant auxiliary services and reagents are also included.

The capital costs estimate for the processing areas is presented in Table 21.7.

Table 21.7: Processing Capital Expenditures

Processing Capital Costs	k US$
610 - Crushing and Ore Handling	**8,831**
611 - Conveyor/Stacker	3,060
615 - Reclaim Circuit	5,771
620 – Grinding	**20,100**
621 - Grinding & Cyclopak	20,086
622 - Media Storage	14
630 – Flotation/Regrind Circuit	**10,720**
631 - Conditioning Tank	2,203
632 - Rougher Cells	2,832
633 - Scavenger/1st Cleaner Cells	1,649
634 - 2nd Cleaner Cells	826
635 - 3rd Cleaner Cells	11
636 - Cyclone & Regrind	3,201
640 – Tailings	**803**
642 - Flotation Tailings	803
650 – Copper Concentrate Filtration; Thickening & Handling	**4,916**
651 - Cu Concentrate Thickening	1,610
652 - Cu Concentrate Filtration	2,983
653 - Load-out, Packaging Concentrate	323
670 – Reagents	**1,450**
671 - Lime Circuit	757
672 - MIBC	161
673 - PAX	255
675 - Na2SiO3	122
676 - Flocculant	155
680 – Plant Services	**539**
681 - Compressed Air	117
682 - Low Pressure Compressed Air	422
Grand Total	**47,359**

21.1.7 Construction Indirect Costs

Construction indirect costs include all the engineering activities as well as site construction management. A full suite of temporary facilities is also included as well as tools and operating and maintenance costs for construction equipment.

Construction Indirect Costs are presented in Table 21.8

Table 21.8: Construction Indirect Capitals

Construction Indirects	k US$
710 - Engineering, CM, PM	**23,159**
711 - Site CM staff and Consultants	7,487
713 - Montreal CM Staff and Consultants	5,991
715 - External Engineering	6,231
716 - Surveying	500
717 - QA/QC	2,200
718 - Commissioning and Vendor's Rep	250
719 - Induction / Travel / Visas / Working Permits	500
720 - Construction Facilities & Services	**2,580**
722 - Construction Temporary Services	1,080
727 - Construction Tools / Consumables	500
729 - Construction Equipment Rentals	1,000
Grand Total	**25,739**

21.1.8 General Services

General Services include all the support departments, generally directly hired by Highland, that will be staffed and organized to assist during the development stage of the Project and will continue their functions during the operating phase; it includes the following:

- General Administration (GM);
- Supply Chain Local;
- HR & Training;

- Health and Safety;
- ESR;
- Security;
- IT;
- Accounting and Finance.

All freight is estimated from quotations or from similar recent projects. Corporate costs are not charged to the Project. Temporary power costs include fuel and maintenance for power consumption the construction and plant needs. Cost estimates are presented in Table 21.9.

Table 21.9: General Services Expenditures

General Service's Owner's Costs	k US$
810 – Departments	**12,280**
811 - General Administration	1,231
812 - Supply Chain	1,295
813 - HR & Training	1,272
814 - ESR	720
815 - Health & Safety	418
816 -Security	875
818 - IT	5,668
819 - Accounting & Financing	800
820 - Logistics / Taxes / Insurance	**5,914**
821 - Freight	5,914
Grand Total	**18,193**

21.1.9 Pre-production and Commissioning Expenditures

The pre-production costs are those of the process plant as mining pre-production costs are covered in Area 526 and Owner’s costs are captured in Areas 811 to 819.

The process plant pre-production includes initial fills as well as salaries and reagents and fuel during the commissioning and ramp-up period to commercial production. Staffing and training of mill personnel is planned progressively in the 12-month period before commissioning.

Pre-production and commissioning expenditures are presented in Table 21.10**Error! Reference source not found.**.

Table 21.10: Pre-Production and Commissioning Expenditures

Area	k US$
950 - Process Plant Pre-Prod. & Commissioning	**9,701**
955 - Process Plant Mgmt and Training	7,091
956 - Process Plant Commissioning	1,425
958 - First Fill	734
959 - Commissioning Spares	451
Grand Total	**9,701**

A 21.5% contingency on all costs was included for a total of $US 90.8 M.

21.2 <u>Sustaining Capital</u>

Sustaining capital of US$459.3 M is required over the life-of-mine ("LoM") for the following main items:

- TDF expansion;
- Mine equipment purchases (additions and replacements);
- Mine development expenditures;
- Process Plant.

A summary of sustaining capital is presented in Table 21.11 and on an annual basis in Table 21.12.

Table 21.11: Summary of Sustaining Capital Costs

Sustaining CAPEX	LoM ($M)	$/t ore	$/lb Cu Payable
Tailings Disposal Facility Expansion	17.56	0.15	0.01
Mine Equipment Purchases	121.49	1.01	0.06
Mine Development Expenditures	291.72	2.43	0.13
Other Plant	28.53	0.24	0.01
Total Sustaining CAPEX	**459.29**	**3.82**	**0.21**

Table 21.12: Sustaining Capital Expenditures

Sustaining CAPEX (M$)	Total	2020	2021	2022	2023	2024	2025	2026	2027	2028	2029	2030	2031	2032	2033	2034	2035	2036	2037	2038	2039	2040	2041	2042	2043	2044	2045	2046	2047
TSF Mgmt.	**17.56**					0.00	0.00	0.00	1.04	0.00	1.09	1.14	1.19	1.25	0.00	1.30	1.35	1.40	0.00	1.46	0.00	1.51	1.56	1.61	1.66	0.00	0.00	0.00	0.00
Water Mgmt.	**0.00**					0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00
Mine Development	**121.49**					28.15	13.99	5.19	13.41	16.00	21.01	8.61	1.87	1.96	1.92	1.88	1.86	1.88	1.85	1.90	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00
Mine Equipment	**291.68**					38.35	28.92	15.25	20.28	8.17	11.27	15.70	7.47	5.73	18.37	5.03	6.09	11.69	11.20	17.90	19.35	18.40	4.99	6.67	5.47	5.82	4.92	2.85	1.78
Process Plant - Other	**28.53**					1.59	1.52	1.46	1.39	1.32	1.25	1.25	1.25	1.25	1.25	1.25	1.25	1.25	1.25	1.25	1.25	1.25	1.25	1.25	1.25	1.25	1.25	0.00	0.00
Total Sustaining Capital	**459.25**					68.09	44.43	21.91	36.11	25.48	34.62	26.70	11.79	10.19	21.54	9.46	10.55	16.23	14.30	22.51	20.60	21.16	7.80	9.53	8.38	7.07	6.17	2.85	1.78

Figure 21.2: Sustaining Capital Expenditures

21.3 Closure Costs and Salvage Value

The closure costs are estimated to US$116 M net and excludes salvage value from plant major equipment.

Closure costs would cover the following activities:

- Tailings reclamation;
- Site closure, dismantling and reclamation;
- Salvaging of plant major equipment;
- Post closure monitoring;
- MDEQ oversight.

The closure cost estimate is presented in Table 21.13 with these costs incurred over a two-year period after commercial operations (i.e. during 2048 and 2049).

Table 21.13: Closure Cost & Salvage Value

Closure Cost Estimate	Unit	Unit Price	Qty	Cost (k $)
TDF Reclamation				
Drain Tube FTB from Texel	sq.m	7.70	10,700,000	82,390
Place and Compact Soil Cover	cu.m	2.25	1,605,000	3,611
Contingency		-	-	12,900
Sub-Total			**12,305,000**	**98,901**
Site Closure & Reclamation				
Place and Compact Soil Cover	cu.m	2	200,000	450
Place and Hydroseed Topsoil	sq.m	2	2,330,000	4,544
Structural Steel Demolition	tonnes	600	2,500	1,500
Concrete Demolition	tonnes	8	35,000	280
Concrete Disposal	tonnes	2	35,000	70
Modular Building Removal	sq.m	50	200	10
Mechanical Pipelines	lot	500,000	1.00	500
Electrical Distribution	lot	500,000	1.00	500
Removal and Disposal of Tanks	lot	10,000	1.00	10
Admin Support	%	0	1.00	1,180
Sub-Total				**9,043**
General Reclamation	**lot**	**56,275**	**1.00**	**56**
Post Closure Monitoring (DCF 5%)	**lot**	**2,900,392**	**1.00**	**2,900**
MDEQ Admin Oversight	**%**	**5.0%**	**1.00**	**5,090**
Total Cost				**115,991**

21.4 Operating Costs

Operating expenditures ("OPEX") are summarized in Table 21.14. The operating costs include mining, processing, General and Administration ("G&A") and royalties. The costs for concentrate transportation to smelters and smelting and refining charges are not considered site operating costs and are therefore excluded from site direct costs.

The transportation costs and smelter conversion charges ("TC/RC") are deducted from gross smelter revenues to estimate the Net Smelter Return ("NSR"). These costs are detailed in Section 19 on Market Studies and Contracts.

The LoM operating cost summary is presented in Table 21.14 and the OPEX by year is presented in Table 21.15. The LoM unit operating cost is estimated at US$1.40/lb of payable copper. The cost profile per lb of copper is relatively stable over the LoM due to the consistent grade profile of the deposit except for the last two years where throughput decreases from the mine.

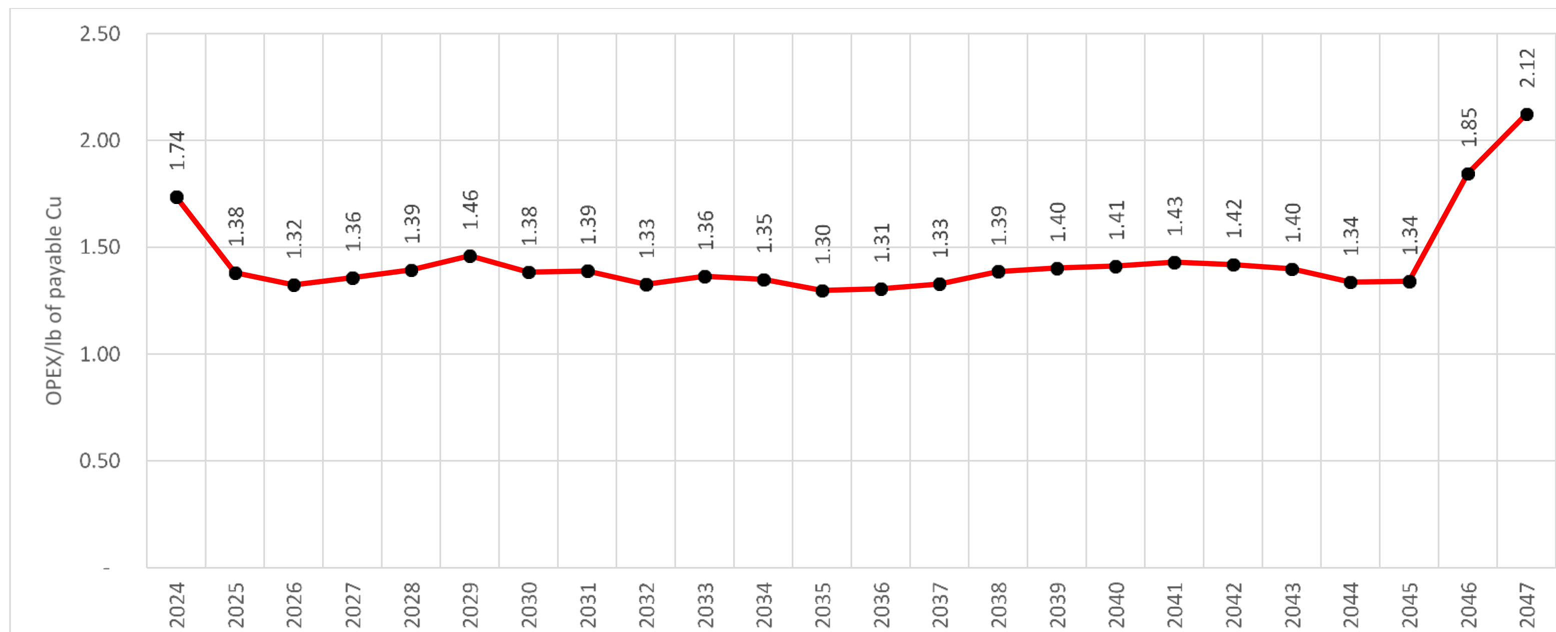
Table 21.14: LoM Operating Cost Summary

LoM OPEX by Area	Total Cost ($M)	Unit Cost ($/tonne milled)	Unit Cost ($/payable lb)	%
Royalties	113	0.94	0.05	3.7%
Mining	2,038	16.96	0.92	66.1%
Processing	740	6.16	0.34	24.0%
General & Administration	193	1.60	0.09	6.3%
Total Site Costs (incl. Royalties)	**3,084**	**25.67**	**1.40**	**100%**

Table 21.15: Annual Operating Costs

OPEX Summary (M$)	Total	2024	2025	2026	2027	2028	2029	2030	2031	2032	2033	2034	2035	2036	2037	2038	2039	2040	2041	2042	2043	2044	2045	2046	2047
Royalties	**113**	3.14	5.52	5.65	8.34	7.96	6.92	5.47	5.68	5.08	4.65	4.43	4.52	4.50	4.28	4.17	4.09	4.08	4.08	4.12	4.20	4.41	4.39	1.89	1.37
Mining	**2,038**	53.27	93.71	92.34	89.85	90.28	93.81	91.74	86.56	90.35	88.80	86.85	85.86	86.95	85.44	87.82	89.24	90.27	90.35	90.13	90.34	91.53	92.40	54.75	44.95
Processing	**740**	19.23	32.85	32.85	32.85	32.85	32.85	32.85	32.85	32.85	32.85	32.85	32.85	32.85	32.85	32.85	32.85	32.85	32.85	32.85	32.85	32.85	32.85	17.30	14.16
G&A	**193**	8.03	8.03	8.03	8.03	8.03	8.03	8.03	8.03	8.03	8.03	8.03	8.03	8.03	8.03	8.03	8.03	8.03	8.03	8.03	8.03	8.03	8.03	8.03	8.03
Total	**3,084**	**83.68**	**140.10**	**138.86**	**139.06**	**139.11**	**141.61**	**138.09**	**133.12**	**136.31**	**134.33**	**132.16**	**131.26**	**132.33**	**130.59**	**132.86**	**134.21**	**135.22**	**135.30**	**135.13**	**135.41**	**136.81**	**137.67**	**81.98**	**68.51**
Unit Cost ($/t milled)	**25.7**	**30.99**	**25.94**	**25.72**	**25.75**	**25.76**	**26.22**	**25.57**	**24.65**	**25.24**	**24.88**	**24.47**	**24.31**	**24.51**	**24.18**	**24.60**	**24.85**	**25.04**	**25.06**	**25.02**	**25.08**	**25.34**	**25.49**	**35.39**	**40.43**
Unit Cost ($/pay. lb Cu)	**1.40**	**1.74**	**1.38**	**1.32**	**1.36**	**1.39**	**1.46**	**1.38**	**1.39**	**1.33**	**1.36**	**1.35**	**1.30**	**1.31**	**1.33**	**1.39**	**1.40**	**1.41**	**1.43**	**1.42**	**1.40**	**1.34**	**1.34**	**1.85**	**2.12**

Figure 21.3: Operating Cost per lb of Payable Copper



21.4.1 Mining Costs

The operating mining costs were evaluated based on the LoM and is supported by supplier quotations, a detailed wage scale and productivity estimates. The mining costs are divided into ten categories that represent the major mining activities. Table 21.16 presents the annual mining costs over the LoM which average $16.96/t.

Table 21.16: Mining Operating Cost Summary

Mine OPEX Summary	LoM Cost (M$)	$/t ore milled	$/lb Payable	%
Mine supervision	97.65	0.81	0.04	4.79%
Drilling & Blasting	703.62	5.86	0.32	34.53%
Stope piping, scaling & serv.	153.99	1.28	0.07	7.56%
Ground support	252.48	2.10	0.11	12.39%
Hauling	174.33	1.45	0.08	8.56%
Mine services and const.	209.16	1.74	0.09	10.27%
Mechanical maintenance	167.91	1.40	0.08	8.24%
Electrical maintenance	191.28	1.59	0.09	9.39%
Technical services	87.12	0.73	0.04	4.28%
Total Mining Cost	**2,037.59**	**16.96**	**0.92**	**100%**

The four main costs for mining are labour (39%), equipment fuel and maintenance (16%), explosives (17%) and ground support (8%).

21.4.2 Processing Costs

The process plant operating costs were evaluated based on estimated reagent consumption rates, supplier quotations, a detailed wage scale and standard industry practice. The process costs are divided into seven categories: labour, reagents, grinding media, liners, maintenance supplies and electrical power. The costs include tailings and water pumping but exclude water treatment costs which are included in the G&A environmental costs.

Total process operating cost summary is presented in Table 21.17 and the annual expenditures over the LoM in Table 21.14.

Reagents are the principal cost item in the mill OPEX represent 31% of cost or US$1.92/t of ore. The reagent consumption rates, reagent prices and resulting unit costs is presented in Table 21.18.

The process plant manpower comprises 75 people, including the laboratory staffing of 7 people.

The power consumption is estimated based on historical power consumption rates for the former White Pine mine. The process plant power includes power for the mill only as power for G&A and mining are provisioned for in each respective budget. The power supply is planned from an owner operated natural gas power plant with an estimated cost of US$0.05/kWh based on current natural gas prices. The power consumption at 15,000 tpd is estimated at 31.0 kWh/t milled.

Table 21.17: Process Operating Cost Summary

Mill OPEX	LoM Cost ($M)	Avg. Cost ($M/yr)	$/t ore	$/lb	%
Mill Labour	134.91	5.62	1.12	0.06	18.2%
Reagents	230.27	9.59	1.92	0.10	31.1%
Grinding Media	37.70	1.57	0.31	0.02	5.1%
Liners	39.74	1.66	0.33	0.02	5.4%
Maintenance Supplies	46.26	1.93	0.39	0.02	6.2%
Operating Supplies	5.34	0.22	0.04	0.00	0.7%
Power	186.17	7.76	1.55	0.08	25.1%
General Other	60.06	2.50	0.50	0.03	8.1%
Total Mill OPEX	**740.45**	**30.85**	**6.16**	**0.34**	**100%**

Table 21.18: Process Plant Reagent Consumption

Reagents	Dosage		Reagent Pricing		Reagent Consumption		Unit Cost (US$/t)
Sodium Isobutyl Xantante (C-3430)	91	g/t	1,840	US$/	491	t/yr	0.17
Methyl Isobutyl Carbinol (MIBC)	45	g/t	2,390	US$/	243	t/yr	0.11
Dowfroth 250 (D-250)	45	g/t	4,000	US$/	243	t/yr	0.18
Alkylaryl Dithiophosphate (A-249)	215	g/t	3,750	US$/	1,161	t/yr	0.81
n-Dodecyl Mercaptan (NDM)	14	g/t	3,840	US$/	76	t/yr	0.05
Sodium Silicates	431	g/t	480	US$/	2,329	t/yr	0.21
Carboxymethyl Cellulose Sodium	144	g/t	2,580	US$/	776	t/yr	0.37
Flocculant	0.3	g/t	3,390	US$/	1	t/yr	0.00
Anti-Scalant	5.5	L/h	4,250	US$/m³	30	m³/yr	0.02
Total							**1.92**

Table 21.19: Grinding Media and Liner Consumption

Grinding Media & Liners	Dosage		Consumable Pricing		Media & Liner Consumption		Unit Cost (US$/t
Ball Mill Grinding Media	248	g/t	1,230	US$/t	305	t/yr	0.31
Regrind Mill Grinding Media	5	g/t	1,766	US$/t	9	t/yr	0.01
Cone Crusher Liners							0.11
Ball Mill Liner							0.22
Total							**0.64**

21.4.3 General and Administration

General and administration ("G&A") includes general management, finance and accounting, supply chain, IT, human resources, health, safety and environment, surface support and corporate and insurance costs.

In most cases, these services represent fixed costs for the site as a whole. The G&A costs exclude certain costs such as transport of concentrates and environmental rehabilitation costs. Water treatment costs are included in environment which represents US$1.80 M/yr over the LoM.

The G&A labor includes 40 people whose total labor cost represents 48% of the G&A OPEX.

A summary of G&A costs is presented in Table 21.20 and the annual expenditures over the LoM in Table 21.14. The average annual G&A budget is US$8.0 M or US$1.60/t of ore.

Table 21.20: General Management and Administration Cost Summary

G&A OPEX by Department	LoM Cost ($M)	Avg. Cost ($M/yr)	$/t ore	$/lb	%
Labour Costs	92.21	3.84	0.77	0.04	47.8%
Training	0.92	0.04	0.01	0.00	0.5%
Insurance	21.89	0.91	0.18	0.01	11.4%
General mgmt. costs	3.00	0.13	0.02	0.00	1.6%
Supply Chain	1.57	0.07	0.01	0.00	0.8%
Surface Support	5.76	0.24	0.05	0.00	3.0%
Information Technology	5.56	0.23	0.05	0.00	2.9%
Human Resources	7.67	0.32	0.06	0.00	4.0%
Health, Safety & Environment (HSE)	54.16	2.26	0.45	0.02	28.1%
Total G&A Costs	**192.74**	**8.03**	**1.60**	**0.09**	**100%**

22. ECONOMIC ANALYSIS

The economic analysis presented in this Report uses an economic model that estimates cash flows on an annual basis for the life of the Project at a level appropriate to a scoping study level of engineering and design.

Cash flow projections are estimated over the LoM based on the sales revenue, operating expenses ("OPEX"), capital expenses ("CAPEX") and other cost estimates. CAPEX is estimated in three categories, initial, sustaining, and closure and reclamation. OPEX estimates include labour, reagents, maintenance, supplies, services, fuel and electrical power. Other costs such as royalties, depreciation and taxes are estimated in accordance with the present stage of the Project.

The financial model results are presented in terms of Net Present Value ("NPV"), payback period, and internal rate of return ("IRR") for the Project. The economic analysis is carried out in real terms (i.e. without inflation factors) in Q3 2019 US Dollars without any project or equipment financing assumptions. The economic results are calculated as of the start of initial capital expenditures with all prior costs treated as sunk costs but considered for purposes of taxation calculations.

22.1 Assumptions

22.1.1 Metal Prices

Metal prices and price scenarios are presented in Section 19.1. The base case copper price for economic evaluation follows a constant price of US$3.00/lb. The silver price is also kept constant at US$16.00/oz.

22.1.2 Fuel

The reference diesel fuel price used for estimating operating costs is US$0.66/L. The diesel fuel price is for off-road or off-highway use by the mine equipment that will not be operated on public roadways. The off-road diesel fuel is not subject to state and federal excise taxes that are applied to retail sales of diesel fuel or for use in vehicles operated on public roadways (Table 22.1). The off-road diesel fuel is dyed red to make it distinguishable. Under the Nonferrous Metallic Minerals Extraction Severance Tax Act, the operation would be exempt of sales tax once in operation.

Table 22.1: Off-Highway Diesel Fuel Price Assumption

Fuel Price	Pre-Production & Operations	
	US$/gal.	US$/L
Retail Diesel Fuel Price	3.000	0.793
Less: Federal Excise Tax	(0.243)	(0.064)
Less: State Tax	(0.263)	(0.069)
Less: Prepaid Sales Tax	-	-
Less: Petroleum Transfer Fee	-	-
Off-Highway Diesel Fuel Price	**2.494**	**0.659**

22.1.3 Exchange Rates

Exchange rates are used to convert certain capital cost and operating cost items in US dollars. The exchange rate assumptions are summarized in Table 22.2.

Table 22.2: Exchange Rate Assumptions

Exchange Rate	Base Value
US$/$C	0.78

22.2 Metal Production and Revenue

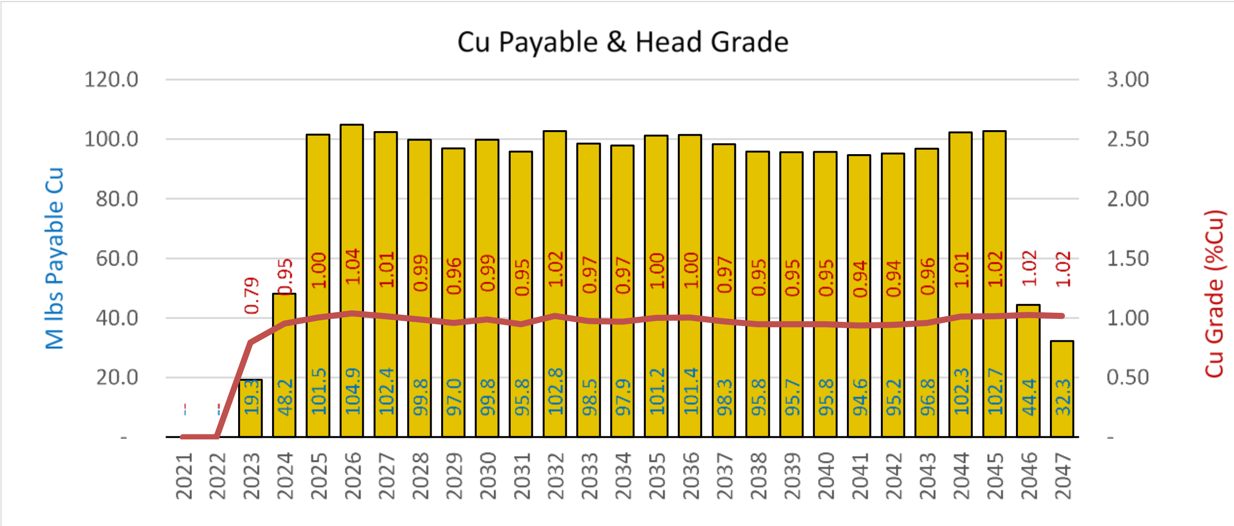
Payable copper produced over the Project life is 1,009 kt (2,224 M lb) with an annual average of 40.4 kt (89 M lb) over the 25-year life which includes 1-year of commissioning and ramp-up. The average payable copper rate is 96.5%. Payable silver production over the LoM is 31.26 M oz with an annual average of 1,250 k oz with an average payable rate of 89.3%. The metal production is presented on an annual basis in Table 22.3.

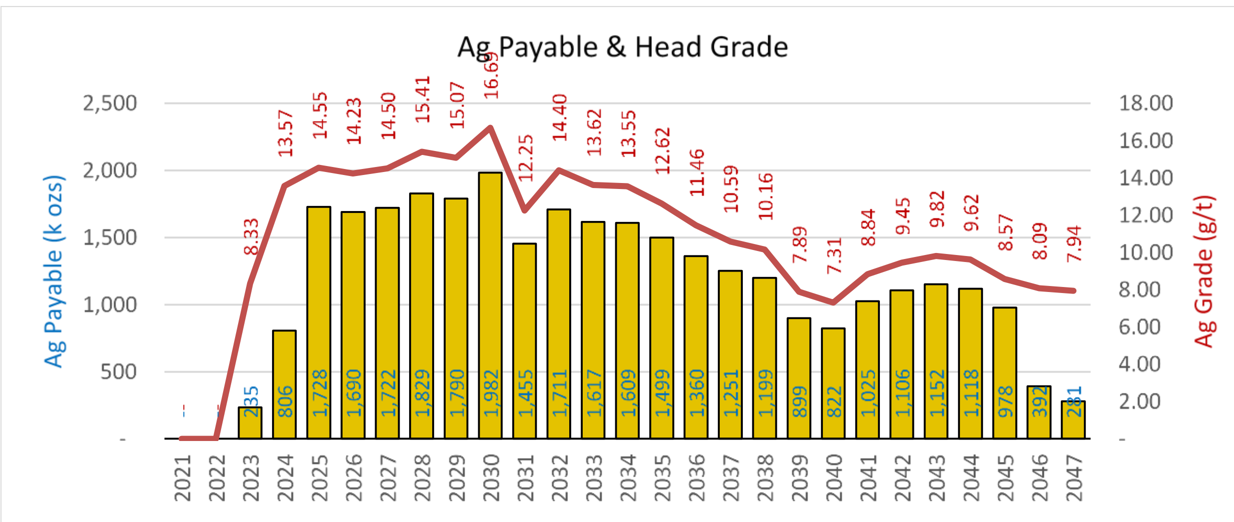
Table 22.3: Metal Production

Production Physicals		Total	2023	2024	2025	2026	2027	2028	2029	2030	2031	2032	2033	2034	2035	2036	2037	2038	2039	2040	2041	2042	2043	2044	2045	2046	2047
Tonnage Processed	kt	**121,405**	1,294	2,700	5,400	5,400	5,400	5,400	5,400	5,400	5,400	5,400	5,400	5,400	5,400	5,400	5,400	5,400	5,400	5,400	5,400	5,400	5,400	5,400	5,400	2,316	1,695
Cu Head Grade	% Cu	**0.98**	0.79	0.95	1.00	1.04	1.01	0.99	0.96	0.99	0.95	1.02	0.97	0.97	1.00	1.00	0.97	0.95	0.95	0.95	0.94	0.94	0.96	1.01	1.02	1.02	1.02
Ag Head Grade	g/t	**11.80**	8.33	13.57	14.55	14.23	14.50	15.41	15.07	16.69	12.25	14.40	13.62	13.55	12.62	11.46	10.59	10.16	7.89	7.31	8.84	9.45	9.82	9.62	8.57	8.09	7.94
Concentrate (dry)	k dmt	**3,421**	29.6	74.1	156.1	161.3	157.5	153.5	149.1	153.5	147.4	158.0	151.4	150.6	155.6	155.9	151.2	147.4	147.1	147.3	145.5	146.5	148.9	157.3	157.9	68.3	49.6
Concentrate (wet)	k wmt	**3,759**	32.5	81.4	171.5	177.3	173.1	168.7	163.9	168.6	162.0	173.7	166.4	165.5	171.0	171.3	166.2	162.0	161.7	161.9	159.9	161.0	163.6	172.9	173.5	75.1	54.5
Cu Contained Metal	kt	**1,188**	10	26	54	56	55	53	52	53	51	55	53	52	54	54	53	51	51	51	51	51	52	55	55	24	17
Cu Contained Metal	M lbs	**2,619**	23	57	119	124	121	118	114	117	113	121	116	115	119	119	116	113	113	113	111	112	114	120	121	52	38
Ag Contained Metal	k ozs	**46,068**	347	1,178	2,526	2,471	2,518	2,675	2,617	2,898	2,127	2,501	2,365	2,352	2,192	1,989	1,838	1,764	1,369	1,268	1,534	1,641	1,705	1,671	1,488	602	432
Cu Recovery	%	**88.00**	88.0	88.0	88.0	88.0	88.0	88.0	88.0	88.0	88.0	88.0	88.0	88.0	88.0	88.0	88.0	88.0	88.0	88.0	88.0	88.0	88.0	88.0	88.0	88.0	88.0
Ag Recovery	%	**76.00**	76.0	76.0	76.0	76.0	76.0	76.0	76.0	76.0	76.0	76.0	76.0	76.0	76.0	76.0	76.0	76.0	76.0	76.0	76.0	76.0	76.0	76.0	76.0	76.0	76.0
Cu Metal Production	kt	**1,045**	9.1	22.6	47.7	49.3	48.1	46.9	45.6	46.9	45.0	48.3	46.3	46.0	47.6	47.6	46.2	45.1	45.0	45.0	44.5	44.8	45.5	48.1	48.3	20.9	15.2
Cu Metal Production	M lbs	**2,305**	20.0	49.9	105.2	108.7	106.1	103.4	100.5	103.4	99.3	106.5	102.0	101.5	104.8	105.0	101.9	99.3	99.1	99.2	98.0	98.7	100.3	106.0	106.4	46.0	33.4
Ag Metal Production	k ozs	**35,012**	263	895	1,920	1,878	1,914	2,033	1,989	2,203	1,616	1,901	1,797	1,787	1,666	1,512	1,397	1,341	1,041	964	1,166	1,247	1,296	1,270	1,131	458	329
Cu Payable Rate	%	**96.5**	96.50	96.50	96.50	96.50	96.50	96.50	96.50	96.50	96.50	96.50	96.50	96.50	96.50	96.50	96.50	96.50	96.50	96.50	96.50	96.50	96.50	96.50	96.50	96.50	96.50
Ag Payable Rate	%	**89.3**	89.15	90.00	90.00	90.00	90.00	90.00	90.00	90.00	90.00	90.00	90.00	90.00	90.00	90.00	89.56	89.40	86.37	85.26	87.96	88.67	88.91	88.05	86.53	85.60	85.44
Cu Payable Metal	kt	**1,009**	8.7	21.8	46.0	47.6	46.5	45.3	44.0	45.3	43.5	46.6	44.7	44.4	45.9	46.0	44.6	43.5	43.4	43.4	42.9	43.2	43.9	46.4	46.6	20.2	14.6
Cu Payable Metal	M lbs	**2,224**	19.3	48.2	101.5	104.9	102.4	99.8	97.0	99.8	95.8	102.8	98.5	97.9	101.2	101.4	98.3	95.8	95.7	95.8	94.6	95.2	96.8	102.3	102.7	44.4	32.3
Ag Payable Metal	k ozs	**31,257**	235	806	1,728	1,690	1,722	1,829	1,790	1,982	1,455	1,711	1,617	1,609	1,499	1,360	1,251	1,199	899	822	1,025	1,106	1,152	1,118	978	392	281
Operating Periods	yrs	**25.0**	1.0	1.0	1.0	1.0	1.0	1.0	1.0	1.0	1.0	1.0	1.0	1.0	1.0	1.0	1.0	1.0	1.0	1.0	1.0	1.0	1.0	1.0	1.0	1.0	1.0

Note: 2023 is part of pre-production and commissioning period

Figure 22.1: LoM Payable Metal Profile





The commissioning and ramp-up schedule to achieve the steady state throughput of 15 ktpd takes place over a two-year period (2023 and 2024). The first year is considered pre-production and commissioning with an average 3.5 ktpd. The first year of operation is continued ramp-up with an average throughput of 7.4 ktpd.

22.3 Capital Expenditures

The capital expenditures include initial CAPEX as well as sustaining capital to be spent after commencement of commercial operations.

22.3.1 Initial Capital Expenditures

The CAPEX for Project construction, including concentrator, mine equipment, support infrastructure, pre-production activities and other direct and indirect costs is estimated to be US$512.5 M. The total initial Project capital includes a contingency of US$90.8 M which is 21.5% of the total CAPEX before contingency excluding pre-production revenue of US$55.76 M. Net of pre-production revenue, the initial CAPEX is estimated at US$456.7 M as presented in Table 22.4. The initial Project CAPEX is spent over a period of four years (2020 to 2023).

Table 22.4: Initial Capital Expenditure Summary

Initial CAPEX	US$ k
000 - General	212
100 - Infrastructure	31,214
200 - Power & Electrical	64,051
300 - Water & TSF Mgmt.	15,078
400 - Mobile Equipment	44,453
500 - Mine Infrastructure	39,183
600 - Process Plant	47,359
700 - Construction Indirects	25,739
800 - General Services & Owner's Costs	22,251
900 - Pre-Production, Commissioning	136,190
Sub-Total Before Contingency	**421,671**
Contingency 21.5%	**90,788**
Total Incl. Contingency	**512,460**
Less: Pre-Production Revenue	**(55,715)**
Total Incl. Contingency & Pre-Prod. Revenue	**456,745**

22.3.2 Sustaining Capital Expenditures

Sustaining capital expenditures during operations are required for additional mine equipment purchases, mine development work, and tailings storage expansion. The total life-of-mine ("LoM") sustaining CAPEX is estimated at $459.3 M with the breakdown presented in Table 22.5.

Table 22.5: Sustaining Capital Expenditure Summary

Sustaining CAPEX	LoM ($M)	$/t ore	$/lb Cu Payable
Tailings Disposal Facility Expansions	17.56	0.15	0.01
Mine Equipment Purchases	121.49	1.01	0.06
Mine Development Expenditures	291.72	2.43	0.13
Other Plant Expenditures	28.53	0.24	0.01
Total Sustaining CAPEX	**459.29**	**3.82**	**0.21**

Note: Ore tonnage and payable copper unit costs during operations period only

22.3.3 Closure and Reclamation

The reclamation and closure cost estimate include the following scope:

- Demolition of infrastructures;
- Salvaging of major equipment;
- Site reclamation, principally for the TDF;
- Post closure monitoring.

The closure and reclamation activities are planned over a two-year period at the end of the mine life (2048 and 2049) with an overall estimate of US$116 M without any salvage value considered.

Table 22.6: Closure and Reclamation Cost Estimate by Stage

Closure Cost Estimate	Cost ($k)
TDF Reclamation	98,901
Site Closure & Reclamation	9,043
General Reclamation	56
Salvage Value	Nil
Post Closure Monitoring	2,900
MDEQ Admin Oversight	5,090
Total Cost	**115,991**

22.3.4 Working Capital

Working capital ("WC") is required to finance supplies in inventory. Given the accessibility of the site, the working capital requirements are considered low compared to remote operations and has been excluded for this PEA.

22.4 Operating Cost Summary

OPEX includes mining, processing, G&A services, concentrate transportation and concentrate treatment and refining charges. The concentrate transportation, treatment charges and refining are deducted from gross revenues to calculate the Net Smelter Return ("NSR"). The NSR for the Project during operations is estimated at US$6,444 M excluding US$55.7 M of NSR generating during pre-production and treated as a reduction of initial capital expenditures. The average NSR over the LoM is US$2.92/lb of payable copper net of silver credits. Detailed operating cost budgets have been estimated from first principles based on detailed wage scales, consumable prices, fuel prices and productivities. The operating costs are detailed in Section 21 of this Report. The average OPEX over the LoM is US$25.67/t of ore or US$1.40/lb of payable copper with mining representing 66% of the total OPEX, or US$16.96/t of ore. A summary of operating cash flow and operating costs is presented in Table 22.7.

Table 22.7: Operating Cost & Summary

Operating Cash Flow	LoM (US$ M)	US$/t ore	US$/lb Cu Payable
Cu Revenue	6,615	55.07	3.00
Ag Credits	496	4.13	0.23
Revenue	7,111	59.20	3.23
Concentrate Transportation Costs	(260)	(2.17)	(0.12)
Treatment & Refining Charges	(407)	(3.39)	(0.18)
Net Smelter Return	6,444	53.65	2.92
Royalties	(113)	(0.94)	(0.05)
Mining Costs	(2,038)	(16.96)	(0.92)
Processing Costs	(740)	(6.16)	(0.34)
G&A Costs	(193)	(1.60)	(0.09)
Total OPEX (including royalties)	**(3,084)**	**(25.67)**	**(1.40)**
Operating Cash Flow	**3,359**	**27.97**	**1.52**

Note: Ore tonnage and payable copper unit costs during operations period only

Table 22.8: LoM C1 & C3 Cost Summary

LoM Costs	Total Cost (US$M)	Unit Cost $/tonne milled)	Unit Cost ($/payable lb)
Mining	2,038	16.96	0.92
Processing	740	6.16	0.34
G&A	193	1.60	0.09
Offsite Costs (transport, TC/RCs)	668	5.56	0.30
By-product credits	(496)	(4.13)	(0.23)
C1 Cost	**3,142**	**26.16**	**1.43**
Depreciation and Closure	575	4.79	0.26
Royalty Costs	113	0.94	0.05
C3 Cost	**3,831**	**31.89**	**1.74**

22.5 Taxes and Royalties

22.5.1 Income Tax

Income for tax purposes is defined as metal revenues minus operating expenses, royalties, Michigan severance tax, reclamation and closure expenses, depreciation and depletion. Depreciation is calculated using the Modified Accelerated Cost Recovery System ("MACRS") method and the unit of production method in accordance with the current U.S. Internal Revenue Service ("IRS") regulations. The federal income tax rate based on new tax reform is 21%. There is no state income tax which is exempt under the Michigan Nonferrous Metallic Minerals Extraction Severance Tax Act. The estimated federal tax paid over the Project life is US$246.6 M.

22.5.2 Michigan Severance Tax

The Nonferrous Metallic Minerals Extraction Severance Tax Act ("MST"), PA 410 of 2012, as amended, levies a specific tax on certain nonferrous metallic minerals for mineral producing properties in the state of Michigan. The tax levied on the eligible mine owner is the Minerals Severance Tax and includes exemption from property taxes levied in this state, taxes levied under part 2 of the Income Tax Act, PA 281 of 1967, Sales tax as levied under PA 167 of 1933, and Use tax as levied under PA 94 of 1937.

The minerals Severance Tax is 2.75% of gross income from mining or the net smelter return, less third-party royalty payments. Over the LoM, the Severance Tax represents US$174.1 M.

22.5.3 Royalties

The owners of the mineral rights (Longyear Mineral Lease) are entitled to fixed annual rental payments and royalty payments. The annual rental fees are increasing over time:

- First through fifth anniversary: US$25,000;
- Sixth through seventh anniversary: US$30,000;
- Eighth and thereafter anniversary: US$1,000,000;
- The rental payments are deductible from the royalty payments.

Royalties are paid on 82% of contained metal in ore mined in the Longyear blocks. The royalty is based on a sliding scale linked to the COMEX price of copper starting at 2% and increasing by one basis point for every cent increase above $3.25/lb. The silver royalty is 2.5% and increasing by same percentage above

$18.00/oz with a cap of 4%. The rental payments are deductible from royalty payments. Over the LoM, the total payments are estimated at US$17.4 M.

Under a transaction with Osisko Gold Royalties, Osisko is to receive a 1.5% NSR royalty which is fixed regardless of copper price. Over the LoM, the Osisko royalty represents a cost of US$97.5 M.

22.6 Economic Model Results

The economic model results are presented in terms of NPV, IRR, and payback period in years for recovery of the initial CAPEX. These economic indicators are presented on both pre-tax and after-tax basis. The NPV is presented both undiscounted ($NPV_{0\%}$) and using a discount rate of 8% ($NPV_{8\%}$). The annual cash flow is summarized in Table 22.10 and graphically in Figure 1.2. A cash flow waterfall for the Project is summarized in Figure 22.3.

The undiscounted after-tax cash flow is estimated at US$1,907 M for the Project. The economic results on a before-tax and after-tax basis are presented in Table 22.9.

Table 22.9: Economic Results Summary

Economic Results Summary	Unit	Before-Tax Results	After-Tax Results
NPV 0%	$M	2,327	1,907
NPV 8%	$M	557	416
IRR	%	19.2%	16.8%
Payback	yrs	4.5	5.2

Figure 22.2: After-Tax Annual Project Cash Flow

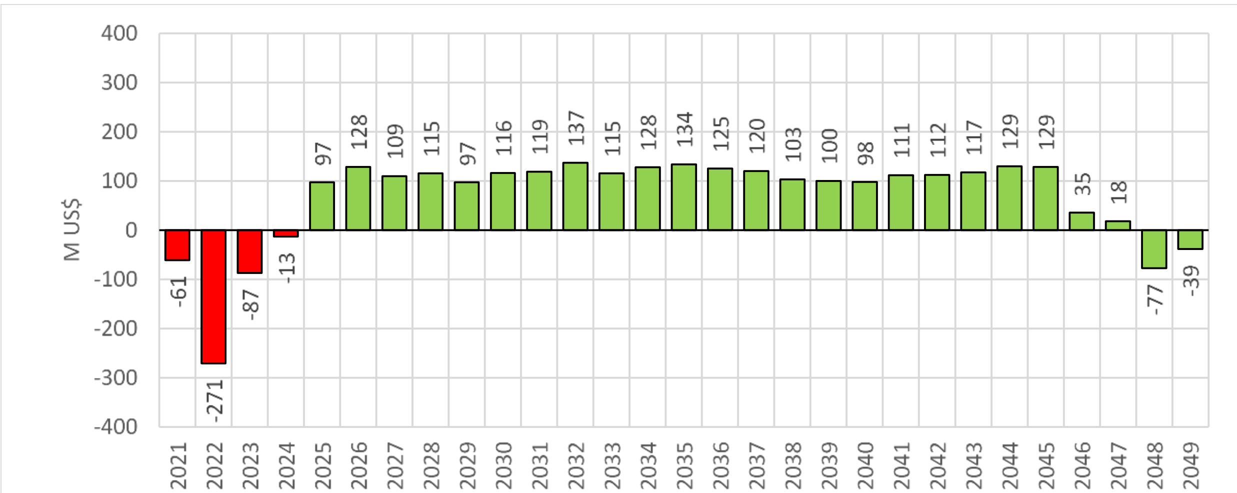


Figure 22.3: After Tax Project Cash Flow Waterfall

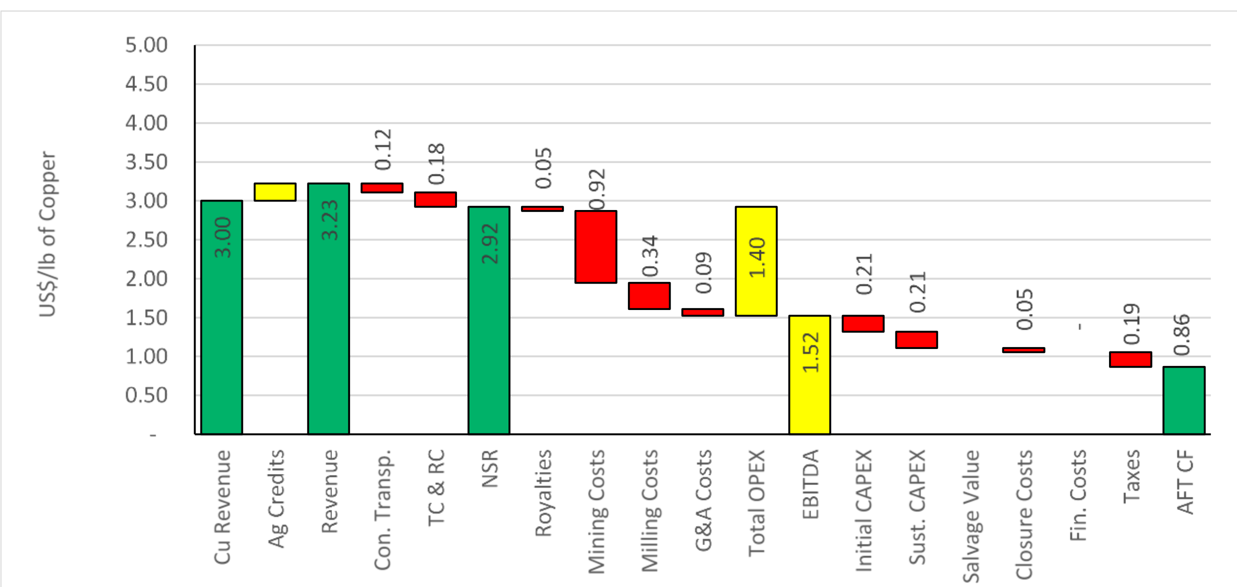


Table 22.10: After-Tax Annual Cash Flow Summary

Cash Flow (M$)	Total	2020	2021	2022	2023	2024	2025	2026	2027	2028	2029	2030	2031	2032	2033	2034	2035	2036	2037	2038	2039	2040	2041	2042	2043	2044	2045	2046	2047	2048	2049
Revenue	**7,111**					157	332	342	335	329	319	331	311	336	321	319	328	326	315	307	301	300	300	303	309	325	324	140	101		
Con. Transport Costs	**-261**					-6	-12	-12	-12	-12	-11	-12	-11	-12	-12	-12	-12	-12	-12	-11	-11	-11	-11	-11	-11	-12	-12	-5	-4		
TC / RC	**-407**					-9	-19	-19	-19	-19	-18	-19	-18	-19	-18	-18	-19	-19	-18	-18	-17	-17	-17	-17	-18	-19	-19	-8	-6		
Net Smelter Return	**6,443**					143	301	310	304	298	290	301	282	304	291	290	297	295	285	278	273	272	272	275	280	294	293	126	92		
Royalties	**-113**					-3	-6	-6	-8	-8	-7	-5	-6	-5	-5	-4	-5	-4	-4	-4	-4	-4	-4	-4	-4	-4	-4	-2	-1		
Mining Costs	**-2,038**					-53	-94	-92	-90	-90	-94	-92	-87	-90	-89	-87	-86	-87	-85	-88	-89	-90	-90	-90	-90	-92	-92	-55	-45		
Processing Costs	**-740**					-19	-33	-33	-33	-33	-33	-33	-33	-33	-33	-33	-33	-33	-33	-33	-33	-33	-33	-33	-33	-33	-33	-17	-14		
G&A Costs	**-192.74**					-8	-8	-8	-8	-8	-8	-8	-8	-8	-8	-8	-8	-8	-8	-8	-8	-8	-8	-8	-8	-8	-8	-8	-8		
Total Operating Costs	**-3,084**					-84	-140	-139	-139	-139	-142	-138	-133	-136	-134	-132	-131	-132	-131	-133	-134	-135	-135	-135	-135	-137	-138	-82	-69		
Working Capital	**0**																														
Operating Cash Flow	**3,359**					59	161	171	165	159	148	162	149	168	157	158	166	163	155	145	138	136	136	140	144	157	155	44	23		
Investment Capital	**-457**	-37	-61	-271	-87																										
Sustaining Capital	**-459**					-68	-44	-22	-36	-25	-35	-27	-12	-10	-22	-9	-11	-16	-14	-23	-21	-21	-8	-10	-8	-7	-6	-3	-2		
Salvage Value	**0**																														
Closure Costs	**-116**																													-77	-39
Taxes	**-420.6**					-4	-20	-21	-19	-18	-17	-20	-18	-21	-20	-20	-21	-21	-20	-19	-18	-18	-17	-18	-19	-21	-20	-6	-3		
Project Cash Flow	**1,907**	-37	-61	-271	-87	-13	97	128	109	115	97	116	119	137	115	128	134	125	120	103	100	98	111	112	117	129	129	35	18	-77	-39

Notes:
- *Pre-production revenue included in investment capital offsetting pre-production costs.*
- *Taxes include federal income tax and Michigan Severance Tax.*

22.7 Sensitivity Analysis

The sensitivity analysis of the economic model was tested with respect to metal prices, initial CAPEX and OPEX for each case. The value of each parameters was raised and lowered 20% to evaluate the impact of such changes on the NPV and IRR. The pre-tax sensitivity results are presented in Table 22.11 and the after-tax sensitivity results in Table 22.12.

The after-tax NPV of the Project is most sensitive to changes in revenue, which is manifested as changes in metal prices or metal grades. For example, a 20% increase in copper price or copper grade increases the $NPV_{8\%}$ from US$415.6 M to US$831.8 M. Similarly, a decrease of 20% in copper price or copper grade reduces the $NPV_{8\%}$ to -US$6.4 M.

Table 22.11: Pre-Tax Sensitivity Results

Variance	Before-Tax Results			
	$NPV_{0\%}$ (M$)	$NPV_{8\%}$ (M$)	IRR (%)	Payback (yrs)
Metal Price Sensitivities				
20%	3,724	1033.2	26.7%	3.1
10%	3,031	797.7	23.1%	3.7
0%	2,327	557.2	19.2%	4.5
-10%	1,623	316.2	14.8%	6.1
-20%	918	75.2	9.8%	8.5
Initial Capital Cost Sensitivities				
20%	2,259	501.0	17.2%	5.1
10%	2,293	529.1	18.1%	4.8
0%	2,327	557.2	19.2%	4.5
-10%	2,362	585.4	20.3%	4.3
-20%	2,396	613.5	21.6%	4.0
Operating Cost Sensitivities				
20%	2,177	505.8	18.3%	4.8
10%	2,252	531.5	18.7%	4.6
0%	2,327	557.2	19.2%	4.5
-10%	2,403	582.9	19.6%	4.4
-20%	2,478	608.6	20.1%	4.3

Table 22.12: After-Tax Sensitivity Results

Variance	After-Tax Results			
	$NPV_{0\%}$ (M$)	$NPV_{8\%}$ (M$)	IRR (%)	Payback (yrs)
Metal Price Sensitivities				
20%	3,126	831.8	23.8%	3.6
10%	2,520	625.9	20.5%	4.2
0%	1,907	415.6	16.8%	5.2
-10%	1,292	204.8	12.7%	7.0
-20%	676	-6.4	7.8%	9.7
Initial Capital Cost Sensitivities				
20%	1,845	361.4	15.0%	5.9
10%	1,876	388.5	15.8%	5.6
0%	1,907	415.6	16.8%	5.2
-10%	1,938	442.7	17.8%	4.9
-20%	1,969	469.7	19.0%	4.6
Operating Cost Sensitivities				
20%	1,772	369.3	15.9%	5.6
10%	1,839	392.4	16.3%	5.4
0%	1,907	415.6	16.8%	5.2
-10%	1,974	438.7	17.2%	5.0
-20%	2,042	461.9	17.6%	4.9

Figure 22.4: After-Tax NPV8% Sensitivity

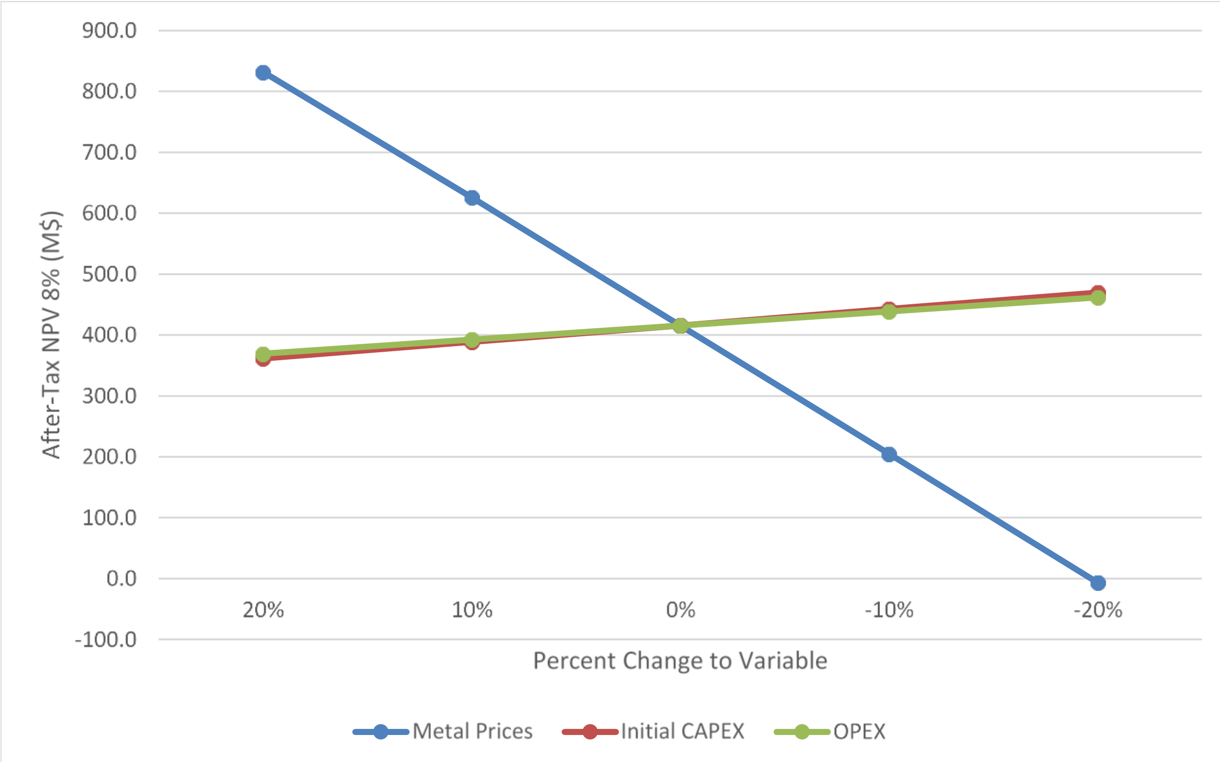
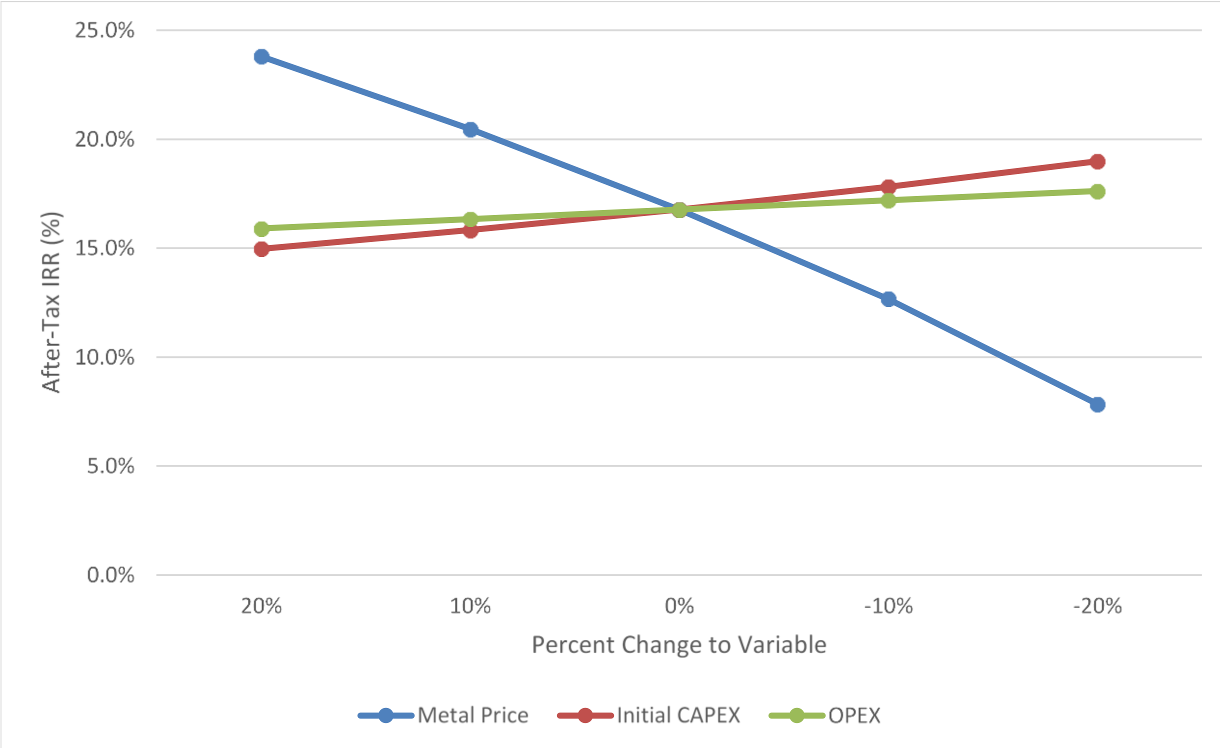


Figure 22.5: After-Tax IRR Sensitivity



23. ADJACENT PROPERTIES

There are no other mineral exploration or development projects adjacent to the White Pine North Project area.

24. OTHER RELEVANT DATA AND INFORMATION

The reader is cautioned that this Preliminary Economic Assessment ("PEA") is preliminary in nature as it includes Inferred Mineral Resources that are too speculative geologically to have the economic considerations applied to them that would enable them to be categorized as mineral reserves, and there is no certainty that the PEA will be realized.

This Report has several cut-off dates for information:

- The effective date of the Current Mineral Resource is August 30, 2019;
- The effective date of this Report is September 22, 2019;
- The amounts are in US$ dollars of Q3 2019 assuming the construction to start in Q1 2020 and the production starting Q3 2023.

Additional Information is not required to make the Technical Report understandable and not misleading.

25. INTERPRETATION AND CONCLUSIONS

25.1 Geology and Mineral Resources

G Mining Services Inc. (“GMSI”) has prepared a Mineral Resource estimate for the White Pine North Project based on data provided up to and including March 2015. No new scientific or technical data has been acquired since March 2015; therefore this mineral resource can be considered as current and effective as of August 30, 2019.

The resource estimate was prepared in accordance with CIM Standards on Mineral Resources and Reserves (adopted May 10, 2014) is reported in accordance with Canadian National Instrument 43-101 - *Standards of Disclosure for Mineral Projects* (“NI 43-101”). The Mineral Resource estimate was prepared under the supervision of Mr. Réjean Sirois, Eng. of GMSI, Vice President, Geology and Resources, an independent “qualified person” (“QP”) as defined in NI 43-101. Geovia GEMS™ and Leapfrog GEO™ software were used to facilitate the Resource estimation process.

In the process of completing the Mineral Resource estimate of the White Pine North Project, GMSI came to the following conclusions:

- GMSI reviewed the available data used in the Mineral Resource estimate, including drill logs, assay certificates, down-hole surveys and additional supporting information sources. GMSI concludes that the drill hole database could be used with confidence in the Mineral Resource Estimate (“MRE”);
- The MRE is based on a database that includes 526 drill holes from available historical drilling by Copper Range Company ("CRC"), and an additional 42 diamond drill holes (with 18 additional wedges) in HQ and NQ diameter core completed by Highland Copper Company Inc. (“Highland”). A total of 274,914 meters (“m”) was drilled by the companies between 1956 and 2015;
- The modelling of the copper mineralization horizons was based on the footwall and hanging wall of the three selected “columns” (sedimentary sequences), namely the Parting Shale (“PS”), the Full Column (“FC”) and the Upper Shale (“US”). These columns were modelled with a minimum true thickness of 2 m. Only the PS column was reported as a Mineral Resource;
- The statistical analysis of the copper and silver assays revealed that the use of grade capping was not necessary;
- Copper and silver uncapped raw assays were composited to the full thickness of the column;
- The variography study based on the zone composites highlighted a weakly anisotropic distribution of copper towards the south-east in the PS column, and a low nugget effect on copper and silver grades;

- The block size dimension (25 m x 25 m x 5 m) was chosen to ensure sufficient definition of mineralisation during block modeling. Since block height is set at 5 m, and that columns have mean heights between 2.18 and 3.02 m, a percent attribute was used in the grade interpolation process. This percent attribute is used when reporting Mineral Resources;
- Grade estimation was undertaken using Ordinary Kriging ("OK") and Inverse Distance Squared ("ID2") into a percentage block model based on the wireframes of the three columns. A three-pass estimation strategy was adopted, with increasingly large search ellipses and relaxed estimation parameters;
- The block model was validated visually and statistically and was found to be a good representation of the composites;
- The Mineral Resources were classified in Indicated and Inferred Mineral Resources, based primarily on estimation pass, and other considerations such as drill spacing, quality of historical data and confidence in grade continuity;
- A 300 m buffer zone around existing workings was excised from the Mineral Resource and, only blocks within mineral leases where Highland has a greater than 25% ownership of the mineral rights were classified as Mineral Resources.
- An underground room-and-pillar mining scenario is judged to be the most adapted to the geometry and dip of the PS, as well as to the tonnage of the deposit;
- The following conceptual mining parameters were used to calculate block values: 1) A flat net smelter return ("NSR") royalty rate of $0.05/lb. Cu payable was applied, which incorporates two royalties on the project (Osisko Gold Royalty and Great Lakes Royalty), 2) No mining loss/dilution, 3) Copper price of $3.00/lb and a silver price of $16/oz, 4) Recovery of 88% for copper and 73.5% for silver, 5) A payable rate of 96.5% for copper and 89.3% for silver, 6) A cut-off grade of 0.9% Cu and 7) Operating costs based on an operating plant at the White Pine site;
- The White Pine North Deposit Underground Indicated Mineral Resources are reported at 133.4 Mt grading an average 1.07% Cu and 14.9 g/t Ag containing 3.15 billion pounds of copper and 63.8 M oz of silver using a lower cut-off grade of 0.9% Cu for the Parting Shale column. Inferred Mineral Resources are reported at 97.2 Mt grading an average 1.03% Cu and 8.7 g/t Ag containing 2.21 billion pounds of copper and 27.2 M oz of Ag using a cut-off grade of 0.9% Cu;
- Mineral Resources that are not Mineral Reserves do not have demonstrated economic viability. The estimate of Mineral Resources may be materially affected by environmental, permitting, legal, title, taxation, socio- political, marketing, or other relevant issues.

GMSI concludes that the Resource evaluation reported in the present Technical Report is a reasonable representation of the Mineral Resources found in the White Pine North Copper Project at the current level of sampling. GMSI believes that there are no significant risks associated with the Project's Mineral Resource estimate, and the varying uncertainties are identified by their respective resource classifications (Indicated and Inferred).

25.2 Mining and Mineral Reserves

Based on geotechnical information, White Pine mine history and mineralization geometry, an underground room-and-pillar method is selected for the White Pine North deposit. This mining method allows for both a good ore selectivity and productivity. However, a series of pillars are left in place to provide roof stability. The mining design was based on a mining rate of approximatively 5.4 Mt/yr. Historically the old White Pine Mine has reached the proposed mining rate. In addition, many assumptions are based on historical data from the old White Pine Mine.

The main conclusions on the mining and mineral reserve estimation are as follows:

- The production schedule is based on mining a fixed target of 5.4 Mt/yr. To achieve this annual production, seven to fourteen production panels must be in production simultaneously;
- The mining method consists of the extraction of a series of entries and cross cuts in the ore leaving pillars in place to support the back. The entries, cross cuts and pillars are sized using a geotechnical analysis of the rock, and experience from the old White Pine Mine with similar ground conditions.
- No geotechnical investigation has been conducted on the underground operations at White Pine North since the closure of the former White Pine Mine. The previous geotechnical work carried out during the operation of the old white pine mine was analyzed and it was used to produce this preliminary study. In addition, a back analysis of the old White Pine was done by Itasca at the beginning of the project.;
- The Mineral Resource included in the mine plan comprises 122 Mt at a copper grade of 0.98%Cu and 11.8 g/t of silver and containing 2.67 billion pounds of copper and 46 M oz of silver.
- Mine equipment selection requires low-profile equipment. Drilling will be done with a low-profile hydraulic-electric jumbo with 2 booms, mucking with 10 t and ground support installed with 1-boom electric-hydraulic bolters. Material handling consists of twelve rock breaker loading stations that feed onto 42 in secondary conveyors located in the stopes which transfer to the main conveyors which transport the ore to the ore storage bins at surface;

- The mine will be accessed via an open box-cut to establish a portal at the mine entrance from surface. Only two drift are excavated from the portal for the first 35 m deep. Then 4 drift are excavated until the first ventilation raise located at a depth of 189 m. From this ventilation raise to the beginning of the West mine section, 6 parallel drift will be excavated to allow a high ventilation flow rate to the Mine. All drifts are set at a width of 6.1 m, and their height varies from a minimum of 3.5 m to a maximum of 6.1 m;
- To achieve and maintain an adequate level of production, the panel must contain at least 12 rooms (headings) in operation simultaneously. The mining cycle includes drilling, blasting, ore mucking, ore transportation to a rock breaker and the stope conveyor, scaling and finally ground support. The mining of the room will be done in two pass approach. In the first pass, larger pillars are left in place. Mining recovery of the first pass is 40%. Once the first is completed, the size of the pillars is reduced to an average mining recovery of 57%;

25.3 Infrastructure

The Copperwood Project requires several infrastructure elements to support the mining and processing operations.

The infrastructure planned for the project includes the following:

- Public access road (County Road 64);
- Site access roads;
- Power generation plant;
- Site electrical distribution;
- Gatehouse;
- Communications network;
- Lake Superior water intake tie-ins;
- Potable water treatment plant tie-ins;
- Sewage treatment tie-ins;
- Covered box-cut for mine access;
- Ore stockpile pad;
- Truck shop, wash bay, warehouse and offices;
- Fuel storage;

- Mill offices and metallurgical laboratory;
- Administration office and assay laboratory;
- Concentrate transload facility;
- Tailings Disposal Facility;
- Effluent Treatment Plant.
- Event pond ditches for surface water management at mill site.

25.4 Environmental and Permitting

Baseline Studies and Impact Analysis

- Highland identified information gaps in 2014 and initiated, but did not complete, required environmental studies
- Legacy data from CRC archives useful for reference but dated and insufficient to fulfill legal requirements for permitting
- Time frame to complete studies, analyses and applications is 12 months.

Permitting Requirements

- Michigan's Public Act 451, the Natural Resource and Environmental Protection Act, sets the framework for all major permit requirements;
    - No Federal permits required under Michigan's delegated authorities.
- Required major permits for the White Pine North Project:
    - Part 632 Nonferrous Mining;
    - Part 31 Wastewater Discharge
    - Part 55 Air Permit;
    - Part 301 Inland Lakes and Streams;
    - Part 303 Wetlands;
    - Part 315 Dam Safety.
- Based on recent Highland experience with the Copperwood Project, estimate 6 to 9 months from application to issue permits.

25.5 Capital Expenditures, Operating Expenditures and Economic Analysis

- The capital expenditures ("CAPEX") for Project construction, including concentrator, mine equipment, support infrastructure, pre-production activities and other direct and indirect costs is estimated to be $US 512.5 M. The total initial Project capital includes a contingency of $US 91 M which is 21.5% of the total CAPEX before contingency excluding pre-production revenue of $US 56 M. Net of pre-production revenue, the initial CAPEX is estimated at $US 456.7 M;
- Sustaining CAPEX during operations are required for additional mine equipment purchases, mine development work, tailings storage expansion and other plant. The total life-of-mine ("LoM") sustaining CAPEX is estimated at $US 456.3 M;
- The NSR for the project during operations is estimated at $US 6,444 M excluding $US 55.7 M of NSR generating during pre-production and treated as pre-production revenue. The average NSR over the LoM is $US 2.92/lb. of payable copper;
- The average operating expenditures ("OPEX") over the LoM is $US 25.67/t of ore or $US 1.40/lb. of payable copper with mining representing 66.1% of the total OPEX, or $US 16.96/t of ore;
- The undiscounted after-tax cash flow is estimated at $US 1,907 M for the Project. The pre-tax NPV8% is estimated at $US 557 M with an 19.2% IRR and 4.5 year payback period. Similarly, the after-tax NPV8% is estimated at $US 416 M with an 16.8% IRR and 5.2 year payback period.

25.6 Risks and Opportunities

The risks and opportunities identification and assessment process is iterative and has been applied throughout the Feasibility Study.

Like all projects, there remain risks and opportunities that could affect the economic results of the Project. Many of the risks and opportunities are general to mining projects and some are specific to the Project which typically need additional information, testing or engineering to confirm assumptions and parameters.

25.6.1 Risks

The risks for the Project that are general or specific include:

- Permit acquisition or delays;
- Ability to attract experienced professionals;
- Declining metal prices;
- Faults creating offsets in the mineralization;

25.6.2 Opportunities

The White Pine North Project has several opportunities that have not been incorporated in the current Feasibility Study Update which would require further engineering, technical information or modifications to current permitting applications.

The significant project opportunities identified are as follows:

- Combination of White Pine Project with Copperwood whereby the latter can provide ore earlier and at higher grade;
- Reduction in pillar with former White Pine mine;
- Mining with a continuous miner or mobile miner;
- Shaft to accelerate access to the White Pine North ore body;
- Metallurgical recovery improvements from flotation process or heap leaching of copper ore and SX-EW option;
- Underground tailings disposal.

26. RECOMMENDATIONS

G Mining Services Inc. ("GMSI") recommends that further work be undertaken to compliment the current Preliminary Economic Assessment ("PEA") and support a future Pre-Feasibility Study ("PFS"), focusing on further upgrades of Inferred resources into the Indicated category.

The PEA is preliminary in nature and includes Inferred Mineral Resources that are too speculative geologically to have economic considerations applied to them that would enable them to be categorized as Mineral Reserves.

The following work is recommended to reduce geological risks, initiate pre-feasibility engineering, metallurgical testworks and environmental baselines, and evaluate further opportunities for the White Pine North Project:

- Infill resource drilling at White Pine North Deposit (eastern sector) to upgrade current Inferred Mineral Resources to Indicated category;
- Confirm mining methods, ventilation and initiate underground geotechnical rock mechanics analysis studies;
- Establish and execute metallurgical testwork program and confirm process flowsheets including preliminary equipment sizing and trade-off studies and other processing alternatives such as heap leaching;
- Pre-feasibility engineering designs including infrastructure, preliminary layouts;
- Starting project definition process for permitting.

The total costs of the recommended work program related to the Mineral Resource are estimated at US$5.34 M (Table 26.1).

Table 26.1: Recommended Work Programs Relating to the Mineral Resource

Description	Included Costs (US$)	Total Costs (k US$)
Geology & Mineral Resources		
Infill Resource drilling at White Pine North Deposit (East portion of the North-East area) to upgrade current Inferred Mineral Resources to Indicated category. 9,000 m of total drilling (8-12 drill holes with varying depths)	Drilling Costs (drilling, logging, assays, etc.) - US$ 200/m.	1,800
Update Mineral Resource Estimate		40
Mining & Mineral Reserves and Geotechnical		
Geotechnical, rock mechanics study	Drilling and analysis	250
Mining Engineering	Including trade-offs	200
Metallurgy & Mineral Processing		
Metallurgical testworks to confirm actual information		600
Mineral Process Pre-feasibility level		800
ESR & Permitting		
Initiating Acid Rock Drainage and geochemistry evaluations, Environmental baseline Study and Wetland delineations		500
Start permitting process		150
Project PFS		
Remaining work to complete PFS	Mainly infrastructure, financial model, etc.	1,000
Total Costs		**5,340**

27. REFERENCES

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