National Instrument NI 43-101 Technical Report

for the

Davidson Project Preliminary Economic Assessment

Smithers Area, British Columbia, Canada

Longitude 127° 17′ 52.1″ W, Latitude 54° 48′ 51.6″ N

April 2, 2024

Prepared For

Moon River Capital Ltd.

Prepared By

A-Z Mining Professionals Limited

B. LeBlanc, P.Eng.

F. Bakker, P.Geo.

DATE AND SIGNATURE PAGE

The technical report titled "National Instrument 43-101 Technical Report for the Davidson Project Preliminary Economic Assessment" (the "Technical Report") with an effective date of February 22, 2024 for Moon River Capital Ltd. The report is prepared in accordance with National Instrument NI 43-101 – Standards of Disclosure for Mineral Projects and Form 43-101F1 – Technical Report. The issue date of this report is April 2, 2024.

The Qualified Persons responsible for the report are:

Name	Signed at	Date
Original Signed and Sealed by Brian C. LeBlanc		
Brian C. LeBlanc, P.Eng. Registration No. 904279972-10	Thunder Bay, Ontario, Canada	April 2, 2024
Original Signed and Sealed by Finley Bakker		
Finley Bakker, P.Geo. Registration No. 18,639 Permit Number 1003901	Campell River, British Columbia, Canda	April 2, 2024
Original Signed and Sealed by John Eggert		
John Eggert, P.Eng. License No. 90397597	Sudbury, Ontario, Canada	April 2, 2024

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1.0 SUMMARY

This technical report provides summary documentation of the Preliminary Economic Assessment (PEA) by A-Z Mining Professionals Ltd. (AMPL) for Moon River Capital Ltd.'s ("Moon River" or "Company") Davidson Property (Project), situated in west central British Columbia approximately 9 kilometers (km) northwest of the town of Smithers.

This PEA assesses the potential economic viability of the Project. The cost estimates fall within the guidance on accuracy for PEAs (±40%). The report is prepared in compliance with the Canadian disclosure requirements of National Instrument 43-101 - Standards of Disclosure for Mineral Projects (NI 43-101) and in accordance with the requirements of Form 43-101 F1. The disclosure is based on reliable information, the professional opinions of independent Qualified Persons, and uses industry best practices and standardised terms.

The Project contemplates development of an underground mine with potentially economic mineralisation processed in an on-site processing facility located underground, resulting in a Life of Mine (LOM) of 20 years.

The PEA indicates the Project has the potential to generate positive economic returns. All references to currency herein are in Canadian Dollars, unless otherwise specified.

1.1 PROPERTY DESCRIPTION

The Davidson Project, formerly known as the Yorke-Hardy, Glacier Gulch, or Hudson Bay Mountain deposit, is located on the east flank of Hudson Bay Mountain, which is at 4,592 metre (m) elevation and is the most dominant topographical feature of the Hudson Bay Mountain Range. Road access is from the town of Smithers, British Columbia, approximately 8.9 km northwest of a portal, which provides access to the Project and is located at 1,067 m elevation (Figure 1.1 – Figure 2 from Atkinson, 1995, below).

The Davidson Property consists of 6 patented claim blocks that encompass the entire resource. In addition, 7 additional claim blocks were recently staked on the western slope of the mountain.

Moon River has the exclusive right to access, develop, and mine the Davidson Property, subject to the provisions of the Davidson Agreement with Roda Holdings Inc. (Roda). Please refer to Section 4.1 for further information.

Figure 1.1. Location of Smithers
Source: Giroux, 2016

1.2 RESOURCES AND RESERVES

Resources used for the PEA were based on the latest Resource estimates calculated and reported in a study completed by A-Z Mining Professionals Ltd. and presented in an NI 43-101 Technical Report entitled "National Instrument NI 43-101 Technical Report for the Davidson Project Resources Update" dated September 13, 2023 and filed on SEDAR+.

Table 1.1 Measured Mineral Resources					
Category	**Cut-off Grade MoS_2**	**Tonnes**	**Grade MoS_2**	**Grade Mo**	**Contained Mo kg**
Measured	>0.10	93,480,000	0.22	0.13	123,300,000
Measured	>0.15	63,523,000	0.26	0.16	99,000,000
Measured	>0.20	39,884,000	0.31	0.19	74,100,000
Measured	>0.25	24,269,000	0.37	0.22	53,800,000
Measured	>0.30	14,828,000	0.43	0.26	37,900,000
Measured	>0.35	9,404,000	0.49	0.29	27,600,000
Measured	>0.40	6,127,000	0.55	0.33	20,200,000
Measured	>0.45	4,006,000	0.61	0.37	14,600,000

Table 1.2 Indicated Mineral Resources					
Category	**Cut-off Grade MoS_2**	**Tonnes**	**Grade MoS_2**	**Grade Mo**	**Contained Mo kg**
Indicated	>0.10	197,999,000	0.17	0.1	201,800,000
Indicated	>0.15	97,533,000	0.21	0.13	122,800,000
Indicated	>0.20	43,625,000	0.27	0.16	70,600,000
Indicated	>0.25	19,627,000	0.32	0.19	37,600,000
Indicated	>0.30	9,291,000	0.39	0.23	21,500,000
Indicated	>0.35	5,277,000	0.43	0.26	13,600,000
Indicated	>0.40	2,912,000	0.48	0.29	8,400,000
Indicated	>0.45	1,619,000	0.54	0.32	5,200,000

Table 1.3 presents the combined Measured and Indicated Mineral Resources at various cut-off grades.

Table 1.3 Measured and Indicated Combined Resources					
Category	**Cut-off Grade MoS_2**	**Tonnes**	**Grade MoS_2**	**Grade Mo**	**Contained Mo kg**
Measured and Indicated	>0.0	394,623,000	0.15	0.09	354,800,000
Measured and Indicated	>0.10	291,479,000	0.18	0.11	314,500,000
Measured and Indicated	>0.15	161,056,000	0.23	0.14	222,000,000
Measured and Indicated	>0.20	83,509,000	0.29	0.17	145,200,000
Measured and Indicated	>0.25	43,896,000	0.35	0.21	92,100,000
Measured and Indicated	>0.30	24,119,000	0.41	0.25	59,400,000
Measured and Indicated	>0.35	14,681,000	0.47	0.28	41,400,000
Measured and Indicated	>0.40	9,039,000	0.53	0.32	28,700,000
Measured and Indicated	>0.45	5,625,000	0.59	0.35	19,900,000

Table 1.4 presents the Inferred Resources.

TABLE 1.4 INFERRED RESOURCES					
Category	**Cut-off Grade MoS_2**	**Tonnes**	**Grade MoS_2**	**Grade Mo**	**Contained Mo kg**
Inferred	>0.0	502,849,000	0.10	0.06	301,400,000
Inferred	>0.10	225,817,000	0.15	0.09	203,000,000
Inferred	>0.15	78,990,000	0.20	0.12	94,700,000
Inferred	>0.20	25,039,000	0.26	0.15	39,000,000
Inferred	>0.25	11,907,000	0.30	0.18	21,400,000
Inferred	>0.30	3,789,000	0.37	0.22	8,400,000
Inferred	>0.35	1,786,000	0.42	0.25	4,500,000
Inferred	>0.40	677,000	0.50	0.30	2,000,000
Inferred	>0.45	404,000	0.55	0.33	1,300,000

It should be noted that Inferred Mineral Resources are considered too speculative geologically to have economic considerations applied to them that would enable them to be categorised as Mineral Reserves. Metallurgical and cost projections are to a PEA level of accuracy. Therefore, there is no guarantee that the economic projections contained in this PEA would be realised.

There are no Reserves for the Project at present.

1.3 PROJECT DESIGN

The Davidson Deposit is located inside Hudson Bay Mountain and does not outcrop on surface. The deposit has an existing portal on the east side of the mountain and over 2,100 m of exploration drifting. The access road and portal can be seen from the town of Smithers. In the mid 2000s, the Project met with local resistance regarding development and mining of the deposit from the eastern side of the mountain. Smithers is a major tourist area, and the mine site would have been highly visible from town. This Project puts the primary mine development on the west side of the mountain with the existing eastern portal used only for initial development. Should mining commence, the east portal would then be shut down and any waste rock generated would be returned underground as backfill.

The Davidson Project is a polymetallic deposit containing molybdenum, tungsten, copper, gallium, and rare earth elements. The focus of this report is the mining and recovery of the molybdenum in the Project. While other metals are present, no work has been done at this time to quantify them or to determine if they are in sufficient quantities to produce a saleable concentrate.

A future mining operation would be an underground mine accessed by twin tunnels developed from the western slope of Hudson Bay Mountain (15,000 m). The existing portal on the east slope of the mountain will provide access to develop the internal ramp system, levels and access to the potentially economic mineralisation and underground infrastructure. When the mine is in production all material mined through the east portal would be returned to the mine as either backfill or low-grade mill feed. The east portal would then serve primarily as an emergency escape-way and as the mine exhaust. All mine access and surface infrastructure would be from the western slope of the mountain.

The mineralised zone is a large irregular shaped mass with an enriched core (bright red area). The plan is to mine the higher-grade core as outlined in Figure 1.2 and Figure 1.3, below.

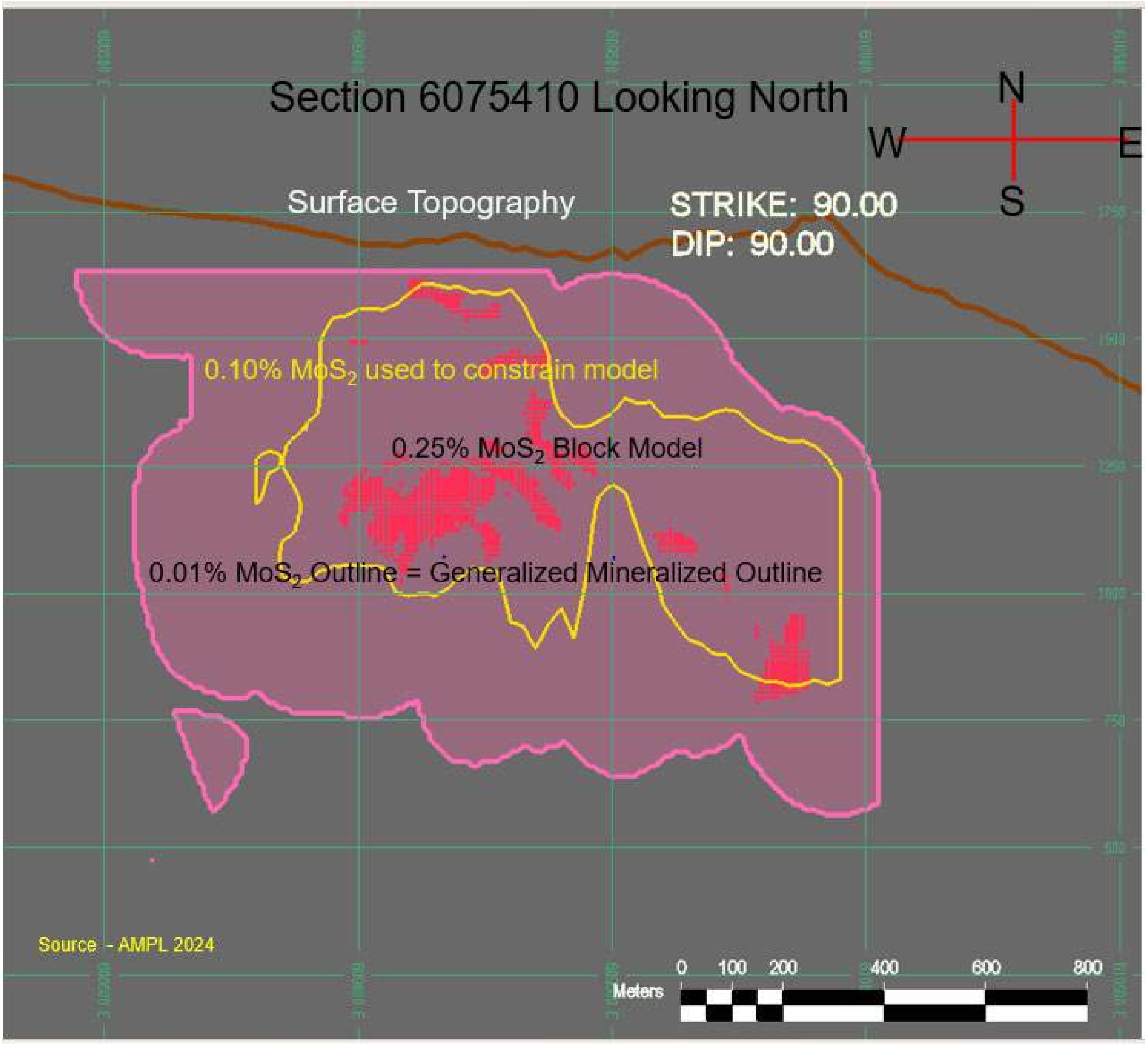


Figure 1.2. Section 6075410 Looking North
Source: AMPL, 2024

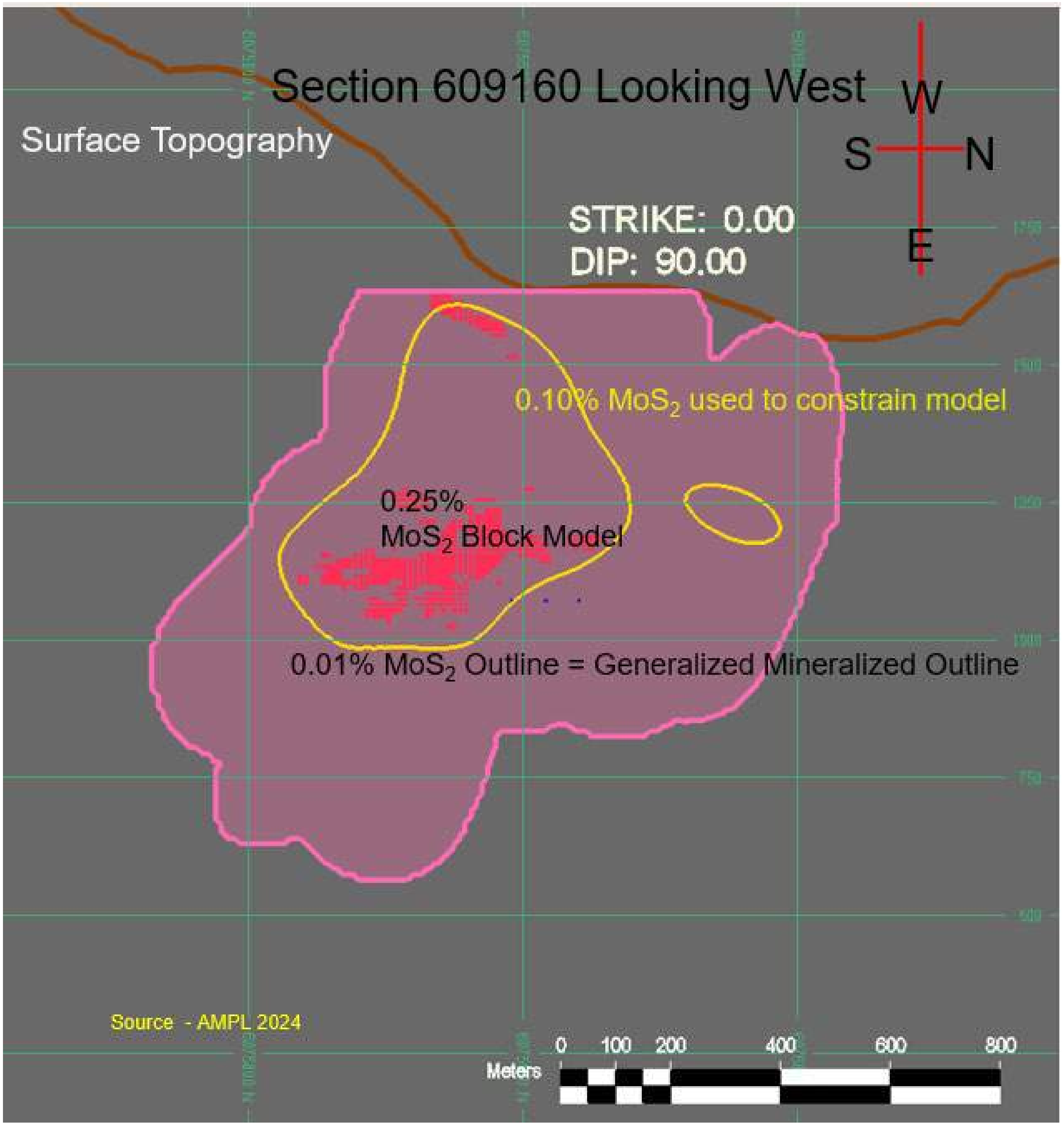


Figure 1.3. Section 609160 Looking West
Source: AMPL, 2024

The potential mine has been designed to produce 7,000 tonnes per day of potentially economic mineralisation. The mineralised zone is highly amenable to bulk mining of large tonnage stopes with inherent economies of scale and low mining costs. Stopes have been designed using sound geotechnical principals and will be approximately 160,000 tonnes each (30 m wide, 45 m deep, and 45 m high), 15 to 16 stopes will need to be mined each year to meet production targets.

The plan is to utilise rubber tired, battery powered mining equipment wherever possible and to automate equipment where possible to reduce manpower requirements. Electrically driven equipment also significantly reduces underground ventilation and mine air heating costs and carbon dioxide (CO_2) emissions. Automation optimises mining manpower and significantly increases productivity of the workforce.

To minimise the surface footprint of the whole operation, the processing plant will be located in specially designed and excavated openings at the top elevation of the mining zones. This eliminates having to move mineralised material from the underground to a surface processing plant 8 km away. This also eliminates the need for a source of backfill material from the surface as well as the transportation of this material to the underground. The mill tailings will provide a ready source of material for a paste backfilling plant as well as significantly reducing the size of the tailings management facility on the surface.

Support infrastructure will be located on the surface and underground at the mine and processing sites and wherever practical, the remainder will be located in the town of Smithers itself.

The entire Project is designed to employ 207 persons in all facets of the operation including mining, processing, maintenance, services, and staff. During pre-production and construction, an outside contracted workforce will be required. Once production is achieved, the plan is to minimise fly-in/fly-out personnel. There is a history of mining in the region and many skilled workers in the area currently work out of town. The Project has a 20-year mine life. Smithers is an attractive area with lots of recreation and good services. The expectation is that the majority of the workforce would come from Smithers and other regional communities.

1.4 MINE PLAN

Underground mining methods would be utilised to extract the potentially economic mineralisation of the deposit.

The mine would be accessed via 2 parallel access drifts, each of approximately 7 km in length, collared and driven from the western slope of Hudson Bay Mountain. During pre-production, the existing portal on the eastern slope of the mountain would be used to develop an internal underground ramp to the top of the mining zone, where an underground processing plant facility would be located and to the bottom of the mining zone where the crushers, coarse ore bins, and an internal winze would be located. Level and stope development would also be done from the eastern portal. After the mine enters production, the existing east portal would be decommissioned and used only as an emergency escapeway and as the main ventilation exhaust. Any waste rock generated during development would be returned to the mine as either backfill or development grade mill feed (Figure 1.4).

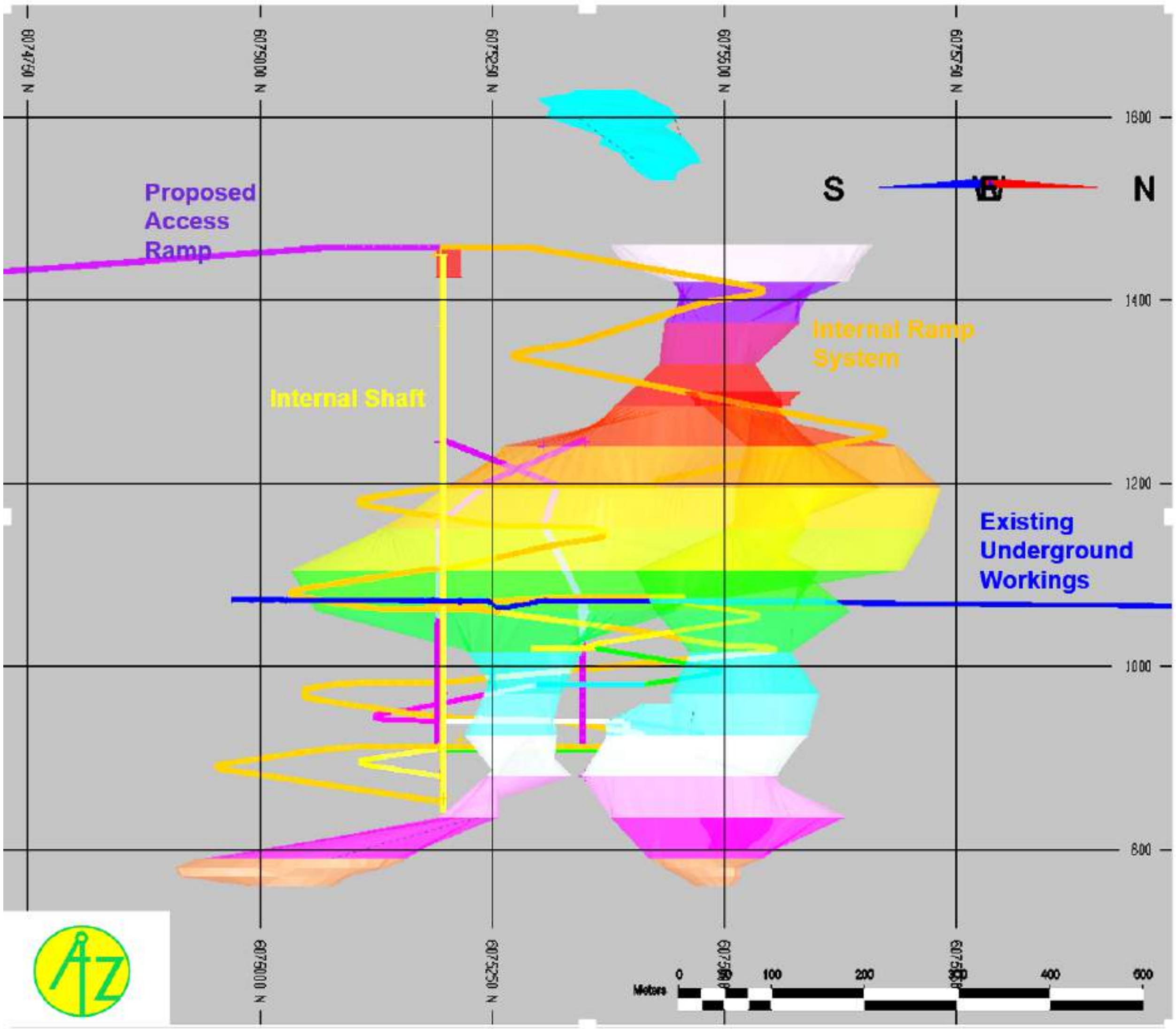


Figure 1.4. Conceptualised Mine Layout
Source: AMPL, 2024

The mineralised zone is a large, irregular shaped mass with an enriched core. The plan is to mine the higher-grade core.

The mining method to be employed will be longhole open stoping with cemented paste (densified tailings) backfill to maximise potentially economic mineralisation recovery. Dilution of 5% has been included in the mined potentially economic mineralisation at a grade of 0.18% MoS_2. Due to the breadth of the potentially economic mineralisation, the stopes would need to be panelled and sequenced.

On each level, the mining areas would be accessed from the main ramp with a footwall drive parallel to the designated footwall of the stope. Levels would be developed parallel to the designated foot wall of the ore zone. The lateral extent of the mining zone is 400 m to 650 m in the central area of the deposit. Two ore-pass systems, 250 m apart, would be developed with dumps on each level and jaw crushers at the bottom of each pass. Levels would be spaced at 45 m vertical intervals from 1,060 m elevation up to 1,420 m elevation where the processing plant would be located. The bulk of the potentially economic mineralised zone lies between 1,060 m and 1,240 m with smaller extensions below 1,060 m and above 1,240 m.

Crosscuts will be driven at 30 m centres down the centreline of each stope to the nominal hanging wall. In some cases, the stopes would be almost 400 m from the foot wall to the hanging wall. All stopes will be filled with paste fill containing 5% cement by volume. Stopes will be large, 30 m along the hanging wall by 45 m deep and 50 m high. Each stope will contain approximately 160,000 tonnes; 15 to 16 stopes will need to be cycled each year to achieve the production targets.

Potentially economic mineralisation will be transported by battery powered, load-haul dump (LHD) units to nearby vertical ore passes, which will deliver the material to jaw crushers at the 940 Level of the mine. The jaw crushers will dump directly into coarse ore bins. Material will be pulled out of the coarse ore bins on the 910 Level, screened and crushed with a secondary cone crusher to -15 millimetres (mm). The fine screened material and the cone crusher discharge will dump onto a conveyor, which will then dump into a bin, which will feed the vertical lift conveyor system in the winze. The vertical lift conveyor will move the crushed material to the 1455 Level and dump onto a conveyor, which dumps into the fine ore bin feeding the underground processing facility (Figure 1.5, below).

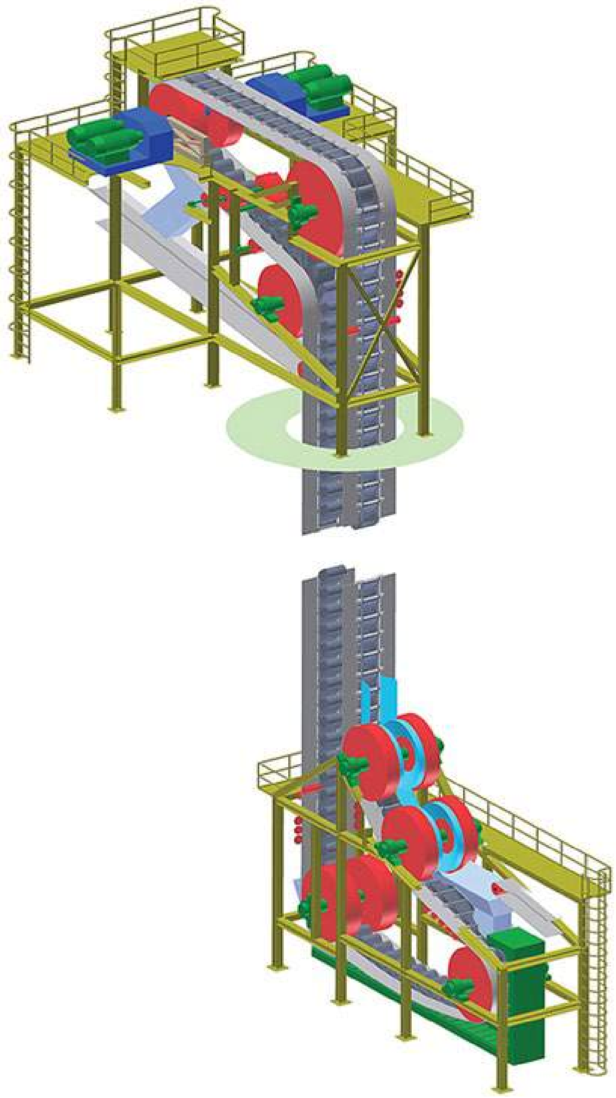


Figure 1.5. Vertical Lift Conveyor System
Source: Pinterest, 2024

Other underground facilities will include:

- Ore processing plant;
- Paste backfill plant;
- Equipment maintenance shops and underground warehouses;
- Explosives storage magazines;
- Refuge stations;
- Fuel bays;
- Materials storage areas; and
- Main de-watering sumps.

1.5 PROCESSING

The processing plant will be located completely underground at the top elevation of the mineralised zones to be mined. Large processing equipment will be located in individual open rooms and interconnected with piping. Other smaller equipment would be installed in groupings in other open rooms. The construction cost of an underground plant is not significantly different from that for a plant located on surface. All underground excavations have been costed under development costs (Figure 1.6 and Figure 1.7, below).

A two-stage crushing system will be located at the bottom of the mine. The crushed material will report to a vertical lift conveyor, which will dump into the fine ore bin feeding the parallel grinding mills. The processing plant will be a conventional flotation plant producing a molybdenum oxide (MoO_2) concentrate for shipment to smelters. The tailings will be primarily made into paste backfill for backfilling of stopes with the balance pumped to a permanent dry stack tailings facility on surface where the water is removed and recycled back to the mine.

The processing plant is expected to have a recovery rate for MoS_2 of 92%.

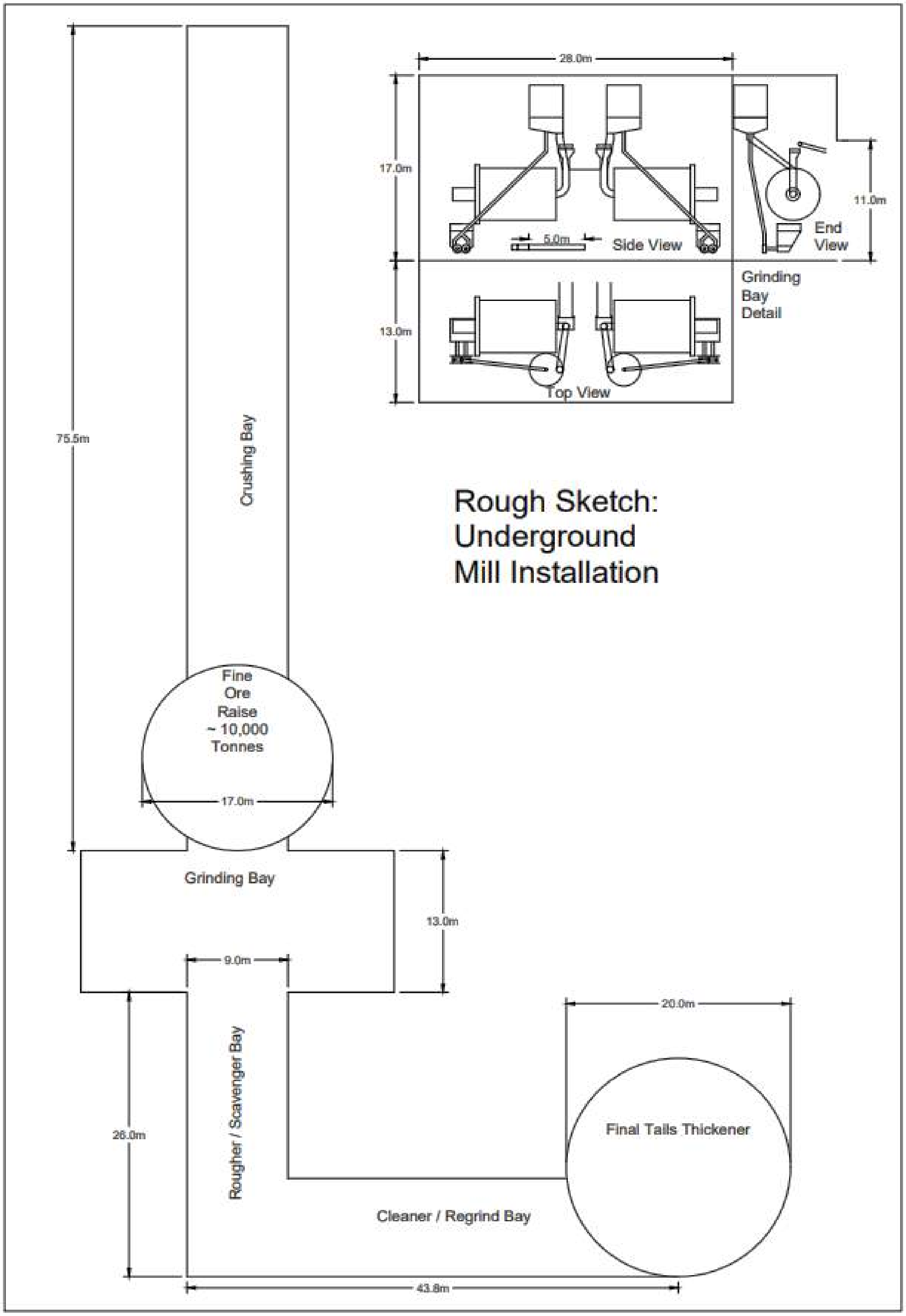


Figure 1.6. Conceptual Underground Process Plant Design
Source: Eggert Engineering, 2024

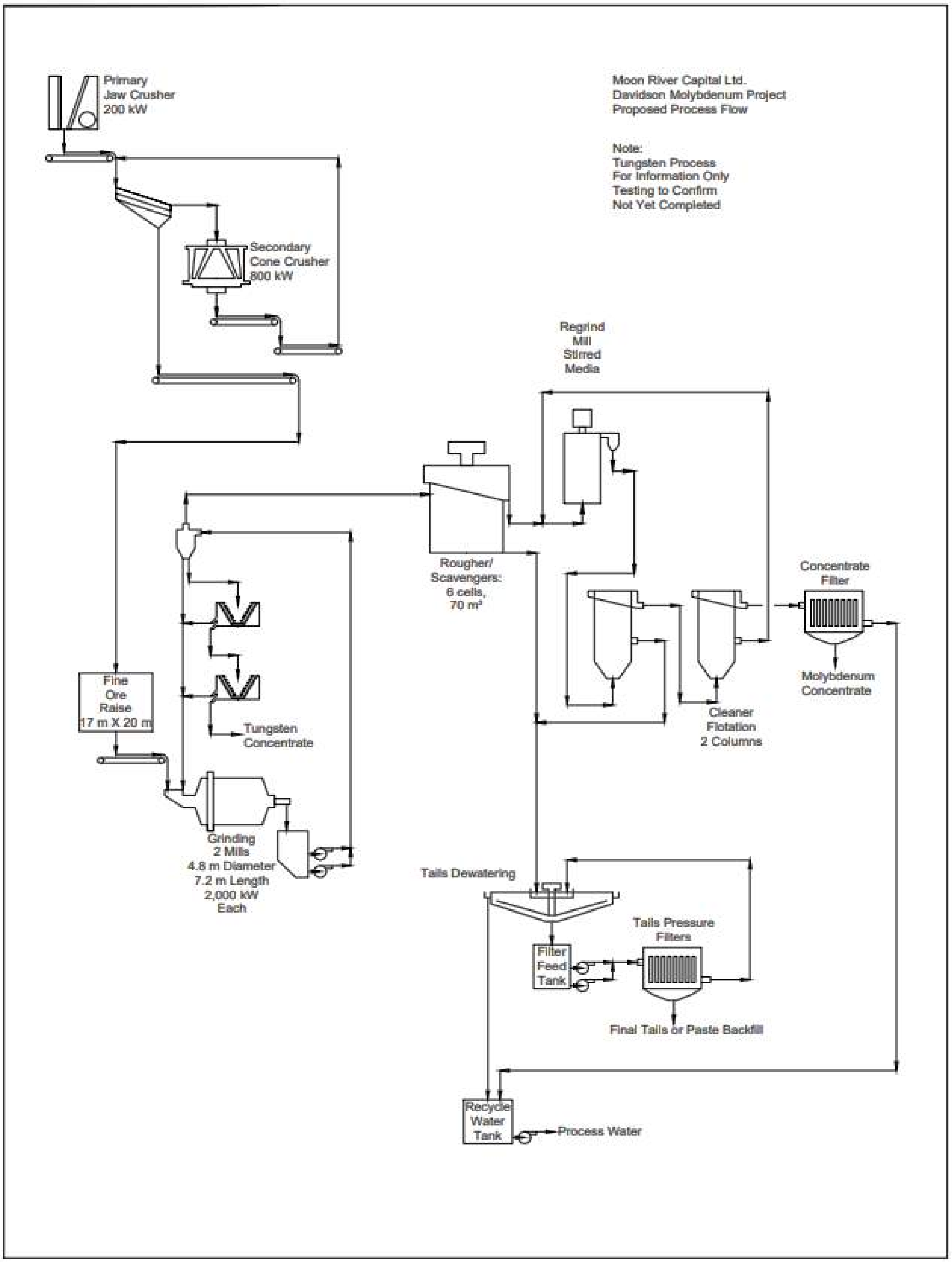


Figure 1.7. Processing Plant Flowsheet
Source: Eggert Engineering, 2024

1.6 INFRASTRUCTURE

The Project is located close to the community of Smithers, which could support and provide services to the mine workforce.

The west side of the mountain is accessed via an active logging road, which connects directly with the main highway through Smithers. The location of the twin portals will be approximately a kilometer north of the logging road. This road is currently active and is highly used by locals for accessing recreation areas and lakes. In order to support a mining project, the road will need to be upgraded and a new road constructed from the existing road to the portal location (Figure 1.8).

Figure 1.8. Site Location

Source: Google Earth™

As the Project is located in a highly used and scenic recreation area, the design of the Project seeks to minimise any disturbance and visual impact. In order to do this, as much infrastructure as possible will be located underground. Offices, warehousing facilities, and the processing plant will all be located underground. Surface infrastructure required would include:

- Upgrading of Access Road;
- Powerline Construction;
- Electrical Sub-stations and Distribution;
- Site Roads and Materials Handling Area;
- Maintenance Shop/Offices/Dry/Warehouse Complex (temporary);
- Two Cement Storage Silos;
- Water Supply System and Water Treatment Plant;
- Dry Stack Tailings Impoundment Area;
- Development Waste Storage;
- Landfill Site; and
- Sewage Disposal Site.

Approximately 20 km of road requires construction or upgrade to allow heavy truck traffic to access the site. Construction will include clearing to the required width of the right-of-way; placing road base, installing culverts, and capping the entire road surface with granular material of suitable type. The switchback road up the east side of the mountain will have to be rehabilitated as well in order to stage development through the east portal.

There is a 500 kilovolt (kV) line south of Smithers, which services the Terrace, British Columbia area. No contact has been made with BC Power; however, it is expected that this line could supply the power necessary for the Project. A dropdown connection to a 44 kV transformer would be made at the 500 kV line and a 17 km power line constructed to the mine site (Figure 1.9, below).

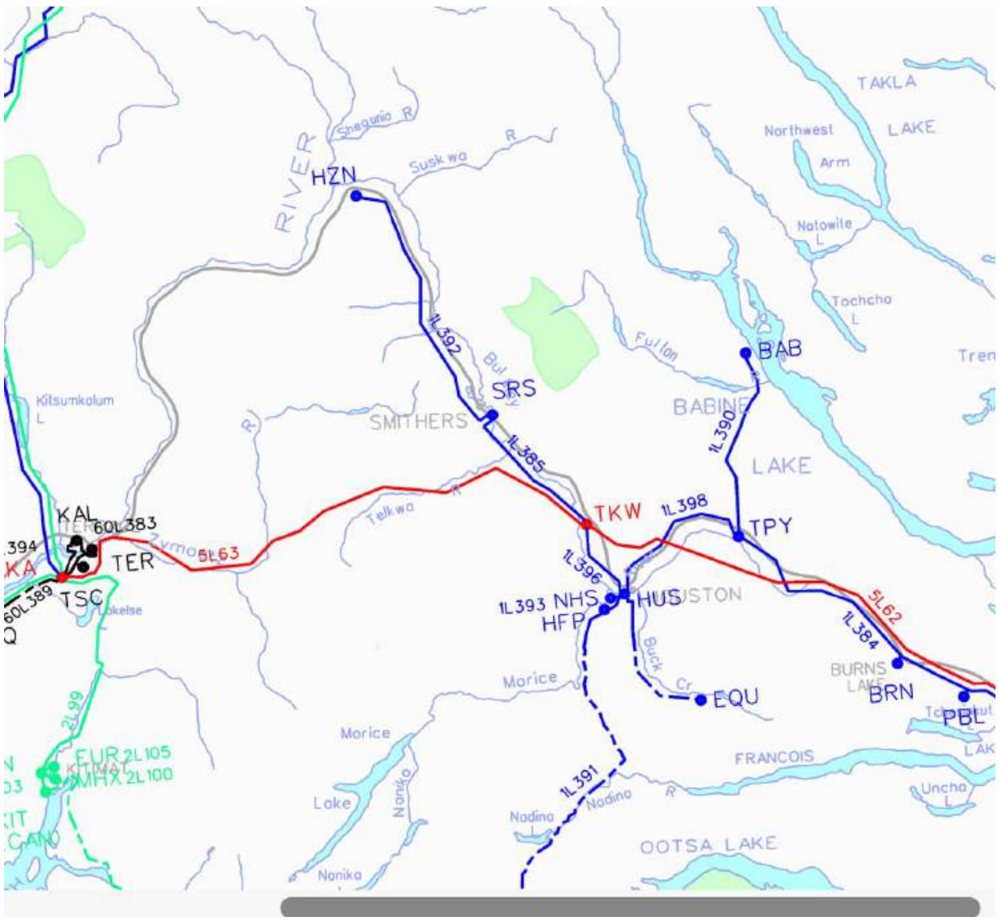


Figure 1.9. BC Hydro Transmission Lines
Source: BC Hydro

Some facilities, including an administration building and purchasing, will be located in Smithers. To minimise onsite laydown areas and surface storage, larger satellite equivalents would be located in Smithers.

1.7 PROJECT SCHEDULE

The expected pre-production construction period would be approximately 3.5 years with mine development being on the critical path. The extreme length of the western access tunnels will take almost 3.3 years to complete. Access for internal development through the eastern portal is critical. All internal ramp systems, level accesses, levels, and ore development access will be driven by two crews working from the east portal. All development of the winze, ore pass systems, internal vent raises, and volume excavations will also be done by the crews from the east portal access. The main construction access will be the east portal. The western access tunnels are scheduled to connect to the internal mine early in the first year of production.

1.8 CAPITAL EXPENDITURES

The estimated project total pre-production capital expenditure, inclusive of contingencies and working capital, is approximately $575 million. A summary of Project pre-production capital expenditures is presented in Table 1.5, below.

TABLE 1.5
PROJECT PRE-PRODUCTION CAPITAL EXPENDITURES
($000)

Component	Year -3	Year -2	Year -1	Year 1
Exploration	$1,000	$1,000	$1,000	
Mine	$34,377	$52,739	$50,440	$24,124
Equipment Leasing	$9,441	$8,952	$8,462	
Processing Plant		$70,000	$50,125	$35,000
Underground Infrastructure		$4,886	$1,815	$23,375
Surface Infrastructure and Mobile Equipment	$23,636	$1,463	$13,361	
Tailings Management Facilities			$9,150	
Owner's Costs	$3,700	$3,700	$3,700	
Contingency	$18,039	$35,685	$34,513	$20,625
Working Capital				$20,679
Mine Closure			$10,000	
Total Capital Expenditures	**$90,193**	**$178,425**	**$182,567**	**$123,803**
Total	**$574,987**			

The capital estimates include the following conditions and exclusions:

- Qualified and experienced construction labour would be available at the time of execution of the Project;
- A water supply capable of supplying the required demand of the processing plant is assumed to be available;
- No extremes in weather have been anticipated during the construction phase; and
- No allowances have been included for construction-labour stand-down costs.

1.8.1 Sustaining Capital

Sustaining capital expenditures are estimated to be $78.6 million, primarily related to mine equipment leasing and replacement and ongoing construction of the tailings management facility and related contingencies.

1.8.2 Working Capital

In addition to the capital costs outlined above, working capital has been estimated at $20.7 million based on 3 months of the estimated operating costs for the year.

1.9 OPERATING COSTS

The estimated total average operating cost (excluding smelting and refining) for the mine is approximately $38.24 per tonne of potentially economic mineralisation. This equates to $21.68 per kilogram (kg) of molybdenum (Mo) ($9.84 per pound). Table 1.6, below, presents a summary table of life of mine average operating costs.

TABLE 1.6
PROJECT OPERATING COSTS SUMMARY

Component	Cost
Diamond Drilling – Infill	$0.50
Underground Mining	$21.07
Processing	$10.94
Tailings Management Facility	$1.34
Mine Indirects	$1.29
Surface Department	$0.90
General & Administration	$2.20
Total Minesite Operating Cost per Tonne	**$38.24**

1.10 ECONOMIC ANALYSIS

The expected cash flow estimates are calculated using the forecast mine plan, operating costs, and capital expenditures incorporating expected long-term metal prices based on the two-year trailing average up to the end of January 2024. The two-year trailing average is based on the London Metal Exchange (LME) prices and is US$47.39 per kg. (US$21.50/pound). The exchange rate used for the calculations was CA$1.35 to US$1.00.

A summary of the expected parameters used for the financial analysis is presented in Table 1.7, below.

TABLE 1.7 CASH FLOW MODEL INPUT PARAMETERS	
Parameter	
Long-term Metal Price (US$)	$47.39 ($21.50/lb)
Exchange Rate	$1.35 $Can per $1 US
Diluted Mineral Resource	49,125,000 tonnes
Dilution (at adjacent mineral grade)	5%
Average Head Grade to Mill	0.34%
Mill Recovery	92
Payability	97%
Pre-Production Capital	$575.0 million
Total Sustaining Capital	$78.6 million
Working Capital	$20.7 million
Reclamation and Closure	$10.0 million
Estimated Operating Costs ($/tonne)	$38.24
Life of Project	20 Years

1.11 FINANCIAL RETURNS

The overall level of accuracy of this study is approximately ±40%.

The Project expected investment and returns based on the base case cash flow parameters for the Project are shown in Table 1.8, below.

TABLE 1.8 EXPECTED PROJECT RETURNS		
	Pre-Tax	**After Tax**
Undiscounted Net Revenue	$5.778 billion	
Undiscounted Total Cash Flow	$2.982 billion	$1.945 million
NPV at 5%	$1.524 billion	$930.6 million
NPV at 8%	$1.043 billion	$601.8 million
IRR	32%	24%
Payback Period	3.3 Years	

Results indicate that at the expected parameters and metals prices, the Project is viable.

1.12 SENSITIVITY ANALYSIS

Sensitivity analyses were performed for capital expenditures, operating costs, mined grades, metal prices, and currency exchange rates using 25% positive and negative variations. The Project is most sensitive to the mined grade, metal price and the exchange rate and less sensitive to capital and operating costs. The lines for grade, metal price and exchange rate are virtually the same for the three factors and overlay each other. The results of the sensitivity analysis at ±25% are presented in Table 1.9 and Table 1.10, below.

TABLE 1.9 NPV 8% DISCOUNT SENSITIVITY ANALYSIS			
	After Tax NPV 8%		
	-25%	Base Case	25%
Capital Cost	705.7	616.8	497.9
Operating Cost	732.1	616.8	471.5
Mined Grade	218.9	616.8	983.9
Metal Price	216.9	616.8	985.9
Exchange Rate	218.9	616.8	983.9

TABLE 1.10 POST TAX IRR SENSITIVITY ANALYSIS			
	After Tax IRR		
	-25%	Base Case	25%
Capital Cost	31%	24%	19%
Operating Cost	26%	24%	21%
Mined Grade	15%	24%	31%
Metal Price	14%	24%	31%
Exchange Rate	15%	24%	31%

The Net Present Value (NPV) and Internal Rate of Return (IRR) sensitivities to variations in key parameters are depicted graphically in Figure 1.10 and Figure 1.11, below. The IRR is most sensitive to variations in metal prices, mined grades and metal prices and less sensitive to capital and operating costs. **Note:** Only 3 lines are visible on the graph as the lines for Mined Grade, Metal Price, and Exchange Rate are virtually identical.

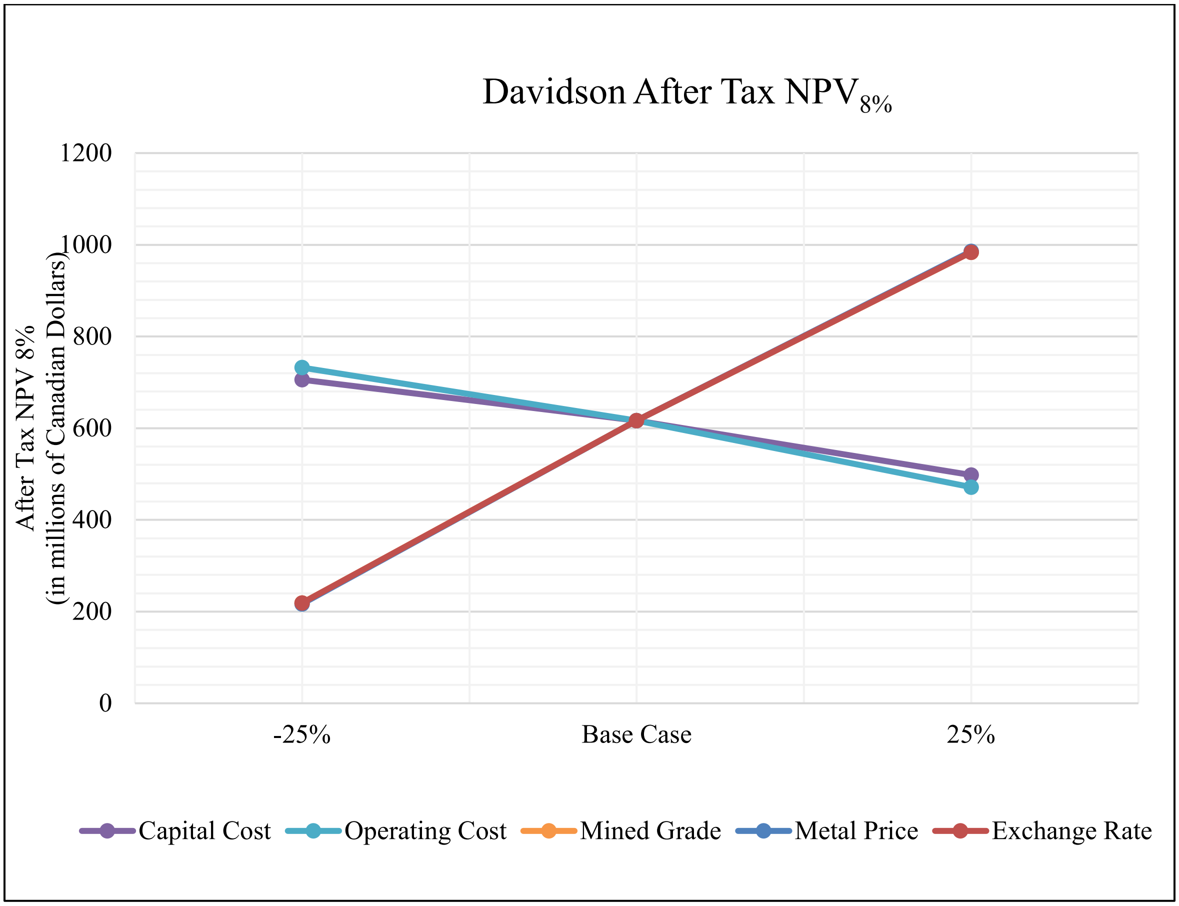


Figure 1.10. NPV at 8% Discount Sensitivity Analysis
Source: AMPL, 2024

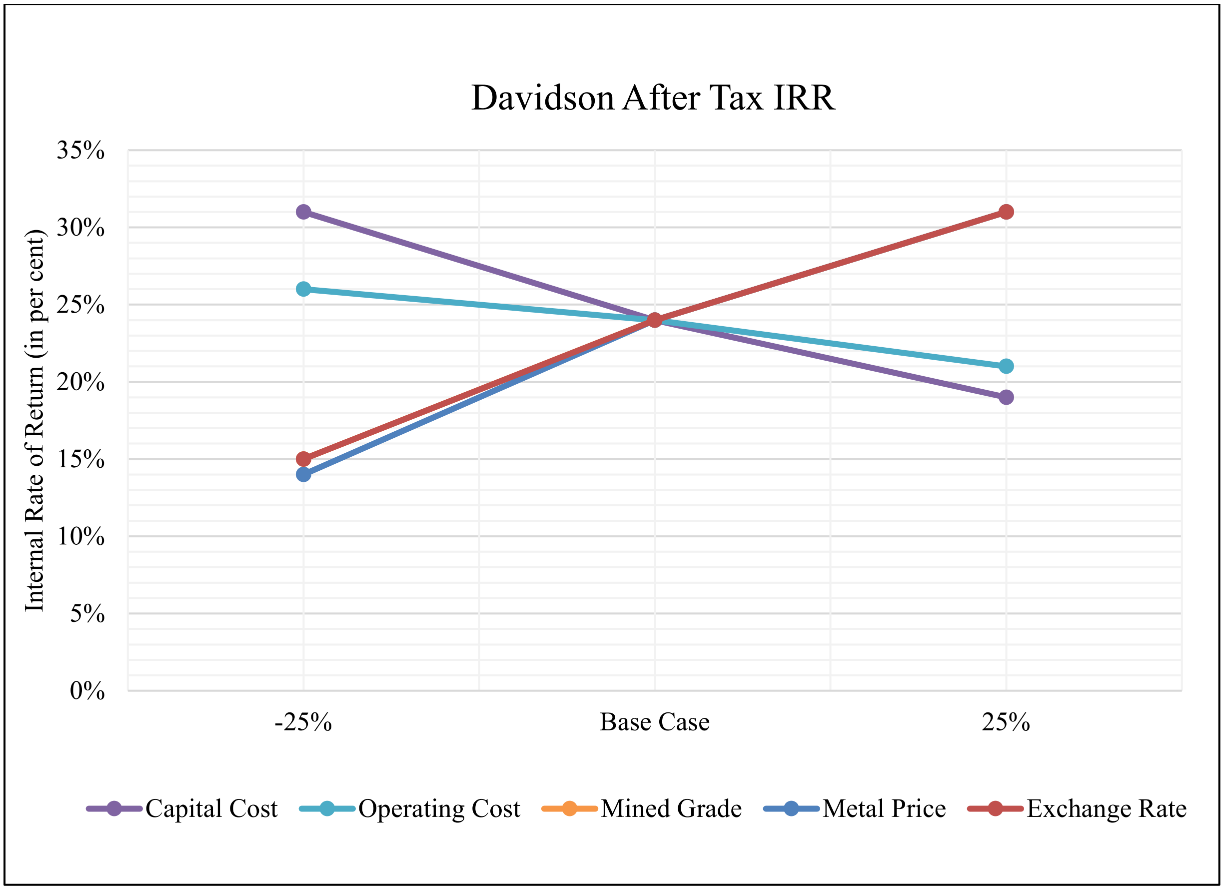


Figure 1.11. IRR Sensitivity Analysis
Source: AMPL, 2024

1.13 CONCLUSIONS

This Preliminary Economic Assessment (PEA) examines the viability of mining the September 13, 2023 NI 43-101 Resource estimate using underground mining methods. The results from this PEA indicate the Davidson Project have the potential to generate positive economic returns.

The Resource of the Davidson Project is comprised of Measured Mineral Resources, Indicated Mineral Resources, and Inferred Mineral Resources. It should be noted that the Inferred Mineral Resources are considered too speculative geologically to have economic considerations applied to them that would enable them to be categorised as Mineral Reserves. Therefore, there is no guarantee that the economic projections contained in this PEA would be realised.

The contemplated plan is to mine the higher-grade core of the mineralised zone. Using a cut-off grade above 0.25% MoS_2, there is a Measured and Indicated Resource of 43.98 Mt at 0.35% MoS_2 and an Inferred Resource of an additional 11.9 Mt at 0.30% MoS_2 available for mining (refer to Table 1.3 and Table 1.4, above). This PEA has identified a diluted potentially mineable Resource of 49.1 Mt at 0.34% MoS_2. (refer to Appendix 1.0).

The engineering design extracts the potential resources at 2.5 Mt per annum and produces $5.84 billion in gross revenue during the 20 year life of the mine.

Based on the study results, the conclusions of AMPL are as follows:

1. The Project provides positive returns based on the parameters and metal prices used in this study and should be progressed further with the aim of bringing the Davidson Property to production.

1.14 RECOMMENDATIONS

Based on the conclusions, AMPL recommends the following:

1. Engage Wet'suwet'en and Gitxsan First Nations in discussions with the aim of establishing a Memorandum of Understanding (MOU) with each.

2. Complete the necessary environmental work for baseline studies, hydrogeology, geochemistry, hydrology, air quality, noise emissions, and effluent receiving water studies as outlined by Ms. M. Tanguay.

3. Conduct further metallurgical testing will be required to advance the Project to a Pre-Feasibility level or higher, including testing for the economic recoverability of tungsten, copper, gallium, and rare earth elements. Sampling requirements are as follows:
   a) The core for the samples should be bagged shortly after logging and splitting with nitrogen injection into the bag, the bags collected in sealable buckets, and nitrogen injection into the bucket prior to sealing. These steps are necessary to assure that sample aging can be eliminated as a potential source of error. It is strongly recommended that a metallurgist or geo-metallurgist be consulted prior to laying out the sample collection drilling program.
   b) The required mass of the samples should be determined in consultation with the metallurgical testing facility.
   c) It is recommended that a facility be chosen that has skilled individuals familiar with process development and assistance in sample and test work selection.
   d) Metallurgical testing should include new sampling of core including:
      i) Per potentially economic mineralisation representative sample;
      ii) Adjoining waste zone samples;
      iii) Variability samples by geography; and
      iv) Variability samples by mine life chronology.
   e) Mineralogical classification of samples, including mineral identification, particularly for tungsten, copper, and rare earth minerals.
   f) A comprehensive comminution testing program including Bond work indices.
   g) Bench testing on a full potentially economic mineralisation composite to further develop the preliminary flowsheet, benchmark reagents, and optimise additions.
   h) Locked cycle testing on a full potentially economic mineralisation composite to estimate cycle times.
   i) Mini-pilot plant testing of the developed flowsheet to finalise flows and cycle times – optional.
   j) Mini-pilot plant testing to assess variability, including impacts of dilution – optional.
   k) De-watering characteristics of concentrates and tailings.

l) Characterisation of tailings, including:
   i) Acid base accounting separately on tailings.
   ii) Solution chemistry of all tailings materials; and
   iii) Suitability of tailings for mine backfill – particularly sulphide tailings.
   iv) Concentrate analysis for salability, including penalty minerals, maximum allowable moisture, etc.
   v) Investigate possible sales and concentrate shipping contracts

4. Complete an oriented core geotechnical drilling program to conduct a detailed rock mechanics analysis for stope geometry and mine design including portal design, stope geometries, and stope sequencing:
   a) Conduct a geotechnical assessment of the bedrock in the area of surface infrastructure and the Tailings Management Facility (TMF).

5. Complete a trade-off study on alternative methods of excavating the twin access drifts with the aim of reducing the development time and capital costs.

6. Further studies are recommended to advance the tailings facility design.
   a) Geotechnical and hydrogeological investigations including:
      i) Laboratory testing to confirm site conditions, identify any potential geologic hazards;
      ii) Characterise foundations and groundwater conditions; and
      iii) Identify suitable borrow sources for construction fill.

   b) Tailings characterisation testing is recommended to better define the:
      i) geochemical,
      ii) physical, and
      iii) settling, as well as filtration properties to validate the TMF design criteria.

   c) Site specific precipitation and evaporation data should be collected and a site-specific water balance model performed to confirm collection pond sizing and discharge water volumes.

   d) A grading plan should be developed that optimises the cut-fill balance for the TMF base grade.

   e) Consider amending the closure cover if it can be demonstrated that the compacted tailings have an equivalent permeability and do not pose a chemical stability risk.

All recommendations should be performed prior to or as part of a follow up Pre-Feasibility Study (PFS) or Feasibility Study (FS). The cost to complete a Pre-Feasibility or Feasibility Study for the Project is estimated to be between $3 million to $5 million.

1.15 CONTACTS WITH THE LOCAL COMMUNITIES

It is recommended that Moon River should increase its interaction and information sharing with the local First Nations and other communities in the region whether they are part of the legal ownership of the surface areas or they are neighbours. The importance of engagement at this stage will prove beneficial moving forward.

2.0 INTRODUCTION

2.1 TERMS OF REFERENCE

This technical report provides summary documentation of the Preliminary Economic Assessment (PEA) performed by A-Z Mining Professionals (AMPL) for Moon River Capital Ltd.'s ("Moon River" or "Company") 100% Davidson Property (Project), situated in west central British Columbia approximately 9 km northwest of the town of Smithers.

This PEA assesses the potential economic viability of the Project. The cost estimates fall within the guidance on accuracy for PEAs (±40%). The report is prepared in compliance with the Canadian disclosure requirements of National Instrument 43-101 - Standards of Disclosure for Mineral Projects (NI 43-101) and in accordance with the requirements of Form 43-101 F1. The disclosure is based on reliable information, the professional opinions of independent Qualified Persons, and uses industry best practices and standardised terms.

The Davidson Project is a polymetallic deposit containing molybdenum, tungsten, copper, and rare earth elements. The focus of this report is the mining and recovery of the molybdenum in the deposit. While other metals are present, no work has been done at this time to quantify them or to determine if they are in sufficient quantities to produce a saleable concentrate.

As of the date of this Report, Moon River Capital Ltd. is a Canadian junior exploration and development company listed on the TSX Venture Exchange (TSXV) with a corporate office at:

Moon River Capital Ltd.
100 King Street West, Suite 7010
Toronto, Ontario M5X 1B1
CANADA

E-mail: pparisotto@moonrivermoly.com

This Report is considered effective as of February 22, 2024 with a filing date of April 2, 2024.

AMPL's Qualified Persons are responsible for the sections of this report identified in their "Certificates of Qualified Persons" submitted with this report to the Canadian Securities Administrators. AMPL has relied on and believes there to be a reasonable basis to rely on the following experts who have contributed the information stated in this report, as noted below:

- Mr. Brian LeBlanc, P.Eng, President and Senior Engineer, AMPL.
- Mr. Finley Bakker, P.Geo, Consulting Resource Geologist and Geology to AMPL.
- Mr. John Eggert, P.Eng, Consulting Metallurgist to AMPL

2.2 SOURCES OF INFORMATION

This Report is based, in part, on internal company technical reports and maps, published government reports, company letters and memoranda, public information, documented results concerning the Project, and discussions held with personnel from the Company regarding all pertinent aspects of the Project as listed in the "References" (Section 27.0) of this report.

The authors have relied on information gathered during a Property visit by Mr. Brian LeBlanc, the preceding National Instrument NI 43-101 Technical Report for the Davidson Property Resource Update of September 13, 2023, detailed reports, digital data, and discussions provided by Mr. Donald Davidson, as well as data and reports provided by Moon River and information in the public realm.

2.3 SITE VISIT

Mr. Brian LeBlanc, P.Eng., a Qualified Person under the terms of NI 43-101, conducted a site visit to the Property from September 27 to 28, 2023. Mr. Donald Davidson, of Roda Holdings Inc. (Roda), acted as a guide (refer to Appendix 4.0).

The objective of the field visit was to inspect the site and existing facilities and determine existing and required mine, processing, and infrastructure requirements for the Project and to gather data from existing reports stored by Mr. Davidson.

2.4 UNITS AND CURRENCY

Unless otherwise stated:

- All units of measurement in the report are in the metric system.
- All currency amounts in this report are stated in Canadian Dollars (CA$).
- Maps are either in UTM coordinates or in the latitude/longitude system.

2.5 GLOSSARY AND ABBREVIATIONS OF TERMS

Abbreviation	**Meaning**
3D	three-dimensional
°C	degrees Celsius
C$ and CA$	currency of Canada
AAS	Atomic Absorption Spectroscopy
Acme	Acme Analytical Labs
AMPL	A-Z Mining Professionals Ltd.
AMAX	American Metal Co.
APT	ammonium paratungstate
ARD	acid rock drainage
bbl	barrels
BCEAA	British Columbia Environmental Assessment Act
BCUC	British Columbia Utilities Commission
BLRMP	Bulkley Land and Resource Management Plan
$CaWO_4$	scheelite
Climax	Climax Molybdenum Corp. of B.C. Ltd.
cm	centimetre
CO_2	carbon dioxide
Darnley Bay	Darnley Bay Resources
DDH	diamond drill hole
DPD	potential discharge points
E	East
EAO	Environmental Assessment Office
EM	Electromagnetic

EMLCI	Energy, Mines and Low Carbon Innovation
EMA	Environmental Management Act
ENV	Ministry of Environment and Climate Change Strategy
EPCM	Engineering, Procurement, and Construction Management
EPIC	Environmental Assessment Offices Project Information Centre
ESSFmc	Engelmann Spruce Subalpine Fir
°F	degrees Fahrenheit
FM	Factory Mutual
FS	Feasibility Study
FTSF	Filtered Tailings Storage Facility
ft^3/tonne	cubic feet per tonne
g	gram
GCL	Giroux Consultants Ltd.
g/t	grams per tonne
ha	hectare
HP	horsepower
Hr	Hydraulic Radius
HLEM	Horizontal Loop Electromagnetics (geophysical survey method)
IAA	Impact Assessment Act
IBA	Impact Benefit Agreement
ICHmc1	Interior Cedar Hemlock
ICP-ES	Inductively Coupled Plasma-Emission Spectrometry
IP	induced polarisation
IRR	Internal Rate of Return
KDC	Kyah Development Corporation
kg	kilogram
km	kilometre
kmph	kilometres per hour
kW	kilowatts
kV	kilo volt
kVA	kilo volt-amperes
LHD	load-haul dump
LLDP	Linear Low Density Polyethylene
LME	London Metal Exchange
LOM	Life-of-Mine
m	metre
m^2	square metre
m^3	cubic metre
MBDLP	Moricetown Band Development Limited Partnership
MCC	master control centre
MCM	thousands of circular mils
mm	millimetre
Mo	molybdenum
MOE	Ministry of Environment
MoO_2	molybdenum oxide
MoO_3	molybdenum trioxide
MoS_2	molybdenum disulphide
Mt	millions of tonnes
MVA	megavolt-amps

N	North
NPV	Net Present Value
NSR	Net Smelter Return
opt	ounces per ton
PAG	potentially acid generating
PDC	process design criteria
P.Geo	Professional Geoscientist
PEA	Preliminary Economic Assessment
PFS	Pre-feasibility Study
ppm	parts per million
QA/QC	Quality Assurance/Quality Control
QP	Qualified Person
SBSmc2	Sub-Boreal Spruce
SG	specific gravity
SM2	Special Management 2
t	tonne (metric)
t/m^3	tonne per cubic metre
TCMC	Thompson Creek Metals Company
TDEM	Time domain electromagnetic
TMF	tailings management facility
THO	Trans-Hudson Orogen
US$	currency of the United States of America
USA	United States of America
UTM	Universal Transverse Mercator
V	volt
VMS	volcanogenic massive Sulphide
VTEM™	versatile time domain electromagnetic
W	Tungsten
WO_3	tungsten trioxide

3.0 RELIANCE ON OTHER EXPERTS

AMPL used the services of the following consultants and firms:

- Dr. W. F. Bawden, Bawden Engineering Ltd., Section 16.0
- Mr. Leon Botham, MSCE, P. Eng., President, Principal Engineer, Newfields, Section 18.0 and Section 21.0
- Ms. Michelle Tanguay, President, 2Tango Environmental Services, Section 20.0

4.0 PROPERTY DESCRIPTION AND LOCATION

4.1 DAVIDSON PROJECT OR YORKE-HARDY DEPOSIT

The Property is situated in west central British Columbia approximately 9 km northwest of the town of Smithers. The majority of the historical exploration activity and current Resource described in this Report is centred at UTM 609360E, 6075510N (NAD83, Zone 9) on topographic map NTS 93L/14 and Mineral Titles Map 093L084.

The Property consists of 6 mineral leases and 1 mineral claims, covering a total area of 2,087.322 hectares (ha). In addition, 7 additional claim blocks were recently staked on the western slope of the mountain. All claims and leases are contiguous and registered to the name of Roda Holdings Inc., a corporation controlled by Mr. Donald Davidson of 6835 Glacier Gulch Road, Smithers, British Columbia.

Moon River is the holder of the exclusive right of access to and from, and to enter upon and take possession of and prospect, develop and mine the Davidson Property, and holds the right to remove and ship therefrom all ore, bullion, concentrates, and minerals recovered in any manner from the Davidson Property all subject to the provisions of the Davidson Agreement (collectively, the “Rights”) with Roda. Roda shall transfer ownership and title to Moon River upon the earlier of: (i) Moon River obtaining bona fide funding commitments in amounts sufficient to construct a mine capable of mining at least 500,000 tonnes of ore per year where registration of title documents is required by the parties providing funding; or (ii), on notice to Roda of commencement of commercial production at levels sufficient to result in the mining of at least 500,000 tonnes of ore within 1 year from commencement of commercial production. In consideration of the Rights, Moon River shall pay Roda $100,000 annually and reimburse Roda for the annual lease and property maintenance payments in connection with the mining leases.

Upon transfer of title from Roda to Moon River, Roda shall reserve to itself and Moon River will grant a 3% net smelter return royalty (NSR). If the NSR payments to Roda in a fiscal year are less than $100,000, Moon River must make a payment to Roda equivalent to the difference between the NSR payments for the fiscal year and $100,000.

As security for the performance of Moon River’s obligations under the Davidson Agreement, Roda also has a first ranking mortgage of and security interest in Moon River’s right, title and interest in the Davidson Agreement, the Davidson Property, and minerals and mineral products extracted or produced therefrom. Roda also has the right to terminate the Davidson Agreement and/or require the transfer back of the Davidson Property in certain circumstances.

Moon River has a right of first refusal in respect of the transfer from Roda to any third party of all or any part of the Davidson Property, the NSR, or any of Roda’s rights under the Davidson Agreement.

Moon River Capital Ltd. entered into an agreement on September 13, 2023 to acquire from Generation Mining Limited (Generation), its interest, rights, and obligations in the Property, and accordingly, the Rights to Moon River to acquire a 100% beneficial interest in the Davidson Property. The agreement between Generation and Moon River and the assignment of the Davidson Agreement to Moon River was registered on title to the Property.

In consideration for the assignment of the Davidson Agreement, Generation received from Moon River: (i) $630,000 in cash; (ii) 9,000,000 common shares in the capital of Moon River; and (iii) to the extent Generation remains a 10% holder of Moon River, (a) the right to nominate one director to the board of directors of Moon River, and (b) the pre-emptive right to retain its pro rata equity interest in Moon River in the event of future equity financings. (Figure 4.1 and Figure 4.2, below).

Figure 4.1. Property Location Map
Source: Hatch, 2007

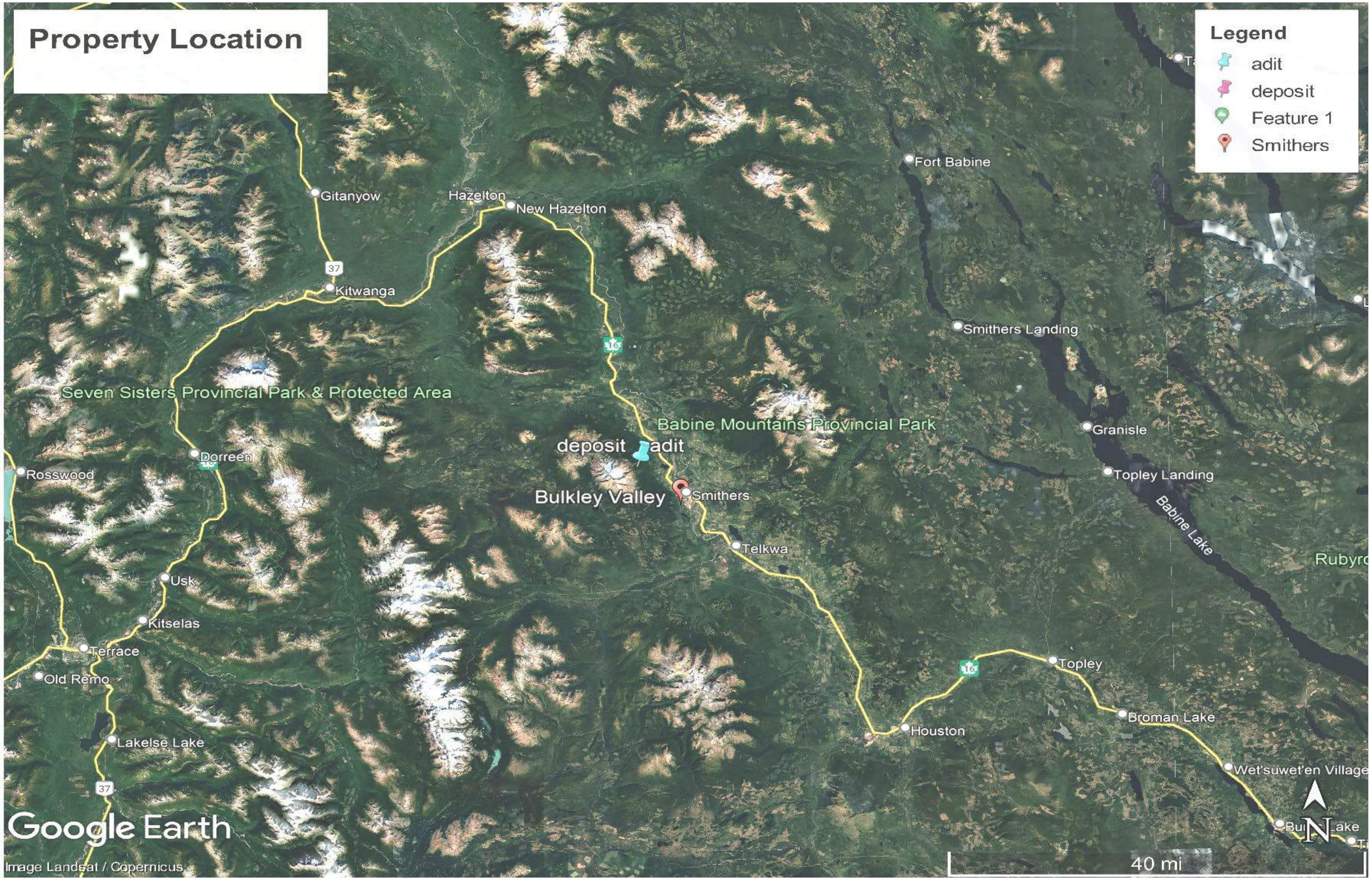


Figure 4.2. Property Location

Source: Google Earth™

The authors did not carry out a title search for this report; however, tenure data was obtained from British Columbia Mineral Titles On-Line and believed to be accurate (see Table 4.1, below).

TABLE 4.1
MINERAL LEASES AND CLAIMS – DAVIDSON PROPERTY

Title Type	Title Number	Issue Date	Good to Date	Term Expiry Date	Area (ha)
Mineral Lease	243455	06/27/62	06/27/24	01/10/25	214.1
Mineral Lease	243475	01/10/68	01/10/24	01/10/25	289.0
Mineral Lease	243476	01/10/68	01/10/24	01/10/25	299.9
Mineral Lease	243477	01/10/68	01/10/24	01/10/25	292.8
Mineral Lease	243478	01/10/68	01/10/24	01/10/25	342.5
Mineral Lease	243479	01/10/68	01/10/24	01/10/25	193.6
Mineral Claim	1102041	02/06/23	02/06/24		279.8
		Total Area			**1,911.6**

All Mineral Leases have been legally surveyed by a British Columbia Land Surveyor and the survey has been approved by the Surveyor General. These are conditions in the Mines Act must be upheld before the Commissioner will grant a mining lease.

The small ‘cells’ that make up the one mineral cell title claim are selected online by using the British Columbia government’s system of electronic ‘online’ staking and choosing a pre-determined polygon ‘net’ of cells that covers the entire province. Each of these cells is based and located on a Universal Transverse Mercator (UTM) geographical description. These claims have not been physically surveyed in the field (see Figure 4.3, below).

AMPL is not aware of any environmental liabilities associated with this property at this time.

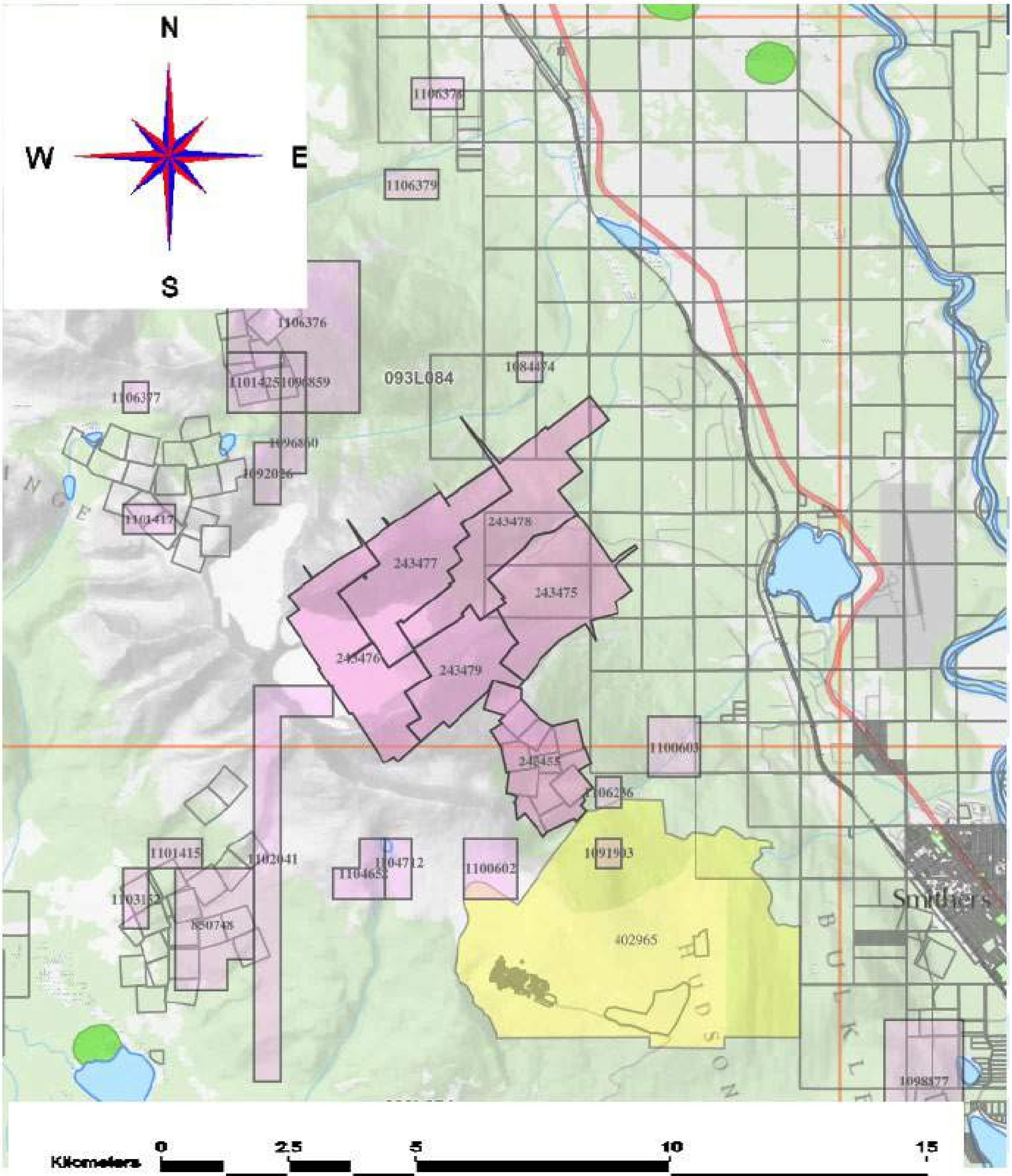


Figure 4.3. Claim Location Map

Source: B.C. (https2://www.mtonline.gov.bc.ca/mtov/map/mto/)

Several outdated Social/Community Impact reviews were conducted. The most recent appears to be the Hatch Feasibility Study titled "Blue Pearl Mining Ltd. Davidson Project, Feasibility Study" – in summary they summarise:

> *There remain some socio-community concerns regarding employee and supply traffic on local roads, especially during shift changes, the change in land use associated with the planned closure of the existing access road to the 1,066 m Ad-it, and noise generated during surface blasting. Although these do not affect the overall feasibility of the Project, these are issues that may need to be addressed further by BPM in consultation with the local community.*

In addition, the Project takes place in the traditional lands of the Wet'suwet'en First Nations (see Figure 4.4, below).

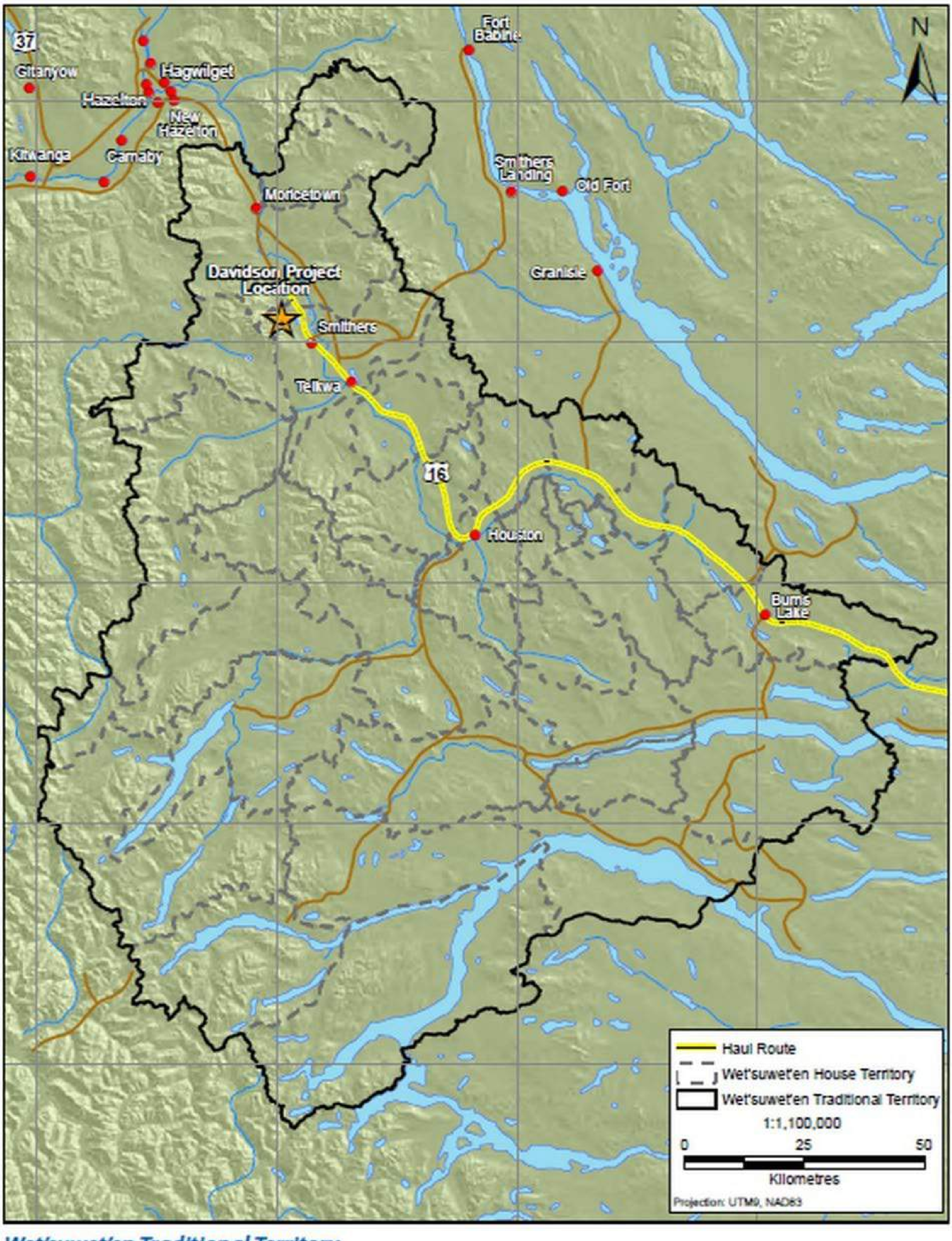


Figure 4.4.   Wet'suwet'en Traditional Territory
Source: AMPL, 2023

5.0 ACCESSIBILITY, CLIMATE, LOCAL RESOURCES, INFRASTRUCTURE, AND PHYSIOGRAPHY

5.1 EXISTING ACCESS, INFRASTRUCTURE, AND CLIMATE

5.1.1 Access and Infrastructure

Smithers is a town in northwestern British Columbia, on the Yellowhead Highway approximately halfway between Prince George and Prince Rupert. With a population of 5,351 in 2016, Smithers provides service coverage for most of the Bulkley Valley. Road access to the property is from the town of Smithers some 8.9 km (5.5 miles) northwest to the portal located at 1,067 m (3,500-ft) elevation. The site is currently inaccessible as the road is blocked with tree blowdowns and washouts in areas. Access will need to be re-established.

The region has excellent infrastructure for mining development. Wireless and wire-line telecommunication services, electrical power, paved highways, and railways are present locally throughout the area. Air Canada, Central Mountain Air, charter airlines, and helicopter companies provide multiple daily flights. Via rails, Jasper-Prince Rupert makes a scheduled stop three times a week in each direction. When Greyhound cancelled bus service in 2019, BC Bus North became the replacement operator for a twice weekly service.

A 138 kV power line is less than 3 km from the portal and the main CN rail line to Prince Rupert parallels the highway at the base of the mountain.

As of the writing of this report, there are no limits on the work being proposed other than that a Notice of Work may be required to reactivate the road. There is no limit on the length of the operating season.

Sufficient water supply exists in the area to supply all needs for a possible mine. The area of the land holdings is considered to be sufficient for the required infrastructure for a mine.

5.1.2 Climate

The Bulkley Valley technically has a subarctic climate, although it is on the borderline of a humid continental climate. Winters are cold and cloudy but highly variable with a January average of -7.2 C (19.0 F). Snow is the main type of precipitation during winter. Warm spells can push temperatures above freezing during the winter months, while cold weather systems can reduce the temperature to less than -20 C (-4 F). The average annual snowfall is 182.7 cm (71.9 inches), with maximum accumulations of snow tending to happen in February when the average snow depth is 29 cm (11 inches).

Summers are warm, with average highs of about 22 C (72 F) and an extreme high of 36.0 C (96.8 F). Nighttime temperatures are often cool, with normal nighttime lows under 10 C (50 F). Depending on the year, there may be very little or a lot of precipitation. Spring and fall are short transition seasons. Smithers receives an average of 508.5 mm (20.02 inches) of precipitation a year, with February through April being the driest months. Smithers receives 1,621 hours of bright sunshine a year, ranging from a minimum of 12% of possible sunshine in December to a maximum of 47% of possible sunshine in August.

5.2 LOCAL RESOURCES

The main industries in the Bulkley Valley are lumber, logging, farming, and tourism; however, mineral exploration and mining have been very important contributions to the local economy. A skilled labour force is readily available. Infrastructure is well developed and includes a 138 kV power line less than 3 km from the Davidson portal and a main CN rail line to Prince Rupert that parallels Highway 16 at the base of Hudson Bay Mountain.

The forestry industry has remained dominant. Agriculture has comprised dairy and beef ranching with opportunities for large-scale greenhouse operations. Tourism resources offer fishing, hunting, and hiking in spectacular terrain. Potential exists for expanding the mining industry, but residents oppose any new coal mines. The open pit Huckleberry Mine, 123 km (76 miles) southwest of Houston, opened in 1997. Owing to low copper and molybdenite prices, production ceased in 2016. At the time, Huckleberry employed 260 people, 80% from Bulkley Valley communities.

Prince George is the largest centre located in the region with a population of approximately 90,000. The economy of Prince George in the first decade of the 21st century has come to be dominated by service industries. The Northern Health Authority, centred in Prince George, has a $450 million annual budget and invested more than $100 million in infrastructure. Part of these investments was the 2012 opening of the BC Cancer Agency's Centre for the North, which includes radiation therapy facilities and associated buildings for modern cancer care.

Education is another key dominant part of this city. The University of Northern British Columbia, the College of New Caledonia, and School District #57 contributes more than $780 million into the local economy annually.

Forestry dominated the local economy throughout the 20th century, including plywood manufacture, numerous sawmills, and three pulp and pellet mills as major employers and customers. The spruce beetle epidemic of the late 1980s and 1990s resulted in a short-term boom in the forest industry as companies rushed to cut dead standing trees before the trees lost value. Sawmill closures (and the creation of 'supermills') occurred around 2005 and the largest pellet mill closed in 2022 due to dwindling supply and lack of a seaport. Mining exploration and development may become the future of Prince George. Prince George estimates that the Nechako Basin contains over 5,000,000 barrels (bbl) (790,000 cubic meters (m^3)) of oil.

Other industry includes two chemical plants, an oil refinery, brewery, dairy, machine shops, aluminum boat building, log home construction, value added forestry product, and specialty equipment manufacturing. Prince George is also a staging centre for mining and prospecting, and a major regional transportation, trade, and government hub. Several major retailers are expanding into the Prince George market, a trend expected to persist. In recent years, several market research call centres have opened in Prince George.

Prince Rupert is a port city in the Province of British Columbia. Its location is on Kaien Island, near the Alaskan panhandle. It is the land, air, and water transportation hub of British Columbia's North Coast, and has a population of 12,220 people as of 2016.

Prince Rupert relies on the fishing industry, port, and tourism. The port possesses the deepest ice-free natural harbour in North America, and the third deepest natural harbour in the world. Situated at 54° North, the harbour is the northwesternmost port in North America linked to the continent's railway network. The port is the first inbound and last outbound port of call for some cargo ships travelling between eastern Asia and western North America since it is the closest North American port to key Asian destinations. The CN Aquatrain barge carries rail cargo between Prince Rupert and Whittier, Alaska, USA.

5.3 PHYSIOGRAPHY

The Property is located in the Bulkley Valley of west central British Columbia, approximately 9 km northwest of the town of Smithers on the southwest flank of Hudson Bay Mountain. The high cirque valley on the east side of the mountain is occupied by the retreating Kathlyn Glacier. There are no roads to the surface projection of the underlying Davidson molybdenum resource other than to the 1,922 m-long access adit driven from the 1,066.8 m (3,500-ft) elevation level. Much of the surface access to the property is by helicopter.

The climate in the Bulkley Valley has cool to moderate summers and longer cold winters with temperatures ranging from maximum highs of 37°C (98°F) to lows of -44°C (111.2°F). Averages are -10°C (50°C) in January to 14°C (57.2°F) in July. The average annual snowfall is 1.5 m (59 inches). Rain can occur in any month and ranges from an average low of 0.004 m (0.17 inches) in February to a high of 0.05 m (1.92 inches) in October.

The Bulkley Valley is sparsely covered with pine, spruce, and balsam with more heavily forested areas on the lower slopes of the Mountain. Tree line is about 1,580 m (5,200-ft) (see Figure 5.1 and Figure 5.2, below).

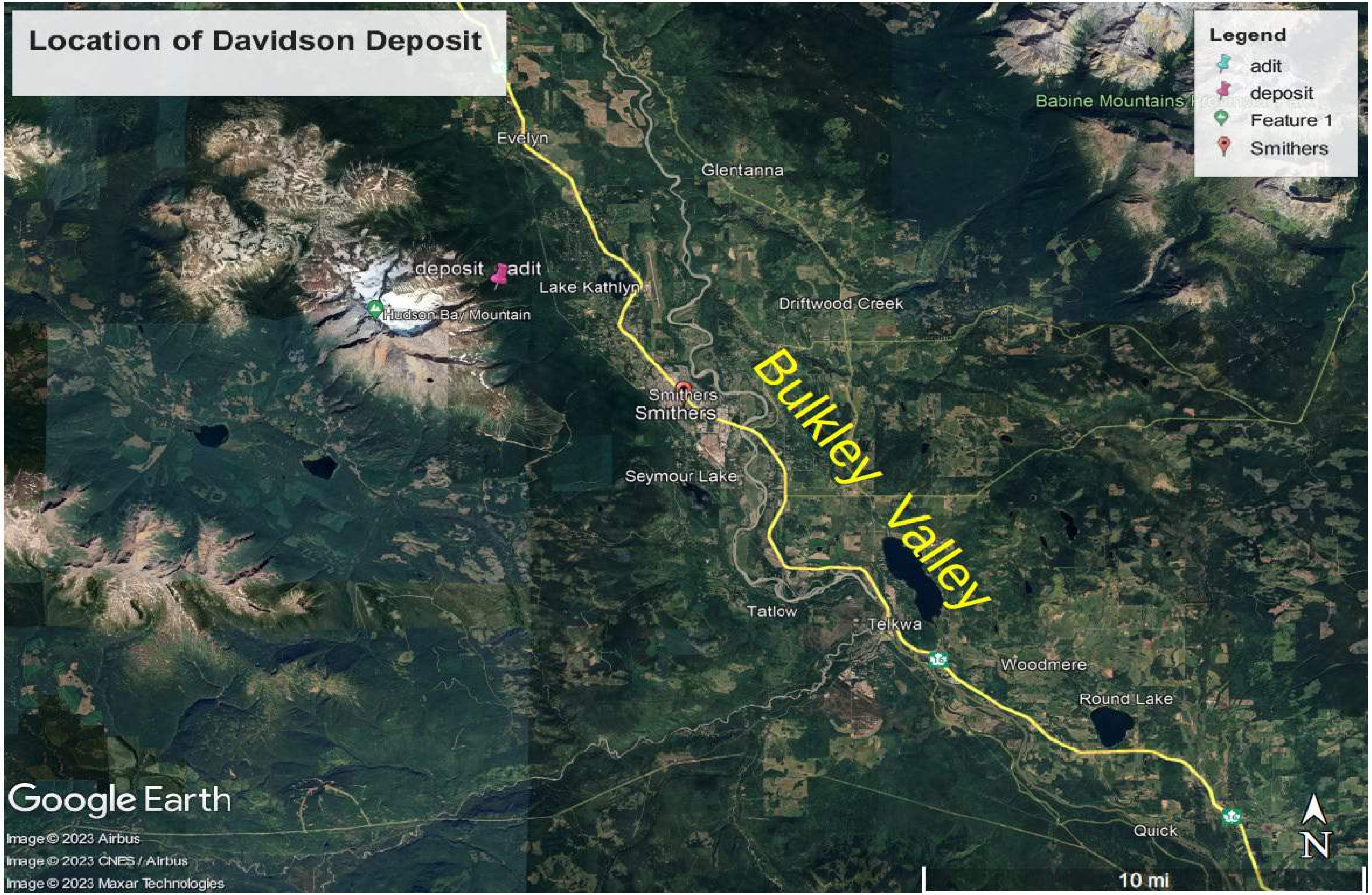


Figure 5.1 Bulkley Valley, British Columbia
Source: Google Earth™

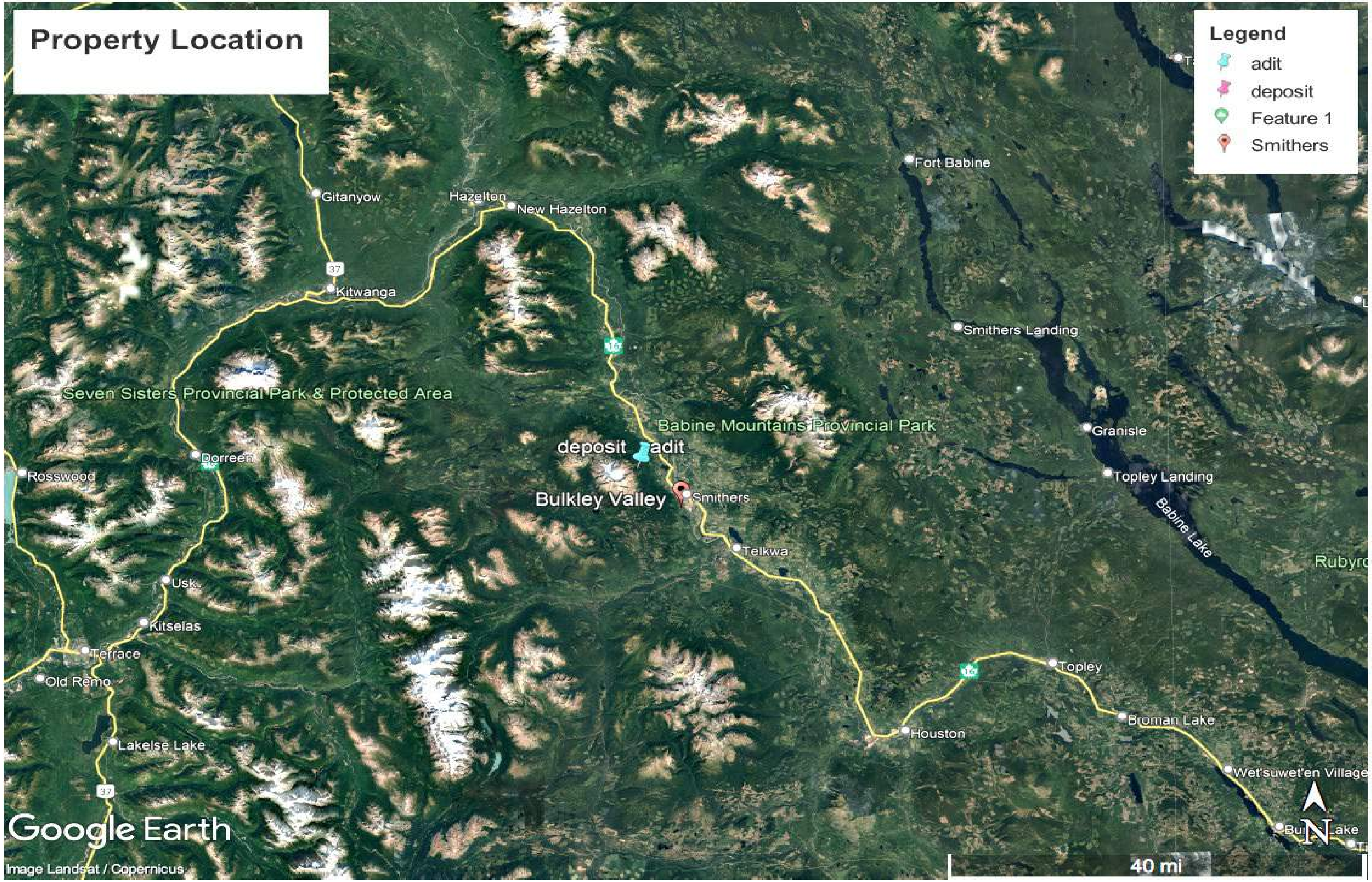


Figure 5.2 ***Smithers, British Columbia***
Source: Google Earth™

6.0 HISTORY

Molybdenum was first reported in an outcrop on Hudson Bay Mountain by the Geological Survey of Canada in 1944. The first claims were staked by William Yorke-Hardy in 1957. The Property was optioned to American Metal Co. (AMAX) from 1957 to 1959 during which time they completed a program of surface trenching and limited drilling.

In 1961, the Property was optioned by Climax Molybdenum Corp. of B.C. Ltd. (Climax). During the period 1961 to 1963, Climax completed a total of 4.420 m (14,502-ft) of diamond drilling identifying two shallow dipping bodies of molybdenite-scheelite (Mo-$CaWO_4$) mineralisation.

In 1966, an adit was collared at an elevation of 1,067 m (3,500-ft) and driven 660 west for 1,708 m (5,600-ft) then due west for 214 m (700-ft) from the east slope of Hudson Bay Mountain, from which two crosscuts were developed for underground drilling. A total of 164 diamond drill holes (DDH) were completed; 41 from surface totaling 23,500 m and 123 holes in fans from underground stations located on roughly 34.907 m centres (100-ft). Climax completed the outright purchase of the Yorke-Hardy in 1971.

A summary of work completed on the Property by Climax between 1962 and 1991 and Blue Pearl between 2006 and 2008 is taken from the BC Government's MINFILE and BC Assessment files. Other notable dates are included (see Table 6.1, below).

TABLE 6.1
SUMMARY OF WORK COMPLETED ON YORKE-HARDY BETWEEN 1962 AND 1991

Year	Description	Reference
1962	Geological Mapping	Assessment Report 471
1963	Airborne Magnetic Survey	Assessment Report 545
1968	Soil geochemical survey: 388 samples	Assessment Report 1730
1968	Soil geochemical survey: 205 samples	Assessment Report 2245
1969	Adit re-opened and ventilated, 1,609 m (5,200 ft) of track ballastede/Grid cutting and geological mapping	
1973	Grid cutting and geological mapping	Assessment Report 4756
1973	Underground diamond drilling, 5 BQ holes, 2,239 m and 273 assays	Assessment Report 4871
1974	Diamond Drilling, 3 holes BX, 146 m	Assessment Report 5041
1976	Diamond Drilling, 2 holes, BQ, 183 m	Assessment Report 5928
1977	Diamond Drilling 2 holes BQ, 69 m	Assessment Report 6480
1979	Diamond Drilling 4 holes HQ, 527 m	Assessment Report 7565
1979	Underground Diamond Drilling 14 holes, 1,884 m	Assessment Report 7780
1981	Preliminary geotechnical and environmental study of a proposed tailings site	Assessment Report 10370
1989	Soil geochemical survey, 264 samples	Assessment Report 18236
1990	Litho geochemical survey, 283 samples	Assessment Report 19569
1990	Soil geochemical survey, 153	Assessment Report 20797
1991	Geochemical surveys, 12 rock, 310 soil samples	Assessment Report 21743
1996	Climax sold the Property to Donald Davidson	

6.1 PREVIOUS WORK

Over the life of this Property, several Resource estimates have been completed.

In 1981, R.C. Steininger utilised all drill holes (DDH-1 to DDH-164) and a sectional technique on cross sections spaced 30 m (100-ft) apart to estimate at a 0.2% MoS_2 cut-off, 22.7 Mt grading 0.401% MoS_2. A tonnage conversion factor of 12.12 cubic feet per tonne (ft^3/tonne) was used for this calculation. These Mineral Resource estimates are viewed as historical Resources and have not been verified by a Qualified Person, as required by NI 43-101 and should not be relied upon. The issuer is not treating this Resource as being current. It is important to note that all these historical and previous Mineral Resource estimates are superseded by the Updated Mineral Resource estimate presented in Section 14.0 of this Technical Report.

In 1981, A. Noble of AMAX Technical Services calculated a Resource within the same 0.2% MoS_2 shell used by Steininger but used kriging and a 12.5 ft^3/tonne tonnage factor and 15.24 × 15.24 × 15.24 m (50 × 50 × 50 ft) blocks. At a 0.2% MoS_2 cut-off, Nobel estimated 53.3 Mt grading 0.275% MoS_2. These Mineral Resource estimates are viewed as historical Resources and have not been verified by a Qualified Person, as required by NI 43-101 and should not be relied upon. The issuer is not treating this Resource as being current. It is important to note that all these historical and previous Mineral Resource estimates are superseded by the Updated Mineral Resource estimate presented in Section 14.0 of this Technical Report.

In 1998, G.H. Giroux, of Giroux Consultants Ltd. (GCL), completed a kriged estimate using the same database of 164 drill holes, a larger mineralised shell, a 15.24 × 15.24 × 7.62 m (50 × 50 × 25 ft) block model, and a tonnage conversion factor of 12.5 ft^3/tonne. At the same 0.2% MoS_2 cut-off, a Resource of 77.63 Mt grading 0.286% MoS_2 was classed Measured plus Indicated – Verdstone Gold Corporation. These Mineral Resource estimates are viewed as historical Resources and have not been verified by a Qualified Person, as required by NI 43-101 and should not be relied upon. The issuer is not treating this Resource as being current. It is important to note that all these historical and previous Mineral Resource estimates are superseded by the Updated Mineral Resource estimate presented in Section 14.0 of this Technical Report.

In February 2005, GCL estimated MoS_2 and WO_3 content by ordinary kriging. These blocks were then classified as Measured/Indicated/Inferred using the kriged estimation error for each block. Based on 1,997 measured specific gravities (SG) from drill core, an average 2.66 was used, with an Imperial tonnage conversion factor of 12.05 ft^3/tonne. A total of 166 drill holes containing 17,737 assays for MoS_2 were available for this analysis. A similar procedure was used to evaluate the 2,613 samples with WO_3. Measured and Indicated Resource with a 0.2% MoS_2 cut-off was estimated at 82.98 Mt of 0.295% MoS_2 and 0.035% WO_3 – NI 43-101 Report for Patent Enforcement and Royalties. These Mineral Resource estimates are viewed as historical Resources and have not been verified by a Qualified Person, as required by NI 43-101 and should not be relied upon. The issuer is not treating this Resource as being current. It is important to note that all these historical and previous Mineral Resource estimates are superseded by the Updated Mineral Resource estimate presented in Section 14.0 of this Technical Report.

In April 2007, GCL completed a new Resource estimate. Based on a cut-off of 0.12% Mo (0.20 MoS_2), the Measured and Indicated Mineral Resources were estimated to be 77.2 Mt with an average grade of 0.169% Mo and contained 288 million pounds of Mo. Measured Mineral Resources were estimated at 45.9 Mt with an average grade of 0.18% Mo and contained 182 million pounds of Mo. Indicated Mineral Resources were estimated at 31.3 Mt with an average grade of 0.154% Mo and contained 106 million pounds of Mo. These estimates do not include the lower zone that returned several high-grade molybdenum drill intercepts in the 2006 to 2007 period – Internal Report for Blue Pearl Mining. These Mineral Resource estimates are viewed as historical Resources and have not been verified by a Qualified Person, as required by NI 43-101 and should not be relied upon. The issuer is not treating this Resource as being current. It is important to note that all these historical and previous Mineral Resource estimates are superseded by the Updated Mineral Resource estimate presented in Section 14.0 of this Technical Report.

In September 2016, GCL completed a revised Resource estimate at the request of Mr. Jamie Levy, President and CEO of Darnley Bay Resources (Darnley Bay). Darnley Bay had entered into an agreement with Roda (Mr. Donald Davidson) to obtain a 100% interest in the Property. These Mineral Resource estimates are viewed as historical Resources and have not been verified by a Qualified Person, as required by NI 43-101 and should not be relied upon. The issuer is not treating this Resource as being current. It is important to note that all these historical and previous Mineral Resource estimates are superseded by the Updated Mineral Resource estimate presented in Section 14.0 of this Technical Report.

Using an Internal Study by Hatch, Giroux considered two underground mining scenarios using two different MoS_2 cut-offs. A 0.20% MoS_2 cut-off corresponding to a bulk mining approach with onsite processing facilities while a cut-off of 0.28% MoS_2 reflected a more selective direct shipping alternative of hauling ore to another mill for processing. Measured plus Indicated Resource estimates were:

- 90.1 Mt, 0.286% MoS_2 and 0.034% WO_3 – 0.20% cut-off (340.5 million pounds Mo and 67.5 million pounds WO_3).

- 34.4 Mt, 0.374% MoS_2 and 0.036% WO_3 – 0.28% cut-off (170.1 million pounds Mo and 27.3 million pounds WO_3).

7.0 GEOLOGICAL SETTING AND MINERALISATION

This section describes the regional geological setting and property-scale geology for the Davidson Project Yorke-Hardy Deposit.

7.1 REGIONAL GEOLOGY

(After Hutter and L'Orsa, 2007)

> *The oldest rocks in the general area of Hudson Bay Mountain are island arc volcanics and sediments of the Lower to Middle Jurassic Hazelton Group (Fig. 3and 4), which form a part of the accreted Stikine terrain. These rocks are followed in age by largely sandy successor basin formations of the Middle to Upper Jurassic Bowser Lake Group and the Lower Cretaceous Skeena Group that were deposited as sediments were eroded from rising landmasses while Stikinia and other terrains collided with North America during Middle to Late Jurassic time. Continued subduction and pressure from advancing Pacific plates during Cretaceous-early Paleogene time resulted in the development of the Skeena fold and thrust belt and in an episode of igneous activity that formed the Bulkley plutonic suite and continental volcanic rocks of the Kasalka Group. A shift in Pacific plate movement from a northerly to a north-westerly direction in Eocene time was accompanied by a trans-tensional regime resulting in the episode of intense volcanism that emplaced the bimodal Ootsa Lake-Endako volcanic assemblages and resulted in the development of basin-and-range structures that account for the Bulkley Valley graben and adjacent fault- block mountain ranges.*
>
> *There are three major suites of granitic intrusive rocks in the region: The Topley plutonic suite (Late Triassic to Middle Jurassic), Bulkley plutonic suite (Late Cretaceous) and the Nanika plutonic suite (Eocene), as outlined by Carter (1981). The Bulkley plutonic suite is represented by a northerly-trending series of intrusions that host or are associated with several porphyry copper-molybdenum systems including the Huckleberry mine and the molybdenum and tungsten bearing system of the Davidson deposit* (see Figure 7.1, below).

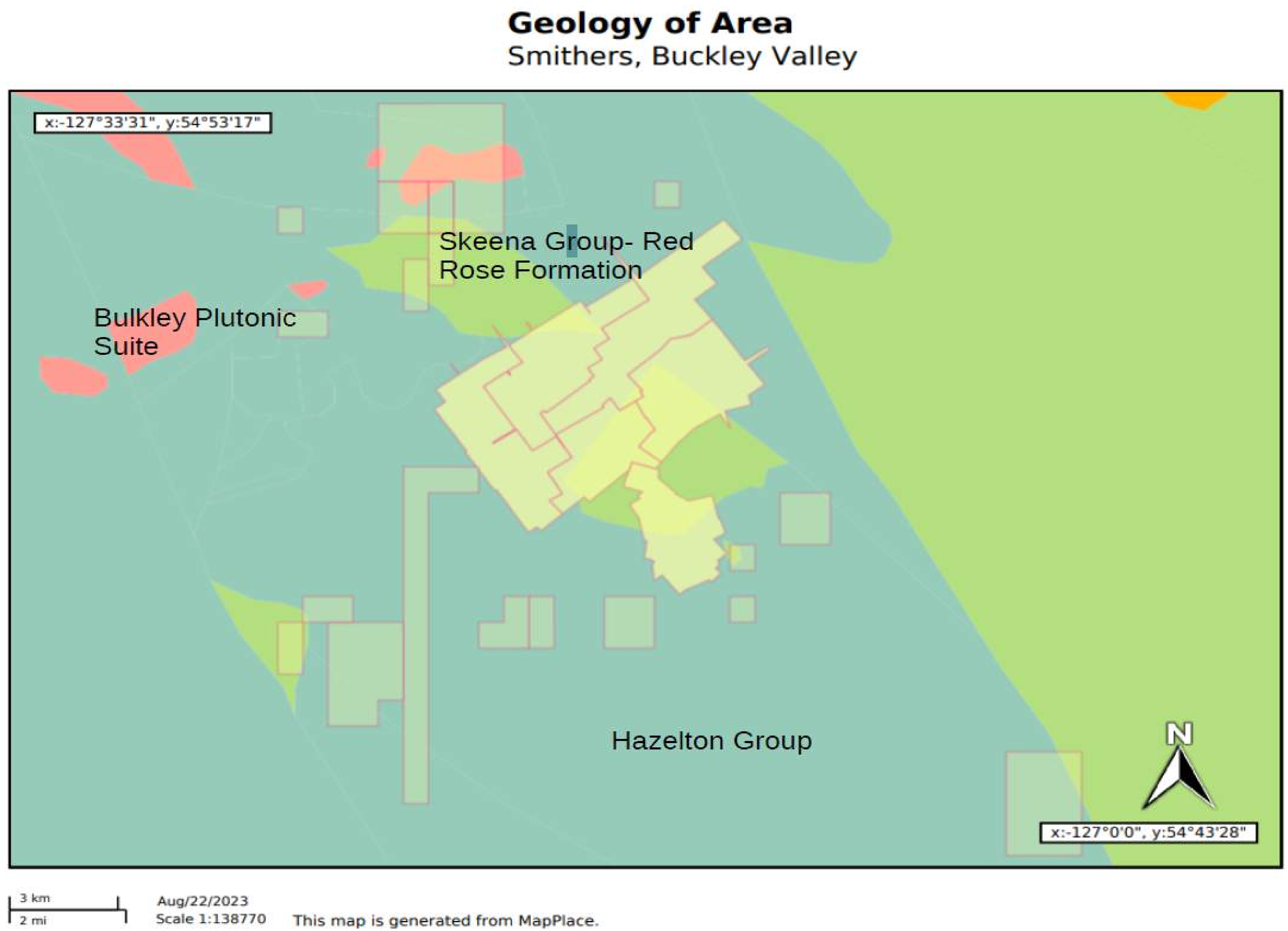


Figure 7.1. ***Regional Geology – Davidson Area***
Source: AMPL, 2023

7.2 PROPERTY GEOLOGY

(After Atkinson, 1995)

Mineralized and altered lithologies include:

- *Early Cretaceous Skeena Group greywacke, sandstone and mudstone with coal seams*
- *Lower to Middle Jurassic Hazelton Group mafic to felsic flows, tuff, breccia and lesser mudstone, conglomerate and limestone*
- *Middle to Late Jurassic granodiorite sill, metabasaltic sills and dykes*
- *Late Cretaceous to Early Tertiary intrusions that include a rhyolite plug, quartz-feldspar porphyry dykes and the Hudson Bay Mountain stock* (see Figures 3 and 4 from Atkinson, 1995).

The granodiorite sill intrudes Hazelton Group volcanic rocks exhibiting concordant and discordant contacts. The sill, defined by drilling, over a 1200 m strike length, dips at 20º southeast steepening to 70º at the 16000 E cross-cut and ranges in thickness from 75 m to 550 m. Emplacement of the sill may be along an east-dipping pre-mineral thrust fault (Kirkham, 1966).

Atkinson suggests the granodiorite sill could be sub-divided into three lithologies based on texture and mineralogy.

- The highest-grade mineralisation is within the basal and southern portions of the sill, characterised by granitic texture. This granitic portion has the highest mafic content of the sill, estimated between 5% to 10%.

- The central and upper part of the sill is more porphyritic with an aphanitic groundmass and euhedral to ragged plagioclase phenocrysts, euhedral quartz phenocrysts, and clots of chlorite, pyrite, and magnetite replacing primary mafic minerals. This porphyritic section normally has intrusive contacts with the other parts of the sill.

- The uppermost and northern sections of the sill are light coloured aplitic granodiorite with intergrowths of quartz and feldspar.

Hazelton volcanic blocks, up to 3 m across, are found within the sill and have been partially digested suggesting interaction with the granodiorite melt. Breccia zones with sub rounded sill fragments contained within a mafic matrix are locally common.

The sill and host Hazelton Group rocks are crosscut by numerous basaltic dykes, sills, and erratically shaped intrusive bodies.

A rhyolite plug intrudes both the Hazelton Group and the granodiorite sill and is truncated by the Hudson Bay stock. This plug is 450 m × 300 m in size and roughly oval in plan. The composition is calc-alkaline quartz-feldspar porphyry.

The Hudson Bay stock, which ranges in composition from quartz monzonite to granodiorite, has been intersected in its east flank by four drill holes at depths ranging from 400 m to 1,000 m.

A sub-radial quartz-feldspar porphyry dyke swarm related to the Hudson Bay stock, has been mapped on surface, underground and intersected in drill holes (see Figure 7.2 and Figure 7.3).

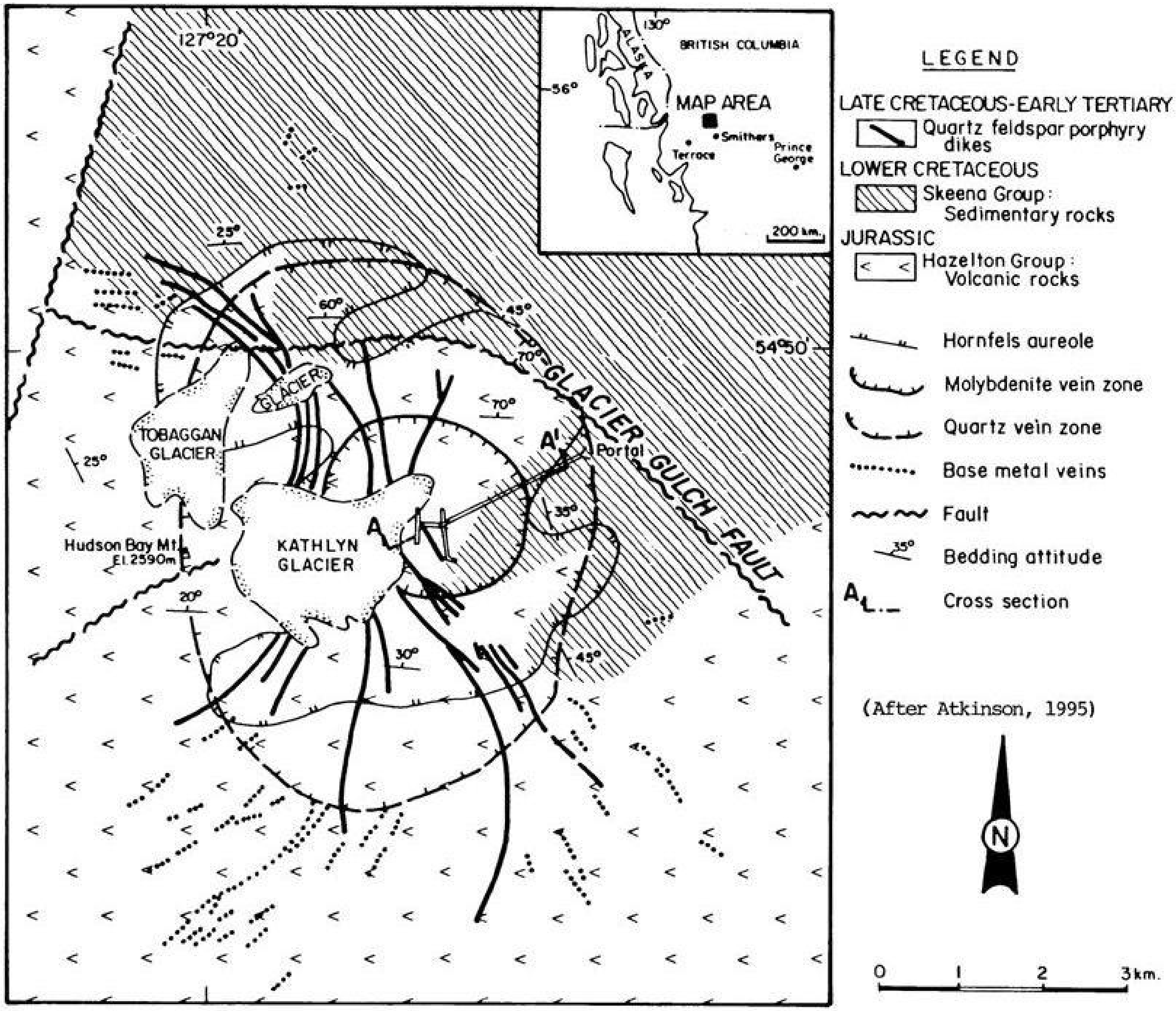


Figure 7.2. Property Geology Plan View – Davidson Property, after Atkinson, 1995

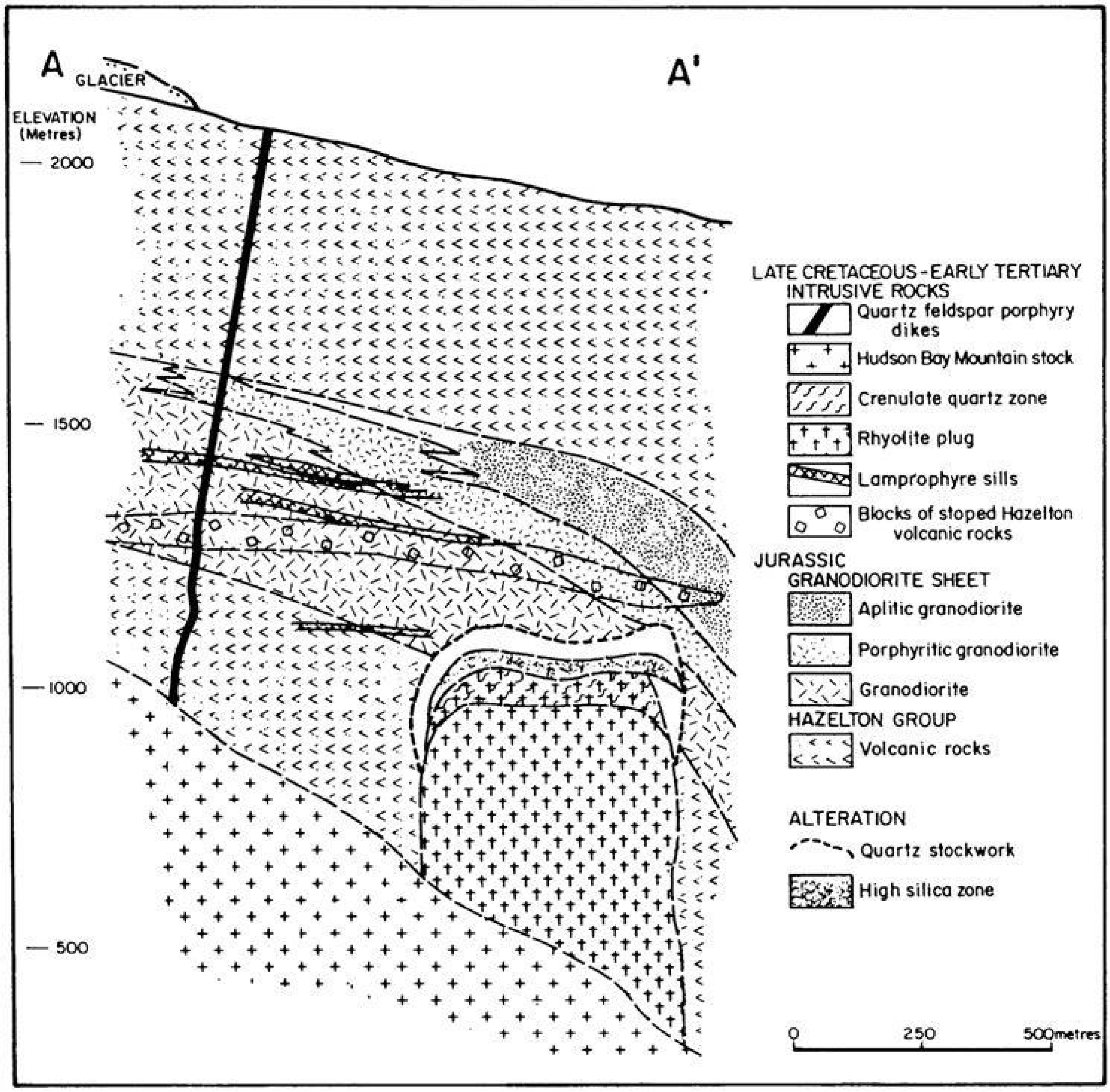


Figure 7.3. Property Geology Cross Section, A-A' West to East, after Atkinson, 1995

7.3 MINERALISATION

The Property is a molybdenite-scheelite porphyry deposit 2.5 km across and extending up to 2 km in depth that consists of moderately to steeply dipping stockwork veins ranging from hairline to 5 mm in width. Stockwork veins exhibit a complex history of crosscutting relationships described by Atkinson (1995) as follows:

- *early stockwork assemblages include andradite garnet, Epidote, chlorite, magnetite and quartz followed by molybdenite occurring as both fine-grained fracture coatings and within veins with quartz and feldspar gangue.*
- *early assemblages are cut by banded veins of fine-grained quartz +molybdenite ± pyrite ± scheelite and less common banded quartz + magnetite up to 1 m wide.*
- *the banded veins are in turn cross cut by magnetite + scheelite and quartz + K-feldspar + scheelite veins (which constitute the principal tungsten mineralizing event).*

- *these veins are themselves cut by pegmatitic quartz + molybdenite ± calcite ± scheelite ± K- feldspar ± pyrite veins up to 10 cm in width. -the youngest veins contain pyrite ± chalcopyrite and calcite.*

The granodiorite sill hosts the high-grade molybdenite zones and has abundant banded and pegmatitic veins. Its more massive composition provided a better host for veins than the more bedded and foliated Hazelton Group lithologies. The rhyolite plug contains mineralisation, is crosscut by mineralised rhyolite dykes, and contains mineralised breccia fragments. The Hudson Bay stock is weakly mineralised and exhibits a sharp decrease in molybdenite grade away from the edges. Finally, the quartz-feldspar porphyry dykes are crosscut in places by pegmatitic quartz-molybdenite veins.

In general, the molybdenite is well crystallised and occurs as stringers, patches, veinlets, and individual grains. The individual grains or crystals ranged in size from as large as 3,000 mm to the smallest size observed being 20 mm. Scheelite and powellite occur as clumps and clusters as large as 300 mm; however, the individual grains or crystals range in size from 4 mm to 40 mm (Enochs, 1980).

The two main zones of molybdenite mineralisation, within the Davidson deposit, have been named the Main and Lower zones, respectively. These are high-grade zones within a much larger but lower grade zone defined by the ≥0.17% MoS_2 shell.

The Main zone is hosted by the granodiorite sheet and is defined by the ≥0.3% MoS_2 grade shell. It is an irregular zone, roughly circular in plan view and elliptical in cross-section, with maximum horizontal dimensions of approximately 450 m and maximum vertical extent of approximately 200 m.

The general mineralised zones within the granodiorite, including the Main zone, has been described by Atkinson (1981) who reported two basic types of molybdenite-bearing quartz veins: **Type 1** (fine-grained molybdenite) and **Type 2** (coarse-grained molybdenite). The Type 1 veins are sub-divided into two sub-types: an early set of narrow (≤ 3 mm) veins that locally form stockworks and a set of much wider (≤ 60 cm) banded veins. The strongest set of banded veins dips to the southeast and east of the 15000 E crosscut, but progressively flattens to the northwest. Type 2 veins are up to 15 cm in width, carry molybdenite crystals ≤ 5 cm in diameter, and may have been the latest quartz-molybdenite veins to be deposited.

The Lower zone, as presently defined, was deposited mainly in the upper part of the rhyolite plug within the ≥ 0.3% molybdenite grade shell. With work still in progress, the zone appears to be elongated to the north-northwest with that dimension being approximately 250 m, and with a maximum width and height of approximately 100 m and 40 m, respectively. Both fine-grained and coarse-grained quartz-molybdenite veins occur in the Lower zone, although the vein type distinctions reported in the Main zone are not as clear in this zone, and the very coarse Type 2 veins are not present. The strongest molybdenite-bearing quartz veins are banded veins, interpreted to be gently southeasterly dipping, which continue past the plug to the southeast. Disseminated molybdenite is present in small amounts locally. There is a multiplicity of vein types still under study in the general area of the Lower zone, including early barren quartz veins, molybdenite-bearing veins with or without magnetite, pyrite or scheelite, and late pyrite-carbonate and finally carbonate veins.

Minor amounts of disseminated and fracture filling pyrite are always present (up to about 2%) within the deposit and chalcopyrite is present in small amounts locally. Veins of these sulphides are generally accompanied by quartz and carbonate minerals, including calcite. Tungsten usually occurs in scheelite and scheelite-powellite in quartz veins, as very fine-grained or coarse disseminations and in fracture-controlled disseminations in the host rock. Disseminated wolframite has been noted in a few intervals. Veins of calcite and other carbonate minerals appear to represent the last stage of vein formation and carbonates are also found disseminated in places.

Pyrrhotite is found with or instead of pyrite in places outside the ore zone. Rarely, pyrite, chalcopyrite, and pyrrhotite are found together in the same vein. Magnetite is found in several vein sets and can be abundant in places.

The rocks associated with the Davidson part of the system are generally silicified, biotitised, and more or less chloritised and some sections are pervasively altered by potassic feldspar and others by a quartz-sericite-pyrite alteration assemblage. Garnet and epidote are found in many sections.

7.4 ALTERATION

(After Atkinson, 1995)

> *A Hornfels aureole, characterized by development of radiating and zoned clots and veins of garnet, epidote, chlorite, Biotite, hornblende and amphiboles, extends from surface where it has been mapped over an area 7 km by 4 km (see Figure 3). Brown to red andradite garnet intergrown with quartz, chlorite, sericite, magnetite, carbonate and occasionally scheelite and rimmed by Epidote becomes increasingly common with depth. In some underground exposures of the sill, 30% of the wall rock is replaced by garnet clots to 10 cm across producing a spotted (appaloosa) texture.*
>
> *Primary igneous textures of the sill have been obliterated by the pervasive loss of mafic minerals and the development of chlorite ± magnetite pre-molybdenite hairline stockworks, clots and veins that may in part be attributed to hydrothermal alteration.*
>
> *Astride the contact of the rhyolite plug with Hazelton Group volcanic rocks and the granodiorite sill, quartz stockwork veins coalesce to form a high silica zone that mimics the shape of the top of the plug (see Figure 4). The high silica zone averages 40 m thick and contains trace fluorite, topaz, magnetite and Biotite.*
>
> *Hydrothermal alteration is fracture controlled. Vein alteration haloes rarely exceed a metre in width. Where veins are numerous, overlapping haloes form zones of pervasive alteration but deposit scale zonation has not been established. Within Hazelton Group rocks, hydrothermal alteration includes Na metasomatism, silicification and destruction of mafic minerals resulting in bleaching of the lithologies. Within the granodiorite sill alteration includes the development of pink potassic alteration which envelops magnetite, quartz, stockwork molybdenite, and pegmatitic quartz-molybdenite veins. Three pulses of hydrothermal fluids are interpreted from the cross-cutting relationships of the alteration envelope.*

8.0 DEPOSIT TYPES

The Davidson deposit is a porphyry molybdenum deposit that shares similar characteristics to the Climax type of molybdenum deposit including mineralised quartz-rich felsic intrusions, multiple mineralisation shells, uni-directional solidification textures, and geological setting (continental back-arc spreading environment). Westra and Keith (1981) classified the deposit as a subset of the Climax type, transitional toward calc-alkaline molybdenum stockwork deposits. Examples of deposits of this transitional type include Questa in New Mexico, USA and Mt. Hope in Nevada, USA. Available geochemical data indicate that the Davidson deposit is characterised by lower fluorine contents than those typical for a Climax type porphyry molybdenum deposit. Bright (1972) reported about 0.1% fluorine in the mineralised zone and about 0.05% fluorine below the mineralised zone, with localised elevated values of up to 2.7% fluorine. Atkinson (1981) reported less than 0.1% fluorine (0.013% to 0.042%) in 9 samples from the known rhyolite plug; there may be other plugs.

An alternative classification, outlined by Sinclair (1995), distinguishes between two classes of porphyry molybdenum deposit according to fluorine content in the intrusive rocks with which mineralisation is genetically associated: a low-fluorine type with generally less than 0.1% fluorine; and a high fluorine type (Climax type) with greater than 0.1% fluorine. The Davidson deposit can be considered as an example of a low fluorine type of porphyry molybdenum deposit.

The Davidson molybdenum-scheelite deposit is considered to be one of the British Columbia Porphyry molybdenite deposits (BC Model #L05: Porphyry Mo (Low F-type) and L07: Porphyry W) that are post-accretion and range in age from 138 to 8 million years.

9.0 EXPLORATION

No further exploration work has been done on the property since the 2016 Resource Update by Giroux.

10.0 DRILLING

The first recorded drilling on the Property (or Yorke-Hardy as it was previously known) was an 11-hole diamond drill program totaling 1,925 m completed in 1958 by AMAX, 5 of which were collared on the glacier. The program resulted in a large area of +0.1% MoS_2 defined but failed to identify additional geologic targets and the Property option was dropped.

In 1961, the Property was re-optioned by AMAX Exploration and 6 long holes numbered 12 to 17 and totaling 3,972 m located a zone of +0.2% MoS_2 from a zone 305 m to 610 m below surface. By 1964, an additional 17,258 m of drilling in 24 DDH (holes 18 to 41) had been completed. From this database, the first preliminary economic appraisal was completed in 1964 and the Project was transferred to Climax Molybdenum Corporation of British Columbia.

In the fall of 1966, an underground adit on the 1,066.8 m (3,500-ft) level was initiated to allow for underground drilling. The adit was collared on the east slope of Hudson Bay Mountain and driven 66° west for 1,708 m then due west for 214 m. In 1967, crosscuts at 15000 E and 16100 E were driven a total of 732 m to provide underground drill stations. Drilling commenced in January 1967 and 9 holes (Holes 42 to 50) were completed totaling 2,830 m in the 16100 E crosscut. During 1967 and 1968, an additional 9,931 m of diamond drilling was completed in Holes 51 to 72. Poor check sampling of assays indicated either sampling or analytical problems. A second economic appraisal was completed in 1969 (Jonson, 1969).

From 1969 to 1972, Climax completed a drill program that used new sampling procedures designed to improve sampling variability. They drilled 20 holes (numbered 73 to 92) totalling 4,318 m from 6 drill stations on the 15000 E crosscut during 1970 and an additional 46 holes (numbered 93 to 141) totalling 12,539 m in 1971. Two bulk sample raises were driven, centred on Drill Holes 81 and 82-82A at 17600 N and 17800 N, respectively. Drill Hole 82A was a twin of 82 drilled because the drillers forgot to take sludge samples in Hole 82. Each raise covered a distance of 46 m. Results from each 3.048 m (10-ft) round in each raised were sealed in 3-tonne crates and shipped to Climax's Pilot Plant in Golden, Colorado, USA.

The crosscut on 161S was extended 244 m at S45E from which Climax drilled an additional 5 holes in 1972-73 totaling 1,818 m to bring the total holes drilled to date to 146.

Further work on sampling protocol resulted in recommendations from Climax to increase the sample size. During the period 1979 to 1980, 6 new drill stations were slashed on odd-numbered sections of the 15000 E crosscut and 18 "up" holes were drilled totaling 3,321 m. The new sampling protocol crushed the entire 3.048 m (10-ft) section of HQ or NQ core (see Table 10.1, below).

TABLE 10.1 SUMMARY OF HISTORICAL DRILLING AT DAVIDSON				
Year	**Operator**	**Number of Holes**	**Hole Numbers**	**Total Meters Drilled**
1958	AMAX	11	1 to 11	1,925.45
1961-64	AMAX	30	21 to 41	21,207.07
1967-68	Climax	31	42 to72	12,747.29
1970	Climax	19	73 to 90	4,313.05
1971	Climax	49	91 to 139	12,495.31
1972-73	Climax	5	140 to 144	1,818.5
1979-80	Climax	20	145 to 164	3,321.39
2006	Blue Pearl	30	165 to 178	7,565.4
			181 to 196	
2007	Blue Pearl	23	197 to 219	7,421.83
	Totals	**218**		**72,815.29**

An additional 53 drill holes (165 to 219) were completed in 2006 (Holes 165 to 196) and 2007 (Holes 197 to 219), under the supervision of J. Hutter of Blue Pearl Mining (a wholly owned subsidiary of Thompson Creek). Drilling was contracted to Hy-Tech Drilling of Smithers, British Columbia. All core was NQ2 in size (50.8 mm diameter). Holes 165 to 196 were collared underground primarily on crosscut 15000 E and part of crosscut 16100 E. Holes 197 through 219 were collared from underground setups on the south end of the 16100 E crosscut.

Historic drilling was primarily spotted on 61 m sections with infill drilling on 30.5 m step-outs.

Drilling results suggest that mineralisation at the Property occurs at two zones; the Main deposit and the Lower deposit.

The 2006 to 2007 holes were drilled to test depth and lateral extents of the known Resource, verify previous historical grades of the Main deposit and better define the Lower deposit. Hole 179/179A was completed for geotechnical and environmental purposes (see Figure 10.1, below).

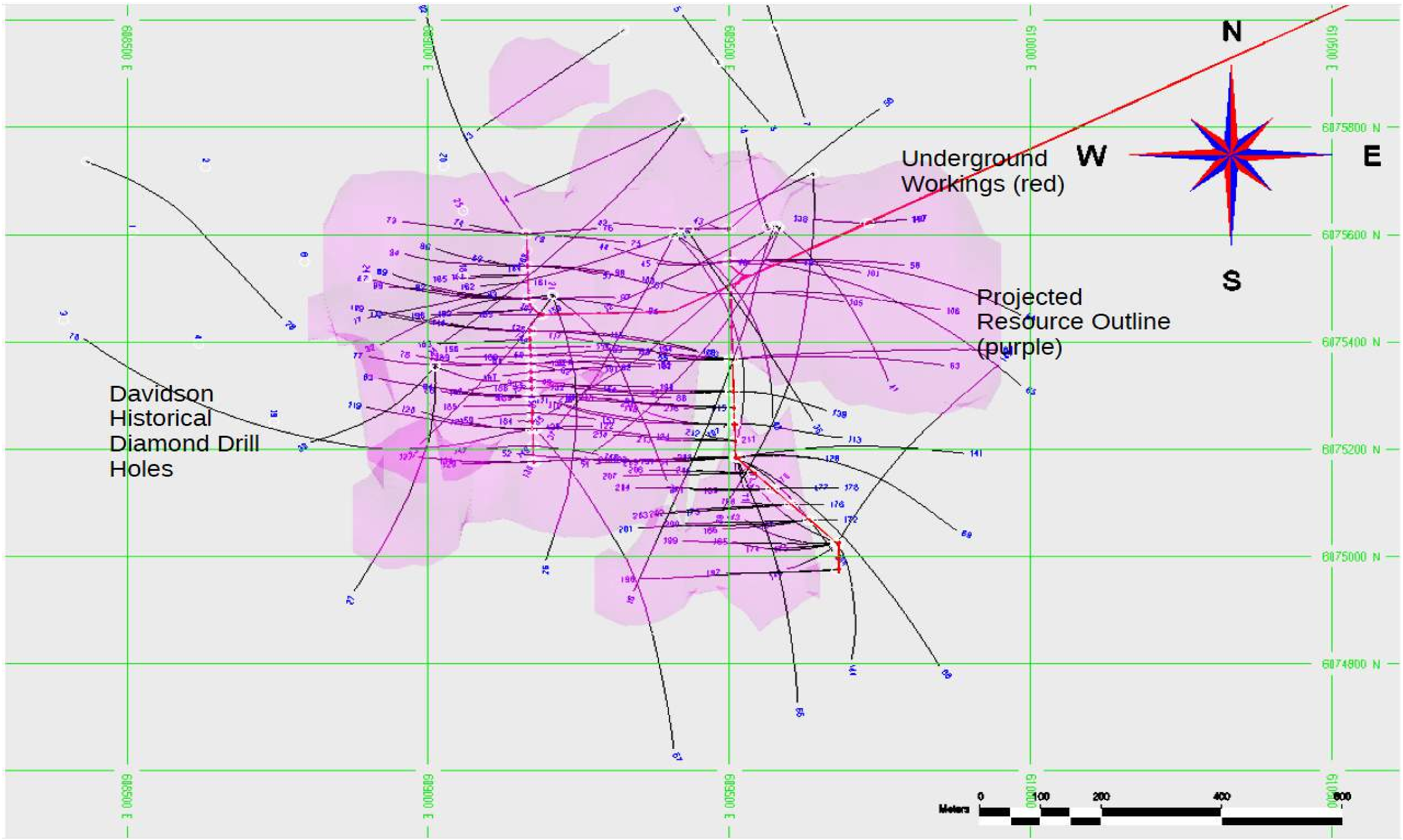


Figure 10.1. Historical Drilling at Davidson
Source: AMPL, 2023

10.1 DRILL HOLE SURVEYING

The survey database used in this Resource calculation suggests that azimuth and dip measurements for drill hole numbers 1 to 9 (including 18 to 22, 24, and 25) were only measured at the collar. All other hole numbers, within sequence 10 through 164, were measured every 30 m starting 15 m below surface. These holes were surveyed with an unknown survey instrument.

During the 2006 to 2007 drill campaigns, drill hole collar location surveys were completed by AllNorth Consultants Ltd. (AllNorth) of Smithers, British Columbia (Hole 165 to 219). Downhole surveys were collected approximately every 30 m (100-ft) downhole starting at least 15 m (50-ft) below the surface using the compass based "Flexit" tool by Fordia. Dip, azimuth, and magnetic intensity readings are collected and transmitted electronically to a surface data receiver. Magnetic disturbance, likely from magnetite, was observed in several of the holes and the survey was repeated by shifting the downhole sensor to lessen the local effects of the magnetite. This was not always successful (Snowden, 2008).

The database showed most holes deviated by azimuth and dip over an acceptable amount of 0-5°/100 m and 0-2°/100 m, respectively, both in positive and negative directions. There is no constant average, and any strong changes going downhole in azimuth are likely due to local concentrations of magnetite. Any future drill program should be aware of possible local and strong magnetic interference during a drill program.

10.2 MINE GRID COORDINATES

The underground workings had been surveyed to a local mine grid by previous operators using transit and tape, the technology available at the time. In 2005/2006, Kelly Grebliunas of AllNorth re-surveyed the workings using modern equipment. The new survey indicated a survey error of approximately 2.5 m (8-ft) over the 2 km distance of the workings. Existing control points were re-established in UTM and tied into the old mine grid to create a new mine grid. The equipment used was a Leica Geosystems Global Positioning Systems GPS Series 500. The level of accuracy achievable with this system using the Rapid Static Method is 5 mm to 10 mm.

The survey was then carried underground using existing control points with a Sokkia Total Station SET500. All available historic drill hole collars were re-surveyed, and the information gained was applied to assign new coordinates for the old drill holes that were no longer visible or could not be accessed. All new drilling was surveyed using the new mine grid. The old mine grid is no longer used. For reporting and engineering requirements, the drill hole surveys are converted to UTM (NAD 83) coordinates.

10.3 DIAMOND DRILL COORDINATES

Drill hole collars were surveyed by AllNorth using a Sokkia Total Station SET500. The initial azimuth and inclination of the drill hole was also surveyed at the collar. This was done by surveying the drill rod or drill slide at the beginning of the drill hole. Downhole surveys were taken at a distance of 15 m (50-ft) from the collar of each drill hole and then at intervals of every 30 m (100-ft). The instrument used was a Flexit tool, supplied by Fordia Ltd. This instrument incorporates a compass and a dip needle, both with electronic readout transmitted to a data pad by radio signal. The survey instrument measures the intensity of the magnetic field in addition to taking azimuth and inclination readings.

As with any compass-based instrument, the azimuth readings are subject to inaccuracies caused by local magnetic fields associated with occurrences of magnetite or pyrrhotite. Pyrrhotite is relatively rare in this deposit, but magnetite is common in veins and occasional coarse disseminations in the intrusive rocks and in veins and widespread fine disseminations in the volcanic rocks. It was often difficult to get reliable readings in the volcanic rocks, but this is not considered to be a serious problem as these rocks are only encountered toward the bottom of the drill holes, where survey errors are considered to be less significant.

Downhole surveying identified a problem with excessive deviation of nearly 3° per 30 m (100-ft) in DDH 165. This was remedied in succeeding holes by the use of a core barrel with an oversized outer diameter, which generally reduced deviation to less than 0.5° per 30 m (100-ft). The larger diameter core barrel can cause problems with drilling in bad ground, but conditions on this Property are generally good enough that any such problems are minimal.

Downhole survey readings resulting in azimuth deviations of greater than 1° per 30 m (100-ft) were viewed as being suspect. All downhole surveys were reviewed during drilling. Suspect surveys, where observed, were highlighted and where practical, the survey was repeated with the downhole instrument being shifted slightly within the drill hole in an attempt to mitigate the disturbance of proximal magnetite. Of 254 initial surveys, 51 were repeated. On analysis of the final results, 60 of the surveys produced results that appeared unreasonable, indicating deviations that would be unlikely or impossible. These 60 surveys were then adjusted to produce a smooth curve that could reasonably be followed by a drill string. In most cases, azimuths would tend to gradually increase with hole depth.

An inherent inaccuracy is present in surveying the rod or slide with a transit at the top of the hole, as the survey points are not very far apart, and therefore, a slight error in the surveyed location of either point induces a significant error in the azimuth or inclination of the hole. Additional sources of error associated with the initial azimuth and inclination survey data include the possibility of slight shifting of the drill between collaring and surveying, and deviation of the drill rod due to uneven ground during collaring. In the event of a discrepancy between the initial azimuth and inclination readings and those determined by downhole surveying, the downhole survey orientations were considered as being correct, if they appeared to be consistent and reasonable. In the event of poor-quality downhole surveys in the first part of the hole (6 of 30 holes), the collar survey was considered to be correct and was used to set the initial azimuth.

Of the 30 holes drilled, 18 had collar azimuth surveys that were within 1° of the adjusted initial downhole survey, 4 more varied by 2° or less, and another 4 varied by 3° or less. The azimuth surveys for DDH 170 varied by nearly 45°, but this hole was inclined very close to vertical, making the azimuth very hard to measure accurately and in any case of little consequence. There was a variance in azimuth surveys for the remaining 3 holes of 3.3°, 3.6°, and 7.5°. A variance of 7.5° is considered excessive; however, the downhole surveys for this hole were of sufficient quality to locate the hole reliably and were taken to be correct.

Downhole measurements of inclination rely only on gravity and, therefore, are not subject to magnetic interference, which causes difficulties with azimuth measurements. In all but three cases, the inclinations used for plotting were those returned by the downhole survey instrument. Changes in inclination never averaged more than 0.4° per hundred feet in any hole, and usually averaged less than 0.2° per hundred feet. In downholes, inclinations would usually tend to decrease slightly with increasing depth, whereas in upholes, the inclinations would tend to increase.

The inclination of the drill hole at the collar was also measured for 22 of the 30 holes using a machinist's protractor (interpolated to 0.1°) as an additional check on the surveying. In most cases, the collar survey, initial downhole survey, and machinist's protractor agreed to within 1°.

Mr. J.M. Hutter of Blue Pearl Mining Inc. reviewed all of the downhole survey data, making modifications where necessary, as previously indicated, and is of the opinion that the downhole survey data suitably define the traces of the drill holes for the 2006 drilling campaign, and is satisfied that the samples are, therefore, sufficiently accurately located in three dimensional (3D) space for a Resource estimation and to support an Indicated and/or Measured Resource classification.

11.0 SAMPLE PREPARATION, ANALYSES, AND SECURITY

Most of the sample preparation, analysis, and security is taken from Giroux and Cuttle's (2016) report. Procedures given are from previous reports and cannot be independently verified. The authors have no reason not to believe their findings and accept them as being adequate. No further diamond drilling has taken place since the last NI 43-101 report in 2016.

11.1 SAMPLING PROCEDURE – 1958 TO 1980 CLIMAX/AMAX

Records on sampling protocols and procedures are incomplete for drill holes 1 to 41. For drill holes 42 to 81 inclusive, the following flowsheets documenting sampling procedures were available in the geology office and core storage facility in Smithers (Smithers facility) and are shown in Section 11.1.2.

11.1.1 Sample Security (Climax/AMAX)

The security protocol for Climax/AMAX era drilling is unknown. As a result, the author must rely on previous work and descriptions by Mr. Davidson (personal communication, June 17, 2023). Mr. Davidson's description of sample handling, shipping, and core storage suggest a normal chain of custody practise by industry standards at the time.

11.1.2 Drill Core Sample Laboratory Preparation (Climax/AMAX)

Samples were prepared and assayed onsite with support from Climax Molybdenum Assay Laboratory in Golden, Colorado, USA. Some of the assay lab equipment, such as the Atomic Absorption Spectroscopy (AAS) analyser, are still at the Smithers facility.

Due to coarse-grained mineralisation (nugget effect) of the molybdenite occurring in clusters or vein stockworks, different sample preparations have been used on the Davidson Project over the years. Climax Molybdenum Assay Laboratory conducted a significant amount of study and check sampling captured in reports by Davidson (1972), Ingamells (1973), and Carson and Pitard (1979). This work led to modifications to the sample reduction scheme for holes up to hole 81 because grinding to 100 mesh caused balling and segregation of MoS_2 (Ingamells, 1973). Davidson (1972), Ingamells (1973), and Carson and Pitard (1979) are available in the Smithers facility. Figure 11.1, below, illustrates the sample preparation procedure.

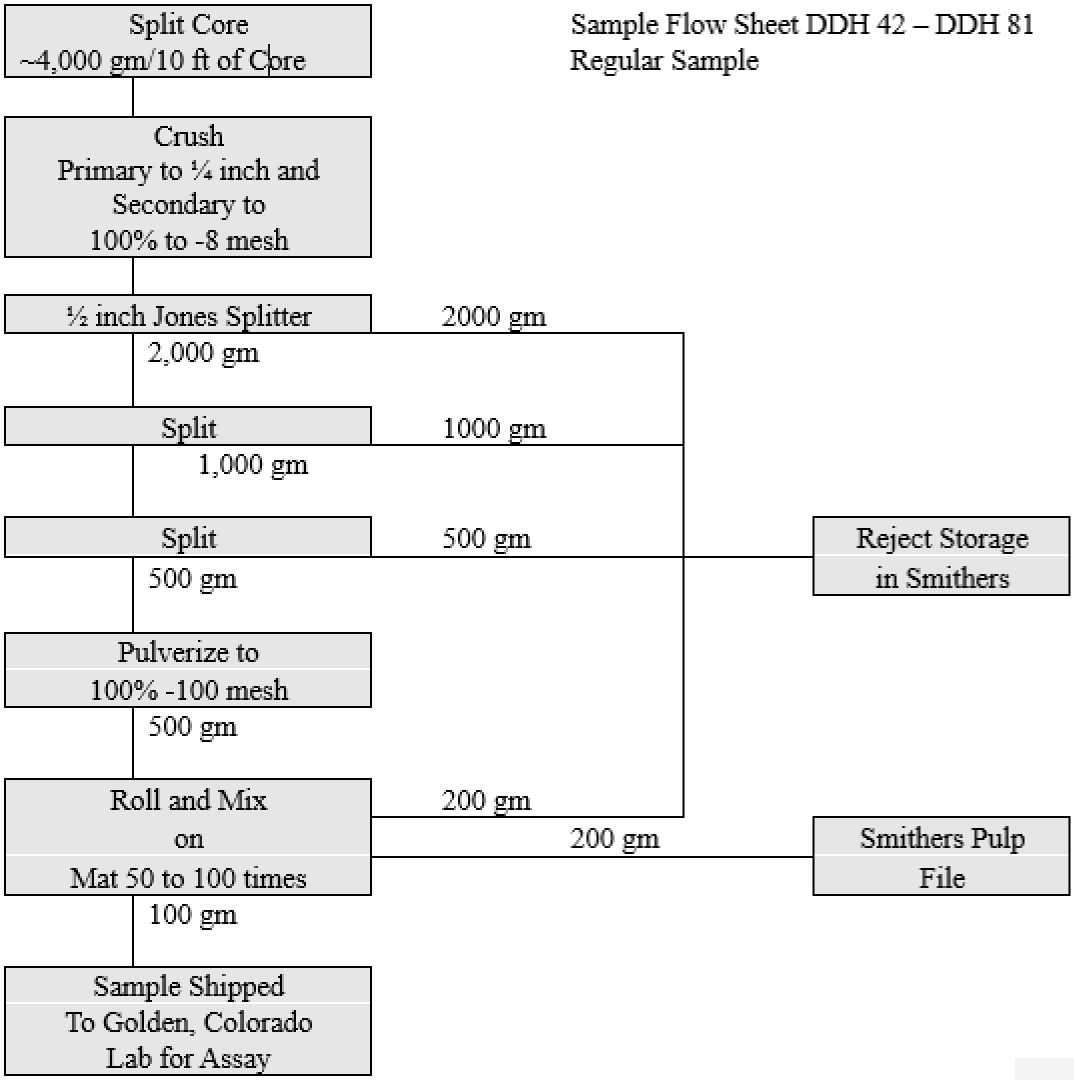


Figure 11.1. Sampling Protocol for Diamond Drill Holes 42 to 81

The following is an excerpt from Giroux and Cuttle (2016) that summarises the sampling procedure for drill holes 82 to 141:

> *For drill holes 82 through 141 the primary and secondary crush of drill core was changed to 100 % passing -6 mesh. The sample was split using a Jones Splitter down to a 1000 gm sample that was then pulverized to 100 % passing -35 mesh as compared to the previous procedure of pulverizing to 100 % passing -100 mesh. It was hoped the coarser grind would improve the MoS2 reproducibility.*
>
> *A study of sampling of Yorke-Hardy drill core by Carson and Pitard (Carson, et.al., 1979) concluded the following:*
>
>> *Due to the very unusual mineral distribution at Yorke-Hardy sampling errors, not normally encountered in core drilling, appear to have been significant. Analysis has shown that there are probably low sources of*

> *error. The core size drilled may not have been large enough to ensure that an accurate sample of the surrounding rock was taken. Again, due to the large nuggets of molybdenite mineralization, reduction, and sub-sampling procedures probably introduced additional errors into the data.*

Carson and Pitard recommended that subsequent drilling be completed with HQ size core and that the entire core be crushed for sampling.

> *The use of whole core is only recommended because the need for a detailed sampling study may outweigh the need for additional core for future study.*

For the drilling (1979), of holes from 145 to 164, the following protocol was implemented. A primary and secondary crush took entire 10-foot section of drill core to 100 % passing -10 mesh. A total of 5 blending passes were made before a final 500 gram split was sent to the pulveriser. The remaining crushed sample was saved as reject. The 500 gram split was pulverized to 100 % passing -20 mesh and again 3 blending passes were made to homogenize the sample. This pulp was then split in half with 250 gm stored in Smithers as a pulp. The remaining 250 grams was split in half with 125 grams sent to Golden Colorado for assay and the remaining 125 grams either assayed as a check sample or combined with the rejects and saved.

11.1.3 Sample Analysis (Climax/AMAX)

Extensive amounts of historic documents were available for review at the Smithers facility and were reviewed by Mr. Salmabadi and shared with the authors. The following from Giroux and Cuttle (2016), in italics, is an agreement with what the authors and Mr. Salmabadi observed.

Initial AMAX drilling was assayed by Coast Eldridge Assayers in Vancouver. From 1966 on, all sample preparation of drill core was handled in a professional manner by Climax trained geologists. Samples were shipped to the Climax Molybdenum Assay Laboratory in Golden, Colorado. MoS_2 and WO_3 assays were done by Spectrographic analysis and colorimetric although there is no record of which process was used for which samples. While the majority of these assays were completed before ISO Standards were developed the Climax Molybdenum Assay Laboratory of Amax was a world leader in molybdenum assays handling samples from both Climax and Henderson mines and therefore there is no reason to believe the results would not have conformed to current standards. No standards or blank results could be found for holes 1 to 72. A report by D. Davidson of Climax Molybdenum Co. (Davidson, 1972) states that no standards were run with assays until DDH 73. Climax prepared internal standards from a weighted mixture of Ottawa sandstone and lubrication grade molybdenite. Standards were prepared in the grades of 0.05, 0.3, 0.9 and 1.5% MoS_2. The standards along with 20 drill core samples were sent to four umpire laboratories, Skyline Labs, Denver, Union Assay Office, Salt Lake, Acme Analytical Laboratories Burnaby, and Loring Laboratories Calgary. The prepared standards were then added to the regular sample stream. Davidson states that results from standards from hole 73 to 81 suggest that regular assays may be 5 to 10 % low for both MoS_2 and WO_3.

The line in the above section from Giroux and Cuttle suggests a small degree of underreporting bias may be present in the Climax/AMAX assays. However, the author is comfortable with using the assays, as reported, due to the apparent unlikeliness of the assays being overestimated and the minimal effect that they may have on the overall Resource.

11.2 SAMPLING PROCEDURES – 2006 TO 2007 DRILLING (BLUE PEARL)

Sampling protocol for hole 165 through 196 was located in an internal report by Snowden Mining Industry Consultants (2008), which was part of a feasibility study report by Hatch Ltd. (May 2, 2008) for Blue Pearl Mining Ltd. Sampling protocols for 197 to 219 were likely very similar, if not the same, to the previous Blue Pearl sampling procedure.

11.2.1 Sample Security (Blue Pearl)

The core was delivered to Blue Pearl's core logging facility by the drill contractor at the end of each shift. The core logging facility was located near the airport on Highway 16, just north of Smithers. The core was logged, marked, and tagged for sections to be manually split. One-half of the core is retained for reference onsite at the logging site and the other half was placed in plastic bags, numbered, and closed with zip ties and sent directly, by commercial trucking, to Acme Analytical Labs (Acme) of Vancouver, British Columbia.

11.2.2 Drill Core Sample Laboratory Preparation (Blue Pearl)

The drill core samples were prepared by Acme using preparation Code R150. This was done by crushing the sample so 70% passes a 10 mesh (-1.68 mm) sieve and a 250-gram split was then pulverised to 95% passing 150 mesh (-0.105 mm). The reject and pulps were then returned after analysis to Blue Pearl's core logging facility in Smithers.

11.2.3 Sample Analysis (Blue Pearl)

One gram of the pulverised -150 mesh material from each core sample was then digested in Aqua Regia (1-part nitric acid to 3 parts hydrochloric acid) and analysed by ICP-ES (Inductively Coupled Plasma-Emission Spectrometry), Acme Code 7AR. Results show a variety of over 25 elements including total molybdenum and tungsten. A conversion factor of 1.6681 is used for % Mo to % MoS_2.

11.2.4 Quality Assurance/Quality Control (QA/QC) (Blue Pearl)

From Snowden, 2008:

A description of the Blue Pearl's protocol for field blanks and certified standards used for holes 165 through 196 is best described by Snowden, 2008. This description is referenced below:

> *Field blank samples facilitate an external check on potential inter-sample contamination during all sample preparation and handling procedures in the lead-up to analysis. Field certified standards allow for an external check on the analytical accuracy of the laboratory. Field blanks and standards were inserted alternatively into the sample stream as part of BPM's Quality Assurance and Quality Control (QAQC)protocol to yield a field control sample frequency of approximately 1:20. Field blanks were obtained from non-mineralized porphyritic andesite of the Kasalka Group. These were obtained from a small quarry on*

Highway 16 near Boulder Creek, located at UTM 603380E, 6107700N. A total of 90 field blank samles were submitted as part of the recent BPM drilling program. The Canmet MP-2 field standard, made from a tungsten-molybdenum ore body in New Brunswick and certified as containing 0.281 ± 0.01% Mo at 95% confidence level, was used initially. Thereafter, for drill holes DDH178 and DDH190 to DDH196, field standards prepared using rock from the Davidson deposit was used. These field standards, designated as BLE-1, BLE-2 and BLE-3, were prepared by CDN Resource Laboratories Ltd. (Vancouver, B.C.) and certified by round-robin assaying at the ACME and ALS Chemex laboratories in Vancouver and by Florin Analytical services, LLC in Reno, Nevada.

According to Mr. Davidson, during the 2007 drilling of holes 197 through 219, similar QA/QC materials and methods were used and followed under the supervision of Mr. Hutter. This included the regular insert of “Kasalka” blanks and field standards BLE-1, 2, and 3 into every batch of samples shipped to Acme; with a frequency of every 20 samples per 1 certified standard and blank.

11.2.5 Field Duplicates (Blue Pearl)

From Giroux and Cuttle 2016:

The 2006 program of field duplicates is best described by Snowden, 2008.

Field duplicates are obtained by splitting half core samples into two quarter core sub-samples, one quarter stored as a representative of the original sample and the other representing the duplicate sample. These samples are collected to assess the mineralization homogeneity and sampling precision. Field duplicate samples were inserted at a frequency of approximately 1:20 for drill holes DDH175 to DDH178vand DDH185 to DDH196. No field duplicate samples were collected for drill holes DDH165 to DDH174, DDH179, and DDH181 to DDH184. A total of 92 field duplicates were analyzed as part of BPM’s recent drilling program.

It is not clear to the authors if a similar protocol for field duplicates was continued for holes 197 through 219 during the 2007-2008 drill programs.

11.2.6 Laboratory Standards and Blanks (Blue Pearl)

From Snowden, 2008:

Field blank samples facilitate an external check on potential inter-sample contamination during all sample preparation and handling procedures in the lead-up to analysis. Field certified standards allow for an external check on the analytical accuracy of the laboratory. Field blanks and standards were inserted alternatively into the sample stream as part of BPM’s Quality Assurance and Quality Control (QAQC) protocol to yield a field control sample frequency of approximately 1:20.

Field blanks were obtained from non-mineralized porphyritic andesite of the Kasalka Group. These were obtained from a small quarry on Highway 16 near Boulder Creek, located at UTM 603380E, 6107700N. A total of 90 field blank samples were submitted as part of the recent BPM drilling program.

The Canmet MP-2 field standard, made from a tungsten-molybdenum orebody in New Brunswick and certified as containing 0.281 ± 0.01% Mo at 95% confidence level, was used initially. Thereafter, for drillholes DDH178 and DDH190 to DDH196, field standards prepared using rock from the Davidson deposit were used. These field standards, designated as BLE-1, BLE-2 and BLE-3, were prepared by CDN Resource Laboratories Ltd. (Vancouver, B.C.) and certified by round-robin assaying at the ACME and ALS Chemex laboratories in Vancouver and by Florin Analytical services, LLC in Reno, Nevada. Details of these field standards are presented in Table 11-1.

Table 11-1: Blue Pearl Field Standard Details

Field Standard	Average Mo Grade (%)	Standard Deviation (% Mo)
BLE-1	0.129	0.007
BLE-2	0.254	0.004
BLE-3	0.369	0.013

A total of 61 Canmet MP-2 standards were inserted into the sample stream for drillholes DDH165 to DDH177 and DDH181 to DDH190. A total of 24 BPM standards (eight BLE-1, seven BLE-2 and nine BLE-3 standard samples) were inserted into the sample stream for drillholes DDH178 and DDH191 to DDH196.

Acme also inserted standards and blanks into each batch of samples received. The standards and blanks were assayed after every 30th sample. No issues with laboratory were identified.

11.2.7 Reject and Pulp Duplicates (Blue Pearl)

From Giroux and Cuttle (2016):

As part of their Quality Management System Acme routinely analyses reject duplicates. During the 2006 drill program on the Davidson Project there were a total of 113 duplicate reject samples analyzed (Snowden, 2008). The authors could not verify results of this program or whether or not a similar program continued during the 2007-8 drill programs.

As part of their Quality Management System Acme routinely analyses splits of the original pulp material as a duplicate. During the 2006 drill program on the Davidson Project there were a total of 117 duplicate pulp samples analyzed (Snowden, 2008).

The authors could not verify results of this program or whether or not a similar program continued during the 2007-8 drill programs.

Snowden (2008) suggests that Blue Pearl had Acme analyse 295 pulps for molybdenum and tungsten from the 2006 drill program using the 7KP method. This analytical process requires a phosphoric acid digest with an ICP-ES finish.

The results of this geochemical check survey were not available to the authors at the time of the 2016 property visit.

11.2.8 Assay Checks by Secondary Laboratory (Blue Pearl)

From Giroux and Cuttle (2016):

> *Blue Pearl submitted approximately 5% of pulps and rejects (216 samples) from the 2006 drill program to ALS Chemex to check for assay accuracy between laboratories. This included material from Blue Pearl's field standards and blanks (Giroux and Cuttle, 2016 cited from Snowden 2008).*
>
> *Results of the check assays during the 2007 programs were not available to the authors at the time of the 2023 property visit.*

11.3 OPINION

In the author's opinion, while the some of the information regarding sample preparation and QA/QC was not available during the preparation of this report, the sample preparation, analyses, and QA/QC measures conducted and described in this section are sufficient to determine that the sample assays in the database for the Davidson Project are suitable for use in the Resource estimate described in this report. Any bias that may have occurred due to clumping or clotting of soft molybdenite on screens would have a conservative influence on Resource estimation.

12.0 DATA VERIFICATION

12.1 VERIFICATION OF DATABASE

A Property visit was conducted by Mr. Ehsan Salmabadi, P.Geo., on behalf of AMPL, on June 17, 18, and 19, 2023 at which time drill hole pulps were sampled, split core samples taken, and diamond drill records scanned and other information regarding QA/QC and the Property was collected from Mr. Donald Davidson, the owner of the claims. Video conferencing was used throughout the site visit with the authors to review data available at the Smithers facility. Mr. Salmabadi reviewed drill core and confirmed historic paper documents used and cited in previous reports. The Davidson assay data that Giroux and Cuttle used for the previous Resource estimate was compared against the originals recorded on handwritten assay certificates. Much of the old analytical equipment is still at the Smithers facility where all the core, pulps, and historic documents are kept. Verdstone Gold Corporation converted Climax/AMAX data to digital format in 1998 and these files were located at the Smithers facility on a series of 3.5-inch floppy discs that were still intact and functioning. The authors were able to access this data and review it in preparation for this report.

Drill holes completed by Blue Pearl (DDH 165 to 219) were stored in core boxes located outside the main warehouse at the Smithers facility, where assay tags in the core boxes corresponded directly to assay sample numbers recorded on the core logs (see Figure 12.1 and Figure 12.2, below). Sample pulps and rejects from the Blue Pearl drill program were stored beside the core boxes outside and sample pulps from preceding drill programs were stored in the main warehouse (see Figure 12.3,below). Specific collars to drill holes were not located in the field as they were collared underground and are currently inaccessible due to the portal being closed off. Other historical collars to holes drilled from the surface on the Hudson Bay glacier have since disappeared according to Mr. Davidson (see Figure 12.4, below).

Figure 12.1. External Storage (Blue Pearl)
Source: AMPL, 2023

Figure 12.2. Internal (Climax/AMAX) Core Storage
Source: AMPL, 2023

Figure 12.3. View of Some NQ Split Core (DDH 189) from Blue Pearl Era Drilling; Some Samples Were Taken to Verify Grade
Source: AMPL, 2023

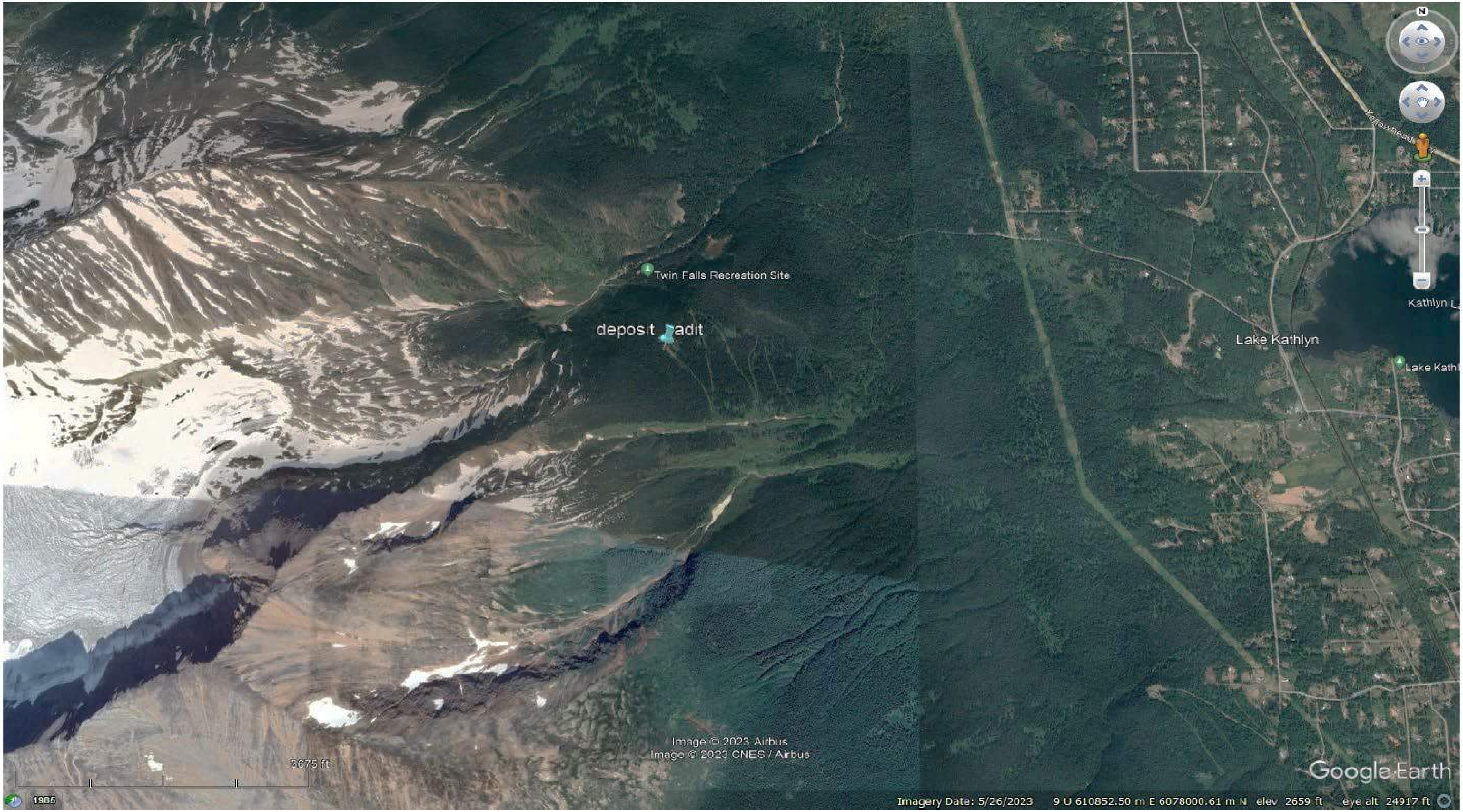


Figure 12.4. ***Google Earth™ View of the Property***
Source: AMPL, 2023

During Mr. Salmabadi's Property visit, molybdenite (MoS_2) was visually identified in drill core and was occasionally accompanied with scheelite ($CaWO_4$), visible under ultraviolet light as a blue-white fluorescing mineral. Scheelite was also observed to fluoresce a greenish-yellow depending on its molybdenum content, whereby molybdenum can replace tungsten in a scheelite crystal lattice.

A well-organised library of drill core, hard copy maps, reports, assay certificates, and 3D plexiglass models of the Davidson molybdenum deposit was also confirmed.

A visit to the underground workings was not possible due to the adit being inaccessible but the core lab and library was readily available.

However, even from Google Earth™, the portal is clearly visible (see Figure 12.5 and Figure 12.6, below). In addition, Mr. Kelly Grebliunas with AScT (listed as one of the experts) undertook the actual surveying of the underground portion of the drifts, converting all data to NAD83, re-surveying old diamond drill holes, and surveying and lining up diamond drills during the 2006 Diamond Drill Program.

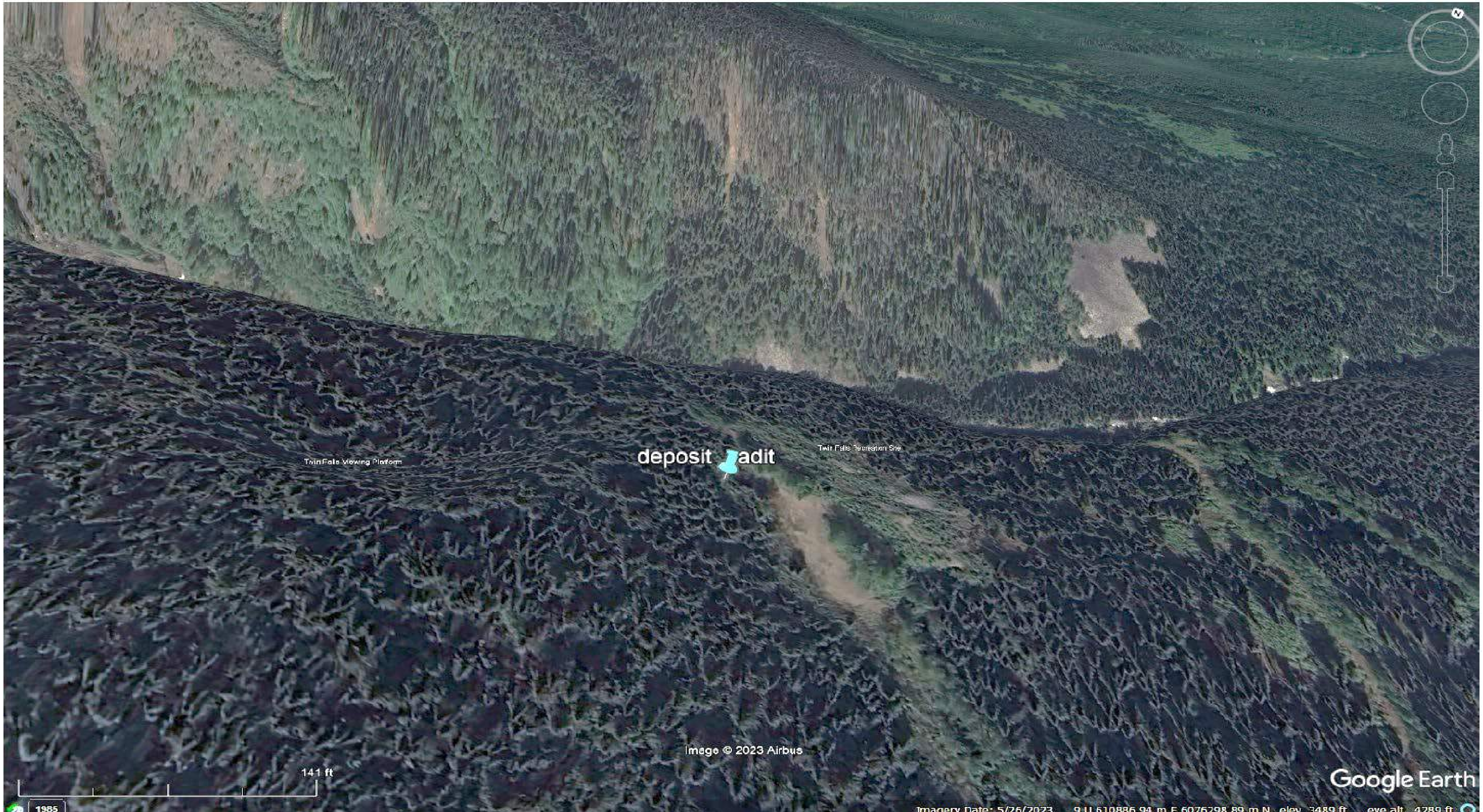


Figure 12.5. Google Earth™ of the Portal
Source: AMPL, 2023

Photo 3 Davidson Portal. No access (utm83 610900E, 6076240N) - Cuttle, 2016

Figure 12.6. View of Portal from 2016 NI 43-101 Report
Source: AMPL, 2023

12.2 HISTORICAL DATA VERIFICATION

(Text in italics is from Giroux and Cuttle, 2016)

12.2.1 Giroux and Cuttle 2004 Data Verification

To verify the drill hole assay data base in 2004, original assay sheets were taken by Giroux at random from the records kept at Smithers and photocopied. A line-by-line verification process was then completed to look for data entry errors in the supplied data base. A total of 2,736 lines of data, which represents 15% of the total data base, were checked with 26 typos found for MoS_2 assays or 0.95% and 12 typos in WO_3 assays or 0.44 %. Most errors were mixing 2 and 7 or 3 and 8 in reading the hand-written sheets and none were considered significant. The errors were corrected, and the frequency of errors was acceptable for this kind of data base.

12.2.2 2004 Duplicate Sample Checks

Several sets of duplicate data were found in the Yorke-Hardy files. The results from duplicate sampling campaigns were entered onto a computer and analyzed. During the drilling of holes 42 to 81 about one in ten samples were taken in duplicate and checked against the original MoS_2 value. The results are shown in Figure 6. The scatter plot shows the original MoS_2 value on the x axis and the duplicate sample on the y axis. Results are reasonable with the best fit regression line through the samples shown slightly above the equal value line indicating slight proportional bias with the duplicates higher than the originals. The Correlation Coefficient is a reasonable 0.9645 and the sampling precision can be calculated at ± 63%. There are several outliers that reflect the nugget effect of mineral clotting on screens in one sample or the other.

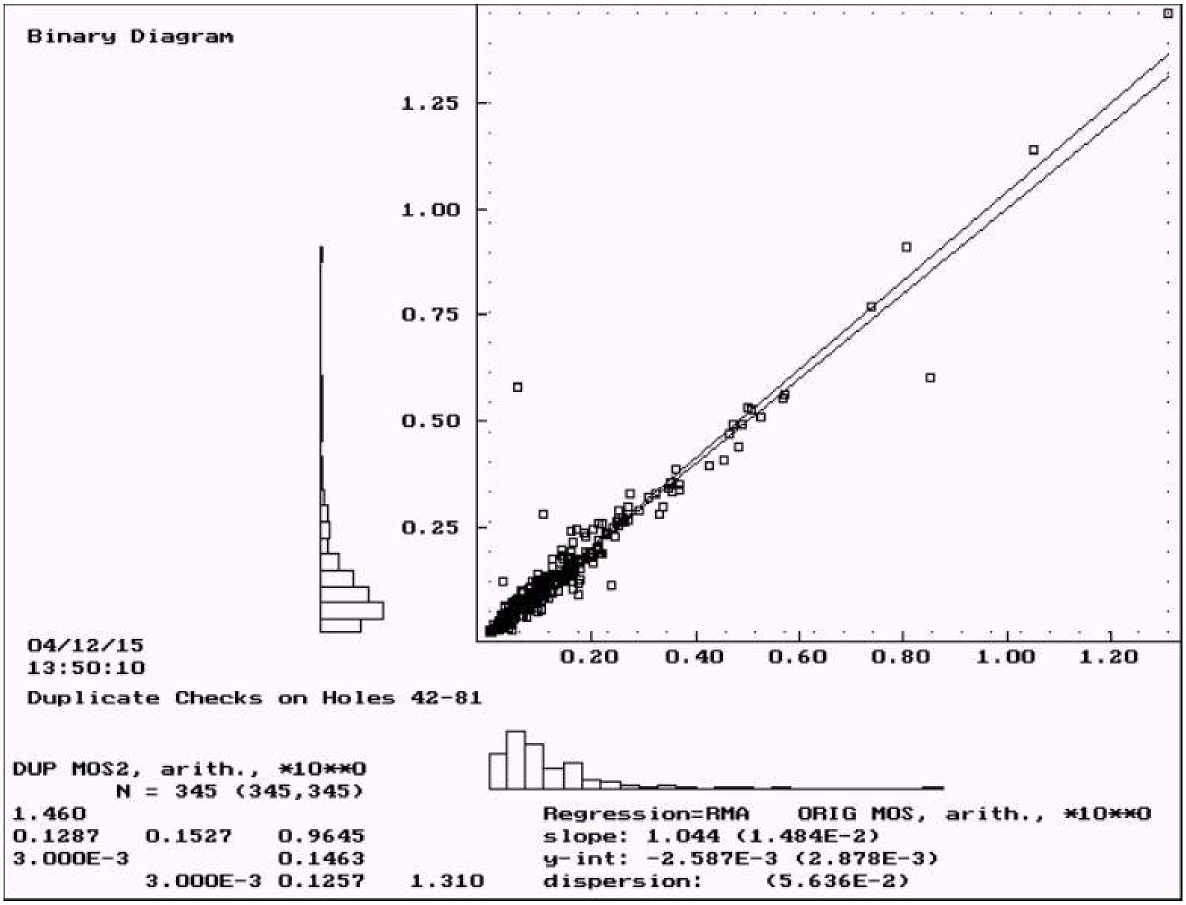


Figure 6. Scatter plot for Original MoS2 versus Duplicate Sample from Holes 42 to 81

During this same time span a system of rolling the pulverized material from 50 to 100 times, prior to analysis, was implemented. To test the effectiveness of this rolling check samples were taken and compared rolled to unrolled. Figure 10 shows a scatter plot with the rolled sample on the x axis and the unrolled sample on the y axis. The best fit regression line is pulled below the equal value line by several outliers. These outliers also bring down the correlation coefficient to 0.8840 and reduce the average sampling precision to ± 116 %.

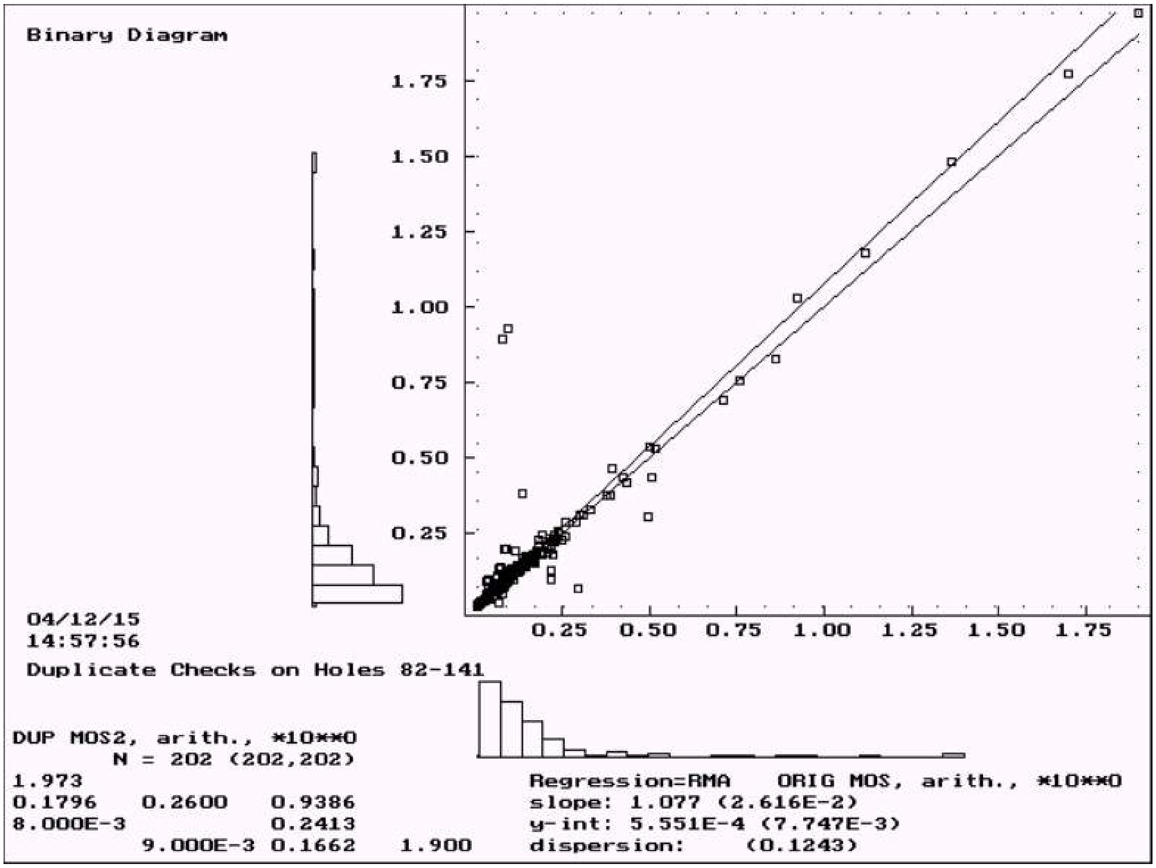


Figure 10. Scatter plot for Original MoS_2 versus Duplicate Sample from Holes 82 to 117

A second set of duplicates was available for the same original samples described above from holes 82 to 117 and a scatter plot comparing the original with the second check sample is shown below as Figure13. The correlation is excellent with a coefficient correlation of 0.991 and the best fit regression line superimposed on the equal value line. The average sampling precision is ± 39% for this data set.

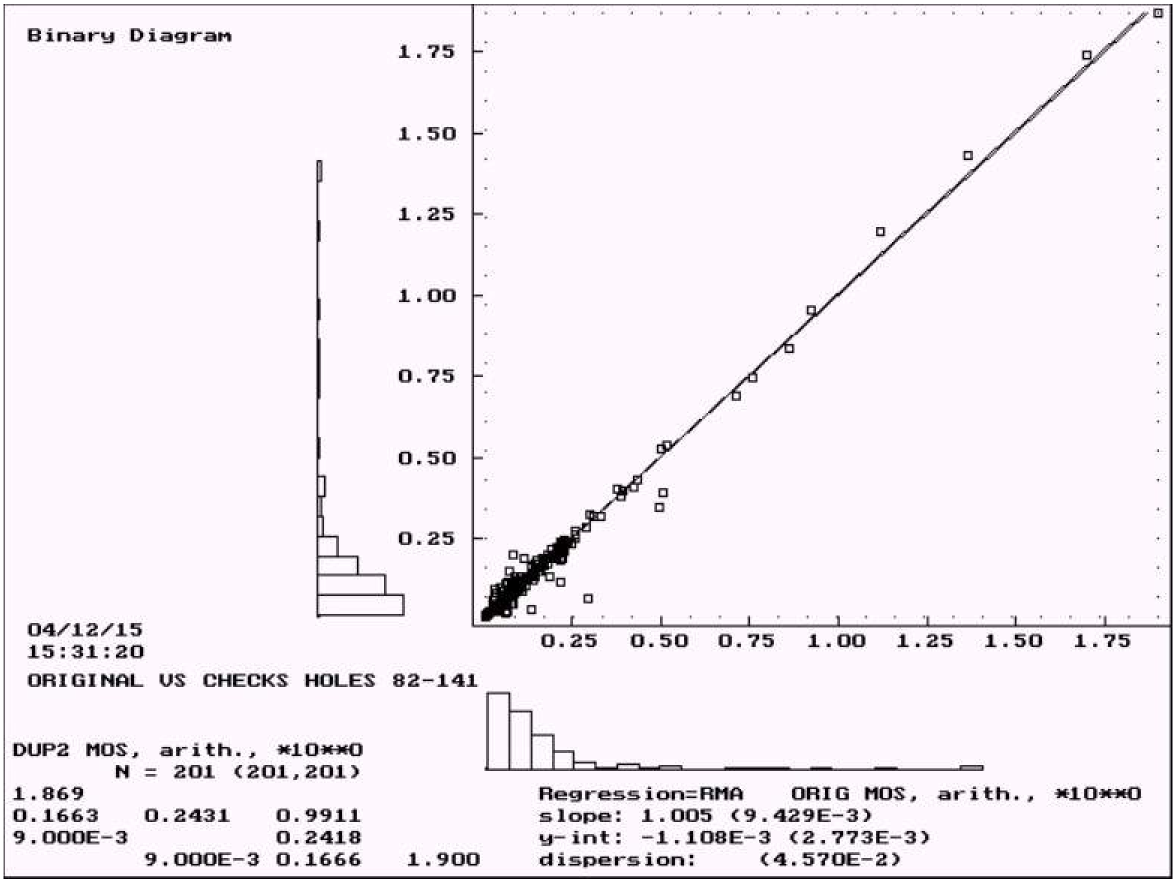


Figure 13. Scatter plot for Original MoS_2 versus 2nd duplicate Sample from holes 82 to 117

In conclusion, the sampling of MoS_2 at the Yorke-Hardy property has been somewhat problematic, and much work and study has been completed by Climax staff to address the problem. The nature of the MoS_2 mineralization, occurring in clots and coarse patches within veins and stockworks has led to clumping and small balls of mineralization occasionally sitting on screens after pulverization. When this happens, the grade reported is obviously lower than it should be. The database as presented is probably conservative in grades and is adequate for a resource estimate.

12.2.3 Bulk Sample

In 1971 two bulk sample raises were driven following drill holes 81 on section 17,600 N and 82 – 82A on section 17,800 N. The procedure was to centre the raise on the drill hole and recover each round in bins under the raise. These bins were loaded into 3 ton crates underground, sealed and shipped to Climax's Laboratory in Golden, Colorado. After each round the raise and bins were washed down. At the lab each round was put through a pilot plant with the recovered grade of MoS_2 reported below in Tables 7 to 10. The results are reasonable, with the first test around drill hole 81 showing a higher average grade from the bulk sample than indicated by drilling (0.312 compared to 0.292 % MoS_2 from drilling). The second test between holes 82 and 82A showed slightly lower grades in the bulk sample (.303 % MoS_2 from the bulk sample compared to 0.349 and 0.323 % MoS_2 from drill holes 82 and 82A respectively). Combining the two tests gives an overall average from drill holes of 0.315 % MoS_2 compared to 0.308 % MoS_2 from the bulk samples.

When the bulk sample is compared round by round with the comparable drill hole assay a wide scatter in grades is observed. Considering the sampling problems encountered with drill hole assays, however, this test shows that overall, the drill holes' average grades are similar to those obtained from a pilot mill test of a large bulk sample.

Hole	From	To	Original Core MoS2 (%)	Sludge MoS2 (%)	Bulk Sample MoS2 (%)
81	0	10	0.093		0.593
81	10	20	0.775	0.310	0.195
81	20	30	0.043	0.056	0.136
81	30	40	0.179	0.060	0.224
81	40	50	0.175	0.250	0.236
81	50	60	0.079	0.155	0.256
81	60	70	0.280	0.095	0.254
81	70	80	0.615	0.583	0.137
81	80	90	0.202	0.369	0.262
81	90	100	0.158	0.149	0.212
81	100	110	0.318	0.276	0.232
81	110	120	0.539	0.606	0.245
81	120	130	0.248	0.258	1.270
81	130	140	0.155	0.185	0.220
81	140	150	0.534	0.559	0.208
		Average	**0.293**	**0.279**	**0.312**

Table 6 MoS2 Bulk Sample from raise around Drill Hole 81

Hole	From	To	Original Core WO3 (%)	Sludge WO3 (%)	Bulk Sample WO3 (%)
81	0	10	0.020		0.074
81	10	20	0.018	0.030	0.063
81	20	30	0.025	0.025	0.032
81	30	40	0.017	0.031	0.028
81	40	50	0.031	0.042	0.043
81	50	60	0.026	0.019	0.046
81	60	70	0.024	0.027	0.061
81	70	80	0.035	0.030	0.042
81	80	90	0.027	0.030	0.040
81	90	100	0.030	0.036	0.035
81	100	110	0.032	0.030	0.048
81	110	120	0.034	0.037	0.051
81	120	130	0.042	0.036	0.067
81	130	140	0.083	0.069	0.046
81	140	150	0.014	0.023	0.037
		Average	0.031	0.033	0.048

Table 7 WO3 Bulk Sample from raise around Drill Hole 81

Hole	From	To	DDH 82 Original Core MoS2 (%)	DDH 82A Original Core MoS2 (%)	DDH 82A Sludge MoS2 (%)	Bulk Sample MoS2 (%)
82 and 82A	0	10	0.199	0.977	0.259	0.282
82 and 82A	10	20	1.186	0.116	0.214	0.537
82 and 82A	20	30	0.198	0.182	0.279	0.114
82 and 82A	30	40	0.509	0.201	0.242	0.126
82 and 82A	40	50	0.121	0.271	0.462	0.165
82 and 82A	50	60	0.241	0.323	0.508	0.106
82 and 82A	60	70	0.204	0.245	0.300	0.347
82 and 82A	70	80	0.130	0.252	0.150	0.325
82 and 82A	80	90	0.379	0.085	0.098	0.316
82 and 82A	90	100	0.126	0.261	0.281	0.198
82 and 82A	100	110	0.200	0.198	0.243	0.390
82 and 82A	110	120	0.307	0.159	0.120	0.681
82 and 82A	120	130	0.739	0.927	1.012	0.348
		Average	**0.349**	**0.323**	**0.321**	**0.303**

Table 8 MoS2 Bulk Sample from raise around Drill holes 82 and 82A

Hole	From	To	DDH 82 Original Core WO3 (%)	DDH 82A Original Core WO3 (%)	DDH 82A Sludge WO3 (%)	Bulk Sample WO3 (%)
82 and 82A	0	10	0.058	0.062	0.061	0.022
82 and 82A	10	20	0.086	0.075	0.088	0.029
82 and 82A	20	30	0.081	0.050	0.028	0.023
82 and 82A	30	40	0.033	0.033	0.029	0.020
82 and 82A	40	50	0.028	0.037	0.039	0.027
82 and 82A	50	60	0.078	0.035	0.038	0.029
82 and 82A	60	70	0.060	0.073	0.077	0.042
82 and 82A	70	80	0.041	0.093	0.068	0.043
82 and 82A	80	90	0.033	0.033	0.036	0.032
82 and 82A	90	100	0.025	0.034	0.039	0.031
82 and 82A	100	110	0.051	0.040	0.039	0.042
82 and 82A	110	120	0.034	0.059	0.054	0.053
82 and 82A	120	130	0.017	0.087	0.097	0.045
		Average	**0.048**	**0.055**	**0.053**	**0.034**

Table 9 WO3 Bulk Sample from Raise around Drill Holes 82 and 82A

12.3 CURRENT (2023) DATA VERIFICATION

The authors verified the Project data from original sources to the degree possible. For the work done by Climax/AMAX (1958 to 1980), there is a large amount of original data available in the geological office and this was used to verify the assay data now in the Project database to the degree possible during the site visit. In terms of the chain of custody, all the digital data in the authors' possession came from Cuttle as Microsoft Excel™ files and were checked with the files found onsite. Checks were performed on a significant number of assays, as is described in following sections.

For data from work by Blue Pearl, the original sources were not found and only a paper print out of the assays and QA/QC were found. The data was digitised and used to check the Microsoft Excel™ files provided by Cuttle.

12.3.1 Climax/AMAX Assays

The Climax/AMAX era molybdenum assays were audited by checking them against primary sources. The original assay sheets were taken at random from the records kept at Smithers, a line-by-line verification process was then completed to check for data entry errors in the Microsoft Excel™ file provided by Cuttle (personal communication, June 3, 2023). A total of 961 lines of data were checked and 2 typos were found for MoS_2 assays. The results were found to be acceptable by the authors.

12.3.2 Blue Pearl Assays

The authors checked Blue Pearl's molybdenum assays against those reported in the Microsoft Excel™ files provided by Cuttle (personal communication, June 3, 2023). To verify the drill hole assay database, scanned and original copies of assay sheets and diamond drill logs were taken by Mr. Salmabadi and the entire series of diamond drill holes from 165 to 219 were entered into spreadsheets so a comparison of data could be made. A total of 546 lines of data were checked and 1 typo was found for MoS_2. The results were examined by the authors and were found to be acceptable. However, it was noted that many of the assays initially received by AMPL and deemed "original" were in fact 10-ft composites of 5-ft intervals. The rationale behind this is not known but when individual assays were "re-composited" by the authors, the results were identical on all samples that were checked.

12.3.3 Assay Table

The author audited the molybdenum assays reported in the Project assay table by checking them against original or near-original sources to the extent such sources were available. Table 12.1, below, summarises the numbers of checks that the authors were able to do, by project operator.

Table 12.2, below, shows the core and pulp check samples taken in June 2023 by Mr. Salmabadi. **Note:** Samples with N/A for WO_3 were either never assayed for WO_3 or were assayed as five contiguous sample composites that were not captured in the sample selection.

TABLE 12.1
MOLYBDENUM ASSAYS OF THE SPLIT-CORE SAMPLES

Drill Hole	From (ft)	To (ft)	From (m)	To (m)	Original Sample ID	Check Assay ID	Original MoS_2 %	Check Assay MoS_2 %	Original WO_3 %	Check Assay WO_3 %
DDH 189	15	20	4.57	6.10	332544	3835557	1.621	2.072	0.223	0.277
DDH 189	20	25	6.10	7.62	332545	3835558	3.837	4.469	0.358	0.398
DDH 189	25	30	7.62	9.14	332546	3835559	1.793	1.783	0.392	0.382
DDH 169	1,070	1,080	326.14	329.18	324646	3835560	0.018	0.027	0.004	0.013
DDH 169	1,080	1,090	332.23	331.93	324647	3835561	0.015	0.023	0.008	0.009

TABLE 12.2
MOLYBDENUM ASSAYS OF THE CORE SAMPLE PULP

Drill Hole	From (ft)	To (ft)	From (m)	To (m)	Original Sample ID	Check Assay ID	Original MoS_2 %	Check Assay MoS_2 %	Original WO_3 %	Check Assay WO_3 %
DDH 60	700	710	213.50	216.55	B230	3835562	0.023	0.030	N/A	0.019
DDH 60	710	720	216.55	219.60	B231	3835563	1.530	1.643	N/A	0.033
DDH 60	1,200	1,210	366.00	369.05	B280	3835564	0.703	0.706	N/A	0.026
DDH 60	1,210	1,220	369.05	372.10	B281	3835565	0.068	0.077	N/A	0.036
DDH 83	460	470	140.30	143.35	E472	3835566	2.17	2.125	0.014	0.089
DDH 83	490	500	149.45	152.50	E475	3835567	0.150	0.163	0.012	0.013
DDH 83	600	610	183.00	186.05	E486	3835568	0.242	0.284	0.057	0.051
				CDN-BL-10	3835569					
DDH 83	740	750	225.70	228.75	E500	3835570	0.811	0.868	0.028	0.027
DDH 84	40	50	12.20	15.25	E520	3835571	0.079	0.080	0.027	0.028
DDH 84	50	60	15.25	18.30	E521	3835572	1.88	1.926	0.072	0.069
DDH 84	60	70	18.30	21.35	E522	3835573	0.103	0.110	0.047	0.047
DDH 84	240	250	73.20	76.25	E540	3835574	0.129	0.145	0.019	0.045
DDH 84	270	280	82.35	85.s40	E543	3835575	0.124	0.147	0.038	0.035
DDH 84	280	290	85.40	88.45	E544	3835576	0.303	0.313	0.004	0.049
DDH 84	290	300	88.45	91.50	E545	3835577	0.231	0.258	0.015	0.021

TABLE 12.2 MOLYBDENUM ASSAYS OF THE CORE SAMPLE PULP										
Drill Hole	**From (ft)**	**To (ft)**	**From (m)**	**To (m)**	**Original Sample ID**	**Check Assay ID**	**Original MoS_2 %**	**Check Assay MoS_2 %**	**Original WO_3 %**	**Check Assay WO_3 %**
DDH 84	420	430	128.10	131.15	E558	3835578	0.553	0.563	0.357	0.400
DDH 84	430	440	131.15	134.20	E559	3835579	0.911	0.934	0.479	0.603
				CDN-MoS-1	3835580					
DDH 84	570	580	173.85	176.90	E573	3835581	0.045	0.042	0.024	0.025
DDH 84	740	750	225.70	228.75	E590	3835582	0.17	0.176	0.016	0.013
DDH 84	750	760	228.75	231.80	E591	3835583	0.42	0.449	0.091	0.089
DDH 58	220	230	67.10	70.15	G613	3835584	0.732	0.719	N/A	0.016
DDH 58	230	240	70.15	73.20	G614	3835585	0.171	0.159	N/A	0.025
DDH 58	360	370	109.80	112.85	G627	3835586	0.558	0.605	N/A	0.083
DDH 58	370	380	112.85	115.90	G628	3835587	1.190	1.180	N/A	0.029
DDH 58	380	390	115.90	118.95	G629	3835588	0.085	0.070	N/A	0.021
DDH 70	30	40	9.15	12.20	M876	3835589	0.722	0.731	N/A	0.019
DDH 70	40	50	12.20	15.25	M877	3835590	0.635	0.575	N/A	0.039
DDH 70	210	220	64.05	67.10	M894	3835591	0.066	0.078	N/A	0.056
DDH 70	220	230	67.10	70.15	M893	3835592	0.930	0.924	N/A	0.048
				CDN-W-4	3835593					
DDH 165	10	20	3.05	6.10	324052	3835594	0.178	0.194	0.015	0.020
DDH 165	20	30	6.10	9.15	324053	3835595	0.065	0.075	0.008	0.012
DDH 165	50	60	15.25	18.30	324056	3835596	0.018	0.020	0.008	0.010
DDH 165	70	80	21.35	24.40	324058	3835597	0.195	0.210	0.004	0.006

12.3.4 Re-sample of Pulps and Core

During the 2023 site visit by Mr. Salmabadi, a selection of pulps was collected from the Smithers storage facility and taken to Vancouver for analysis at Bureau Veritas. A total of 41 samples was taken including 32 pulp samples from Climax era drilling, 4 pulps from Blue Pearl drilling, and 5 split core samples from Blue Pearl drill core shown in Table 8, above. Bureau Veritas re-pulverised the pulps to homogenise the sample with 85% < 75 µm. Molybdenum and tungsten was assayed using a four-acid digestion – ICP-ES/ICP-MS analysis. Results for molybdenum in ppm were converted to percent and then to MoS_2 by dividing by 0.5994. Molybdenum makes up 59.94% of molybdenite (MoS_2).

The results are shown as a scatter plot (see Figure 12.7 and Figure 12.8, below). The molybdenum database is valid and adequate for the estimation of a Mineral Resource. However, in the author's opinion, the tungsten assay data is not adequate for a Mineral Resource estimate due to inconsistent assaying where some holes were not assayed, limited QA/QC documentation from Climax and AMAX era drilling, and a lack of available documentation on the metallurgical recovery of tungsten at such low grades and the ability to make a saleable concentrate.

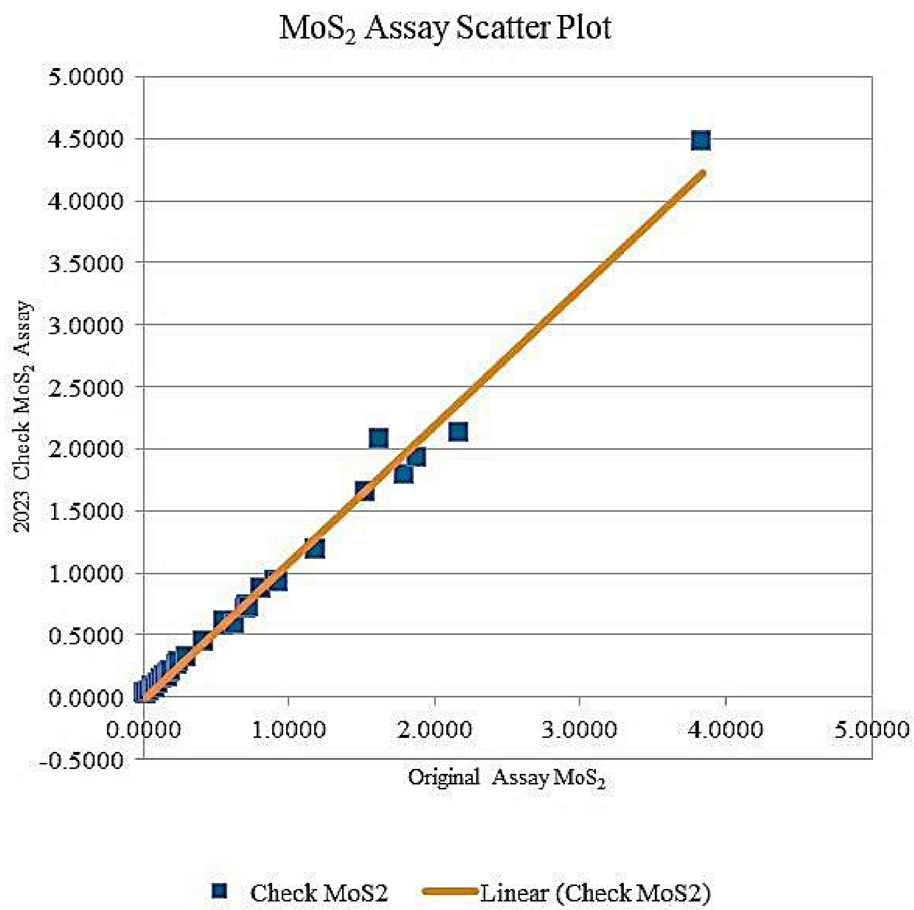


Figure 12.7. Scatter Plot Showing Original MoS_2 (x axis) versus 2023 Check MoS_2 from Pulps

Source: AMPL, 2023

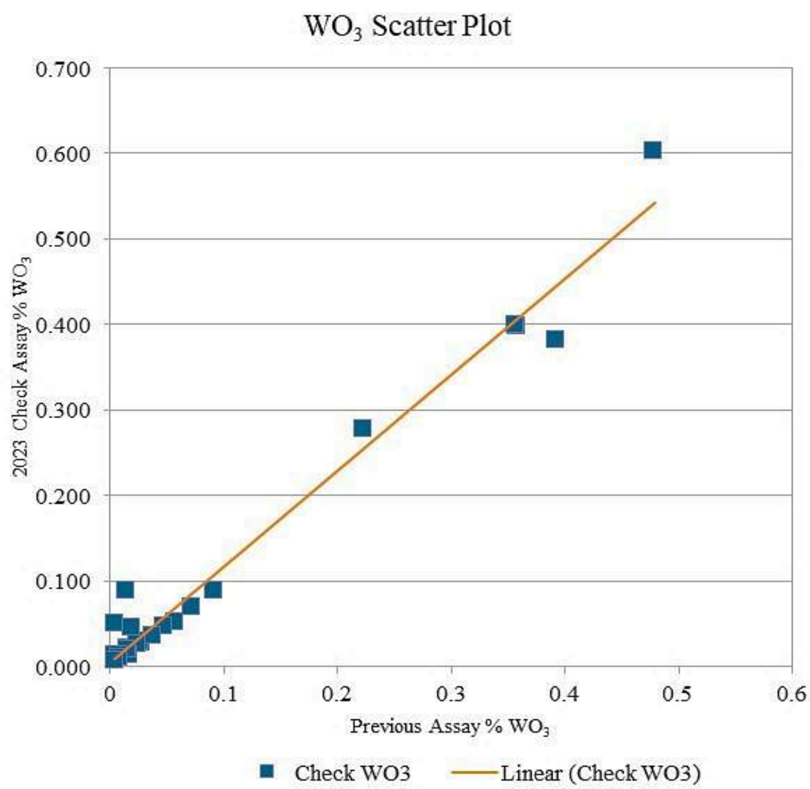


Figure 12.8. Scatter Plot Showing Original WO_3
Source: AMPL, 2023

12.3.5 Summary Comment with Respect to Assay Data

In the author's opinion, the assay table (Table 12.1, above) is a sufficiently accurate compilation of historical assays for use in a Resource estimate, providing that the varying levels of supporting documentation are considered.

12.3.6 Collar Coordinates

12.3.6.1 Checks Against Project Source Documents

The diamond drill hole data was received as three separate ".csv" files. A collar file, a downhole survey file, and an assay file. It is assumed that these were exported from a Gems™ program.

To verify collar coordinates, Mr. Kelly Grebliunas with AscT was contacted. Mr. Grebliunas was responsible for the surveying of both old and current drilling as well as the underground portion of the mine in 2006. AllNorth provided field notes as well as survey locations.

Microsoft Excel™ spreadsheets were provided to AMPL. The collar coordinates that were supplied were compared to those received from the Client, via Mr. Cuttle.

These comparisons are given in Table 12.3 and Table 12.4, below.

TABLE 12.3
COLLAR COORDINATES COMPARISON FROM ALLNORTH AND CLIENT – DDH 47 TO 164

DDH #	Coordinates from AllNorth UTM Coordinates - Collar (m)			Coordinates Received from Client UTM Coordinates - Collar (m)			Difference UTM Coordinates - Collar (m)		
	Northing	Easting	Elevation	Northing	Easting	Elevation	Northing	Easting	Elevation
47	6,075,307.5593	609,502.1280	1,069.4921	6,075,307.78	609,502.20	1,068.50	0.22	0.07	-0.99
49	6,075,307.4634	609,500.5296	1,069.7767	6,075,307.68	609,500.61	1,068.79	0.22	0.08	-0.99
51	6,075,185.4539	609,506.1291	1,069.9235	6,075,185.74	609,506.24	1,068.94	0.29	0.11	-0.98
52	6,075,185.5252	609,506.7293	1,069.5144	6,075,185.81	609,506.84	1,068.53	0.28	0.11	-0.98
58	6,075,550.3369	609,506.1405	1,071.0682	6,075,550.44	609,506.13	1,070.08	0.10	-0.01	-0.99
64	6,075,550.6416	609,506.5993	1,069.1070	6,075,550.75	609,506.59	1,068.11	0.11	-0.01	-1.00
65	6,075,550.5841	609,506.0819	1,068.2917	6,075,550.69	609,506.07	1,067.30	0.11	-0.01	-0.99
102	6,075,603.2637	609,159.6366	1,071.2449	6,075,603.23	609,159.79	1,070.25	-0.03	0.15	-0.99
104	6,075,183.0215	609,511.6609	1,069.4997	6,075,183.31	609,511.77	1,068.51	0.29	0.11	-0.99
107	6,075,246.7940	609,508.1129	1,069.1874	6,075,247.05	609,508.21	1,068.20	0.26	0.10	-0.99
109	6,075,246.7880	609,507.1129	1,068.8300	6,075,420.36	609,165.79	1,078.45	173.57	-341.32	9.62
110	6,075,246.6785	609,505.5170	1,069.3644	6,075,246.93	609,505.61	1,068.38	0.25	0.09	-0.98
111	6,075,246.6642	609,504.7965	1,069.0644	6,075,246.92	609,504.89	1,068.08	0.26	0.09	-0.98
113	6,075,245.9080	609,509.3002	1,068.9638	6,075,246.16	609,509.39	1,067.98	0.25	0.09	-0.98
116	6,075,368.6201	609,503.5801	1,068.8916	6,075,368.81	609,503.64	1,067.90	0.19	0.06	-0.99
117	6,075,368.6000	609,501.3501	1,068.8840	6,075,368.73	609,501.41	1,067.89	0.13	0.06	-0.99
134	6,075,176.3937	609,172.8060	1,069.4491	6,075,176.58	609,172.99	1,068.50	0.19	0.18	-0.95
138	6,075,621.2189	609,725.5324	1,066.1761	6,075,621.36	609,725.40	1,066.61	0.14	-0.13	0.43
139	6,075,307.2640	609,507.5348	1,068.9982	6,075,307.49	609,507.61	1,068.01	0.23	0.08	-0.99
140	6,075,621.1038	609,732.6367	1,066.1475	6,075,621.25	609,732.50	1,066.72	0.15	-0.14	0.57
141	6,075,185.7469	609,519.2641	1,070.5605	6,075,186.03	609,519.37	1,069.57	0.28	0.11	-0.99
142	6,075,024.6964	609,677.9247	1,070.6737	6,075,025.11	609,678.01	1,069.68	0.41	0.09	-0.99
143	6,075,023.7938	609,677.9584	1,070.6108	6,075,024.21	609,678.05	1,069.62	0.42	0.09	-0.99
144	6,075,020.5771	609,680.7670	1,071.0951	6,075,020.90	609,680.94	1,070.01	0.32	0.17	-1.09
145	6,075,026.3705	609,679.7626	1,070.6561	6,075,026.79	609,679.85	1,069.67	0.42	0.09	-0.99
146	6,075,022.8787	609,679.4220	1,070.6764	6,075,023.30	609,679.51	1,069.69	0.42	0.09	-0.99
147	6,075,206.6391	609,172.9457	1,074.6719	6,075,206.81	609,173.13	1,072.33	0.17	0.18	-2.34
149	6,075,206.4976	609,174.0094	1,072.6669	6,075,206.67	609,174.19	1,072.08	0.17	0.18	-0.59
150	6,075,267.4422	609,171.3839	1,073.1692	6,075,267.58	609,171.61	1,072.22	0.14	0.23	-0.95
151	6,075,267.5070	609,173.8232	1,073.2019	6,075,267.65	609,173.91	1,072.29	0.14	0.09	-0.91
152	6,075,267.4255	609,172.3240	1,073.7648	6,075,267.57	609,172.51	1,072.77	0.14	0.19	-0.99
153	6,075,328.7016	609,169.5004	1,073.2841	6,075,328.82	609,169.68	1,072.29	0.12	0.18	-0.99
154	6,075,329.3005	609,172.0236	1,073.2419	6,075,329.42	609,172.20	1,072.25	0.12	0.18	-0.99
155	6,075,328.7103	609,171.3779	1,073.6315	6,075,328.82	609,171.56	1,072.64	0.11	0.18	-0.99
156	6,075,389.2767	609,168.5656	1,073.2075	6,075,389.36	609,168.75	1,072.21	0.08	0.18	-1.00
157	6,075,389.2932	609,170.2035	1,073.3354	6,075,389.38	609,170.39	1,072.34	0.09	0.19	-1.00
158	6,075,389.1116	609,169.2963	1,073.4572	6,075,389.20	609,169.48	1,072.46	0.09	0.18	-1.00
159	6,075,450.5159	609,190.5427	1,073.2152	6,075,450.58	609,190.72	1,072.22	0.06	0.18	-1.00
160	6,075,572.0660	609,164.0600	1,073.4590	6,075,572.06	609,164.21	1,072.46	-0.01	0.15	-1.00
161	6,075,510.0403	609,166.2101	1,073.7407	6,075,511.16	609,166.20	1,071.50	1.12	-0.01	-2.24
162	6,075,510.1631	609,164.1988	1,072.9999	6,075,510.83	609,165.62	1,071.09	0.67	1.42	-1.91
163	6,075,451.3696	609,187.5647	1,072.7826	6,075,451.43	609,187.74	1,071.79	0.06	0.18	-0.99
164	6,075,449.7131	609,186.8839	1,072.8162	6,075,449.77	609,187.06	1,071.82	0.06	0.18	-1.00

TABLE 12.4
COLLAR COORDINATES COMPARISON FROM ALLNORTH AND CLIENT – DDH 165 TO 196

	Coordinates from AllNorth				Coordinates Received from Client				Difference		
DDH #	UTM Coordinates - Collar (m)				UTM Coordinates - Collar (m)				UTM Coordinates - Collar (m)		
	Northing	Easting	Elevation		Northing	Easting	Elevation		Northing	Easting	Elevation
165	6075023.883	609675.820	1072.183	165	6075024.3	609675.91	1069.62		0.42	0.09	2.56
166	6075067.626	609630.541	1069.608	166	6075067.84	609630.85	1069.11		0.21	0.31	0.50
167	6075067.536	609631.098	1069.624	167	6075067.95	609631.42	1069.12		0.41	0.32	0.50
168	6075096.984	609600.143	1069.502	168	6075097.34	609600.25	1069.2		0.36	0.11	0.30
169	6075126.360	609568.918	1069.168	169	6075126.71	609569.34	1068.66		0.35	0.42	0.51
170	6075126.419	609569.143	1069.248	170	6075126.72	609569.23	1068.74		0.30	0.09	0.51
171	6075126.282	609568.544	1069.700	171	6075126.63	609568.68	1068.71		0.35	0.14	0.99
172	6075067.109	609636.809	1070.309	172	6075067.49	609636.9	1069.17		0.38	0.09	1.14
173	6075096.911	609599.607	1070.173	173	6075097.26	609599.71	1069.19		0.35	0.10	0.98
174	6075024.034	609675.795	1070.623	174	6075024.45	609675.89	1069.52		0.42	0.09	1.10
175	6075023.806	609676.014	1071.054	175	6075024.22	609676.1	1069.67		0.41	0.09	1.38
176	6075096.422	609605.729	1069.949	176	6075096.78	609605.83	1068.96		0.36	0.10	0.99
177	6075125.797	609573.780	1069.776	177	6075126.13	609573.88	1068.79		0.33	0.10	0.99
178	6075125.840	609574.130	1069.765	178	6075126.17	609574.23	1068.78		0.33	0.10	0.99
179	6076545.184	611928.773	696.043		missing from Dbase – Portal 2 hole						
181	6075525.036	609162.670	1073.956	181	6075525.06	609162.84	1072.96		0.02	0.17	1.00
182	6075461.928	609171.001	1073.477	182	6075461.98	609171.19	1072.49		0.05	0.19	0.99
183	6075404.936	609165.254	1073.794	183	6075405.01	609165.44	1072.8		0.07	0.19	0.99
184	6075252.431	609171.400	1076.220	184	6075252.48	609169.56	1073.19		0.05	-1.84	3.03
185	6075282.817	609168.891	1074.087	185	6075282.96	609169.3	1072.91		0.14	0.41	1.18
186	6075313.675	609169.669	1075.871	186	6075313.8	609169.85	1074.87		0.12	0.18	1.00
187	6075343.572	609168.187	1075.745	187	6075343.68	609168.37	1072.2		0.11	0.18	3.54
188	6075374.515	609167.091	1076.577	188	6075374.61	609167.28	1075.58		0.09	0.19	1.00
189	6075374.441	609166.604	1074.527	189	6075374.53	609166.79	1073.53		0.09	0.19	1.00
190	6075314.060	609173.953	1073.051	190	6075314.19	609174.13	1072.06		0.13	0.18	0.99
191	6075344.002	609172.743	1075.022	191	6075344.11	609172.93	1073.72		0.11	0.19	1.30
192	6075344.124	609173.466	1072.357	192	6075344.23	609173.65	1071.36		0.11	0.18	1.00
193	6075374.717	609172.184	1074.399	193	6075374.81	609172.37	1073.41		0.09	0.19	0.99
194	6075374.710	609172.406	1072.934	194	6075374.8	609172.59	1071.94		0.09	0.18	0.99
195	6075525.077	609163.516	1074.456	195	6075525.1	609163.69	1073.46		0.02	0.17	1.00
196	6075461.799	609171.484	1073.851	196	6075461.85	609171.67	1072.25		0.05	0.19	1.60

In general, the collar coordinates match well with surveyed collars from AllNorth. However, hole 109 appears to have been misplotted or mislabeled. The author was unable to locate the original drill log. It was assumed that the error may be in the diamond drill hole identification underground. This hole was not used in the estimate.

Generally speaking, the coordinates used and those later supplied by AllNorth show some variances, particularly in elevation. When the diamond drill holes are viewed in 3D, relative to the underground drift, as supplied by AllNorth, it appears that their numbers are more correct.

The author cannot explain why the numbers are slightly different or why they were truncated. However, for the purposes of this Resource, they are deemed acceptable as maximum elevation difference is 3.5 m, block size is 10 m, and the vertical component of the deposit often exceeds 200 m. For detailed planning purposes, these discrepancies need to be investigated.

12.3.6.2 Field Checks

Mr. Salmabadi was unable to have any collars physically inspected because the road to the Property was cut off by fallen trees. The site visit did not allow for enough time to have the road cleared. In addition, most of diamond drill holes were also drilled from underground and are currently inaccessible. Other historical collars to holes drilled from the surface on the Hudson Bay Glacier have since disappeared.

12.3.6.3 Summary Comment Respecting Drill-Hole Locations

In the author's opinion, the drill hole locations in the database are sufficiently accurate for use in a Resource estimate. The locations of most of the drill holes are well documented by AllNorth. There is little documentation available for the locations of earlier drill holes. Locations are available in drill logs for drill holes, but these are on a local grid and the author does not have a key for converting local grid references to UTM. However, despite the inability to do field checks, the author feels confident that the collar surveys were reasonably well done, and the data is reliable.

12.3.7 Downhole Surveys

The following, in italics, is from Hatch (2008) and was originally documented by Snowden in 2006:

> *Drillhole collars were surveyed by Kelly Grebliunas of Allnorth Consultants Ltd., using a Sokkia Total Station SET500. The initial azimuth and inclination of the drill hole were also surveyed at the collar. This was done by surveying the drill rod or drill slide at the beginning of the drillhole. Downhole surveys were taken at a distance of 15 m (50 ft) from the collar of each drillhole and then at intervals of every 30 m (100 ft). The instrument used was a Flexit tool, supplied by Fordia Ltd. This instrument incorporates a compass and a dip needle, both with electronic readout transmitted to a data pad by radio signal. The survey instrument measures the intensity of the magnetic field in addition to taking azimuth and inclination readings.*
>
> *As with any compass-based instrument, the azimuth readings are subject to inaccuracies caused by local magnetic fields associated with occurrences of magnetite or pyrrhotite. Pyrrhotite is relatively rare in this deposit, but magnetite is common in veins and occasional coarse disseminations in the intrusive rocks and in veins and widespread fine disseminations in the volcanic rocks. It was often difficult to get reliable readings in the volcanic rocks but this is not considered to be a serious problem as these rocks are only encountered towards the bottom of the drill holes, where survey errors are considered to be less significant.*
>
> *Down hole surveying identified a problem with excessive deviation of nearly 3° per 30 m (100 ft) in DDH165. This was remedied in succeeding holes by the use of a core barrel with an oversize outer diameter which generally reduced deviation to less than 0.5° per 30 m (100 ft). The larger diameter core barrel can cause problems with drilling in bad ground, but conditions on this property are generally good enough that any such problems are minimal.*
>
> *BPM notes that, with the use of an oversize core barrel, deviations of the drill hole generally averaged less than 0.5° per 30 m (100 ft) during the 2006 drilling campaign. BPM regard downhole survey readings resulting in azimuth deviations of greater than about 1° per 30 m (100 ft) as being suspect. All downhole surveys were reviewed by BPM's Jim Hutter during drilling. Suspect surveys, where observed, were highlighted and where practical the survey was repeated with the downhole instrument being shifted slightly within the drill hole in an attempt to mitigate the disturbance of proximal magnetite. Of 254 initial surveys, 51 were repeated. On analysis of the final results, 60 of the surveys*

produced results that appeared unreasonable, indicating deviations that would be unlikely or impossible. These 60 surveys were then adjusted to produce a smooth curve that could reasonably be followed by a drill string. In most cases azimuths would tend to gradually increase with hole depth.

An inherent inaccuracy is present in surveying the rod or slide with a transit at the top of the hole, as the survey points are not very far apart, and therefore a slight error in the surveyed location of either point induces a significant error in the azimuth or inclination of the hole. Additional sources of error associated with the initial azimuth and inclination survey data include the possibility of slight shifting of the drill between collaring and surveying, and deviation of the drill rod due to uneven ground during collaring. In the event of a discrepancy between the initial azimuth and inclination readings and those determined by downhole surveying, BPM considered the downhole survey orientations as being correct if they appeared to be consistent and reasonable.

In the event of poor-quality downhole surveys in the first part of the hole (six of thirty holes), the collar survey was considered to be correct and was used to set the initial azimuth.

Of the 30 holes drilled, 18 had collar azimuth surveys that were within one degree of the adjusted initial downhole survey, four more varied by two degrees or less, and another four varied by three degrees or less. The azimuth surveys for DDH 170 varied by nearly 45 degrees, but this hole was inclined very close to vertical, making the azimuth very hard to measure accurately and in any case of little consequence. There was a variance in azimuth surveys for the remaining three holes of 3.3, 3.6 and 7.5 degrees. A variance of 7.5 degrees is considered excessive, however the downhole surveys for this hole were of sufficient quality to locate the hole reliably and were taken to be correct.

Downhole measurements of inclination rely only on gravity and therefore are not subject to magnetic interference which causes difficulties with azimuth measurements. In all but three cases the inclinations used for plotting were those returned by the downhole survey instrument. Changes in inclination never averaged more than 0.4 degrees per hundred feet in any hole, and usually averaged less than 0.2 degrees per hundred feet. In downholes, inclinations would usually tend to decrease slightly with increasing depth, whereas in upholes the inclinations would tend to increase.

The inclination of the drillhole at the collar was also measured for 22 of the 30 holes using a machinist's protractor (interpolated to 0.1°) as an additional check on the surveying. In most cases the collar survey, initial downhole survey and machinist's protractor agreed to within one degree.

J.M Hutter of Blue Pearl Mining Inc. has reviewed all of the downhole survey data, making modifications where necessary as previously indicated, and is of the opinion that the downhole survey data suitably define the traces of the drillholes for the 2006 drilling campaign, and is satisfied that the samples are therefore sufficiently accurately located in 3-D space for a resource estimation and to support an Indicated and/or Measured resource classification.

12.3.8 Mine Grid Coordinate

The following, in italics, is from Hatch (2008) and was originally documented by Snowden in 2006.

> *The underground workings had been surveyed to a local mine grid by previous operators using transit and tape, which was the technology available at the time. A re-survey in 2005/2006 using modern equipment indicated a survey error of approximately 2.5 m (8 feet) over the 2 km distance of the workings. The re-survey was done by Kelly Grebliunas of Allnorth Consultants Ltd, an engineering firm with offices in Smithers and several other locations in British Columbia. Existing control points were re-established in UTM and tied into the Mine Grid.*
>
> *Equipment used was a Leica Geosystems Global Positioning Systems GPS Series 500. The level of accuracy achievable with this system using the Rapid Static Method is 5 to 10 mm. The survey was then carried underground using existing control points with a Sokkia Total Station SET500. All available historic drillhole collars were re-surveyed, and the information gained was applied to assign new coordinates for the old drillholes that were no longer visible or could not be accessed. All new drilling was surveyed using the 2005/2006 mine grid. This mine grid, which was used for geological purposes, has the same surface control points as the old mine grid, but the underground coordinates are somewhat different due to the error in the old surveying. The old mine grid is no longer used. For reporting and engineering requirements, the drillhole surveys are converted to UTM (NAD 83) coordinates.*

12.3.9 Site Inspection

During the period June 17 to June 19, 2023, Mr. Salmabadi conducted a site inspection of the Davidson Property. He spent time at the geology office and core storage facility located in Smithers and spent hours reviewing drill core and paper records.

Mr. Salmabadi collected five split-core samples of drill core. These samples were kept in the custody of the author until the samples were sealed in plastic bags and closed with a numbered single-use plastic “zap strap”. The samples were always in the possession of Mr. Salmabadi and were delivered to Bureau Veritas in Vancouver, British Columbia. In addition to the core samples, 36 pulps from the Climax/AMAX era drill cores samples and 5 samples from Blue Pearl/Blue Pearl era drill core samples were collected and delivered to Bureau Veritas in Vancouver, British Columbia. The results are previously discussed in Section 12.3.4.

12.3.10 Author’s Summary Statement

In the authors’ opinion, the assays, drill hole locations, and downhole surveys recorded in the Project’s database are of adequate quality to uphold the Resource estimate described in this report.

Drill core evaluation in the field supports the geological characteristics and interpretations of this deposit as presented in this report.

13.0 MINERAL PROCESSING AND METALLURGICAL TESTING

No new testing was performed in the preparation of this report. All the testing and the design basis were developed or summarised by others in previous reports. A number of the reports were written prior to adoption of NI 43-101. As such, these reports may not meet the standards required of the instrument for studies beyond a PEA. The current study is a PEA, and hence, this work is sufficient.

The recoveries of molybdenum to concentrate and concentrate grade assumptions used in this report are based on extensive metallurgical testing. A number of reports on the metallurgy of the property have been prepared over the years. The following reports were used in the preparation of the process design detailed in Section 17.0 of this report. In each of the descriptions below, “The report” refers only to the report being summarised.

- 1978 – Amax – Preliminary Process Evaluation – Tungsten By-Product, Carson, Wick
- 1980 – Amax – Yorke-Hardy Data Evaluation, Enochs
- 1981 – Kilborn – Preliminary Design and Cost Estimate
- 2008 – Hatch – Feasibility Study.

Recent testing (post 1981) has included extensive mineralogical work including QEMSCAN and similar studies. Given the very extensive bench and pilot scale testing of processes that provide excellent guidelines for process parameters, such as target grind sizes, retention times, etc., this data is of limited value in progressing the Project to an economic process. Future testing should focus on optimising the unit processes already identified by varying process parameters previously identified. New diamond drill core should be analysed appropriately to confirm previous results.

13.1 1978 – AMAX – PRELIMINARY PROCESS EVALUATION – TUNGSTEN BY-PRODUCT

The report asserts that the property has been under study since 1964 and several continuous pilot plant tests had been conducted. The report asserts that:

> *In 1976, a pilot plant program was conducted to evaluate the technical feasibility of recovering tungsten from the molybdenite rougher tailings.*

A report *Mineral Processing Yorke-Hardy Report Number 3, dated April 1, 1977* is referenced, but was unavailable.

Gravity recovery was by use of Reichert Cones. This technology has been superseded by more efficient equipment. The Reichert Cones produce relatively low-grade concentrates. These were leached and subsequently ammonium paratungstate (APT) was produced. The report asserts:

> *A single set of final process recommendations for the total Yorke-Hardy venture do not now exist.*

That is, the process design was incomplete. At the time the report was written, gravity recovery techniques were inferior to those available today. In particular, the use of gravity recovery on the molybdenum flotation tailings may not be the ideal option.

Design basis information from the report was as follows:

- Plant feed rate: 416 short tons per hour – corresponds to an operating throughput of 10,000 tonnes per day.
- Remaining design basis information was related to tungsten treatment only.

13.2 1980 – AMAX – YORKE-HARDY DATA EVALUATION

This report is a comprehensive summary of work previously completed on the Property. The report was commissioned to evaluate the feasibility of a 2,000 short tons per day mine with feed grade of 0.5% MoS_2 0.06% WO_3 ore.

The report was commissioned by Climax Molybdenum Company.

The grind was 42% plus 100 mesh. This is roughly a p80 of about 53 µm. Recovery was 88% MoS_2 with a grade of 90% MoS_2.

Design basis information from the report was as follows:

Ore is within a host rock of granodiorite.

In 1971, the Colorado School of Mines Research Institute determined bond indices in kilowatt hours per short ton:

- Wic: 9.86
- Wir: 15.14 to 17.08 kWh/st
- Wib: 13.62 to 14.61 kWh/st

Pilot plant testing in 1971 found it to be 16.0 kWh/st.

In plant retention times as determined by pilot plant trials are:

- Rougher: .... 18 minutes
- First Cleaner:...................................... 15 minutes
- First Cleaner Scavenger:.................... 15 minutes
- Second to Fourth Cleaners:................ 12 minutes each

The report stated that there was insufficient data for mill design. Missing data includes feed and discharge sizes of re-grind circuits and estimates of work indexes for these streams. This means re-grind mill sizing will be an estimation.

Metallurgical test reports were extracted in the appendix of the main report. These covered testing from 1964 to 1977.

The report indicated that oxidation of the stored samples resulted in reduced recovery and quality of concentrates. This implies that stockpiled material will be detrimentally impacted. This must be considered if low-grade, long-term stockpiles are to be treated at the end of mine are planned.

13.3 1981 – KILBORN – PRELIMINARY DESIGN AND COST ESTIMATE

This report does not contain any supporting information. It does provide a detailed description of the mill, including flowsheets, general arrangements, and a comprehensive design basis. It is assumed that that the testing detailed in the 1980 – AMAX report was used as the source of the design basis. The design basis information from the report is the basis for the process design detailed in the section "Recovery Methods". Modern methods, including tank cell flotation, coarse particle flotation, column flotation, and stirred media grinding were not generally available at the time the testing was performed. The section below makes assumptions on potential improvements to the circuit using more modern equipment.

The author has assumed that the report is based on short tons (st), not metric tonnes.

Design Basis information from Kilborn:

- Ore Bulk Density — 110 lb/ft³ – Equivalent to 1.41 mt/m³
- Bond ball mill work index — 14 kWh/st – This is the minimum from testing – this is too low.
- Primary grind — 10% +35 mesh (author assumes this is Tyler mesh hence 420 microns. If Canadian mesh, would be 500 microns)
- Circulating load — 300%
- Cyclone underflow — 76% solids (author assumes this is weight by weight w/w)
- Cyclone overflow — 40% solids
- Mill discharge — 76% solids
- Rougher concentrate feed rate — 224 short tons per hour (st/hr)
- Rougher concentrate mass pull — 7 st/hr – this is 5.13% of total feed
- Rougher circuit retention time — 18 min
- Scavenger concentrate mass pull — 4.5 st/hr
- Scavenger circuit retention time — 15 minutes
- Fourth cleaner concentrate mass pull — 0.7 st/hr – this is 0.31% of total feed
- Total cleaner circuit retention time — 51 minutes

There are two points on the size distribution. 10% + 420 µm and 42% + 149 microns. This gives an approximate p80 between 53 µm and 74 µm. Grinding calculations assume a p80 of 53 µm – this is conservative.

13.4 2008 – HATCH – FEASIBILITY STUDY

The Hatch report is of limited use for process design purposes. Some further testing was performed, but these often included blends of Endako material. Metallurgical testing did not include a pilot plant; hence, it is not suitable for a NI 43-101-compliant FS. Previous pilot plant data does not reflect new technology. Testing was to a PFS level. The primary purpose of the Hatch study was to evaluate the effectiveness of using the Endako mill to process the ore.

Design Basis information from Hatch:

- Ore SG.: 2.66

Comminution:

- SPI..86
- “Autogenous”..................................17.8 kWhr/t
- Wir – Bond Rod..............................14.0 to 15.5 kWhr/t
- Wib – Bond Ball ....15.5 to 17.4 kWhr/t

13.5 DESIGN BASIS

This report uses the following information for the proposed mill design. Note that this information is considered by the author to be suitable for a PEA level NI 43-101 report. Much of the data was generated from testing performed prior to 1980. Many of these were full pilot plant scale test programs. As such, if the mill were to duplicate the design developed based on these tests, the design could be developed to be suitable for a FS level NI43-101 report. Using this design would eliminate the use of new technologies. This report does not assert any potential improvements in performance through the use of new technologies. This should be evaluated if further evaluation of the Property is indicated.

The design basis summarised below was used to develop the information required to estimate equipment sizes for a new mill. The information is generated based on a process design criteria (PDC) developed to a level suitable for a PFS level NI43-101 report. To improve the level, further data would need to be generated. Pilot plant studies would be necessary for a FS level study.

In addition, the data is provided to equipment suppliers to determine the size of proposed equipment.

13.5.1 Operational Constraints

These data are relatively independent of the mineralisation and process. Note that times do not include availability and utilisation. As such, the mill may not operate every day of the year. Down time is accounted for in the mass balances by utilisation factors (Table 13.1, below).

TABLE 13.1
OPERATING DAYS/YEAR

Parameter	Value	Units	Source	Comments
Days per Year	365	d	JGE	
Hours Per Day	24	h	JGE	

13.5.2 Mineralisation

These data detail the characteristics of the mineralisation (Table 13.2, below).

TABLE 13.2
MINERALISATION CHARACTERISTICS

Parameter	Value	Units	Source	Comments
Annual Tonnage	2,500,000	tpa	BL	
Specific Gravity	2.66		Hatch	
Bulk Density	1.44	t/m³	Kilborn	Was given as 110 lb/ft³
Run of Mine Moisture	2%	w/w	JGE	Assumption for mass balance

13.5.3 Crushing

These data detail the information necessary to determine the mass flows in the crushing circuit proposed (Table 13.3, below).

TABLE 13.3
MASS FLOWS: CRUSHING CIRCUIT

Parameter	Value	Units	Source	Comments
Availability	50%		JGE	% of time crushing per day
Fine Ore Capacity – % of Daily Tonnage	2.66		Hatch	
Fine Ore Raise Maximum Diameter	20	m	BL	For sizing of raise
Final Crusher Product p80	12,500	µm	JGE	

13.5.4 Grinding

These data detail the information necessary to determine the mass flows in the grinding circuit proposed (Table 13.4, below).

TABLE 13.4
MASS FLOWS: GRINDING CIRCUIT

Parameter	Value	Units	Source	Comments
Primary Grinding Availability	95%		JGE	% of time crushing per day
Circulating Load	300%		Kilborn	
Cyclone Underflow SG	2.66		JGE	Assumption, may be higher
Cyclone Feed Density	61.6%		JGE	Needed for 40% flotation feed density
Cyclone Underflow Density	75%		JGE	First approximation
Ball Mill Feed Density	75%		JGE	First approximation
Flotation Feed p80	240	µm	Hatch	Based on 10% +420 µm; 42% +149 µm

13.6 GRAVITY

These data detail the information necessary to determine the mass flows in the gravity circuit proposed. Note that there is no testing to confirm the performance of the gravity circuit for recovery of tungsten (Table 13.5, below).

TABLE 13.5
MASS FLOW: GRAVITY CIRCUIT

Parameter	Value	Units	Source	Comments
Primary Cyclone Underflow to Gravity	25%		JGE	First approximation – no testing to confirm
Per Concentrator Water Addition	20	m^3/hr	JGE	First approximation
Number of Primary Concentrators	2		JGE	First approximation

13.7 FLOTATION

These data detail the information necessary to determine mass flows in the flotation circuit proposed (Table 13.6, below).

TABLE 13.6 **MASS FLOWS: FLOTATION CIRCUIT**				
Parameter	**Value**	**Units**	**Source**	**Comments**
% of Fresh Mill Feed to Rougher Concentrate	5.13%		Kilborn	
% of Fresh Mill Feed to Final Molybdenum Concentrate	0.31%		Kilborn	
Concentrate SG – Assumed, may be higher	2.66		JGE	
Concentrate Density – Assumed, may be lower	40%		JGE	
Rougher Retention Time	18	min	Kilborn	
Number of Rougher Cells	3		JGE	
Scavenger Retention Time	15	min	Kilborn	
Number of Scavenger Cells	3		JGE	
Cleaner Retention Time	51	min	Kilborn	
Number of Cleaner Columns or Equivalent	2		JGE	
Flotation Cell Volume Effectiveness	80%		JGE	

13.8 DE-WATERING

These data detail information necessary to determine mass flows in the dry stack tails process and the paste fill process. It is proposed that a single pressure filter circuit be used for both. Filtered tails would be combined with fresh slurry to produce paste fill of a suitable density (Table 13.7, below).

TABLE 13.7 **MASS FLOWS: DRY STACK TAILINGS CIRCUIT**				
Parameter	**Value**	**Units**	**Source**	**Comments**
Paste Fill Density	80%		JGE	Based on Golder Pastetec Design Basis
Dry Stack Tailings Density	95%		JGE	To be confirmed

13.9 EXPECTED PERFORMANCE

The performance estimate from AMAX – 1980 indicated MoS_2 recovery to final concentrate of 88% with a concentrate grade of 90% MoS_2. The performance estimate from Hatch – 2008 indicated recovery to final concentrate of 92% with similar grades. The newer value can be considered appropriate, given improvements in flotation technology.

14.0 MINERAL RESOURCE ESTIMATE

At the request of Moon River Capital Limited, AMPL Professionals were retained to produce an updated Resource Estimate on the Davidson Property located in Smithers, British Columbia. There has been no additional drilling since 2007 on the Property. The effective date for this estimate is September 13, 2023.

Mr. Finley Bakker, P.Geo. is the Qualified Person responsible for the Resource estimate. Mr. Bakker is a Qualified Person by virtue of education, experience, and membership in a professional association. He is independent of the Company applying all the tests in Section 1.5 of the NI 43-101. Mr. Bakker did not make a visit to the Property. Mr. Eshan Salmabadi, P.Geo. visited in his stead in June 2023.

There appears to be no issues or factors that could materially affect the Mineral Resource estimate. This includes no issue involved with environmental permitting, legal, title, taxation, socio-economic, marketing, political, mining, metallurgical, or infrastructure.

14.1 DATA ANALYSIS AND VERIFICATION

For the 2023 update of the Davidson Resource, no additional drilling data was incorporated since the previous NI 43-101 in 2016, as no additional drilling has been performed since that time. All units are metric.

A comparison between the 2016 Resource and the recent 2023 Resource was completed.

Much of data received to undertake the 2023 update of the Davidson Project Resource could best be described as second hand/one step removed. As such, it was important to verify the model often using non-traditional methods.

14.1.1 Comparison of a Physical 3D Model with a Computer-Generated Model

Climax Molybdenum created several models based on different cut-offs, as shown in Figure 14.1, below. The red model is based on a 0.1% MoS_2 cut-off. The digital model, created using MineSight™ software (see Figure 14.2, below), is reasonably close, although it used more recent drill holes as well.

Figure 14.1. Physical 3D Model of Deposit
Source: AMPL, 2023

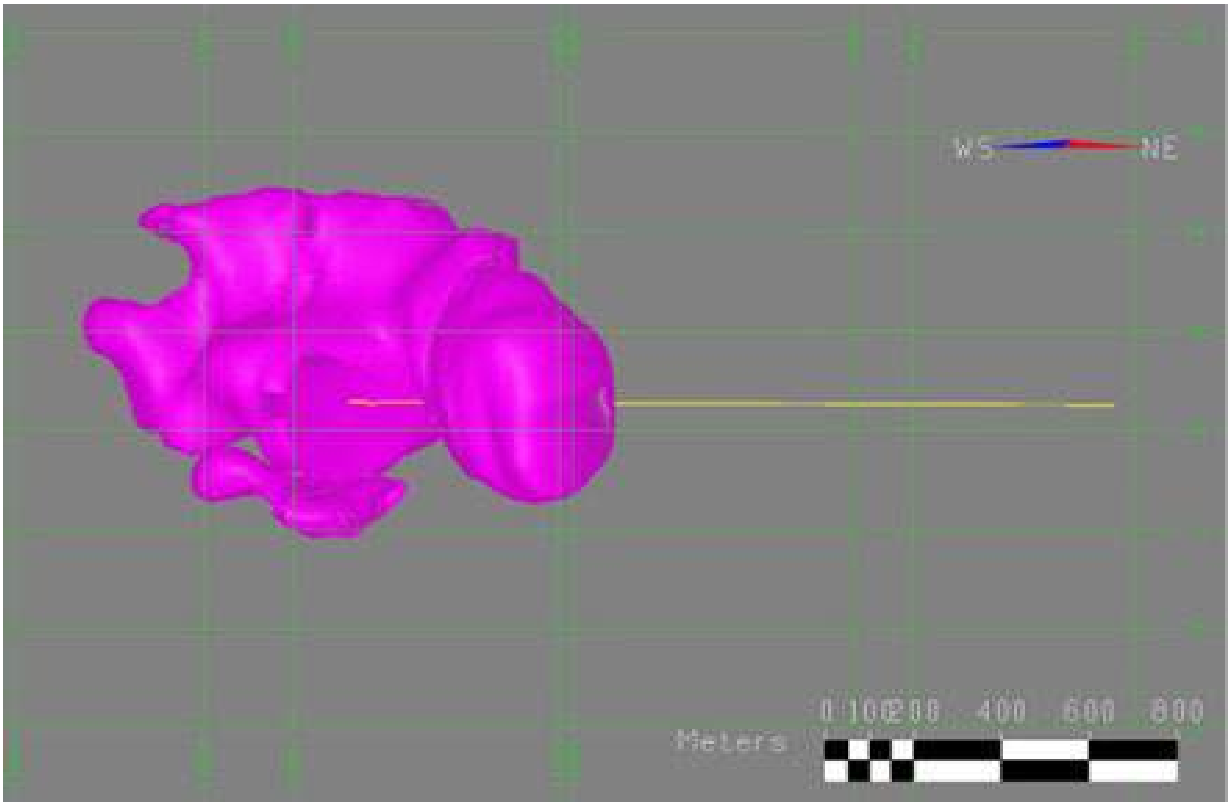


Figure 14.2. Computer Generated Model
Source: AMPL, 2023

While AMPL did not have access to more recent digital models of the deposit, they were able to access previous detailed physical models.

The physical models show a reasonable resemblance to the 3D wire-frame generated in MineSight™ with the exception of some outliers based on grade in the digital model (see Figure 14.3 and Figure 14.4, below).

Figure 14.3. Plexiglass Model Showing Sections
Source: AMPL, 2023

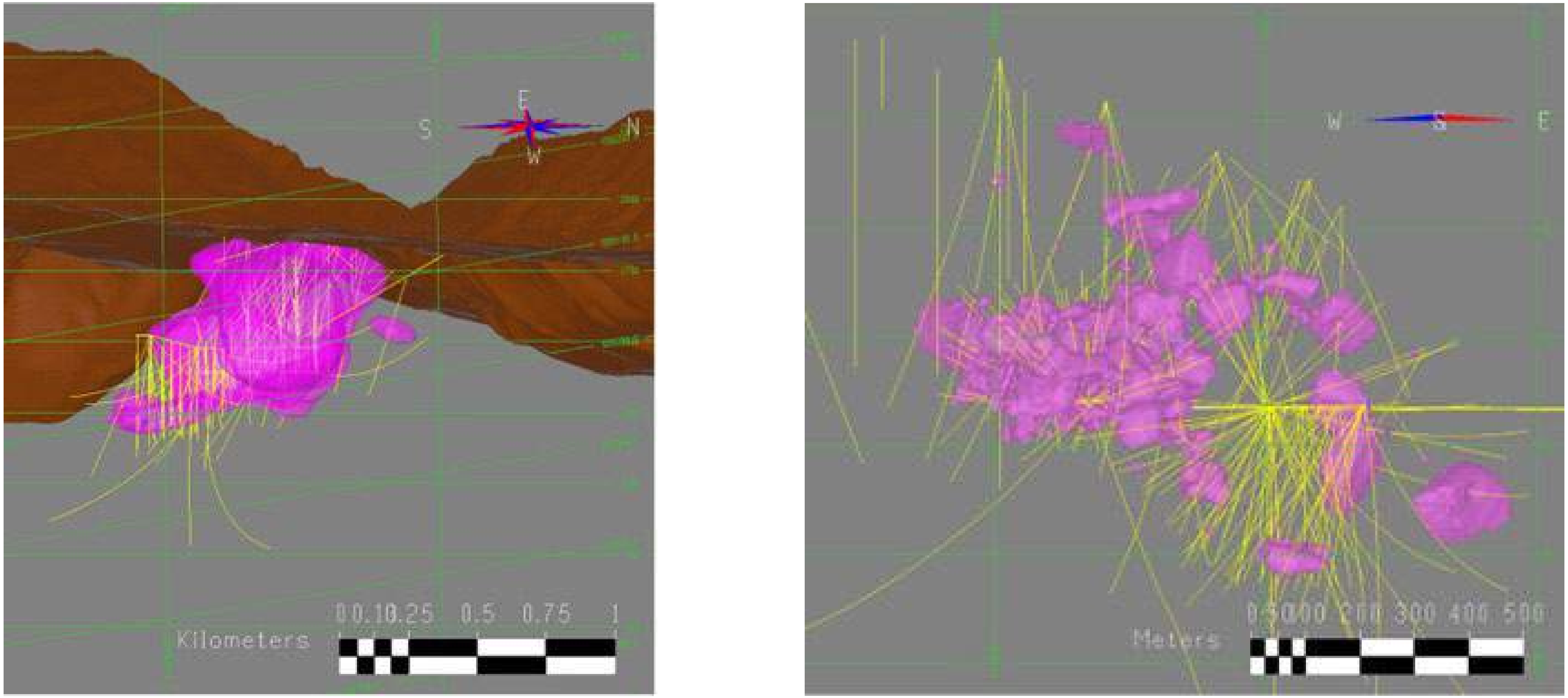


Figure 14.4. Comparison of Plexiglass Model with Computer Generated Model
Source: AMPL, 2023

14.2 DIAMOND DRILL HOLE DATA

14.2.1 Diamond Drill Downhole Assays

Diamond drill hole assays were received as ".csv" files, which were extracted from a previous Resource model. In addition, Mr. Salmabadi found original documents detailing sampling for holes 165 through to 190. It immediately became apparent that there was a discrepancy in the "raw data" received. The original logs were Imperial and much of the assaying was done over 1.52 m (5-ft) intervals. The data received indicated that all assaying was done over 3.04 m (10-ft) intervals. As a result, all holes from 165 through to 190 were manually entered into a spreadsheet. It was obvious that the assays had been averaged over 3.04 m (10-ft) intervals. Checks of the drill holes comparing intervals did not find any errors but there appears to be no logical explanation as to why this method of dealing with the data was employed. On some sheets, the 1.52 m (5-ft) intervals were already combined and made the comparison much easier. No errors were found.

14.2.2 Diamond Drill Downhole Surveys

There was no way to physically check downhole surveys, but visual inspection of diamond drill traces is reasonable with holes flattening and deviating to the right as would be expected with the rotation of rods. It would appear that the holes were pushed hard with expected results (see Figure 14.5 and Figure 14.6, below).

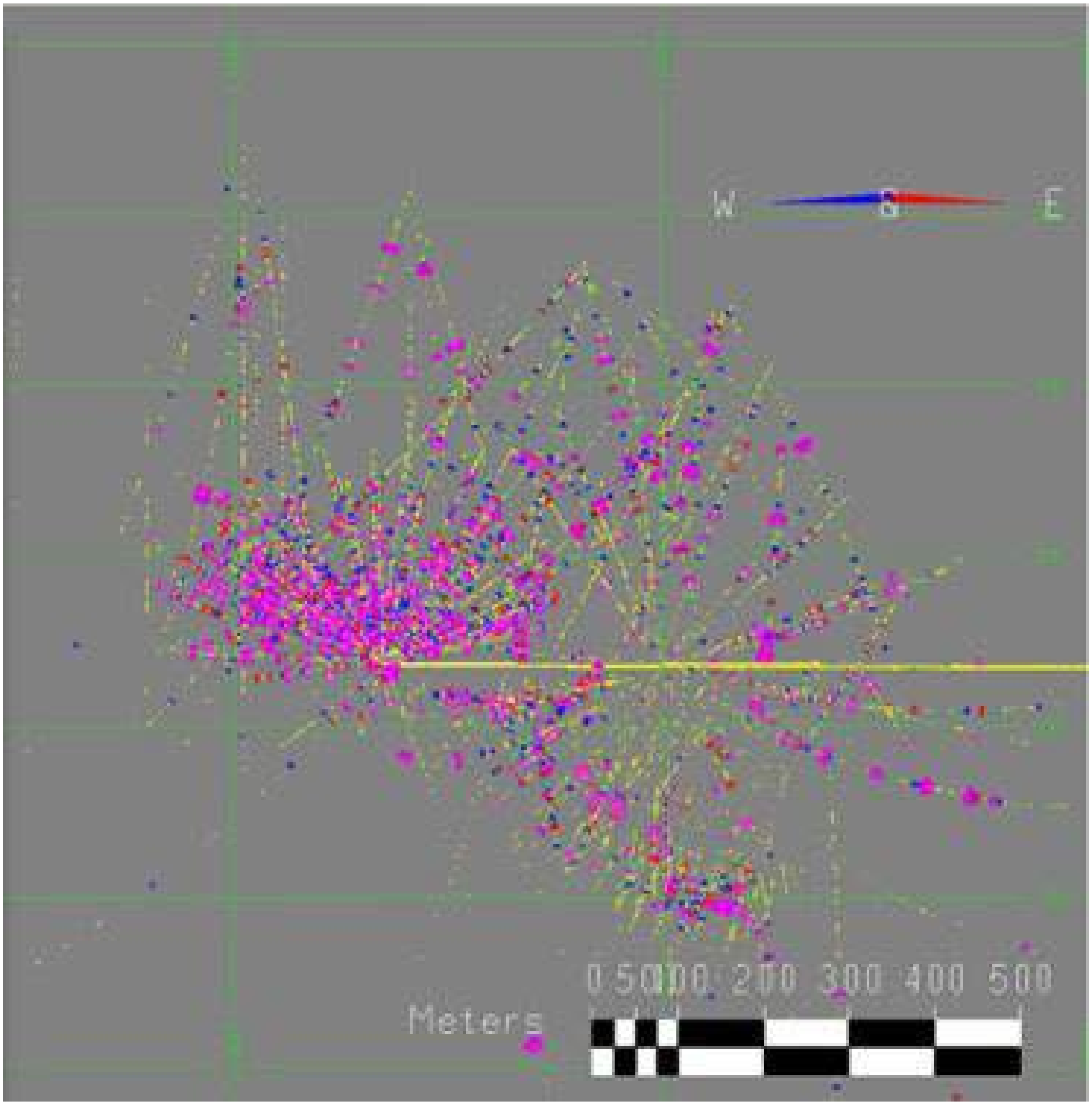


Figure 14.5. Showing Intercepts of >0.10% MoS_2 (in Cyan)
Source: AMPL, 2023

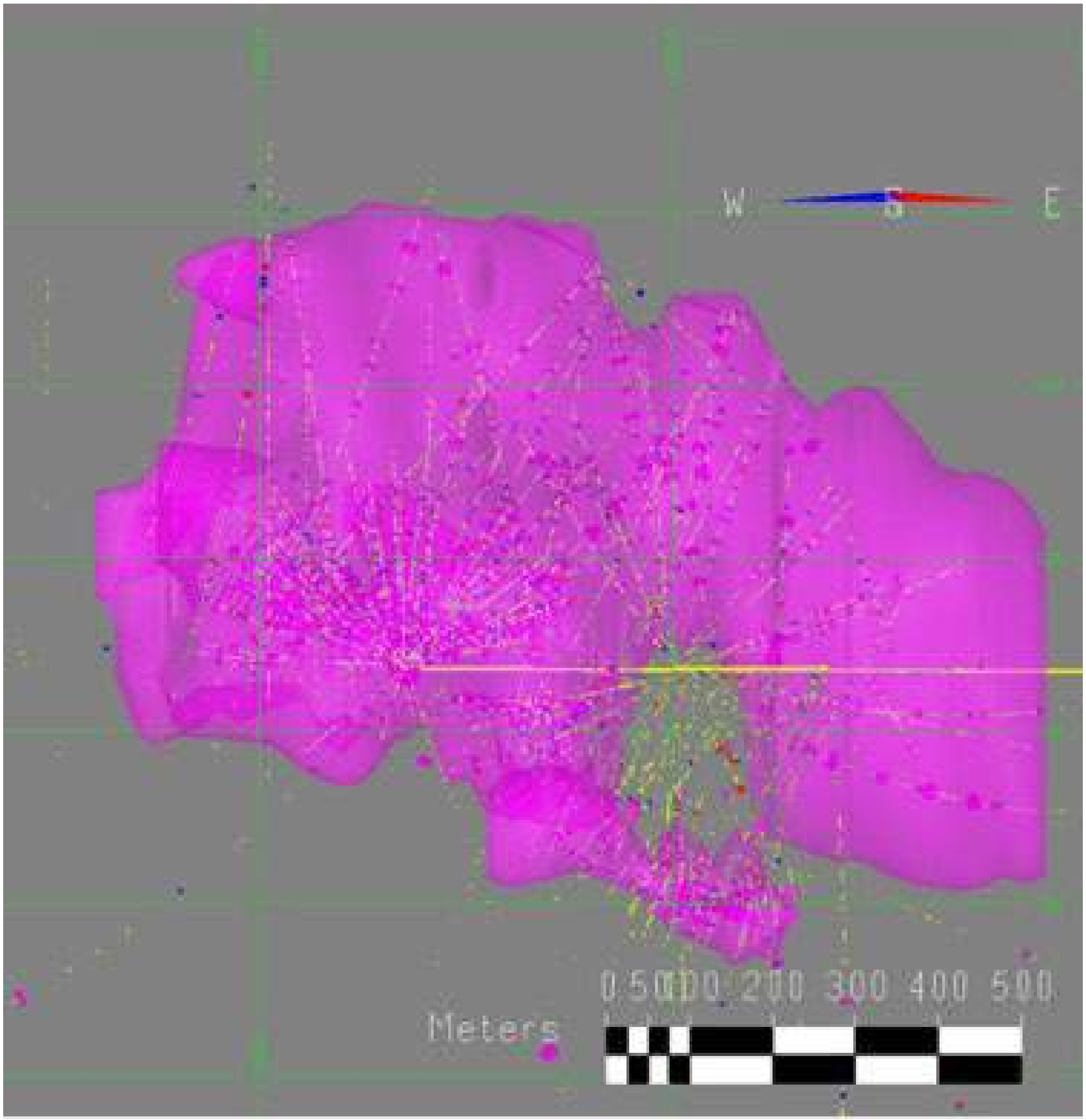


Figure 14.6. Showing 3D Wire Frame Built Around 0.10% MoS_2 Intercepts
Source: AMPL, 2023

14.2.3 Composites

For the current Resource estimate, a mineralised solid was constructed around a roughly designed and manually constrained 0.1% MoS_2 grade shell to constrain the estimate. Two sets of composites were created. Set one used 5 m composites and was limited by the 0.1% MoS_2 grade shell. The second set of composites involved entire length composites by each zone. The second set is listed in Table 14.1, below.

TABLE 14.1
LIST OF INTERCEPTS USED IN THE MODEL (MAIN LENS)

DH-ID	LENS	FROM	-TO-	LENGTH	MOS_2	DH-ID	LENS	FROM	-TO-	LENGTH	MOS_2
10	4	14.63	174.04	159.41	0.120	65	4	91.40	527.30	435.90	0.200
12	4	5.49	305.49	300.00	0.150	67	4	0.00	64.01	64.01	0.300
12	4	305.49	593.45	287.96	0.110	70	4	0.00	146.30	146.30	0.140
13	4	304.80	381.00	76.20	0.170	71	4	204.22	257.74	53.52	0.130
14	4	67.06	286.51	219.45	0.100	72	4	182.80	286.51	103.71	0.180
15	4	83.78	433.73	349.95	0.120	73	4	76.20	245.06	168.86	0.230
16	4	6.10	338.33	332.23	0.130	74	4	51.82	213.36	161.54	0.180
16	4	640.08	722.38	82.30	0.100	75	4	73.15	220.68	147.53	0.130
16	4	838.20	973.53	135.33	0.120	76	4	67.06	243.84	176.78	0.150
17	4	3.05	303.05	300.00	0.210	77	4	0.00	268.22	268.22	0.250
17	4	303.05	737.62	434.57	0.270	78	4	0.00	274.32	274.32	0.450
18	4	195.07	558.70	363.63	0.220	79	4	54.86	213.36	158.50	0.160
21	4	76.20	376.20	300.00	0.130	80	4	0.00	213.36	213.36	0.310
21	4	376.20	551.69	175.49	0.220	81	4	0.00	213.97	213.97	0.250
22	4	152.40	452.40	300.00	0.150	82	4	0.00	241.40	241.40	0.340
22	4	452.40	679.70	227.30	0.190	83	4	0.00	262.13	262.13	0.390
23	4	67.06	367.06	300.00	0.160	84	4	0.00	243.84	243.84	0.270
23	4	367.06	690.37	323.31	0.150	85	4	0.00	214.79	214.79	0.630
24	4	484.63	562.97	78.34	0.110	86	4	0.00	149.96	149.96	0.220
25	4	377.95	533.40	155.45	0.110	87	4	0.00	274.32	274.32	0.140
26	4	70.10	301.75	231.65	0.120	88	4	0.00	242.93	242.93	0.270
26	4	393.19	701.04	307.85	0.120	89	4	0.00	283.46	283.46	0.190
27	4	478.54	633.98	155.44	0.120	90	4	0.00	183.18	183.18	0.230
29	4	76.20	376.20	300.00	0.140	91	4	0.00	181.05	181.05	0.250
29	4	376.20	816.86	440.66	0.220	92	4	0.00	274.32	274.32	0.260
31	4	13.11	362.71	349.60	0.130	93	4	0.00	211.23	211.23	0.180
31	4	478.54	789.43	310.89	0.170	94	4	27.43	243.84	216.41	0.200
32	4	76.20	376.20	300.00	0.130	95	4	0.00	303.89	303.89	0.240
32	4	376.20	627.89	251.69	0.200	96	4	30.48	243.84	213.36	0.210
33	4	143.26	443.26	300.00	0.130	97	4	0.00	182.88	182.88	0.140
33	4	443.26	713.23	269.97	0.250	98	4	39.62	242.93	203.31	0.190
34	4	67.06	367.06	300.00	0.110	99	4	0.00	231.65	231.65	0.120
34	4	367.06	731.18	364.12	0.210	100	4	27.43	244.45	217.02	0.170
35	4	12.19	312.19	300.00	0.140	101	4	42.67	290.17	247.50	0.130
35	4	312.19	670.25	358.06	0.260	103	4	67.06	292.91	225.85	0.140
37	4	0.00	300.00	300.00	0.130	104	4	210.31	350.52	140.21	0.120
37	4	300.00	624.84	324.84	0.120	105	4	48.77	274.32	225.55	0.120
38	4	0.00	411.48	411.48	0.160	106	4	54.86	369.72	314.86	0.140
39	4	124.97	545.59	420.62	0.180	108	4	158.50	213.36	54.86	0.360
39	4	832.10	908.61	76.51	0.090	109	4	0.00	304.80	304.80	0.250
40	4	0.00	39.62	39.62	0.090	110	4	79.25	305.71	226.46	0.150
40	4	60.93	463.30	402.37	0.160	111	4	70.10	305.41	235.31	0.220
41	4	9.14	309.14	300.00	0.110	112	4	0.00	320.04	320.04	0.410
41	4	309.14	529.44	220.30	0.110	113	4	204.22	271.27	67.05	0.170
42	4	24.38	338.94	314.56	0.140	114	4	0.00	259.08	259.08	0.250
43	4	21.34	366.06	344.72	0.120	115	4	0.00	274.32	274.32	0.270
44	4	39.62	335.28	295.66	0.150	116	4	100.58	152.40	51.82	0.060
45	4	39.62	313.94	274.32	0.120	117	4	54.86	225.55	170.69	0.130
46	4	42.67	295.66	252.99	0.160	118	4	0.00	344.42	344.42	0.210
47	4	121.86	188.98	67.12	0.210	119	4	0.00	271.27	271.27	0.190
48	4	39.62	322.48	282.86	0.160	120	4	0.00	228.60	228.60	0.210
49	4	60.93	249.94	189.01	0.190	121	4	0.00	170.08	170.08	0.230
50	4	51.82	335.89	284.07	0.300	122	4	0.00	153.01	153.01	0.210
51	4	88.39	265.18	176.79	0.130	123	4	0.00	321.11	321.11	0.210
52	4	73.15	316.99	243.84	0.100	124	4	0.00	207.26	207.26	0.190
52	4	341.38	364.85	23.47	0.140	125	4	0.00	119.79	119.79	0.230
53	4	109.73	270.66	160.93	0.100	126	4	0.00	213.36	213.36	0.290
55	4	146.30	225.55	79.25	0.120	127	4	0.00	201.47	201.47	0.310
57	4	27.43	396.06	368.63	0.170	128	4	201.17	280.42	79.25	0.090
58	4	51.82	304.66	252.84	0.200	129	4	0.00	137.16	137.16	0.170
59	4	27.43	274.62	247.19	0.230	129	4	149.35	177.70	28.35	0.060
60	4	79.25	446.23	366.98	0.120	130	4	0.00	112.78	112.78	0.200
61	4	48.77	243.84	195.07	0.140	131	4	0.00	168.25	168.25	0.100
62	4	0.00	304.66	304.66	0.210	132	4	60.96	272.80	211.84	0.170
63	4	64.01	365.59	301.58	0.120	133	4	0.00	169.47	169.47	0.240
64	4	73.15	460.25	387.10	0.150	134	4	0.00	157.58	157.58	0.100

TABLE 14.1
LIST OF INTERCEPTS USED IN THE MODEL (MAIN LENS)
(CONTINUED)

DH-ID	LENS	FROM	-TO-	LENGTH	MOS_2	DH-ID	LENS	FROM	-TO-	LENGTH	MOS_2
135	4	0.00	84.12	84.12	0.100	177	4	243.84	265.18	21.34	0.120
136	4	0.00	67.06	67.06	0.160	181	4	27.43	158.19	130.76	0.170
137	4	0.00	125.82	125.82	0.150	182	4	0.00	194.98	194.98	0.240
138	4	0.00	163.37	163.37	0.140	183	4	0.00	214.79	214.79	0.450
140	4	0.00	108.81	108.81	0.180	184	4	0.00	95.97	95.97	0.250
142	4	243.84	344.42	100.58	0.270	185	4	0.00	145.39	145.39	0.280
143	4	246.89	344.42	97.53	0.220	186	4	0.00	99.01	99.01	0.300
146	4	259.08	347.31	88.23	0.120	187	4	0.00	105.11	105.11	0.350
147	4	0.00	128.32	128.32	0.270	188	4	0.00	106.68	106.68	0.390
148	4	0.00	146.00	146.00	0.190	189	4	0.00	163.46	163.46	0.460
149	4	0.00	128.02	128.02	0.330	190	4	0.00	216.10	216.10	0.200
150	4	0.00	128.63	128.63	0.300	191	4	0.00	166.04	166.04	0.330
151	4	0.00	148.13	148.13	0.240	192	4	0.00	203.00	203.00	0.240
152	4	0.00	119.79	119.79	0.250	193	4	0.00	170.69	170.69	0.240
153	4	0.00	147.83	147.83	0.340	194	4	0.00	214.79	214.79	0.220
154	4	0.00	153.01	153.01	0.270	195	4	27.43	175.56	148.13	0.160
155	4	0.00	153.92	153.92	0.330	196	4	0.00	228.50	228.50	0.200
156	4	0.00	198.12	198.12	0.370	197	4	259.08	344.42	85.34	0.170
157	4	0.00	183.18	183.18	0.370	198	4	304.80	338.33	33.53	0.160
158	4	0.00	152.40	152.40	0.380	199	4	277.37	344.42	67.05	0.160
159	4	0.00	121.92	121.92	0.210	200	4	252.98	313.94	60.96	0.130
160	4	51.82	121.92	70.10	0.170	201	4	258.96	271.27	12.31	0.080
161	4	21.34	123.44	102.10	0.200	201	4	298.70	335.28	36.58	0.240
162	4	27.43	134.11	106.68	0.300	202	4	243.84	280.42	36.58	0.130
163	4	0.00	121.92	121.92	0.260	203	4	280.42	307.85	27.43	0.130
164	4	0.00	12.19	12.19	0.090	206	4	219.46	231.65	12.19	0.050
165	4	256.03	347.47	91.44	0.300	207	4	155.45	188.98	33.53	0.080
166	4	243.84	326.14	82.30	0.250	209	4	158.50	204.22	45.72	0.050
167	4	231.65	331.32	99.67	0.210	210	4	112.78	219.46	106.68	0.080
168	4	228.50	310.90	82.40	0.190	211	4	182.88	262.13	79.25	0.200
169	4	207.26	298.70	91.44	0.270	213	4	173.74	219.46	45.72	0.170
170	4	201.17	307.85	106.68	0.130	214	4	91.44	256.34	164.90	0.120
171	4	237.74	277.37	39.63	0.150	217	4	106.63	213.36	106.73	0.260
173	4	225.55	304.80	79.25	0.170	218	4	76.17	237.74	161.57	0.160
174	4	252.98	353.57	100.59	0.200	219	4	67.06	265.18	198.12	0.180
175	4	256.03	341.38	85.35	0.100	82A	4	0.00	45.72	45.72	0.300

These were identified as Lens 4. Material outside of the main wire frame was identified as Lens 5. Composites 5 m in length were created between these boundaries. In addition, average grade composites were created for intersections/piercements of the wire frame (indicated as Lens 4). The statistics for these composites are shown below in Figure 14.7 and Figure 14.8, below.

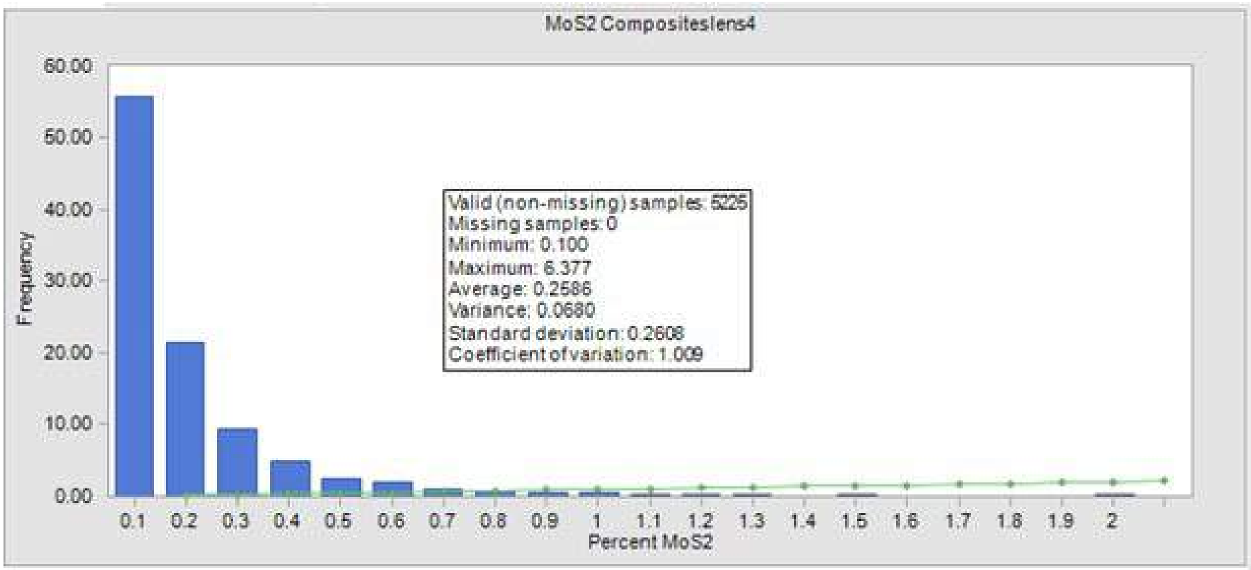


Figure 14.7. ***Main Zone (Lens 4) Histogram and Statistics Based on Composites – Main Zone (Lens 4) Grade Tonnage Curve (Frequency)and Statistics Using Composites***
Source: AMPL, 2023

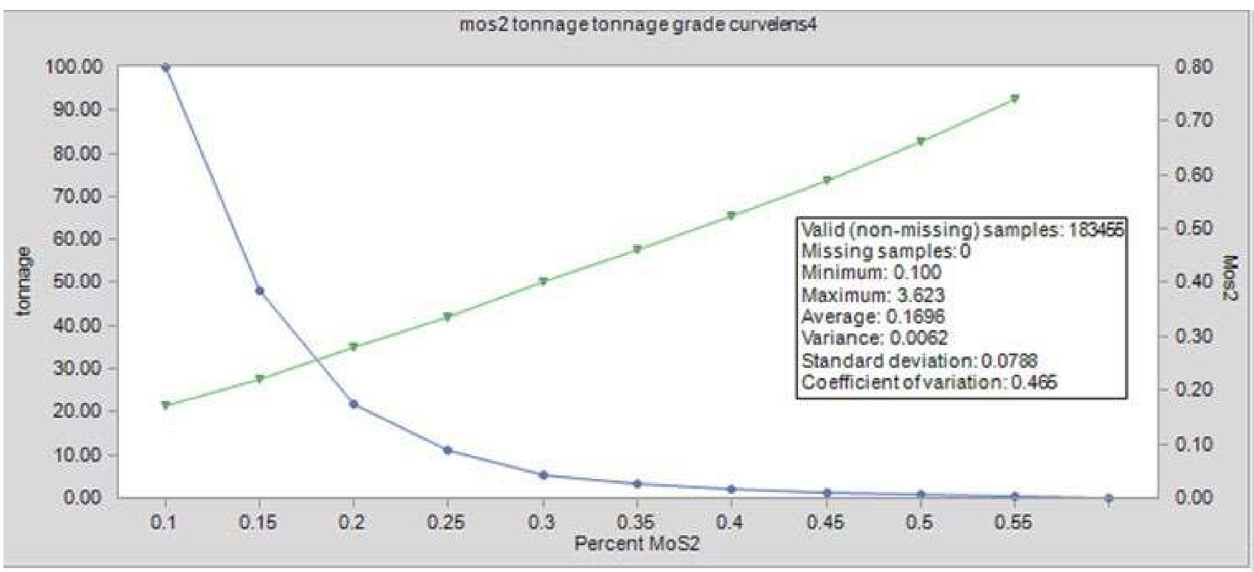


Figure 14.8. ***Main Zone (Lens 4) Grade Tonnage Curve (Frequency)and Statistics***
Source: AMPL, 2023

Table 14.2, below, was created for intervals outside of the zone of influence created by the wire frame model.

TABLE 14.2
INTERVALS OUTSIDE OF THE ZONE OF INFLUENCE

DH-ID	LENS	FROM	-TO-	LENGTH	MOS_2	DH-ID	LENS	FROM	-TO-	LENGTH	MOS_2
1	5	0.00	153.62	153.62	0.010	40	5	463.30	747.98	284.68	0.080
2	5	0.00	188.37	188.37	0.010	41	5	0.00	9.14	9.14	0.100
3	5	0.00	144.78	144.78	0.010	42	5	0.00	24.38	24.38	0.060
4	5	0.00	154.84	154.84	0.010	43	5	0.00	21.34	21.34	0.080
5	5	0.00	144.17	144.17	0.070	44	5	0.00	39.62	39.62	0.080
6	5	0.00	135.33	135.33	0.000	45	5	0.00	39.62	39.62	0.090
7	5	0.00	241.10	241.10	0.060	46	5	0.00	42.67	42.67	0.080
8	5	0.00	204.83	204.83	0.080	47	5	0.00	121.86	121.86	0.100
9	5	0.00	232.26	232.26	0.050	47	5	188.98	213.36	24.38	0.080
10	5	0.00	14.63	14.63	0.050	48	5	0.00	39.62	39.62	0.090
10	5	174.04	229.21	55.17	0.050	49	5	0.00	60.93	60.93	0.080
11	5	0.00	97.84	97.84	0.050	49	5	249.94	306.63	56.69	0.070
12	5	0.00	5.49	5.49	0.110	50	5	0.00	51.82	51.82	0.050
13	5	0.00	304.80	304.80	0.070	51	5	0.00	88.39	88.39	0.050
13	5	381.00	593.75	212.75	0.050	51	5	265.18	278.89	13.71	0.140
14	5	0.00	67.06	67.06	0.080	52	5	0.00	73.15	73.15	0.080
14	5	286.51	610.51	324.00	0.080	52	5	316.99	341.38	24.39	0.050
15	5	0.00	83.78	83.78	0.090	53	5	0.00	109.73	109.73	0.080
16	5	0.00	6.10	6.10	0.030	54	5	0.00	301.45	301.45	0.060
16	5	338.33	640.08	301.75	0.060	55	5	0.00	146.30	146.30	0.080
16	5	722.38	838.20	115.82	0.070	55	5	225.55	282.24	56.69	0.040
17	5	0.00	3.05	3.05	0.000	56	5	0.00	335.13	335.13	0.070
17	5	737.62	765.05	27.43	0.090	57	5	0.00	27.43	27.43	0.120
18	5	0.00	195.07	195.07	0.060	58	5	0.00	51.82	51.82	0.100
19	5	0.00	300.00	300.00	0.030	59	5	0.00	27.43	27.43	0.060
19	5	300.00	603.81	303.81	0.040	60	5	0.00	79.25	79.25	0.130
20	5	0.00	352.04	352.04	0.050	61	5	0.00	48.77	48.77	0.080
21	5	0.00	76.20	76.20	0.060	63	5	0.00	64.01	64.01	0.060
21	5	551.69	637.64	85.95	0.080	64	5	0.00	73.15	73.15	0.070
22	5	0.00	152.40	152.40	0.070	64	5	460.25	500.48	40.23	0.060
22	5	679.70	800.10	120.40	0.070	65	5	0.00	91.40	91.40	0.070
23	5	0.00	67.06	67.06	0.090	65	5	527.30	578.85	51.55	0.090
24	5	0.00	300.00	300.00	0.020	66	5	0.00	300.00	300.00	0.060
24	5	300.00	484.63	184.63	0.070	66	5	300.00	518.77	218.77	0.050
25	5	0.00	377.95	377.95	0.040	67	5	64.01	364.01	300.00	0.070
25	5	533.40	560.83	27.43	0.070	67	5	364.01	664.01	300.00	0.070
26	5	0.00	70.10	70.10	0.060	67	5	664.01	916.23	252.22	0.010
26	5	301.75	393.19	91.44	0.070	68	5	0.00	300.00	300.00	0.050
26	5	701.04	858.32	157.28	0.080	68	5	300.00	617.22	317.22	0.070
27	5	0.00	300.00	300.00	0.080	69	5	0.00	300.00	300.00	0.050
27	5	300.00	478.54	178.54	0.090	69	5	300.00	678.48	378.48	0.050
27	5	633.98	868.98	235.00	0.060	70	5	146.30	446.30	300.00	0.050
28	5	0.00	300.00	300.00	0.050	70	5	446.30	746.30	300.00	0.030
28	5	300.00	600.00	300.00	0.040	70	5	746.30	945.79	199.49	0.010
28	5	600.00	868.98	268.98	0.040	71	5	0.00	204.22	204.22	0.050
29	5	0.00	76.20	76.20	0.060	72	5	0.00	182.80	182.80	0.050
29	5	816.86	949.76	132.90	0.090	72	5	286.51	586.51	300.00	0.060
30	5	0.00	300.00	300.00	0.060	72	5	586.51	754.68	168.17	0.030
30	5	300.00	697.38	397.38	0.060	73	5	0.00	76.20	76.20	0.030
31	5	0.00	13.11	13.11	0.070	74	5	0.00	51.82	51.82	0.050
31	5	362.71	478.54	115.83	0.070	75	5	0.00	73.15	73.15	0.060
31	5	789.43	795.22	5.79	0.070	76	5	0.00	67.06	67.06	0.080
32	5	0.00	76.20	76.20	0.040	77	5	268.22	301.75	33.53	0.060
32	5	627.89	769.62	141.73	0.050	79	5	0.00	54.86	54.86	0.050
33	5	0.00	143.26	143.26	0.050	83	5	262.13	274.32	12.19	0.030
33	5	713.23	739.44	26.21	0.040	89	5	283.46	304.80	21.34	0.070
34	5	0.00	67.06	67.06	0.050	94	5	0.00	27.43	27.43	0.050
35	5	0.00	12.19	12.19	0.100	96	5	0.00	30.48	30.48	0.040
36	5	0.00	445.92	445.92	0.070	98	5	0.00	39.62	39.62	0.060
37	5	624.84	916.84	292.00	0.060	100	5	0.00	27.43	27.43	0.070
38	5	411.48	711.48	300.00	0.050	101	5	0.00	42.67	42.67	0.120
38	5	711.48	992.12	280.64	0.060	102	5	0.00	300.00	300.00	0.040
39	5	0.00	124.97	124.97	0.050	102	5	300.00	485.55	185.55	0.040
39	5	545.59	832.10	286.51	0.060	103	5	0.00	67.06	67.06	0.060
40	5	39.62	60.93	21.31	0.090	104	5	0.00	210.31	210.31	0.050

14.3 SEMI-VARIOGRAM ANALYSIS

Three dimensional variograms were generated using MSDA™ software, an add on program to MineSight™/Hexagon™/MinePlan™ software (see Figure 14.9, below).

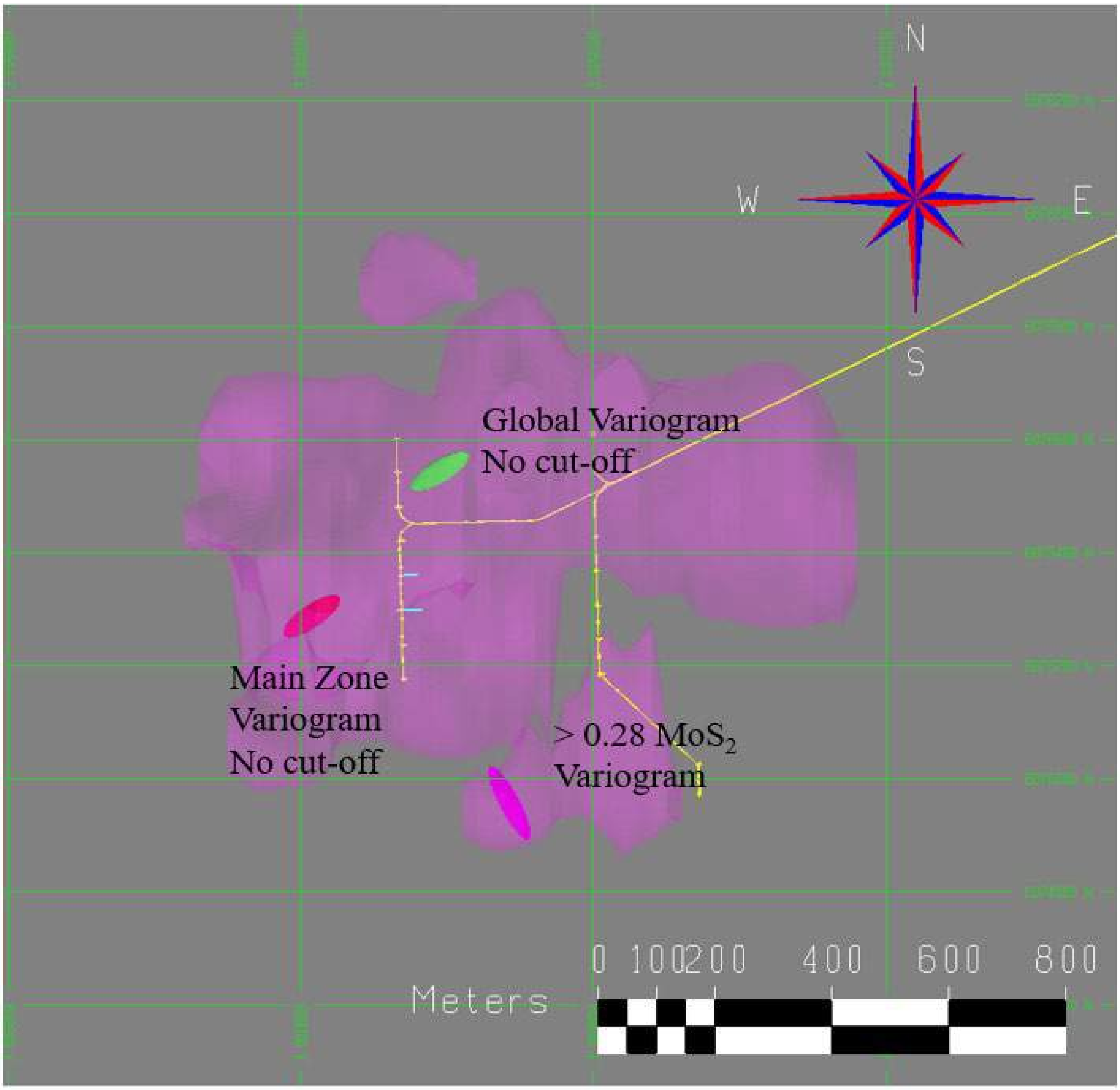


Figure 14.9. Variograms in Plan View
Source: AMPL, 2023

The importance of these variograms may have ramifications – previous models have generally referred to two semi flat zones – the variograms would appear to indicate that there is a very significant near vertical component to the mineralisation – with selective mining, such as room and pillar, this may be of considerable concern (see Figure 14.10, below). For bulk mining, not as much but should be considered as having a possible impact on zonation of the Resource as well as orientation of diamond drilling.

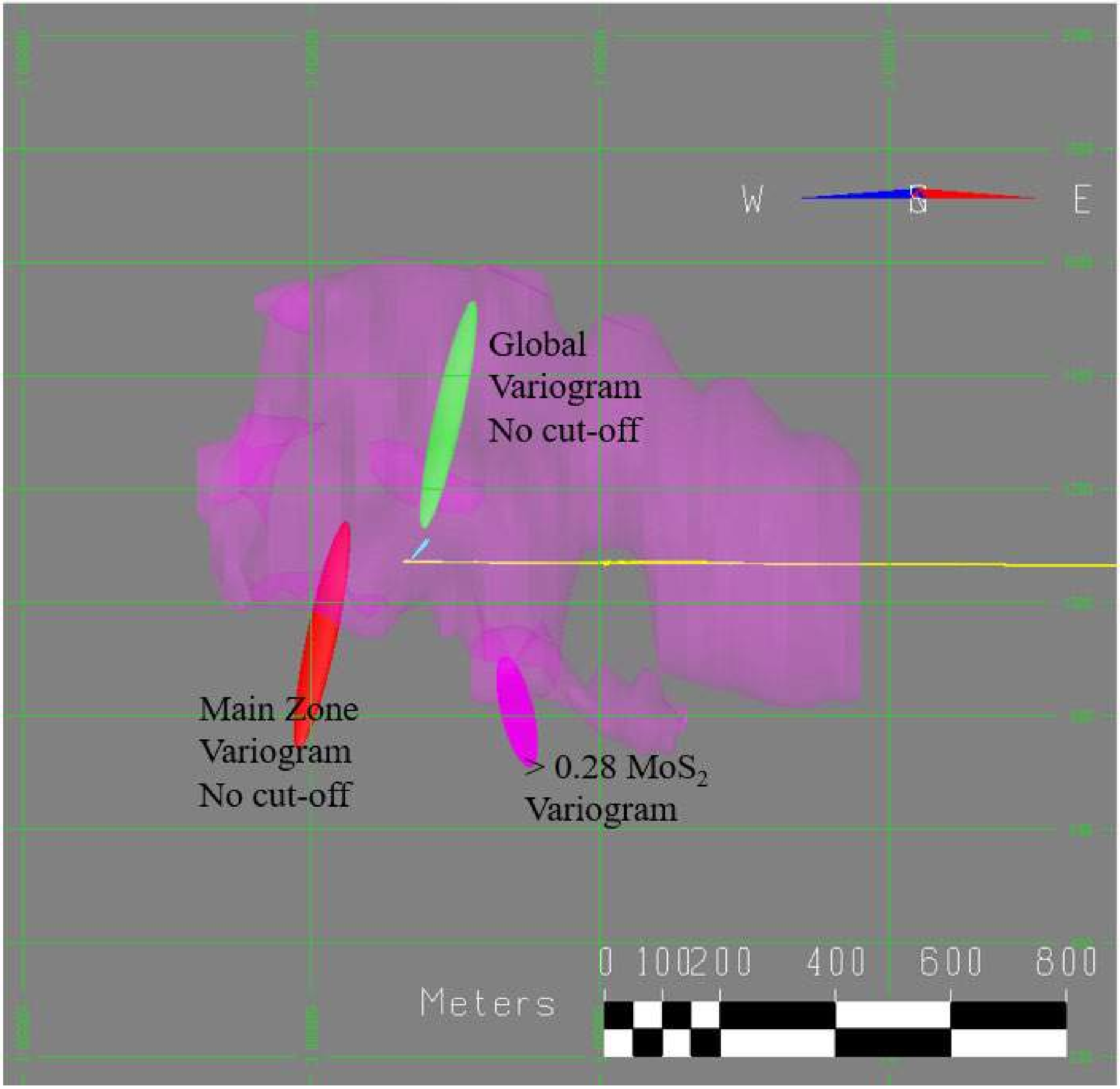


Figure 14.10. Sections Looking North of the Variograms
Source: AMPL, 2023

The orientation of the variograms in section is also interesting. Modelling has generally considered the deposit to have two semi-flat lenses. The variograms seem to indicate a strong vertical component to the zonation of mineralisation. Again, the ramifications could result in under-reporting of mineralisation since much of the drilling tends to mimic the orientation of the mineralisation with the assumption that the zonation is flat and not vertical.

14.4 BLOCK MODEL

A 3D block model was created using MineSight/™Hexagon™/MinePlan™ software. The dimensions are given in Figure 14.11, below.

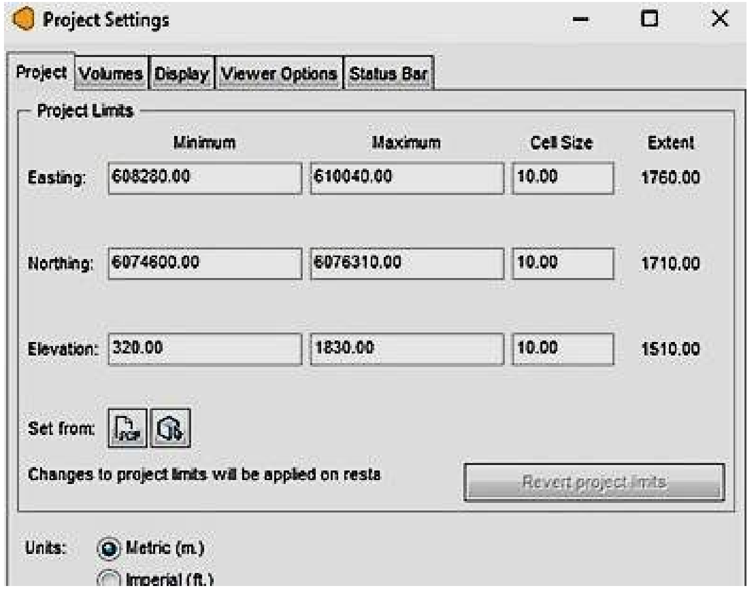


Figure 14.11. 3D Block Model Dimensions
Source: AMPL, 2023

The block model consists of blocks 10×10×10 m in dimension. Earlier models were 50×50×25-ft and 15×15×5 m. Due to the apparent near vertical dimension of the variograms, the smaller vertical component was expanded to 10 m.

14.5 GRADE INTERPOLATION

Two lens codes (domains) were created. Lens 4 was material inside what was a manually created 0.1% MoS_2 shell. Lens 5 was material outside the shell. Where both lenses occupied the same block, preference was given to the material inside the 0.1% MoS_2 shell (*i.e.*, overwrote Lens 5).

Grades for MoS_2 were interpolated in three passes – the first pass would generally be used to calculate "Inferred Resource" (see Figure 14.12, below).

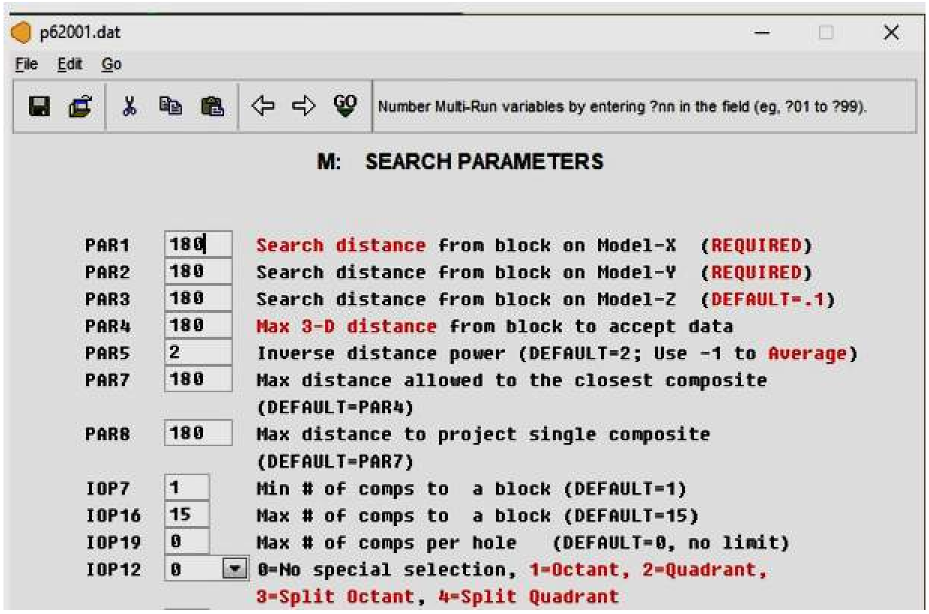


Figure 14.12. "First Pass"

Source: AMPL, 2023

The second pass would generally be used to calculate "Indicated Resource" – a two hole minimum was applied (see Figure 14.13, below).

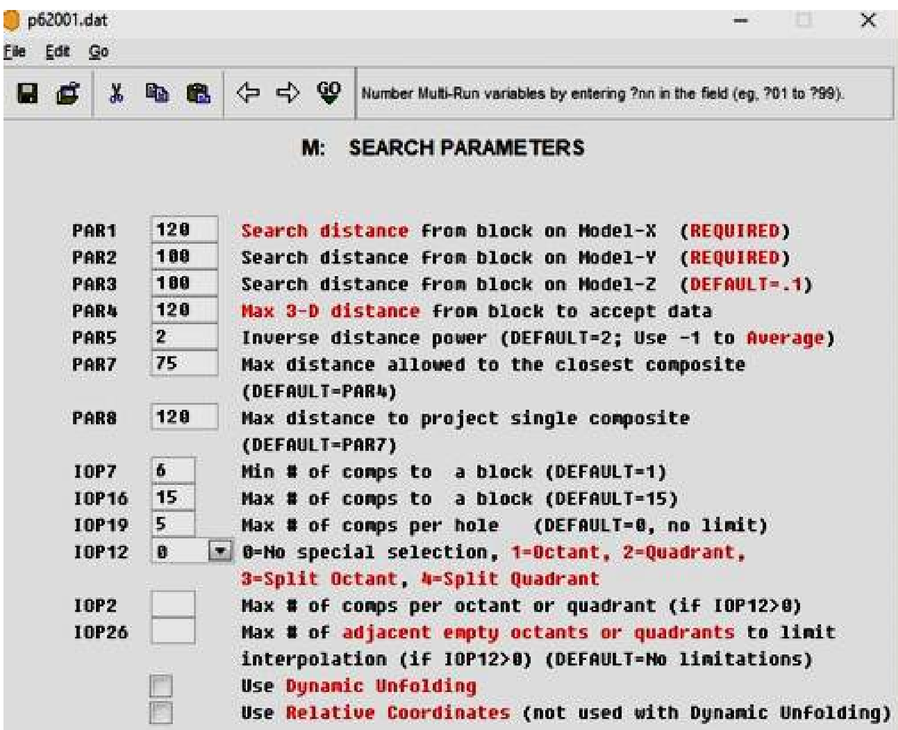


Figure 14.13. "Second Pass"

Source: AMPL, 2023

The third pass would generally be used to calculate "Measured Resource" – two hole minimum and ID3 (see Figure 14.14, below).

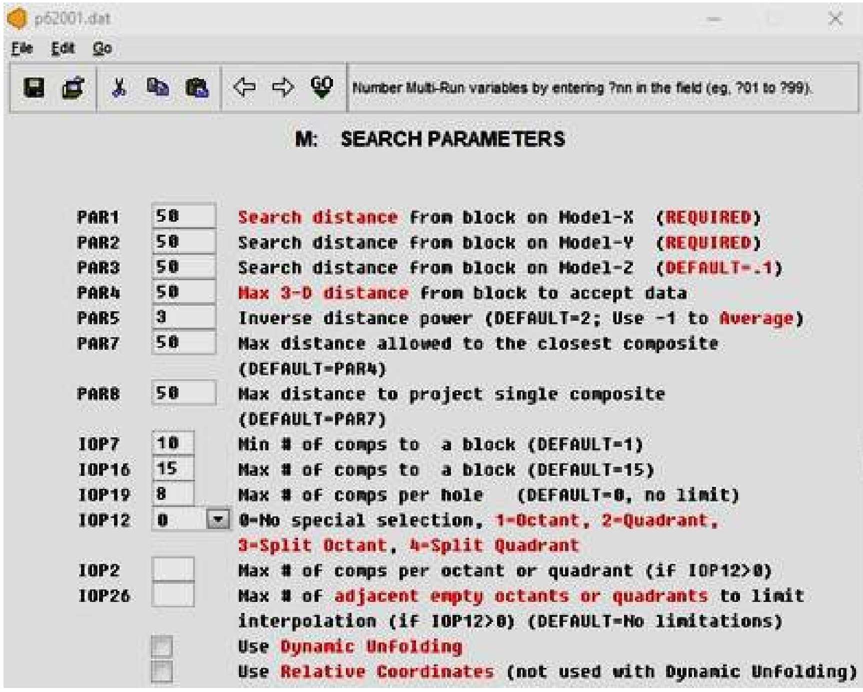


Figure 14.14. "Third Pass"

Source: AMPL, 2023

14.6 BULK DENSITY

During the site visit by Mr. Salmabadi, a significant amount of measured specific gravity (SG) measurements was found in the records. This correlated to work reported to Giroux (2016). He assigned an average SG of 2.66 to rock. An internet search gives an average SG 2.6 to 2.7 for granodiorite as well. Therefore, a SG of 2.66 was used.

14.7 RESULTS

Tungsten was modeled but not reported. Firstly, there are a significant number of missing assays that would have to be entered as null values negatively impacting the calculated grade. Secondly, the grade previously reported of 0.036 WO_3 is very low and the author has concerns whether this small amount could be economically recovered and if it was recovered, whether it would be saleable as a concentrate of sufficient quality. Contamination of WO_3 with molybdenum is always a concern.

14.8 CLASSIFICATION

Based on the study herein reported, delineated mineralisation of the Property is classified as a Resource according to the following definitions from National Instrument 43-101 and from CIM (2014).

> *In this Instrument, the terms "Mineral Resource", "Inferred Mineral Resource", "Indicated Mineral Resource" and "Measured Mineral Resource" have the meanings ascribed to those terms by the Canadian Institute of Mining, Metallurgy and Petroleum, as*

> *the CIM Definition Standards (May 2014) on Mineral Resources and Mineral Reserves adopted by CIM Council, as those definitions may be amended.*

The terms Measured, Indicated, and Inferred are defined by CIM (2014) as follows.

> *A Mineral Resource is a concentration or occurrence of solid material of economic interest in or on the Earth's crust in such form, grade or quality and quantity that there are reasonable prospects for eventual economic extraction. The location, quantity, grade or quality, continuity and other geological characteristics of a Mineral Resource are known, estimated or interpreted from specific geological evidence and knowledge, including sampling.*
>
> *The term Mineral Resource covers mineralization and natural material of intrinsic economic interest which has been identified and estimated through exploration and sampling and within which Mineral Reserves may subsequently be defined by the consideration and application of Modifying Factors. The phrase 'reasonable prospects for economic extraction' implies a judgment by the Qualified Person in respect of the technical and economic factors likely to influence the prospect of economic extraction.*
>
> *The Qualified Person should consider and clearly state the basis for determining that the material has reasonable prospects for eventual economic extraction. Assumptions should include estimates of cut-off grade and geological continuity at the selected cut-off, metallurgical recovery, smelter payments, commodity price or product value, mining and processing method and mining, processing and general and administrative costs. The Qualified Person should state if the assessment is based on any direct evidence and testing. Interpretation of the word 'eventual' in this context may vary depending on the commodity or mineral involved. For example, for some coal, iron, potash deposits and other bulk minerals or commodities, it may be reasonable to envisage 'eventual economic extraction' as covering time periods in excess of 50 years. However, for many gold deposits, application of the concept would normally be restricted to perhaps 10 to 15 years, and frequently to much shorter periods of time.*

Inferred Mineral Resource

> *An 'Inferred Mineral Resource' is that part of a Mineral Resource for which quantity and grade or quality are estimated on the basis of limited geological evidence and sampling. Geological evidence is sufficient to imply but not verify geological and grade or quality continuity. An Inferred Mineral Resource has a lower level of confidence than that applying to an Indicated Mineral Resource and must not be converted to a Mineral Reserve. It is reasonably expected that the majority of Inferred Mineral Resources could be upgraded to Indicated Mineral Resources with continued exploration.*
>
> *An 'Inferred Mineral Resource' is based on limited information and sampling gathered through appropriate sampling techniques from locations such as outcrops, trenches, pits, workings and drill holes. Inferred Mineral Resources must not be included in the economic analysis, production schedules, or estimated mine life in publicly disclosed Pre-Feasibility or Feasibility Studies, or in the Life of Mine plans and cash flow models of developed mines. Inferred Mineral Resources can only be used in economic studies as provided under NI 43-101.*

There may be circumstances, where appropriate sampling, testing, and other measurements are sufficient to demonstrate data integrity, geological and grade/quality continuity of a Measured or Indicated Mineral Resource, however, quality assurance and quality control, or other information may not meet all industry norms for the disclosure of an Indicated or Measured Mineral Resource. Under these circumstances, it may be reasonable for the Qualified Person to report an Inferred Mineral Resource if the Qualified Person has taken steps to verify the information meets the requirements of an Inferred Mineral Resource.

Indicated Mineral Resource

An 'Indicated Mineral Resource' is that part of a Mineral Resource for which quantity, grade or quality, densities, shape and physical characteristics are estimated with sufficient confidence to allow the application of Modifying Factors in sufficient detail to support mine planning and evaluation of the economic viability of the deposit. Geological evidence is derived from adequately detailed and reliable exploration, sampling and testing and is sufficient to assume geological and grade or quality continuity between points of observation. An Indicated Mineral Resource has a lower level of confidence than that applying to a Measured Mineral Resource and may only be converted to a Probable Mineral Reserve.

Mineralization may be classified as an Indicated Mineral Resource by the Qualified Person when the nature, quality, quantity and distribution of data are such as to allow confident interpretation of the geological framework and to reasonably assume the continuity of mineralization. The Qualified Person must recognize the importance of the Indicated Mineral Resource category to the advancement of the feasibility of the project. An Indicated Mineral Resource estimate is of sufficient quality to support a Preliminary Feasibility Study which can serve as the basis for major development decisions.

Measured Mineral Resource

A Measured Mineral Resource is that part of a Mineral Resource for which quantity, grade or quality, densities, shape, and physical characteristics are estimated with confidence sufficient to allow the application of Modifying Factors to support detailed mine planning and final evaluation of the economic viability of the deposit. Geological evidence is derived from detailed and reliable exploration, sampling and testing and is sufficient to confirm geological and grade or quality continuity between points of observation. A Measured Mineral Resource has a higher level of confidence than that applying to either an Indicated Mineral Resource or an Inferred Mineral Resource. It may be converted to a Proven Mineral Reserve or to a Probable Mineral Reserve.

Mineralization or other natural material of economic interest may be classified as a Measured Mineral Resource by the Qualified Person when the nature, quality, quantity and distribution of data are such that the tonnage and grade or quality of the mineralization can be estimated to within close limits and that variation from the estimate would not significantly affect potential economic viability of the deposit. This category requires a high level of confidence in, and understanding of, the geology and controls of the mineral deposit.

Modifying Factors

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

14.8.1 Reported Tonnages and Grade

A flow chart (Figure 14.15, below) shows how the Resource estimate tonnage and grade were reported. From an initial gross in-situ tonnage of over 813 Mt, the deposit was broken down into tonnage within the 0.1% MoS_2 wire frame (Main zone) and material outside the Main zone (Outliers). The tonnage was then broken down to material that met a minimum grade of 0.30% MoS_2, as 0.30% Mo was chosen as the cut-off grade for this report. Using the calculations of "Measured", "Indicated", and "Inferred", as discussed in Section 14.5, each zone was then further broken down to one of these categories. Measured and Indicated, outside the Main zone, was then automatically downgraded to Inferred and Inferred was not reported. This was done since continuity was difficult to establish. The Measured and Indicated and Inferred for the Main zone were reported as calculated.

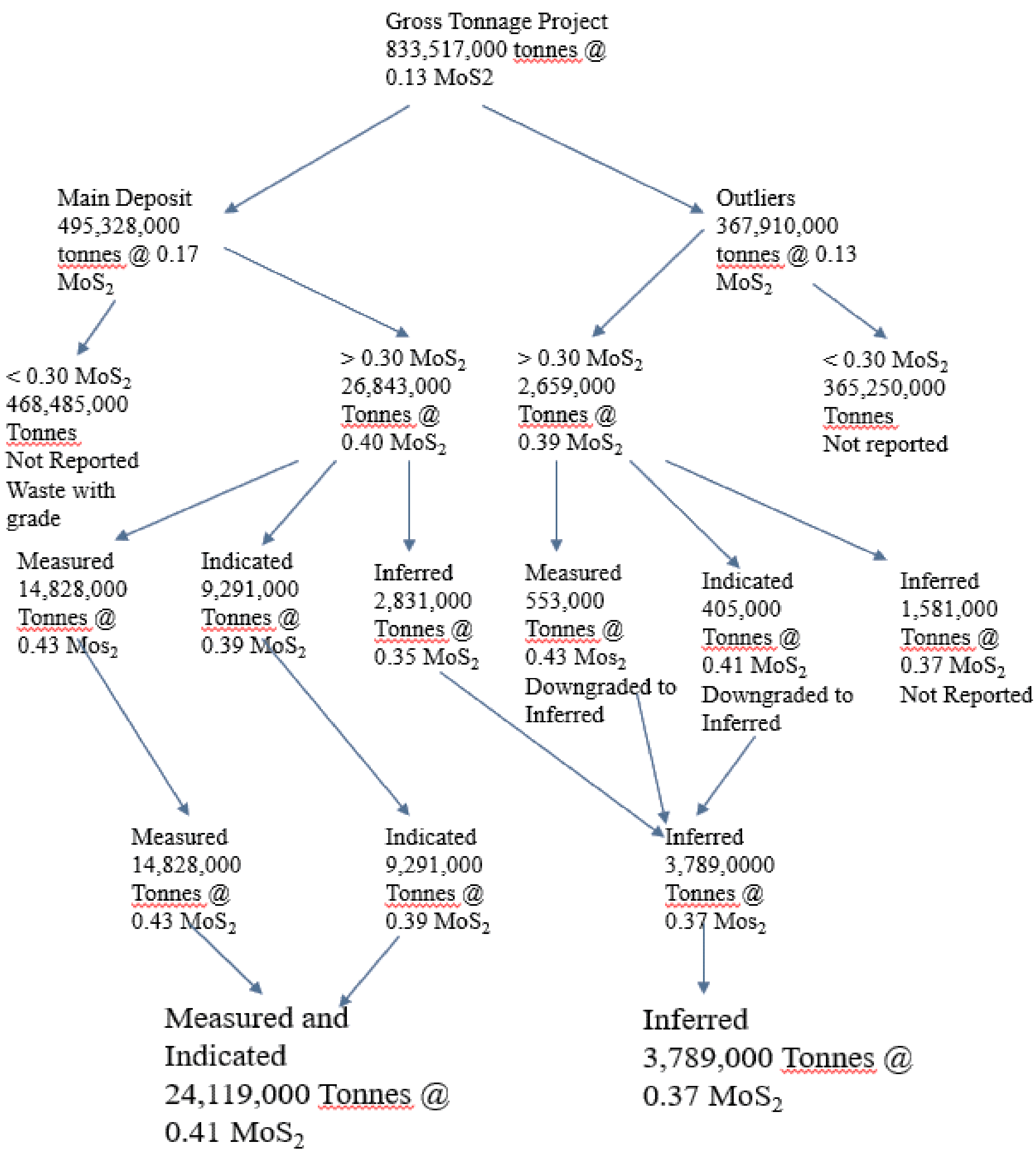


Figure 14.15. Flow Chart of Results
Source: AMPL, 2023

14.8.2 Calculation of Cut-off Grade

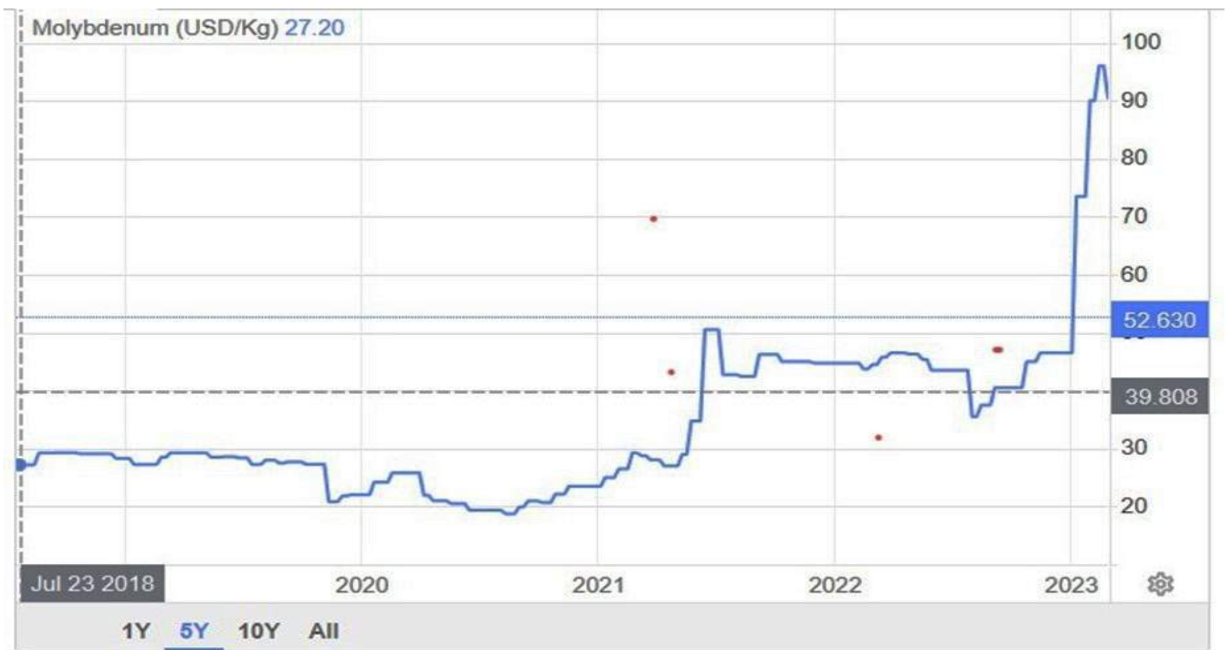
Based on the authors' experience, a >0.3% MoS_2 cut-off was chosen as the in-situ ore value as this cut-off should be below all in costs for a bulk tonnage underground mine. For the purposes of the Resource calculation, the following assumptions were made (see Table 14.3 and Figure 14.16, below).

TABLE 14.3 ASSUMPTIONS USED FOR CALCULATION OF CUT-OFF GRADE	
Molybdenum Price per kg	US$39.00
Molybdenum Price per lb	US$17.73
% Molybdenum in MoS_2	59.94%
Price per lb	$17.73
Recovery	90%
Exchange Rate (US$:CA$)	1.33
Estimated In-situ Value	CA$83.95

Figure 14.16, below, shows the molybdenum price in US$ per kg (2.205 lbs/kg) for the past 5 years with the average price over this period being US$39.81 per kg. Based on this price history, a long-term price for this report of US$39 per kg of molybdenum was selected for the cut-off grade calculation.

Using the parameters presented in Table 14.3, the calculated in-situ value at a cut-off grade of 0.3% MoS_2 is as follows:

Grade/100 × 1,000 kg/tonne × 59.94 (% Mo in MoS_2) × 0.90 % recovery/100 × US$39/kg × 1.33 = CA$83.95 In-situ Value



The Molybdenum contract is made available for trading with a unit of measure of dollars per pound. One contract unit represents 1,322.77 pounds, the equivalent of 60% of one metric ton. The unit size was chosen to reflect standard trading terms in the physical molybdenum market, in which molybdenum oxide is priced on a "molybdenum contained" basis. Typically, the molybdenum content of molybdenum oxide is around 60%. This means that traders can trade the equivalent size in the futures size without having to adjust their futures trade for the "molybdenum contained" factor.

Figure 14.16. Molybdenum Prices from 2005 to 2023

Source: www.tradingeconomics.com

14.9 RESULTS

Table 14.4, below, gives the in-situ values for all the cut-off grades considered up to the base case. Depending upon the mining method chosen, all cut-off grades have a reasonable prospect for eventual economic extraction. The base case for extraction of >0.3% MoS_2 was estimated for a large underground mining operation. As the deposit outcrops on the surface, lower grades were included for possible extraction by open pit mining or underground caving methods. No mining method has been selected at this time as no economic evaluations have been done.

TABLE 14.4
IN-SITU VALUES AT VARIOUS CUT-OFF GRADES

Cut-off Grade MoS_2	5 Year Average In-situ Value	In-situ Value at Current Pricing	Possible Extraction Method
>0.10	$27.98	$41.61	Surface/Cave
>0.15	$41.97	$62.42	Surface/Cave
>0.20	$55.96	$83.23	Underground/Cave
>0.25	$69.95	$104.03	Underground/Cave
>0.30	$83.95	$124.84	Underground

In the tables shown below, all tonnages have been rounded to the nearest 1,000 tonnes.

Table 14.5 and Table 14.6, below, present the Measured and Indicated Mineral Resources for the Property at various cut-off grades.

TABLE 14.5
MEASURED MINERAL RESOURCES

Category	Cut-off Grade MoS_2	Tonnes	Grade MoS_2	Grade Mo	Contained Mo kg
Measured	>0.10	93,480,000	0.22	0.13	123,300,000
Measured	>0.15	63,523,000	0.26	0.16	99,000,000
Measured	>0.20	39,884,000	0.31	0.19	74,100,000
Measured	>0.25	24,269,000	0.37	0.22	53,800,000
Measured	>0.30	14,828,000	0.43	0.26	37,900,000
Measured	>0.35	9,404,000	0.49	0.29	27,600,000
Measured	>0.40	6,127,000	0.55	0.33	20,200,000
Measured	>0.45	4,006,000	0.61	0.37	14,600,000

TABLE 14.6 INDICATED MINERAL RESOURCES					
Category	**Cut-off Grade MoS_2**	**Tonnes**	**Grade MoS_2**	**Grade Mo**	**Contained Mo kg**
Indicated	>0.10	197,999,000	0.17	0.1	201,800,000
Indicated	>0.15	97,533,000	0.21	0.13	122,800,000
Indicated	>0.20	43,625,000	0.27	0.16	70,600,000
Indicated	>0.25	19,627,000	0.32	0.19	37,600,000
Indicated	>0.30	9,291,000	0.39	0.23	21,500,000
Indicated	>0.35	5,277,000	0.43	0.26	13,600,000
Indicated	>0.40	2,912,000	0.48	0.29	8,400,000
Indicated	>0.45	1,619,000	0.54	0.32	5,200,000

Table 14.7 and Table 14.8, below, show various tonnage and cut-off grades that were considered for the report. Based on the cut-off grade shown in Table 14.7, 0.30% MoS_2 was considered the most reasonable. If average Mo prices are sustained at a higher level, as has been the case since 2021 and especially since 2022, as shown in Figure 14.16, then a much lower cut-off grade could be used. Table 14.7 shows tonnes for Measured and Indicated Mineral Resources. Table 14.8 shows tonnes for Inferred Resources.

TABLE 14.7 MEASURED AND INDICATED COMBINED RESOURCES					
Category	**Cut-off Grade MoS_2**	**Tonnes**	**Grade MoS_2**	**Grade Mo**	**Contained Mo kg**
Measured and Indicated	>0.0	394,623,000	0.15	0.09	354,800,000
Measured and Indicated	>0.10	291,479,000	0.18	0.11	314,500,000
Measured and Indicated	>0.15	161,056,000	0.23	0.14	222,000,000
Measured and Indicated	>0.20	83,509,000	0.29	0.17	145,200,000
Measured and Indicated	>0.25	43,896,000	0.35	0.21	92,100,000
Measured and Indicated	>0.30	24,119,000	0.41	0.25	59,400,000
Measured and Indicated	>0.35	14,681,000	0.47	0.28	41,400,000
Measured and Indicated	>0.40	9,039,000	0.53	0.32	28,700,000
Measured and Indicated	>0.45	5,625,000	0.59	0.35	19,900,000

TABLE 14.8 INFERRED RESOURCES					
Category	**Cut-off Grade MoS_2**	**Tonnes**	**Grade MoS_2**	**Grade Mo**	**Contained Mo kg**
Inferred	>0.0	502,849,000	0.10	0.06	301,400,000
Inferred	>0.10	225,817,000	0.15	0.09	203,000,000
Inferred	>0.15	78,990,000	0.20	0.12	94,700,000
Inferred	>0.20	25,039,000	0.26	0.15	39,000,000
Inferred	>0.25	11,907,000	0.30	0.18	21,400,000
Inferred	>0.30	3,789,000	0.37	0.22	8,400,000
Inferred	>0.35	1,786,000	0.42	0.25	4,500,000
Inferred	>0.40	677,000	0.50	0.30	2,000,000
Inferred	>0.45	404,000	0.55	0.33	1,300,000

14.10 COMPARISON TO PREVIOUS 2016 RESOURCES

Table 14.9 and Table 14.10, below, refer to the previous Mineral Resource by Giroux, G.H. (2016) – *Updated Technical Report* and *Resource Estimate, Davidson Molybdenum Deposit*, Smithers, British Columbia, Canada and are included for comparison only. These Mineral Resource estimates are viewed as historical Resources and have not been verified by a Qualified Person, as required by NI 43-101 and should not be relied upon. It is important to note that all these historical and previous Mineral Resource estimates are superseded by the Updated Mineral Resource estimate presented in Section 14.0 of this Technical Report.

TABLE 14.9
2016 MEASURED AND INDICATED RESOURCE

MoS_2 Cut-off (%)	Tonnes>Cut-off (tonnes)	Grade>Cut-off			Million Pounds Mo	Million Pounds MoO_3	Million Pounds WO_3
		MoS_2 (%)	Mo (%)	WO_3 (%)			
0.19	103,610,000	0.274	0.164	0.034	375.21	562.82	77.68
0.20	**90,080,000**	**0.286**	**0.171**	**0.034**	**340.50**	**510.75**	**67.53**
0.21	78,300,000	0.298	0.179	0.035	308.39	462.59	60.43
0.22	68,860,000	0.310	0.186	0.035	282.13	423.20	53.14
0.24	53,880,000	0.332	0.199	0.035	236.42	354.64	41.58
0.26	42,920,000	0.353	0.212	0.036	200.24	300.37	34.07
0.28	**34,420,000**	**0.374**	**0.224**	**0.036**	**170.14**	**255.21**	**27.32**
0.30	27,700,000	0.394	0.236	0.036	144.25	216.37	21.99
0.32	22,510,000	0.414	0.248	0.037	123.17	184.75	18.36
0.34	18,470,000	0.433	0.260	0.037	105.70	158.55	15.07
0.36	15,040,000	0.452	0.271	0.037	89.85	134.77	12.27
0.38	12,120,000	0.472	0.283	0.037	75.61	113.41	9.89
0.40	9,770,000	0.491	0.294	0.037	63.40	95.10	7.97
0.42	7,880,000	0.511	0.306	0.037	53.22	79.83	6.43
0.44	6,350,000	0.531	0.318	0.037	44.56	66.85	5.18
0.46	5,050,000	0.552	0.331	0.037	36.84	55.26	4.12

TABLE 14.10
2016 INFERRED RESOURCES

MoS_2 Cut-off (%)	Tonnes>Cut-off (tonnes)	Grade>Cut-off MoS_2 (%)	Grade>Cut-off Mo (%)	Grade>Cut-off WO_3 (%)	Million Pounds Mo	Million Pounds MoO_3	Million Pounds WO_3
0.19	12,750,000	0.237	0.142	0.034	39.94	59.91	9.56
0.20	**10,620,000**	**0.245**	**0.147**	**0.033**	**34.39**	**51.58**	**7.73**
0.21	8,520,000	0.256	0.153	0.033	28.83	43.24	6.20
0.22	6,880,000	0.265	0.159	0.032	24.10	36.15	4.85
0.24	4,250,000	0.288	0.173	0.032	16.18	24.27	3.00
0.26	2,740,000	0.310	0.186	0.031	11.23	16.84	1.87
0.28	**1,850,000**	**0.329**	**0.197**	**0.031**	**8.04**	**12.07**	**1.26**
0.30	1,300,000	0.347	0.208	0.031	5.96	8.94	0.89
0.32	880,000	0.365	0.219	0.031	4.25	6.37	0.60
0.34	610,000	0.380	0.228	0.032	3.06	4.60	0.43
0.36	410,000	0.397	0.238	0.032	2.15	3.23	0.29
0.38	250,000	0.413	0.248	0.032	1.36	2.05	0.18
0.40	130,000	0.434	0.260	0.033	0.75	1.12	0.09
0.42	80,000	0.451	0.270	0.034	0.48	0.72	0.06
0.44	40,000	0.474	0.284	0.035	0.25	0.38	0.03
0.46	20,000	0.509	0.305	0.034	0.13	0.20	0.01

A comparison between the 2016 Mineral Resource and the 2023 Mineral Resource is given in Table 14.11, below.

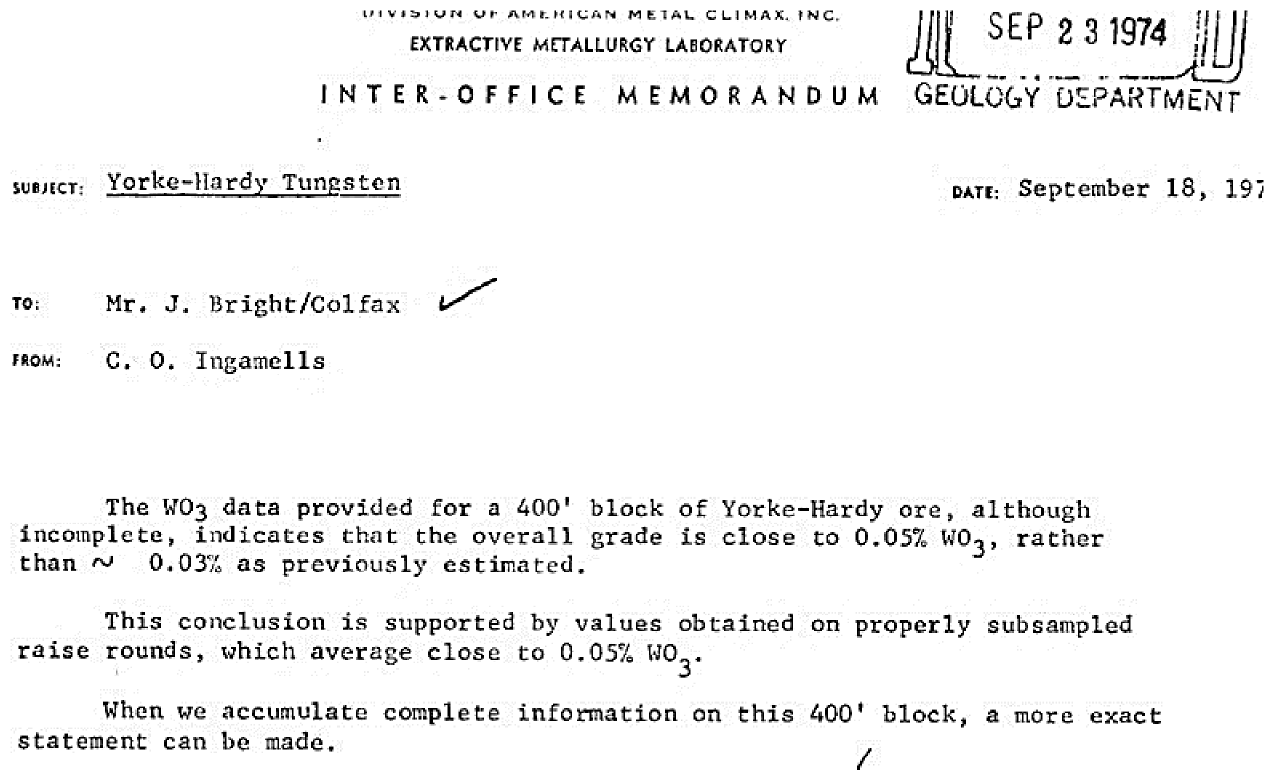
TABLE 14.11
COMPARISON OF 2023 RESOURCE TO PREVIOUS 2016 RESOURCE 0.30% USING MoS_2 CUT-OFF

	2023 Resource			2016 Resource		
	Tonnes	Grade MoS_2	Grade Mo	Tonnes	Grade MoS_2	Grade Mo
Measured and Indicated	24,119,000	0.41	0.25	27,700,000	0.39	0.24
Inferred	3,789,000	0.37	0.22	1,300,000	0.35	0.21

The MineSight™ model and resource estimated, generated from the drill hole data by AMPL, compares quite favourably with the previous resource estimate by Giroux. The difference in metal content is approximately 3%, which is essentially identical and well within the expected discrepancies between differing softwares (normally 5% to 10%).

The first item to note is that in this report tungsten (WO_3) is not reported. It was calculated but it was noted that there was a considerable amount of missing assays and/or the wrong assay method was used. References were made to check assays but nothing definitive was found in the data search. In addition, the grade of tungsten was very low (approximately 1/10 that of molybdenum) and at these concentrations, it is the opinion of the author that even if some of the tungsten is recoverable, it may not be saleable. The tungsten market is very limited and requirements of a saleable concentrate can be very stringent. It is, therefore, felt that at this stage, without further work, tungsten should not be reported in the same degree of confidence as MoS_2. It is the opinion of the author that at this stage, tungsten could be reported in all its entirety as an Inferred Mineral Resource but would require a separate report/table.

While some work has been undertaken, AMPL is of the opinion that the memo below (Figure 14.17, below) has not been fully resolved.



DIVISION OF AMERICAN METAL CLIMAX, INC.
EXTRACTIVE METALLURGY LABORATORY

INTER-OFFICE MEMORANDUM

SEP 23 1974
GEOLOGY DEPARTMENT

SUBJECT: Yorke-Hardy Tungsten

DATE: September 18, 197

TO: Mr. J. Bright/Colfax

FROM: C. O. Ingamells

The WO_3 data provided for a 400' block of Yorke-Hardy ore, although incomplete, indicates that the overall grade is close to 0.05% WO_3, rather than ~ 0.03% as previously estimated.

This conclusion is supported by values obtained on properly subsampled raise rounds, which average close to 0.05% WO_3.

When we accumulate complete information on this 400' block, a more exact statement can be made.

Figure 14.17. Inter-office Memo

14.11 BLOCK MODEL VALIDATION

The block model was verified for tonnage and grade using various MinePlan™ functions (see Table 14.12, below).

1. Query function (essentially a spearing of solids routine.
2. PitRes™ – A Resource reporting tool in Hexagon.
3. UG1Res™ – A second Resource reporting tool using different parameters.

It is the opinion of the authors that the variances are acceptable. PitRes™ was used for all Resource calculations.

TABLE 14.12 VALIDATION OF RESULTS			
	Volume of Main Zone	**Variance**	
Query Function	634,214,105	-440	0.00%
PitRes™	634,213,665		
UG1Res	622,468,914	11744751	1.89%
	Volume of Measured Wire Frame	**Variance**	
Query Function	173,380,390	-3147469	-1.85%
PitRes™	170,232,921		
UG1Res	167,080,456	3152465	1.89%
	Volume of Indicated Wire Frame	**Variance**	
Query Function	173,380,390	-3147469	-1.85%
PitRes™	170,232,921		
UG1Res	167,080,456	3152465	1.89%
	Volume of Inferred Wire Frame	**Variance**	
Query Function	3,065,366,079	-33144337	-1.09%
PitRes™	3,032,221,742		
UG1Res	2,980,815,170	51406572	1.72%

14.12 COMMENTS ON SELECTED SECTION PLAN VIEWS

Figure 14.18 through Figure 14.21 show selected level plans and long sections looking north. It is apparent that while the deposit, as a whole, has considerable continuity at higher grades, it becomes more diffuse. On the negative side, this may create some difficulty in designing large contiguous open stope mining blocks. Conversely, it also indicates that most "waste" material can have significant grade, and as such, may become "incremental" Resource significantly reducing the effects of dilution in designing mining blocks as well as to possibly allow driving access drifts in "waste" containing some grade, providing some contribution of value from development material.

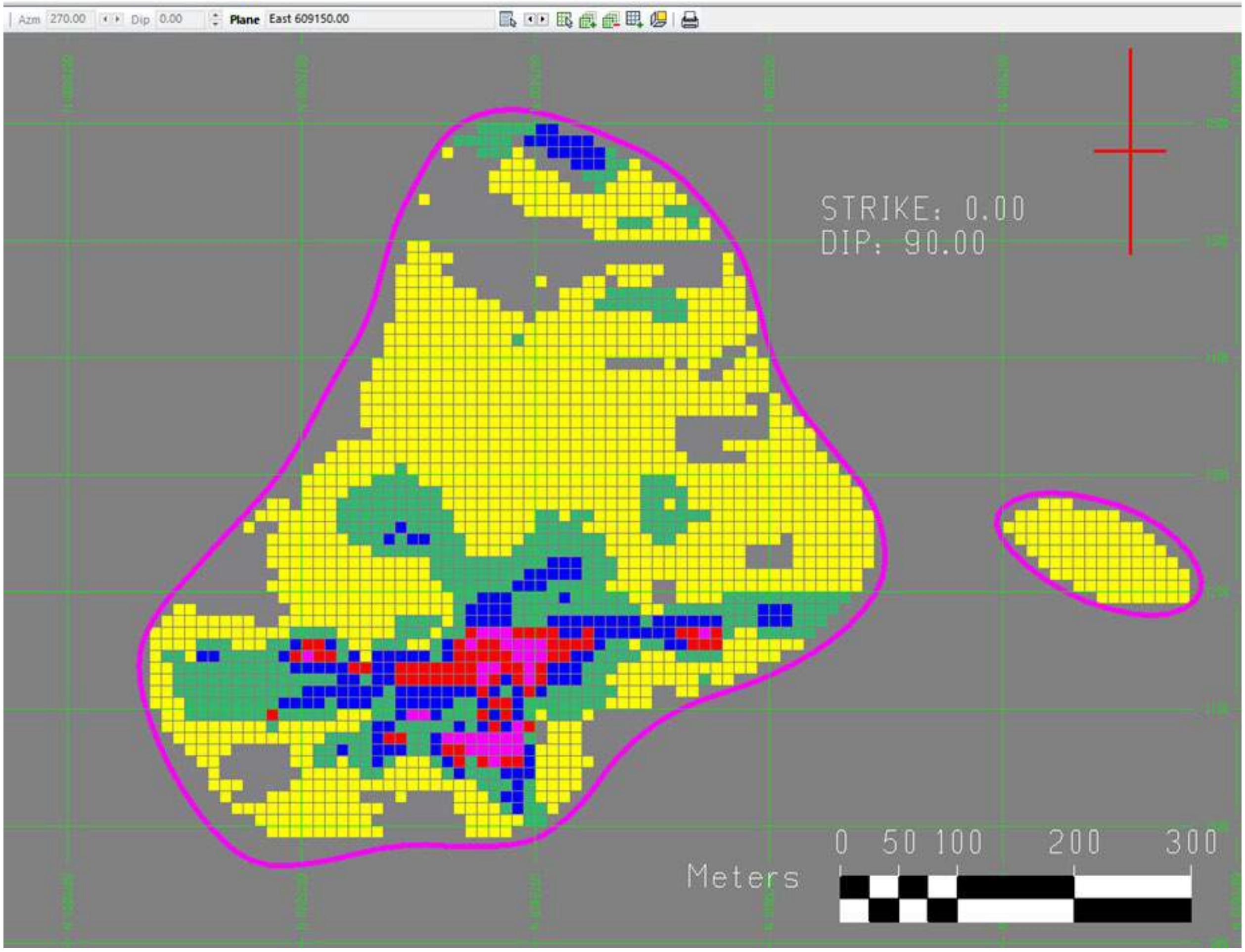


Figure 14.18. Section 609150 – East
Source: AMPL, 2023

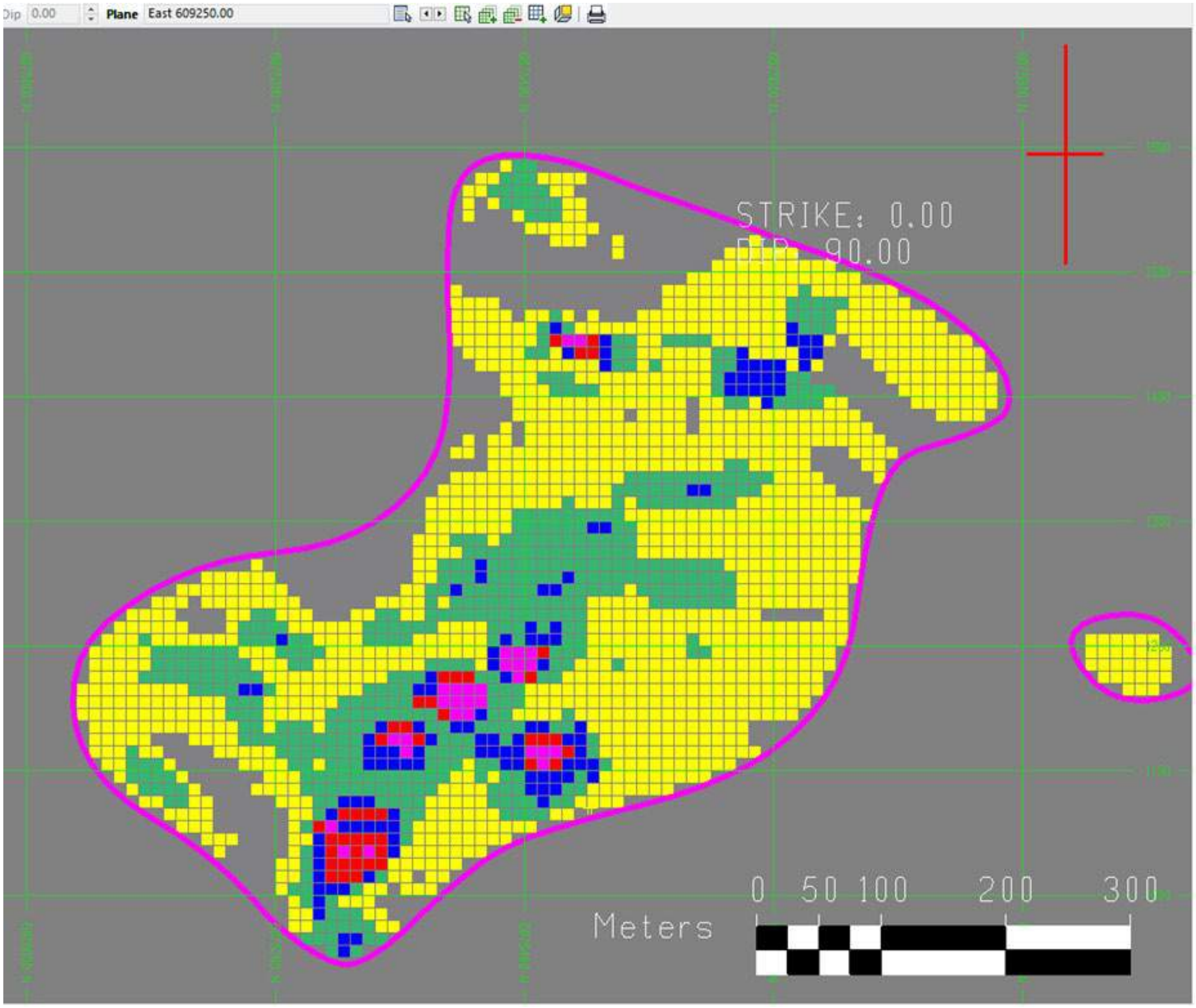


Figure 14.19. Section 609250 – East
Source: AMPL, 2023

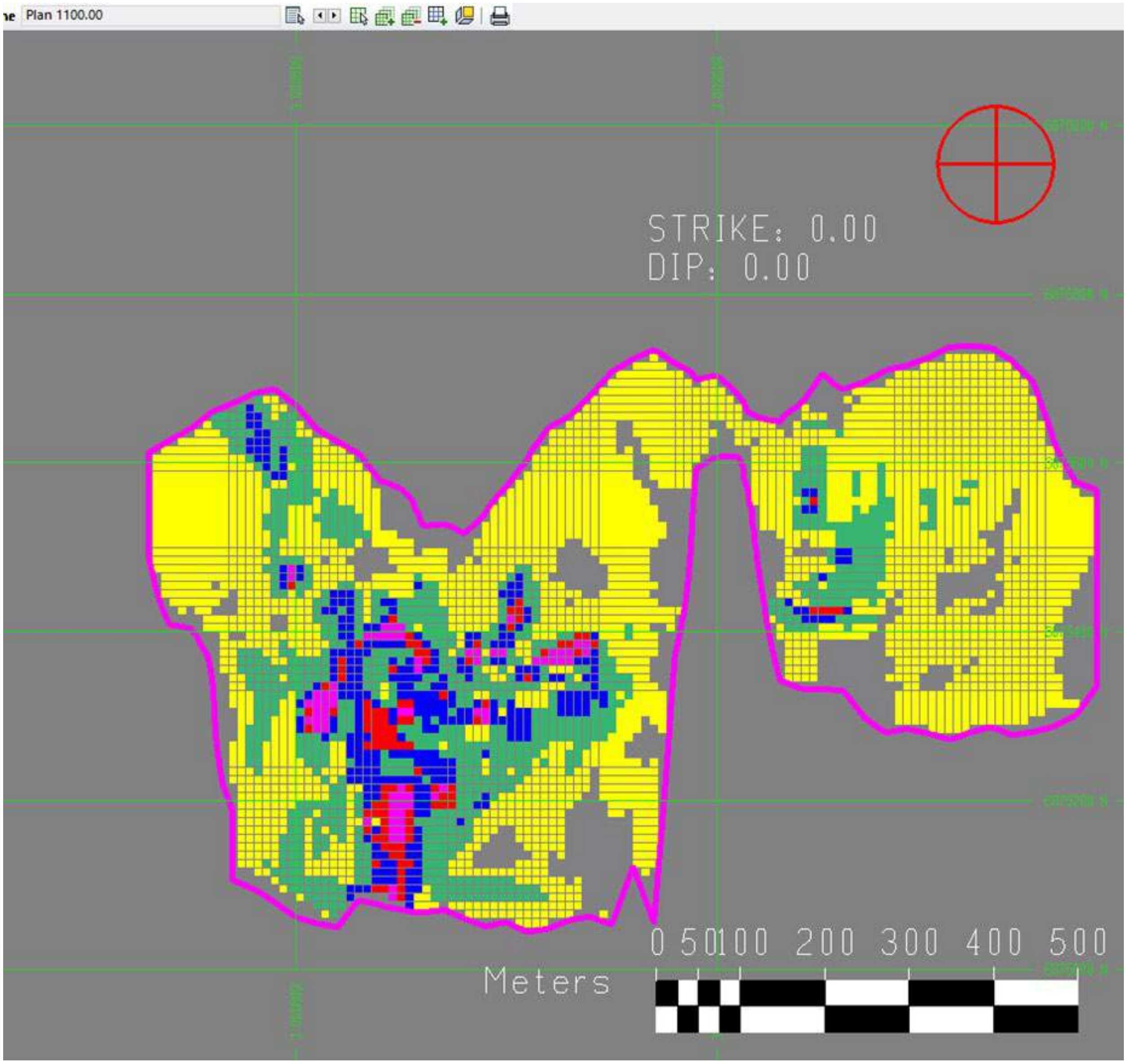


Figure 14.20. Plan View 1100

Source: AMPL, 2023

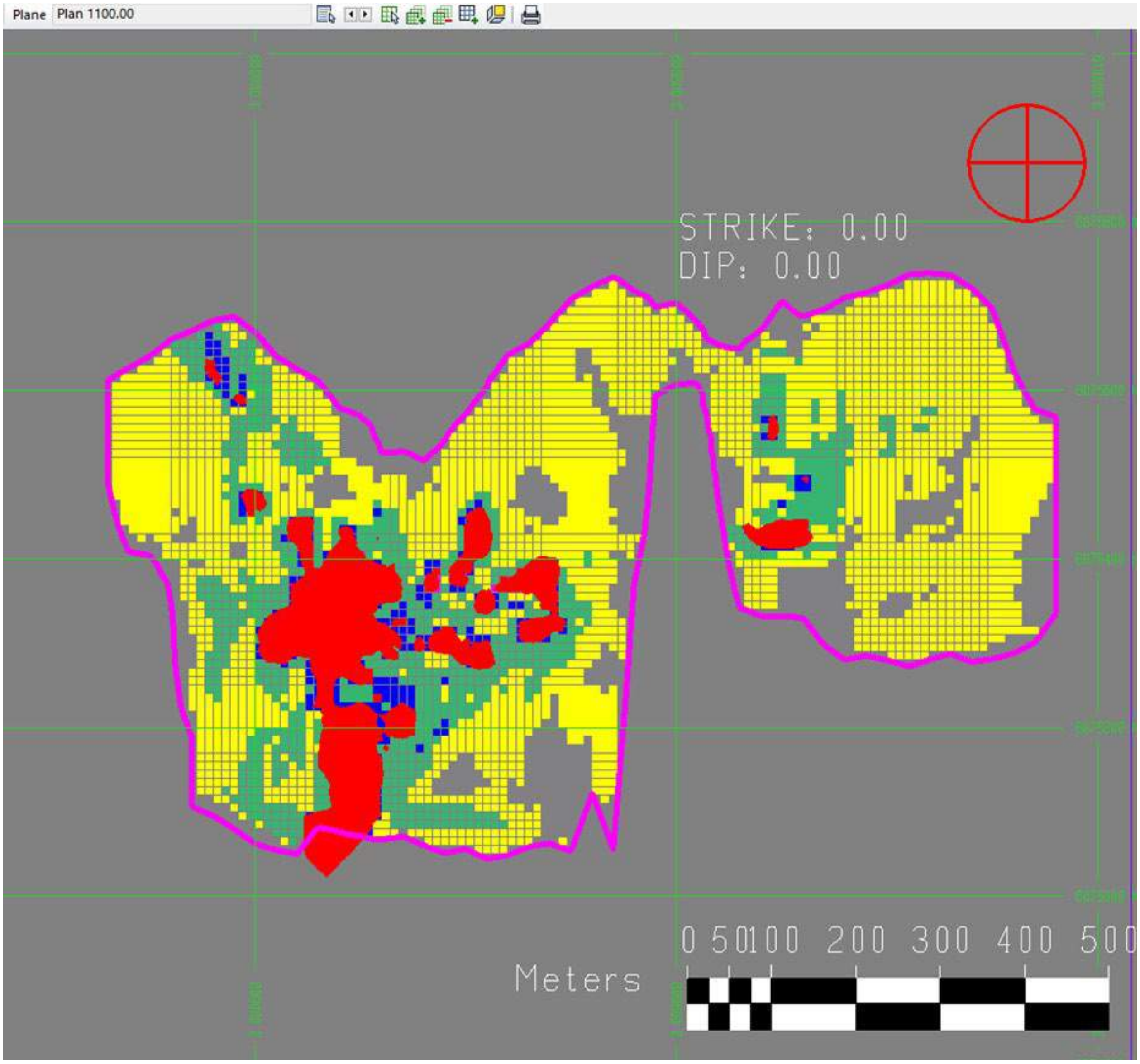


Figure 14.21. Plan View 1100 with Measured Superimposed in Red

Source: AMPL, 2023

15.0 MINERAL RESERVES

At this time, the Davidson Property has no Mineral Reserves. Reserves can only be determined with a Pre-Feasibility or Feasibility Study.

16.0 MINING METHODS

The mineralised zone is located inside Hudson Bay Mountain between the 940 m and 1,440 m elevations. Previous exploration work consisted of an adit from the 1,066 m elevation, on the east side of the mountain to enable underground drilling and the taking of a bulk sample. This adit has been abandoned since 2006 and would need to be enlarged and rehabilitated to be considered for use.

16.1 GENERAL

In the mid 2000s, the Project met with local resistance regarding development and mining of the deposit from the eastern side of the mountain. Smithers is a major tourist area and the mine site would have been highly visible from town. This Project puts the primary mine development on the west side of the mountain with the existing eastern portal used only for initial development. Once mining commences, the portal will be shut down and any waste rock generated will be returned underground as backfill.

The roadway to the existing portal entrance is a switchback up the eastern slope of the mountain that is in poor condition and needs to be rehabilitated before it could be used. A “new” drainage system was put into use about 15 years ago and has resulted in washouts and caving in several areas. The “old” drainage system consisted of ditching on the upstream side of the road loops and discharging off the ends of the switchbacks. This system worked for over 50 years and should be re-established as part of the rehab.

16.1.1 Portal Excavation and Incline Drive

The main mine access will be located on the western slope of the mountain and consist of twin 4.5 m × 5.0 m inclined tunnels from approximately 980 m elevation and will be driven at a 6% grade up to the 1,420 m elevation. This elevation is the upper boundary of mining and will be where the underground milling facility and paste fill plant are located.

The 4.5 m × 5.0 m inclines will be driven in tandem with ventilation cross overs and re-muck stations every 250 m and safety bays every 100 m. As each ventilation cross over is completed, the previous one will be sealed with a shotcrete wall and used as a storage, sump, or electrical sub. One tunnel will act as the ventilation intake and the other as the exhaust system. Auxiliary ventilation will only be required between the lead vent cross over and the next vent cross over.

Simultaneous with the development of the main access tunnels, the old portal on the east slope will be reactivated and slashed out. This will provide access for development of the mine ramp system, foot wall drives, and potentially economic mineralisation access crosscuts. This will also provide access to both the top and bottom of the mine for development of the ore pass/crusher systems, the ore storage bin, loading pocket, and shaft bottom. The upper level of the mine will give access for the early development of the underground processing facility, coarse ore bin, and drive for the vertical lift conveyor system. The tunnel slashing and ramp development will be driven using metal ducting until internal ventilation raises can be established (Figure 16.1, below).

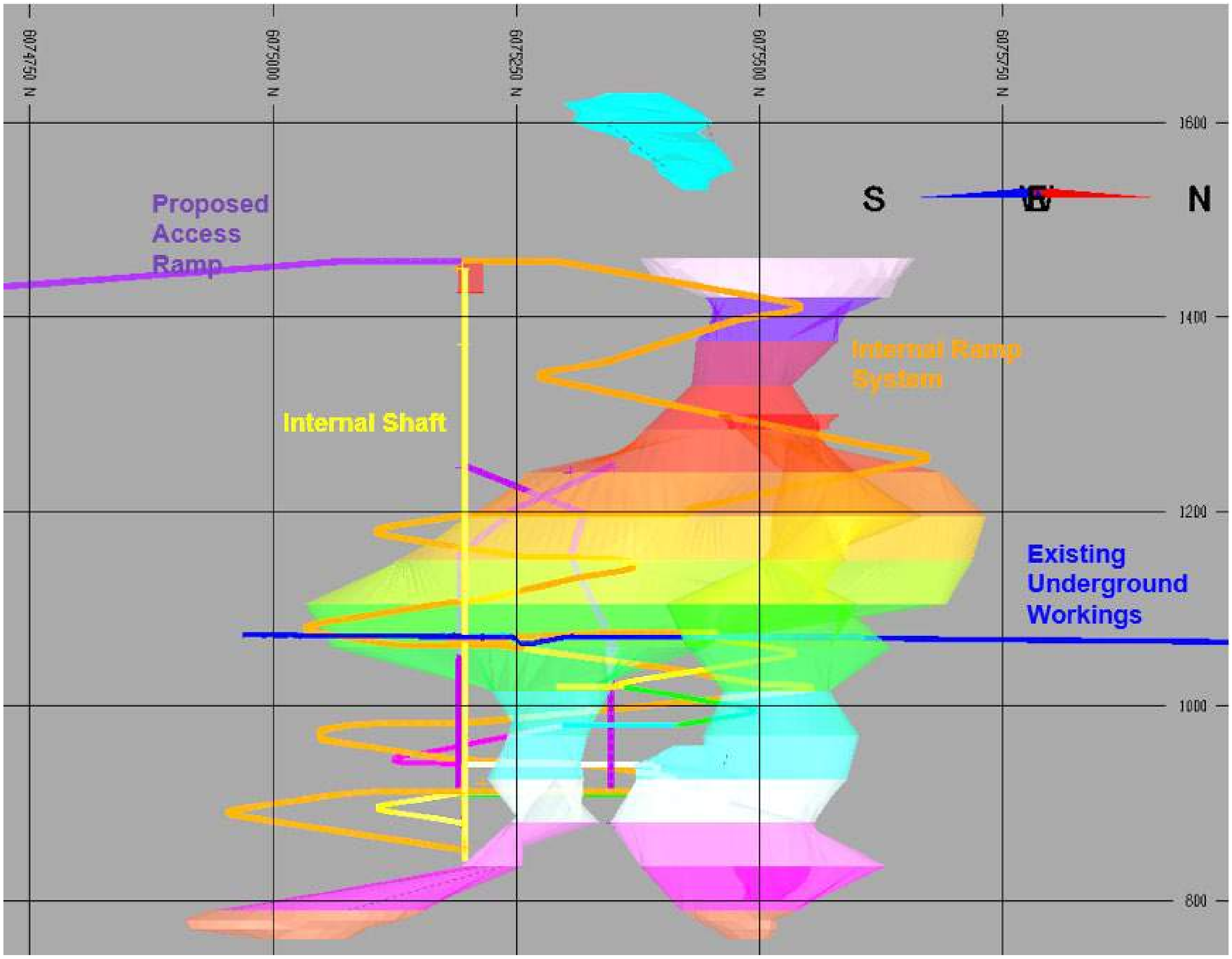


Figure 16.1. Mine Schematic
Source: AMPL, 2024

Two boom electric hydraulic jumbos will be used for the tunneling and will be supported with electric LHDs, bolters, and ancillary equipment. Almost all equipment will be electrically powered. At this time, only the jumbos, shotcrete sprayer, graders, and tractors are not available as electrically powered units.

The electrical power system for the drill jumbos will be 1,000-volt (V) power, allowing for electric subs to be spaced approximately 1 km apart, greatly reducing the costs of sub-stations and power cable. One-thousand-volt equipment is available worldwide and is the norm in most areas of the world.

Both accesses will require an area sufficient to accommodate the ventilation system (including heating), compressor, trailer, water, and generator. This area also needs to be graded to accommodate drainage away from the mine entrance and collected for treatment before releasing into the environment. Any overburden removed will be stored for future use on the mine closure.

Roadbed material would be crushed development (Figure 16.2, below).

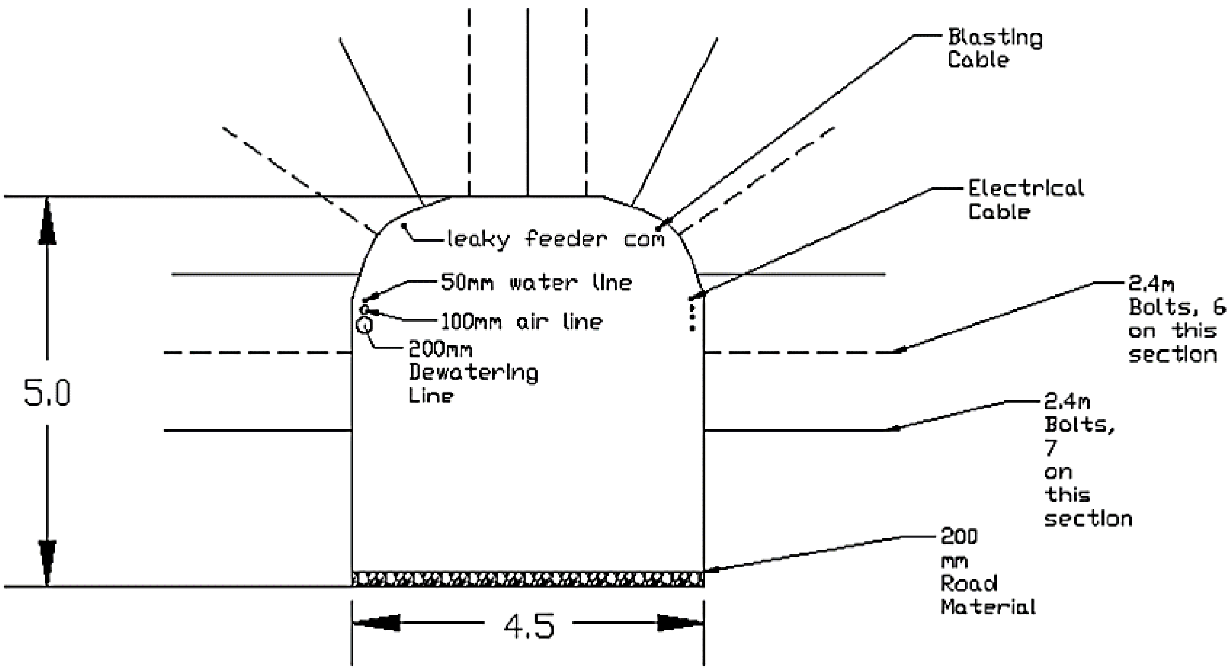


Figure 16.2. Typical Ramp Cross Section
Source: AMPL, 2024

16.2 UNDERGROUND MINE DESIGN

The mineralised zone is a large amorphous mass with higher concentrations toward the centre of the mass. The plan is to mine the higher-grade centre of the mass. There is no defined hanging wall or foot wall due to the massive nature of the deposit. A nominal hanging wall has been determined by evaluating the geotechnical data and nominating the most stable direction as the side wall. In this case, the side wall would be an east-west face and the hanging wall would be north-south facing.

On each level, the mining areas would be accessed from the main ramp by a 4.5 m × 4.0 m wide access drift. A foot wall drift 4.5 m × 4.0 m wide will be developed parallel to the designated FW of the ore zone. The lateral extent of the mining zone is 400 m to 650 m in the central area of the deposit. Two ore-pass systems, 250 m apart, will be developed with dumps on each level and jaw crushers at the bottom of each pass. Levels will be spaced at 45 m vertical intervals from 1,060 m elevation up to 1,420 m elevation where the processing plant will be located. The bulk of the potentially mineable mineralised zone lies between 1,060 m and 1,240 m with smaller extensions below 1,060 m and above 1,240 m.

Crosscuts will be driven at 30 m centres down the centreline of each stope to the nominal hanging wall. In some cases, the stopes would be almost 400 m from the foot wall to the hanging wall. The proposed mining method is longhole open stoping with paste backfill. Due to the breadth of the potentially economic mineralisation, the stopes will need to be panelled and sequenced. All stopes will be filled with paste fill containing 5% cement by volume. Stopes will be large, 30 m along the hanging wall by 45 m deep and 50 m high. Each stope will contain approximately 160,000 tonnes; 15 to 16 stopes will need to be cycled each year to achieve production targets (Figure 16.3).

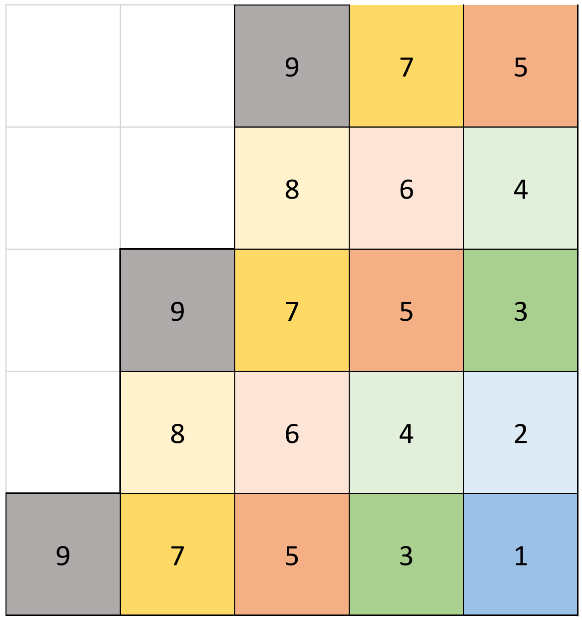


Figure 16.3. Mining Sequence of Panel Stopes

Source: AMPL, 2024

Drilling will be with rubber tire mounted ITH drills complete with rod carousels and automation packages. The holes will be 150 mm and up to 52 m long. The crosscut accesses will be driven 5 m wide to allow for cable bolt support in the mid span of the stope and will accommodate two drills in each heading. Only one driller will be required in each stope to operate both drills and the automation package will allow the drills to continue operation during shift change. Only the undercut of the initial stope in the vertical sequence will be required to be silled to the boundaries as the undercut of the next vertical stope will be drilled and blasted from the drill horizon. Maximum exposure of the hanging wall will be 50 m high by 30 m wide. This will give an Hydraulic Radius (Hr) for the hanging wall of 9.4. The Hr of the sidewalls will be 11.8 and the Hr of the back will be 9. To lessen blast damage a pre-shear ring will be drilled at the hanging wall with a 110 mm ITH drill (Figure 16.4, below).

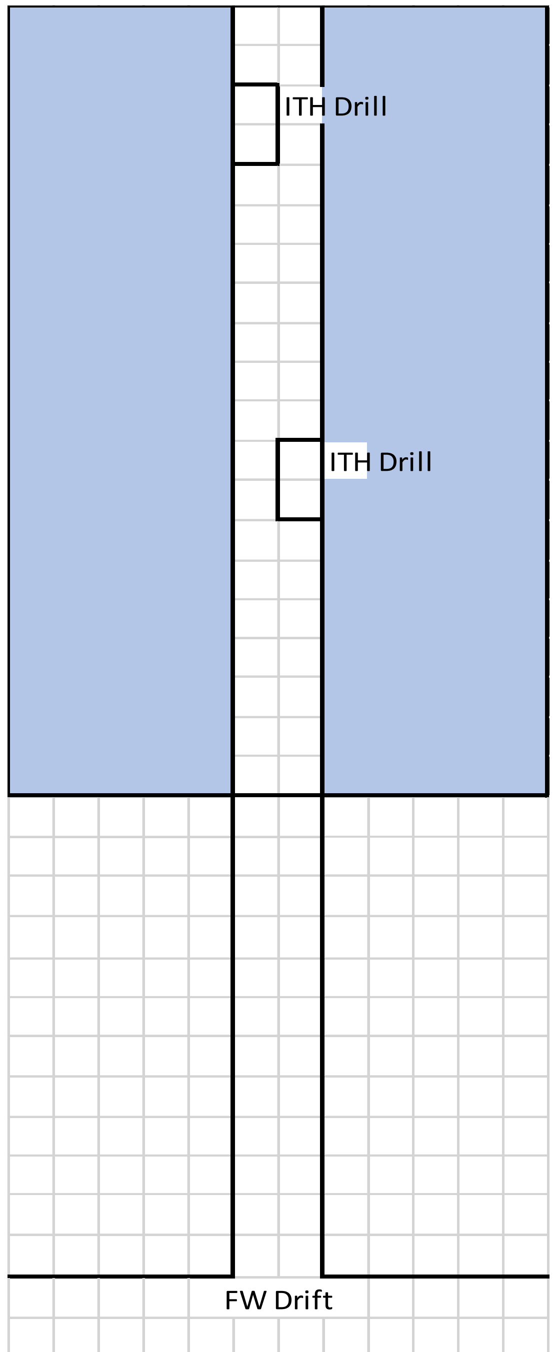


Figure 16.4. Stope Layout
Source: AMPL, 2024

Underground development, including excavation of ramps, accesses, and haulage drifts will employ diesel powered, rubber tired 2 boom electric/hydraulic drill jumbos. All other equipment will be battery powered including 7 m^3 LHD units, 50 tonne haul trucks, bolters, ANFO loaders, and scissor lifts with work platforms. Mining will utilise battery powered rubber tired mobile equipment including ITH drills, LHDs, and haul trucks.

16.3 GEOTECHNICAL CONSIDERATIONS

For the purposes of this study, the geotechnical design has been based on a review, by Dr. W. F. Bawden, of previous geotechnical work completed on the Project. His comments are summarised below:

- *The orebody is hosted in a granodiorite, a strong stiff rock. The rock mass quality is good to very good (GSI = 65 TO 75).*
- *For the purpose of this study the orebody has been assumed to be dry due to lack of hydrogeological data.*
- *The available data indicate that the rock mass is dissected by several joint sets (i. e. blocky). A statistical analysis of joint set densities is not possible with the existing data and thus joint set dominance cannot be determined. There are several steeply dipping joint sets but also at least two low angle dip sets (≤ 35°) and two or more sets dipping between ~40° and 80°.*
- *The far field in situ stress state is unknown. Two possibilities are evaluated in this report: (i) a gravitational stress field and (ii) a stress field where the maximum principle stress is horizontal. There is limited field evidence suggesting that (i) is more probable (joint surface staining in the adit indicating water flow plus one striated fracture exhibiting water flow).*

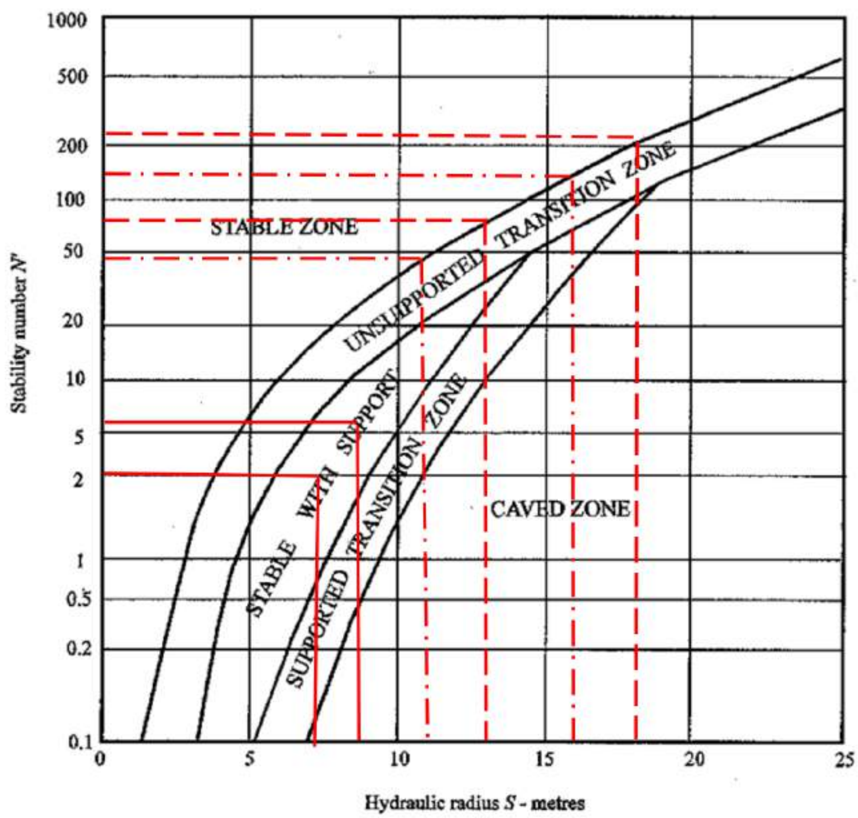


Figure 21: HR range for maximum principle stress horizontal: back ——— ; east – west wall — — — north – south wall – · – · – · – · –

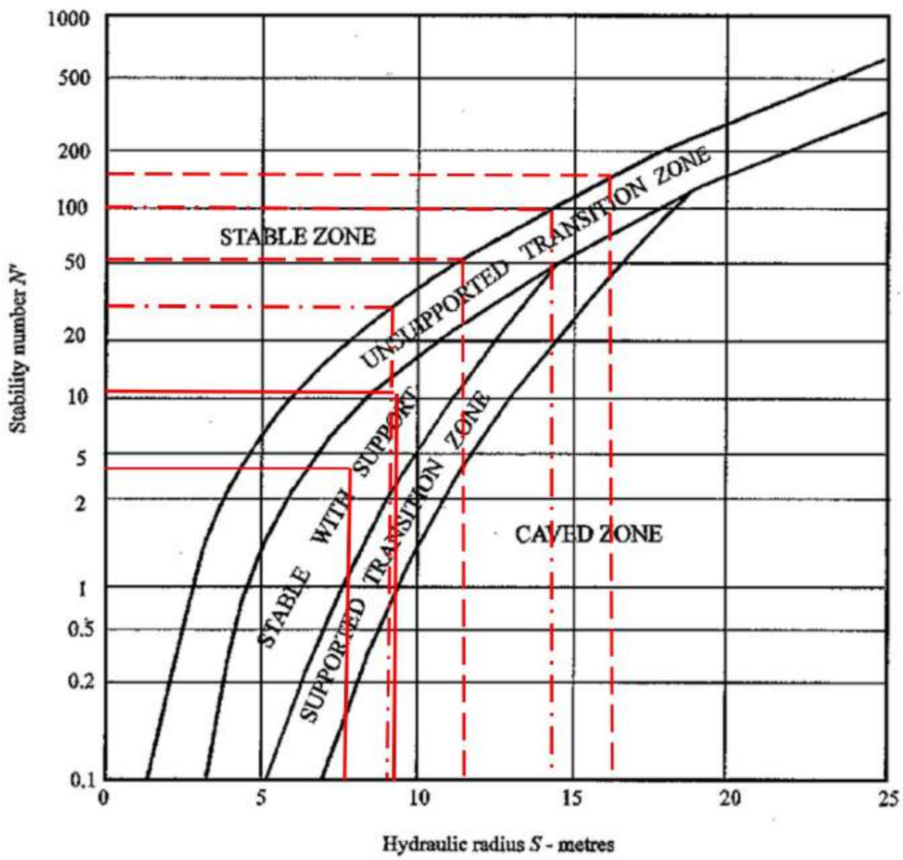


Figure 22: HR range for gravitational stress field: back ; east – west wall
north – south wall

- *Analysis of the existing structural data indicates that stope backs will be exposed to numerous wedges and will require deep secondary (cable bolt) support. Vertical stope walls are assumed to be unsupported. Analysis of the structural data indicates that an east – west orientation is favored for the longest stope wall with end walls being north – south. Stope stability analysis was conducted using the empirical Stability Graph technique. Resulting hydraulic radius (HR) ranges for the two limiting stress conditions are:*

Maximum Principle stress horizontal	Hr
Back	7.5 - 8.5
East-West walls	13.5 - 18
North-South walls	11 - 16
Gravitational stress field	Hr
Back	7.5 - 9
East-West walls	11.5 - 16.5
North-South walls	9-14

- *Secondary stopes or pillars, unless cut extremely thin, should not experience any overstress.*
- *Stopes must be filled to prevent any possible surface deformation. Cemented paste backfill is recommended as the filling medium for operational efficiency and cost savings with the mine cycle.*
- *The existing geotechnical database is sufficient for the present PEA study. It would, however, require a significant upgrade to be adequate for a feasibility level study. Geotechnical drill holes will be required on the centreline of the portal, the main decline as well as any permanent infrastructure for the mine surface structures.*

Future test work, including oriented core drilling, will be required to characterise the rock strengths and quality of both the ore zones and the waste rock for the next phase of study.

Lateral development will be supported with 1.8 m long resin grouted rebar on a 1.2 m × 1.2 m pattern and welded wire mesh screen (1.5 m × 2.7 m sheet with 5.6 mm wire thickness, 100 mm × 100 mm apertures) on the backs and walls to within 1.5 m of the floor on the walls. Screen sheets will be installed with 0.2 m overlap.

16.4 MINE ACCESS AND LEVEL DEVELOPMENT

16.4.1 Main Access

The main access will be located on the western slope of the mountain and will be a twinned inclined tunnel. The heading size will be 5.0 m high × 4.5 m wide and will be driven at a +6% grade. One tunnel will be used for access and the second for egress of men and materials. During development, one tunnel will be used for fresh air intake and the other for exhaust. Upon completion of development and connecting with the internal ramp system, both tunnels will act as a fresh air intake.

All personnel, equipment, and materials will be transported into and from the mine via this main ramp system. All development grade ore and waste rock (as required only) will be transported from the underground in 50 tonne electrically powered haulage trucks.

The east exploration drift will be slashed out and an internal ramp system developed from 1060 up to 1450 and from 1060 down to 900. The development of the ramp system will be a priority in order to construct the necessary infrastructure while the main access tunnels are being driven. Upon completion and connection with the western access tunnels, the east tunnel will act solely as a mine exhaust tunnel.

16.4.2 Level Access and Development

On each level, the mining areas would be accessed from the main ramp by a 4.5 m high × 4.0 m wide access drift driven in the foot wall. The proposed mining method is longhole open stoping using 150 mm drill holes. Stoping will take place in panels, which are nominally 30 m wide (along strike, 45 m across and 45 m high. All walls will be vertical and the hydraulic radii of all the exposed faces fall within the stable or stable with support (back) areas utilising the Matthew's Stability Graph analytical method.

Underground development, including excavation of ramps, accesses, and haulage drifts, will employ diesel powered, rubber tired 2 boom electric/hydraulic drill jumbos, and electrically powered 7 m^3 LHD units, 50 tonne haul trucks, bolters, ANFO loading units, and scissor lifts with work platforms. Production mining will utilise electrically powered rubber tired mobile equipment including In-The Hole drill units as well as a single boom drill with extension rods for cable bolting, LHDs, and haul trucks.

16.5 ROCK HANDLING

Initially, ramp and level development will utilise 50 tonne haulage trucks to bring the rock to the surface. Development of the internal ramp system will include two ore pass systems complete with a flat jaw crusher at the bottom and 5,500 tonne coarse ore bins beneath each crusher. The crushers will be located on the 940 Level. On the 910 Level, a cone crusher will be set up to process the coarse ore from either crusher and send it to a Pocketlift Conveyor for hoisting the crushed material to the fine ore bin feeding the processing plant.

Once these systems are in place development grade potentially economic mineralisation can be sent through the ore pass and hoisting system to the mill (Figure 16.5 and Figure 16.6, below).

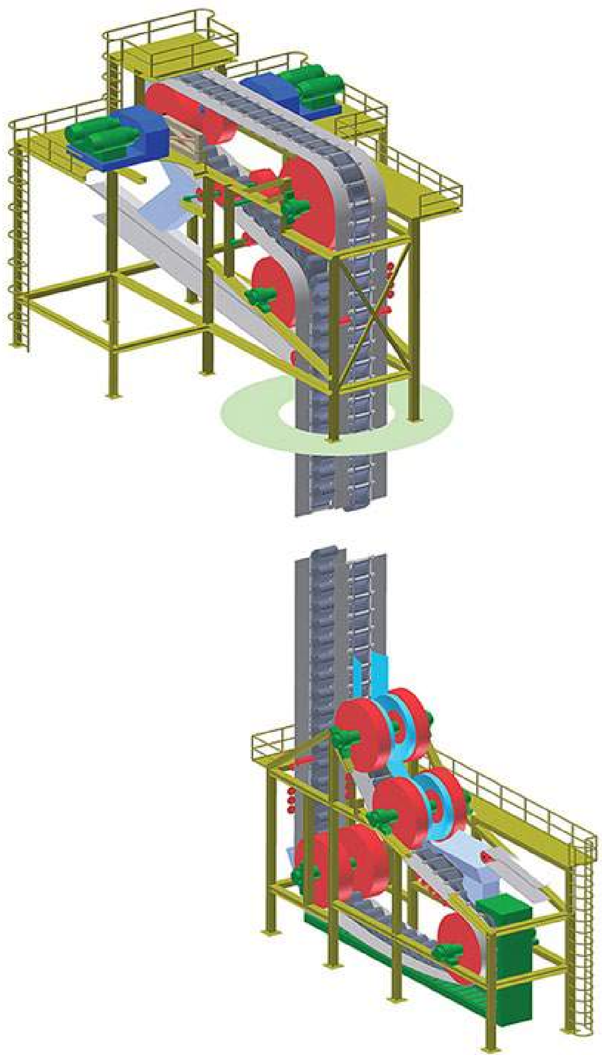


Figure 16.5. Vertical Lift Conveyor System
Source: Pinterest, 2024

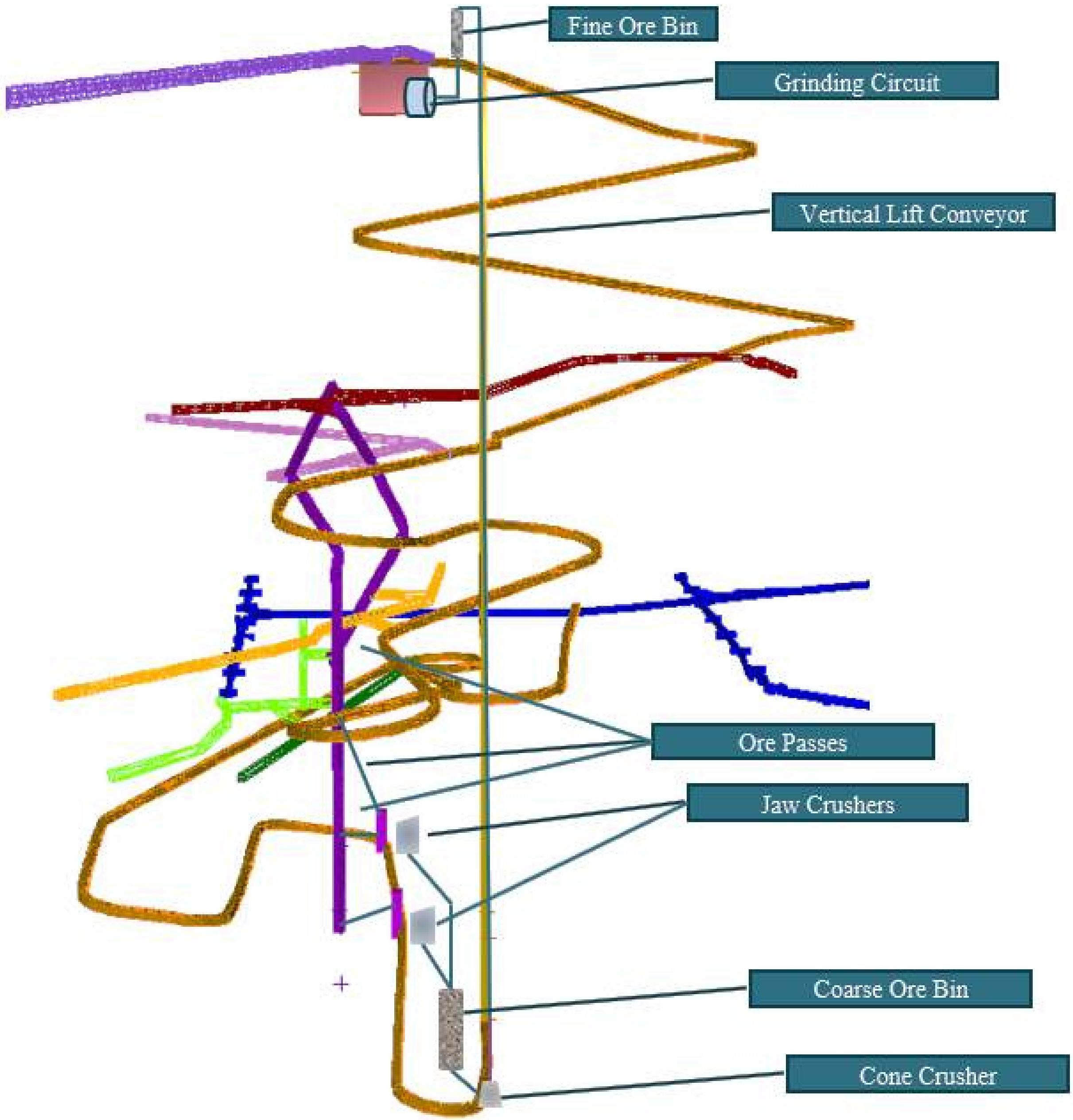


Figure 16.6. Davidson Ore Flow System

Source: AMPL, 2024

16.6 UNDERGROUND SERVICES AND INFRASTRUCTURE

Underground infrastructure will include:

- Breakdown maintenance shop;
- Fuel stations;
- Explosives and detonator magazines;
- Refuge stations;

- Main dewatering sumps;
- Main storage areas;
- Latrines;
- Electrical substations; and
- Mine wide wireless communication and control system.

Mine surface support facilities located in the area of the portal will include a surface ventilation fan set-up, two cement silos, maintenance shop, explosives magazines, mine rescue station, power sub-station, compressor station, small warehousing facility, laydown yard, and a water storage pond.

16.6.1 Electrical Distribution

The power line would be connected to a surface sub-station located near to the mine portal. Power from the main sub-station would feed the main underground power line, a 500 thousands of circular mils (MCM) cable, installed in the main access ramp from the surface. This power line would feed portable sub-stations located on levels central to the working areas. Portable power centres would supply loads on the nearby levels and transform power down to 4,160 V and 1,000 V, as required.

On the surface, the sub-station would also provide 4,160 V feeds to drive ventilation fans and other power requirements for the underground mine surface facilities. The system would utilise a switch room/master control centre (MCC) panel near the ramp portal.

The main underground mine electrical feed will consist of a 4,160 V, armoured 3 conductors, 5 kV, 500 MCM teck cable installed in the ramp. A grounding conductor will also be hung in the ramp in conjunction with the 4,160 cable. This will supply the electrical power for the underground processing plant as well as all other underground electrical requirements. Equipment underground will be powered by 750 kilovolt-amperes (kVA) portable sub-stations located in the electrical sub-station openings. The sub-stations will step power down to 1,000 V for mining equipment and 120V for smaller, electrical powered equipment.

Table 16.1, below, presents the connected load list for underground and estimated electrical power consumption during peak mine development and production periods.

TABLE 16.1
ESTIMATED CONNECTED LOAD

Unit	Quantity	Load Factor	Operating Hours/Day	Consumption Per Unit (kW)	Total Installed	Total Monthly
Vertical Lift Conveyor	1	75%	24	1,491	1,491	805,140
Main Ventilation Fans						
Underground Exhaust	1	100%	24	300	300	216,000
Surface Intake	2	100%	24	300	600	432,000
Auxiliary Ventilation	12	75%	17	22	264	100,980
	2	75%	17	37	74	28,305
Pumps						
Ramp Sumps	6	75%	19	44	264	112,860
Main Sump	2	88%	19	150	300	149,625
Compressed Air						
Compressor 1	1	75%	24	186	186	100,440
Compressor 2	1	75%	24	186	186	100,440
Underground Equipment						
Jumbo	1	9%	2.5	180	180	1,215
Bolter	1	15%	4	90	90	1,620
Scoop Trams	8	75%	18	180	1,440	583,200
Truck	3	60%	18	240	720	233,280
Longhole Drills	5	90%	20	220	1,100	594,000
Services	9	0.75	20	100	900	405,000
Miscellaneous	1	0.8	20	100	100	48,000
Total Monthly Power Consumption (kWh)						**3,912,105**

16.6.1.1 Electrical Cabling

The electrical cabling would be hung from messenger cable that will be installed on the opposite side of the drift from the air/water lines. Bosserman clips will be used to hold the cables.

The central blasting cable will also be installed on the same side as the electrical bundle except it will be suspended on its own brackets attached to roof anchors.

16.6.2 Compressed Air

Compressed air would be supplied by 3 compressors in a compressor room located off the main intake ramp. They would provide approximately 23.8 m^3 per minute at a minimum pressure 8.3 bar (120 psi) to the underground mine. Two compressors would operate at ¾ capacity with the third compressor as a back-up if one is being repaired or maintained. The compressors would supply the main compressed air pipeline located in the main access ramp from the surface. Residual heat from the compressors would augment the heat in the air in the intake ramp.

16.6.3 Service Water

The underground mine would require approximately 400,000 m^3 of service water per year for use in drilling, dust suppression, etc. This water will be supplied from ground water and recycled water from the underground settling sumps. Previous hydrological studies done on the proposed mining area of the Project estimate groundwater inflows of 1,200 l/min to 2,400 l/min due to glacial melt. One thousand two hundred litres per minute (1,200 l/min) would supply approximately750,000 m^3 of fresh water per year. This is sufficient to supply the mining requirements as well as the processing requirements. Additional make-up water for processing will come from recycled mine water.

A water storage pond on the surface, which will store water recycled from the underground mine. All service water requirements will be met by water pumped out of the mine and sent to the surface water storage pond. Water would be sent underground in a pipeline located in the trackless access ramp from the surface. This will feed the main distribution lines on the levels, which would send water to the stope access crosscuts. Water pressures and volumes would be controlled by installing water stations, at appropriate vertical intervals within the mine, which would house a transfer station and holding tanks.

16.6.4 Mine Communications and Control Systems

An 802.11 network (WiFi) voice, video, and data transmission network will connect the mine and the surface operations. The system is comprised of access points (transmits data to and from clients' computers, tags, PLCs, etc.) installed in the mine drifts, which facilitate communication between clients and transfers data to a database server and control system on the surface. Wired telephones will be located at key infrastructure locations, such as the refuge stations. Key personnel (such as mobile mechanics, crew leaders, and shift supervisors) and mobile equipment operators (such as loader, truck, and utility vehicle operators) will be supplied with handheld mobile telephones, suitable for use underground, for contacting over the 802.11 network.

16.6.5 Mine De-Watering

The long-term mine de-watering system will include water collection sumps located on each level. The sumps would be located near the point where the ramp and level access crosscuts intersect and would be designed to prevent water entering the ramp from the levels. Overflow drill holes from the sumps would send water to the main water collection sumps, for settling, recirculation, and/or discharge from the mine. The main collection sumps would be located on the 910 Level. Each main sump would be comprised of two sets of dirty water and clear water sumps. Dirty water sumps would be sub-divided by removable timber baffle walls into three compartments to aid in settling of solids. The dirty water sumps would be used one set at a time and slimes removed from the non-operational sump with LHDs. Water would overflow from the dirty water sumps into a clear water sump (Figure 16.7, below).

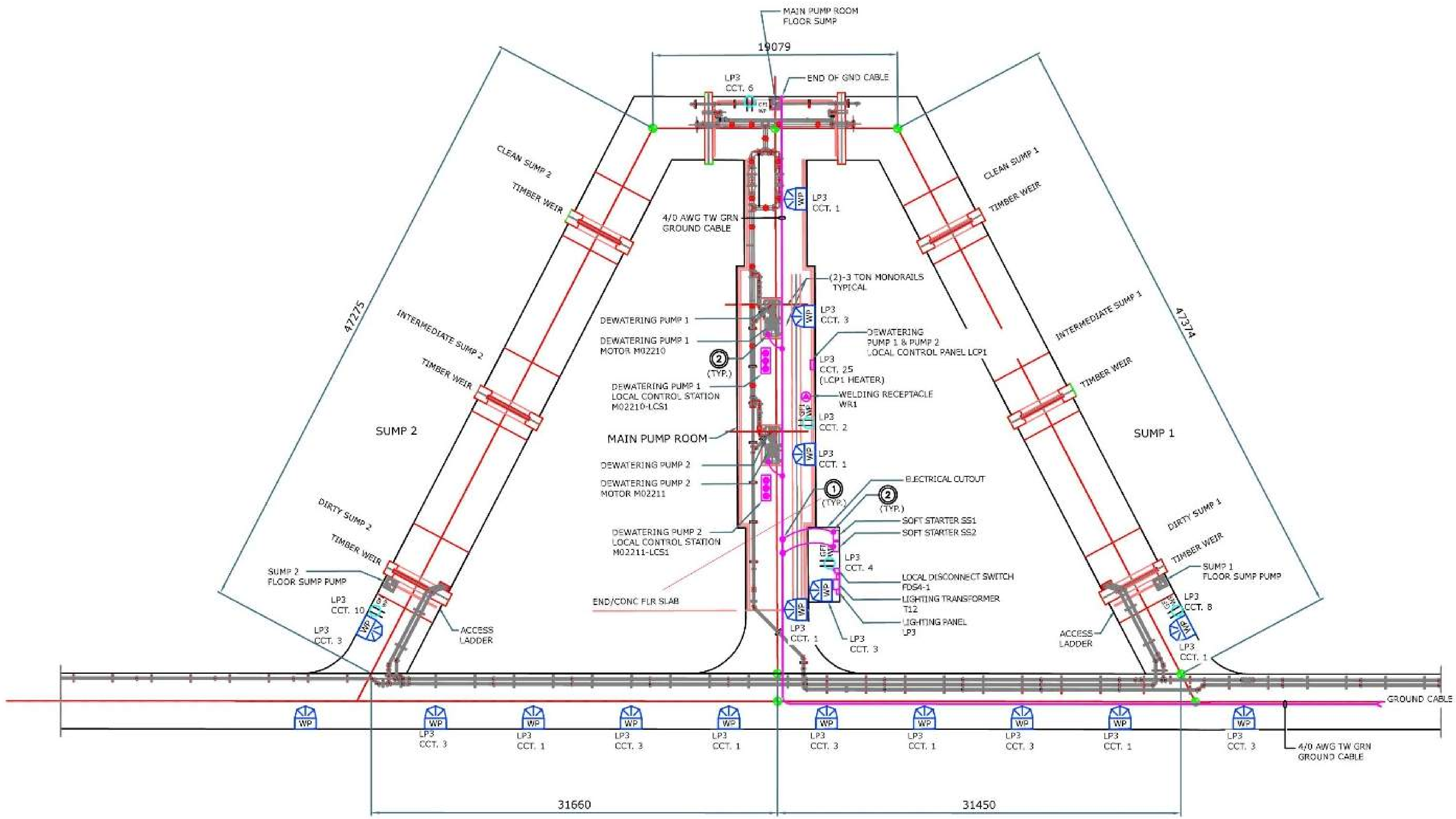


Figure 16.7. Typical Sump Arrangement
Source: AMPL, 2023

Each clear water sump, similar in size to the dirty water sumps, would be utilised to treat and store clear water prior to recirculation within the mine or to discharge. Water would be pumped to a large holding tank at the mill elevation or to a surface holding pond for underground process water or discharged to the water treatment facility on the surface.

16.6.6 Maintenance Shop

A small breakdown shop will be set up during the pre-production period in both the east and west accesses in an abandoned re-muck off the ramp. This shop will be used until a permanent breakdown shop is located midway in the mine. The mobile equipment maintenance shops would be used to perform all breakdown and service maintenance on mobile mining equipment. Major equipment rebuilds will be done in a facility in Smithers.

The permanent shop would be constructed on the 1150 Level, off the ramp. The shop would consist of a main shop area for one large piece of equipment or a couple of smaller units. The facility configuration would consist of an access drift leading to the main shop area, two additional repair bays, a welding area, wash bay area, parts storage warehouse, tool crib, electrical room, lunchroom, and supervisor's office.

The main shop area would be equipped with an overhead bridge crane in each repair bay. The electrical room, meeting room, and office would be isolated by steel hinged doors. The lunchroom would be equipped with wooden benches and tables and the office would be equipped with computer workstations connected to the mine information management system (Figure 16.8, below).

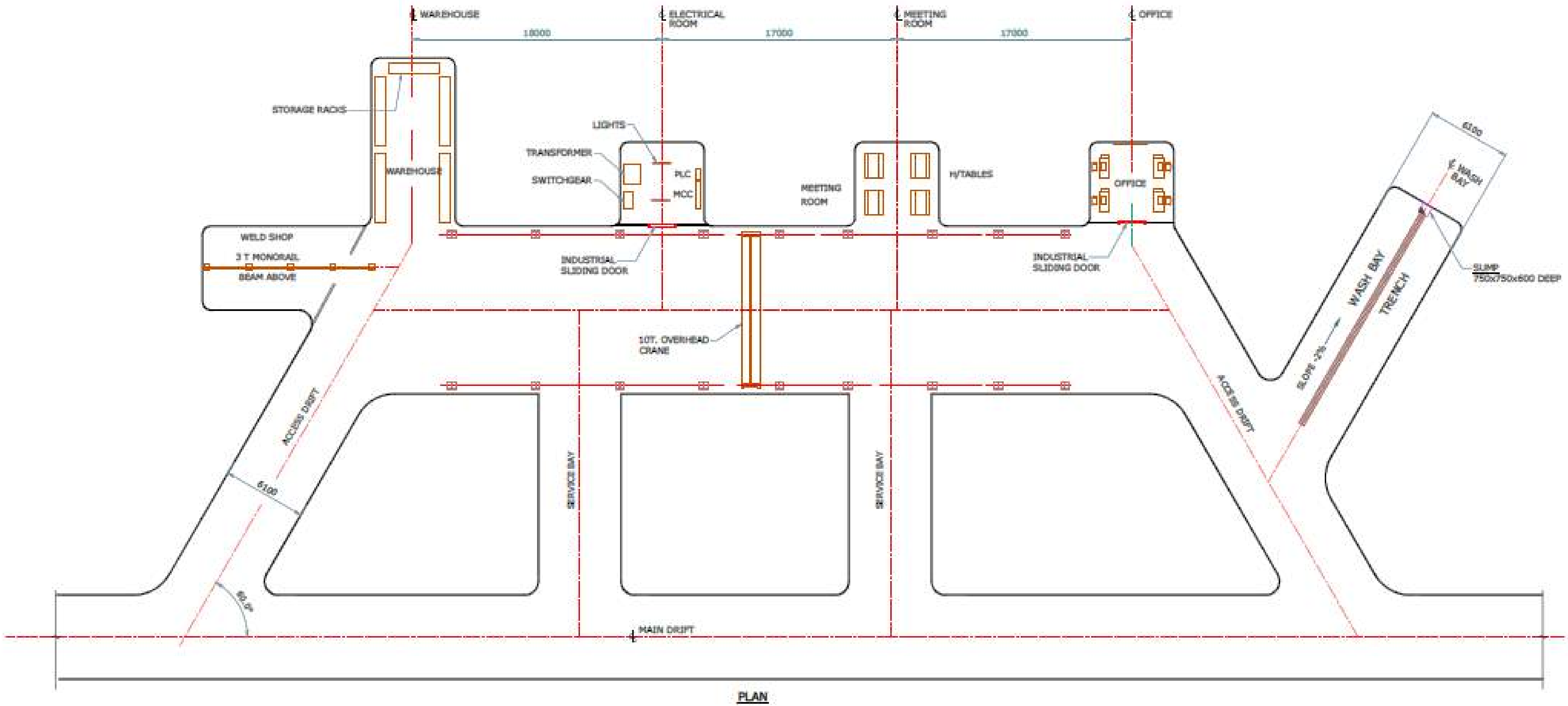


Figure 16.8. Underground Maintenance Shop
Source: AMPL, 2023

16.6.7 Fuel Stations

Portable self-contained fueling and lubrication stations will be located on levels where mining equipment is parked. The units have built in isolation doors and fire suppression.

SatStat® fuel station bladders will be filled at the surface tank farm and transported to the underground fueling station on a flat-bed utility vehicle. The SatStat® bladder will be set into the stationary SatStat® fueling station from which fuel will be dispensed by equipment operators. Each bladder has a capacity of 1,000 litres. The station will be equipped with heat-sensitive fire suppression from Ansul. A second SatStat® station storing oils and lubricants will be located near the fuel station. Several of these fueling and lubrication stations will be placed on different levels of the mine (Figure 16.9).

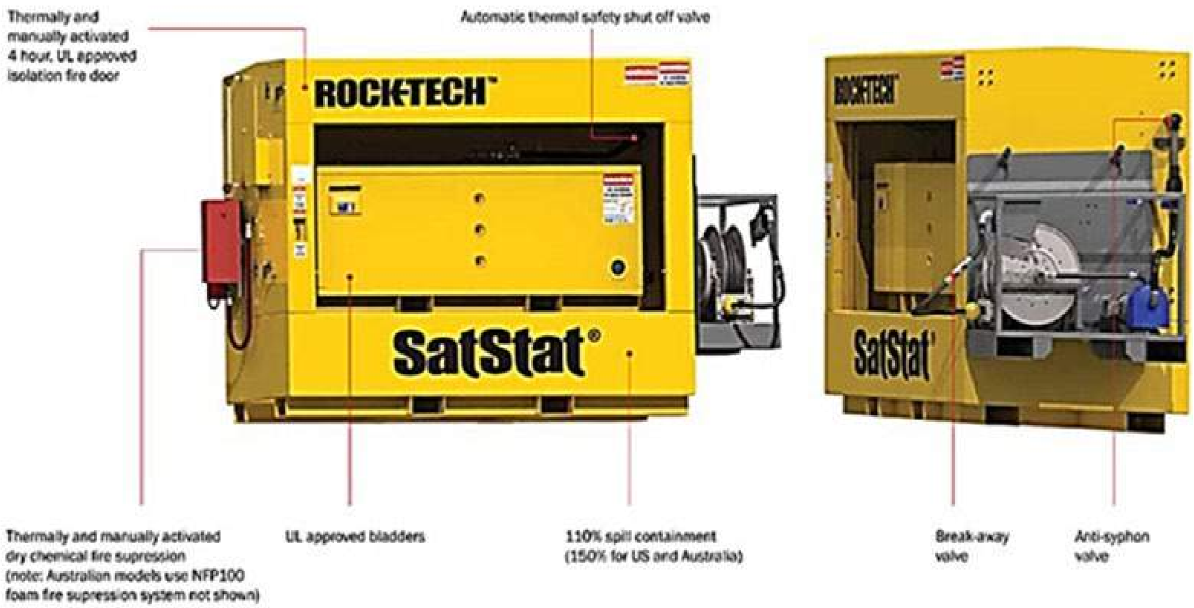


Figure 16.9. Fail Safe Fuel and Lubrication Systems
Source: Rock-Tech Sales and Service Ltd.

16.6.8 Refuge Stations

Main refuge stations would be located approximately every 80 to 90 m vertical intervals on the 910, 980, 1060, 1150, 1240, 1330, and 1420 Levels. Refuge stations would be fitted with a double door entry system in concrete walls at one end. The facility would include wooden benches and tables, hand washing station, and other equipment and supplies, as well as a supervisor's desk and other associated furniture. The refuge stations would also be equipped with safety and rescue equipment. Compressed air and water lines would be connected from the mine's supply system to lines inside the refuge station. The facility would be fitted with an electric heater unit and be vented through intake and exhaust ventilation ducts to the outside (Figure 16.10, below).

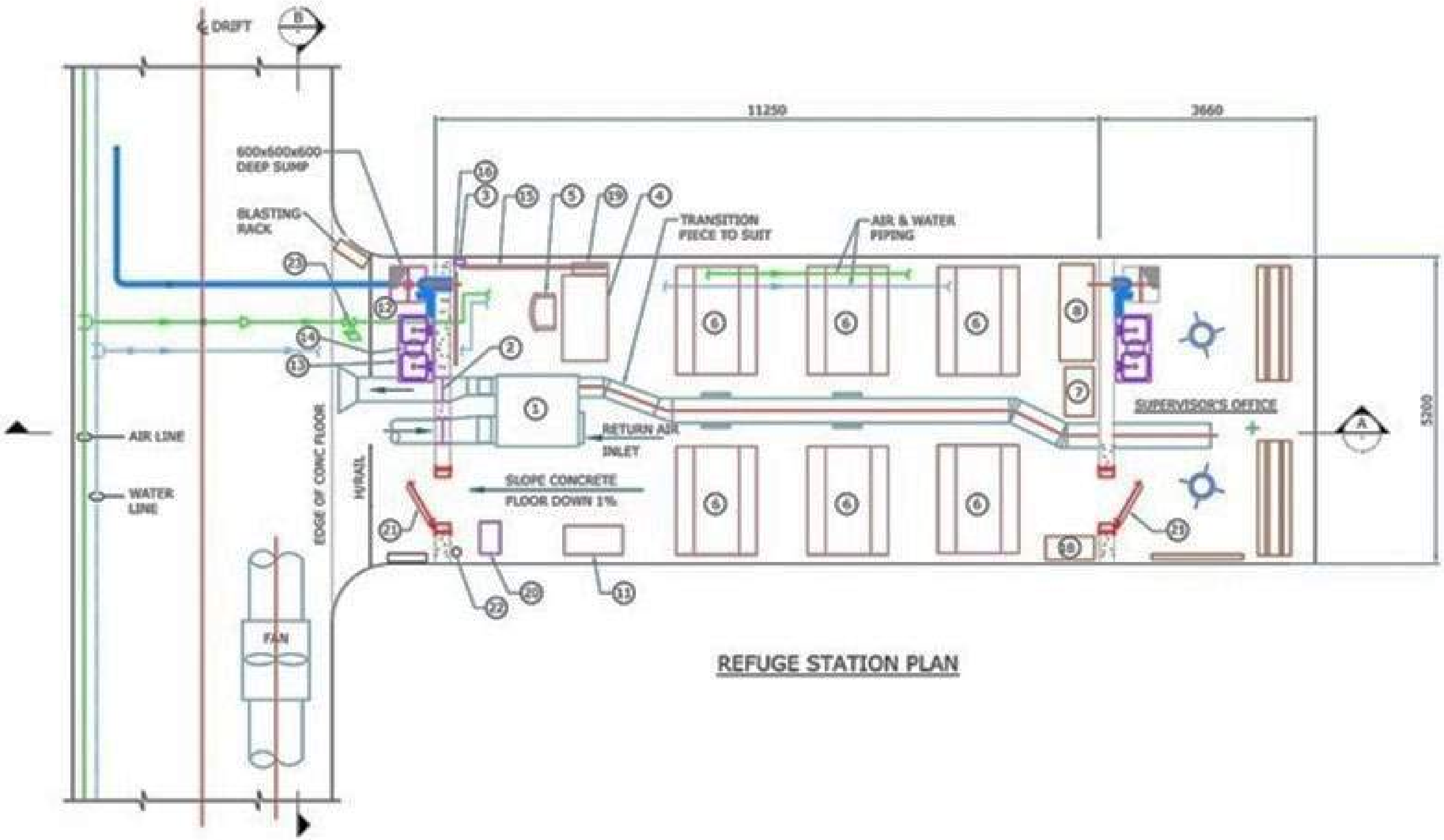


Figure 16.10. Standard Refuge Station
Source: AMPL, 2023

16.6.9 Explosives Storage

All blasting would utilise ANFO explosives. ANFO would be delivered in bulk bags, to the explosives magazines. Other stick explosives would be stored in this magazine as well.

Explosives magazines would be located on every main level. The explosives magazine floor would be gravel and the magazine entrance would include a concrete wall with doors to allow access for mobile equipment and people traffic. Both sides of the magazine would be fitted with wooden shelving on which bulk explosives bags can be placed. This magazine would require a fire suppression system. A flashing red light would be mounted by the entrance to indicate its location (Figure 16.11, below).

16.6.10 Detonator Magazine

Detonator magazines would be located near the explosives magazines. The magazines would be equipped with a gravel floor and suitable wooden shelving to allow stacking of detonator boxes on each side. The entrance would be blocked with timber posts and screen, with a man door in the wall. A flashing red light would be mounted by the entrance to indicate its location.

16.6.11 Materials Storage Areas

Storage areas, specially constructed for the purpose for storing mining consumables including pipe and fittings, ground support materials, ventilation supplies, etc., would be developed on every third level. The storage areas would include shelving and low wooden racking to safely store articles. Materials and parts would be palletised or placed in specially designed containers (for bulk materials and parts) for sending underground via the ramp. Service vehicles would transport the bulk materials to the storage areas. Materials would be distributed from the storage areas to workplace storage areas by service vehicles.

16.6.12 Restrooms

Portable toilet units, equipped with a mine toilet and small sink, would be located on appropriate working levels and near the refuge stations. Servicing of these will be contracted to the supplier.

16.6.13 Surface Support Facilities

Surface support facilities would include a mine dry, small warehouse/shop/office complex, cement storage silos; explosives magazines, laydown yard, mine rescue station, water storage pond, and power sub-station.

A small maintenance shop facility would be provided to perform major equipment repairs and rebuilds. A description of the shop facility is contained in the infrastructure section of this report. The warehouse for mine items only would be a combination of pallet (large or bulk items) and shelved (smaller items) storage.

The explosives storage area for the mine would be located 500 m from the mining and other facilities. The magazines would be housed in metal shipping containers and located so they can be observed by security located at the services site. The magazines would not be in direct line of sight of the mine or other facilities to protect mine personnel, equipment, and facilities.

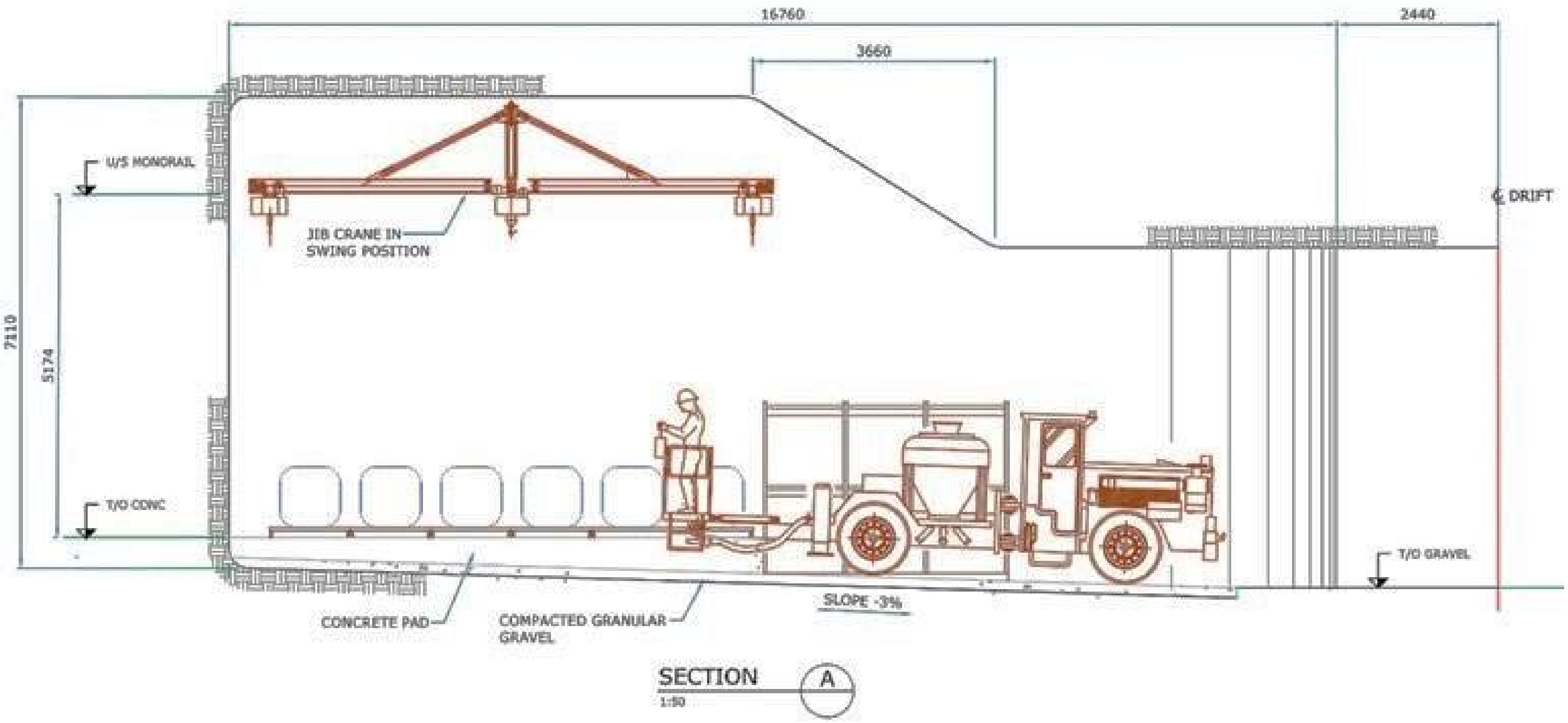


Figure 16.11. Section Through a Standard Explosives Storage Area
Source: AMPL, 2023

A laydown yard would be constructed near the portals to store materials and equipment required for the underground mine. This laydown yard would have raised timber stands on which to place large materials, such as screen, pipe, etc. as well as gravel graded areas for storing equipment and materials. A storage building would store equipment requiring protection from the elements.

A fully equipped mine rescue station is required on the property. The mine rescue station would be equipped with all the necessary equipment, including self-contained breathing apparatus, flame lamps, gas testing equipment, rescue equipment, etc., and supplies and chemicals required to operate the station. There would be enough equipment to, in an emergency, have three 5-person mine rescue teams operating or on standby at any one time.

All underground mine water would be sent to a water storage pond and reused or discharged.

16.7 MINING METHODS

One mining method will be employed at the Davidson Project. The mineralised zone is a massive zone of some 700 Mt with a higher-grade core. The central core is where mining will take place in levels spaced nominally 45 m apart. The stopes will be 30 m across the hanging wall, 45 m down the side walls, and 45 m high. A longhole mining method will be employed utilising 6-inch blast holes with stoping at 45 m vertical spacing. Due to the wide extent of the mineralised mining zone, the stopes will need to be panelled and worked in sequence. Stoping will proceed vertically for two lifts before mining resumes on the bottom level and on the third lift simultaneously. This sequence will allow two vertical blocks to be mined in the same stope without interference. The sequencing will also be a primary/secondary sequence with all stopes requiring cemented paste fill.

Mining horizons would be developed on each main level from 940 to 1420. The bulk of the mineralised mining zone is between the 1060 and 1240 levels and extends between 400 m and 650 m on each level and in cases is 400 m from the nominal foot wall to the nominal hanging wall. The mineralised mining zone progresses east and north as the elevation increases. There is a high-grade chute below the 1060 Level, which will be mined down to 900 Level and moves eastward at depth. There is another high-grade chute above the 1060 Level that moves east and north above the level up to the 1420 Level.

A 4.5 m high undercut over the full width and length of the stope would be silled on the bottom level of the stoping sequence of the potentially economic mineralisation block. Ground support would consist of 1.5 m resin rebar and screen. This would serve as the void for the stope blasting. Successive lifts would be “silled” by drilling and blasting the overcut of the stope at the drilling horizon. The longhole drills would drill 150 mm vertical drill holes (approximately 45 m to 52 m in length) in rings parallel to the foot wall and hanging wall of the potentially economic mineralisation. The drills would be fully automated and could be set-up to drill during shift change. One operator in each stope will be able to operate two drills. Drill holes would be loaded with ANFO and Nonel detonators and blasted in horizontal slices into the undercut below. An ITH reaming head will be used to drill two 30-inch holes in the stope to provide the initial cut for blasting the stope. Potentially economic mineralisation would be removed from the undercut by LHDs and transported to the nearest ore pass dump.

Stope mucking would utilise electrically powered 8.0 m^3 bucket LHDs mucking in the draw points. One operator would be able to operate two LHDs from a central control room.

The stopes would be mined in a primary/secondary sequence. Primary stopes would be those where all stope walls are in rock. Secondary stopes are those where the stope walls along strike in the ore consist of backfill. All mined out stopes will be backfilled with cemented paste fill. Once mining commences, all material removed from the east exploration drift will be returned to the underground as either backfill material or incremental mill feed.

16.8 DILUTION AND EXTRACTION

Expected dilution and mining recovery for the proposed mining method would be approximately 5% and 84%, respectively, with these factors included in the potentially mineable Resources. The dilution would, in most cases, be close to the stope grade due to the massive nature of the deposit.

The dimensions of the stopes have been established using an allowable hydraulic radius (open stope area divided by perimeter) that depends on the rock quality and using an empirical design method. If the stopes were to remain open after mining, then sill pillars and rib pillars would be required to prevent the collapse of the hanging wall, but significant ore would be left unmined. To minimise pillars and prevent the possibility of ground failure, stopes will be backfilled utilising paste backfill.

Table 16.2, below, show the following geotechnical design criterion that has been used for the stopes at the Davidson Project.

TABLE 16.2
GEOTECHNICAL DESIGN CRITERIA

Maximum Principle Stress Horizontal	Hr
Back	7.5 – 8.5
East-West Walls	13.5 – 18
North-South Walls	11 – 16
Gravitational Stress Field	Hr
Back	7.5 – 9
East-West Walls	11.5 – 16.5
North-South Walls	9 – 14

The stope design keeps the Hr at the low end of the allowable parameters. The design of the back is toward the upper end of the limits and will require cable bolting of the back for support. Twelve-meter twin-strand bulbed cables on a 2.5 m × 2.5 m pattern will be installed in the central back area of the stope in a narrow fan pattern. Cables should be tensioned and installed with plates.

At the Davidson Project, longhole mining will be the primary mining method. The levels have been spaced at 45 m intervals and the drilling has been sized at 6-inch blast holes. A central crosscut will be driven at the top of each stope and an ITH drill will be required to drill the holes in a fan pattern from this crosscut. Mining will progress upwards from the bottom of the mining block; thus, necessitating silling to be done only on the bottom level.

Both primary and secondary stopes will be filled with cemented paste fill as the panel mining sequence exposes walls on all sides in the secondary stopes.

Additional geotechnical drilling will be required at the Davidson Project to improve rock quality data along strike and at depth and aid in optimising stope geometry and support requirements.

The potentially mineable underground resource is estimated to be 49,125,000 tonnes at a grade of 0.34% MoS_2. This PEA relies on Measured and Indicated Mineral Resources (approximately 79% of the total resource tonnes) but also Inferred Mineral Resources.

It should be noted that the Inferred Mineral Resources are considered too speculative geologically to have economic considerations applied to them that would enable them to be categorised as Mineral Reserves. For the PEA, the metallurgical recovery is based on early-stage test work. Also, the cost projections range in accuracy from the PEA to the Feasibility level. Therefore, there is no guarantee that the economic projections contained in this PEA would be realised.

16.9 MINING OPERATIONS

16.9.1 Drilling

A rubber tired, electrically powered ITH drill capable of drilling up to 6-inch holes will be required to drill off the longhole stopes. The drill will be fully automated and equipped with a carousel for rod handling. Two drills, operated by one operator, will be required to drill off each stope. The drills will be set up to continue drilling during shift change (Figure 16.12, below).

Figure 16.12. Sandvik ITH Drill
Source: Sandvik

16.9.2 Blasting

All stoping will be blasted with ANFO. All explosives will be initiated using electric initiation systems connected to a central blasting system. The longhole stopes will be taken in three lifts, the first two (5 m-7 m and 10 m-15 m) to create sufficient void to blast the bulk of the stope in the final blast (25 m-30 m). Two 30-inch holes will be drilled with the ITH reaming head to act as slots for blasting. Slots will be pulled to a sufficient height to allow the stope to break before each stope blast is initiated.

16.9.3 Ground Support

As the mine is developed and the nature of the rock, the mineralisation, and the geotechnical features of the area are revealed by excavation, the mine design may require changes in the field. Such changes are to be undertaken by competent, qualified, and authorised professional engineers. Variability of the rock mass will require ongoing design decisions using the construction layouts to reflect the reality of the situation in progress. All decisions shall be documented and approved by the management team onsite.

Provisional rock support shall be as follows (Figure 16.13, Figure 16.14, Figure 16.15, and Figure 16.16, below):

- Until rock parameters are derived from exploration/geotechnical drilling and the ground control design has been designed and approved by a qualified, competent, and certified geotechnical professional(s), the following is the estimated ground support for the excavations at the Davidson Project:
  - 1.8 m length × 20 mm diameter rebar bolts installed with resin on a 1.2 m × 1.2 m pattern;
  - Weld mesh 100 mm × 100 mm squares installed in required areas only;
  - Fiber reinforced shotcrete applied to appropriate depth in required areas only; and
  - 6.0 m cement grouted cable bolts installed in areas greater than a 5.0m diameter span

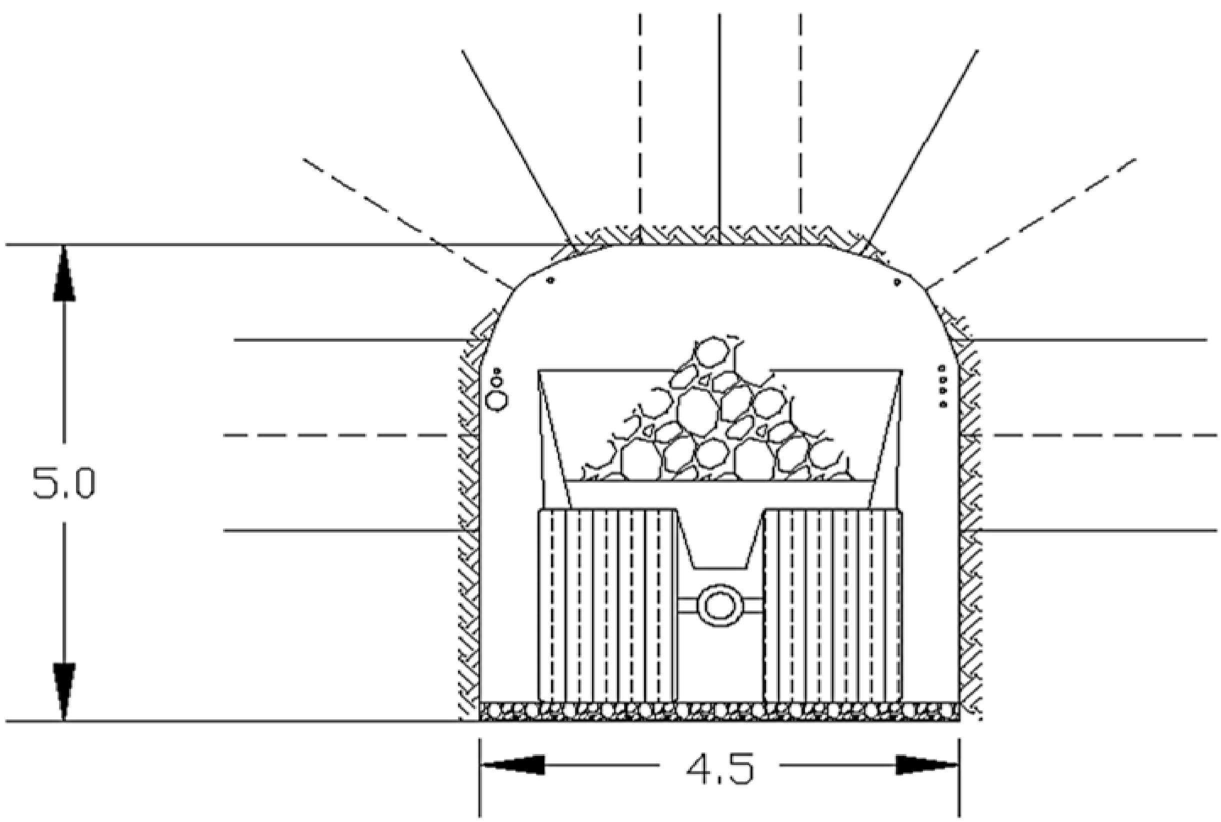


Figure 16.13. Section of a Typical Drift
Source: AMPL, 2024

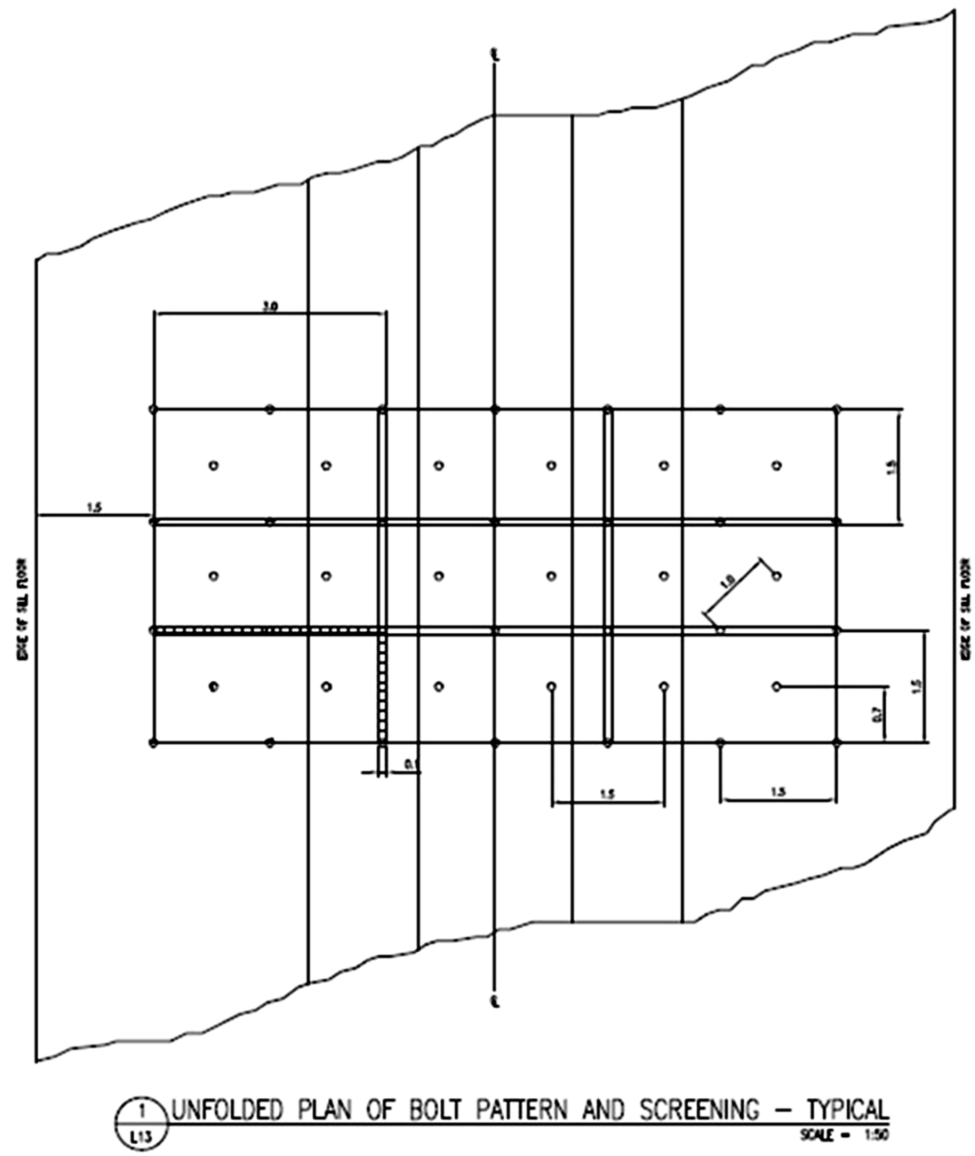


Figure 16.14. Nominal Bolting and Screening Pattern
Source: AMPL, 2024

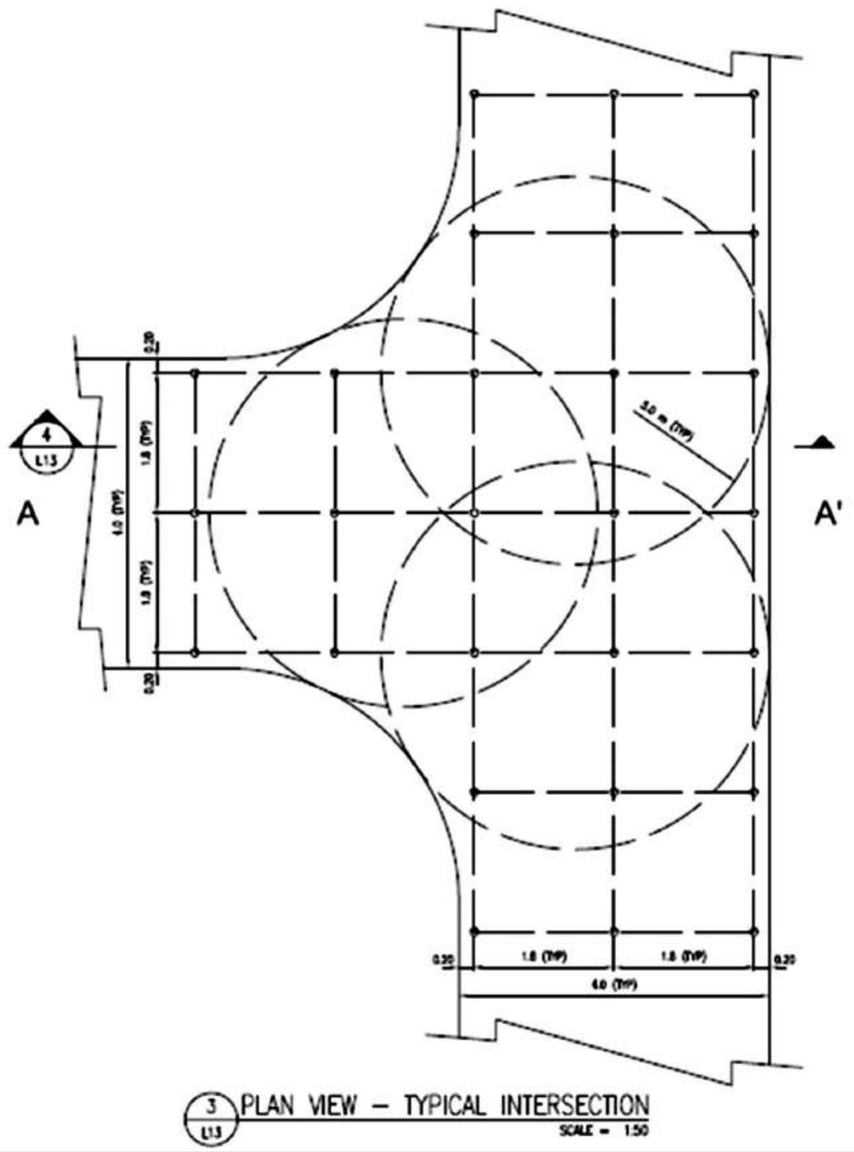


Figure 16.15. Plan of Cable Bolting Pattern in Intersections
Source: AMPL, 2024

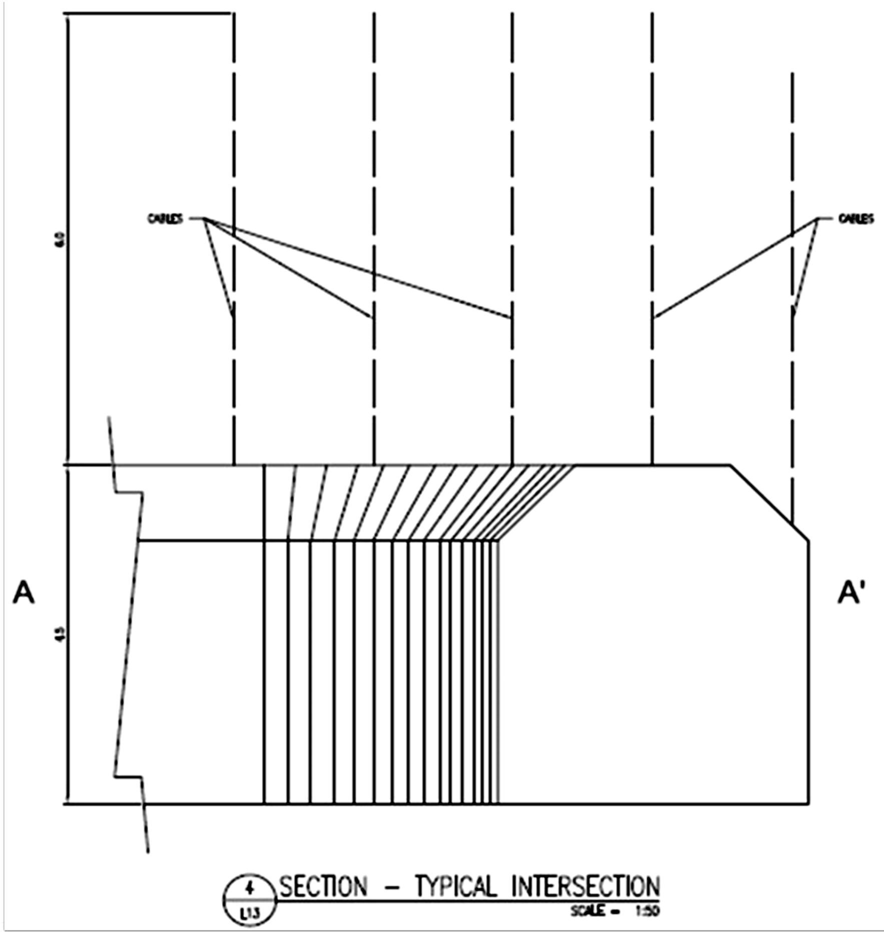


Figure 16.16. Section Showing Cable Bolts in an Intersection
Source: AMPL, 2024

16.9.4 Mucking

Mucking of the production stopes will be done utilising 8.0 m^3 remote capable electrically powered LHD units and report to the ore pass on the level.

The muck will report to a jaw crusher, coarse ore bin, and then a cone crusher before being hoisted to the 1455 Level by a vertical pocket conveyor. The vertical pocket conveyor will dump onto a belt, which dumps into the fine ore bin feeding the mill.

The shaft or, in this case, the 8-ft × 10-ft raise, is part of the overall hoisting system that comprises: the vertical lift conveyor system; the power drive units at the top of the raise and including the dumping and/or off-loading arrangements to dump the crushed material into the fine ore bin feeding the processing plant.

16.10 MINING EQUIPMENT

The pre-production mine development group will consist of three development crews. Equipment for the crew is shown in Table 16.3. Once the mine is in production, two of the crews will be demobilised and the equipment reduced accordingly.

Equipment for the mine production, services, and construction and maintenance groups is also presented in Table 16.3, below.

TABLE 16.3
MINE EQUIPMENT FLEET

Type	Electric	Development	Production	Services and Construction	Maintenance	Staff	Total
Electric/Hydraulic 2 Boom Jumbo		3					**3**
3.0 m^3 LHD				3			**3**
8.0 m^3 LHD	Y	2					**2**
8.0 m^3 LHD (Automated)	Y	1	8				**9**
Spare Battery and Charger LHD			2				**2**
Haulage Trucks (50 tonnes)	Y	3	1				**4**
Scissor Screener Bolter	Y	3					**3**
ANFO Loader	Y	2					**2**
Longhole Drill Rig	Y		5				**5**
Reaming Head			2				**2**
Grader			3				**3**
Cable Bolt Unit	Y		1				**1**
Scissor-Lift Truck	Y	2	1	1			**4**
Cassette Truck Power Unit	Y			5			**5**
Man Carrier Cassette				3			**3**
Flat Deck Cassette				4			**4**
Mine Mate Crane cassette				2			**2**
Transmixer Cassette				3			**3**
SS2 Shotcrete Sprayer				1			**1**
ML5 Multi-lift Basket	Y				1		**1**
Tractor		2	2	3	5	4	**16**
Pneumatic Trailer (20 tonne)				2			**2**
S36 Drills		2					**2**
Handheld Drills (jackleg/stoper)		12		4			**16**

Underground operations and maintenance personnel will be transported to their working places in personnel carriers. During the shift, workers will travel around the mine in light utility vehicles, such as Toyota Landcruiser or Hilux vehicles, equipped with bench seats in the box for people to sit on. Service vehicles, for materials and parts, will consist of flat bed or pickup trucks with a box, which can hold palletised, containerised, or individual items. Mine staff, engineering, and geology personnel will travel in light utility vehicles.

16.11 MINE BACKFILLING

It is expected that all stopes will have to be backfilled in order to eliminate any potential surface deformations and to maximise the recovery of the potentially economic resource. All stopes will be backfilled with cemented paste backfill (CPB) containing 5% cement. The paste fill will be readily available from the mill tailings. Fill can be delivered to the stopes at approximately 7,000 tonnes per day.

16.11.1 Underground Distribution System

The fill would be delivered to the top of the stopes by the paste fill lines or by bore holes. Fill fences, constructed at the stope entrances, would consist of a shotcreted barricade in the crosscut equipped with drainage pipes for decanting water. All stopes, including secondary stopes, will require filling with CPB to maximise resource recovery. Wherever possible, waste development will be disposed of as fill along with the CPB.

16.12 VENTILATION

The ventilation system is designed to adequately dilute the exhaust gases produced by diesel equipment. The required air volume was calculated as 0.05 m^3 per second (100 ft^3 per minute (cfm)) per brake horsepower of diesel equipment, as per Canadian standards for Tier 3 diesel engines. Where Tier 4 diesel engines are available with equipment, a reduced ventilation volume of 0.025 m^3 per second (50 cfm) may be allowed for this equipment. The horsepower rating of the underground equipment was determined and utilisation factors was applied to estimate the total amount of air required (see Table 16.4, below).

TABLE 16.4
VENTILATION REQUIREMENTS

Units	Quantity	Engine (kW)	Engine (HP)	Total Installed (HP)	Utilisation	Total cfm (000) Required
Electric/Hydraulic 2 Boom Jumbo	1	110	148	148	25%	37
3.0 m^3 LHD	3	71.5	96	288	50%	144
Grader	3	110	148	444	50%	222
SS2 Shotcrete Sprayer	1	110	148	148	20%	29.6
Tractor	16	55	74	1,184	30%	355.2
Total Required Ventilation in cfm (000)						**787.8**

The mining operation to support the diesel portion of the mining equipment fleet would require ventilation air volumes of approximately 40 m^3 to 45 m^3 per second (80,000 cfm to 90,000 cfm). The requirement has been increased to 125,000 cfm to provide sufficient air volumes to clear production blast smoke. The ventilation system would consist of a push-pull system utilising ventilation raises and the main access ramps (Figure 16.17 below).

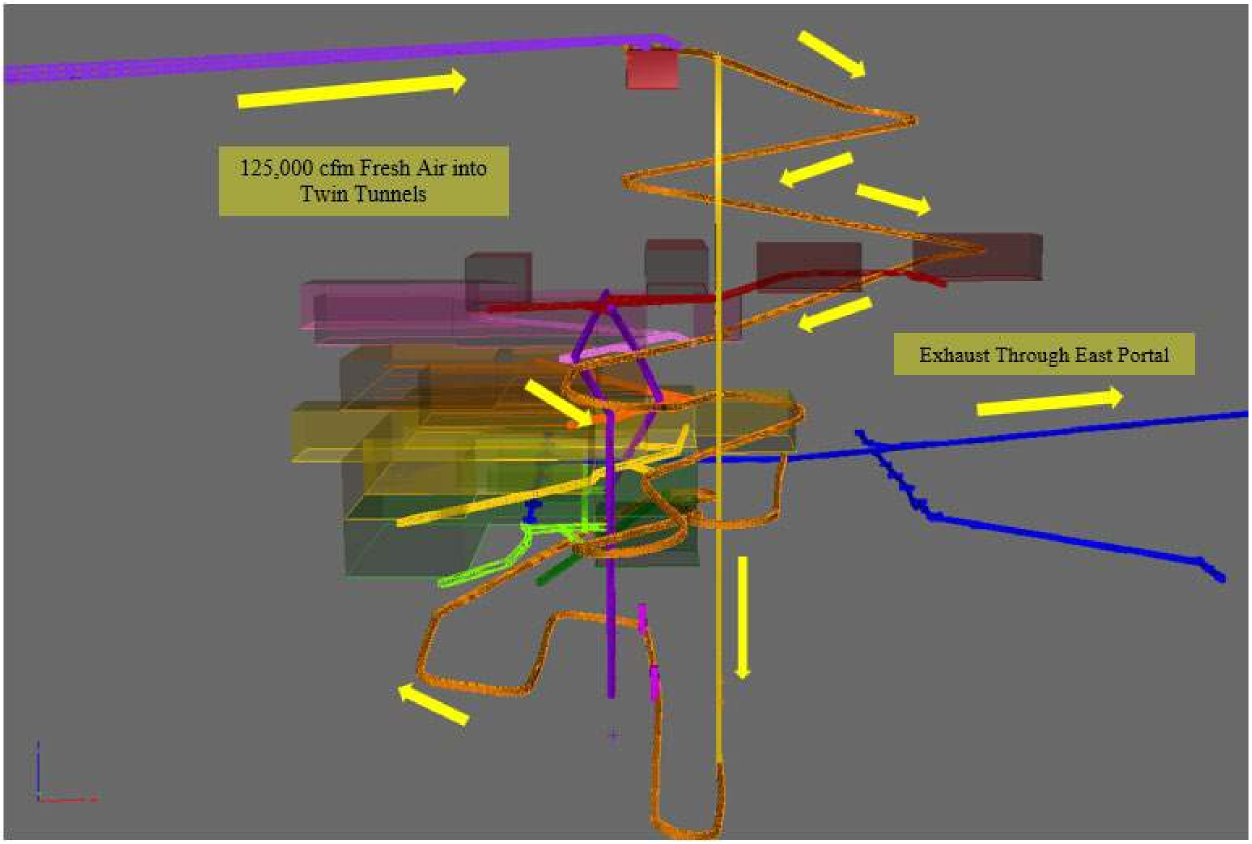


Figure 16.17. Main Ventilation System
Source: AMPL, 2024

16.12.1 Development

The twin ramp system would be a closed-circuit during development, with one tunnel acting as intake and the other as exhaust airways. Ventilation crossovers will be driven between the tunnels every 250 m and the auxiliary ventilation system moved ahead. Previous crossovers will be sealed with a shotcrete barricade to prevent short circuiting.

Development from the East Portal will consist of slashing out the existing drift and then driving up ramp and down ramp to develop the resource body and the necessary infrastructure. Initial legs of the development will be driven utilising metal, low friction ducting. A ventilation raise will be driven parallel to the ramp system to allow fresh air to advance as the ramp progresses upwards and downwards.

16.12.2 Production

Two 3.3 m × 3.3 m ventilation raises would be developed from 910 m to 1,455 m at either end of the production levels. Once the west access tunnels have connected with the rest of the mine development, the permanent ventilation system will use both tunnels as main air intakes and the east portal as an exhaust. The level ventilation raises will transfer fresh air to the levels as required and the air exhaust through the ramp system and the east portal. Air would flow along a level, be picked up by auxiliary ventilation fans, and pushed into stope accesses. From there, air would flow in the LHD mucking drift and up the pilot raise in the centre of the stope to the main foot wall drift on the level above the stope. Air would travel in the main foot wall drifts on the levels then to the exhaust raise and from there to the East Portal exhaust drift.

A large, low pressure exhaust fan will be located in the East Portal exhaust drift. If required, low pressure fans would be connected to the twin ramps near the west portals to assist air movement from the surface.

Fresh air delivery to the stopes will be controlled using auxiliary ventilation fans and ducting. Ventilation regulators, doors, and bulkheads will also be used to control the airflow in the mine.

The ramp and other lateral development will use 50 horsepower (HP) and 75 HP fans depending on the heading length. Minimal ventilation is required for the equipment, but sufficient ventilation has to be supplied to clear the blast smoke. Development headings are sized to accommodate large ducting (122 mm), to reduce head losses.

Auxiliary ventilation delivery to stopes will typically use 30 HP to 50 HP fans, with 91 mm (36-inch) flexible ducting.

16.13 DEVELOPMENT AND PRODUCTION SCHEDULES

Mine production will be 7,000 tonnes per day or 2,500,000 tonnes per year. Development is scheduled to meet stope mining requirements, on each yearly basis. Priority will be given to developing the twin access ramps from the west side of the mountain as well as the internal ramp systems which will be driven from the east exploration drift. Three independent development crews will be utilised, each with a priority heading. Development crews will be contracted for the first four years and will generally have multiple headings available for advancing at any time.

Crew 1 will start in Year -3 and will be the west tunnelling crew, which is scheduled to advance 1.5 rounds per day in each heading, or 4,410 m of advance per year. This rate is aggressive but achievable by a contractor as the crew will always be in a multi-face situation. It will take approximately 3.5 years to complete this access and connect to the inner mine development and underground processing plant location.

Crew 2 will start in Year -3 and will slash out the exploration drift from the east portal to 4.5 m × 5.0 m. Once the slashing is completed, the development of the up ramp will be the priority. Upon completion of the up ramp, the priority will be to develop the underground areas for the mill and paste plant, the fine ore bin, and the vertical lift conveyor drive area. Secondary headings will be the level accesses and foot wall and crosscut development. Scheduling is to advance 2 rounds (4.2 m length) per day, of 4.5 m × 5.0m or 4 m × 4.5 m headings, for a total of 2,94 0m of advance per year (not including safety bays, slashing, cut outs, etc.). Crew 2 will demobilise at the end of Year 1.

Volume excavations will be from the top down. A central raise would be driven for the various bins and the thickener and the overcuts openings silled out with traditional development methods. Vertical lifts can then be mined using the small track drill, such as the Boart S36 drill. For the large grinding section, overcuts and undercuts can be silled out and the remaining block between drilled with a track drill and blasted.

Crew 3 will start in Year -2 and drive the down ramp as a priority and then establish the crusher cut-outs and the coarse ore bins as well as the loading pocket area. Raise crews will then be able to drive the shaft, the ore passes, and ventilation raises. Crew 3 will demobilise at the end of Year -1.

16.13.1 Productivities

With potentially economic mineralisation development, stoping, and backfilling, the following parameters were used in determining stope requirements:

- Two 30-inch pilot raises will be drilled using ITH drills in each stope.

- Stopes are large, being 30 m wide, 45 m long, and 45 m high. Matthew's Stability Analysis was used in designing the stopes. All faces meet the criteria of being stable or stable with support. The back falls into the stable with support category and will be supported with 12 m double strand bulbed cables drilled in a fan pattern from the drill crosscut.

- Each stope will be approximately 160,000 tonnes and 15 to 16 stopes will need to be cycled each year to maintain the production rate of 2,450,000 tonnes per year.

- To meet daily production requirements will require 12 to 14 stopes in the mining sequence; 4-5 stopes in the drill cycle, 4-5 stopes in the mucking cycle, and 4 stopes in the filling cycle.

- Drilling off a stope is scheduled at 3 to 4 weeks, blasting and mucking out at 2.5 months, and backfilling at 16 to 18 days.

- Development has been scheduled so it is well ahead of the mining requirements and mining takes place on more than one level simultaneously.

- Panel mining allows two stopes in a crosscut to be mined vertically in one year. After the lead stope has been mined vertically, the third stope on the third level can be mined simultaneously with the second stope on the first level. From this point, two stopes can always be mined in a crosscut at the same time.

16.13.2 Underground Mine Development Schedule

The development schedule ensures development is in place approximately 6-12 months before production mining is required.

Development metres are based on preliminary level plans generated from the block model with lateral development centre lines applied to the plans to access all the stoping areas scheduled in the potentially economic mineralisation production schedule. Ramping and raising connect the different levels with quantities determined, accordingly. A 10% additional development factor was applied to all ramp metres to account for safety bays and flattening out for level accesses. Table 16.5 and Table 16.6, below, presents the development schedule for LOM.

TABLE 16.5
LIFE OF MINE VERTICAL DEVELOPMENT SCHEDULE

	Year -3	Year -2	Year -1	Year 1
Alimak Shaft		660		
Ramp Vent Raise Up	410			
Loading Pocket Vent Raise		250		
North Vent Raise			410	
South Vent Raise			410	
Ore Pass Systems to 940 Crusher Level, including fingers		400	300	700
Coarse Ore Bins Below Crushers	#2 Cr	128	m^3 Eq	
	#1 Cr	128	m^3 Eq	
Fine Ore Bin for Mill		250	m^3 Eq	
Mill Thickener and Central Raise		130	m^3 Eq	
Cement Storage Silo		25	m^3 Eq	
Total Vertical Development	**410**	**1,971**	**1,120**	**700**

TABLE 16.6
LIFE OF MINE LATERAL DEVELOPMENT SCHEDULE

		Year -3	Year -2	Year -1	Year 1	Year 2-16
Twin Ramps West Side	14982	4410	4410	4968	1194	
East Exploration Drift Slashing	1700	1700				
Internal Up Ramp	3207	960	1745	502		
Internal Down Ramp	1403		1403			
1450 Level			218			
1420 Level			627	1342		
1375 Level			50			
1330 Level			50			
1285 Level			50	205		
1240 Level			50		846	
1195 Level			50	333	900	
1150 Level		50	50		1250	
1105 Level		50	423	590	1045	
1060 Level		50	430	1135		
1020 Level			65			
980 Level			65			
940 level			300	265		
910 Loading Pocket			304	584		
880 Level				366		
Lateral Development Totals		**7220**	**10290**	**10290**	**5235**	**16230**

16.13.3 Mine Production Schedule

The mine production schedule is based on mining 7,000 tonnes per day of potentially economic mineralisation, for 357 days per year for a yearly mine production rate of 2,500,000 tonnes.

The production schedule is derived from scheduling all the potentially economic mineralisation above 0.25% MoS_2 between the selected mining levels (Table 16.7, below).

TABLE 16.7
LIFE OF MINE PRODUCTION SCHEDULE

	Year 1	Year 2	Year 3	Year 4	Year 5	Year 6	Year 7
Tonnes	1,625,000	2,500,000	2,500,000	2,500,000	2,500,000	2,500,000	2,500,000
Grade	0.50%	0.45%	0.45%	0.40%	0.40%	0.40%	0.34%
	Year 8	**Year 9**	**Year 10**	**Year 11**	**Year 12**	**Year 13**	**Year 14**
Tonnes	2,500,000	2,500,000	2,500,000	2,500,000	2,500,000	2,500,000	2,500,000
Grade	0.34%	0.34%	0.31%	0.31%	0.31%	0.31%	0.31%
	Year 15	**Year 16**	**Year 17**	**Year 18**	**Year 19**	**Year 20**	**Total**
Tonnes	2,500,000	2,500,000	2,500,000	2,500,000	2,500,000	2,500,000	**49,125,000**
Grade	0.30%	0.28%	0.28%	0.25%	0.25%	0.25%	**0.34%**

16.14 MINE SURFACE INFRASTRUCTURE

Surface facilities will generally be centred near the twin portals.

Surface support facilities will include explosives magazines, two cement silos, a small shop and mine rescue station, a mine change house, power sub-station, laydown yard, and water collection ponds.

16.14.1 Explosives Magazines

The explosives magazine would be located 500 m from any facility, including the mine portal. The actual magazines would be provided and permitted by the explosives supplier.

The area would be cleared and a gravel base laid. The shipping containers used to store the explosives and detonators would be raised off the ground to assist in the transfer of explosives from the delivery trucks to the magazines. The area would be fenced around its entire perimeter with a locked gate access. The area would be provided with lighting. Outside the fencing, a berm of several metres height would be constructed to contain any potential explosions in the magazines.

16.14.2 Other Facilities

A mine dry and small warehouse/maintenance shop/office complex will be constructed near to the twin portals. A mine laydown yard will be constructed near the portals to store materials and equipment required for the underground mine. This laydown yard would have raised timber stands on which to place large materials, such as screen, pipe, etc., as well as gravel graded areas for storing equipment and materials. Any materials requiring cold storage will be stored in the ventilation crossovers and re-mucks in the twin ramp system. The main warehouse will be located underground. Spare components requiring heated storage can be stored in a facility in Smithers.

All underground mine discharge water would be sent to the water treatment facility and re-used or discharged.

A fully equipped mine rescue station is required on the property and will be incorporated into the dry/warehousing building. The mine rescue station will be equipped with all the necessary equipment, including self-contained breathing apparatus, flame lamps, gas testing equipment, rescue equipment, etc., and supplies and chemicals required to operate the station. There will be enough equipment to, in an emergency, have two 5-person mine rescue teams operating or on standby at any one time.

The mine will be technically supported by the geology and engineering departments. The geology department will be responsible for mapping and interpretation, sampling of production drill holes, grade control, and Reserve estimations. They will also undertake any exploration work on the Property to prove up new Mineral Resources for potential mining. The engineering department will be responsible for mine planning, production scheduling, surveying, geotechnical design, collecting, and reporting performance statistics for the mine and any other technical requirements that support the operation.

16.15 FLOTATION

Flotation will comprise a rougher scavenger circuit followed by two stages of concentrate cleaning with re-grind. Note that this differs from previous designs that incorporated four stages of cleaning with multiple stages of re-grind. Previous designs did not use modern upgrading equipment, such as column flotation and stirred media re-grind.

Cyclone overflow from the grinding circuit will be floated in a rougher circuit comprising 3 tank cells and a scavenger circuit, also comprising 3 tank cells. Each cell will be 4.5 m diameter × 4.5 m high and have a volume of 70 m^3. Retention time will slightly exceed the total of 33 minutes. The tailings from the rougher circuit are final tails.

Rougher concentrate will flow to the cleaner flotation circuit. The circuit will have two flotation columns or equivalent, such as the Glencore Jameson Cell or the Woodgrove staged flotation reactor. Rougher concentrate combines with the second column cell tails to feed the re-grind mill. First cleaner concentrate will feed the second cleaner cell. First cleaner concentrate will combine with the rougher circuit tails, and hence, are final tails. Re-grind will be via either a Glencore Isa mill or a Metso HIG mill. Both are stirred media mills ideally suited to controlled concentrate regrind.

16.16 GRADE CONTROL

Grade control will be conducted by geological technicians and performed on a daily basis. Material grades will be measured and compared throughout several locations of the process, including the muck pile in each heading, concentrator feed belts, and concentrate and tailings handling locations.

16.17 UNDERGROUND PERSONNEL

The underground workforce is anticipated to be initially contracted during pre-production then transition to an owner operated workforce in Year 1 of operation. Significant training will be required throughout the entire project life. The local skillset is mainly industrial. Timber and mechanical industries are prevalent, which carry skillsets that are similar to mining, such as equipment operation and repair. More highly specialised skillsets will take longer to train. Table 16.8, Table 16.9, and Table 16.10, below, shows the anticipated manpower compliment in the mine.

TABLE 16.8 UNDERGROUND MINE MANPOWER COMPLEMENT	
Production	**Complement**
Blasting	16
Mucking	16
Drilling	12
Cable Bolting	4
Backfilling	8
Total	**56**
Services	
Serviceman	12
Grader Operator	8
Construction/Services/Backfill Leader	1
Construction/Services/Backfill Helper	4
Lamp Room/Dryman	4
General Labourer	4
Crushermen	4
Total	**37**
Development	
Development Miner	6
Total Mine Department	**99**

TABLE 16.9 UNDERGROUND MAINTENANCE COMPLEMENT	
Maintenance Department	
Leadhand Mechanic	4
Mobile Mechanic	4
Mechanic	12
Mechanics Helper	4
Electrician	8
Electrician Helper	4
Stationary Mechanic	4
Total Maintenance Department	**40**

TABLE 16.10
MANAGEMENT AND SUPPORT STAFF

Management and Support Staff	
General Manager	1
Comptroller	1
Accountant	2
Head of Health/Safety and Security	1
Environmental Manager	1
Environmental Technician	2
Office Clerk/Secretary	1
Purchasing Agent	1
Warehouseman	4
Warehouse Stocktaker	1
Medical Services (Contract)	1
Security Contract	3
Mine Superintendent	1
Mine General Foreman	2
Mine Supervisor	4
Mine Services Supervisor	1
Mine Trainer/H&S Coordinator	2
Clerk/Secretary	1
Chief Engineer	1
Mine Engineer	2
Mine Planning Technician	2
Ventilation/Surveyor Technician	2
Chief Geologist	1
Mine Geologist	2
Geological Technicians	4
Total Management and Support Staff	**44**

17.0 RECOVERY METHODS

The 7,000 tonnes per day processing facility will be built entirely in the underground mine. The mill will be located above the main potentially economic mineralisation zones at the 1420 Level elevation. Locating the mill underground will serve several purposes; reduce the surface footprint and visual impact, negate the necessity of moving 7,000 tonnes of material from the mine to an outside processing facility, and provide a ready source of backfill material by utilising the mill tailings as paste fill. No other source of backfill is currently available and mining without backfill would reduce the overall ore recovery by approximately 50%. Putting a quarry on the surface for cemented rockfill would require permitting a 5,000 tonne per day open pit waste mine and would necessitate crushing and moving this material underground to be used as fill (Figure 17.1, below).

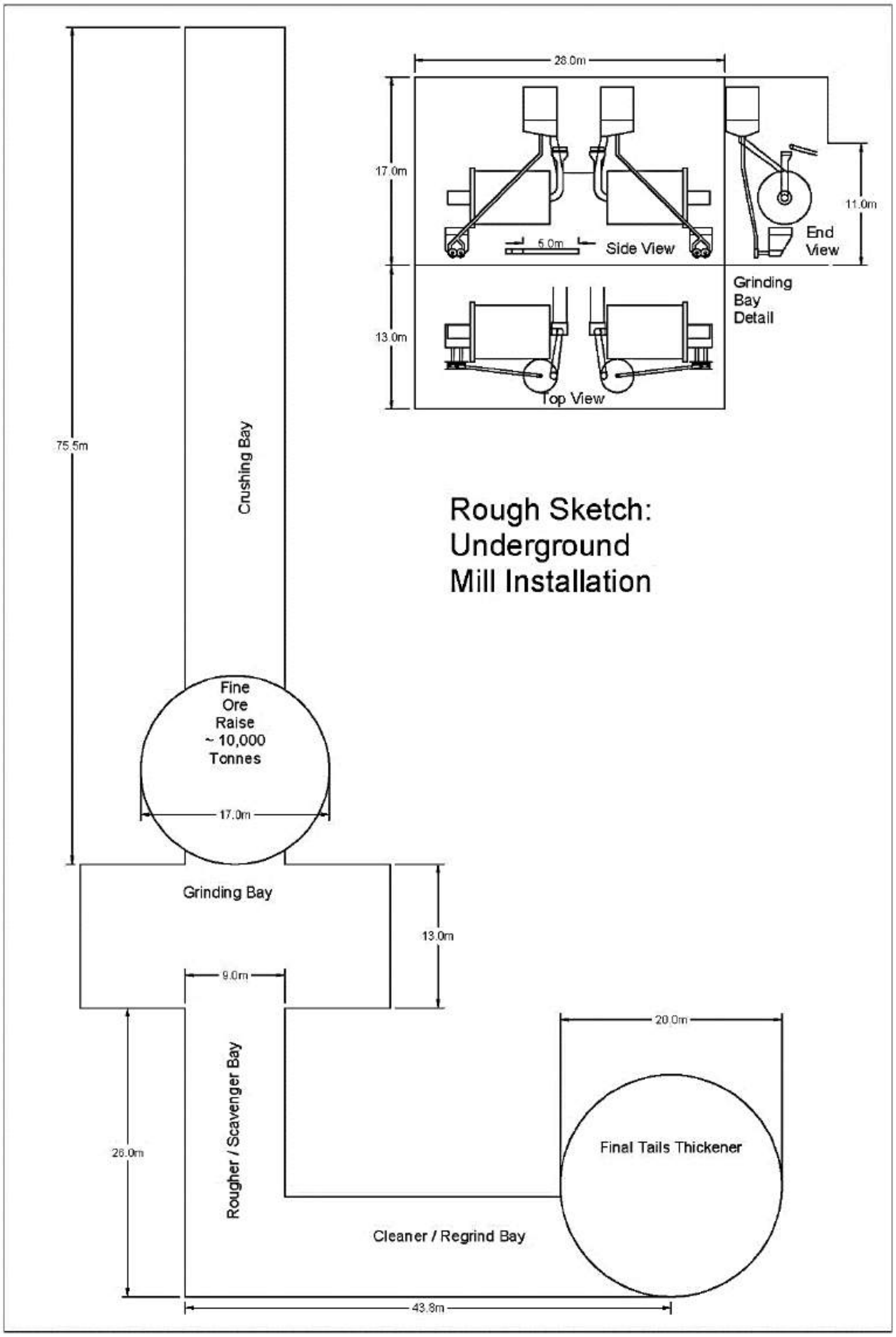


Figure 17.1. Underground Processing Plant Preliminary Design

Source: Eggert Engineering, 2024

The following flowsheet description is based primarily on designs from previous studies on the Davidson Project. Process criteria, include:

- Production rate of 2,450,000 metric tonnes per year;
- Final primary grind size (p80);
- Flotation stage retention times;
- Expected mass pull to flotation concentrates; and
- Crushing and grinding work indices.

Modifications have been made that assume improvements in the process were made possible by more modern equipment. These changes must be confirmed by bench scale and pilot scale testing to confirm the assumptions made.

Figure 17.2, below, is a simplified overall flowsheet of the mill. The flowsheet is to a PEA level only. The equipment layouts, sizes, etc., must be confirmed by further testing. If the Project progresses to a PFS, locked cycle tests will suffice. If the Project progresses to an FS, pilot plant tests will be necessary. These should be sufficient for detailed equipment specification and design for installation.

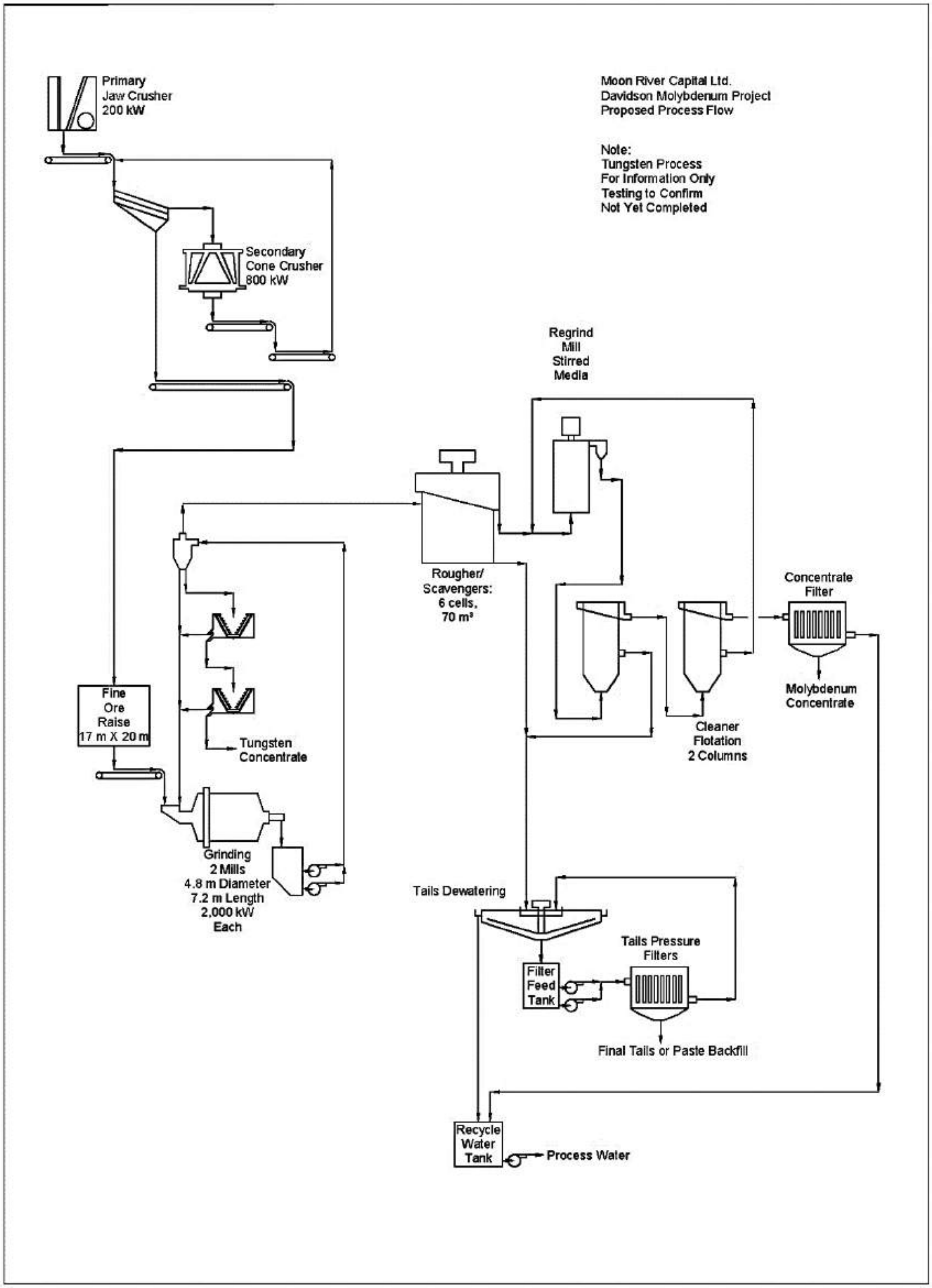


Figure 17.2. Processing Plant Flowsheet
Source: Eggert Engineering, 2024

17.1 CRUSHING

Crushing will be accomplished in two stages. The first stage is an open circuit jaw crusher. Feed to the jaw will be by a pan feeder located on the discharge of the coarse ore bin. The pan feeder will be controlled by a level sensor at the feed to the crusher. There will be two independent primary jaw crushers at the bottom of the two ore pass systems. Each will feed directly into a coarse ore bin and each bin will independently feed the secondary cone crusher.

Primary crusher discharge will be conveyed to a double deck screen. The top deck will be relatively coarse to protect the second sizing deck. The screen will be sized to accept 100% of the primary crusher discharge.

The fine side from the screen will discharge to a conveyor common to the screen and the secondary cone crusher. The coarse side of the screen will feed a surge bin ahead of the cone crusher. Feed to the cone crusher will be by a belt conveyor equipped with a variable frequency drive and a metal detector to protect the cone crusher. The speed of the drive will be controlled by the level in the cone crusher to assure choke feeding to the crusher. The level in the cone crusher feed bin will control the feed rate to the jaw crusher.

The secondary cone crusher will operate in closed circuit with the screen. Fine material from the crushing circuit will be conveyed to the vertical lift conveyor system, which will dump the crushed material into the fine ore bin. The raise will be sized to hold 24 hours of grinding circuit feed or at least 5,000 m^3.

17.2 GRINDING

Grinding will be accomplished by two 4.8 m diameter × 7.2 m long ball mills operating in parallel. The mills will be sized to achieve a target power while operating as an overflow discharge mill. Expected mill power is 2,000 kW per mill. There will be no discharge grate in the mill. The mill will have a trommel screen, which will eliminate some of the wood and plastic from the fresh feed.

Feed to the grinding circuit will be by two apron feeders per mill to reduce rat hole development in the fine ore bin. An alternate design comprising two fine ore raises, one per mill is being considered.

The required opening to accommodate the grinding circuit is 9 m wide × 37 m long × 18 meters high.

The fine ore raise will be 15 m^2 × 25 m high. This will provide up to 24 hours of material to allow for crusher maintenance.

Ore will be classified by hydrocyclones to produce feed for the flotation circuit. The underflow from the cyclones will be split with approximately 75% of the underflow flowing directly to the ball mill feed and 25% being directed to a gravity concentrator – either a Falcon Concentrator or a Knelson Concentrator. Tails from the gravity concentrator will combine with the remainder of the cyclone underflow, and hence, be directed to the ball mill feed. Concentrate from the concentrator will be further treated by a second concentrator. Concentrate from the second concentrator will be the tungsten concentrate. Tails from the second concentrator will periodically be discharged to the feed of the ball mill.

17.3 DE-WATERING

Two streams require de-watering. The concentrate stream must be suitable for sale. This requires a relatively dry material, which would be produced by a pressure filter. The mass flow of concentrate is relatively small.

A 'Larox' style pressure filter is specified. This filter has a smaller foot-print than a plate and frame filter, that make it more suited to an underground installation.

The final tails stream will feed a paste backfill plant for use in cemented backfill in the mined-out workings. The paste backfill will be 80% solids. This will be achieved by pressure filtering a portion of the final tails to 90% to 95% solids and combining this with the remaining final tails to produce paste fill at 80% solids. One hundred percent of the final tails will be used for paste backfill, when openings underground are available. When openings are not available for backfill underground, the paste will be deposited on the surface as thickened tails. The specification for the moisture of the filter cake is to allow the tails to be deposited as a dry stack tails.

17.4 DESIGN RISKS

The process design this study proposes is not unusual. The installation underground is. Though this should not present any significant issues in terms of installation, it may result in unexpected costs. If the Project continues to a PFS or FS, the increased detail will mitigate these risks.

Crushing efficiency may require that the stope ore be blasted in such a way as to produce a relatively small top size of less than 50 cm. The proposed crushing and grinding circuit can accommodate material up to 90 cm.

The proposed circuit includes gravity concentration in the primary grinding circuit to produce a tungsten concentrate. This has not been studied or tested. As such, there are no expectations of any tungsten production. Though this is flagged here as a risk, it also presents a significant opportunity.

Previous reports (AMAX, 1980) have indicated that there may be radioactive materials in the potentially economic mineralisation. This needs confirmation and may need to be evaluated.

There is no information in any of the reports of the acid generating potential of the potentially economic mineralisation. AMAX (1980) indicates that the potentially economic mineralisation becomes oxidised over time implying sulphide oxidation, and hence, acid generation. This could be mitigated by directing 100% of the cleaner flotation tails to cemented backfill with desulphurised tails only reporting to the surface.

17.5 OPERATING EXPENSES

For the purposes of this PEA, the operating expenses, as summarised in the 2008 Hatch Study, of $16.30 were used as the baseline. The Hatch costs are excessively high for a project of this nature. An alternate review was conducted to assess the expected OPEX for the processing facilities. Estimated mill OPEX, as detailed below, is $10.94 per tonne or $25,500,000 per year in 2024 Canadian Dollars.

17.6 OPEX DETAILS

The OPEX elements are summarised as follows:

- Mill Consumables are estimated at $8 per tonne – This is considered conservative and provides a contingency to the OPEX.
- Crusher Power is 2 jaws at 200 kW, 1 cone at 800 kW ancillary equipment at 400 kW.

- Total Crusher Power is 1,600 kW.
- Mill Power is 2 mills at 2,000 kW each. Ancillary equipment at 3,100 kW
- Total Mill Power is 7,100 kW.
- Power for cost estimating is 8,700 kW × 24 hours/day = 208,800 kWhr/day. At $0.061/kWhr power is $12,820 per day. Power cost is $1.82 per tonne.
- Environmental costs are estimated at $750,000 per year or $0.30 per tonne

Manpower costs estimates are presented in Table 17.1, below, which assumes 4 persons are required to provide 24 hours per day, 7 days a week coverage for one position.

TABLE 17.1
ESTIMATED MANPOWER COSTS

Position	Salary	Number per Shift	Per Day	Annual Compensation	Total Cost ($)
Labourer	$40,000	2	8	$320,000	**$464,000**
Grinding Operator	$55,000	1	4	$220,000	**$319,000**
Flotation Operator	$60,000	1	4	$240,000	**$348,000**
De-watering Operator	$50,000	1	4	$200,000	**$290,000**
Millwrights	$65,000	2	2	$130,000	**$188,500**
Electricians	$65,000	2	2	$130,000	**$188,500**
Instrumentation Tech	$65,000	2	2	$130,000	**$188,500**
Assayers	$70,000	2	2	$140,000	**$203,000**
Shift Supervisors	$80,000	1	3	$240,000	**$348,000**
Metallurgist	$90,000	1	1	$90,000	**$130,500**
General Foreman	$90,000	1	1	$90,000	**$130,500**
Mill Manager	$125,000	1	1	$125,000	**$181,250**
Total Manpower Costs				**$2,055,000**	**$2,979,750**

Total OPEX is, hence, estimated as $10.94 in Table 17.2, below:

TABLE 17.2
TOTAL OPEX COST

Component	Cost
Manpower	$2,979,750
Mill Reagents/Consumables	$20,000,000
Environmental	$750,000
Power	$4,551,214
Total Annual Mill OPEX	**$27,356,214**
Total Cost per Tonne Mined	**$10.94**

18.0 PROJECT INFRASTRUCTURE

The Project is located nearby to Smithers, which could support and provide services to the mine workforce. This section describes the infrastructure required to support the mining operation.

18.1 EXISTING INFRASTRUCTURE

This Project is a combination of greenfield and brownfield areas. There is an existing underground portal and limited underground development on the east face of Hudson Bay Mountain while the new twin access drifts will be driven from a new location on the west face of the mountain.

The right-of-way on the east side of the mountain has been established by a main provincial highway, a gravel road off the highway, and a narrow road constructed in the 1960s with switchbacks up the mountain to the existing portal. This road is not much more than a track through the bush and is in need of upgrading before it could be used to support a development project from the east portal. In the photo below (Photo 18.1), the track of the road can be seen by the yellow tops of the trees.

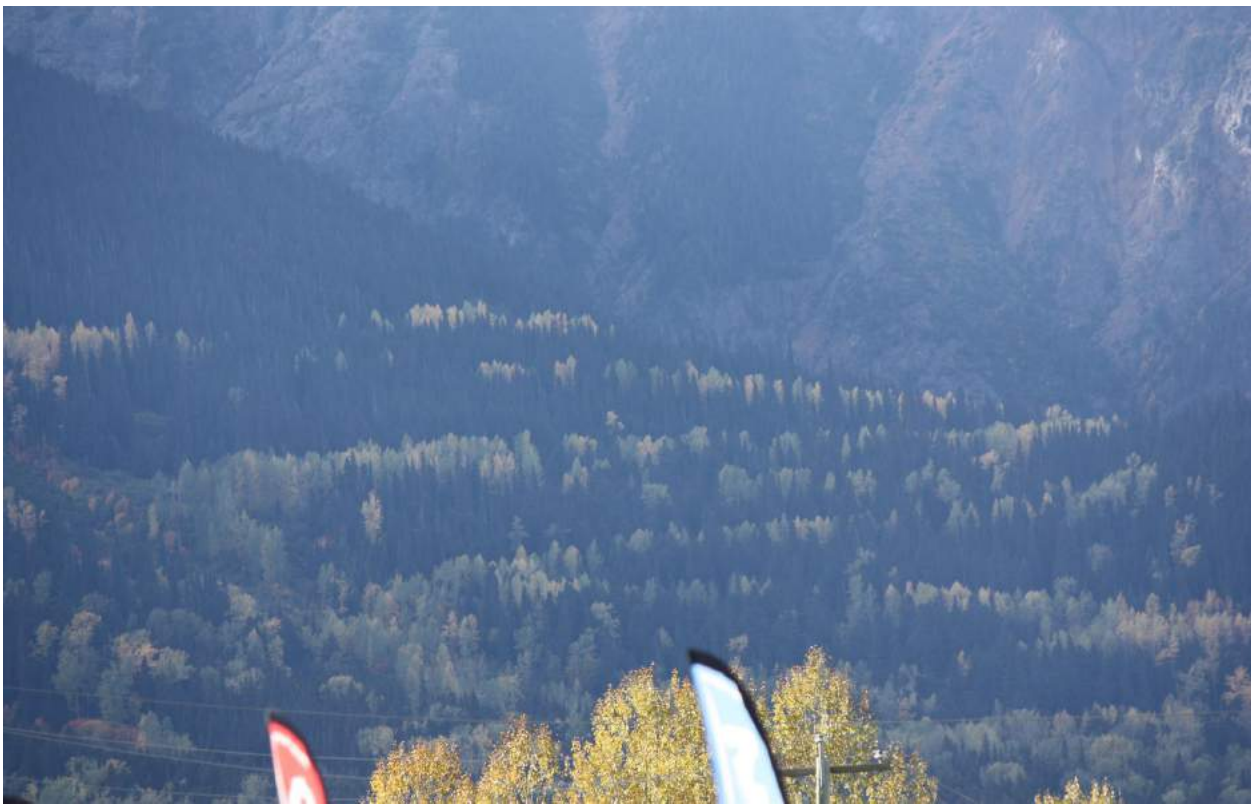


Photo 18.1. Track of the Road Seen by the Yellow Tops of the Trees
Source: AMPL, 2023

The west side of the mountain is accessed via an active logging road, which connects directly with the main highway through Smithers. The location of the twin portals will be approximately a kilometer north of the logging road. This road is currently active and is highly used by locals for accessing recreation areas and lakes. In order to support a mining project, the road will need to be upgraded and a new road constructed from the existing road to the portal location (Photo 18.2, below).

Photo 18.2. Overview of the Mine Road
Source: Goggle Earth™

As the Project is located in a highly used and scenic recreation area, the design of the Project seeks to minimise any disturbance and visual impact. As much infrastructure as possible will be located underground in order to do this. Offices, warehousing facilities, and the processing plant will all be located underground.

Surface infrastructure required would include:

- Upgrading of Access Road;
- Powerline Construction;
- Electrical Sub-stations and Distribution;
- Site Roads and Materials Handling Area;
- Maintenance Shop/Offices/Dry/Warehouse Complex (temporary);
- Two Cement Storage Silos;
- Water Supply System and Water Treatment Plant;
- Dry Stack Tailings Impoundment Area;
- Development Waste Storage;
- Landfill Site; and
- Sewage Disposal Site.

A site plan for the project is shown in Figure 18.1, below.

Figure 18.1. Site Plan

Source: AMPL, 2024

18.2 MINE ROAD ACCESS

Approximately 20 km of road requires construction or upgrade, to allow heavy truck traffic to access the site. Construction will include clearing to the required width of the right-of-way; placing road base, installing culverts, and capping the entire road surface with granular material of suitable type. The switchback road up the east side of the mountain will have to be rehabilitated as well in order to stage development through the east portal.

18.3 POWER LINE TO SITE

Primary electrical power for the mine would be provided from the main surface sub-station connected to the outside power grid. There is a 138 kV line that services Smithers and the surrounding communities and goes up the east side of Hudson Bay Mountain as far as Hazelton. The 2006 Hatch Study indicated that this line would need to be upgraded in order to supply the Project with sufficient power. This Project is expected to have much higher electrical demands with a processing plant onsite and a vertical conveyor hoisting system and crushing systems underground. Electrical demand is estimated at 25 kV.

There is a 500 kV line south of Smithers that services the Terrace, British Columbia area. No contact has been made with BC Power; however, it is expected that this line could supply the power necessary for the Project. A dropdown connection to a 44 kV transformer would be made at the 500 kV line and a 17 km power line constructed to the mine site (Figure 18.2, below).

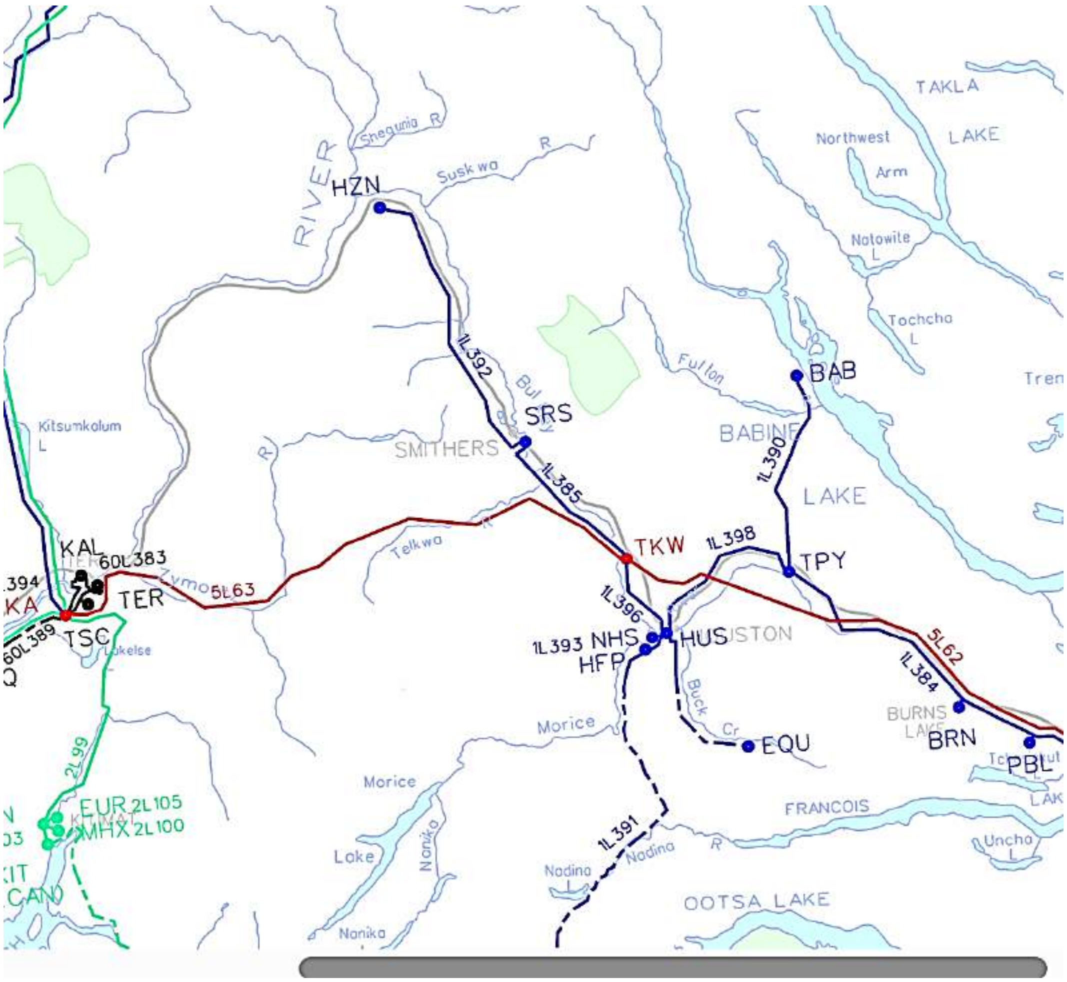


Figure 18.2. BC Power Grid in Smithers Area

Source: BC Hydro

At the site, the power would further be reduced to supply the underground with the necessary power as required (4,160 V for large motors, 1,000 V, and 600 V for other uses) with panels established for low-voltage, single phase (220/110 V) needs.

The new sub-station feed to the underground will be set up with two independent lines, with a disconnect switch allowing use of either line-up independently and interconnection to power systems for emergency needs.

Back-up diesel generation for pumps, fans, and the ore processing facility would be required.

18.4 WATER SUPPLY

Plant and process water, as well as fire water, will be sourced from a river or lake close to the site. All development from the east portal will utilise a storage tank and flows from DDH in the exploration drive. In the 2006 Hatch Feasibility Study, inflows from the DDH were estimated at 105 litres per minute, more than sufficient to support development from the east portal.

On the west side of the mountain, Aldrich Lake is close to the portal and could supply all water necessary for development and processing.

18.4.1 Plant and Process Water

A main objective of the design is to minimise the overall water usage requirements. It is anticipated that 80% to 90% of the water used in the process would be recycled from the mine/mill process water pond, with 10% to 20% being made up with fresh water from the fresh water source.

Process water will be treated, as necessary, to maintain low turbidity. Any water being sent back to the environment will be treated to meet Provincial regulations.

Gland water will be taken from the make-up water to ensure minimal turbidity in the process. Water testing of the fresh water source will be carried out prior to detailed design to assess the need for filtration of this water source.

18.4.2 Fire Water

Fire water will be drawn from Aldrich Lake and stored in a fire water tank adjacent to the mill facility. Diesel-powered generators will power the fire pumps throughout the plant and the tank will be of sufficient size to meet Factory Mutual (FM) requirements for the facility.

18.4.3 Potable Water

The potable water system also includes the process water system that needs to meet or exceed dissolved solids that may interfere in the extraction process, notwithstanding the ability to use as a source for drinking and bathing. Potable water and clean service water will be treated with a combination of reverse osmosis filters and chlorination to ensure the water meets all regulatory guidelines. Potable water will be pumped to a storage tank and kept for use in all drinking and bathing.

18.5 PROCESSING FACILITY

The processing facility will be located in the underground, above the ore zone at the 1420 Level. Locating the mill underground will serve several purposes; reduce the surface footprint and visual impact, negate the necessity of moving 7,000 tonnes of material from the mine to an outside processing facility, and provide a ready source of backfill material by utilising the mill tailings as paste fill. No other source of backfill is currently available and mining without fill would reduce the overall ore recovery by over 50%. Putting a quarry on the surface for cemented rockfill would require permitting a 5,000 tonne per day open pit waste mine and would necessitate crushing and moving this material underground to be used as fill.

The only facilities required on the surface would be the Filtered Tailings Storage Facility (FTSF) and the two cement storage silos capable of storing 250 tonnes of cement each. The cement dust would then be transferred underground via stainless steel pneumatic trailers to an underground silo associated with the paste fill plant. Approximately 250 tonnes per day of cement would be required to maintain paste filling.

The FTSF will store filtered tailings from processing of the ore, will be located south of Hudson Bay Mountain, and will consist of a side hill impoundment (Figure 18.3, below). Perimeter earthfill embankments will be constructed to provide containment around the north and east perimeters of the facility. Natural topography will provide containment along the west and south perimeters of the facility.

Figure 18.3. Filtered Tailings Storage Facility
Source: Newfields, 2024

Based on current estimates, the FTSF will store a total of approximately 25 Mt of filtered tailings. It is not known if the tailings will be potentially acid generating (PAG). As a result, at this stage, it is expected that the facility will be fully lined using a 60 mil (1.5 mm) thick Linear Low Density Polyethylene (LLDPE) liner system with a sand bedding layer below the liner. Perimeter seepage and runoff collection ditches will be provided to collect any seepage and runoff from the facility.

18.6 WATER TREATMENT PLANT

Pricing of $10 million has been included for a water treatment plant to treat water from the mine and surface facilities before discharging to the environment.

18.7 SHOP AND WAREHOUSE FACILITY

A small breakdown shop will be set up during the pre-production period in both accesses in an abandoned re-muck off the ramp. This shop will be used until a permanent breakdown shop is located midway in the mine. The mobile equipment maintenance shops would be used to perform all breakdown and service maintenance on mobile mining equipment. Major equipment re-builds will be done in a facility in Smithers.

The permanent shop would be constructed on the 1150 Level, off the ramp. The shop would consist of a main shop area for one large piece of equipment or a couple of smaller units. The facility configuration would consist of an access drift leading to the main shop area, two additional repair bays, a welding area, wash bay area, parts storage warehouse, tool crib, electrical room, lunchroom, and supervisor’s office.

The main shop area would be equipped with an overhead bridge crane in each repair bay. The electrical room, meeting room, and office would be isolated by steel hinged doors. The lunchroom would be equipped with wooden benches and tables and the office would be equipped with computer workstations connected to the mine information management system. The shop and warehouse will be located in the underground. A main shop will be located on the 1150 Level and consist of two main service bays, a central contained in one building. There will be one small shop facility on the surface to service mobile equipment and one for stationary/electrical and specialty gear; $1.5 million has been included for equipping the main shop.

A main underground warehousing facility would be provided with areas for pallet shelving storage of materials and parts, a lockup area for supplies, and office space for purchasing and warehousing personnel. Laydowns for large material and equipment can be located in walled off ventilation cross overs and disused re-mucks along the main tunnel access. Excavation costs have been included in capital development and $200,000 has been included for construction of the floors and walls.

As both of the west side twin access tunnels will be fresh air intakes, a mine rescue station can be located at the first cross over/re-muck.

18.8 CAMP

No camp facility will be constructed at the site. The town of Smithers is 12 km away from the mine site and should be able to provide accommodations for workers that do not relocate to Smithers. An allowance for 50% of the onsite workforce has been allocated in the cash flow model at $150 per day per person.

18.9 FUEL STORAGE

Fuel pads and waste oil depots need to be constructed to ensure any spillage will be contained and, in the event of a fire, a method to prevent the spread to other infrastructure or surrounding bush. An earthen structure and catchment pad is included in the design; $100,000 has been included for construction of the containment and purchase of the fuel tank. Minimal fuel facilities will be required as the majority of equipment is electrically powered.

18.10 PROPANE

Propane will be stored in large, high pressure propane tanks supplied by the propane supplier. The propane tanks will have a protective berm surrounding them to prevent any damage caused by a potential explosion. Initially, storage facilities will be required at both the east and west portals; however, once the west ramps have joined up with the rest of the mine, the east side heating system will no longer be required; $50,000 has been included for the propane storage.

18.11 QUARRY/ROCK DUMP

A local quarry will supply aggregate for the road construction. According to Mr. Davidson, who has been associated with the property since 1966, initial testing of the waste development from the east portal shows no acid rock drainage (ARD) potential. During a site visit in September 2023, no visible staining was apparent on the waste pile. Subject to additional testing, the waste from the underground will be used as road base material for the development drives on the west side. A small, portable crusher can be set up to crush the material.

Should the waste prove to have ARD potential, the waste stockpile will be located in the Zymoetz River watershed, as will the dry stack tailings facility. All run-off from the waste stockpile will be captured and treated at the water treatment plant prior to release to the environment.

Berms and drainage systems containing water and preventing seepage are designed to handle all waste rock from underground pre-production development.

18.12 EFFLUENT POND

Water management will be a series of collection ditches and ponds used to collect impacted water from around the Property outside of the dry stack tailings facility. Water drawn from the tailings facility will be either treated before release or re-circulated back into the processing facility as process water. The collected surface impact water, along with mine discharge water, is pumped into a raw water collection pond. This water is then treated through a water treatment facility. Treated effluent water that achieves background or better water quality is then discharged into a clean water holding pond. Water from the clean water holding pond is then re-used in the mining and milling process and excess water is allowed to discharge to the environment via several septic fields named potential discharge points (PDP). These discharge points function in such a way to ensure the released water weeps (disperses) back into the ground water below the surface as it would if the project did not take place.

18.13 SMITHERS ADMINISTRATION OFFICE

The main administration, purchasing, and accounting facilities will be located in the community of Smithers. All other activities will be located in underground offices.

18.14 EPCM COSTS AND FIRST FILLS/SPARES

The EPCM costs on the infrastructure works is estimated at CA$1.75 million and the first fills/spares for equipment is CA$150,000. **Note:** Please refer to Table 21.3, below, for detailed cost estimates on the underground infrastructure.

19.0 MARKET STUDIES AND CONTRACTS

The mine will produce a molybdenum concentrate that will be sold to a smelter(s) for further processing to metal.

Smelter payment prices are based on paying mines for molybdenum metal contained in MoO_2 concentrate minus smelter charges. This PEA has used the 2-year average moving price for February 2022 to end of January 2024, published by Metal Platts, for the long-term price to be paid by smelters. Metal Platts is the primary recognised source for prices for molybdenum concentrate sales pricing worldwide.

The long-term MoO_2 price, based on the 2-year moving price included in this study, is US$47.39 per kilogram (kg) or US$21.50 per pound (lb).

20.0 ENVIRONMENTAL STUDIES, PERMITTING, AND SOCIAL OR COMMUNITY IMPACT

The Davidson Project area and mineral deposit have been the subject of a past project development proposal by Thompson Creek Metals Company (TCMC), formerly Blue Pearl Mining Limited. In 2008, a Feasibility Study was completed by Hatch Ltd., which proposed to develop the Davidson molybdenum deposit as an underground mine producing 2,000 Mt of ore per day for 10 years with ore hauled to the Endako mill for processing. The application for an environmental assessment was submitted by TCMC in September 2008 to the Environmental Assessment Office (EAO), initiating a review period. In December 2008, the EAO suspended the time limit for the environmental assessment completion to allow TCMC to provide additional information required for the EAO to be able to complete the EA. In 2011, TCMC informed the EAO that they would not request the Project be re-initiated, and the EAO officially terminated the EA.

Baseline social and biophysical data was collected between 2005 and 2009 to support the design and assessment of the Project. The data, studies, and reports are helpful for background information and to inform and refine study designs for future engineering and baseline work, but they will not suffice for future assessment and permitting processes because they are dated and non-continuous. Since 2008, the regulatory and social requirements for assessment and engagement in British Columbia have changed significantly. The Project details, assessment application, and documentation of all aspects of the environmental assessment process for the Project proposed by TCMC in 2008 are available at the BC Ministry of Environmental and Climate Change Strategy website for the Environmental Assessment Offices Project Information Centre (EPIC) https://www.projects.eao.gov.bc.ca/p/588510eeaaecd9001b817e54/project-details.

The current Project contemplated by Moon River differs significantly from that previously proposed in location, footprint area, project components, proposed site access routing, and concentrate handling. Project-specific scoping for environmental, social, economic, heritage, and health studies baseline data collection will be conducted and reviewed periodically as the Project advances through trade-offs and engineering studies.

20.1 TERRESTRIAL ECOLOGY

The Biogeoclimatic Ecosystem Classification Subzone/Variant Map for the Bulkley Subunit (2021) indicates that the Project area and supporting infrastructure area are within three bio-geoclimatic sub-zone classifications; Engelmann Spruce Subalpine Fir (ESSFmc), Interior Cedar Hemlock (ICHmc1), and Sub-Boreal Spruce (SBSmc2). Most of the Project area comprises young and mature trees, predominately conifers, with some mixed conifer-broadleaf stands present. Wetlands and very wet forests are sensitive ecosystems in the area. The identification and location of possible rare, endangered, or culturally significant plant species, as well as any invasive plants, will also form part of the baseline effort required to support environmental assessment and permitting processes and will be informed by engagement activities with stakeholders and First Nations.

20.1.1 Terrain Soils and Surficial Geology

The Project is within the Bulkley Ranges on the boundary between the Skeena and Hazelton Mountains. The surface elevation at the mineral resource area is approximately 1,900 m, and the contemplated mine waste, tailings, and portal entrance area are between 850 m and 1,000 m. The townsite of Smithers is 600 m, and the top of Hudson Bay Mountain, the dominant mountain in the Bulkley Ranges, is 2,500 m. The mountainous terrain is steep, and snow avalanche activity is prominent in alpine terrain and gullies, with avalanche scars extending below the treeline. The surficial geology is of glacial origin, with deep till deposits and localised glaciofluvial and glaciolacustrine sediments. Exposed sediment, gravels, and cobbles indicate active deposition of fluvial and colluvial sediments, and alluvial fans are associated with the Bulkley Range mountains. The contemplated mine waste, tailings, and portal entrance area is in an area of relatively low relief, at the base of the mountain with gentle rolling hills, dense forests, lakes, and meadow to the west.

Terrain mapping for hazard identification has been conducted for the forest stewardship plan that overlaps the Project area. Terrain stability classes IV and V are found in the steep mountainous and glacier areas near Hudson Bay Mountain but not elsewhere in the Project area.

20.2 WILDLIFE AND WILDLIFE HABITAT

Several species of conservation concern potentially occur within the bio-geoclimatic zones in the Project area. The mammal species that are either blue-listed (special concern) or red-listed (threatened or endangered) include Caribou (Northern Mountain Population), Wolverine, Grizzly Bear, Mountain Goat, Hoary Bat, and Little Brown Bat. There is a large, identified area of defined critical habitat for federally listed Woodland Caribou (southern mountain population) 15 km south from the farthest south point of the contemplated Davidson Project.

Migratory avian species that may use the Project area for parts of their life cycle include: Northern Goshawk, Western Grebe, Great Blue Heron, Short-eared Owl, Upland Sandpiper, American Bittern, Rough-legged Hawk, Swainson Hawk, Smith’s Longspur, Common Nighthawk, Long-tailed Duck, Black Swift, Rusty Blackbird, Peregrine Falcon, Gyrfalcon, American White Pelican, Red-necked Phalarope, American Golden-Plover, Eared Grebe, California Gull, Surf Scotter, and Wandering Tattler. Lewis Woodpecker, Lark Sparrow, California Gull, and Band-tailed Pigeon are also possible in the Project area and are resident birds that do not migrate. Multi-season bird surveys will be required to characterise use of the area by resident and migratory birds.

Species will be identified as valued eco-system components for habitat suitability, critical habitat identification, and population studies, but the final determination will include input from the species at risk atlas, First Nations, and stakeholders in the Project area.

20.3 WATER RESOURCES

Surficial hydrology, hydrogeology, and water quality studies will be required for the Davidson Project, which encompasses both the watershed in the Mineral Resources mining area and the watersheds to the east in the area of the Project supporting infrastructure, tailings, waste rock, and portal area. There are two major watersheds that are divided by the height of land that runs directly west of the Mineral Resource inclusive of Hudson Bay Mountain. These are Zymoetz River and Bulkley River. These are further divided into sub-watersheds within both major watershed areas. Kathlyn Creek watershed on the east side includes the existing adit area and has been the subject of an application for a designated Community Watershed, which in British Columbia, is for the purpose of drinking water protection. A query of the current, published map of Community Watersheds within the mineral tenure areas of the Davidson Project did not intersect any identified Community Watershed. The environmental baseline study will be designed to capture upstream and downstream sites within watersheds to support the assessment of possible changes resulting from the Project development. The study will be designed in alignment with the BC Ministry of Environment’s (MOE) Water and Air Baseline Monitoring Guidance Document for Mine Proponents and Operators (Interim Version) and will be designed to ensure an adequate baseline for human health risk assessment processes. Proposed study areas, sample parameters, and sample locations will be reviewed with the appropriate provincial agencies, First Nations, and other stakeholders as part of the baseline study design.

20.3.1 Aquatic Eco-Systems

Previous studies have been conducted in areas that overlap the Project area to characterise water quality, sediment quality, and benthic invertebrate communities. Some elevated metal concentrations were observed across the local sampling area, including elevated metals within the adit area waterbodies (**Note**: Mineralised molybdenum outcrops in several locations of the study area). Elevated metal concentrations in area samples likely indicate a localised limnology influenced by the geology. The previous results of sampling in rivers, lakes, and streams that intermittently exceeded standard guidance for toxicity and metals indicate that site-specific water quality values may need to be created to recognise the unique values of the existing water bodies in the area.

20.3.2 Groundwater Quality and Quantity

Groundwater investigations at the site were limited to the Toboggan Creek watershed. As noted above, a full baseline will be required to define the existing hydrogeological conditions and groundwater flow paths to support the assessment of potential environmental effects on groundwater resources in all Project infrastructure areas, which is a significantly expanded area from the previous study area. The previous work in the Mineral Resource area and below in the valley bottom provides background information on hydrogeological flows in the mining area primarily indicating that there is a thick, impermeable layer of glacial till/clay in many areas and that groundwater flow generally follows topography. Groundwater quality sampling previously conducted indicated that water samples collected from underground had elevated molybdenum and arsenic concentrations but that these water quality conditions were not encountered at the mountain base.

20.3.3 Fisheries

The mineral tenure area of the Davidson Project includes a portion of the Toboggan Creek watershed, a watershed classified as a Fisheries Sensitive Watershed (F-6-004) that is subject to an order that requires special management to protect fish. Specifically, the hydrological conditions must be conserved to ensure no adverse impacts on fish habitat within the watershed.

Fish and fish habitats are protected under legislation and there are authorisation processes for the protection of both. The Federal Fisheries Act and the Metal Mining Effluent Regulations enabled by the Act provide for specific protection and monitoring related to the potential adverse effects of mining operations.

The data collected to date on fish and fish habitat in the area and studies conducted by the government and area environmental organisations will be used to refine future studies required to design, evaluate, and permit the mine. Comprehensive fisheries studies are expected to be required in a local study area that encompasses the contemplated portal, waste storage, tailings area, and mine area.

20.4 CLIMATE

A strong precipitation gradient exists between the west coast and Smithers, which is approximately 200 km inland. As moist maritime air masses from the coast move inland, they release much of their moisture on the windward slopes of the Coast Mountains before reaching the Project area.

The average annual precipitation measured at the Smithers Airport is 509 mm, compared with 1,295 mm at Terrace and 2,551 mm at Prince Rupert (Environment Canada, 2023). Maximum participation is in October when an average of 62.3 mm of rainfall occurs.

The temperature, as measured at the Smithers Airport weather station, typically varies from -11°C to 24 C and is rarely below -24°C or above 29°C (Environment Canada, 2023). Winds are generally between 6 kilometres per hour (kmph) and 12 kmph with gusts between 13 kmph and 20 kmph. The Project area's local topography, differential heating, and wind circulation will affect microscale weather patterns and may differ from those at the Smithers Airport. Site-specific and updated meteorology will be required as baseline data to support refined project engineering and regulatory processes.

20.4.1 Meteorology and Hydrology

A meteorology station installation and seven hydrometric stations were constructed in the area previously, and though they were subsequently removed, they will provide background data and study refinements for hydrology and meteorology studies that will be conducted to reflect the proposed project design and infrastructure corridors. Data from the removed stations demonstrated a wide variety of hydro-climatic conditions measured in a 3-year period, which included both a record snowpack year and a 1-in-100-year dry conditions year. This variability reinforces the need for a robust data set to develop accurate predictions. A meteorology station equipped with instrumentation to measure wind speed, wind direction, temperature, relative humidity, and precipitation will be installed within the Project area and hydrometric stations equipped with pressure transducers, dataloggers, and a staff gauge will be installed in watersheds potentially impacted by the Davidson Project.

Several long-standing stations in the region have robust, decades-long datasets, including a snow course location on the south side of Hudson Bay Mountain. Meteorology and hydrometric stations will need to be deployed to collect site-specific data to inform specific mine and infrastructure site conditions. This data will be complemented by the long-term regional data from existing government meteorology and hydrometric stations to develop estimates of run-off, peak flows, and low flows.

20.4.2 Air Quality

The air pollutants of concern in the region and area are particulate matter. Fine particulates are created from sources such as forest harvest debris burning, beehive burners, residential heating, and forest fires. Fugitive dust is limited to that which is mobilised during the early spring on dirt roads. Baseline dust and particulate background data will be required to evaluate the potential incremental effects of the Project on air quality.

20.4.3 Visual Quality

An inventory of landforms and topography, vegetation, human use elements, and scenic vistas and viewpoints will need to be collected to create the baseline dataset for viewshed effects analysis. This process includes the identification of key observation points by the groups of people who would be most affected by visual changes and a computer modelling process to map areas visible from specific viewpoints. The baseline data collection for the visual quality effects assessment will be informed by public engagement efforts.

20.5 SOCIO-ECONOMIC ENVIRONMENT

20.5.1 Overview

Public participation is a required element of the environmental assessment process and the regulations governing the environmental assessment process also contain provisions for the mandatory consultation with potentially affected First Nations, stakeholders, organisations, and members of the public. Effective consultation will contribute to project and project components that meet the expectations of stakeholders and rightsholders and incorporate local perspectives and traditional knowledge into the project design and operation. Consultation processes must provide opportunities for identification and resolution of issues and must be structured to fit the needs of the First Nations, stakeholders, potentially affected parties, and organisations.

The Davidson project is near to existing communities and identifies the values of the region and community. Consultation on the previously proposed mine project was extensive with many individuals, special interest groups and organisations providing comments and documenting concerns. Updated stakeholder mapping will be conducted to ensure that engagement is scoped appropriately and that all interested parties are provided with project information and opportunities to provide comments.

20.5.2 Regional Area

The recently announced closure of the sawmill in Houston and the placement of the Huckleberry Mine into care and maintenance in 2016 have emphasised the considerable socio-economic connectedness of the Buckley Valley communities and the need to plan for resilience.

Smithers, Telkwa, Houston, and the Regional District of Bulkley-Nechako have joined a group of twenty-one regional and local governments of northwest British Columbia to form the Northwest BC Resource Benefits Alliance (RBA). The RBA has been provided funding by the provincial government to work with stakeholders, including project proponents, labour, First Nations, local business, and the social services sector, with an aim to creating a funding agreement with the province to ensure that the region benefits from the capital spending on large projects (*i.e.*, mining, transmission lines, pipelines, LNG plant). Economic activity in the region often falls outside of municipal boundaries or regional district service areas so it does not generate the local government revenue required to improve the physical and social infrastructure needed to fully realise the benefits. The RBA is working towards a resource/development benefits arrangement with the British Columbia provincial government to see more of the wealth created in the region, stay in the region to support economic development, and sustain local communities.

20.5.3 Regional District

The Project area is within the Regional District of Bulkley-Nechako (RDBN) and is partially within the Smithers-Telkwa Rural Official Community Plan (OCP). The RDBN provides local government services to rural residents and unincorporated communities within 77,000 km^2 the district encompasses. Within the OCP, the Project area is designated as “Rural Resource”. The Rural Resource designation is intended to preserve lands within the plan area for a variety of activities, inclusive of mineral and aggregate extraction. The OCP has recommendations for the provincial ministry responsible for Mineral Resources not to issue permits for extraction or processing until the applicant demonstrates mitigation measures to minimise or nullify the effects of the activity. The recommendations also include requesting the province consider not allowing work camps at new mines within reasonable driving distance of a community to promote local residency.

20.5.3.1 Smithers

The Town of Smithers is nearest to the Project area and is a regional hub. It is approximately halfway between Prince George (371 km) and Prince Rupert (346 km) along Highway 16. The City of Terrace, population 17,682 (StatsCanada, 2023), is the nearest city.

Via Rail Canada offers service from Smithers to towns along the route to Prince Rupert, but as the passenger service is predominately focused on tourism and runs only three times a week, it is not a regular commuting route. Transit BC operates the Bulkley Nechako Regional Transit System that provides transit three times a week between Smithers and Prince George, with stops in all communities along the way.

The Town of Smithers' core community service facilities include a 25-bed hospital, a regional airport, two elementary schools, one high school, a community college, outdoor parks, a recreation centre (pool, ice rink, fitness), a public library and town hall, and emergency service (police, ambulance, fire department). Amenities include a mix of stores, restaurants, and accommodations. As the largest town in Bulkley Valley, Smithers provides services to neighbouring communities, including Telkwa and Houston.

The population of Smithers has remained largely stable since 2001 at approximately 5,400 people. The median household income is $85,000, and the median age is 39.6 in 2022 (Statistics Canada, 2022). Nearly 8% of the household incomes reported in the last census were above $150,000. Housing prices almost doubled between 2011 and 2021; the average housing purchase in 2021 was $406,800 (Statistics Canada, 2022).

The sources of primary sector employment are agriculture, mining, and forestry, with high levels of participation both in the primary sector and supporting secondary industries, such as trades and transport. Retail, tourism, public administration, accommodation, education, and professional services are other industries with significant participation. The unemployment rate was 8.9% in 2016 and 6% in 2022 (Statistics Canada, 2022).

Tourism, particularly adventure tourism, has increased in the Bulkley Valley, with Smithers being the centrepiece. The official tourism website for Smithers is "Get Good Natured." Many adventure tour operators operate out of Smithers and there are several associations and societies with mandates to promote and advance outdoor pursuits, including Bulkley Valley Backpackers, Smithers Mountain Bike Association, Smithers Snowmobile Association, and Bulkley Backcountry Ski Society.

20.5.3.2 Telkwa

The Village of Telkwa is at the confluence of the Bulkley and Telkwa Rivers along Highway 16. It hosts a population of 1,474 residents (Statistics Canada, 2022), an 11% increase since 2016. It is a short commute to Smithers (15 km) or Houston (42 km). A separately elected Mayor and Council govern it and the village has a grocery store, restaurants, a pharmacy, a village library, and recreational infrastructure. The community has an elementary school but not a high school; older students must take a bus to attend school in Smithers.

20.5.3.3 Houston

The Town of Houston has a population of 3,052 (Statistics Canada, 2022), with an estimated 2,000 additional residents in the surrounding area to which the District of Houston provides core services (roads, health services, emergency services). It is 65 km from Smithers and is situated on the confluence of the Bulkley and Morice Rivers. Houston has both elementary and secondary school levels. It was historically a forestry town with a sawmill and a large transportation sector but now has growing mining and tourism sectors. The economic development strategy for the community has four main pillars. Business retention and expansion is one and the support of mining companies and activities is expressly noted as an action step under this pillar.

20.5.3.4 Land Use Planning

Bulkley Land and Resource Management Plan (BLRMP) was primarily developed around the concepts of bio-diversity, sustainability, and several other values, including sub-surface resources. The BLRMP covers 762,0734 ha of Crown land in the Skeena region of northwestern British Columbia and includes the communities of Smithers, Telkwa, Moricetown, and Fort Babine. Most objectives within the plan are policy directions except for limited resource management zones with legal standing directions relating to specific values. The Project is within Glacier Gulch Resource Management zone, Unit -10-1, with objectives for visual quality, rare eco-systems, water quality for domestic consumption, and fish. It is designated as Special Management 2 (SM2) zone. It is a zone where industrial activities will need to be carried out sensitively to minimise impacts on identified values such as wildlife habitat, visual quality, recreation, or sensitive soils. Within the SM2 zone, legally designated areas subject to visual quality objectives overlap the Davidson Project Mineral Resource area and consistency with the objectives must be considered for any proposed development within these areas. Within the BLRMP, Glacier Gulch is specifically recognised as a popular local recreation area with mineral potential, explicitly noting the presence of a molybdenum deposit.

The proposed portal, waste storage, and tailings areas are within the Copper River Resource Management zone (sub-unit 12-2) of the BLRMP, which has objectives for water quality for fish habitat, visual quality preservation within the Copper River corridor and recreational focus points, and preservation of important riparian eco-systems.

A designated Ungulate Winter Range (u-6-007 Unit 8) established in 2019 by the British Columbia government overlaps the Project area entirely. The general wildlife measures within the corresponding order pertain to minor tenures with an exemption process for authorisations for industrial activity, including accessing mineral rights.

A large area designated as an important fossil area overlaps the entire Bulkley Valley and the Davidson Project area. This designation means that a Fossil Impact Assessment may be required as part of the assessment process.

The overarching regional land and resource planning objectives include resource development, and it is stated within the plan that the environmental assessment process addresses resource management objectives within the plans.

20.5.3.5 Land and Resource Use

Land tenures in the study area include mineral claims and leases, commercial recreation and trapping licenses, and a community forest agreement. The Hudson Bay Mountain Resort, an alpine ski hill development, is located south of the Project area. Well-established hiking routes, mountain biking, and cross-country skiing trails and snowmobile areas comprise non-tenured recreational uses. There are three Forest Recreation Sites and numerous Forest Recreation Trails near the Project.

The Community Forest Agreement area overlaps the entire Davidson Project mineral tenure area. Community Forest Agreements are area-based forest licenses managed by local governments, community groups, or First Nations. The Community Forest Agreement overlapping the Project area is a licence held jointly by the Town of Smithers and the Village of Telkwa, in partnership with the Office of the Wet'suwet'en in a collaboration called the Wetzin'kwa Community Forest Corporation. It is governed by a seven-person volunteer board. The licensee must regularly supply a 5-year landscape-level forest stewardship plan for approval by the government. Within the plan, they must demonstrate alignment with provincial government objectives for the protection of values.

None of these tenures are exclusive use nor do they preclude the development of Mineral Resources.

There is one titled private lot (0.03 km^2) on the northeast shore of Aldrich Lake near the contemplated tailings. The owner will be contacted and engaged as part of stakeholder engagement activities. Additionally, directly west of the titled private lot is an area reserved by the provincial government for environmental, recreational, or conservation purposes. These are exclusive use tenures and while not areas required for Project development, they are proximal to them and will require individual engagement to ensure effects can be identified and mitigated.

20.5.3.6 Archaeology

Baseline archeological evaluations and studies have been conducted on the Project area as part of the environmental assessment of the previous project proposal. Two recorded archaeological sites near Hudson Bay Mountain were cataloged in provincial databases and 15 additional sites were identified in field surveys in 2006. An archaeology impact assessment will be required in the Project development area and the supporting infrastructure corridor footprint as archaeological baseline data to ensure protection of cultural heritage and inform environmental assessment processes.

20.5.4 First Nations

Consultation with Indigenous rightsholders is an integral part of the project development, design, and environmental assessment process, informing the identification and mitigation of potential social, cultural, economic, and environmental impacts. The consultation process for major mine development must address long-standing concerns with stewardship of the land and the cumulative ecological and social impacts within the territory. Traditional ecological knowledge is expected to be utilised alongside Western science in the review processes and the development of mitigation strategies.

Most projects proceed through refined project evaluation and development under an established and formal agreement with potentially affected First Nations. These agreements, often called Participation Agreements, are negotiated between the project proponent and the identified leadership and go beyond basic government-mandated processes toward a true partnership model. They provide a framework for proactive engagement and ensure early and continuous collaboration between the proponent and the First Nations in all planning stages. They provide economic capacity for the First Nation to participate in engagement, project development, and review with customised terms and benefits that reflect the specific needs values and concerns of the Nation involved. These agreements usually comprise clauses to ensure there are benefits from the project assessment work effort and include terms related to employment, training, contracting, and environmental and cultural protection during the project evaluation stage.

Once the project advances to environmental assessment, negotiations for the development of an Impact Benefit Agreement (IBA) will commence, usually starting with the development of a term sheet. IBAs are increasingly common agreements between a resource development proponent and an impacted community, most often an Indigenous community. The key clauses include commitments to contribute financially, ensure economic participation in employment, training, and contracting opportunities, and increase environmental safeguards that often exceed regulatory environmental requirements to protect areas of cultural or ecological importance.

The process for sharing tax revenue from new mines and significant mine expansions with First Nations is well established in British Columbia. Currently, there are 17 active Economic and Community Development agreements addressing a percentage share of provincial mineral tax revenue. These agreements are negotiated on a case-by-case basis between the Government of British Columbia and eligible First Nations proximate to the project.

20.5.4.1 Office of the Wet'suwet'en

The Project lies within Wet'suwet'en territory, which includes much of the Bulkley Valley, including Moricetown, Smithers, Telkwa, Houston, and Burns Lake. The Wet'suwet'en community structure is divided into five clans:

1. Gilseyhu (Big Frog),
2. Laksilyu (Small Frog),
3. Gitdumden (Wolf/Bear),
4. Laksamshu (Fireweed), and
5. Tsayu (Beaver clan)

which are further sub-divided into 34 houses (Woos, et al., 2006).

Each house has titles and a territory associated with it. The Project area and potential zone of influence lie within both the territories of the Cas'Yex (Grizzly House) of the Gitumden Clan and the Kwen Beegh Yex (House Beside the Fire) of the Laksilyu (Small Frog Clan). The Wet'suwet'en territory is called the Yintah and it is necessary to receive permission from the appropriate Nations' representatives to conduct any work in their territory.

20.5.4.2 Witset First Nation

Witset First Nation, formerly Moricetown and originally "Kyah Wiget", is a First Nations band of Wet'suwet'en peoples operating under the Indian Act and governed by a Chief and Councillors elected on two-year cycles. Witset First Nation has seven reserves that comprise settlement areas and the Witset (Moricetown) community all of which are within the larger Wet'suwet'en territory.

Witset First Nation owns 100% of the issued shares of Kyah Development Corporation (KDC). KDC acts as the General Partner for the Moricetown Band Development Limited Partnership (MBDLP) that owns several assets and businesses and additionally holds agreements with the British Columbia Government relating to revenue sharing of logging revenues.

Additional First Nations will be identified within the zone of influence as supporting the Project infrastructure as it is refined and will include those First Nations whose rights are potentially affected by powerline right-of-way, road corridor development, routes used for shipping materials and products, and social-economic or environmental influences. The potential impacts of the Project will need to be communicated, understood, evaluated, then either mitigated or accommodated. Additionally, cumulative effects on rights will need to be addressed, meaning not just the Project but the combined impacts of past, present, and reasonably foreseeable future human activities on Indigenous rights.

20.5.5 Rightsholder and Stakeholder Engagement

Formal engagement plans are required for Project development and for the environmental assessment process. A formal engagement plan must be drafted and submitted for approval in the early engagement phase of the Project assessment and includes identification of potentially affected Indigenous Nations and communities, methods of engagement, mechanisms for gathering and considering feedback, communication plans for informing groups of opportunities to participate, and information on how the engagement results will refine the Project. The engagement plan will be evaluated and adjusted periodically and adapted as needed to ensure effectiveness and include additional rightsholders or stakeholders if they are identified.

The identification of public stakeholders will be conducted in a stakeholder mapping process. Engagement with the public and area stakeholder groups is expected to include land tenure holders, businesses, guide outfitters, recreational users, and special interest groups. This engagement will coincide with baseline collection activities and communications will include data collection for socio-economic baseline studies. Information will be provided regularly during the Project's progression and opportunities for feedback will be provided.

Engagement with federal, provincial, regional, and municipal government and government-funded agencies will be required and included in the engagement plans. The government engagement will provide administrative officials with knowledge about the Project and will ensure communication regarding expectations relating to Project development activities are jointly understood. Items for discussion with the government will include land and resource management, protected areas, environmental and social baseline studies, and effects assessment criteria.

20.5.6 Existing Project Site Environmental Factors

20.5.6.1 Davidson Project – Adit Area

The mineral tenure area of the Davidson Project includes historical underground mine workings of approximately 2,100 m, an adit, an adit access road, and waste rock placed proximal to the adit. In 2013, a Draft Care and Maintenance Plan and Closure Update was submitted by Thompson Creek Metals to the EMLCI. However, mine closure plans are not publicly available in British Columbia, and the content of the closure plan and the final status of the rehabilitation measures in this area is currently unknown. The water quality baseline study design will encompass the adit area and associated drainage to define existing conditions and inform any activities required in this previously disturbed area.

20.5.6.2 Duthie Mine: Henderson/Sloan Creek

The Duthie Mine, a past-producing silver, lead, and zinc underground mine is located on the west slope of Hudson Bay Mountain, within a sub-watershed draining to Aldrich Lake. Duthie Mine was mined primarily in the 1920 and 1950s. Rehabilitation measures were initiated in 1993 and the work effort mainly comprised aggregating 2,6000 m^3 of tailings from an estimated area of 40,000 square metres (m^2), away from stream flow and upslope from the areas of maximum groundwater discharge. Perimeter diversion ditches were then created to divert groundwater, which would otherwise become contaminated away from the pile. There are continuing pre-contact and post-contact water monitoring sites associated with the Duthie Mine drainage area registered within the British Columbia Water Resources Atlas.

The baseline water quality study design will include the sub-watershed draining to Aldrich Lake to define existing conditions and capture any water quality discrepancies created by the Duthie Mine drainage. The finalised locations of surface infrastructure for the Project will determine whether the Duthie Mine's potential contamination contributions to the watershed water will need to be considered in water quality modelling or the cumulative effects assessment.

20.5.7 Water Management, Waste Management and Monitoring

20.5.7.1 Overall Water Management Strategy

The Project will need refined water modelling and operational management strategies for service water in the underground mine and mill, mine infiltration, paste backfill plant, dry-stacked tailings, waste rock areas, and any infiltration within the access ramp. Underground voids are expected to supply the water for the mine, mill, and paste backfill recycling.

20.5.7.2 Key Water Management Techniques

Low permeability paste backfilling will reduce water flow through mine workings. Collection systems and mine plan sequencing will factor in backfilling rates to optimise the operational water balance. Excess contact water from underground, unsuitable for the mill or paste backfill, will be treated at the water treatment facility before discharge.

20.5.7.3 Tailings and De-Watering

Tailings alternatives will be evaluated as part of engagement activities and engineering and environmental studies. Dry stack deposition will be specifically considered during studies due to the minimisation of environmental risks and long-term liability, increased social acceptance, and maximisation of water recycling for processing.

Excess mill tailings, not used for paste backfill, will be pumped to a dry-stack tailings facility. Water removed during de-watering of tailings will be recycled or treated at the water treatment facility before discharge. The dry stack tailings facility surface will be designed with diversion ditches to prevent infiltration, a seepage collection system to prevent contact with groundwater or nearby surface water bodies and will be progressively reclaimed to limit precipitation infiltration. Collected seepage water may be recycled to support the mine operations.

20.5.7.4 Water Treatment Facility

The water treatment facility will be required for the treatment of water encountered during the development stage of the Project. Refined studies and modelling of constituents of concern will inform the final design of the treatment facility, but at a minimum, it is expected that it will be required to treat total suspended solids, molybdenum, and ammonia. Developing site-specific discharge criteria is part of the Environmental Management Act (EMA) Permit Process.

20.5.7.5 Waste Rock

Development rock will be temporarily stored on the surface, and once appropriate stopes have been mined out, some or all, of this rock will be backhauled underground. To limit contact water, waste rock storage areas will be designed with diversion ditches and seepage collection systems. Refined water mine planning will inform the need for additional waste rock management and the final design of the waste rock storage facility.

20.5.7.6 Acid Rock Drainage/Metal Leaching (ARD/ML)

Previous work has been undertaken to assess ARD potential. The assessment and prediction of ARD/ML concluded that significant portions of the rock should release ARD. Yet, to date, no ARD has developed in the 58 years since the existing waste rock pile and adit were created.

It is recognised that there are difficulties with applying standard ARD/ML modelling to molybdenum mines. Molybdenum ores often have a combination of minerals that both generate and neutralise acidity when exposed to water. Additionally, the chemical reaction involved in ARD/ML release from molybdenum ores can be very slow, masking the results in short-term laboratory tests. Given that the previous ARD/ML studies provided predictions that do not align with long-term site conditions, ARD/ML studies will be reinitiated for the Project and include static and kinetic analysis. Static testing will include geo-chemical characterisation, geo-chemical analysis, trace element content, and mineralogy properties. Kinetic testing will include trickle leach cells, tailing humidity cells, composite cells, and field bins. Given the outcomes of the previous work, particular attention will be paid to designing the studies to investigate the attenuation and neutralisation potential of all lithologies that will be disturbed by mine activities.

Characterisation to understand the ARD and ML potential, ARD/ML, and the neutralisation potential of the Mineral Resource material and waste rock is required to inform project design and mitigations and meet environmental assessment and permitting requirements. ARD/ML predictions form the basis of modelling used for numerous aspects of environmental assessment, ecological risk evaluation, and closure planning. ARD/ML data on all rock types and created rock mixes (*i.e.*, tailings, paste backfill) will be collected concurrently with the geological and geo-chemical data collection necessary to refine the Project's economic evaluation.

20.5.8 Waste Management, Monitoring, and Water Management

20.5.8.1 Water Management

Note: In this case, the processing plant is underground, most of the tailings will be used as backfill, only desulphidised tailings will be stored on the surface and any water released will be treated to meet Provincial Water Quality Objectives before release.

20.5.9 Licenses, Permits, and Approvals

20.5.9.1 Environmental Assessment

The Davidson Project meets designated project thresholds for assessment under both the British Columbia Environmental Assessment Act (BCEAA) and the federal Impact Assessment Act (IAA). The BCEAA, IAA, and accompanying regulations establish the framework for delivering environmental assessments; however, the scope, procedures, and methods of each assessment are specific to the circumstances of the proposed Project. Each environmental assessment is focused on the issues relevant to the Project and whether the Project should proceed. Proposed mining projects are required to obtain an Environmental Assessment Certificate before the issuance of operational permits, such as a Mines Act Permit, Environmental Management Act Permit, Water License, or Explosives Storage and Use Permit.

When projects meet thresholds for both BCEAA and IAA EAs, substitution agreements or coordinated environmental assessments between the two levels of government can be utilised to streamline the process and when these are utilised, the BC EAO takes the lead in integrating the provincial and federal processes into a harmonised review. The federal elements under the Fisheries Act, Species at Risk Act, Navigation Protection Act, Migratory Birds Convention Act, and supporting regulation for each are incorporated into the requirements of the assessment and the relevant federal agencies provide guidance, expertise, and review of the assessment process.

Both the BCEAA and the federal IAA underwent considerable changes in 2018. The changes focused the assessment processes on early engagement with the public and Indigenous Nations, increased Indigenous involvement, and created clearer timelines for stages of the review process. A readiness decision on whether a project should proceed to an environmental assessment was added as a gatekeeping step. The readiness decision is made after an Initial Project Description and corresponding Engagement Plan has been approved and actioned and a Detailed Project Description and Summary of Engagement compiled. The EAO then seeks consensus with the participating Indigenous Nations and a decision option is selected. Options for the readiness decision include requiring a revised Detailed Project Description, proceeding to environmental assessment, recommending the minister exempt the project from environmental assessment, or recommending the minister terminate the project from the process.

20.5.9.2 Concurrent Approval Regulation

The Concurrent Approval Regulation outlines a process that allows a proponent to apply for concurrent review of other provincial approvals (*e.g.*, licences and permits) for a proposed project that is undergoing an environmental assessment. This allows for the timely issuance of other required approvals if an environmental assessment certificate is granted. Where EAO allows for the concurrent review of permit applications, authorisations are generally made within 60 days of the issuance of an environmental assessment certificate. This approach requires detailed engineering and complex modelling in advance of the certainty of receipt of the environmental assessment certificate and the follow-up. There are risks and benefits to pursuing concurrent approvals and it is best used in situations where the project scope is well-defined and unlikely to change significantly. Not all applications for concurrent review are approved as it depends on the project’s complexity, potential impacts, and the specific approvals required. The concurrent approval regulation is a provincial regulation and the streamlining benefits may not be realised when a federal environmental assessment is also required.

20.5.9.3 Federal Licenses and Approvals

Successful completion of a substituted or coordinated environmental assessment does not automatically grant all federal permits. Application for specific permits need to be submitted separately. The assessment elements can be addressed concurrently with an environmental assessment but applications for Fisheries Authorisations, Explosives Act Licenses, Transport of Dangerous Goods permits, and similar need to be compiled and submitted for approvals.

20.5.9.4 Clean BC Act

Large projects undergoing environmental assessment may need to demonstrate how they align with CleanBC targets and sector-specific goals to contribute to the province's emissions reduction goals. Early consideration of designing to reduce emissions during project planning will improve the chances of approval and competitiveness. CleanBC promotes fuel switching from diesel to electricity for mine vehicles, equipment, and processes where feasible.

20.5.10 British Columbia Utilities Commission

British Columbia Utilities Commission (BCUC) reviews and approvals are required to connect directly to the main provincial power grid. There are specific pre-application consultations, detailed application processes, and BCUC reviews that cumulate in a decision document issued by a review panel that approves, denies, or approves the project with specific conditions. The BCUC approvals process focuses on public utility services, economic feasibility, and the technical aspects of power delivery whereas the environmental assessment addresses environmental and broader socio-economic impact assessments. For major mine projects, the environmental assessment usually happens first and the EAO and the BCUC may coordinate consultation processes so that the BCUC can consider the findings of the environmental assessment process during their review and avoid re-examining the same issues.

20.5.11 Mine Closure Requirements and Financial Assurance

The BC Ministry of Energy, Mines and Low Carbon Innovation (EMLCI) released an interim Major Mines Reclamation Security Policy in 2022 to provide direction on financial assurance calculation while the Ministry of Environment and Climate Change Strategy (ENV) advanced a mandate to create the necessary legislative changes to ensure that the cost of environmental clean-up is entirely the responsibility of the owners of large industrial projects. This mandate, termed the Public Interest Bonding Strategy, is an interagency, multi-stakeholder effort expected to take 2-3 years to complete.

The interim Major Mines Security Policy states that new and existing mines (whether in operation, in care and maintenance, or closed) having less than 5 years of remaining Mineral Reserves are required to post security equal to the entire reclamation liability. A partial exploration incentive is available to mines that have a minimum of 5 years of remaining Reserves. The policy seeks to fully bond mines moving forward and reduce the differential between reclamation liabilities and reclamation securities for existing mines.

Reclamation liability cost estimates are calculated based on an approved reclamation and closure plan. Closure plans must be developed and updated throughout the LOM, as part of the initial permit process, every 5 years after that, in support of permit amendments and 12 months before the planned date of mine closure. The reclamation security required is based on the NPV of the peak estimated liability during the 5 years. Liability estimates are calculated based on 100 years and discounted according to the liability held. The required content of a liability cost estimate is comprehensive. It includes reclamation costs for land forming and re-vegetation, engineering and administration, equipment and structure removal, water treatment capital and operating costs, maintenance, monitoring, labour rates based on third-party contractors, and a default contingency of 15%. Special approvals are required to allow for the use of salvage value or the value of any other assets or revenue stream in offsetting the reclamation liability amounts.

21.0 CAPITAL AND OPERATING COSTS

21.1 CAPITAL EXPENDITURES

The capital expenditures estimates are based on budget pricing from suppliers for critical components, consultants, contractors, and a review of other Canadian projects. Smaller equipment and facilities component costs were factored based on industry norms for the type of facility being constructed and, where possible, adjusted to reflect local conditions. Capital expenditures estimates are within ±40%.

Labour rates are based on contractor costs in the region and country, for similar types of work. Where costs were either not available or irrelevant, costs from other similar projects in Canada were used. The rates used include all cost and profit components payable to contractors.

All expenditure estimates are in 2024 constant Canadian Dollars.

21.1.1 Basis for Estimates

The capital expenditures estimate includes the following:

- Mine development, mining equipment mobile (leased) and fixed, and associated consumables and maintenance parts for development and infrastructure;
- Processing plant equipment and construction;
- Project infrastructure equipment and materials;
- Construction materials;
- Labour;
- Temporary buildings and services;
- Construction support services;
- Spare parts;
- Initial fills (inventory);
- Freight;
- Vendor supervision;
- Owner's cost;
- Engineering, Procurement, and Construction Management;
- Commissioning and start up; and
- Contingency.

21.1.2 Direct Costs

Direct costs are all costs associated with permanent facilities. This includes mine development openings, equipment and material costs, as well as underground mine, processing plant, and infrastructure construction, and installation costs.

Mine infrastructure costs for facilities, such as maintenance shops, mine de-watering, refuge stations, etc., were developed based on the conceptual plans and general arrangements presented earlier. Wherever possible, equipment and material budget prices and contractor budget installation costs were used.

Other major equipment expenditure estimates are based on quotes obtained from suppliers and installation costs estimated as part of this study.

During the pre-production and sustaining development periods, all materials and equipment pricing are based on quotes obtained from Canadian or United States suppliers.

All major equipment expenditures include freight only. Applicable taxes and duties have not been included in the capital expenditure estimates.

Commodity pricing for earthwork, concrete, steel, architectural, and piping are based on Canadian costs and suppliers. Labour rates and equipment usage rates used throughout the estimate are based on mining contractor and other sources information.

Pricing used are expected contractor rates for rock excavation and transport during the pre-production stage.

Labour rates generally reflect Canadian contractor rates. The mine labour costs are based on four types of estimates:

- Contractor budget prices for undertaking the tasks associated with constructing a specific installation.
- Average industry rates a contractor will be expected to charge for performing specific tasks.
- Lateral and raise development costs based on expected productivity and labour, materials, and equipment costs for such an underground development program.
- All labour costs include government mandated contributions and the costs for Company provided benefits.

21.1.3 Indirect Costs Estimate

The indirect costs include all costs associated with temporary construction facilities and services, construction support, freight, vendor representatives, spare parts, initial fills and inventory, Owner's costs, Engineering, Procurement, and Construction Management (EPCM), commissioning, and start-up assistance.

The costs for construction facilities include all temporary facilities, services and operation, site office operations, security buildings and services, construction warehousing and material management, construction power and utilities, site transportation, medical facilities and services, garbage collection and disposal, and surveying.

Spare Parts – The cost for spare parts is factored based on equipment costs where the vendors did not provide cost for spares needed for the first year of operation.

Initial Fills (Inventory) – The estimated cost for initial fills is based on 3 months of operating requirements.

Freight – The freight costs are based on delivery to the site from point of manufacture and based on supplier estimates or average Canadian project costs.

Taxes and Duties – Taxes and duties have been excluded.

Engineering, Procurement, and Construction Management (EPCM) – EPCM has been calculated only on specific construction activities, such as the processing plant. All site, underground installations and underground processing plant rooms development will be supervised by the Moon River Project Management team.

Capital Cost Qualifications and Exclusions – All surface and underground processing plant construction work will be executed by contractors.

Capital expenditures estimates exclude:

- Sunk costs;
- Taxes and duties;
- Deferred capital;
- Financing and interest during construction;
- Additional exploration drilling;
- Escalation;
- Corporate withholding taxes;
- Legal costs;
- Metallurgical testing costs;
- Condemnation testing; and
- Salvage revenues.

All expenditure estimates are in 2024 constant Canadian Dollars.

21.1.4 Underground Mining

Underground capital cost estimates are based on budget pricing from suppliers, consultants, and contractors provided with general specifications to ensure equipment or service provided is specific to the Project and includes all costs specific to the Project and application. Some small equipment and facilities component costs were factored based on norms for the type of facility being constructed and adjusted to reflect local conditions.

Construction and installation labour rates are based on Owner/Operator costs for the types of work envisaged for the Project.

The underground equipment fleet will be leased by Moon River.

The mine pre-production capital expenditures are estimated to total $277 million including a 25% contingency. The breakdown of the pre-production mine capital expenditures is presented in Table 21.1, below.

TABLE 21.1
MINE DEPARTMENT CAPITAL COSTS
($000)

Component	Year -3	Year -2	Year -1	Year 1
Exploration	$1,000	$1,000	$1,000	$-
Mine Development	$34,377	$52,739	$50,440	$24,124
Equipment Leasing	$9,441	$8,952	$8,462	$-
Underground Infrastructure	$-	$4,886	$1,815	$23,375
Contingency at 25%	$11,205	$16,894	$15,429	$11,875
Total	**$277,015**			

The initial capital expenditure for the underground mine will include collaring of the twin access drifts portals and development of the access drifts 7 km to the top elevation of the potentially economic mineralisation. Simultaneously, the internal ramp system will be driven from the existing east portal to the top and bottom of the potentially economic mineralisation to allow excavation and construction of the underground processing plant, underground crusher systems, and internal winze and vertical lift conveyor system. Early production levels will be established on the 1060, 1105, 1150, 1195 and 1240 Levels.

Mine development will also include development of the initial ventilation system, installation of mine fans and heaters, installation of a pumping system, and reticulation systems for electricity, communications network, compressed air, process water, and mine drainage water.

The pre-production period is expected to be 3.5 years and also includes construction of the underground processing plant, crusher systems, vertical lift conveyor system and related surface infrastructure.

The mine development capital development expenditures estimates are shown in Table 21.2, below.

TABLE 21.2
MINE CAPITAL DEVELOPMENT

Area	Year -3	Year -2	Year -1	Year 1	Cost/m	Total ($M)
Twin Ramp System	4410	4410	4968	1194	$4,608	$69.0
Slash out Existing Exploration Drive	1700				$4,608	$7.8
Internal Ramp System	960	3,148	502		$4,608	$21.2
Level development and Volume Excavation	150	2732	4820	4019	$4,608	$54.0
Raising	410	1310	1120	700	$2,699	$9.6
Contingency at 25%						$40.4
Total						**$202.1**

Pre-production leasing costs for underground equipment is estimated to be $26.9 million.

Underground mine infrastructure capital expenditure estimates for the Project are shown in Table 21.3, below.

TABLE 21.3
UNDERGROUND MINE INFRASTRUCTURE CAPITAL EXPENDITURE ESTIMATES

Component	Quantity	Units	Unit Cost	Total Cost	Year -3	Year -2	Year -1	Year 1
SURFACE INFRASTRUCTURE								
Mine Portal								
Surface Intake Vent Fan Installation	2	L.S.	$350,000	**$700,000**	$700,000			
Mine Air Heaters	2	L.S.	$225,000	**$450,000**	$450,000			
Explosives Magazines (Supplier Provided)	2	L.S.	$25,000	**$50,000**	$50,000			
Compressors	3	L.S.	$267,932	**$803,795**	$535,864			$267,932
Cement Silos for backfill	2	L.S.	$1,000,000	**$2,000,000**				$2,000,000
Mine Rescue and Fire Fighting Equipment	2	L.S.	$407,241	**$814,483**	$407,241			
Total Surface Infrastructure				**$5,518,278**	**$2,843,105**			**$2,267,932**
Mobilise, Setup, and Demobilise	2	L.S.	$100,000	**$200,000**	$100,000			$100,000
UNDERGROUND SUPPORT SERVICES FACILITIES								
Pocket Lift Conveyor System	1	L.S.	$44,000,000	**$44,000,000**			$22,000,000	$22,000,000
Exhaust Ventilation Fans Installations	1	L.S.	$350,000	**$350,000**				$350,000
Mechanicad Ducting	1	L.S.	$1,000,000	**$1,000,000**	$340,000	$340,000	$250,000	$70,000
Maintenance Shop	1	L.S.	$1,500,000	**$1,500,000**				$1,500,000
Fueling Station (Marcotte)	4	L.S.	$90,000	**$360,000**	$360,000			
Explosives and Detonators Magazines Construction and Equipping	4	L.S.	$86,000	**$344,000**		$172,000		
Main Storage Area Construction and Equipping	2	L.S.	$36,000	**$72,000**		$36,000		$36,000
Main De-watering Sump Construction and Equipping	1	L.S.	$500,000	**$500,000**				$500,000
Discharge Line	1	L.S.	$146,964	**$146,964**				$146,964
Refuge Station Construction and Equipping	7	L.S.	$140,000	**$980,000**	$140,000	$280,000	$280,000	$280,000
Portable Toilets	7	L.S.	$5,000	**$35,000**	$5,000	$10,000	$10,000	$10,000
Total Underground Support Services Facilities				**$49,287,964**	**$845,000**	**$838,000**	**$22,540,000**	**$24,892,964**
MINE SERVICES								
Portable Substations	11	each	$211,810	**$2,329,910**	$211,810	$635,430	$635,430	
Mine Communication	1	L.S.	$1,000,000	**$1,000,000**			$200,000	$800,000
Computers, Peripherals & Software	2	L.S.	$110,000	**$220,000**	$40,000	$70,000		
Engineering & Geology Equipment	2	L.S.	$44,000	**$88,000**	$44,000			$44,000
Paste Backfill Distribution System	1	L.S.	$1,500,000	**$1,500,000**				$500,000
Underground Booster Fans and Auxiliary Ventilation	2	L.S.	$ 529,500	**$794,250**	$264,750	$264,750		
Mine Lamps	125	each	$200	**$25,000**	$8,000	$7,000		$ 10,000
Total Mine Services				**$5,957,160**	**$568,560**	**$977,180**	**$835,430**	**$1,354,000**
Total Mine Infrastructure Expenditures			**$53,974,647**	**$60,963,402**	**$4,356,665**	**$1,815,180**	**$23,375,430**	**$28,614,895**

21.1.5 Processing Plant

Total pre-production capital expenditures for the processing plant and tailings management facility (TMF) are estimated to be $205.3 million including a 25% contingency. The TMF costs were supplied by an outside engineering consultant and have been spread over the LOM. Table 21.4, below, shows the estimated costs for the construction of the mill and TMF. The mill was sized to accommodate a daily production rate of 7,000 tonnes per day from the underground mine.

TABLE 21.4
PROCESSING PLANT AND TMF CAPITAL EXPENDITURE ESTIMATES

Equipment	Cost
Crusher	
Pan Feeder	$500,000
Jaw Crusher	$800,000
Conveyor – Jaw Discharge	$1,500,000
Conveyor – Screen Fine to FOR	$750,000
Conveyor – Cone Discharge	$750,000
Conveyor – Screen Feed	$750,000
Cone Crusher	$2,000,000
Cone Crusher Feeder	$500,000
Processing Plant	
Ball Mill Feeders – 4	$3,000,000
Ball Mills – 2	$20,000,000
Pumps – Allocation	$5,000,000
Cyclones	$1,000,000
Gravity Concentrators	$2,000,000
Rougher Scavenger Flotation Cells – 6	$6,000,000
Cleaner Flotation Columns – 2	$3,000,000
Concentrate Pressure Filter	$500,000
Tails/Paste Pressure Filters – 4	$12,000,000
Paste Pump	$2,000,000
Equipment Cost	$62,050,000
Installation Cost	$93,075,000
Initial Tailings Facility	$9,149,739
Subtotal	**$164,274,739**
Contingency at 25%	$41,068,685
Total Cost	**$205,343,424**

21.1.6 Surface Infrastructure

Total pre-production capital expenditures for the infrastructure and surface department are estimated to be approximately $43.9 million, including a 25% contingency. The breakdown of expenditures is presented in Table 21.5, below. Major expenditure components are for access road upgrading, power supply and distribution, site preparation, waste rock and ore storage pads, shops and offices, and water supply and treatment.

TABLE 21.5
SURFACE INFRASTRUCTURE

Component	Year -3	Year -2	Year -1	Cost
Main Access Road + Property Access Roads + East Access Road	$3,500,000			$3,500,000
Topsoil Stripping and Grubbing	$250,000	$125,000	$125,000	$500,000
Main Operations Pads Prep and Buildings Earthworks	$187,500		$62,500	$250,000
Transmission Line to Site (17 km at $250,000/km)	$4,250,000			$4,250,000
Power Distribution Onsite	$3,500,000			$3,500,000
Drop Down Transformer at Grid Connection	$5,000,000			$5,000,000
Potable Water System	$150,000			$150,000
Mobilise/Demobilise and Earthworks Management	$100,000			$100,000
Sewage System	$300,000			$300,000
Cold Storage			$200,000	$200,000
Fuel Storage	$100,000			$100,000
Propane	$50,000			$50,000
Water Treatment Plant (Aecom cost + $95K estimate for electric and piping)			$10,000,000	$10,000,000
Quarry/Waste Dump	$1,000,000		$1,000,000	$2,000,000
Dry/Warehouse/Office/Shop Complex	$2,000,000			$2,000,000
Effluent Pond		$50,000		$50,000
Process Water Line and Pumphouse	$400,000			$400,000
Subtotal	**$20,787,500**	**$175,000**	**$11,387,500**	**$32,350,000**
EPCM at 8% Owner Management	$1,663,000	$14,000	$911,000	$2,588,000
First Fills		$75,000		$75,000
Spare Parts		$75,000		$75,000
Surface Equipment Lease	$1,185,894	$1,124,403	$1,062,912	$3,373,210
Subtotal	**$22,450,500**	**$339,000**	**$12,298,500**	**$35,088,000**
Contingency at 25%	$5,612,625	$84,750	$3,074,625	$8,772,000
Total Surface Mine Infrastructure	**$28,063,125**	**$423,750**	**$15,373,125**	**$43,860,000**

21.1.7 Project Indirects and Owner's Costs

Project Indirects and Owner's costs, including a 25% contingency, are estimated at $13.9 million over the year pre-production period. Owner's costs also include all equivalent General and Administration (G&A) costs, which would be incurred during the construction phase.

21.1.8 Total Capital Expenditures

The estimated Project pre-production capital expenditure, inclusive of contingencies and working capital, is approximately $575 million. The total expenditures include EPCM, contractor overheads, and a 25% contingency on all estimated expenditures. A summary of Project pre-production capital expenditures is presented in Table 21.6, below. A working capital allowance of $20.7 million is estimated to be required.

The capital estimates include the following conditions and exclusions:

- Qualified and experienced construction labour would be available at the time of execution of the Project;
- A water supply capable of supplying the required demand of the processing plant is assumed to be available;
- No extremes in weather have been anticipated during the construction phase; and
- No allowances have been included for construction-labour stand-down costs.

TABLE 21.6
TOTAL PROJECT CAPITAL EXPENDITURE ESTIMATES
($000)

Component	Year -3	Year -2	Year -1	Year 1
Exploration	$1,000	$1,000	$1,000	
Mine	$34,377	$52,739	$50,440	$24,124
Equipment Leasing	$9,441	$8,952	$8,462	
Processing Plant		$70,000	$50,125	$35,000
Underground Infrastructure		$4,886	$1,815	$23,375
Surface Infrastructure and Mobile Equipment	$23,636	$1,463	$13,361	
Tailings Management Facilities			$9,150	
Owner's Costs	$3,700	$3,700	$3,700	
Contingency	$18,039	$35,685	$34,513	$20,625
Working Capital				$20,679
Mine Closure			$10,000	
Total Capital Expenditures	**$90,193**	**$178,425**	**$182,567**	**$123,803**
Total	**$574,987**			

21.1.9 Working Capital

Working Capital has been estimated at $20.7 million based on 3 months of the estimated operating costs for the year.

21.1.10 Sustaining Capital

Sustaining capital is estimated at $78.6 million for the LOM and consists of continuing expansion and construction of mine facilities and equipment, equipment leasing and replacement, expansion of the TSF, and staged closure costs for the TSF.

21.1.11 Reclamation and Closure Costs

There will be very little site infrastructure to dismantle and remove as most of the infrastructure, including the processing plant, is located underground. Reclamation and closure costs have been estimated at $10 million to remove the existing site infrastructure and reclaim the affected area, seal the three portals, and maintain water monitoring from the TMF for a period of time.

21.2 OPERATING COST ESTIMATES

21.2.1 Basis for Estimates

Operating costs are based on Canadian and other country normal prices from suppliers and other similar type projects, for consumables and parts. The cost of power is based on online posted rates for the Province of British Columbia.

Critical operating cost components are based on the following costs:

- The diesel fuel price is assumed to be $1.75 per litre.
- The electrical power cost is assumed to be $0.061 per kWh.

Labour costs for the operating period are based on the manpower schedules presented for each department and the associated labour costs. Labour rates are based on contractor costs in the region and country, for similar types of work. Where costs were not available, costs from other similar projects were used. The rates used include all cost and profit components payable to contractors.

All costs are quoted in constant 2024 Canadian Dollars.

21.2.2 Mining

Individual costs for underground mining have been estimated for manpower, equipment operating, maintenance, and materials consumptions from first principles. The total underground mining cost is estimated to be $21.07 per tonne of potentially economic mineralisation, with the cost breakdown presented in Table 21.7, below.

TABLE 21.7 UNDERGROUND MINING COSTS	
Components	**Total Cost ($/t)**
Stope Development	$2.66
Cable Bolting	$0.43
Longhole Drilling Operating Costs	$0.50
Longhole Blasting	$1.81
Stope Mucking	$2.20
Longhole Drilling Manpower	$0.22
Total Stoping Cost, per tonne	**$7.82**
Services Equipment	$0.30
Heating Costs	$0.09
Electrical Power	$0.55
Backfill	$7.20
Crushing	$0.50
Powerlift conveyor hoist	$1.00
Services Manpower	$3.60
Total Mining Cost per Tonne	**$21.07**

Mines services and overheads costs include all other non-direct stoping costs for the operation. Mine services operating costs are associated with maintaining underground facilities and services (power, water supply, etc.), operating and maintaining ventilations fans, supplies for safety and training, including personal protective equipment and mine rescue, and operating and maintaining all support mobile equipment used in the mine.

The mining costs are based on costs from Canadian suppliers and underground contractors.

21.2.3 Processing Plant and Tailings Management

The operating costs for the processing plant and the TMF are detailed in Table 21.8, below.

TABLE 21.8 PROCESSING AND TAILINGS MANAGEMENT COSTS	
Component	**Cost**
Manpower	$2,979,750
Mill Reagents/Consumables	$20,000,000
Environmental	$750,000
Power	$4,551,214
Total Annual Mill OPEX	**$27,356,214**
Total Cost per Tonne Mined	**$10.94**
TMF Cost/Tonne Placed	$4.48
TMF Cost/Tonne Mined	$1.34
Total	**$12.29**

21.2.4 General and Administration (G&A) Costs

The estimates for G&A costs encompass all operating costs associated with operating the offices and providing materials and supplies for staff functions.

The total yearly G&A costs are estimated to be approximately $5.5 million (presented in Table 21.9, below), of which approximately $2.1 million is for salaries and benefits. Employee burdens account for approximately 35% of the total salary for each employee. Annualised site G&A costs are estimated at $2.20 per tonne of potentially economic mineralisation processed.

The mine management and administration roster and costs have been estimated in Table 21.9, below. A total of 19 people would be employed in this area, most of which would be staff positions. They would be responsible for the management, administration, personnel, accounting, purchasing needs, and distribution of material to the operation, site security, health and safety, and environmental issues. The total costs for G&A labour are $0.86 per tonne of potentially economic mineralisation processed.

Table 21.9 Estimated G&A Roster and Costs			
Position	**Complement**	**Annual Compensation ($)**	**Total Cost ($)**
General Manager	1	$200,000	$270,000
Comptroller	1	$95,000	$128,250
Accountant	2	$70,000	$189,000
Head of Health/Safety and Security	1	$90,000	$121,500
Environmental Manager	1	$150,000	$202,500
Environmental Technician	2	$70,000	$189,000
Office Clerk/Secretary	1	$70,000	$94,500
Purchasing Agent	1	$80,000	$108,000
Warehouseman	4	$70,000	$378,000
Warehouse Stock Taker	1	$60,000	$81,000
Medical Services (Contract)	1	$90,000	$90,000
Security Contract	3	$98,000	$294,000
Total Complement	**19**		**$2,145,750**

21.2.5 Concentrate Transport Charges

Transportation charges of $120 per tonne of concentrate have been included in the cash flow model.

21.2.6 Project Total Operating Costs

The estimated total average operating cost (excluding smelting and refining) is approximately $38.24 per tonne. Table 21.10 presents a summary table of LOM average operating costs for each department on a cost per tonne of potentially economic mineralisation basis.

Table 21.10 Mine Site Operating Costs	
Component	**Cost**
Diamond Drilling – Infill	$0.50
Underground Mining	$21.07
Processing	$10.94
Tailings Management Facility	$1.34
Mine Indirects	$1.29
Surface Department	$0.90
General & Administration	$2.20
Total Minesite Operating Cost	**$38.24**

21.2.7 Exclusions

For the purpose of this study, value added taxes and other taxes, along with import duty costs, have not been included. Exploration costs, including future infill and definition drilling and all costs associated with areas beyond the Property limits, have also not been included. In addition, salvage value of the infrastructure at the end of the Project life have not been included.

22.0 ECONOMIC ANALYSIS

The expected cash flow estimates are calculated using the forecast mine development and production plan (using diluted potentially economic Measured, Indicated, and Inferred Mineral Resources), operating costs, and capital expenditures incorporating expected long-term metal prices based on the 24-month trailing average pricing as of January 31, 2024 (Table 22.1, below).

**TABLE 22.1
COMMODITY PRICING AND EXCHANGE RATE**

Commodity	Price
MoS_2	$47.39
Exchange rate (US$/CA$)	$0.74

The cut-off determination for mining was based on a Break-Even NSR cut-off value. The Resource was broken into stoping blocks, with an in-situ dollar value calculated for each block. Dilution was included based on the surrounding rock for each stope with the appropriate grades. The break-even cut-off grade for the Resource was determined to be 0.11% MoS_2 utilising the commodity price used in the cash flow model.

A summary of the expected parameters used for the financial analysis is presented in Table 22.2, below.

**TABLE 22.2
CASH FLOW MODEL INPUT PARAMETERS**

Parameter	
Long-term Metal Price (US$)	$47.39 ($21.50/lb)
Exchange Rate	$1.35 CA$ per $1 US
Diluted Mineral Resource	49,125,000 tonnes
Dilution (at adjacent mineral grade)	5%
Average Head Grade to Mill	0.34%
Mill Recovery	92
Payability	97%
Pre-Production Capital	$575.0 million
Total Sustaining Capital	$78.6 million
Working Capital	$20.7 million
Reclamation and Closure	$10.0 million
Estimated Operating Costs ($/tonne)	$38.24
Life of Project	20 Years

The cash flow analysis has been conducted on the assumption of 100% equity investment and excludes any element or impact of financing arrangements. All exploration and acquisition costs incurred prior to the production decision are excluded from the cash flows.

Capital expenditures, as shown in the capital section, would be incurred over a 3.5-year period, which is reflected in the discounted cash flow calculations. The cash flows include sustaining capital and capital expenditures contingency of approximately 25%.

Net Revenue is based on payments for metals produced, less the costs for metal sales, shipping, and smelter and refinery charges.

The expected cash flow analysis used the metal prices indicated above. The discounted cash flow analysis has been based on 2024 Constant Canadian Dollar values.

The potentially mineable underground resource is estimated to be 49.1 Mt at a grade of 0.34% MoS_2 per tonne. This PEA relies on Measured, Indicated, and Inferred Mineral Resources.

It should be noted that the Inferred Mineral Resources are considered too speculative geologically to have economic considerations applied to them that would enable them to be categorised as Mineral Reserves. Metallurgical recoveries and capital and operating cost estimates are to a PEA level of accuracy. Therefore, there is no guarantee that the economic projections contained in this PEA would be realised.

22.1 TAXES

Federal corporate income and British Columbia provincial corporate income and mining taxes, including allowed deductions for tax purposes, were included in the cash flow model.

22.2 FINANCIAL RETURNS

The overall level of accuracy of this study is approximately ±40%.

The Project expected investment and returns, based on the expected cash flow parameters, are shown in Table 22.3, below. The payback on capital investment is approximately 3.3 years.

TABLE 22.3
EXPECTED PROJECT RETURNS

Pre-Tax		
NPV	5%	$1,523,623,681
	8%	$1,042,890,161
	10%	$814,606,913
IRR		32%
After-Tax		
NPV	5%	$930,632,326
	8%	$601,808,873
	10%	$447,396,466
IRR		24%

Results indicate that at the expected parameters and metals prices, the Project is viable.

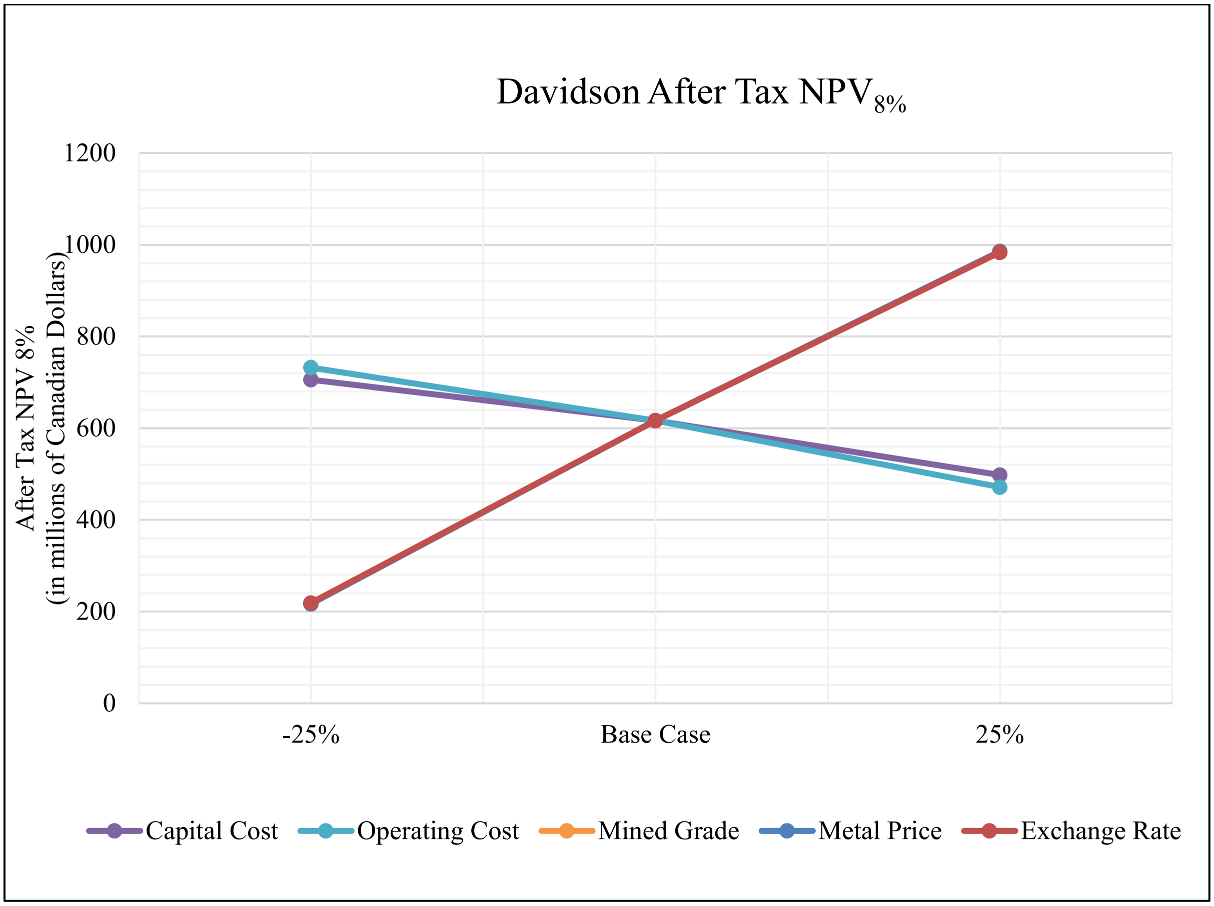
22.3 SENSITIVITY ANALYSIS

Sensitivity analyses were performed for capital expenditures, operating costs, mined grades, metal prices, and currency exchange rates using 5% to 25% positive and negative variations. The Project is most sensitive to changes in metals prices, resource grades, and exchange rates and reasonable acceptable to changes in the other variables. The results of the sensitivity analysis at are presented in Table 22.4 and Table 22.5, below.

TABLE 22.4 SENSITIVITY ANALYSIS NPV AT 8% DISCOUNT RATE			
	After Tax NPV 8%		
	-25%	Base Case	25%
Capital Cost	705.7	616.8	497.9
Operating Cost	732.1	616.8	471.5
Mined Grade	218.9	616.8	983.9
Metal Price	216.9	616.8	985.9
Exchange Rate	218.9	616.8	983.9

TABLE 22.5 SENSITIVITY ANALYSIS IRR			
	After Tax IRR		
	-25%	Base Case	25%
Capital Cost	31%	24%	19%
Operating Cost	26%	24%	21%
Mined Grade	15%	24%	31%
Metal Price	14%	24%	31%
Exchange Rate	15%	24%	31%

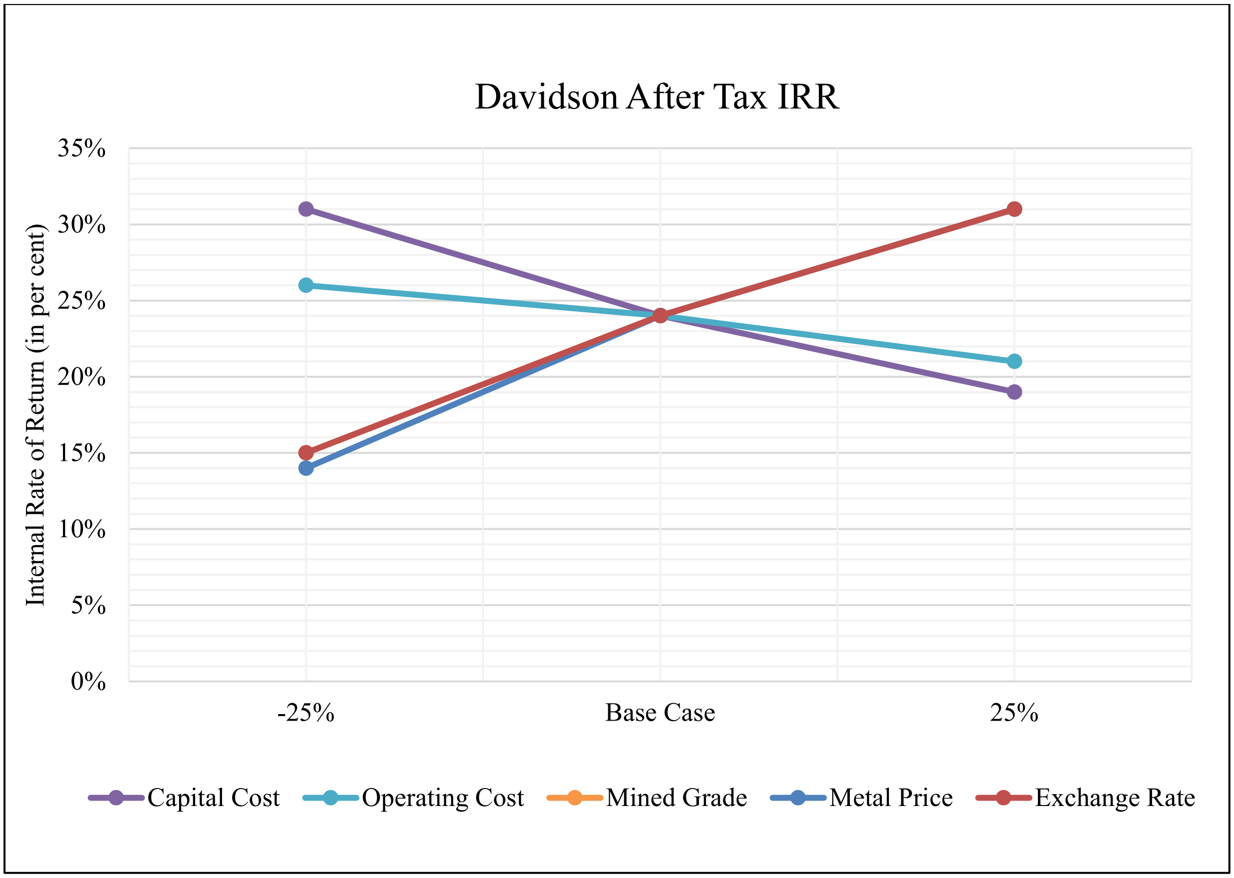
The NPV and IRR sensitivities to variations in key parameters are depicted graphically in Figure 22.1 and Figure 22.2, below. The IRR is most sensitive to variations in metal prices and mined grades and less sensitive to capital and operating costs.



Note: The lines for Grade, Metal Price and Exchange Rate are virtually the same and overlay each other.

Figure 22.1. NPV at 8% Discount Sensitivity Analysis

Source: AMPL, 2024



Note: The lines for Grade and Metal Price are virtually the same and overlay each other.

Figure 22.2. IRR Sensitivity Analysis

Source: AMPL, 2024

23.0 ADJACENT PROPERTIES

The authors do not have any specific information regarding neighbouring or adjacent molybdenum mineral properties other than the Endako molybdenum mine located 160 km to the southeast of the Davidson Property.

The Duthie Silver and Gold Mine, located 6 km to the southwest of the Davidson Property, is a precious and base metal vein type occurrence that has seen intermittent production between 1923 to 1988.

24.0 OTHER RELEVANT DATA AND INFORMATION

Numerous internal pre-feasibility and feasibility stage studies were completed by Climax over their property tenure.

24.1 ENVIRONMENTAL STUDIES – CLIMAX

The Map Production Division of the British Columbia Government Surveys and Mapping Branch completed a series of 1:25,000 maps for the Land Inspection Division in May 1974. These base maps, which cover all the claims and mining leases of the Davidson Property, consist of the following:

1. Soils, Forest Capability
2. Recreational Capability
3. Agricultural Capability
4. Climate Capacity of Agriculture
5. Ungulate Capability
6. Waterfowl Capability
7. Fisheries
8. Mineral Deposit Land Use
9. Present Land Use
10. Topography
11. Surface Land Status
12. Under Surface Land Status
13. Forest Cover
14. Water Resources.

These maps would prove most useful for future environmental studies.

24.2 INTERNAL REPORTS TO BLUE PEARL MINING/THOMPSON CREEK METALS

In May 2008, Hatch Ltd was the lead consultant in completing an internal report to Blue Pearl Mining. Contributors to this report include:

1. Snowden Mining Industry Consultants for geological setting, mineral and deposit type, historical exploration and drilling, and QA/QC.

2. Rescan Environmental Services for Environmental studies, such as water and air quality, meteorological, and noise studies.

3. McIntosh Engineering of Tempe, Arizona, USA for Mining.

4. Hatch Ltd for metallurgical testing, capital cost estimate, operating cost estimate and project economics.

The report outlines a planned underground mine designed to produce 730,000 tonnes of material per year or about 2,000 tonnes per day over a calendar year. The material would be mined by blast hole stoping with cemented backfill. Extraction of the Resource would be along a 2.9 km newly constructed adit at the 700 m elevation level to avoid visual impact on the northeastern side of Hudson Bay Mountain.

In August 2008, Rescan Environmental completed an application for an Environmental Assessment Certificate for Blue Pearl Mining.

In February 2013, Rescan Environmental Services Ltd of Vancouver, British Columbia completed an internal study titled “2012 Freshwater Baseline Report” prepared for Thompson Creek Metals Company Inc. It presents the results of the ongoing water quality monitoring program for the Davidson Project.

In April 2013, Rescan Environmental prepared a ‘draft’ copy of a “Care and Maintenance Plan and Closure Report Update” for Thompson Creek Metals Company Ltd.

No other relevant data or information about the Davidson Property is known.

25.0 INTERPRETATION AND CONCLUSIONS

This PEA examines the viability of mining the September 13, 2023 NI 43-101 Resource estimate using underground mining methods. The results from this PEA indicate the Davidson Project have the potential to generate positive economic returns.

The contemplated plan is to mine the higher-grade core of the mineralised zone. Using a cut-off grade above 0.25% MoS_2, there is a Measured and Indicated Resource of 43.98 Mt at 0.35% MoS_2 and an Inferred Resource of an additional 11.9 Mt at 0.30% MoS_2 available for mining. (refer to Table 1.3 and Table 1.4, above). This PEA has identified a diluted potentially mineable Resource of 49.1 Mt at 0.34% MoS_2 (refer to Appendix 1.0).

The engineering design extracts the potential resources at 2.5 Mt per annum and produces $5.84 billion in gross revenue during the 20-year LOM.

Based on the study results, the conclusions of AMPL are as follows:

1. The Project provides positive returns based on the parameters and metal prices used in this study and should be developed further with the aim of bringing the Davidson Property to production.

26.0 RECOMMENDATIONS

Based on the conclusions, AMPL recommends the following.

26.1 CONTACTS WITH THE LOCAL COMMUNITIES

Based on the conclusions, AMPL recommends the following technical and social direction:

1. Engage Wet'suwet'en and Gitxsan First Nations in discussions with the aim of establishing a Memorandum of Understanding (MOU) with each.

2. Complete the necessary environmental work for baseline studies, hydrogeology, geochemistry, hydrology, air quality, noise emissions, and effluent receiving water studies as outlined by Ms. M. Tanguay.

3. Conduct further metallurgical testing will be required to advance the Project to a Pre-Feasibility level or higher, including testing for the economic recoverability of tungsten, copper, gallium, and rare earth elements. Sampling requirements are as follows:
   a) The core for the samples should be bagged shortly after logging and splitting with nitrogen injection into the bag, the bags collected in sealable buckets, and nitrogen injection into the bucket prior to sealing. These steps are necessary to assure that sample aging can be eliminated as a potential source of error. It is strongly recommended that a metallurgist or geo-metallurgist be consulted prior to laying out the sample collection drilling program.
   b) The required mass of the samples should be determined in consultation with the metallurgical testing facility.
   c) It is recommended that a facility be chosen that has skilled individuals familiar with process development and assistance in sample and test work selection.
   d) Metallurgical testing should include new sampling of core including:
      iv) Per potentially economic mineralisation representative sample;
      v) Adjoining waste zone samples;
      vi) Variability samples by geography; and
      vii) Variability samples by mine life chronology.
   e) Mineralogical classification of samples, including mineral identification, particularly for tungsten, copper, and rare earth minerals.
   f) A comprehensive comminution testing program including Bond work indices.
   g) Bench testing on a full potentially economic mineralisation composite to further develop the preliminary flowsheet, benchmark reagents, and optimise additions.
   h) Locked cycle testing on a full potentially economic mineralisation composite to estimate cycle times.
   i) Mini-pilot plant testing of the developed flowsheet to finalise flows and cycle times – optional.
   j) Mini-pilot plant testing to assess variability, including impacts of dilution – optional.
   k) De-watering characteristics of concentrates and tailings.
   l) Characterisation of tailings, including:
      i) Acid base accounting separately on tailings.
      ii) Solution chemistry of all tailings materials; and
      iii) Suitability of tailings for mine backfill – particularly sulphide tailings.

iv) Concentrate analysis for salability, including penalty minerals, maximum allowable moisture, etc.
v) Investigate possible sales and concentrate shipping contracts

4. Complete an oriented core geotechnical drilling program to conduct a detailed rock mechanics analysis for stope geometry and mine design including portal design, stope geometries, and stope sequencing:
   a) Conduct a geotechnical assessment of the bedrock in the area of surface infrastructure and the Tailings Management Facility (TMF).

5. Complete a trade-off study on alternative methods of excavating the twin access drifts with the aim of reducing the development time and capital costs.

6. Further studies are recommended to advance the tailings facility design.
   a) Geotechnical and hydrogeological investigations including:
      i) Laboratory testing to confirm site conditions, identify any potential geologic hazards;
      ii) Characterise foundations and groundwater conditions; and
      iii) Identify suitable borrow sources for construction fill.

   b) Tailings characterisation testing is recommended to better define the:
      i) geochemical,
      ii) physical, and
      iii) settling, as well as filtration properties to validate the TMF design criteria.

   c) Site specific precipitation and evaporation data should be collected and a site-specific water balance model performed to confirm collection pond sizing and discharge water volumes.

   d) A grading plan should be developed that optimises the cut-fill balance for the TMF base grade.

   e) Consider amending the closure cover if it can be demonstrated that the compacted tailings have an equivalent permeability and do not pose a chemical stability risk.

All recommendations should be performed as part of a follow up PFS or FS. The cost to complete a Pre-Feasibility or Feasibility Study for the Project is estimated to be between $3 million to $5 million.

It is recommended that Moon River should increase its awareness to the communities in the region whether they are part of the legal ownership of the surface areas or they are neighbours. The importance of engagement at this stage will prove beneficial moving forward.

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CERTIFICATE OF QUALIFICATIONS

I, Brian C. LeBlanc, residing in Thunder Bay, Ontario, P7G 1M6, Canada, do hereby certify that:

1. I am President and a Principal at A-Z Mining Professionals Ltd.
2. This certificate applies to the report titled "National Instrument 43-101 Technical Report for the Davidson Project Preliminary Economic Assessment" for Moon River Capital Ltd. (the "Technical Report"), with an effective date of February 22, 2023.
3. I am a graduate of Michigan Technological University in 1986 with a Degree in Mining Engineering.
4. I am licensed by the Professional Engineers of Ontario, Registration No. 904279972-10.
5. I have read the definition of "Qualified Person" set out in National Instrument 43-101 ("**NI 43-101**") and certify that, by reason of my education, affiliation with a professional association (as defined in NI 43-101) and past relevant work experience, I am a "Qualified Person" for the purposes of NI 43-101.
6. My relevant experience for the purpose of the Technical Report is:
   a) Extensive and progressively more senior engineering and operational duties at base metals, gold and nickel mining operations and development projects.
   b) Fourteen (14) years of experience directing and overseeing several scoping level, pre-feasibility level, and feasibility level studies for mines and mining companies.
   c) Mill Operator – Giant Yellowknife Mines, 1974 – 1975.
   d) Crusher Operator/Screening Plant Operator/Loadout Operator/Surveyor – Steep Rock Iron Mines Ltd., 1976 – 1979.
   e) Mine Planner/Chief Surveyor – Nanisivik Mines Ltd., 1981 – 1984.
   f) Mining Engineer/Underground Supervisor/Mine General Foreman/Technical Services Superintendent/Mine Superintendent – Williams Mine, 1986 – 2003.
   g) Manager of Mining – Kinross' Kubaka Mine (Russia), 2003 – 2004.
   h) Technical Services Superintendent – Lac Des Isles Mines, 2004 – 2006.
   i) Project Superintendent – Redpath Indonesia, 2006 – 2007.
   j) Project Manager for Ontario – North American Palladium Ltd., 2007 – 2010.
   k) General Manager/Vice President/President – NordPro Mine & Project Management Services Ltd, 2010 – 2014.
   l) President, A-Z Mining Professionals Limited, February 2014 to Present.

7. I assisted in preparation of the Technical Report and Peer Review for Sections 1.0 thru 15.0. I co-authored and am responsible for Sections 1.0 and 16.0 thru 26.0 of the Technical Report.
8. I have completed a personal inspection of the Property that is the subject of the Technical Report.
9. As of the effective date of the Technical Report, to the best of my knowledge, information, and belief, the Technical Report contains all scientific and technical information that is required to be disclosed to make the Technical Report not misleading.
10. I am independent of both the issuer and the vendor applying all the tests in Section 1.5 of NI 43-101.
11. My prior involvement with the project was co-authoring "National Instrument 43-101 Technical Report titled, "National Instrument NI 43-101 Technical Report for the Davidson Project Resources Update Longitude 127° 17′ 52.1″ W, Latitude 54° 48′ 51.6″ N," the "Technical Report"), with an effective date of August 11, 2023.
12. I have read NI 43-101, Form 43-101F1 and the Technical Report, and the Technical Report has been prepared in compliance therewith.

Effective Date: 22nd day of February 2024

Signing Date: 2nd day of April 2024

Original Signed and Sealed by Brian C. LeBlanc

Brian C. LeBlanc, P. Eng, Registration No. 904279972-10

CERTIFICATE OF QUALIFICATIONS

I, Finley Bakker, P.Geo., residing in Campbell River, B.C., V9H-1C6, Canada, do hereby certify that:

1. I am a Professional Geoscientist Registration No. 18,639 at Finley Bakker Consulting Permit Number 1003901.
2. This certificate applies to the report titled "National Instrument 43-101 Technical Report for the Davidson Project Preliminary Economic Assessment" for Moon River Capital Ltd. (the "Technical Report"), with an effective date of February 22, 2023.
3. I am a graduate of McMaster University with a Hons. Bachelor of Science in Geology (1979)
4. I am a licensed Professional Geologist with EGBC (1991) in the Province of British Columbia, Canada (Registration No. 18,639).
5. I co-authored and assisted in the preparation of Sections 1.0 through 15.0 and am responsible for Section 14.0.
6. I have read the definition of "Qualified Person" set out in National Instrument 43-101 ("**NI 43-101**") and certify that, by reason of my education, affiliation with a professional association (as defined in NI 43-101) and past relevant work experience, I am a "Qualified Person" for the purposes of NI 43-101.
7. My relevant experience for the purpose of the Technical Report is:
    a) Practiced my profession continuously since 1979.
    b) Thirty-six (36) years experience utilizing MineSight™ software.
    c) Forty-four (44) years experience calculating Resources and Reserves.
    d) Chief Geologist at four mines.
    e) Have also held the positions of Senior Resource Geologist, Exploration Manager and Superintendent of Technical Services.
8. I have not completed a personal inspection of the Property that is the subject of the Technical Report.
9. As of the effective date of the Technical Report, to the best of my knowledge, information, and belief, the Technical Report contains all scientific and technical information that is required to be disclosed to make the Technical Report not misleading.
10. I am independent of the issuer applying all the tests in Section 1.5 of NI 43-101.
11. I am also independent of the Vendor and the Property.
12. My prior involvement with the project was co-authoring "National Instrument 43-101 Technical Report titled, "National Instrument NI 43-101 Technical Report for the Davidson Project Resources Update Longitude 127° 17′ 52.1″ W, Latitude 54° 48′ 51.6″ N," the "Technical Report"), with an effective date of August 11, 2023.
13. I have read NI 43-101, Form 43-101F1 and the Technical Report, and the Technical Report has been prepared in compliance therewith.

Effective Date: 22nd day of February 2024

Signing Date: 2nd day of April 2024

Original Signed and Sealed by Finley Bakker

Finley Bakker, P.Geo., Registration No. 18,639, Permit Number 1003901

CERTIFICATE OF QUALIFICATIONS

I, John Eggert, residing at 158 David Street, Sudbury, Ontario, P3E 1T4, Canada, do hereby certify that:

1. I am a Professional Engineer at Eggert Engineering Inc.
2. This certificate applies to the report titled "National Instrument 43-101 Technical Report for the Davidson Project Preliminary Economic Assessment" for Moon River Capital Ltd. (the "Technical Report"), with an effective date of February 22, 2023.
3. I am a graduate of Queen's University at Kingston in 1990 with a BSc in Mining Engineering.
4. I am licensed by the Professional Engineers Ontario (License No. 90397597).
5. I have read the definition of "Qualified Person" set out in National Instrument 43-101 ("**NI 43-101**") and certify that, by reason of my education, affiliation with a professional association (as defined in NI 43-101) and past relevant work experience, I am a "Qualified Person" for the purposes of NI 43-101.
6. I have worked in operations, technical and managerial positions in Canada. I have been an independent engineer for thirteen years. I have performed mill designs, metallurgical accounting, cost estimations, operations management, due diligence reviews and report writing for mining projects in Canada, the USA and Mexico.
7. I authored and assisted in preparation of the Technical Report and take responsibility for Section 13.0 and Section 17.0.
8. I have not completed a personal inspection of the Property that is the subject of the Technical Report.
9. As of the effective date of the Technical Report, to the best of my knowledge, information and belief, the Technical Report contains all scientific and technical information that is required to be disclosed to make the Technical Report not misleading.
10. I am independent of the issuer applying all the tests in Section 1.5 of NI 43-101.
11. I have not had prior involvement with the Property that is the subject of the Technical Report.
12. I have read NI 43-101, Form 43-101F1 and the Technical Report, and the Technical Report has been prepared in compliance therewith.

Effective Date: 22nd day of February 2024.

Signing Date: 2nd day of April 2024.

Original Signed and Sealed by John Eggert

John Eggert, P.Eng., License No. 90397597
President, Eggert Engineering Inc.

APPENDIX 1.0 DAVIDSON MOLYBDENUM PROJECT CASH FLOW

DAVIDSON MOLYBDENUM PROJECT BC, CANADA	Tonnes	Grade MoS_2	Grade Mo	
Tonnage > 0.25%	46,785,714	0.34	0.26	
5% Dilution	2,339,286	0.18	0.23	
Total Tonnes	49,125,000	0.34	0.259	Note: Includes dilution at .27%, grade of Tonnage between 0.25% and 0.3% cut-off

Description	Unit	Unit Rate		Year (3)	(2)	(1)	1	2	3	4	5	6	7	8	9	10	11	12	13	14	15	16	17	18	19	20	Total
PRODUCTION																											
Ore Mined	tonnes						1,625,000	2,500,000	2,500,000	2,500,000	2,500,000	2,500,000	2,500,000	2,500,000	2,500,000	2,500,000	2,500,000	2,500,000	2,500,000	2,500,000	2,500,000	2,500,000	2,500,000	2,500,000	2,500,000	2,500,000	**49,125,000**
Grade Mined	% MoS2		1.00				0.500%	0.450%	0.450%	0.400%	0.400%	0.400%	0.340%	0.340%	0.340%	0.310%	0.310%	0.310%	0.310%	0.310%	0.300%	0.280%	0.280%	0.250%	0.250%	0.250%	**0.336%**
Waste Mined	tonnes																										**0**
Contained Metal MoS2	ktonnes																										
Ore Processed	tonnes						1,625,000	2,500,000	2,500,000	2,500,000	2,500,000	2,500,000	2,500,000	2,500,000	2,500,000	2,500,000	2,500,000	2,500,000	2,500,000	2,500,000	2,500,000	2,500,000	2,500,000	2,500,000	2,500,000	2,500,000	**49,125,000**
Grade Processed	% MoS2						0.500%	0.450%	0.450%	0.400%	0.400%	0.400%	0.340%	0.340%	0.340%	0.310%	0.310%	0.310%	0.310%	0.310%	0.300%	0.280%	0.280%	0.250%	0.250%	0.250%	**0.336%**
Contained Metal MoS2	tonnes						8,125	11,250	11,250	10,000	10,000	10,000	8,500	8,500	8,500	7,750	7,750	7,750	7,750	7,750	7,500	7,000	7,000	6,250	6,250	6,250	**165125**
Processing Plant Recovery	%	92.0%					92.0%	92.0%	92.0%	92.0%	92.0%	92.0%	92.0%	92.0%	92.0%	92.0%	92.0%	92.0%	92.0%	92.0%	92.0%	92.0%	92.0%	92.0%	92.0%	92.0%	92.0%
Recovered Metal MoS2	tonnes						7,475	10,350	10,350	9,200	9,200	9,200	7,820	7,820	7,820	7,130	7,130	7,130	7,130	7,130	6,900	6,440	6,440	5,750	5,750	5,750	**151915**
Mo CONCENTRATE																											
Conversion MoS2 to Mo Oxide	%	59.9%					59.9%	59.9%	59.9%	59.9%	59.9%	59.9%	59.9%	59.9%	59.9%	59.9%	59.9%	59.9%	59.9%	59.9%	59.9%	59.9%	59.9%	59.9%	59.9%	59.9%	59.9%
Roaster Recovered Mo Oxide	%	99.7%					99.7%	99.7%	99.7%	99.7%	99.7%	99.7%	99.7%	99.7%	99.7%	99.7%	99.7%	99.7%	99.7%	99.7%	99.7%	99.7%	99.7%	99.7%	99.7%	99.7%	99.7%
Mo Oxide Concentrate Grade	%	55.5%					55.5%	55.5%	55.5%	55.5%	55.5%	55.5%	55.5%	55.5%	55.5%	55.5%	55.5%	55.5%	55.5%	55.5%	55.5%	55.5%	55.5%	55.5%	55.5%	55.5%	55.5%
Dry Mo Oxide Concentrate Produced	tonnes						8,049	11,144	11,144	9,906	9,906	9,906	8,420	8,420	8,420	7,677	7,677	7,677	7,677	7,677	7,430	6,934	6,934	6,191	6,191	6,191	163,576
Moisture Content	%	8.0%					8.0%	8.0%	8.0%	8.0%	8.0%	8.0%	8.0%	8.0%	8.0%	8.0%	8.0%	8.0%	8.0%	8.0%	8.0%	8.0%	8.0%	8.0%	8.0%	8.0%	8.0%
Wet Mo Oxide Concentrate Produced	tonnes						8,693	12,036	12,036	10,699	10,699	10,699	9,094	9,094	9,094	8,291	8,291	8,291	8,291	8,291	8,024	7,489	7,489	6,687	6,687	6,687	176,662
Mo Oxide Concentrate Contained Metal	tonnes						4,467	6,185	6,185	5,498	5,498	5,498	4,673	4,673	4,673	4,261	4,261	4,261	4,261	4,261	4,123	3,849	3,849	3,436	3,436	3,436	90,785
Mo Oxide Concentrate Contained Metal	kg						4,467,073	6,185,179	6,185,179	5,497,937	5,497,937	5,497,937	4,673,246	4,673,246	4,673,246	4,260,901	4,260,901	4,260,901	4,260,901	4,260,901	4,123,452	3,848,556	3,848,556	3,436,210	3,436,210	3,436,210	90,784,677
REVENUE & SMELTER CHARGES($000)																											
Mo Oxide Price - $US	$US/kg	$47.39	1.00	$21.50			$47.39	$47.39	$47.39	$47.39	$47.39	$47.39	$47.39	$47.39	$47.39	$47.39	$47.39	$47.39	$47.39	$47.39	$47.39	$47.39	$47.39	$47.39	$47.39	$47.39	
Mo OXIDE GROSS REVENUE	**$US/lb**	**$21.50**					**$211,676,743**	**$293,090,875**	**$293,090,875**	**$260,525,222**	**$260,525,222**	**$260,525,222**	**$221,446,439**	**$221,446,439**	**$221,446,439**	**$201,907,047**	**$201,907,047**	**$201,907,047**	**$201,907,047**	**$201,907,047**	**$195,393,916**	**$182,367,655**	**$182,367,655**	**$162,828,264**	**$162,828,264**	**$162,828,264**	**$4,301,922,726**
Roasting Charge	$US/kg	$5.69					$49,429	$68,441	$68,441	$60,836	$60,836	$60,836	$51,711	$51,711	$51,711	$47,148	$47,148	$47,148	$47,148	$47,148	$45,627	$42,585	$42,585	$38,023	$38,023	$38,023	**$1,004,557**
Transportation Cost	$US/wmt	$120.00					$1,043,122	$1,444,323	$1,444,323	$1,283,842	$1,283,842	$1,283,842	$1,091,266	$1,091,266	$1,091,266	$994,978	$994,978	$994,978	$994,978	$994,978	$962,882	$898,690	$898,690	$802,402	$802,402	$802,402	**$21,199,449**
T0TAL NET REVENUE	**$US**						**$210,584,191**	**$291,578,111**	**$291,578,111**	**$259,180,543**	**$259,180,543**	**$259,180,543**	**$220,303,462**	**$220,303,462**	**$220,303,462**	**$200,864,921**	**$200,864,921**	**$200,864,921**	**$200,864,921**	**$200,864,921**	**$194,385,407**	**$181,426,380**	**$181,426,380**	**$161,987,839**	**$161,987,839**	**$161,987,839**	**$4,279,718,719**
TOTAL NET REVENUE	**$CAD**	$1.35					**$284,288,658**	**$393,630,450**	**$393,630,450**	**$349,893,733**	**$349,893,733**	**$349,893,733**	**$297,409,673**	**$297,409,673**	**$297,409,673**	**$271,167,643**	**$271,167,643**	**$271,167,643**	**$271,167,643**	**$271,167,643**	**$262,420,300**	**$244,925,613**	**$244,925,613**	**$218,683,583**	**$218,683,583**	**$218,683,583**	**$5,777,620,271**
MINESITE OPERATING COSTS ($000)		**($/t)**																									
Diamond Drilling - Infill		$0.5	1.00				$812,500	$1,250,000	$1,250,000	$1,250,000	$1,250,000	$1,250,000	$1,250,000	$1,250,000	$1,250,000	$1,250,000	$1,250,000	$1,250,000	$1,250,000	$1,250,000	$1,250,000	$1,250,000	$1,250,000	$1,250,000	$1,250,000	$1,250,000	**$24,562,500**
Underground Mining		$21.07	1.00				$34,235,572	$52,670,110	$52,670,110	$52,670,110	$52,670,110	$52,670,110	$52,670,110	$52,670,110	$52,670,110	$52,670,110	$52,670,110	$52,670,110	$52,670,110	$52,670,110	$52,670,110	$52,670,110	$52,670,110	$52,670,110	$52,670,110	$52,670,110	**$1,034,967,665**
Equipment Leasing							$20,571,590	$19,428,764	$11,292,533	$10,639,246	$9,985,959			$6,000,000	$6,000,000	$6,000,000											**$89,918,092**
Processing		$10.94	1.00				$17,781,539	$27,356,214	$27,356,214	$27,356,214	$27,356,214	$27,356,214	$27,356,214	$27,356,214	$27,356,214	$27,356,214	$27,356,214	$27,356,214	$27,356,214	$27,356,214	$27,356,214	$27,356,214	$27,356,214	$27,356,214	$27,356,214	$27,356,214	**$537,549,597**
Tailings Management Facility		$4.48	0.30				$2,184,000	$3,360,000	$3,360,000	$3,360,000	$3,360,000	$3,360,000	$3,360,000	$3,360,000	$3,360,000	$3,360,000	$3,360,000	$3,360,000	$3,360,000	$3,360,000	$3,360,000	$3,360,000	$3,360,000	$3,360,000	$3,360,000	$3,360,000	**$66,024,000**
Mine Indirects		$1.29	1.00				$2,096,250	$3,225,000	$3,225,000	$3,225,000	$3,225,000	$3,225,000	$3,225,000	$3,225,000	$3,225,000	$3,225,000	$3,225,000	$3,225,000	$3,225,000	$3,225,000	$3,225,000	$3,225,000	$3,225,000	$3,225,000	$3,225,000	$3,225,000	**$63,371,250**
Surface Department		$0.90	1.00				$1,456,325	$2,240,500	$2,240,500	$2,240,500	$2,240,500	$2,240,500	$2,240,500	$2,240,500	$2,240,500	$2,240,500	$2,240,500	$2,240,500	$2,240,500	$2,240,500	$2,240,500	$2,240,500	$2,240,500	$2,240,500	$2,240,500	$2,240,500	**$44,025,825**
General & Administration		$2.20	1.00				$3,576,788	$5,502,750	$5,502,750	$5,502,750	$5,502,750	$5,502,750	$5,502,750	$5,502,750	$5,502,750	$5,502,750	$5,502,750	$5,502,750	$5,502,750	$5,502,750	$5,502,750	$5,502,750	$5,502,750	$5,502,750	$5,502,750	$5,502,750	**$108,129,038**
Total Minesite Operating Cost	**$**	**38.24**	**1.00**	**$0**			**$82,714,563**	**$115,033,338**	**$106,897,107**	**$106,243,820**	**$105,590,533**	**$95,604,574**	**$95,604,574**	**$101,604,574**	**$101,604,574**	**$101,604,574**	**$95,604,574**	**$95,604,574**	**$95,604,574**	**$95,604,574**	**$95,604,574**	**$95,604,574**	**$95,604,574**	**$95,604,574**	**$95,604,574**	**$95,604,574**	**$1,968,547,967**
OPERATING INCOME ($000)																											
	$			**$0**			**$201,574,096**	**$278,597,112**	**$286,733,343**	**$243,649,913**	**$244,303,200**	**$254,289,160**	**$201,805,100**	**$195,805,100**	**$195,805,100**	**$169,563,070**	**$175,563,070**	**$175,563,070**	**$175,563,070**	**$175,563,070**	**$166,815,726**	**$149,321,040**	**$149,321,040**	**$123,079,010**	**$123,079,010**	**$123,079,010**	**$3,809,072,304**

ROYALTIES ($000)													0	0	0	0	0	0	0	0	0						
Davidson	3.0%						$8,528,660	$11,808,913	$11,808,913	$10,496,812	$10,496,812	$10,496,812	$8,922,290	$8,922,290	$8,922,290	$8,135,029	$8,135,029	$8,135,029	$8,135,029	$8,135,029	$7,872,609	$7,347,768	$7,347,768	$6,560,507	$6,560,507	$6,560,507	**$173,328,603**
EBITDA	**$**			**$0**		**$0**	**$193,045,436**	**$266,788,199**	**$274,924,430**	**$233,153,101**	**$233,806,388**	**$243,792,348**	**$192,882,810**	**$186,882,810**	**$186,882,810**	**$161,428,041**	**$167,428,041**	**$167,428,041**	**$167,428,041**	**$167,428,041**	**$158,943,117**	**$141,973,272**	**$141,973,272**	**$116,518,503**	**$116,518,503**	**$116,518,503**	**$3,635,743,701**
BOOK DEPRECIATION (STRAIGHT LINE)																											
Cumulative Processed Tonnes							1,625,000	4,125,000	6,625,000	9,125,000	11,625,000	14,125,000	16,625,000	19,125,000	21,625,000	24,125,000	26,625,000	29,125,000	31,625,000	34,125,000	36,625,000	39,125,000	41,625,000	44,125,000	46,625,000	49,125,000	
Years Left	20.0			20.0	20.0	20.0	19.0	18.0	17.0	16.0	15.0	14.0	13.0	12.0	11.0	10.0	9.0	8.0	7.0	6.0	5.0	4.0	3.0	2.0	1.0	0.0	
Opening Balance				$0	$90,192,738	$268,617,533	$451,184,261	$544,724,672	$551,823,572	$522,985,752	$493,541,878	$462,695,511	$431,849,143	$401,002,776	$382,656,408	$350,768,374	$330,317,514	$303,893,125	$270,127,222	$236,361,320	$202,595,417	$168,829,514	$147,563,611	$110,672,708	$73,781,806	$26,890,903	
Additions				$90,192,738	$178,424,795	$182,566,727	$123,802,893	$35,768,619	$1,819,045	$1,319,994	$0	$0	$0	$12,500,000	$0	$11,437,174	$6,607,363	$0	$0	$0	$0	$12,500,000	$0	$0	-$10,000,000	$6,607,363	
Total Depreciation				$0	$0	$0	$30,262,482	$28,669,720	$30,656,865	$30,763,868	$30,846,367	$30,846,367	$30,846,367	$30,846,367	$31,888,034	$31,888,034	$33,031,751	$33,765,903	$33,765,903	$33,765,903	$33,765,903	$33,765,903	$36,890,903	$36,890,903	$36,890,903	$26,890,903	
Closing Balance				$90,192,738	$268,617,533	$451,184,261	$544,724,672	$551,823,572	$522,985,752	$493,541,878	$462,695,511	$431,849,143	$401,002,776	$382,656,408	$350,768,374	$330,317,514	$303,893,125	$270,127,222	$236,361,320	$202,595,417	$168,829,514	$147,563,611	$110,672,708	$73,781,806	$26,890,903	$6,607,363	
DD&A				**$0**	**$0**	**$0**	**$30,262,482**	**$28,669,720**	**$30,656,865**	**$30,763,868**	**$30,846,367**	**$30,846,367**	**$30,846,367**	**$30,846,367**	**$31,888,034**	**$31,888,034**	**$33,031,751**	**$33,765,903**	**$33,765,903**	**$33,765,903**	**$33,765,903**	**$33,765,903**	**$36,890,903**	**$36,890,903**	**$36,890,903**	**$26,890,903**	
Net Operating Profit				**$0**	**$0**	**$0**	**$162,782,954**	**$238,118,479**	**$244,267,565**	**$202,389,234**	**$202,960,021**	**$212,945,980**	**$162,036,442**	**$156,036,442**	**$154,994,775**	**$129,540,006**	**$134,396,289**	**$133,662,138**	**$133,662,138**	**$133,662,138**	**$125,177,214**	**$108,207,369**	**$105,082,369**	**$79,627,600**	**$79,627,600**	**$89,627,600**	**$2,988,804,352**
CAPITAL EXPENDITURES ($000)																											
Exploration	$		1.00	$1,000,000	$1,000,000	$1,000,000																				$0	**$3,000,000**
Mine	$		1.00	$34,377,033	$52,739,041	$50,440,360	$24,123,972																			$0	**$161,680,406**
Equipment Leasing			1.00	$9,441,098	$8,951,559	$8,462,021								$ 10,000,000								$10,000,000					**$46,854,678**
Processing Plant	$		1.00		$70,000,000	$50,125,000	$35,000,000	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	**$155,125,000**
Underground Infrastructure	$		1.00		$4,886,165	$1,815,180	$23,375,430	$28,614,895	$1,455,236	$1,055,995																	**$61,202,902**
Surface Infrastructure & Mobile Equipment	$		1.00	$23,636,394	$1,463,403	$13,361,412	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	**$38,461,210**
Tailings Management Facilities	$		1.00			$9,149,739	$0	$0	$0	$0	$0	$0	$0	$0	$0	$9,149,739	$5,285,890	$0	$0	$0	$0	$0	$0	$0	$0	$5,285,890	**$28,871,258**
Owner's Costs	$		1.00	$3,699,666	$3,699,668	$3,699,670	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	**$11,099,003**
Contingency	$	25%	1.00	$18,038,548	$35,684,959	$34,513,345	$20,624,850	$7,153,724	$363,809	$263,999	$0	$0	$0	$2,500,000	$0	$2,287,435	$1,321,473	$0	$0	$0	$0	$2,500,000	$0	$0	$0	$1,321,473	**$126,573,614**
Working Capital	$						$20,678,641																				**$20,678,641**
Mine Closure	$		1.00			$10,000,000																			-$10,000,000		**$0**
Total Capital Expenditures	**$**		**1.00**	**$90,192,738**	**$178,424,795**	**$182,566,727**	**$123,802,893**	**$35,768,619**	**$1,819,045**	**$1,319,994**	**$0**	**$0**	**$0**	**$12,500,000**	**$0**	**$11,437,174**	**$6,607,363**	**$0**	**$0**	**$0**	**$0**	**$12,500,000**	**$0**	**$0**	**-$10,000,000**	**$6,607,363**	**$653,546,711**
CORPORATE INCOME TAX ($000)																											
Tax Payable																											
Federal Corporate Income Tax							$6,689	$18,636	$23,577	$21,436	$23,930	$26,990	$21,646	$21,555	$22,093	$19,150	$20,113	$20,430	$20,759	$21,001	$20,078	$17,781	$17,752	$14,606	$14,861	$14,988	**$388,071**
Provincial Corporate Income Tax							$6,018	$17,114	$21,339	$19,160	$21,017	$23,429	$18,759	$18,428	$19,001	$16,304	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	**$180,571**
Mining Tax							$10,110	$31,568	$37,039	$31,503	$31,759	$33,058	$26,235	$23,830	$25,455	$20,556	$21,964	$22,823	$22,823	$22,823	$21,686	$17,787	$19,412	$16,000	$17,300	$15,141	**$468,872**
Total Tax Payable							**$22,817**	**$67,318**	**$81,955**	**$72,099**	**$76,707**	**$83,477**	**$66,640**	**$63,813**	**$66,549**	**$56,010**	**$42,078**	**$43,253**	**$43,582**	**$43,824**	**$41,764**	**$35,568**	**$37,164**	**$30,606**	**$32,161**	**$30,129**	**$1,037,514**
PROJECT CASHFLOWS ($000)																											
Project Pre-Tax Cashflow				**-$90,192,738**	**-$178,424,795**	**-$182,566,727**	**$69,242,542**	**$231,019,580**	**$273,105,385**	**$231,833,108**	**$233,806,388**	**$243,792,348**	**$192,882,810**	**$174,382,810**	**$186,882,810**	**$149,990,867**	**$160,820,678**	**$167,428,041**	**$167,428,041**	**$167,428,041**	**$158,943,117**	**$129,473,272**	**$141,973,272**	**$116,518,503**	**$126,518,503**	**$109,911,140**	**$2,982,196,990**
Project Cumulative Cashflow				-$90,192,738	-$268,617,533	-$451,184,261	-$381,941,718	-$150,922,139	$122,183,246	$354,016,354	$587,822,742	$831,615,090	$1,024,497,899	$1,198,880,709	$1,385,763,518	$1,535,754,385	$1,696,575,063	$1,864,003,104	$2,031,431,144	$2,198,859,185	$2,357,802,302	$2,487,275,573	$2,629,248,845	$2,745,767,347	$2,872,285,850	$2,467,713,442	
Project After Tax Cashflow				**-$90,192,738**	**-$178,424,795**	**-$182,566,727**	**$46,425,215**	**$163,701,417**	**$191,149,929**	**$159,734,116**	**$157,099,211**	**$160,315,749**	**$126,243,126**	**$110,569,674**	**$120,334,010**	**$93,980,418**	**$118,743,118**	**$124,174,706**	**$123,846,189**	**$123,603,813**	**$117,179,234**	**$93,905,762**	**$104,809,501**	**$85,912,396**	**$94,357,516**	**$79,781,770**	**$1,944,682,609**
Project After Tax Cumulative Cashflow				-$90,192,738	-$268,617,533	-$451,184,261	-$404,759,046	-$241,057,629	-$49,907,700	$109,826,416	$266,925,628	$427,241,377	$553,484,503	$664,054,176	$784,388,186	$878,368,604	$997,111,722	$1,121,286,428	$1,245,132,617	$1,368,736,430	$1,485,915,664	$1,579,821,426	$1,684,630,927	$1,770,543,323	$1,864,900,839	$1,565,697,434	

APPENDIX 2.0 DAVIDSON MOLYBDENUM PROJECT PEA GEOMECHANICAL REVIEW_FINAL_REPORT

A-Z MINIING

DAVIDSON MOLYBDENUM PROJECT PEA GEOMECHANICAL REVIEW

PEA GEOMECHANICAL REVIEW

1. EXECUTIVE SUMMARY

1.1 The orebody is hosted in a granodiorite, a strong stiff rock. The rock mass quality is good to very good (GSI = 65 TO 75).

1.2 For the purpose of this study the orebody has been assumed to be dry due to lack of hydrogeological data.

1.3 The available data indicate that the rock mass is dissected by several joint sets (i. e. blocky). A statistical analysis of joint set densities is not possible with the existing data and thus joint set dominance cannot be determined. There are several steeply dipping joint sets but also at least two low angle dip sets (≤ 35°) and two or more sets dipping between ~40° and 80°.

1.4 The far field in situ stress state is unknown. Two possibilities are evaluated in this report: (i) a gravitational stress field and (ii) a stress field where the maximum principle stress is horizontal. There is limited field evidence suggesting that (i) is more probable (joint surface staining in the adit indicating water flow plus one striated fracture exhibiting water flow).

1.5 Analysis of the existing structural data indicates that stope backs will be exposed to numerous wedges and will require deep secondary (cable bolt) support. Vertical stope walls are assumed to be unsupported. Analysis of the structural data indicates that an east – west orientation is favored for the longest stope wall with end walls being north – south. Stope stability analysis was conducted using the empirical Stability Graph technique. Resulting hydraulic radius (HR) ranges for the two limiting stress conditions are:

Maximum principle stress horizontal	
Back	**7.5 – 8.5**
East – west walls	**13.5 - 18**
North – south walls	**11 - 16**
Gravitational stress field	
Back	**7.5 - 9**
East – west walls	**11.5 – 16.5**
North – south walls	**9 - 14**

1.6 Secondary stopes or pillars, unless cut extremely thin, should not experience any overstress.

1.7 Stopes must be filled to prevent any possible surface deformation. Cemented paste backfill is recommended as the filling medium for operational efficiency and cost savings with the mine cycle.

1.8 The existing geotechnical database is sufficient for the present PEA study. It would, however, require a significant upgrade to be adequate for a feasibility level study.

2. TABLE OF CONTENTS

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3. INTRODUCTION AND TERMS OF REFERENCE

Bawden Engineering Limited was contracted by A-Z Mining to conduct a review of available geotechnical data for the Davidson Molybdenum Property Preliminary Economic analysis [PEA] on December 5, 2023. All project data and reports were provided to Bawden Engineering by A-Z Mining and are listed in the references section of this report. No site visit or further geotechnical investigation has been conducted by Bawden Engineering for this study. All geotechnical data analysis and resulting geomechanical stope and pillar stability analysis and backfill recommendations were conducted by Dr. W. F. Bawden.

4. PROJECT BACKGROUND INFORMATION

The Davidson molybdenum deposit (formerly know as the Yorke-Hardy deposit) is located within the eastern flank of Hudson Bay Mountain nine kilometers northwest of Smithers BC. The deposit is situated approximately 300 to 450 m below surface on the east side of Hudson Bay Mountain. Access to the deposit for additional 2006 exploration drilling was from an existing 2 km long adit with the portal at an elevation of 1066 m (Figure 1). Elevated molybdenum levels were first noted in 1944 and the deposit was delineated through extensive exploration drilling between 1957 and 1980. (After Golder Associates, 2006).

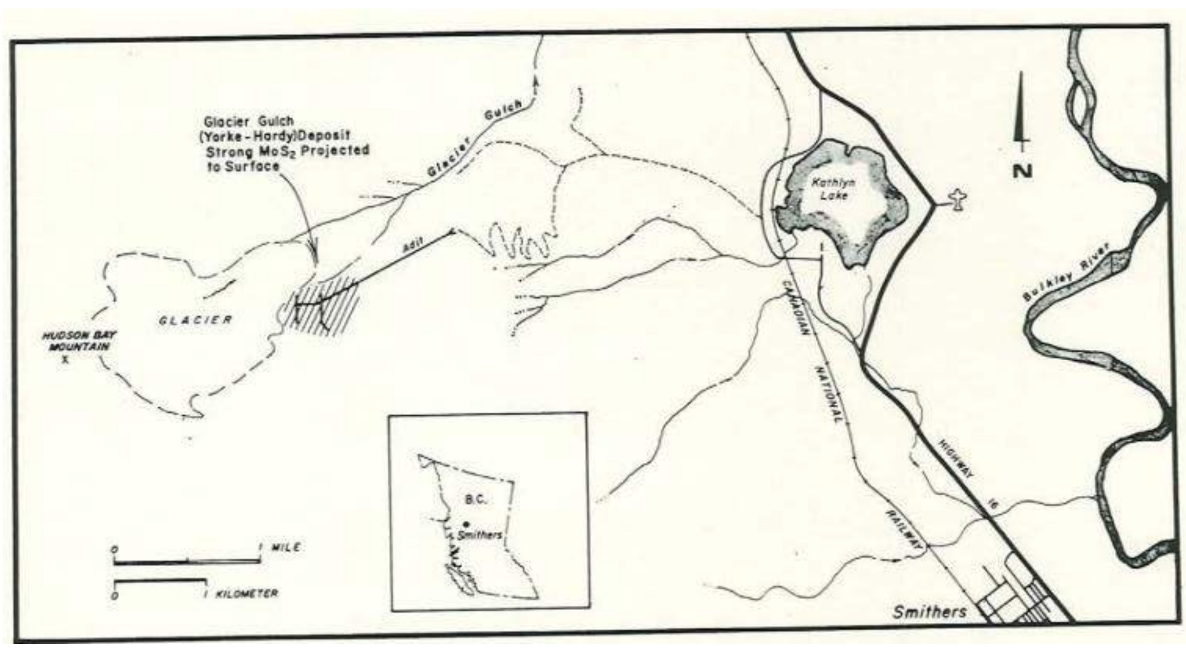


Figure 1: Site location (After White, 1981)

5. GENERAL GEOLOGICAL CONDITIONS

The general geological conditions are described in White (1981). The local geology is composed of a complex mix of igneous and volcanic formations. Figure 2 shows a transverse section through the deposit following the adit (azimuth 245°). The orebody is hosted within the Granodiorite sheet formation.

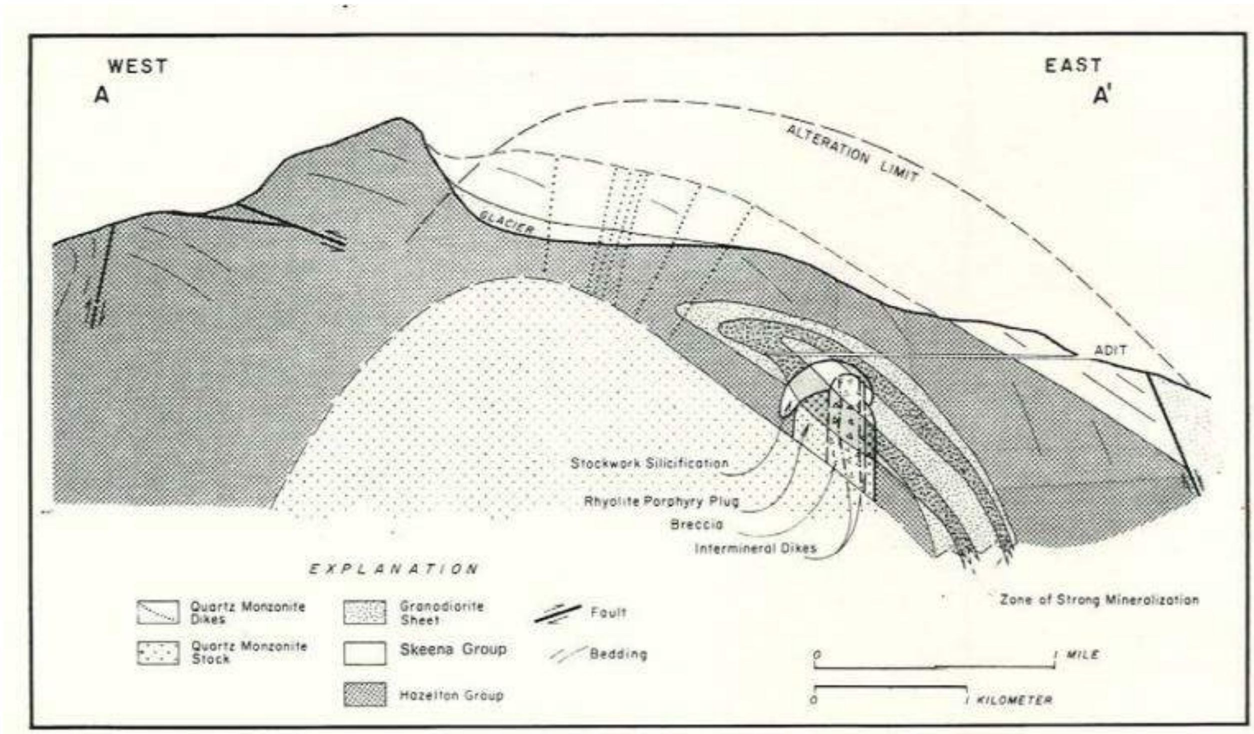


Figure 2: Diagrammatic East-West section along Hudson Bay Mountain (After White, 1981)

6. GEOTECHNICAL DATA SUMMARY AND INTERPRETATION

The majority of geotechnical data on the Davidson Deposit is contained in the report "Geotechnical Evaluation of the Yorke-Hardy Deposit" (White, 1981). This work included line mapping in the exploration adit, intact rock testing and basic rock mass classification. Additional geotechnical data was collected during a site visit by Golder (2006) where additional structural and oriented core data was obtained from an infill drilling program. This data is summarized and analyzed below.

6.1 Intact Rock Properties

Rock samples from the earlier exploratory drilling (White, 1981) were tested at the Colorado School of Mines. Results of this testing campaign are provided in Table 1. The key data from this study pertinent to the present PEA analysis is the Uniaxial Compressive Strength (UCS) values. This testing resulted in average UCS values for mineralized granodiorite of ~ 217MPa (SD = 103 MPa) and for nonmineralized granodiorite of ~176 MPa (SD = 64 MPa). Additional test samples were taken from the infill drilling program during the Golder 2006 campaign and these results are provided in Table 2. It is assumed that this data is all from mineralized granodiorite. The data shows UCS test values only. The data from DDH 187 (315.35 – 316.1 ft) and DDH-188 (231.15-231.85 ft) (red boxes) is not included in average UCS calculations since core photos indicated obvious failure along preexisting structure. The average and SD UCS values from this test program are 163 MPa and 53 MPa respectively.

Figure 3 shows a recent compilation of intact rock properties. Granodiorite is highlighted by the red box.

	Uniaxial Compression strength (psi)			Brazilian Disc Tension strength (psi)			Young's Modulus (psi x 106)			Angle of Internal Friction (degrees)	Cohesion psi	Specific gravity			Density			Poison's ratio		
	Mean	SD	No.	Mean	SD	No.	Mean	SD	No.			Mean	SD	No.	Mean	SD	No.	Mean	SD	No.
Mineralized Granodiorite	31,496	14,920	5	1,530	840	4	12.16	1.46	4	42.3	6,958	2.644	0.026	9	165.0	1.6	9	.241	-	4
Unmineralized Granodiorite	25,483	9,322	3	1,680	770	4	15.55	-	2	36.4	5,710	2.643	0.062	10	164.9	3.9	10	.138	-	2

Table 1: Summary of Intact Rocks tests (After White, 1981)

DRAFT

Borehole No.	Depth (feet)	Density (Kg/m^3)	UCS (MPa)
DDH-185	298.40 – 299.30	2704	116.5
DDH-185	323.80 – 324.65	2737	217.65
DDH-185	339.90 – 340.75	2670	75.93
DDH-185	430.70 – 431.55	2643	164.55
DDH-186	171.95 – 172.70	2671	141.3
DDH-186	225.40 – 226.15	2613	94.11
DDH-186	300.95 – 301.75	2656	238.37
DDH-187	254.35 – 255.00	2669	138.19
DDH-187	315.35 – 316.10	2668	60.11
DDH-188	185.20 – 186.30	2632	222.16
DDH-188	231.15 – 231.85	2656	27.04
DDH-188	325.70 – 326.80	2668	177.79
DDH-189	397.75 – 398.80	2660	213.75
DDH-189	492.90 – 493.70	2663	153.61

Table 2: Intact Rock properties (Golder 2006). Results highlighted in red not included in UCS data summary

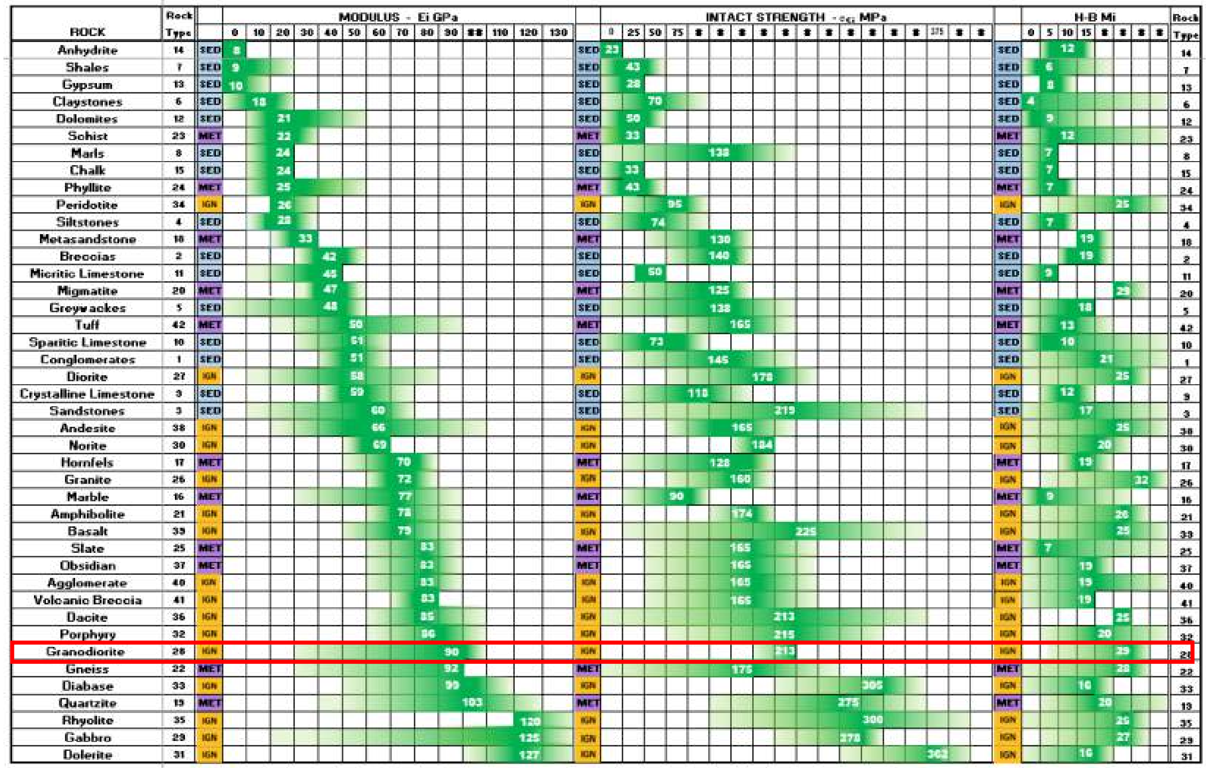


Figure 3: Average valus of intact rock modulus *Ei*, uniaxial compressive strength σ_{ci} and Hoek-Brown constant m_i for various rock types sorted in terms of increasing rock stiffness. (After Hoek, 2023)

UCS values for the studies referenced above fall within the general UCS data for granodiorite (highlighted in red box on Figure 3). The high SD's are normal in rock UCS testing, especially for such small sample sizes.

6.2 Rock Fabric

The earliest geotechnical structural data is provided by White (1981). This data was obtained from line mapping conducted in the Adit (Figure 4). Mapping was conducted separately in mineralized Granodiorite (line1), unmineralized Granodiorite (line 2) and the Hazelton Volcanics (line 3). Only the mapping in the Granodiorites will be discussed further in this report.

Figure 5 shows Schmidt stereonet plots of the mapping data for all three areas. Figure 6 shows the summary of fracture set statistics (mineralized and unmineralized granodiorites only). Column 2 of Figure 6, number of observations, is of particular importance. It is generally accepted that a minimum of ~ 100 observations is the minimum to provide a statistically sound analysis. While the total number of observations for line 1 (nonmineralized granodiorite) approaches that limit it is highly skewed by the 60 observations recorded for Set rank 1. This would imply a very strong, closely spaced individual joint set that would impose a strong anisotropy to this rock mass. Nowhere in the White (1981) report or the later Golder (2006) reports is any such dominant joint set mentioned. It is possible that this was a typographical error in the White report. Otherwise the number of recorded observations is trivial rendering a truly statistical data evaluation impossible.

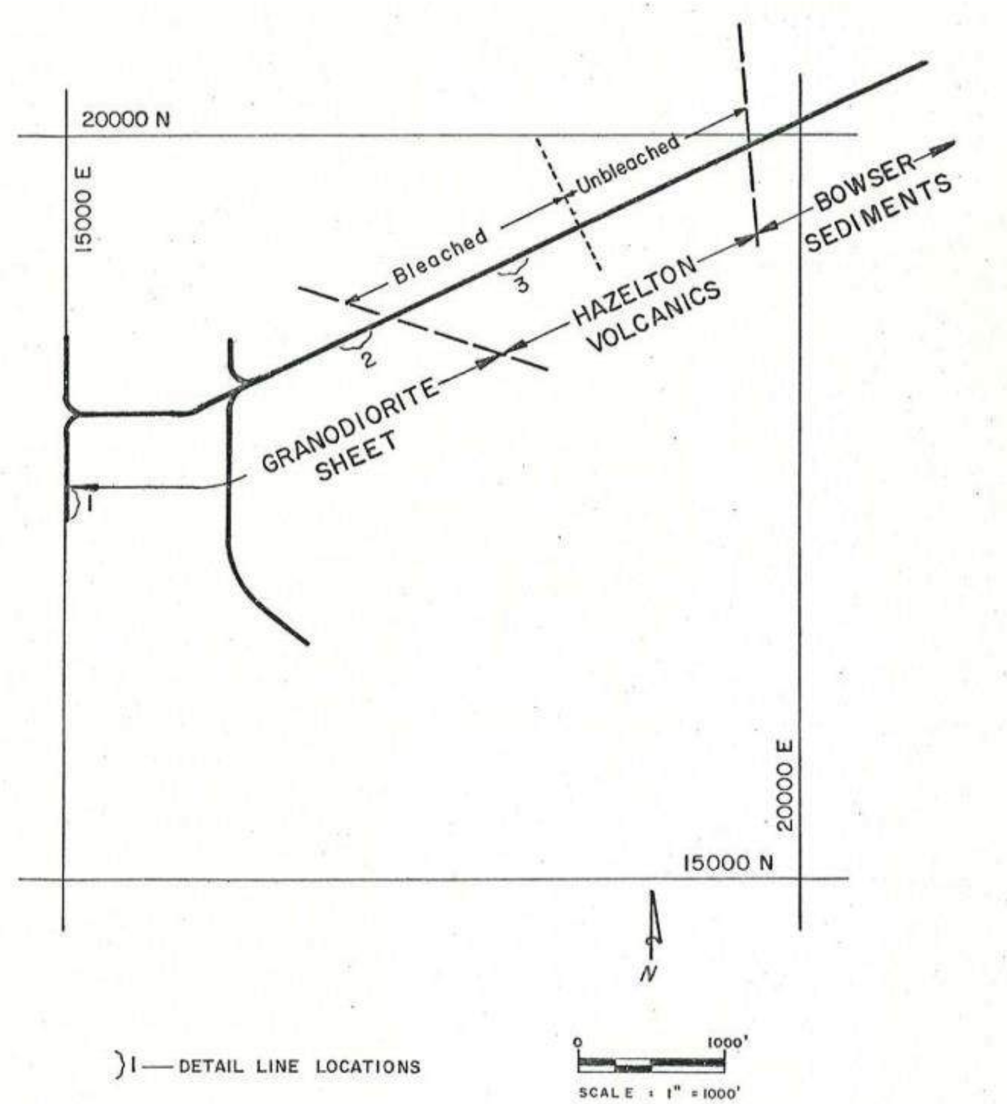


Figure 4: Sketch of generalized geology (3500 level) showing detail of line locations (After White 1981)

It is assumed that the data shown in Figure 6 represents the average dip/dip direction for each of the indicated set rankings. In order to make this data more easily interpretable the average dip/dip direction for each ranked set has been replotted as individual great circles on a lower hemisphere Schmidt stereonet and is displayed individually for nonmineralized and mineralized granodiorite in figure 7(a) and (b) respectively. In Figure 7(a) set rankings 5 and 6 were not plotted due to the very low number of observations relative to rank set 1. This will be discussed further later.

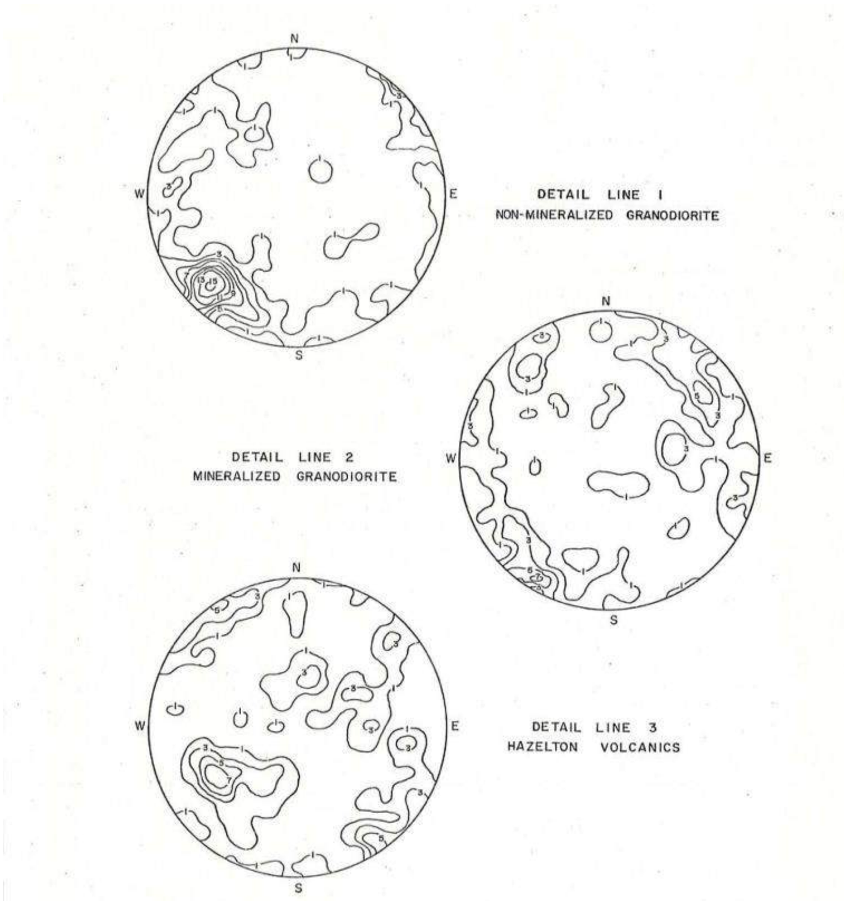


Figure 5: Schmidt plot of rock fractures (lower hemisphere per cent plot of pole densities – after White1981)

The Golder 2006 report provides geotechnical structural data based on an underground site visit to the adit plus oriented core logging data from the 2006 infill drill hole program. Two major structural features were mapped in the adit: (1) a sub horizontal fault ~ 80 m east of the intersection of the northwest and south east exploration drifts [picture numbers 3 & 4, Golder (2006)]. The second was a subvertical fault between drill stations 11 and 22 [pictures 5 & 6, Golder (2006)].

Oriented core drilling was done as part of the 2006 infill drilling program. The Golder 2006 report notes that “the structural data obtained during the geotechnical logging of the 2006 infill holes was not as comprehensive as originally anticipated. … The length of core drilled and geotechnically logged in the data provided to Golder in May 2006 was approximately 5000 feet, of this a little over 1000 feet contained orientation data for the identified structural features. Only 20% of the of the total number of joints, 801 oriented out of 3853 joints, resulted in the data from the first set of logs being smaller than anticipated. A second set of logs were provided to Golder in July 2006. This set was better with a total of 3800 feet of drilling and 1760 feet of core that had been successfully oriented.

The total length of core from the infill drilling was about 9000 feet with 3000 feet of the total length having been successfully oriented.” (Golder 2006, Section 4.3 Structure)

Detail Line 1 - Granodiorite - Out of Ore

Set Rank No.	Number of Observations	Dip Direction (Degrees)	Standard Deviation	Dip (Degrees)	Standard Deviation	Spacing (feet)
1	60	44.0	12.4	77.9	11.2	1.0
2	11	131.8	10.3	83.8	11.7	5.6
3	10	91.9	11.2	83.1	14.9	6.0
4	10	345.5	9.3	80.8	8.2	1.9
5	4	138.1	17.4	38.2	7.0	4.7
6	3	43.9	15.1	54.4	7.0	11.7
7	3	300.7	49.7	27.7	14.1	9.5

Detail Line 2 - Granodiorite - In Ore

Set Rank No.	Number of Observations	Dip Direction (Degrees)	Standard Deviation	Dip (Degrees)	Standard Deviation	Spacing (feet)
1	18	280.5	10.9	87.1	13.2	2.8
2	17	229.7	15.5	66.0	3.1	1.1
3	16	33.3	6.4	88.2	7.4	2.3
4	12	5.8	10.8	82.8	21.4	5.9
5	10	265.0	10.8	40.5	6.0	1.3
6	9	52.3	11.8	68.2	5.0	1.9
7	8	143.5	7.5	75.7	7.9	9.0
8	3	333.5	168.2	14.8	3.6	8.2

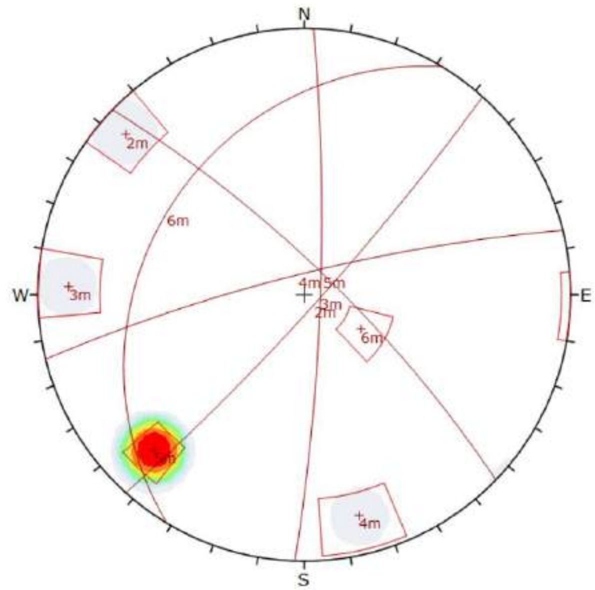
Figure 6: Summary of fracture set statistics (mineralized and unmineralized granodiorites only – After White 1981

The structural data discussed above was plotted on a lower hemisphere Schmidt stereonet. The stereonet figure was not included in the Golder report [page left blank] although all of the detailed drill logs were provided in an appendix to the report. An independent evaluation of the nature of the joint set clustering is therefore not possible without reinterpretation and plotting of raw data from the drill hole logs (outside of the scope for this study). No mention is made in the Golder report as to the cause of the poor orientation results (e. g. bad ground, inexperienced drillers, etc.). The joint sets identified on the stereonet are provided in Table 3 with the mean planes shown as great circles on a stereonet in Figure 8.

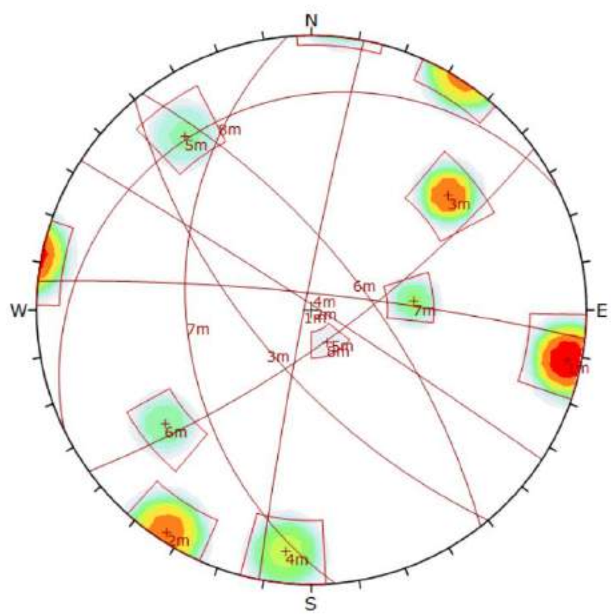
Golder (2006) states that “previous scan line mapping of three locations along the exploration drift identified 3 fracture sets (White, 1981):

- A north west striking fracture set dipping steeply to the north east.
- A north east striking set dipping to the south east.
- A north to north west striking set dipping to the west.”

The mean joint set orientations are provided in Table 4. The corresponding planes between the nonmineralized and mineralized granodiorite in Tables 3 and 4 are labelled as 'Identifier' in those respective tables.



(a)



(b)

Figure 7: Stereonet great circle projections of average joint ranked sets (a) nonmineralized; (b) mineralized granodiorite (Modified from White 1981)

Set	Dip	Dip Direction	Identifier
1	53	251	H
2	83	217	C
3	83	46	A
4	33	4	E
5	11	353	E

Table 3: Joint Set Data for Granodiorite obtained from oriented core drilling infill holes (Modified from Golder, 2006)

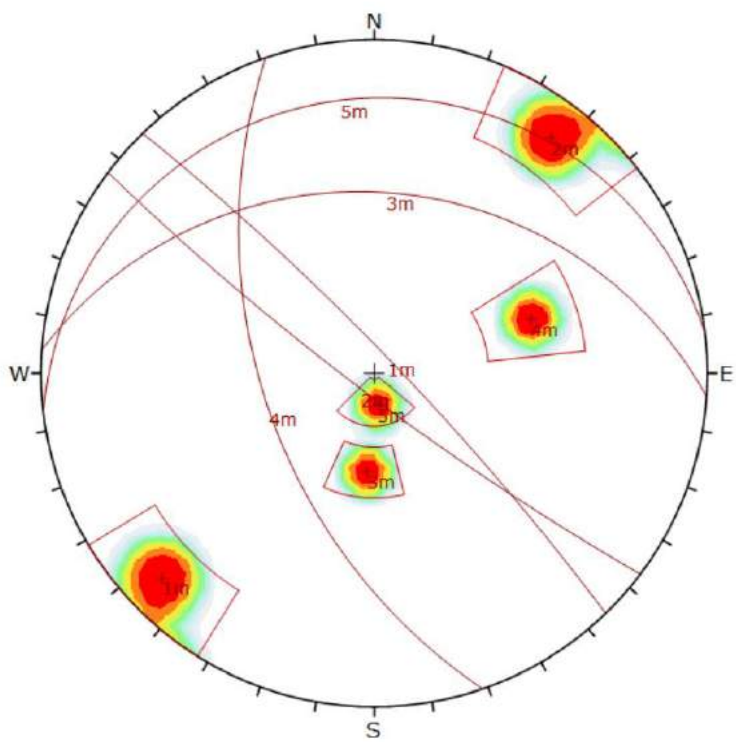


Figure 8: Stereonet great circle projection of mean data sets form 2006 in fill drill oriented core data (modified from Golder 2006)

NONMINERALIZED			
ID	Dip	Dip Direction	Identifier
2m	84	132	D
3m	83	92	B
4m	81	346	C
5m	78	44	A
6m	28	301	E
Mineralized			
ID	Dip	Dip Direction	Identifier
1m	87	281	B
2m	88	33	A
3m	66	230	F
4m	83	6	C
5m	76	144	D
6m	68	52	G
7m	41	265	H
8m	15	334	E

Table 4: Summary of Fracture Set Statistics [Modified from White, 1981]

Golder (2006) further states "Joint sets identified in the 2006 data did not match those identified by White." Following careful analysis of all available rock mass structural data I disagree with this Golder assessment. I have evaluated the three great circle stereonets [Figures 7a & b and 8] and, recognizing the high SD's normally associated with any joint fabric structural sampling, have identified what I believe are the same structural families in all three stereonets (identified by identifier letters). The 2006 Golder infill drill data provides the largest structural data set available. However, as noted in Golder (2006) "the sparsity of the data obtained from the oriented core and the drilling of all of the infill exploration holes along similar azimuths will have resulted in the orientation data being restricted" (i. e. blind zone on stereonet). This further explains why the data sets upon first inspection appear different.

Figure 9 shows a more detailed plan of the exploration adit complex. A number of valuable photos of the rock mass were obtained during the Golder 2006 site visit. A few of the relevant photos are included in the following discussion. The sub horizontal fault [Photos 3, 4] was tight. The sub vertical fault [Photo 5] was open for a considerable distance and there had been significant movement along this fault [Golder 2006].

The intact granodiorite exposed in the adit is generally massive, strong and stiff [Golder 2006, Photo 1]. Figure 10 [Golder Photo 7] shows the northeastern exploration drift south of drill station 15. The ground condition is blocky with multiple visible discontinuity surfaces highlighted. The fracture surface highlighted in the center is striated, iron stained and was producing a moderate quantity of water. Additional iron staining can be observed on a fracture surface on the left (west) wall also caused by water flow. Note the limited trace length of the majority of factures in the photo.

Figure 11 (Golder Photo 8) shows a view looking north up the northwest adit. Two high persistence, appearing to be approximately N-S striking fractures, the upper dipping east and the lower dipping west are evident.

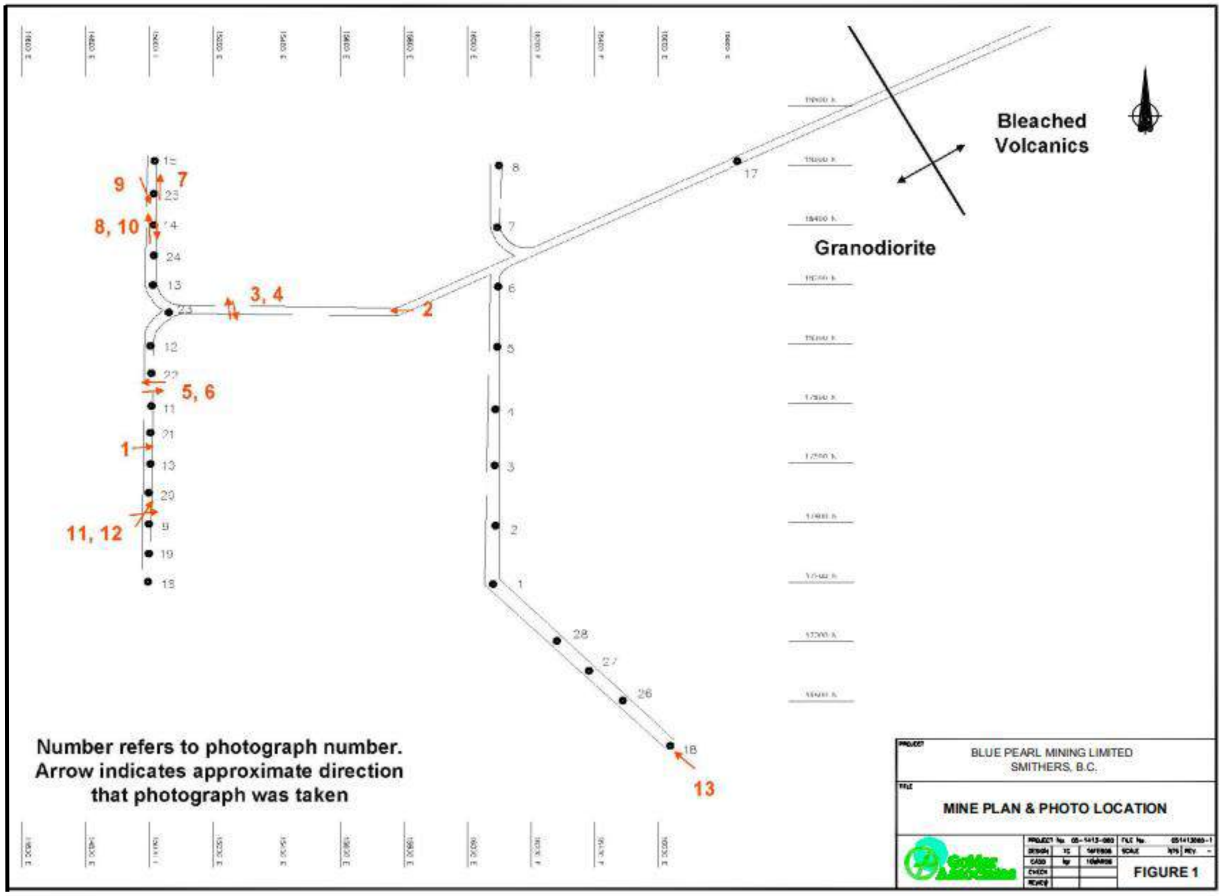


Figure 9: Detailed plan of exploration adit layout. Numbered black dots are drill stations (After Golder, 2006)

Figure 12 (Golder Photo 9) is a view south down the northeast adit showing a high persistence fracture dipping ~ 70° east. Figure 13 (Golder Photo 11) shows a combination of sub horizontal and steeply dipping fractures of varying persistence again creating blocky ground.

Golder photo 14 shows drill station 18 at the end of the southeastern exploration drift. This is the largest excavation existing underground with approximate dimensions of 8m H x 6m W x 8m L. The excavation is supported by 1.8 m long mechanical bolts and galvanized mesh. Note the iron staining on the walls. The present (2023) condition of this excavation is not known.

The importance of the photos shown is to indicate that some of the fracture sets can have high persistence members (e. g. the B and H joints (Table 4)). Unfortunately, joint strikes were not recorded for these features. It is however possible that the conjugate joints shown in Figure 11 are part of the F and G joints (Table 4).

A further observation from the data is that, by observation from Figure 7 (b), multiple potential wedges would be expected in flat stope backs. A further observation is that several sub horizontal joints were identified (e. g. Golder Photo 2 and others). Due to the nearly horizontal inclination of the adits however the sub horizontal jointing will naturally be underestimated in the mapping. Two separate sub horizontal joints were identified in the infill drilling logs however (Set E – Table 3).

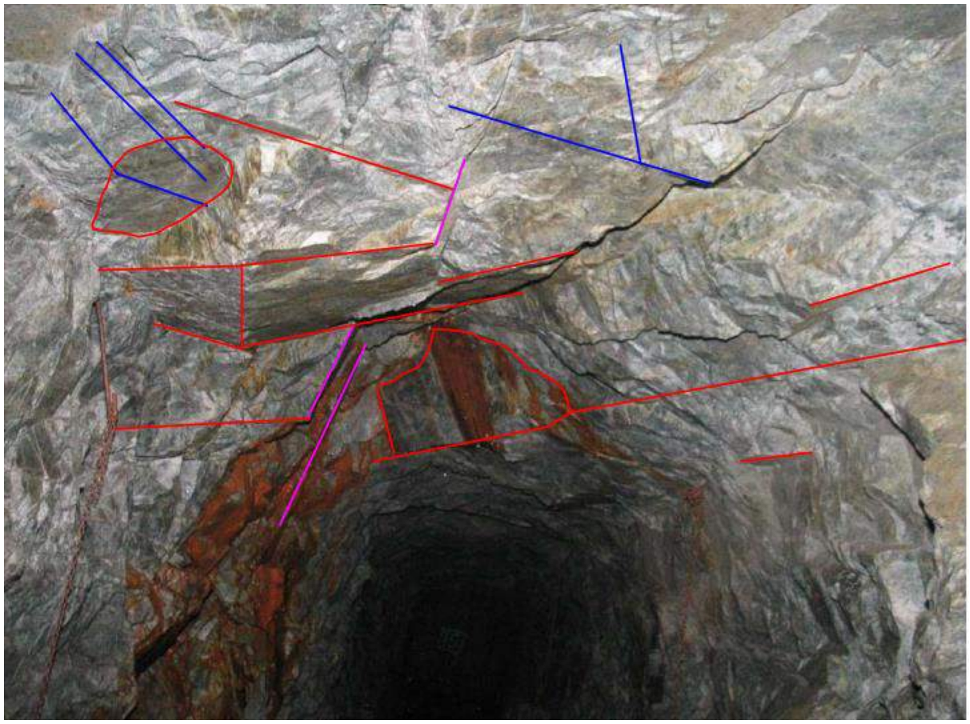


Figure 10: Golder Photo 7

Figure 11: Golder Photo 8

Figure 12: Golder Photo 9

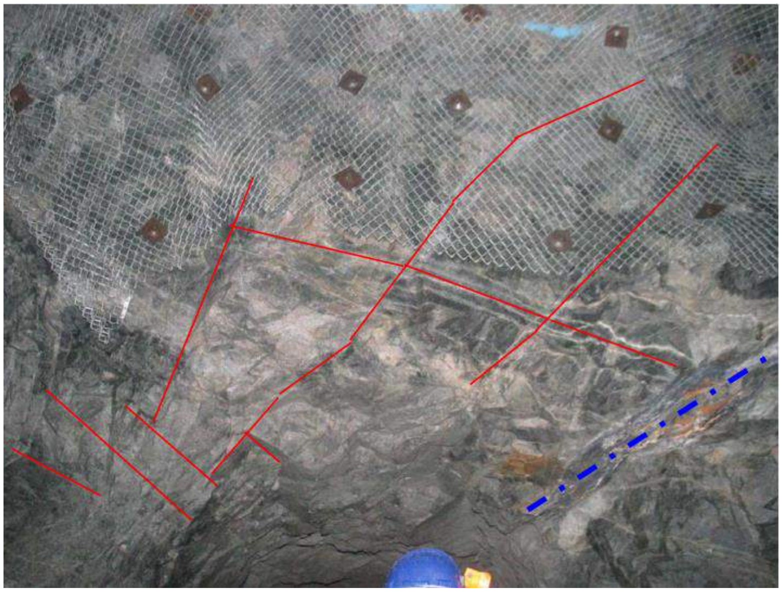


Figure 13: Golder Photo 11

Figure 14: Golder Photo 13

A final observation from the Golder 2006 photos is that the fracture surfaces generally appear to be planar and rough.

6.4 Far Field Stress

There have been no insitu stress (ISS) measurements conducted at or near the Davidson deposit. Both White (1981) and Golder (2006) suggest that the horizontal stress is likely to form the maximum principal stress (σ_1). While for a number of reasons this may be feasible, because of the deposit location on the flank of Hudson Bay Mountain (Figure 2) I question this assumption (see Figure 15). Additionally the observation of fracture staining from water flow and one fracture making significant flow may suggest that the horizontal stresses have been relaxed as indicated in Figure 15(b). For the purpose of this study I will use upper and lower bound far field stress assumptions to test sensitivity. The upper and lower bound ISS values used for the stope and pillar stability analyses (later in this report) are provided in Table 5. The orebody depth of 450 m used in Golder 2006 is also assumed in this report for far field ISS calculations.

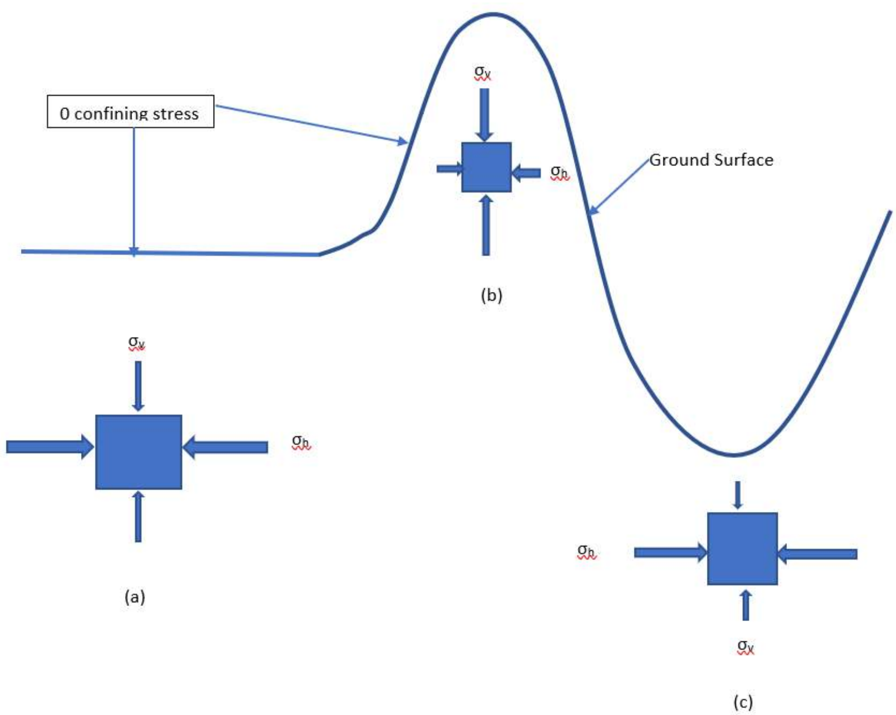


Figure 15: Stress conditions under varying terrain: (a) $\sigma_v \neq \sigma_h$; (b) $\sigma_v > \sigma_h$ due to stress relaxation of valley walls; (c) $\sigma_v << \sigma_h$ due to stress concentration below valley

Upper Bound	$\sigma_v = \Upsilon D = 0.027D = \sigma_h$
	$\sigma_H = 1.2\ \sigma_v$
Lower Bound	$\sigma_v = \Upsilon D = 0.027D$
	$\sigma_H = \sigma_h = 0.7\ \sigma_v$

Table 5: Far Field stress assumptions

Basically the actual far field insitu stress state is unknown and the values used in this report are a best estimate by the author.

6.5 Rock mass Classification

Due to the scant and high uncertainty nature of the geotechnical database for the Davidson property I have elected to use the Geological Strength Index system for rock mass classification. For this I used examination of the Golder 2006 photos from the adit complex to visually assess the appropriate GSI range. The origin and use of GSI is discussed in detail in Hoek et al (1995). The most recent iteration of the GSI plot is shown in Figure 16. The estimated GSI range for the Davidson mineralized granodiorite [the formation most relevant to this study] is shown with the red circle on Figure 16. The limiting GSI values of 65 and 75 are used later in the stope stability section.

6.6 Groundwater

The only groundwater information available is from the Golder 2006 observations from the adit incorporated in section 6.2. For the remainder of this report the potential influence of groundwater will be ignored (i. e. excavations will be assumed to be dry).

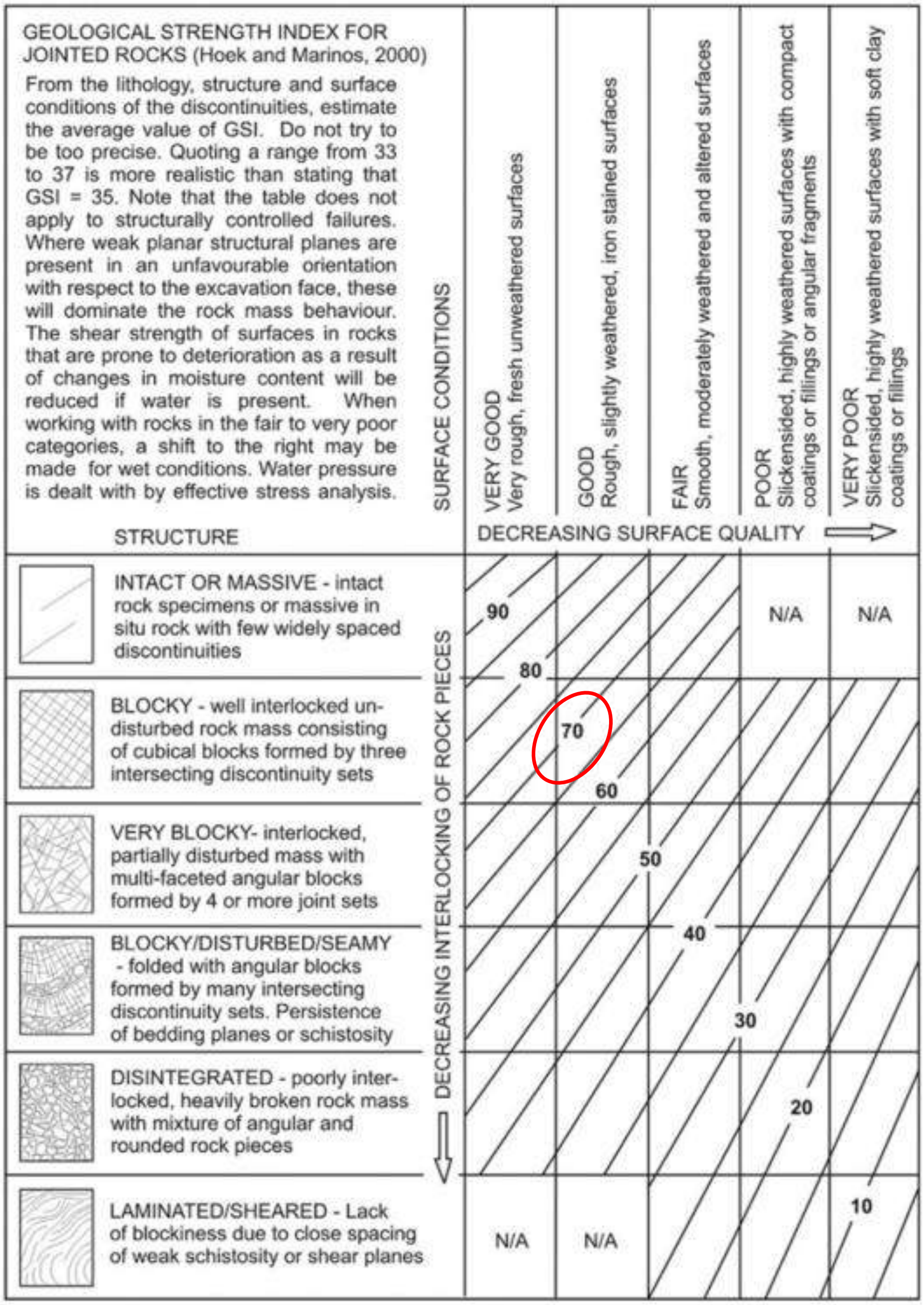


Figure 16: Characterization of blocky rock masses based on interlocking and joint conditions (After Hoek, 2023). Estimated GSI for the Davidson property is indicated by the red circle.

7. MAXIMUM STOPE DIMENSION ANALYSIS

The Stability Graph stope stability analysis (Potvin, 1988) is used herein for the evaluation of maximum stope or chamber size analysis. The methodology to utilize the Stability Graph is provided in Hoek et al (1995 – chapter 14). The Stability Graph is shown in Figure 17.

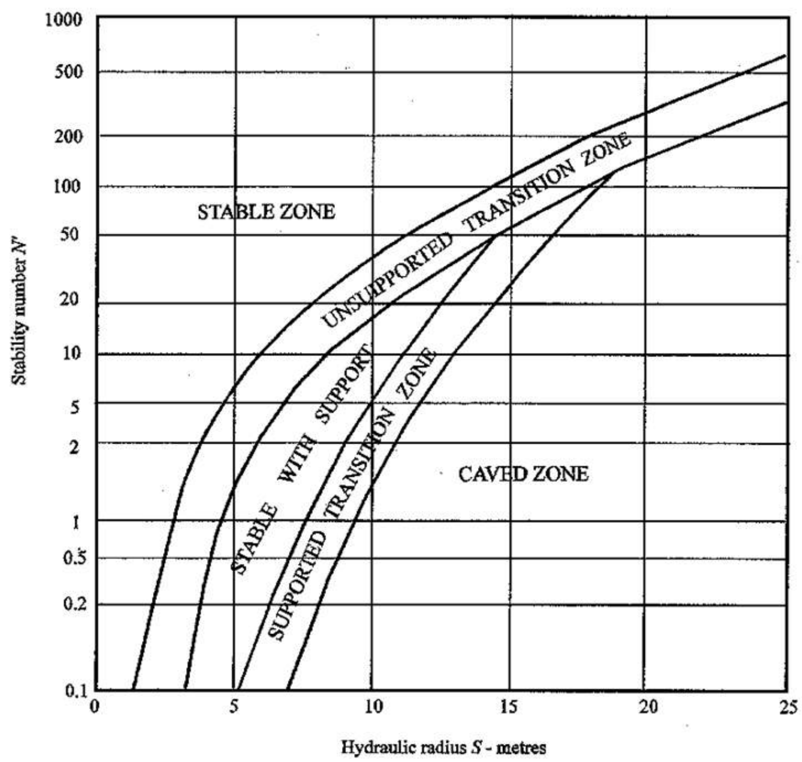


Figure 17: Stability Graph

On this graph the Stability Number (N' – vertical axis) represents the geomechanical data and is calculated as:

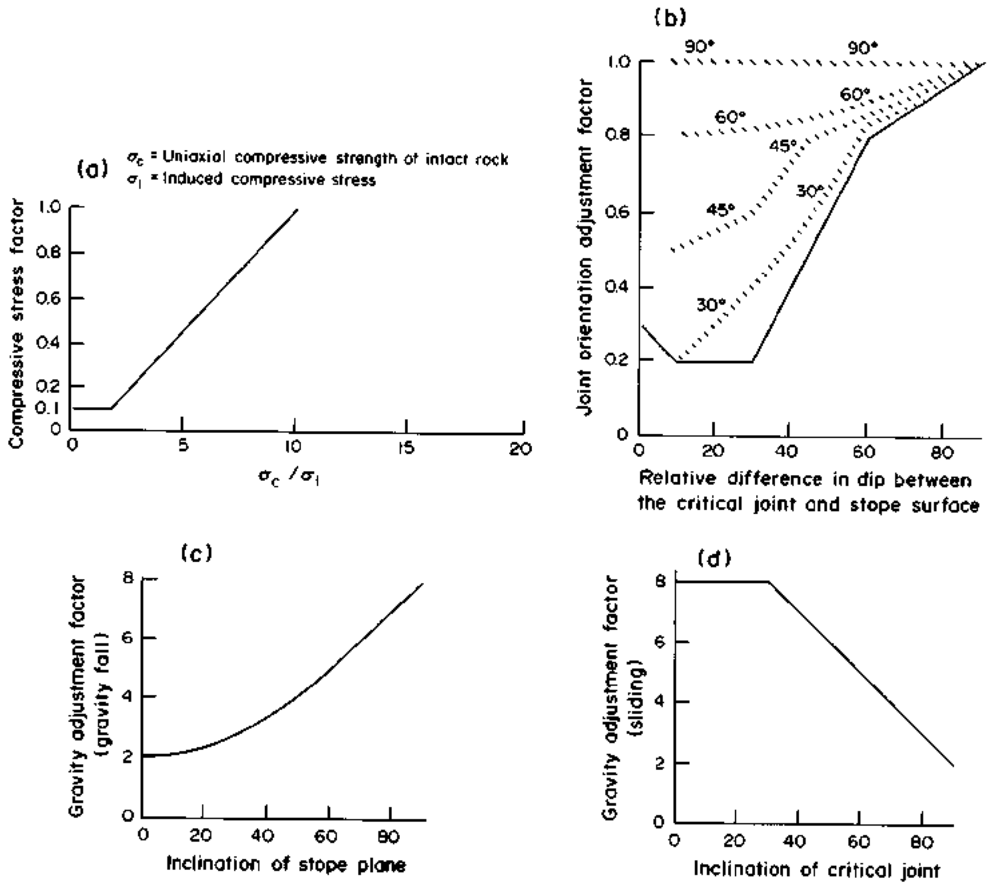
$$N' = Q'*A*B*C \qquad (1)$$

Q' is calculated from the assessed GSI values using:

$$GSI = 9\ln Q' + 44 \qquad (2)$$

Factors A, B, and C are derived based on the local mine induced stress, most critical structure and orientation of each individual wall based on the charts in Figure 18.

Inverting equation (2) gives a Q' range of ~ 10 – 30.



σ

Figure 19: Charts to determine Stability Graph Factors A, B and C

Factor A is controlled by the ratio of the Unconfined compressive strength (σ_c) against the maximum compressive stress (σ_1) at the center of the wall. Since no numerical modelling has been conducted σ_1 is estimated using a simple Kirsch assumption (i. e. $\sigma_{MAX} = 2\sigma_1$). Using the upper bound ISS assumption stope backs will experience excess compressive stress while stope wall stresses will reduce. Using the lower bound ISS assumption stope walls will experience excess compressive stress while stope back stresses will reduce. All stress values are based on a 450 m depth and a mineralized granodiorite UCS of 170 MPa. The relevant Factor A values (Figure 20) are:

- Upper bound stress conditions: A_{back} = 0.53; A_{wall} = 1.0.
- Lower bound conditions: A_{back} = 1.0; A_{wall} = 0.7.

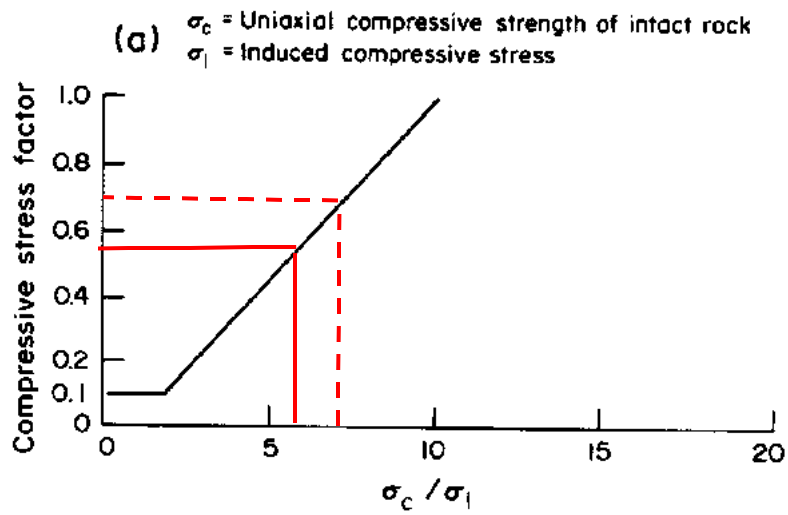


Figure 20: Factor A: UB stope back – solid red line: LB stope walls – dashed red line.

Factor B is strongly dependent on the specific wall orientation with respect to critical structures. Figure 7(b) presents the most comprehensive plot of potential structures in the mineralized granodiorite. All of the mapping and core logging data indicate one or more low angle (dip 15 – 30 degrees) joint sets. Factor B for flat stope backs is therefore taken as 0.2.

Stope walls are all assumed to be vertical. Many of the fracture sets identified are very steep (Figure 7) Subvertical joints (dip ≥ ~80°) will have minimal impact on these stope walls. Fracture sets with dip steeper than about 35° and strike within about 30° of the walls however can cause significant wall fall off (e. g. stope identifiers F, G and H in Figure 7b). The Stability Graph favors rectangular over square shapes. I therefore recommend making the stope long axis east – west as then the most problematic joint sets listed above are either perpendicular to these walls or intersect at an obliques angle that restricts wedge development. This gives a Factor B value of 1.0 for east-west walls.

The North – south end walls would only be impacted by joint set 7m (Figure 7b – identifier H (Table 4)). This would only impact the east wall however Figure 12 (Golder Photo 9) shows an approximately north-south fracture dipping steeply east that would impact the west wall. This joint orientation has not been picked up in either the adit mapping or the oriented core. The structural database, however, is still too sparse to preclude this as a potential joint set. Due to the uncertainty concerning the existence of an east dipping north-south joint set, a Factor B value of 0.6 is used for north-south end walls for this analysis.

Factor C = 2 for flat backs and 8 for vertical walls is used.

The resulting upper and lower bound N' values for (1) the horizontal maximum principle stress and (2) the gravitational stress field upper and lower bound Q' values are provided below.

- Maximum principle stress horizontal ($\sigma 1 = \sigma_H = 1.2\ \sigma_v$)
    - Back N' = 2 – 6
    - East – west walls N' = 80 - 240

  - North – south walls N' = 48 – 144
- Maximum principle stress – gravitational stress field ($\sigma 1 = \sigma_v = \Upsilon D = 0.027D$)
  - Back N' = 4 – 12
  - East – west walls N' = 56 – 168
  - North – south walls N' = 34 – 102

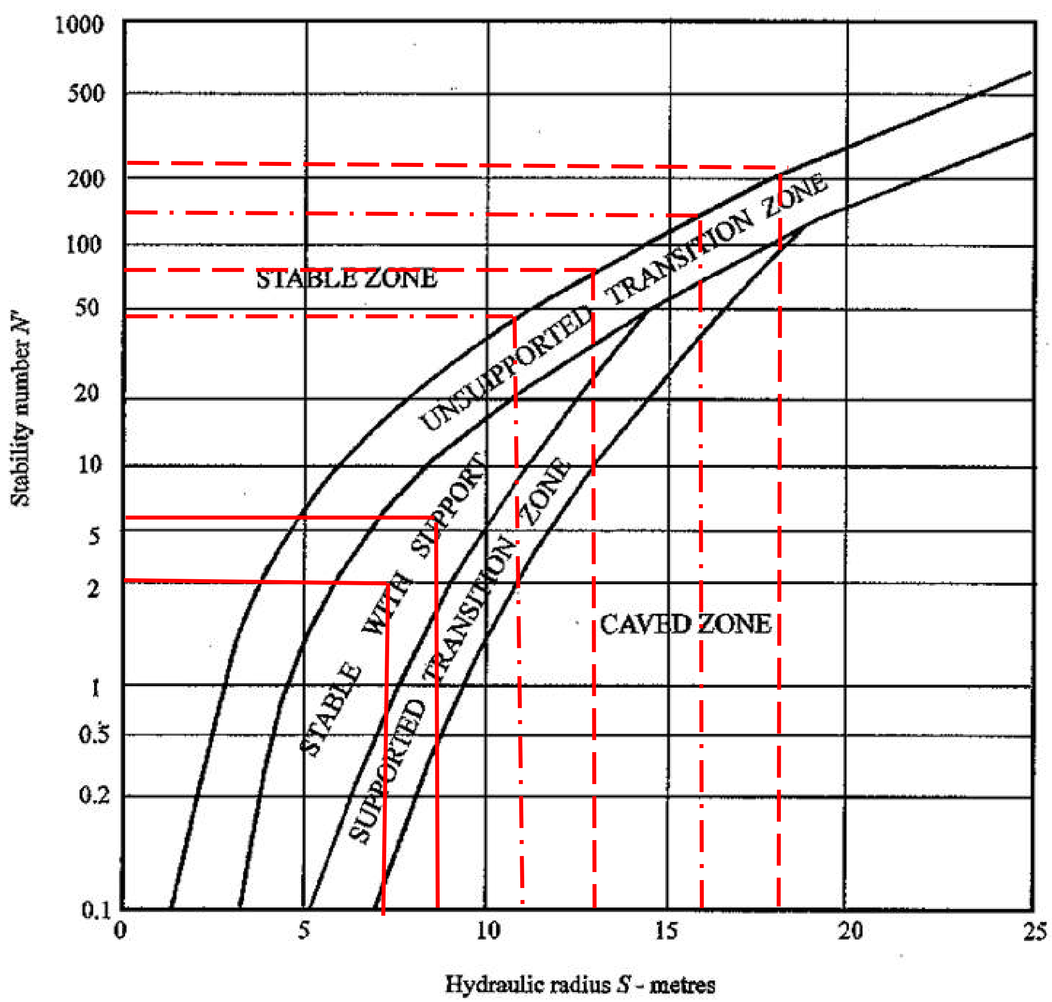


Figure 21: HR range for maximum principle stress horizontal: back ; east – west wall north – south wall

The resulting HR ranges are:

- Back 7.5 to 8.5
- East – west walls 13.5 to 18
- North – south walls 11 to 16.

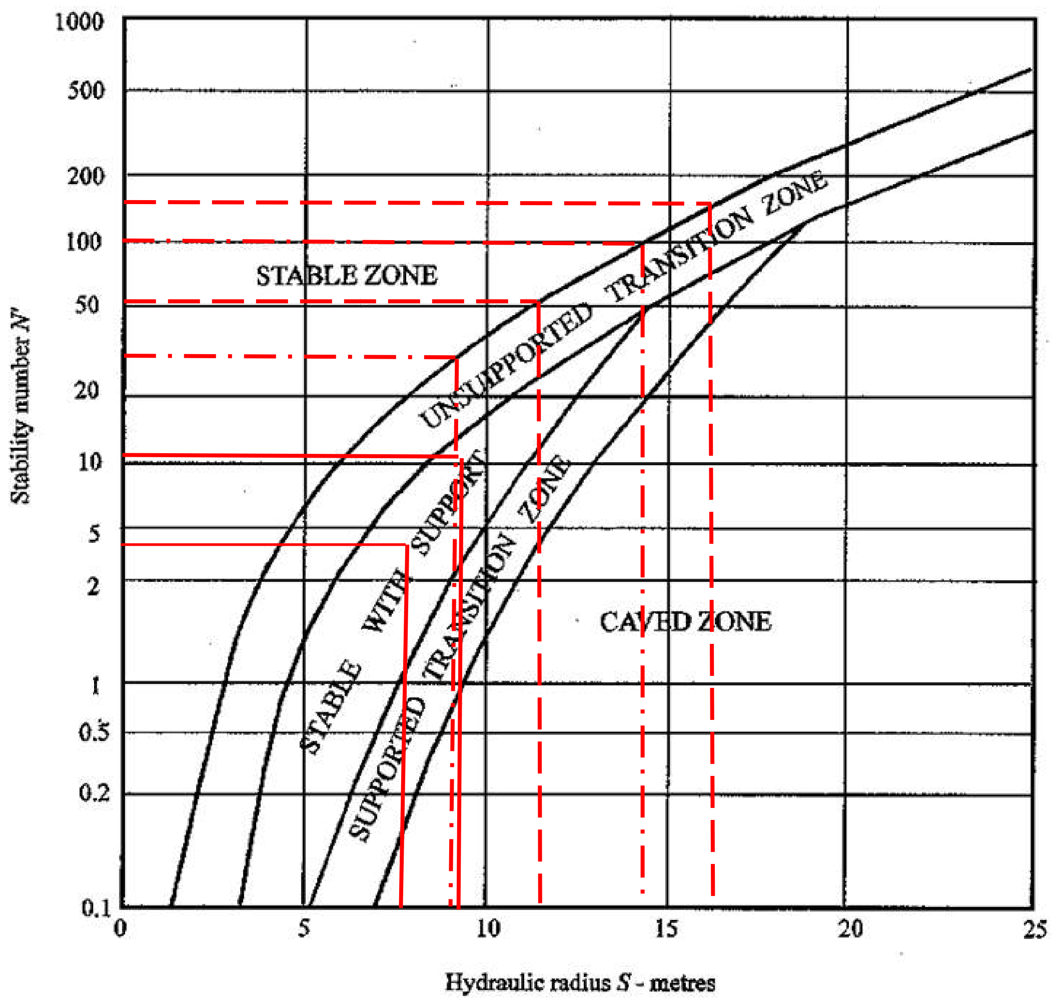


Figure 22: HR range for gravitational stress field: back ; east – west wall north – south wall

The resulting HR ranges are:

- Back 7.5 to 9
- East – west walls 11.5 to 16.5
- North – south walls 9 to 14.

For both far field ISS assumptions the back hydraulic radius estimates for the back assume back cable bolt support. At this point assume cable lengths of 1/3 span + 2m. Walls are all assumed to be unsupported. Considering the limited and highly uncertain nature of the existing geotechnical database I recommend using the lower bound HR estimates for both stress field assumptions. Should this project ultimately reach the production stage stope cable bolt support should be instrumented using SMARTcables to optimize cable bolt lengths for actual filed conditions.

For PEA estimating purposes the back HR values could be pushed to an absolute maximum at the leading edge of the Unsupported Transition Zone. Because of the significant limitations and uncertainties with the geotechnical data base however this would invoke significantly higher risk of potential stope back failure.

8. OTHER FACTORS

Secondary stopes or pillars, unless cut extremely thin, should not experience any overstress. Mine development should be stable assuming standard support in good condition. It is expected that all stopes will have to be backfilled in order to eliminate and potential surface deformations. Either cemented hydraulic backfill (CHF) or cemented paste backfill (CPB) could be used. I would strongly recommend the use of CPB as it has a number or significant operational benefits, including minimizing excess drainage and pumping costs plus loss of binder in CHF. There is additionally reduced risk of barricade failure although in both cases fill barricades have to be carefully engineered and monitored. The Davidson deposit lends itself to large stopes. This is beneficial since the fill rise rate will be relatively slow allowing the paste so setup such that I expect continuous pours with CPB to be possible. This provides a major cost advantage to the mining cycle. Further details on this is beyond the scope of this study.

9. CONCLUSIONS AND RECOMMENDATIONS

The orebody is hosted in a granodiorite, a strong stiff rock. The rock mass quality is good to very good (GSI = 65 TO 75). For the purpose of this study the orebody has been assumed to be dry due to lack of hydrogeological data.

The available data indicate that the rock mass is dissected by several joint sets (i. e. blocky). A statistical analysis of joint set densities is not possible with the existing data and thus joint set dominance cannot be determined. There are several steeply dipping joint sets but also at least two low angle dip sets (≤ 35°) and two or more sets dipping between ~40° and 80°.

The far field in situ stress state is unknown. Two possibilities are evaluated in this report: (i) a gravitational stress field and (ii) a stress field where the maximum principle stress is horizontal. There is limited field evidence suggesting that (i) is more probable (joint surface staining in the adit indicating water flow plus one striated fracture exhibiting water flow).

Analysis of the existing structural data indicates that stope backs will be exposed to numerous wedges and will require deep secondary (cable bolt) support. Vertical stope walls are assumed to be unsupported. Analysis of the structural data indicates that an east – west orientation is favored for the longest stope wall with end walls being north – south. Stope stability analysis was conducted using the empirical Stability Graph technique. Resulting hydraulic radius (HR) ranges for the two limiting stress conditions are:

Maximum principle stress horizontal	
Back	7.5 – 8.5
East – west walls	13.5 - 18
North – south walls	11 - 16
Gravitational stress field	
Back	7.5 - 9
East – west walls	11.5 – 16.5

North – south walls	9 - 14

Assume cable bolt length of 1/3 span + 2m. This provides a 2 m minimum anchorage above the assumed peak of the stress arch. Once mine production begins instrumentation can be used to fully optimize cable bolt length and spacing.

Secondary stopes or pillars, unless cut extremely thin, should not experience any overstress.

Stopes must be filled to prevent any possible surface deformation. Cemented paste backfill is recommended as the filling medium for operational efficiency and cost savings with the mine cycle.

In order to upgrade the geotechnical database to the Feasibility Study (FS) level an extensive geotechnical drilling program would be required with holes at various azimuths and dips around the full 360° range. If additional exploration development is done then line mapping in all development should be conducted in order to better quantify minimum and where possible maximum fracture set persistence. Drill holes should be minimum NQ size and should be drilled using oriented core drilling technique. Holes should then also be surveyed using either an optical or acoustic logging technique.

____

W. F. Bawden Ph.D., P. Eng.

President

10. REVIEWED LITERATURE AND REFERENCES

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APPENDIX 3.0 NEWFIELDS REPORT

NewFields
Perspective. Vision. Solutions.

680.240003.000

A-Z Mining Professionals Ltd.
781 Community Hall Road
Thunder Bay, ON P7G 1M6

March 27, 2024

ATTN: Mr. Brian LeBlanc, P.Eng.

**RE: PRELIMINARY ECONOMIC ASSESSMENT DESIGN
FILTERED TAILINGS STORAGE FACILITY
DAVIDSON MOLYBDENUM PROJECT, SMITHERS, BC, CANADA**

Dear Brian,

Moon River Capital (MRC) retained A-Z Mining Professionals Ltd. (AMPL) to complete a Preliminary Economic Assessment (PEA) for the Davidson Molybdenum Project (Davidson). AMPL retained NewFields Canada Mining & Environment ULC (NewFields) to complete a PEA level design of a filtered tailings storage facility (FTSF) for the project.

The purpose of this technical memorandum is to summarize the details of the FTSF design, including preliminary layouts and cross-sections of the FTSF, water balance, estimation of construction material quantities, and a preliminary construction cost estimate.

1. BACKGROUND

1.1. Site and Project Description

Based on information provided to NewFields, the project includes development of a mine with a mineral resource of approximately 25 million tonnes (Mt). The project is located near Smithers, British Columbia and is accessible by road, with electrical infrastructure nearby.

Preliminary plans call for the FTSF to be developed to the south of Hudson Bay Mountain, approximately 10 km to the west of Smithers.

1.2. Objectives and Scope of Work

NewFields' scope of work for the PEA included the following tasks:

- Collect and review relevant available data;
- PEA level design of the FTSF, including site plan and typical cross sections;
- Material take-offs (MTOs) and construction cost estimates related to the FTSF at a PEA level; and,

- Preparation of the PEA technical memorandum.

2. FILTERED TAILINGS STORAGE FACILITY PEA DESIGN

2.1. Design Basis and Assumptions

It is understood that the FTSF will store filtered tailings from processing of the ore. The FTSF will be located south of Hudson Bay Mountain and will consist of a side hill impoundment (Figure A-1). Perimeter earthfill embankments will be constructed to provide containment around the north and east perimeters of the facility. Natural topography will provide containment along the west and south perimeters of the facility.

Based on current estimates, the FTSF will store a total of approximately 25 Mt of filtered tailings. It is not known if the tailings will be potentially acid generating (PAG). As a result, at this stage, it is expected that the facility will be fully lined using a 60 mil (1.5mm) thick Linear Low Density Polyethylene (LLDPE) liner system with a sand bedding layer below the liner. Perimeter seepage and runoff collection ditches will be provided to collect any seepage and runoff from the facility.

For preliminary design of the FTSF, assumptions and considerations included:

- Dry density of filtered tailings (in place): 1.5 tonnes per cubic metre (t/m^3), resulting in a total filtered tailings storage volume of approximately 16.7 million cubic metres (Mm^3);
- Filtered tailings may be PAG and will be contained in a fully lined storage facility. The facility will be contained by a combination of earthfill dams and natural topography;
- The FTSF will be developed in 2 cells, with the first cell providing containment for approximately 50 percent of the total life-of-mine tailings. The perimeter dams will have an average height of approximately 6 metres (m) and will provide containment of runoff from the surface of the facility. The maximum FTSF stack height will be approximately 30 m;
- FTSF dyke interior and exterior slopes will be 3 horizontal to 1 vertical (H:V) with a 6 m crest width;
- Contact water will be collected in the facility and either returned to the process plant for reuse in milling, or treated and discharged to the environment;
- Surface water diversion channels (SWDC) will be constructed to divert runoff around the FTSF;
- The top surface of the FTSF will be graded to promote surface water drainage; and,
- Closure of the FTSF will include installation of a liner and waste rock cover over the filtered tailings.

2.2. FTSF Design

The general layout of the FTSF was designed to accommodate the estimated storage volumes and satisfy the project constraints described in Section 2.1. The FTSF will have a final top elevation approximately 910 metres above sea level (masl) and a footprint of approximately 75 hectares (ha). The liner system will include a prepared foundation (cleared, grubbed and stripped of all organic and unsuitable materials), a liner bedding layer (0.3m thick) of compacted sand and a 60 mill (1.5mm) LLDPE geomembrane liner.

Closure of the FTSF would include decommissioning of the seepage collection and diversion system, installation of a synthetic liner and waste rock or soil cap on top of the FTSF to provide long-term physical stability.

Conceptual sketches of the FTSF in plan and cross section are presented in Figures A2 and A3 for Phase 1 development and in Figures A4 and A5 for Phase 2 development. A summary of design details related to the FTSF are presented in Table 1.

Table 1: FTSF Design Summary

Design Component	Volume (m^3)	Footprint (m^2)	Elevation (masl)	Crest Width (m)	Upstream Slope	Downstream Slope
FTSF Storage Volume	16,700,000	750,000	910	--	3H:1V	3H:1V
FTSF Containment Dam	540,000	--	890	6	1.5H:1V	3H:1V

From the bottom to top, the liner system will be composed of prepared subgrade, 0.3 m of bedding sand/gravel and synthetic liner.

2.3. Material Quantities and Cost Estimates

The estimated material quantities and estimated costs for the construction of the FTSF are presented in Table 2. The estimate is broken down by phase of construction of the facility.

The capital and operating costs were estimated based on unit rates in NewFields' cost database for similar projects in similar locales and adjusted based on NewFields' recent project experience. The PEA quantity estimates were based on the 3-dimensional (3D) models of the FTSF developed for the project using topographic data obtained from public sources. Stripping volumes were estimated based on footprint areas for topsoil stripping and stockpiling. It is noted that the PEA estimate includes costs for stripping and stockpiling topsoil from the entire FTSF footprint to be used as soil cover for closure.

Cost estimates do not include mobilization, construction QA/QC, or other engineering or administrative costs.

Table 2. Material Quantities and Cost Estimate Summary

ITEM	DESCRIPTION	UNIT	UNIT RATE	QUANTITY	ESTIMATED COST (CAD)	
1.00	**SITE PREPARATION**					
1.01	Mobilization	EST	7%		$	1,812,964
1.02	Demobilization	EST	3%		$	776,984
1.03	Storm Water and Sediment Management	EST	2%		$	381,827
				Site Preparation Subtotal	***$***	***2,971,776***
2.00	**FTSF EARTHWORKS - Stage 1**					
2.01	Clearing and Grubbing	SM	$ 0.26	282,000	$	73,320
2.02	Topsoil Stripping - Excavate, Haul, and Stockpile	CM	$ 5.49	141,000	$	774,090
2.03	Prepared Sand Till - Scarify, Moisture Condition, and Compact	SM	$ 0.94	282,000	$	265,080
2.04	Earthfill - Load, Haul, and Place	CM	$ 7.20	324,000	$	2,332,800
2.07	Sand Bedding - Process, Load, Haul, and Place	CM	$ 13.35	84,600	$	1,129,410
2.08	Anchor Trench - Excavate and Backfill	M	$ 39.41	2,250	$	88,673
				Stage 1 Earthworks Subtotal	***$***	***4,663,373***
3.00	**FTSF EARTHWORKS - Stage 2**					
3.01	Clearing and Grubbing	SM	$ 0.26	282,000	$	73,320
3.02	Topsoil Stripping - Excavate, Haul, and Stockpile	CM	$ 5.49	141,000	$	774,090
3.03	Prepared Sand Till - Scarify, Moisture Condition, and Compact	SM	$ 0.94	282,000	$	265,080
3.04	Earthfill - Load, Haul, and Place	CM	$ 7.20	216,000	$	1,555,200
3.07	Sand Bedding - Process, Load, Haul, and Place	CM	$ 13.35	84,600	$	1,129,410
3.08	Anchor Trench - Excavate and Backfill	M	$ 39.41	1,500	$	59,115
				Stage 2 Earthworks Subtotal	***$***	***3,856,215***
4.00	**GEOSYNTHETICS (Includes Ponds) - Stage 1**					
4.01	60mil LLDPE Double Sided Textured Geomembrane - Supply (10% Allowance for Waste and Overlap)	SM	$ 8.82	302,000	$	2,663,640
4.02	60mil LLDPE Double Sided Textured Geomembrane - Install	SM	$ 2.45	302,000	$	740,935
				Stage 1 Geosynthetics Subtotal	***$***	***3,404,575***
5.00	**GEOSYNTHETICS (Includes Ponds) - Stage 2**					
5.01	60mil LLDPE Double Sided Textured Geomembrane - Supply (10% Allowance for Waste and Overlap)	SM	$ 8.82	302,000	$	2,663,640
5.02	60mil LLDPE Double Sided Textured Geomembrane - Install	SM	$ 2.45	302,000	$	739,900
				Stage 2 Geosynthetics Subtotal	***$***	***3,403,540***
6.00	**CLOSURE COVER**					
6.01	60mil LLDPE Double Sided Textured Geomembrane - Supply (10% Allowance for Waste and Overlap)	SM	$ 8.82	604,000	$	5,327,280
6.02	60mil LLDPE Double Sided Textured Geomembrane - Install	SM	$ 2.45	604,000	$	1,479,800
6.05	Sand Cover - Process, Load, Haul, and Place	CM	$ 13.35	282,000	$	3,764,700
				Closure Cover Subtotal	***$***	***10,571,780***
				DIRECT CONSTRUCTION COST	***$***	***28,871,258***
				DIRECT CONSTRUCTION COST/CM TAILINGS	***$***	***1.15***
8.00	Contingency	EST	15%		***$***	***4,330,689***
				DIRECT CONSTRUCTION COST+ CONTINGENCY	***$***	***33,201,946***
9.00	**INDIRECTS**					
9.01	Engineering	EST	8%		$	2,656,156
9.02	Construction Management	EST	7%		$	2,324,136
9.03	QA/QC	EST	5%		$	1,660,097
9.04	Third-Party Surveying	EST	3%		$	996,058
				Subtotal for Indirects	***$***	***7,636,448***
				TOTAL COST	**$**	**40,838,394**
				TOTAL COST/TONNE TAILINGS	**$**	**1.63**
10.00	**OPERATING COST**					
10.01	Filtered Tailings - Load, Haul, and Place	t	$ 4.48	25,000,000	$	112,000,000
				TOTAL OPERATING COST	**$**	**112,000,000**
				TOTAL OPERATING COST/TONNE TAILINGS	**$**	**4.48**

3. LIMITATIONS

NewFields has prepared this document in a manner consistent with the level of care and skill ordinarily exercised by the engineering and geoscience professions practicing in similar conditions within the jurisdiction that the services are provided, subject to time limits and physical constraints applicable to this work. No other warranty, express or implied, is made.

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4. CLOSURE

We trust that the information contained within this document fulfills your requirements at this time. We appreciate the opportunity to work with AMPL on this project. If you require additional information, please do not hesitate to contact the undersigned.

Best Regards,

NewFields Canada Mining & Environment ULC

Prepared By:

Leon Botham, MSCE, P.Eng.
Principal Engineer

LCB/lcb

Attachments:
A - Figures

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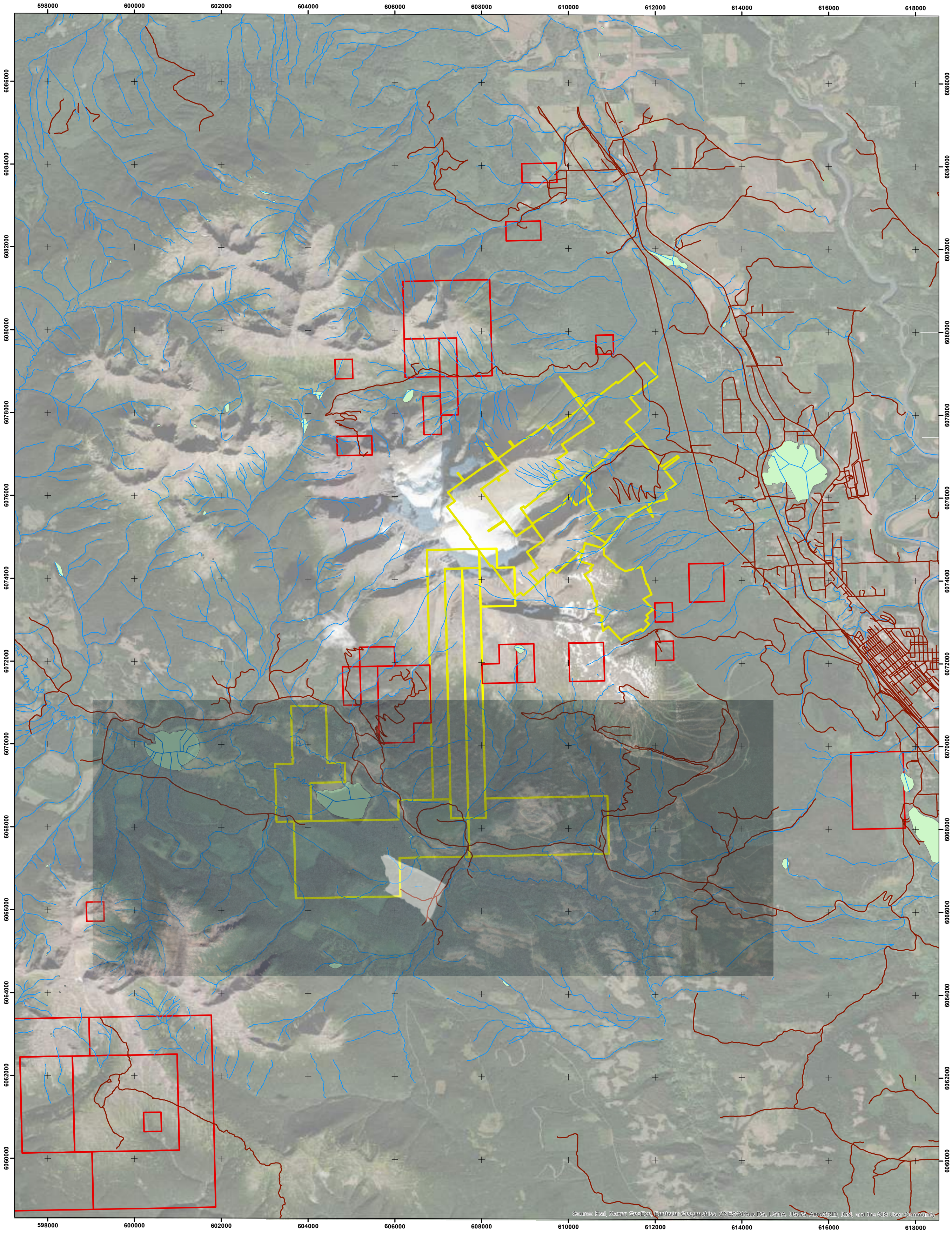
Attachment A

Figures



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Legend

- TSF
- Davidson, Donald Alexander
- Roda Holdings Inc.
- Water Channel
- Roads
- Other Property

NewFields

CLIENT
A-Z Mining Professionals Ltd.

PROJECT
Davidson Molybdenum Project

TITLE
Tailings Storage Facility - Location

FILENAME

FIGURE NO. A-1

REVISION A

NewFields

CLIENT
A-Z Mining Professionals Ltd.

PROJECT
Davidson Molybdenum Project

TITLE
TSF - Stage 1 Plan View

FILENAME

FIGURE NO.
A-2

REVISION
A

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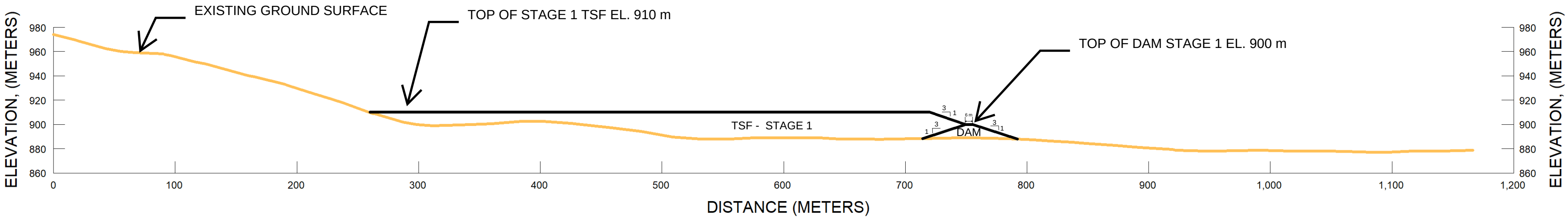
SECTION A-A



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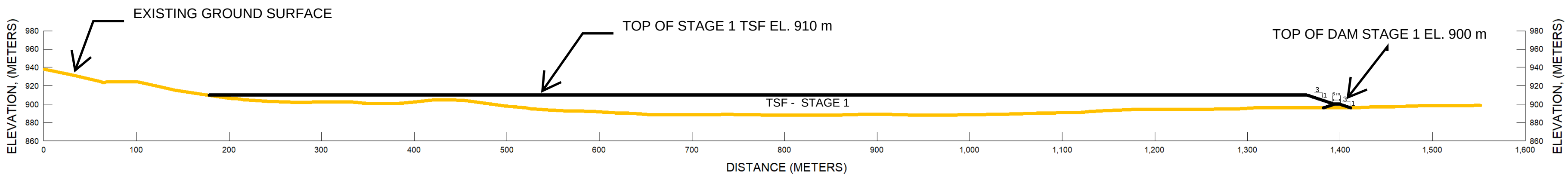

NewFields

CLIENT
A-Z Mining Professionals Ltd.

PROJECT
Davidson Molybdenum Project

TITLE
TSF - Stage 1 Cross Sections

FILENAME

FIGURE NO.
A-3

REVISION
A

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NewFields

CLIENT
A-Z Mining Professionals Ltd.

PROJECT
Davidson Molybdenum Project

TITLE
TSF - Stage 2 Plan View

FILENAME

FIGURE NO.
A-4

REVISION
A

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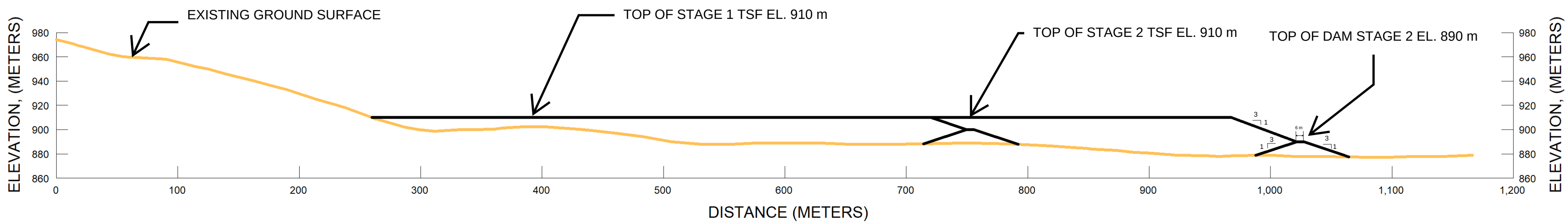
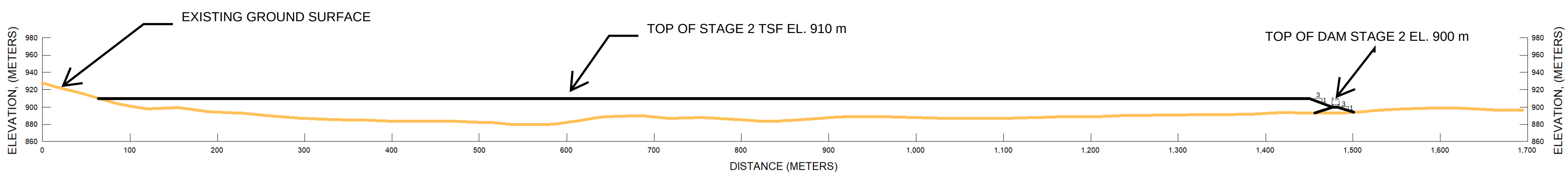
SECTION C-C



SECTION D-D



NewFields	CLIENT A-Z Mining Professionals Ltd.
PROJECT Davidson Molybdenum Project	
TITLE TSF - Stage 2 Cross Sections	FILENAME
	FIGURE NO. A-5 / REVISION A

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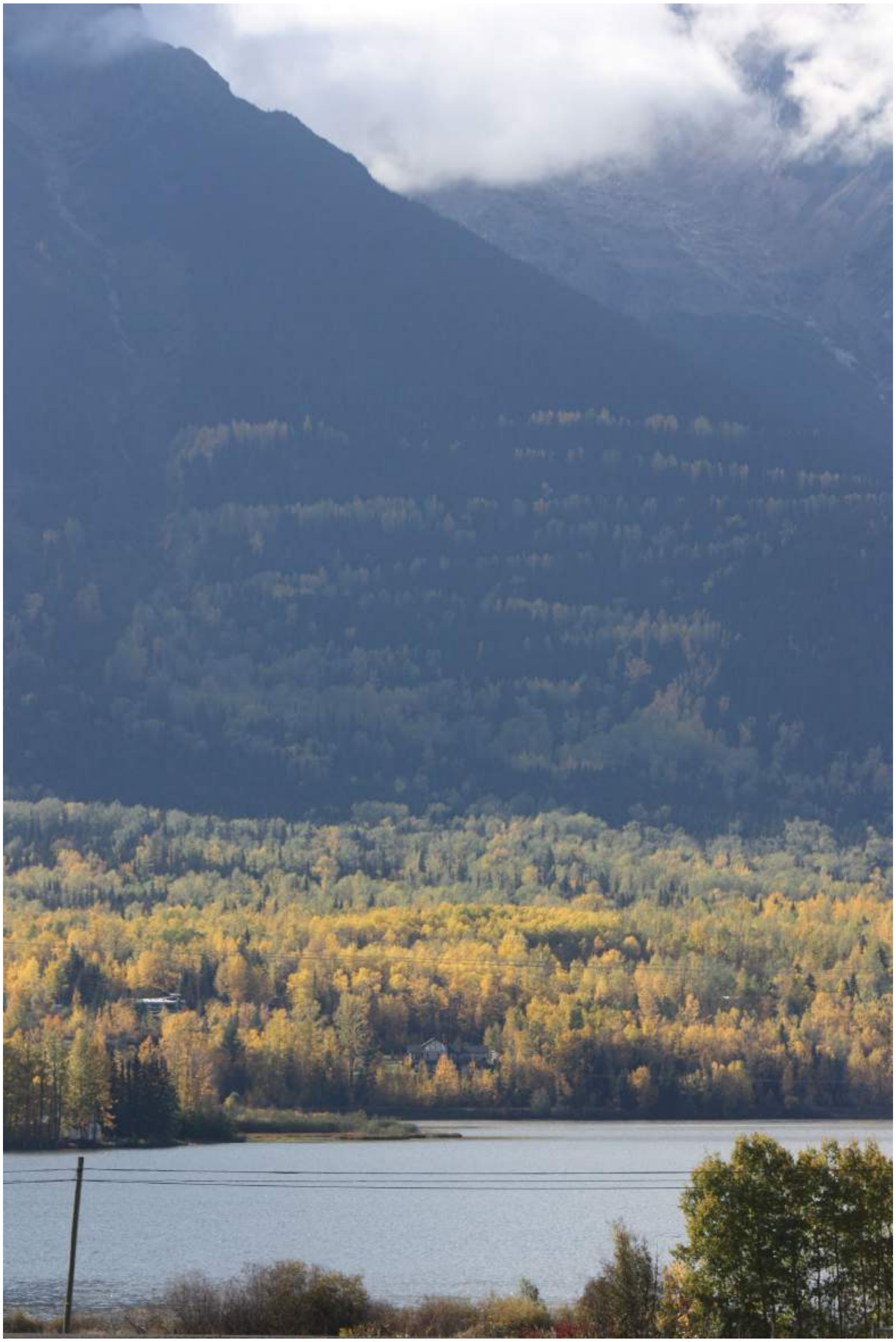
APPENDIX 4.0 SITE VISIT

October 5, 2023

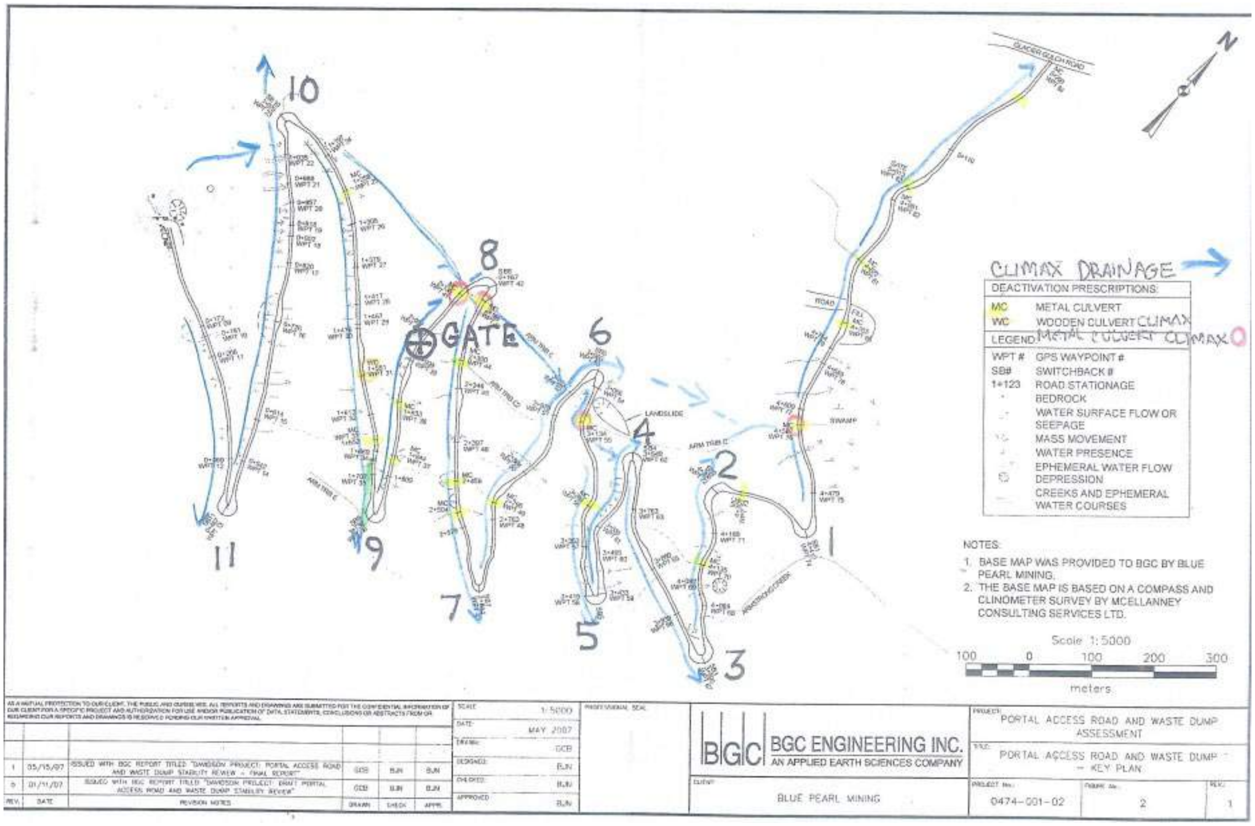
On September 27 and 28, 2023 I visited the site of the Davidson Project in Smithers, B.C. Canada. On September 27, I met with Mr. Donald Davidson at the site of his office/warehouse/core storage facility near the Smithers airport. Shortly afterwards we left to meet Mr. Scott Rowsell of Pro-Tech Forest Resources Ltd. Mr. Rowsell had a side by side 4 wheeler to make the trip up to the old portal. The week previous to my visit Mr. Rowsell had spent half a day cutting access through fallen trees along the roadway. The road was overgrown in most areas and switch-backed up the mountain side.

The roadway can be seen by the yellow tops of the deciduous trees



This view is from the Smithers Airport parking lot

There were 3 major washouts along the roadway which appear to have been caused by culvert installations made in the past 15 years.



Original Climax Drainage Schematic

Mr. Davidson says there were no issues with the roadway until the drainage plan was changed by McElhanney Engineering and culverts were installed to drain the ditches from one level to the next on the switchbacks. The original plan had all the ditching on the upstream side of the road and discharging past the end of the switchback.

The current ditches are quite shallow as they have filled in over the years with detritus from the mountain side and the vegetation and should be re-established.

Washouts along the road

In one area the road was only wide enough for the 4 wheeler to make it past a landslide.

Narrow 4 wheeler access only, large pile of sloughed material on either side of roadway

The portal has been blocked off by rock that was trucked in for the purpose. This rock shows visible signs of staining. (ARD)

Portal, rocks show signs of Fe staining

The pad area up near the portal was constructed mostly of mine waste from the tunnel excavations in the 1960's. This rock does not show any signs of staining. There is very little land area up by the portal, enough to stage a small development project, but not enough to stage a mining operation. The following photo was taken from the same spot as the first portal photo.

Flat staging area by Portal

Edge of waste dump, no signs of staining

Following the trip up to the portal we returned to the office area and started going through some of the extensive literature on past studies. I initially identified four reports that I felt would be worth scanning so as to have electronic copies. Mr. Rowsell took two of the reports to be scanned at Pro-Tech and the other two were brought to a facility in downtown Smithers, Mills Office Productivity.

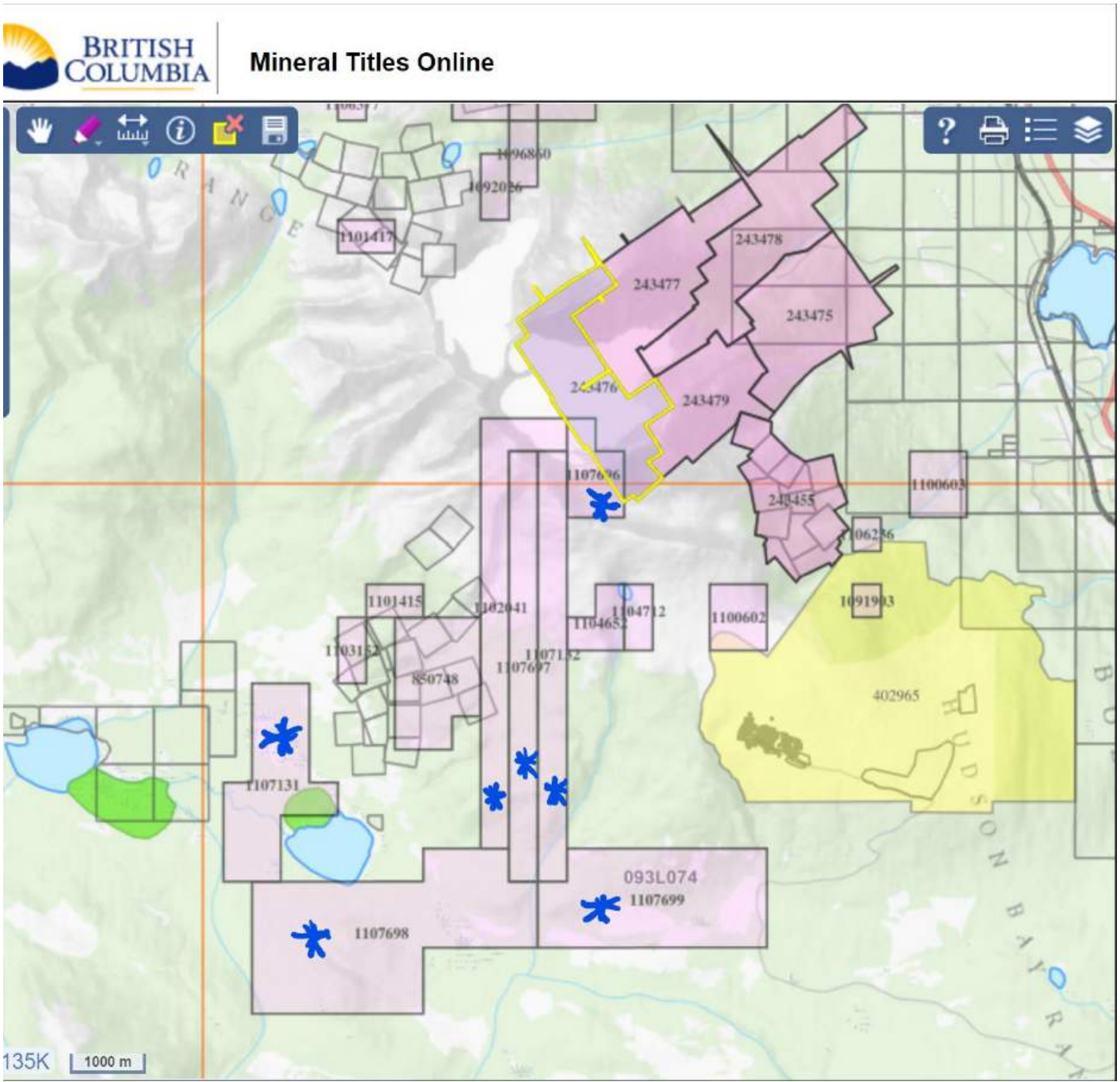
After dropping off the reports we drove around to the back side of Hudson's Bay Mountain in Mr. Rowsell's truck. We stopped at a gravel pit on the south slope and drove along the main logging haul road and into the bottom of the Duthie Mine rehabilitation area. Access off the main haul road was limited as many of the roads Mr. Davidson wanted to travel were badly overgrown. The area just north of the haul road is a gently sloping table land before the steep slopes of the mountain. We then proceeded south on another logging road to get a better view of the overall area.

Duthie Mine Waste Dump

Looking north-west towards the southern flank of Hudson's Bay Mountain

On September 28, 2023 I again met with Mr. Davidson at his office/warehouse/core storage facility. His daughter Trish was also there. We went through a large part of his library of study material and I identified another 6 reports for scanning. I brought them to Mills Office Productivity for scanning. I was able to get copies of all the scanned documents before I had to leave for the airport.

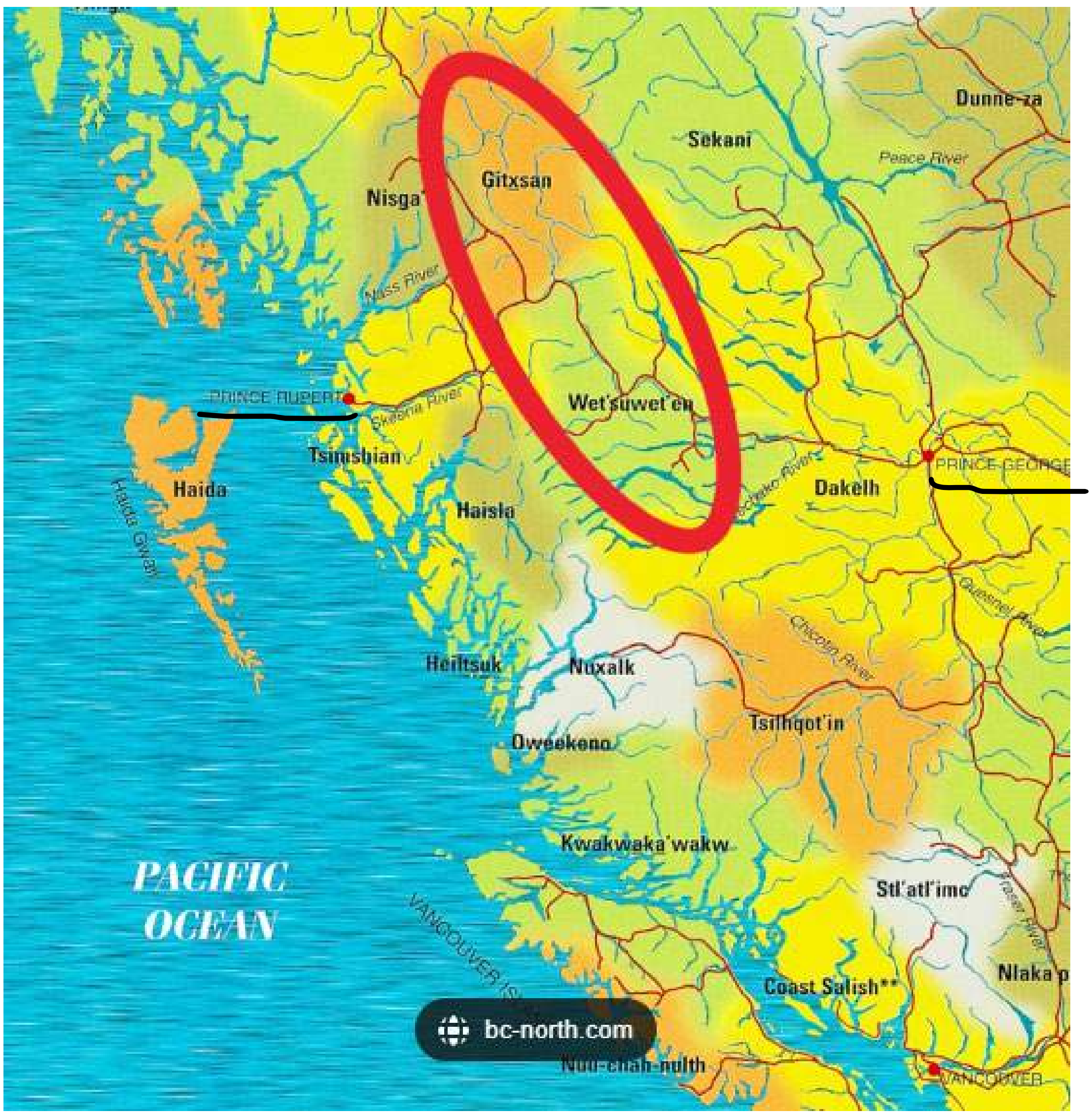
During the morning we went through a number of topo maps looking at the southern flank of the mountain. Mr. Davidson had identified an area where it would be possible to collar a portal. We had not been able to access that area on the previous day. Mr. Davidson had previously staked a narrow corridor to provide access from the main logging road to the main claim block, Claim # 1102041. This claim block only adjoined the main claim block at one narrow point of contact. He had done some further staking with another narrow corridor to the east of the first one, Claim # 1107132, and with a claim block to the west of Aldrich Lake, claim 1107131, which he thought would be suitable for a tailings pond.

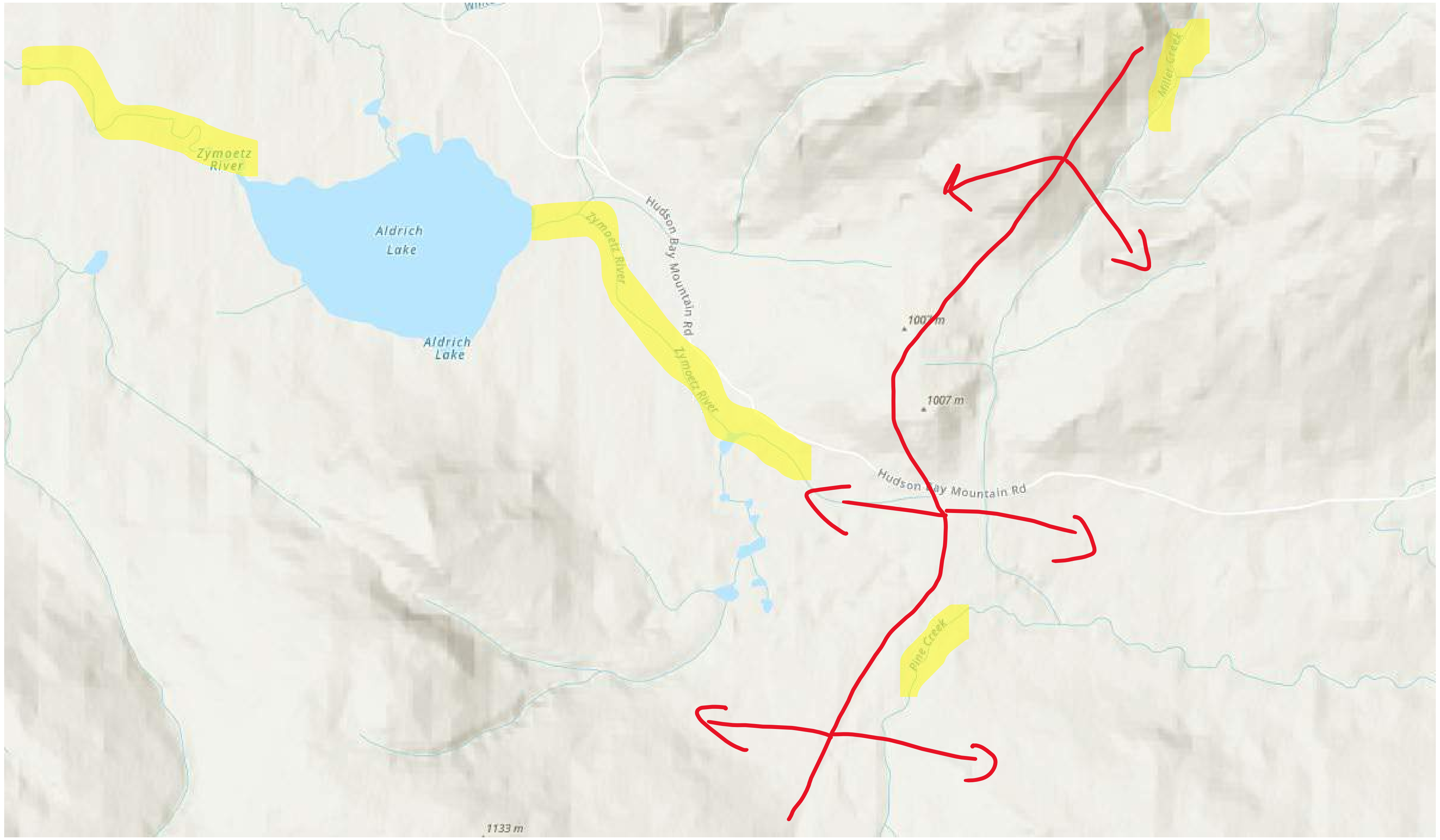


I felt there would be insufficient ground for the size of the project, for both development waste and tailings disposal and especially at the narrow access point. I identified additional claims that I felt should be staked before someone else staked them. I phoned Paul to discuss and he told me to proceed. Mr. Davidson subsequently staked the additional claims. Claim 1107696 expands the access point to the main claim block and should allow good access for whatever access drift will be required to be driven. Claim block 1107697 fills in the space between to other two narrow claims.

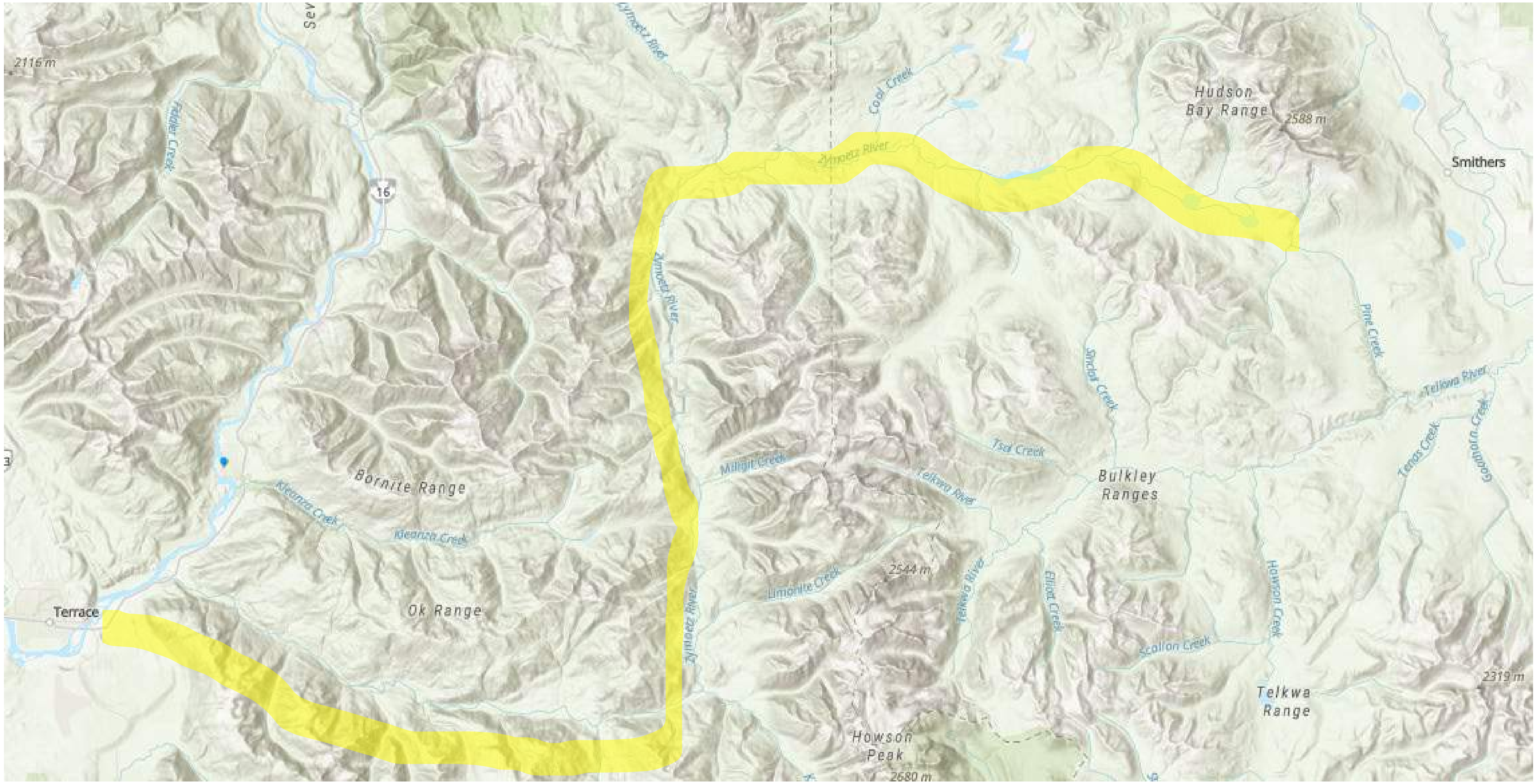
One of the main issues that will arise is the watershed. Just below the staked area there is a height of land with the water reporting to two different watersheds. To the east is Miller Creek which is directly below the narrow access blocks and which reports to Pine Creek which reports to the Telkwa River. This is a sensitive area as there are many communities along the Telkwa. Claim 1107699 is in this watershed. There are large clear-cut areas in this block which would be suitable for development waste disposal as there does not appear to be any issues with ARD. This area is in the traditional territory of the Wet'suwet'en First Nation.

West of Miller Creek is a height of land and the watershed reports to the Zymoetz River which flows out of Aldrich and Dennis Lakes. Claim 1107698 is in the Zymoetz River watershed. There are no communities along this watershed until Terrace, some 115 km away. This area is also fairly flat and has been clear-cut in several areas. This area was chosen as a possible dry stacked tailings disposal area. The Zymoetz River runs through the traditional territory of the Gitxsan First Nation.





Zymoetz River/Telkwa River Watersheds



Zymoetz River: Aldrich Lake to Terrace BC

In the afternoon we continued digging through the files and identifying useful material. A great deal of work was done by Rescan on the environmental requirements for the original Feasibility Study by Hatch. Unfortunately most of this material is "stale dated" and will have to be redone. Also, most of it was for the front side of Hudson's Bay Mountain. The back side of the mountain may host different flora and fauna and have other environmental issues. There were 37 volumes of environmental reports, which should be reviewed by our Environmental Associate. Some of these may still be useful, such as hydrology reports, ethnohistorical reports and ARD potential. However, most of the work will have to be redone before the Project could proceed to a Feasibility Study.

A minimum of two years worth of Environmental Baseline Data collection will be required, further soil geochemistry, evaluation of possible endangered species and plants, an archaeological study, groundwater studies and an Environmental Management Plan are a few of the studies that will need to be done. A Closure Plan will need to be developed. Also, a Memorandum of Understanding will need to be reached with the local First Nations. Everything needs to be in place before completing a Feasibility Study. A "Road Map" detailing all requirements, timelines and estimated costs for this work will be included in the pricing for the PEA.

Recommendations:

It is recommended that consideration be given to engaging a biologist to begin the baseline studies in the spring of 2024 as a minimum of two spring freshets and two fall migration seasons will need to be done.

The current portal will serve as both a ventilation source/outlet and as a second egress from the mine. The road and the drainage system will need to be repaired and the portal area rehabilitated.

Begin consultation with the local First Nations. Meetings with the local communities should also be considered. Moon River should also consider engaging someone local as the company representative.

Brian LeBlanc, P. Eng.
President
A-Z Mining Professionals Ltd.