Preliminary Economic Assessment
Pickett Mountain Project

Penobscot County, Maine, USA

68.468°W Longitude and 46.134°N Latitude

Effective Date: September 14, 2020

Filing Date: October 29, 2020

Prepared For

WOLFDEN
Wolfden Resources Corporation

By

A-Z Mining Professionals Limited

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1.0 Executive Summary

The Pickett Mountain Property area has been explored from the 1950s to present day. Getty Oil made the first significant discovery in the early 1980s. The property was sold to Chevron, which employed the polygonal resource calculation method and stated the historic figure of 3.15 million short tons of resource with an estimated grade of 9.66% zinc, 4.30% lead, 1.24% copper, 0.029 opt gold, and 2.96 opt silver (Laverty, 1983; Riddell, 1983).

The polymetallic deposit is separated into two main zones: the West Zone and the East Zone. The focus of mining will be zinc. The West Zone is narrow but high grade while the East Zone is broader and has a larger volume with a large portion of it currently sub-economic.

1.1 Resources and Reserves

Wolfden Resources (Wolfden) acquired the property in 2017, through its subsidiary Wolfden Mt. Chase LLC, with the goal of conducting advanced exploration to further the property toward an economic assessment. With the addition of the Wolfden drilling between 2017 and October of 2018, the estimate of the mineral resources, as of January 2019 using a 9% ZnEq cut-off grade, are shown in Table 1.1, below [**Note:** Historical resources are reported in tons and ounces per ton; current resources are reported as tonnes and grams per tonnes].

TABLE 1.1
MINERAL RESOURCE STATEMENT – JANUARY 2019 WITH 9% ZNEQ CUT-OFF

Category	Tonnes	% Zn	% Pb	% Cu	g/t Ag	g/t Au	Density	% ZnEq
Indicated	2,050,000	9.88	3.93	1.38	101.58	0.92	3.99	19.32
Inferred	2,030,000	10.98	4.35	1.2	111.45	0.92	4	20.61

For the purposes of this document, mineral resources were updated using a 7% Cut-off grade shell. The updated estimate is shown in Table 1.2, below. Note, no other information was included or deducted from the estimate and the same methodology as those used in 2019 were applied.

TABLE 1.2
MINERAL RESOURCE STATEMENT – UPDATED SEPTEMBER 2020 WITH 7% ZNEQ CUT-OFF

Category	Tonnes	% Zn	% Pb	% Cu	g/t Ag	g/t Au	Density	% ZnEq
Indicated	2,177,000	9.25	3.68	1.32	96.4	0.9	3.98	18.23
Inferred	2,294,000	9.79	3.88	1.15	101.1	0.9	3.99	18.62

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 and cost projections are to PEA level of accuracy. Therefore, there is no guarantee that the economic projections contained in this Preliminary Economic Assessment would be realized.

The Pickett Mountain deposit, as currently defined to a depth of 875m, has significant infill and expansion opportunities. The local exploration target expansion range is 6 to 10 million tonnes grading 12% to 20% ZnEq, based on the current geological model, without the addition of other lenses. This target size is derived from the interpretation of the drilling, geological structure, geology, and surface sampling carried out on the property to date. The potential quantity and grade of the target is conceptual in nature. There has been insufficient exploration of this target to define a Mineral Resource and it is uncertain if further local exploration will result in this target being delineated as a Mineral Resource.

There are no reserves for the project at present.

1.2 Project Operating Plan

The focus of the project has been to get the regulatory issues resolved while conducting the engineering components. To that end, the approval for the project is currently in Month Six of a six- and one-half month stage of an overall 13½ month rezoning process for the permitting of the mine and mine infrastructure. The project is filing its findings and plan to the Land Use Planning Commission (LUPC), which will certify to the Maine Department of Environmental Protection (MDEP) that it will meet or exceed all requirements in protection of the environment. Underground production rates have been set at 1,300 tpd for all underground mining operations. Initially mining will begin on the upper West Zone.

The 1,200 tpd processing plant will be developed and commissioned, which is expected to take approximately one year. Initially, plant feed will originate from the West Zone and then the East Zone.

1.3 Mine Plan

The bulk of the high value material sits in narrow veins, which is conducive to raise platform mining. The mine will be accessed from the surface by a decline collared in the hillside. An excavation will be cut into the hillside to create a large free face in which to collar the portal. The floor of the cut and the first two rounds of the decline will be designed to be upgrade at a sufficient grade to allow water to drain away from the portal. The decline will be driven at a nominal 15% downgrade and access the upper sections of the ore zone at 100m intervals.

Initially, waste rock and mineralised rock will be trucked to the surface via low profile haul trucks. To hoist the mineralised material during commercial production, a small vertical shaft will be developed from the underground to create an opening such that a 10-tonne skip and counterweight can be employed to move the ore to surface (Figure 1.1).

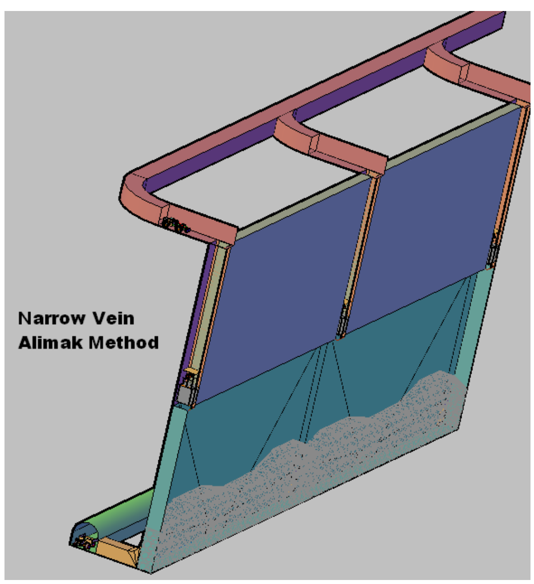


Figure 1.1 Narrow Vein Mining Using Alimak Raise Platforms

1.4 Processing

Pickett mountain potentially economic mineralisation will be processed in a conventional sequential process flowsheet for recovery of copper, lead, and zinc concentrates.

The ore will be processed in a three-stage crushing circuit followed by a ball mill to produce a ground product with a P_{80} of 37 micrometers. The slurry will be conditioned with chemicals and a rougher copper concentrate floated. The concentrate will be reground to P_{80} of 21 micrometers and cleaned several times to produce a marketable-grade copper concentrate.

The tailings from rougher and first cleaner flotation will be combined as feed to the lead circuit. Lead rougher concentrate will be floated following conditioning of the slurry with reagents. The lead rougher concentrate will be reground to P_{80} of 14 micrometers and cleaned several times to produce a marketable lead concentrate.

The lead rougher and first cleaner flotation tailing will be combined as feed to the zinc circuit. The slurry will be conditioned with reagents and rougher zinc concentrate will be floated. The concentrate will be reground to P_{80} of 22 micrometers and cleaned several times to produce a marketable-grade zinc concentrate.

The combined zinc rougher and first-cleaner flotation tailing will constitute the final tailing.

1.5 Infrastructure

The Pickett Mountain Project is a green field mining project amid a logged area that has access roads used by the foresters to reach timber lots. A right-of-way has been established and it requires upgrading to meet safety standards for higher volumes of traffic that will occur with the advent of construction and operation of the mine. A total of $19.5 million will be required to supply and install the infrastructure for the project. Infrastructure includes the tailings storage facility and associated collection ponds as well as water run-off catchments surrounding the property. There are water treatment plants for industrial effluent as well as domestic sewage field beds. The plant design includes storage of soils and organics such that the operator may use it to complete restoration upon closure. There are roads to connect the various facilities and pads around the site including areas for cold storage waste rock storage and potential resource storage. A mill will be built on site to process material hoisted from underground and the main access for mine workings is a portal connecting to the decline. Electrical power will be brought to site by the utility supplier in the area.

1.6 Capital Expenditures

The initial capital cost estimate for the Project is summarised in Table 1.3 by major area. All costs are expressed in United States Dollars unless otherwise stated and are based on 2020 pricing and deemed to have an overall accuracy of ±40%. Equipment pricing was based on a combination of budget quotations and actual equipment costs from recent similar A-Z Mining Professionals Ltd. projects and are considered to be representative of the Project.

TABLE 1.3
CAPITAL COST ESTIMATE BY AREA

Component	Expenditures ($ millions)		Total Expenditures ($ millions)
	Year -2	Year -1	
Mine Development	6.75	14.69	21.44
Underground Infrastructure	3.21	6.95	10.16
Surface Infrastructure	10.00	10.11	20.11
Mining Equipment	0.99	0.99	1.98
Contingency	-	-	10.74

1.7 Operating Costs

The estimated total average operating cost for the life of mine (excluding smelting and refining) for the Pickett Mountain Project is approximately $93.08 per tonne. Table 1.4 presents a summary table of life-of-mine average operating costs for each department on a cost per tonne basis of potentially economic mineralisation.

TABLE 1.4
PROJECT TOTAL OPERATING COSTS

Department	Cost
Underground Mining	$ 47.73
Processing	$ 31.25
Dry Stack Placement of Tailings	$ 1.30
Surface Services	$ 2.63
General and Administration	$ 7.95
Environmental and Sustainable Development	$ 2.21
Total Cost	**$ 93.08**

1.8 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 industry consensus pricing supplied by Wolfden Resources Corporation (Table 1.5).

TABLE 1.5
COMMODITY PRICING

Commodity	Consensus Pricing
Zinc	$ 1.15
Copper	$ 3.00
Lead	$ 1.00
Gold	$ 1,500.00
Silver	$ 18.00

A summary of the expected parameters used for the financial analysis is presented in Table 1.6.

TABLE 1.6
CASHFLOW MODEL INPUT PARAMETERS

		Average Mill Head Grade:	Payability	Average Long Term Pricing
Undiluted Mineral Resources ~50/50 Indicate & Inferred	4,471,000 tonnes at grades of 9.51 % zinc, 1.23% copper, 3.77% lead, .88 g/t gold and 98.67 g/t silver			
Estimated Mining Dilution	10% at 0 grade			
Projected Mining Recovery	85%			
Zinc %		8.56	0.85	$ 1.15
Copper %		1.11	0.95	$ 3.00
Lead %		3.40	0.95	$ 1.00
Gold g/t		88.80	0.95	$ 1,500
Silver g/t		0.79	0.93	$ 18.00
Pre Production Capital, incl Working Capital	$ US 147.4 million			
Total Sustaining Capital	$ US 100 million			
Financial Assurance Trust: Reclamation & Closure	$ US 13.7 million			
Royalties	None			
Estimated Operating Costs ($/Tonne)	$ US 93.08 /tonne			
Life of Mine	9.7 years			

1.9 Sensitivity Analysis

Sensitivity analyses were performed for metal prices, capital expenditures, operating costs, mined grades, smelter charges, and recoveries with ranges up to 20% positive and negative variations. The project is most sensitive to changes in metals prices and reasonably sensitive to changes in all the other variables.

The NPV and IRR sensitivities to variations in key parameters are depicted graphically in Figure 1.2 and Figure 1.3. The IRR is most sensitive to variations in metal prices and mined grades and less sensitive to capital and operating costs.

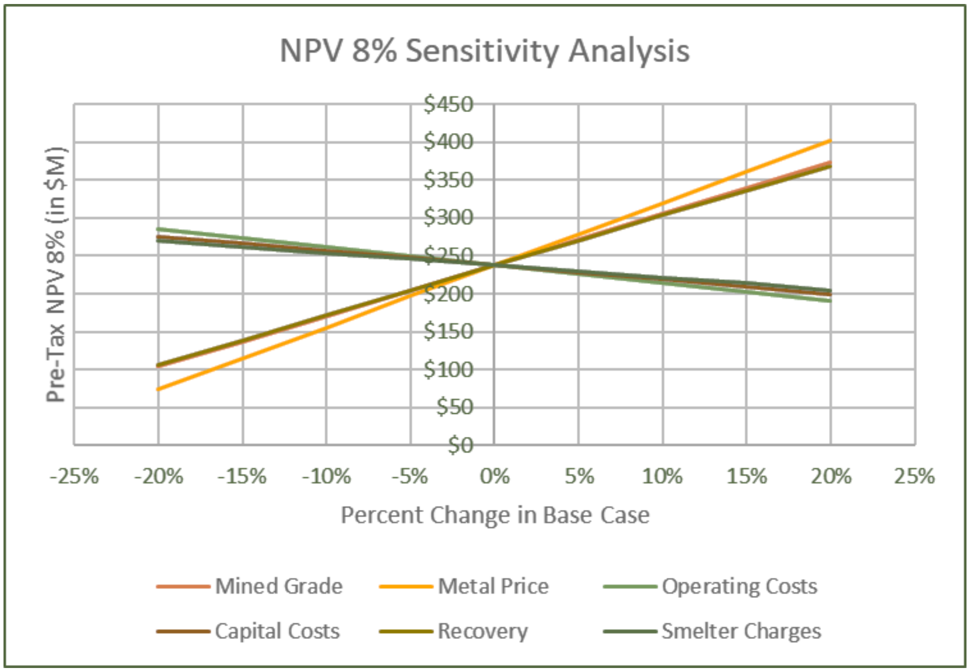


Figure 1.2 NPV at 8% Discount Sensitivity Analysis

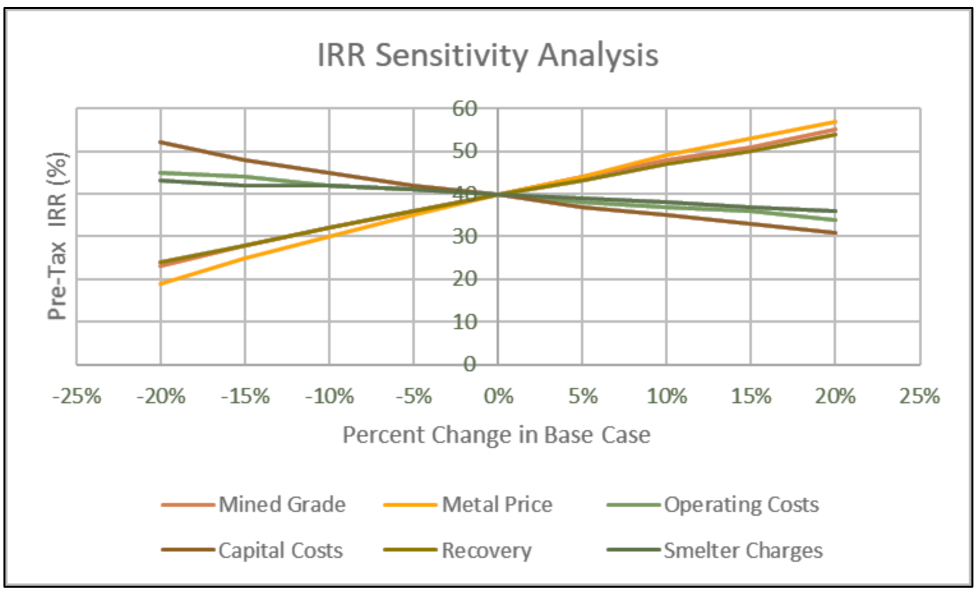


Figure 1.3 IRR Sensitivity Analysis

1.10 Conclusions

This Preliminary Economic Assessment has identified a diluted mineral resource of 4.2 million tonnes at 8.56% zinc, 1.11% copper, 3.4% lead, .79 g/t gold, and 88.8 g/t silver. This resource is comprised of 50% Indicated Resources and 50% Inferred 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. Mineral Resources that are not mineral reserves do not have demonstrated economic viability; therefore, there is no guarantee that the economic projections contained in this Preliminary Economic Assessment would be realized.

The deposit would be mined by underground mining methods with metals extracted in a processing plant custom built for the purpose. The mine site infrastructure facilities would be minimised but include a processing plant, small surface shop, warehouse, office complex, water treatment facility, dry stack tailings facility, and transformer and power distribution. Water for the project is assumed for this study to be provided from a well(s) near to the project initially then mainly recycled within the project site.

The mine would operate at 432,000 tonnes per annum and produce $1.36 billion in cash flow during the life of the mine.

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

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 deposit to production.

2. The proposed project would be considered a small to medium sized underground mining operation, which can be developed for production at a reasonable cost in a near-term horizon, provided regulatory approval and permits are acquired.

3. The mined grade of potentially economic mineralisation is an important variable for the success of the operation as are operating costs. Operating management efforts during mine production must be focused on these parameters.

4. The scoping level test work has indicated that a sequential flotation process will produce marketable-grade copper, lead, and zinc concentrates. Arsenic and antimony levels were high in copper concentrates produced in open-cycle and locked-cycle tests. Additional geo-metallurgical test work will provide additional information on the impurities in the marketable-grade copper concentrate to determine if penalties need to be paid. In addition, blending of ores from different areas in the mine will keep impurities (As/Sb) below penalty levels.

5. The following conclusions can be drawn based on the historical and current scoping level metallurgical studies:

    - The sequential flotation process is a process of choice for recovering marketable-grade concentrates of copper, lead, and zinc that include in each, quantities of previous metals.

    - Blending of material into the mill and/or final copper concentrate may be required to maintain low levels of arsenic and antimony, below the penalty limits for the concentrate.

    - Metal recoveries of 78% to 88% for copper, lead, and zinc are expected in the selected flowsheet while maintaining high quality concentrates.

    - Further testing is needed to optimise metal recoveries and reagent quantities in order to maximise revenue and reduce Capex and Opex for the milling circuit.

The project will be required to first obtain, from the Maine LUPC, approval of a rezoning petition that will allow for mining in this unorganised township. The petition was submitted and has been accepted by the LUPC as complete for its review. Based on initial soil and wetland field survey,s in addition to desktop studies as described in the petition, it was concluded that the preliminary designs of the proposed project would have no undue impact on the natural resources and could be completed in a manner that would fit harmoniously within the surrounding area, and therefore, would satisfy the goals and specific requirements of the LUPC.

The socio-economic analysis completed for the petition indicated there is adequate local capacity to provide municipal services and a sufficient labor pool to be employed and trained as employees. Wolfden correspondence and presentations with the towns proximal to the project resulted in concurrence letters from these towns in support of the project and that it would not pose an undue burden on municipal services provided by these communities and that other purchased services (solid waste, communications, power) had adequate capacity.

Following a rezoning approval, Wolfden will need to obtain a Maine Metallic Mining permit from the MEDEP under the Maine Chapter 200 rules. The Chapter 200 rules, with respect to metallic mines, only allow for underground mining methods and require tailings disposal as dry stacked tailings, in lined facilities, to be closed with a final cover of equal hydraulic performance. It is technically and financially feasible for the project to meet these two requirements. Review of the requirements for mine design, mine operation, mine closure, water collection and treatment, and reclamation and environmental monitoring, did not identify technical or operational requirements that could not be met by a well-designed and responsibly managed project. The future Baseline Characterisation Studies needed to support an MEDEP permit application are being developed and discussed with the MEDEP.

As part of Chapter 200 rules, the MEDEP will require that a financial assurance trust fund is established, prior to the issuance of a mining permit. In accordance with Section 17 of the Chapter 200 rules, the project will need to continuously maintain a financial assurance trust, as a condition of the mining permit, until the MEDEP determines that all reclamation, closure, post-closure maintenance and monitoring, and corrective actions have been completed and for as long as the MEDEP determines that the mining operation and any associated waste material could pose an unreasonable threat to public health and safety or the environment. The financial assurance trust must include sufficient funds for the following:

a) the cost to investigate all possible releases of contaminants at the site, monitor all aspects of the mining operation, close the mining operation in accordance with the closure plan, conduct treatment activities as necessary for all fluids and wastes generated by the mining operation and those post closure for a minimum of 100 years, implement remedial activities for all possible releases, and maintenance of structures and waste units as if these units have released contaminants to the groundwater and surface water, conduct corrective actions for potential environmental impacts to groundwater and surface water resources, as identified in the environmental impact assessment, and conduct all other necessary activities at the mine site in accordance with the environmental protection, reclamation and closure plan; and

b) the cost to respond to a worst-case catastrophic mining event or failure, including, but not limited to, the cost of restoring, repairing, and remediating any damage to public facilities or services, to private property, or to the environment resulting from the event or failure.

A filter cake TMF sized to contain the projected life of mine filtered tailings within the siting constraints identified can be constructed. Contact water from the TMF can be collected in an adequately sized pond constructed at the base of the TMF on the south side of the facility. The TMF would have a maximum elevation of 380 m, which is approximately the elevation of the treetops as measured from the ground surface at the topographic divide at the south side of the facility. There is potential for expansion of the TMF using the land to the south and there is flexibility for phased construction and progressive reclamation of the TMF.

1.11 Recommendations

It is recommended that infill drilling should continue with the aim of upgrading the Inferred Resources to Indicated Resources.

1. Metallurgical test work should be undertaken to optimise the process parameters for the proposed process flowsheet including confirmation that the process flowsheet is capable of processing variable ores from the mine and determine whether the variable ores respond to mechanical sorting technologies.

2. Perform sufficient test work to size equipment for the planned throughput. Some test work may be required at the Vendor's facility (*i.e.*, regrind mill sizing, thickener size, etc.).

Perform a detailed rock mechanics analysis for stope geometry and mine design including oriented core geotechnical drilling.

Continue to advance the project toward production by undertaking an advanced exploration program in parallel with finalising the project design and capital requirements. The goal of the Advanced Exploration Program will be to confirm resources with the objective of converting Mineral Resources to Mineral Reserves.

Complete a geochemical characterisation of simulated processed tailings for possible metal leaching and acid rock drainage (MI/ARD) potential.

It is recommended that Wolfden proceeds with the rezoning and mine permitting process, using contacts it has established with LUPC and MEDEP and engage with them in a proactive and collaborative fashion. Aligning with the State of Maine on how to meet legislative requirements for financial assurance trust fund should be prioritised. Future feasibility studies and designs should seek to avoid and minimise impacts to environmental, natural, cultural, scenic, and recreational resources to the extent possible. While the current mine development plan for treated water is subsurface infiltration, going forward Wolfden should consider other alternatives to provide greater flexibility and redundancy for the management of re-infiltration of treated water if soil site conditions warrant. These alternatives should, at a minimum, consider spray irrigation, which is an established method for treated waters in the State of Maine and which could be designed to operate year-round.

Further studies are recommended to advance the tailings facility design including geotechnical and hydrogeological investigations including laboratory testing to confirm site conditions, identify any potential geologic hazards, characterise foundations and groundwater conditions, and identify suitable borrow sources for construction fill. Tailings characterisation testing is recommended to better define the geochemical, physical, settling, and filtration properties to validate the TMF design criteria. 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. A grading plan should be developed that optimises the cut/fill balance for the TMF base grade. 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 Pre-Feasibility Study or Feasibility Study. The cost to complete a Pre-Feasibility or Feasibility Study for the Pickett Mountain Project is estimated to be between US$3 million to US$5 million plus the cost of any additional infill drilling to upgrade the mineral resources.

2.0 Introduction

2.1 Terms of Reference

This technical report was prepared by A-Z Mining Professionals Limited (AMPL) for the purpose of providing a National Instrument (NI) 43-101 Technical Report describing the geology and previous exploration history for the base metal deposit known as the Pickett Mountain Project (formerly known as Mount Chase) in Penobscot County, Maine, USA. This report also provides a Mineral Resource estimate for the Pickett Mountain base-metal deposit.

The Pickett Mountain Project property was acquired in 2017 by Wolfden Mt. Chase LLC, a wholly-owned subsidiary of Wolfden Resources Corporation (Wolfden), in an arm's length, third party transaction of US$8.5 million.

AMPL was retained by Mr. Jeremy Ouellette, Vice President Corporate Development, to prepare this Preliminary Economic Assessment for Wolfden.

As of the date of this Report, Wolfden is a Canadian junior exploration and development company listed on the Canadian TSX Venture stock exchange (WLF.TSXV) and with a corporate office at:

1100 Russell Street, Unit 5
Thunder Bay, Ontario P7B 5N2
Canada
Tel: 807-624-1136

This Report is considered effective as of September 14, 2020 with a filing date of October ??, 2020.

AMPL's qualified persons are responsible for the areas in 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:

- Finley Bakker, P. Geo, Contract Resource Geologist to AMPL
- Jerry Grant, P. Geo, Contract Geologist, QA/QC, and Geology
- Brian LeBlanc, P. Eng, President and Senior Engineer, AMPL
- Eric Hinton, P. Eng, Principal and Senior Engineer, AMPL
- Darryl Boyd, HBSc, Principal and Manager of Regulatory Compliance, AMPL
- Frank Palkovits, P.Eng, Mining Engineer, President of Mine Paste Ltd.
- Eric Sellars, P.Eng, Consultant Geotechnical Engineer, SLR
- Deepak Malhotra, President, Pro Solv Consulting, LLC
- Peter Baker, Senior Program Manager, Wood Environment and Infrastructure, Inc.
- Peter Thompson, Senior Project Manager/Senior Hydrologist, Cedere Associates, LLC

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 technical personnel from the company regarding all pertinent aspects of the project as listed in the "References" (Section 27.0) of this report.

AMPL has not conducted detailed land status evaluations, but has obtained tenure information from previous technical reports, public documents, and statements by Wolfden regarding property status and legal title to the project, which is owned by Wolfden Mt Chase LLC.

2.3 Site Visit

Mr. Alan Aubut, P. Geo., previously of A-Z Mining Professionals Ltd. (AMPL), a qualified person under the terms of NI 43-101, conducted a site visit to the Pickett Mountain Property on September 27, 2017. A site visit to a core storage facility, housing the Pickett Mountain core and maintained by Huber Engineered Woods at their Easton, Maine production facility, was conducted on September 26, 2017. Mr. Jerry Grant, P.Geo, a qualified person under the terms of NI 43-101 and acting as a consulting geologist for Wolfden, worked on site on the 2018, 2019, and 2020 drill programs, verified current and previous data, and oversaw the QA/QC results and program. An updated site visit was not achieved during this study due to health and safety concerns and travel restrictions related to the Covid-19 pandemic ongoing throughout the duration of this study.

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 US dollars ("US$")
- Gold (Au) and Silver (Ag) assay values are reported in ounces per tonne (opt)
- Copper (Cu), Lead (Pb), and Zinc (Zn) assay values are reported in percent (%)
- Maps are either in UTM coordinates or in the latitude/longitude system

2.5 Glossary and Abbreviations of Terms

Abbreviation	Meaning
°C	degrees Celsius
C$ and CA$	currency of Canada
Ag	silver
Altius	Altius Minerals Corporation
AMPL	A-Z Mining Professionals Ltd.
Au	gold
Chevron	Chevron Resources Company, a subsidiary of Chevron Oil
cm	centimetre
Cu	copper
DDH or ddh	diamond drill hole
E	east
EM	electromagnetic
g	gram
g/t	grams per tonne
Getty	Getty Mineral Company, a subsidiary of Getty Oil
ha	hectare
HLEM	Horizontal Loop electromagnetics (geophysical survey method)
IP	induced polarization
km	kilometre
kW	kilowatts
MDEP	Maine Department of Environmental Protection
m	metre
mm	millimetre
Mt	millions of tonnes
N	north
NSR	net smelter return
opt	ounces per ton
P.Geo	Professional geoscientist
PEA	Preliminary Economic Assessment
Pb	lead
ppm	parts per million
QP	Qualified Person
t	tonne (metric)
t/m^3	tonne per cubic metre
TDEM	time domain electromagnetic
US$	currency of the United States of America
USA	United States of America
USGS	United State Geological Survey
UTM	Universal Transverse Mercator
VTEM™	Versatile time domain electromagnetic
WCC	Woodward-Clyde Consultants
Wolfden	Wolfden Resources Corporation
Zn	zinc
ZnEq	zinc equivalent

3.0 Reliance on Other Experts

The updated Mineral Resource estimate, used for the purposes of this report, using a 7% cut-off grade, has been prepared by Independent Qualified Persons (QP), Mr. Finley Bakker (P. Geo.) and has an effective date of September 14, 2020. The updated estimate is based on the same methodology as the January 7, 2019 estimate, without the insertion or deletion of any data, that was prepared by Independent Qualified Persons (QP), Mr. Finley Bakker (P. Geo.), Mr. Jerry Grant (P. Geo.), and Mr. Brian LeBlanc (P. Eng.), of A-Z Mining Professionals Ltd.

Qualified Person, Mr. Ron DeGagne of Environmental Applications Group, has relied on information supplied by Mr. Peter Baker, Senior Program Manager, Wood Environment and Infrastructure Inc. of Portland, Maine and Mr. Peter Thompson, Senior Hydrogeologist of Credere Associates LLC of Westbrook, Maine for information relating to the environmental studies, permitting, water management, and closure.

Qualified Person, Dr. Deepak Malhotra, Ph.D., President, Pro Solv Consulting, LLC and consultant to Resource Development Inc., is responsible for all matters pertaining to metallurgy, specifically Sections 1.4, 13.0, 17.0, and 21.1.5.

Qualified Persons, Mr. Frank Palkovits, P.Eng, Mining Engineer, President of Mine Paste Ltd. and Mr. Eric Sellars, P.Eng, Consultant Geotechnical Engineer, SLR, are responsible for all matters pertaining to the design, construction, and operation of the Tailings Management Facility.

4.0 Property Description and Location

The Pickett Mountain Property is in northeastern Maine, in the southeast quarter of Township 6, Range 6, Penobscot County. The Property consists of 2,781 hectares (6870 acres) of private land. It is about 16 km (10 miles) north of the village of Patten and about 153 km (95 miles) north of Bangor (Figure 4.1). It is approximately 53 km (33 miles) from the Canadian border and is approximately 67 km (42 miles) due west of the town of Woodstock, New Brunswick.

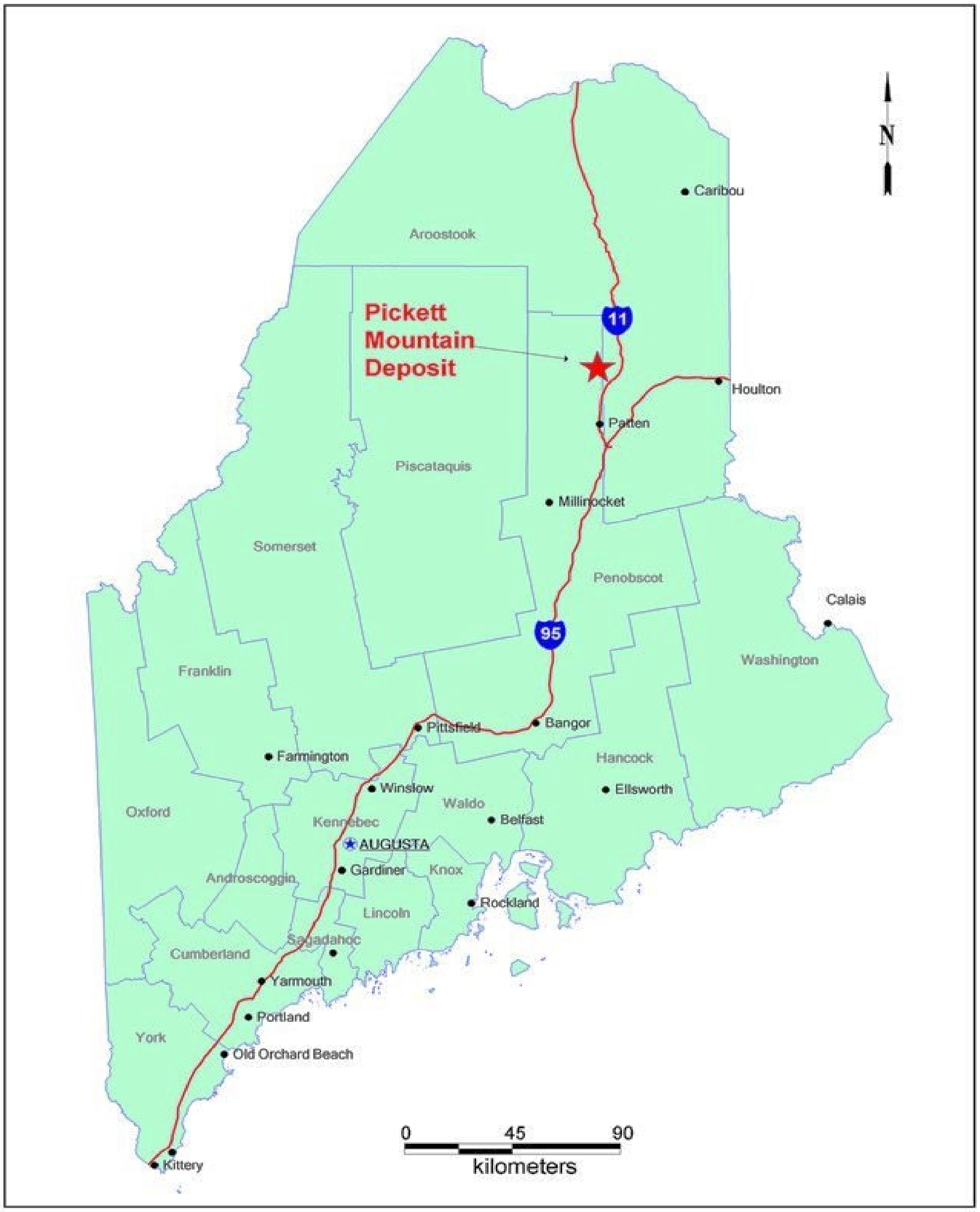


Figure 4.1 Pickett Mountain Project Location Map

4.1 Land Tenure

Wolfden acquired, through its indirect wholly-owned subsidiary Wolfden Mt. Chase LLC, all of the mineral, timber, oil, and surface rights, exclusive of the surface area of great ponds (lakes that include the waters of Pickett Mountain Pond, Pleasant Lake and Mud Lake) covering approximately 2,781 hectares (Figure 4.2). More specifically, the Property consists of the southeast quarter of Township 6, Range 6, in

Penobscot County, Maine. The only known encumbrances are two small surface rights parcels on the north shore of Pleasant Lake and a small surface rights lease on the south side of Pleasant Lake for recreation purposes.

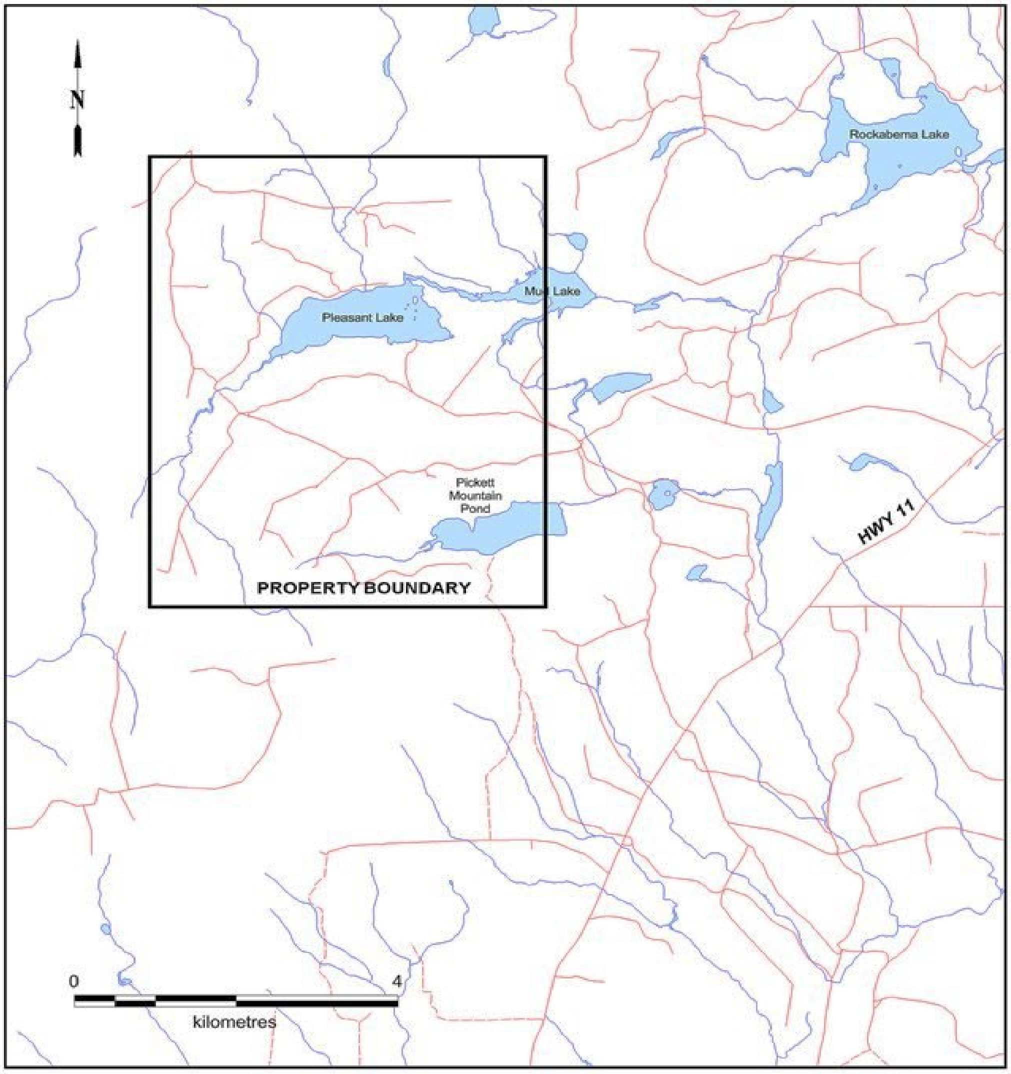


Figure 4.2 Pickett Mountain Property Map

Wolfden advises that it does not require any permits to complete the contemplated exploration work on the Property. The authors are not aware of any other significant factors and risks that may affect access, title, or the right or ability to perform work on the Property as currently contemplated. There are no known environmental liabilities to which the Property is subject to.

4.2 Purchase Agreement

On November 15, 2017, Wolfden Mt. Chase LLC acquired a 100% interest in the Pickett Mountain Project for a cash purchase price of US$8.5 million (the "Acquisition") from a third-party vendor. To fund the acquisition, the Company granted a 1.35% gross sales royalty on the Pickett Mountain Project to a subsidiary of Altius Minerals Corporation for cash consideration of US$6 million and completed a non-brokered private placement of 20,200,000 subscriptions at a price of C$0.25 per subscription receipt for gross proceeds of C$5,050,000.

5.0 Accessibility, Climate, Local Resources, Infrastructure and Physiography

5.1 Accessibility

Access to the Property from State Highway 11 is by an 8.4 km long, well used logging road. From State Highway 11, there are paved primary and secondary highways with access to Interstate Highway 95 at Island Falls, a total distance from the Property of about 36 km. The presence of existing infrastructure permits exploration to be carried out year-round.

5.2 Climate

The climate of Northern Maine is a typical humid continental climate. The average annual temperature in Patten is 4.2°C. In a year, the average rainfall is 1,002 mm. Between the driest and wettest months, the difference in precipitation is 42 mm. During the year, the average temperatures vary by 30.1°C. Summer temperatures typically vary between 6°C and 25°C while winter temperatures usually range between 2°C and -17°C with an average January temperature of -11°C (Figure 5.1). The region usually receives approximately 63 to 105 mm of precipitation per month with November being normally the wettest month (www.Climate-Data.org).

	January	February	March	April	May	June	July	August	September	October	November	December
Avg. Temperature (°C)	-11.5	-10	-4	3.2	10.1	15.8	18.6	17.3	12.4	6.4	0.1	-8
Min. Temperature (°C)	-17.8	-16.9	-10.3	-2.6	3.1	8.9	11.9	10.6	5.8	0.6	-4.4	-13.4
Max. Temperature (°C)	-5.1	-3.1	2.4	9.1	17.2	22.8	25.4	24.1	19	12.3	4.6	-2.6
Avg. Temperature (°F)	11.3	14.0	24.8	37.8	50.2	60.4	65.5	63.1	54.3	43.5	32.2	17.6
Min. Temperature (°F)	-0.0	1.6	13.5	27.3	37.6	48.0	53.4	51.1	42.4	33.1	24.1	7.9
Max. Temperature (°F)	22.8	26.4	36.3	48.4	63.0	73.0	77.7	75.4	66.2	54.1	40.3	27.3
Precipitation / Rainfall (mm)	69	63	67	74	83	85	98	98	85	87	105	88

Figure 5.1 Patten, ME Historical Weather Data
(From https://en.climate-data.org/location/140940/)

5.3 Local Resources

The nearest community to the Property is Patten, Maine, located 16 km south-southeast along Highway11. It has a population of approximately 1,000 and is the site where Wolfden established its operational base for the project. By taking Secondary Highway 159 east, approximately 14.5 km, one can connect to Interstate Highway 95 at Island Falls. There, it is possible to connect to a railway operated by the Maine Northern Railway.

5.4 Infrastructure

The area is well supported by local infrastructure, including well maintained roads, highways, and access to rail in the town of Sherman Station (27 km from the Property), as well the state's electric grid that runs along Highway 11.

5.5 Physiography

The Property lies within rolling hills just to the northeast of a range of hills with the highest elevation being at nearby Mount Chase at 744m above sea level. The average surface elevation is about 366m. The area is well wooded with a mixture of hardwood and softwood. Hardwood species present include maple, beech, and birch with lesser ash. Softwood includes spruce and some pine and cedar.

6.0 History of the Property

Exploration in Maine for massive sulphides commenced soon after 1953 when the Brunswick #6 deposit was discovered in neighbouring New Brunswick. This early work concentrated on the volcanic rocks known to exist along the Maine coast and resulted in two deposits being found and developed: Cape Rosier and Blue Hill. Intermittent exploration continued in northern and western Maine through to the 1970s. In 1967, a consortium of exploration companies operated under the name “The Northeast Joint Venture.” This group eventually discovered the base metal deposit at Bald Mountain in 1977 (Scully, 1988).

The first documented mineral exploration work in the immediate area was done by Humble Oil and Refining Company in 1968. Their subsidiary, North American Exploration Co., completed regional geochemical surveys that resulted in a 915m by 1,830m grid being established around Pickett Mountain and distinct anomalies were detected (Luethe, 1989).

1978 – 1984: In 1978, Getty Mineral Company (Getty) explored the area and again using a regional geochemical sampling program located an anomalous area close to Pickett Mountain (see Figure 6.1 to Figure 6.3, below). The program involved collecting stream, seep, and soil samples averaging about 30 samples per square mile. This program was followed by a more detailed soil sampling program that further defined the geochemical anomaly. During the summer of 1979, a Max-Min horizontal loop electromagnetic (HLEM) and magnetic surveys were conducted (see Figure 6.4, below). A bedrock conductive source was identified and drilled in the fall. This drilling intersected massive sulphides within volcanics. The initial drill program consisted of 12 holes totaling 1,473m (Luethe, 1989).

During 1980, Getty undertook additional geophysics. In 1981, 10 diamond drill holes were completed totaling 1,602m to test some outlying targets. The drilling failed to locate any massive sulphides. In 1982, an EM-37 survey was undertaken (see Figure 6.4, below) to test for deeper mineralisation. An airborne “Input” survey was flown over the Property in 1983.

Hole 23 was drilled in 1982 and intersected significant sulphide mineralisation. A total of 28,020m in 96 holes were drilled between 1982 and 1984. During this same period, preliminary metallurgical testing, baseline environmental studies, and a pre-feasibility study were completed.

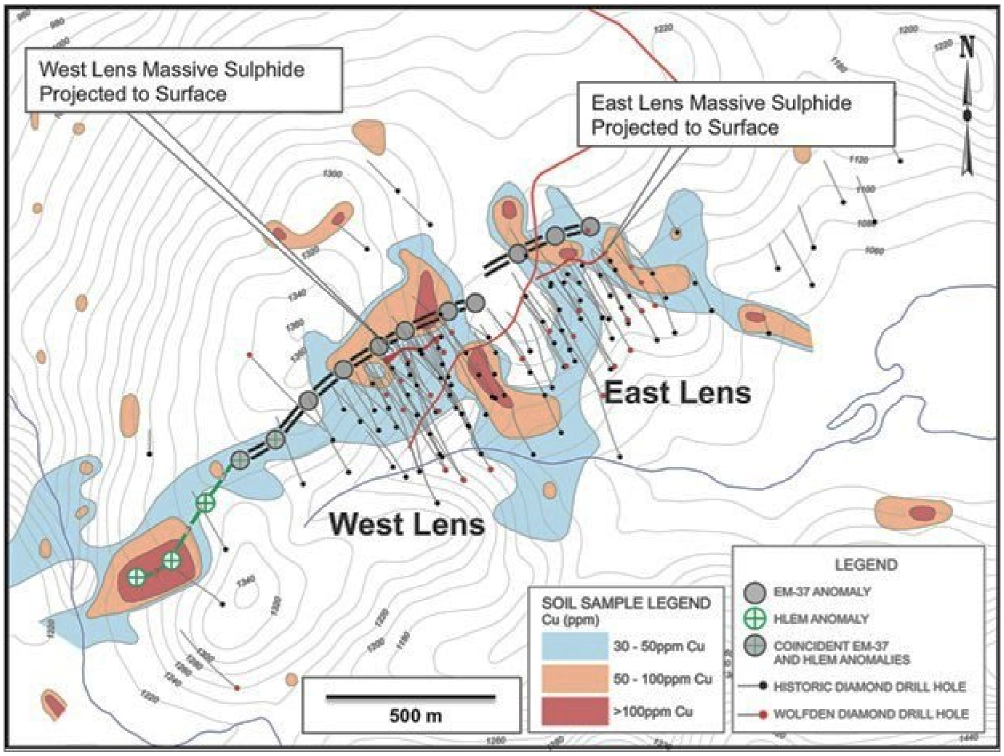


Figure 6.1 Historic Soil Sampling Over the Pickett Mountain Property – Cu

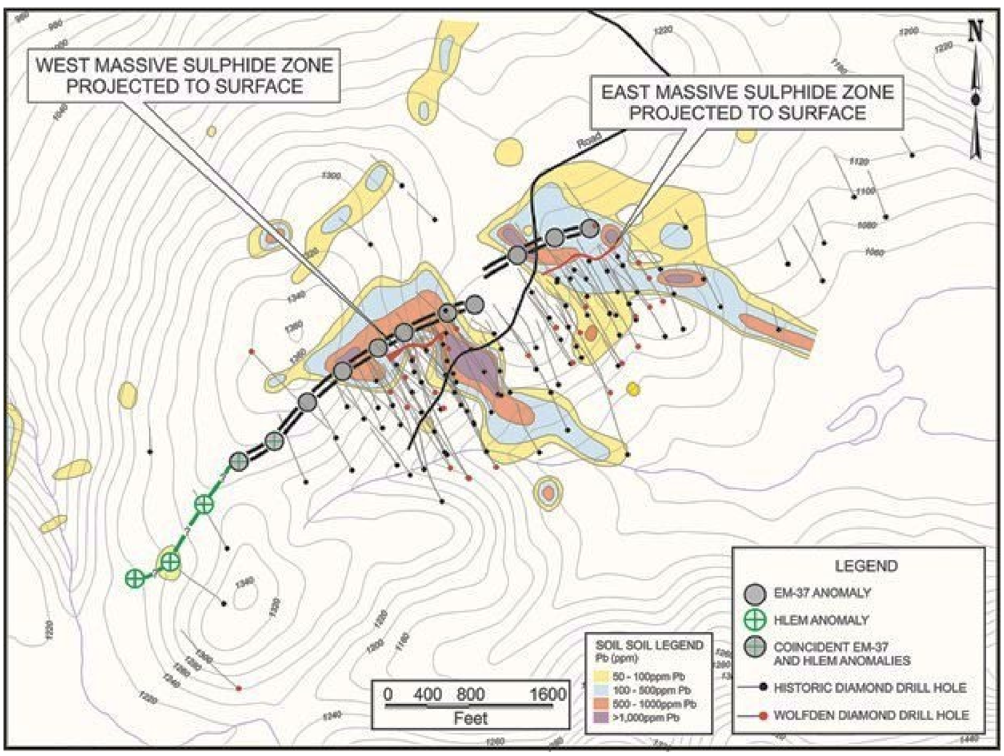


Figure 6.2 Historic Soil Sampling Over the Pickett Mountain Property – Pb

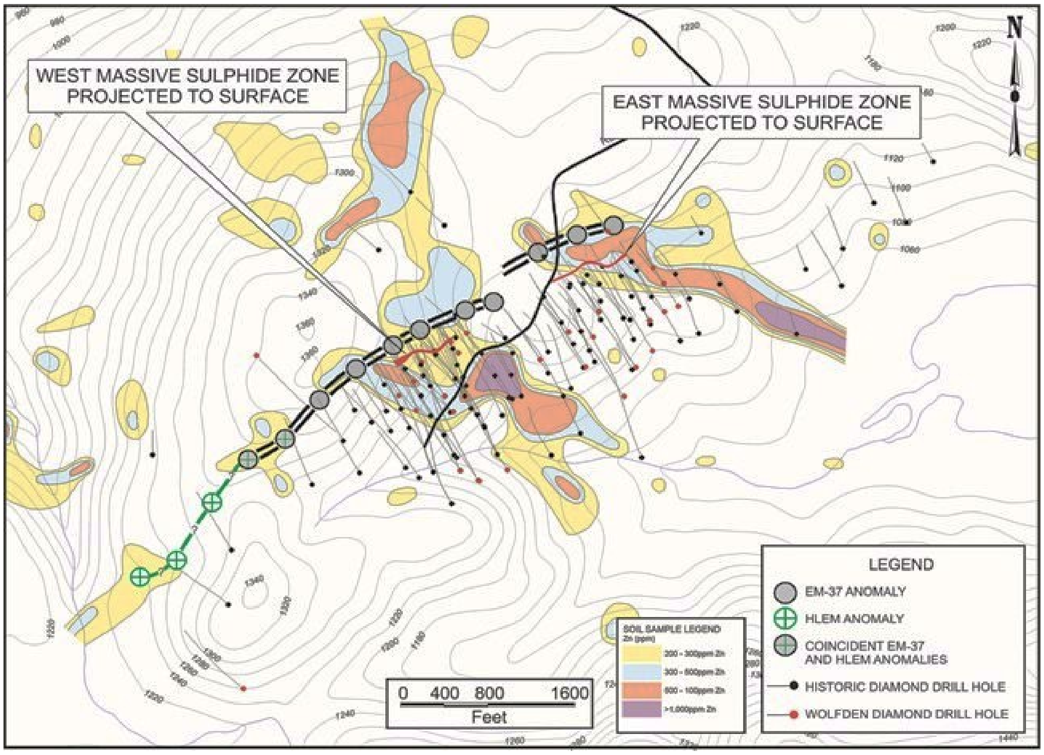


Figure 6.3 Historic Soil Sampling Over the Pickett Mountain Property – Zn

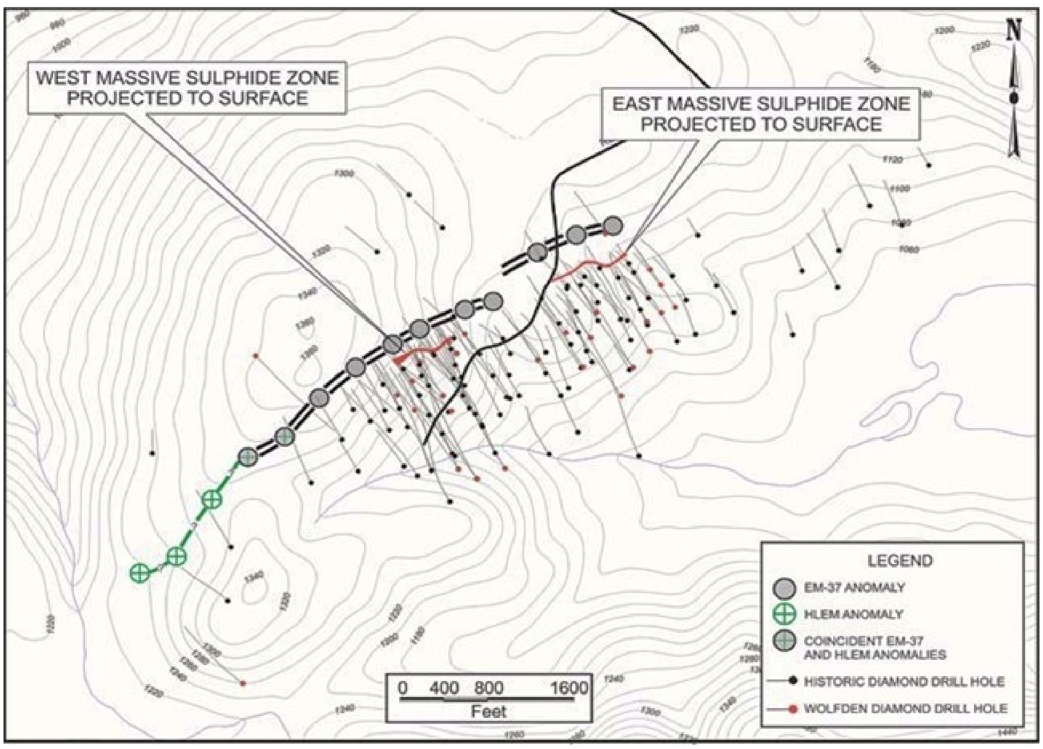


Figure 6.4 Compilation of Historical Geophysical Surveys

An historical resource estimate was undertaken using the "Contour Method" for Getty in 1983. The methodology used involved creating thickness and grade-thickness grids that used an eight-foot thickness and 4% total sulphide cut-off, with any area not meeting either threshold not being included in the calculation. As it was still early in the exploration of the deposit, no geologic interpretation was used to limit the deposit size. Using an average tonnage factor of 8.25 cubic feet (a density of 4.1 t/m^3) per ton, to a depth of approximately 1,300 feet (400m), the estimated resource was 3.15 million tons with an average grade of 9.66% Zn, 4.30% Pb, 1.24% Cu, 2.96 opt Ag, and 0.029 opt Au (Laverty, 1983; Riddell, 1983). This historical resource does not use the classification terms "Inferred Mineral Resource," "Indicated Mineral Resource," and "Measured Mineral Resource" that have the meanings ascribed to them by the Canadian Institute of Mining, Metallurgy and Petroleum, as the CIM Definition Standards on Mineral Resources and Mineral Reserves adopted by CIM Council, as amended. The authors have not done sufficient work to classify this historical estimate as a current Mineral Resource and Wolfden is not treating this historical estimate as a current Mineral Resource and they are included in this section for illustrative purposes only and should not be disclosed out of context.

With the purchase of Getty Oil by Texaco in late 1984, the project was terminated and the leases put up for sale.

1985 – 1989: Chevron Resources Company purchased the Getty lease in October 1985 and then immediately renewed exploration on the Property primarily looking for additional massive sulphides along strike. Additional geophysical surveys, including a proprietary deep penetrating EM survey were completed. An additional 16 drill holes totalling 6,038m were drilled. Sulphides were intersected although no significant massive sulphides were located (Luethe, 1989).

In the second half of 1988, work was carried out in the vicinity of Getty hole 66-84-90. A detailed re-evaluation and a revised geologic interpretation was completed. Additional metallurgical work was also completed (Luethe, 1989).

Chevron completed another historical resource estimate using the updated geological interpretation. This estimate involved using the polygonal method to a depth of approximately 1,300 feet (400m). Grades were converted to zinc equivalent (% ZnEq = % Zn + (% Pb × 0.53) + (% Cu × 1.64) + (opt Ag × 0.45). Using a minimum horizontal thickness of 5 feet and an arbitrary cut-off grade of 11% ZnEq the resource was estimated to be 2.5 million tons averaging 11.42% Zn, 4.94% Pb, 1.62% Cu, and 3.3 opt Ag. Even though it has some of the highest grades intersected by drilling, the #1 lens was excluded as only 4 holes had tested the lens (Luethe, 1989). This historical resource does not use the classification terms "Inferred Mineral Resource," "Indicated Mineral Resource," and "Measured Mineral Resource" that have the meanings ascribed to them by the Canadian Institute of Mining, Metallurgy and Petroleum, as the CIM Definition Standards on Mineral Resources and Mineral Reserves adopted by CIM Council, as amended. The authors have not done sufficient work to classify this historical estimate as a current Mineral Resource and Wolfden is not treating this historical estimate as a current Mineral Resource and they are included in this section for illustrative purposes only and should not be disclosed out of context.

To the best of the knowledge of the authors of this report, the last historical work completed on the project and any related accessible data from that work was in 1989.

7.0 Geological Setting and Mineralisation

7.1 Regional Geology

The Pickett Mountain project is in the northern Appalachian orogenic belt. The Appalachians are a Paleozoic orogen that formed along the northern margin of Gondwana in the Neoproterozoic and early Paleozoic. It has been subdivided into 5 domains based on stratigraphic and structural contrasts: Humber, Notre Dame, Ganderia, Avalonia, and Meguma, as shown in Figure 7.1 (Hibbard, et al., 2007; Fyffe, et al., 2009). The Pickett Mountain project is located within the Ganderia Zone.

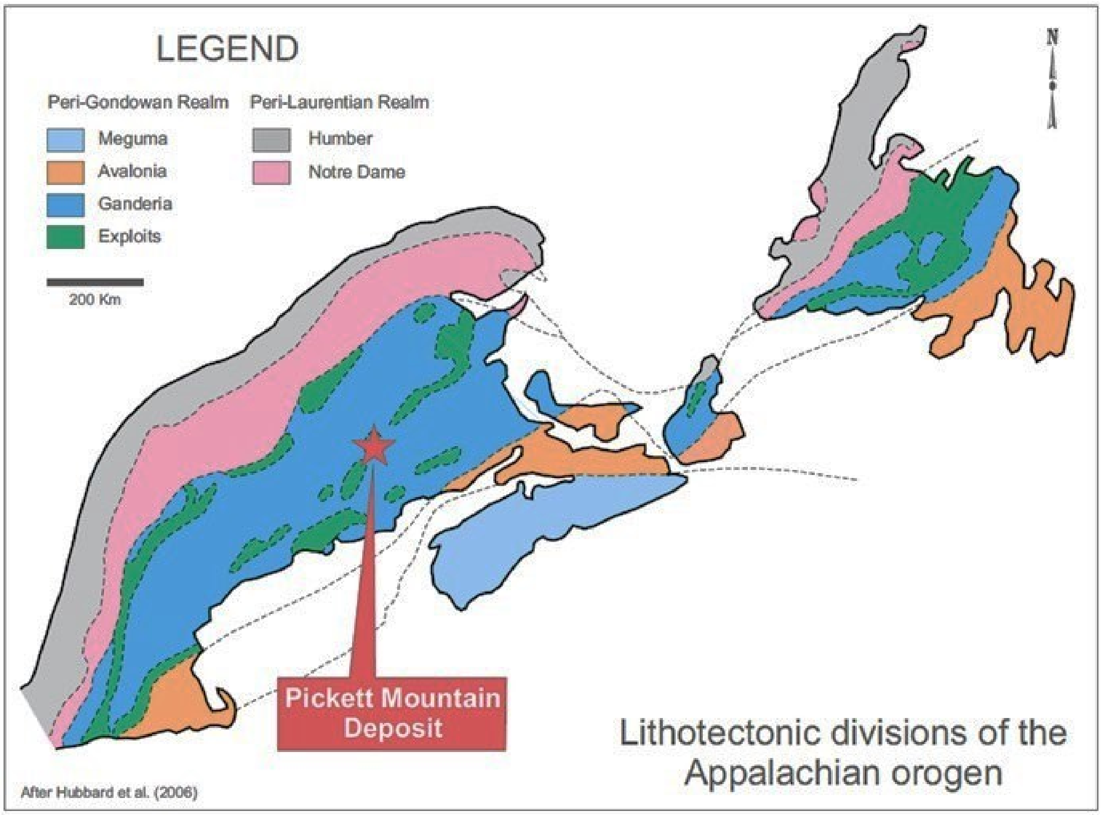


Figure 7.1 Lithotectonic Divisions of the Northern Appalachian Orogen
(**Source:** Adapted from Hibbard, et al., 2006)

The Ganderia Zone consists of Late Neoproterozoic to Early Ordovician rocks that are predominantly continent-derived, quartz-rich sediments and with Neoproterozoic volcanic and plutonic rocks (Fyffe, et al., 2009). These have undergone multiple stages of deformation, metamorphism, and plutonism and record the development and destruction of a continental margin (Williams, 1978).

The Property covers a portion of the southeast limb of the southwest plunging Weeksboro-Lunksoos Lake Anticlinorium that is cored by the Grand Pitch Formation, made up of complexly folded shale and siltstone with interbedded quartzite and greywacke and believed to be of Early Cambrian age (Figure 7.2). The stratigraphic sequence within the Anticlinorium and above the unconformity is illustrated in Figure 7.3.

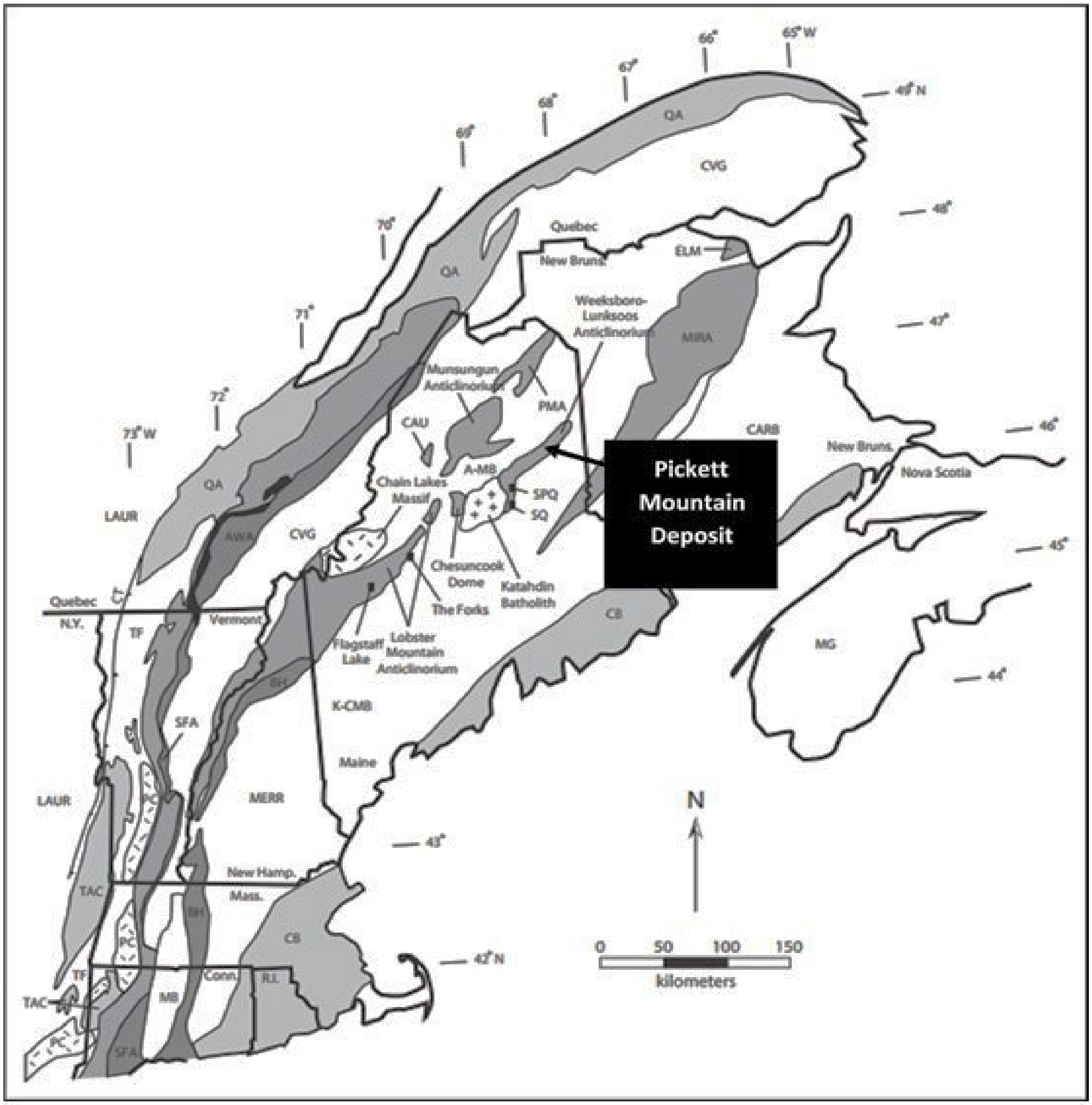


Figure 7.2 Generalised Geology of the Northern Appalachians

(Source: Schoonmaker, et al., 2017)

Pre-Devonian units are shaded. LAUR = autochthonous Laurentian margin, QA = Quebec Allochthons, TAG = Taconic Allochthons, TF = transported Laurentian margin and basin deposits, PC = Precambrian massifs, SFA-AWA = Shelburne Falls arc, Ascot-Weedon arc, and related oceanic rocks, including ophiolitic fragments (black), MB = Mesozoic basin, CVG = Connecticut Valley Gaspe Synclinorium, BH = Bronson Hill Arc, MERR = Merrimack Synclinorium, CAU = Caucomgomoc inlier, A-MB = Aroostook-Matapedia belt, SPQ = Shin Pond quadrangle, SQ = Stacyville quadrangle, PMA = Pennington Mtn. Anticlinorium, MIRA = Miramichi Highlands, K-CMB = Kearsarge–Central Maine belt, ELM = Elmtree-Belledune inlier, CARB = Carboniferous cover rocks, CB = Coastal belt, MEG = Meguma terrane

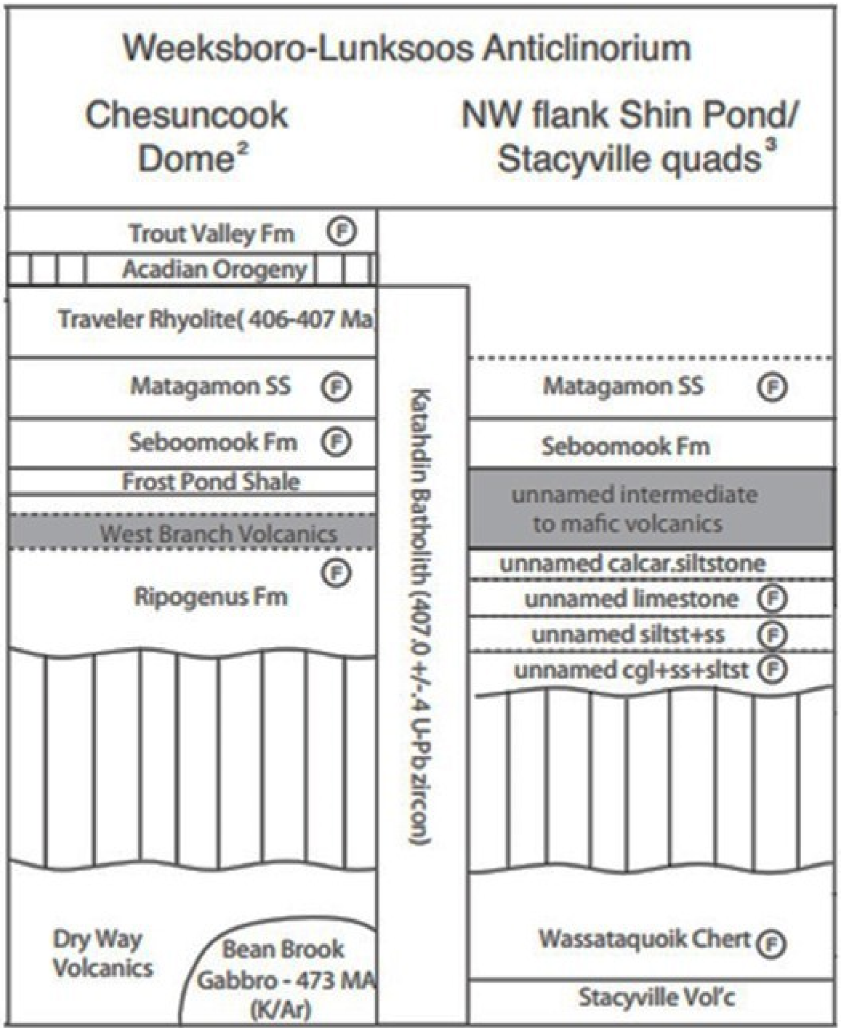


Figure 7.3 Stratigraphic Section for the Weeksboro-Lunksoos Lake Anticlinorium, North-Central Maine, Showing Ordovician Through Devonian Rocks

All units shown lie unconformably above the Cambrian Grand Pitch Formation (**Source:** Adapted from Schoonmaker, et al., 2011).

7.2 Local Geology and Structure

The local stratigraphy documented in this section is thought to be equivalent to the lower-most Ordovician-age volcanic rocks (Dry Way Volcanics and Stacyville Volcanics) illustrated on Figure 7.3. The geology of the Pickett Mountain deposit locale, as mapped in 2018, is illustrated in Figure 7.4 and a cross section of the deposit and associated lithotypes are depicted on Figure 7.5.

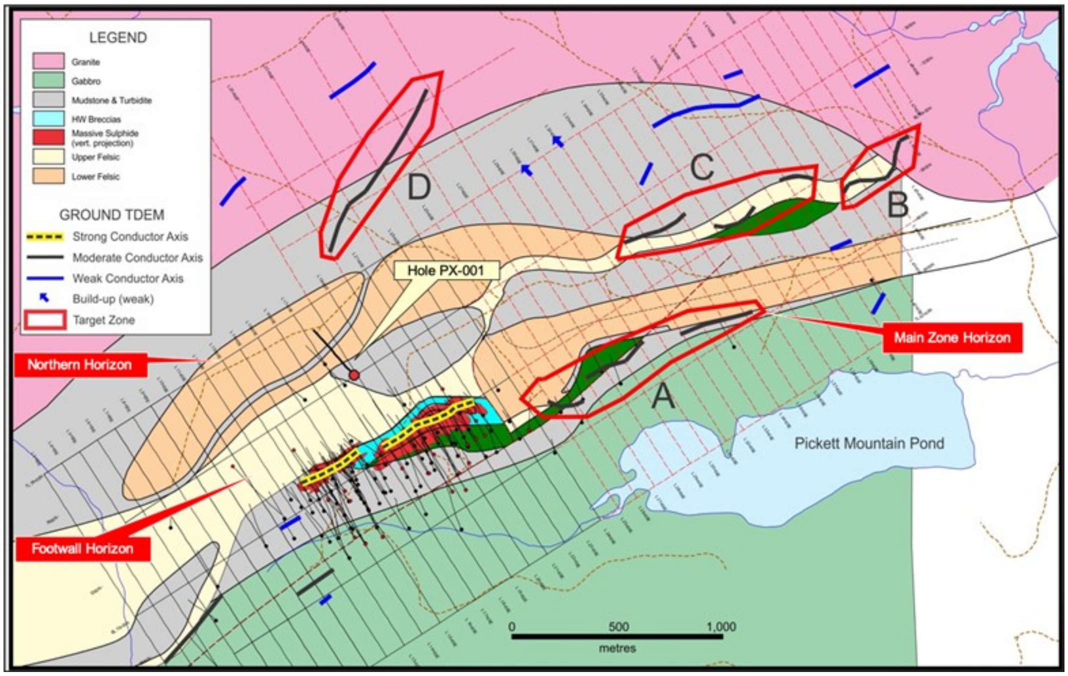


Figure 7.4 Geology Plan Map of the Pickett Mountain Deposit

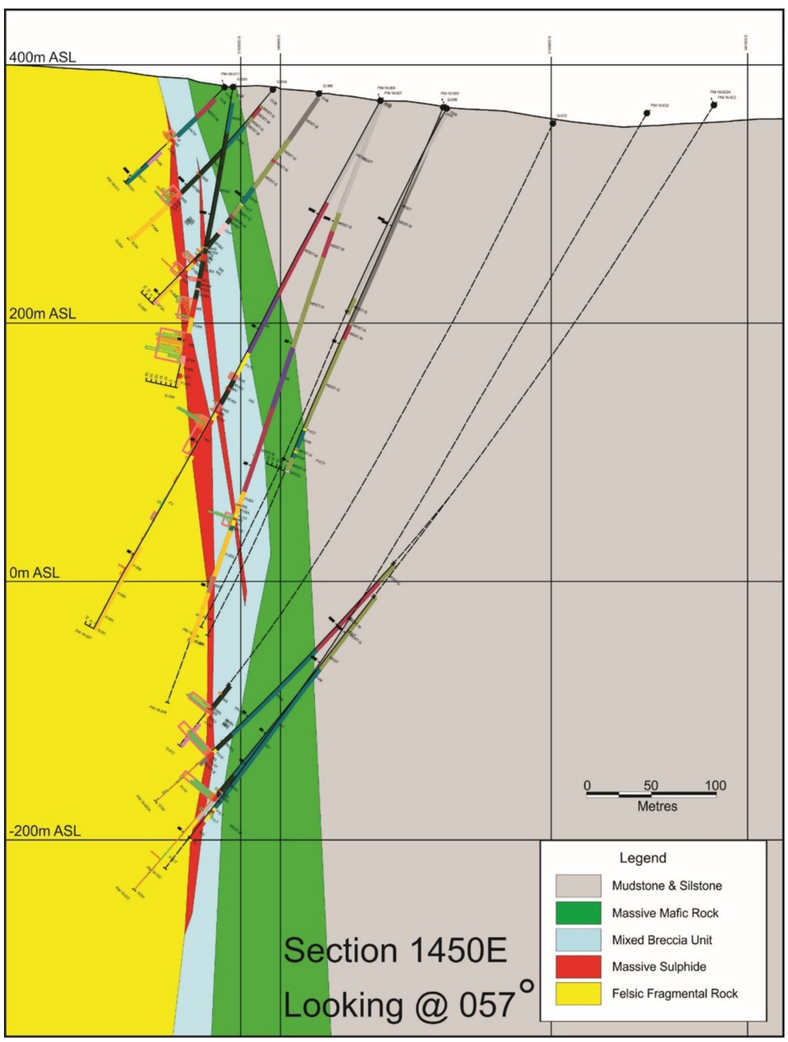


Figure 7.5 Cross Section of the Pickett Mountain Deposit

In 2018, geological mapping was completed in the deposit area as well as in the northwest portion of the Property. Outcrop exposure is quite poor; mapping was augmented with the logged drill-hole geology in the deposit area. Three main rock units were observed in the outcrop: footwall felsic volcanic rock, hanging wall mudstone-siltstone, and hanging wall massive mafic rock.

In the deposit area, the contact between the footwall and hanging wall rocks is occupied by an assemblage of mafic and felsic flows and breccia, mudstone, and massive sulphide. Generally, contacts and bedding strike northeast and dip steeply to the southeast. Repetitions of the contact between the footwall and hanging wall rocks suggest folding. The W1 and W2 Lenses are planar and steeply dipping. The E1 Lens is similarly oriented at its west edge, but the strike rotates clockwise and the dip shallows eastward, as it becomes affected by an interpreted synclinal fold nose with an axis that plunges towards the southwest.

7.2.1 Stratigraphy

7.2.1.1 Footwall Felsic Rocks

The lowermost rock units appear to be comprised of three pulses of felsic volcanism and a clear time break represented by a heterolithic debris flow. These rocks are intruded by a unit of massive, quartz-feldspar porphyritic rock described herein as QFP and locally by a massive, medium to coarse grained feldspar quartz porphyritic felsic unit. An alternative theory is that the QFP cores a fold and that the initial and latter felsic sequences are stratigraphically equivalent.

The initial felsic volcanic unit is generally massive in character, often flow banded, sporadically hematitic, and often magnetic. It contains scattered, millimetre-sized quartz phenocrysts and forms a prominent magnetic high to the north (FW) of the West, and, in part, to the East Lens. Magnetic inversion modeling indicates that this unit does not extend much deeper than 150m vertically, but drilling is still required to validate this. Along strike to the east, this unit disappears and an argillaceous sequence appears; the nature of this ‘facies’ change is unknown at this time. Up stratigraphy, this massive unit gives way to a heterolithic debris flow, that is locally calcareous, locally base-metal bearing, and weakly to intensely altered. It is believed that the FW massive sulphide zone is associated with the upper contact of this unit. Locally, a quartz crystal tuff lies above the debris flow, this unit contains millimetre-sized quartz phenocrysts that locally agglomerate to form grape-like clusters. The next sequence is commonly felsic volcanic breccia, consisting of rounded, oblate fragments in a matrix of similar composition and texture, but a slightly different colour. Quartz and more commonly, feldspar phenocrysts, are generally round and less than 1 mm. Sections of the breccia contain abundant blocky patches of dark, fine-grained felsic rock with scattered 0.5 mm plagioclase phenocrysts thought to be fiamme, although wall rock rip-ups have also been reported. The fragmental rock also includes sections of tuff and lapilli tuff, which are compositionally similar to the volcanic breccia. Thin aplite dykes are reported in the drill logs. The foliation is usually penetrative and the aspect ratio of the fragments is 2:1:1. Sericitisation is always present and commonly minor, but increases to strong in the deposit area.

Several different types of generally massive textured ‘QFP’ have been recognised in core and outcrop. The QFP varies from quite coarse grained with centimetre-sized quartz phenocrysts with little to no obvious feldspar crystals, and, sometimes hornblende crystals to locally QFP with abundant 1-2 mm rounded quartz and feldspar phenocrysts in a fine-grained, hard, felsic matric. The quartz eyes tend to clump together in 0.5-1.0 cm masses somewhat resembling raspberries. In one area, below the East Lens sulphide zone, the QFP, with the mm-size quartz crystals, is obviously fragmental in character and extrusive. In other areas, the massive and extremely consistent texture of the QFP unit suggests that it is intrusive. The foliation appears as anastomosing 0.5 cm-spaced cleavage, and alteration varies from very weak to intense, with abundant chlorite and sulphides.

7.2.1.2 Massive Sulphide

The massive sulphide is fine-grained and weakly to moderately banded, with the banding defined by centimetre to decimetre scale variations in the content of pyrite, sphalerite, galena, chalcopyrite, and gangue minerals. Other minerals present in varying amounts include calcite, chlorite, tetrahedrite, arsenopyrite, and magnetite.

The massive sulphide attains a maximum horizontal width of up to 25m (E1 Lens).

7.2.1.3 Breccia Unit

In the deposit area, a disrupted assemblage of rock types separates the deposit contact and a stratigraphically overlying massive mafic flow. The assemblage unit is 150m wide horizontally in the footwall to the East Zone, but thins to the west, pinching out near the West Zone.

The unit is not exposed on the surface; the drill logs suggest the unit is dominantly mafic breccia, with fist-sized mafic bombs in hyaloclastite. Other rocks include massive felsic and pyroclastic flows (which have Zr/Ti ratios distinctly lower than those of the footwall felsic rock), black and maroon mudstone (similar to those in the mudstone-siltstone unit), maroon chert, and semi-massive and massive sulphide.

A tentative interpretation of this unit is a flow breccia at the front of, and then overridden by the overlying mafic flow.

7.2.1.4 Massive Mafic Flow

This thick unit was initially mapped as anorthosite, as it consists almost entirely of fine-grained, equant plagioclase with <5% clinopyroxene. The rock is featureless and massive and has been named massive mafic flow because of the associated breccias.

7.2.1.5 Mudstone and Siltstone

Mudstone, with lesser siltstone, is the uppermost unit observed. The mudstone is dark green to black or, in a 200m thick horizon, alternating medium green and maroon. The siltstone is light beige and occurs in 5 cm to 30 cm beds. Bedding is otherwise faint to absent.

7.2.2 Metamorphism

Chlorite is the only prograde metamorphic mineral observed suggesting lower greenschist grade metamorphism.

7.2.3 Structure

Similar felsic volcanic rocks and mudstone-siltstone are repeated across several contacts throughout the mapped area. Regional USGS mapping of nearby stratigraphic units indicates contacts repeated by closely spaced anticlines and synclines, or folding, in nearby stratigraphic units rather than a history of alternating volcanism and sedimentation. The deposit horizon is rotated into an interpreted syncline east of the East Zone, also arguing for fold repetitions of the contact.

Foliations in the rocks are axial planar to the interpreted folds near contacts but tend to be more northerly away from contacts. It is suggested that these foliations record a later flattening that produced cross-folding in the deposit area.

7.3 Mineralisation

The mineral zone at Pickett Mountain is a volcanogenic massive sulphide deposit that strikes at approximately 057°. It has been traced by drilling approximately 900m along strike and to 750 vertical metres below surface. It consists of 4 primary lenses and several minor lenses that likely reflect the original formation of the mineralisation. It is stratabound and is hosted primarily by an intermediate to felsic lapilli tuff to volcanic breccia unit (Scully, 1988).

Primary minerals of economic interest are chalcopyrite, galena, and sphalerite intercalated with variable amounts of pyrite. Accessory minerals include tetrahedrite and minor arsenopyrite. There are two primary lenses of massive sulphide that have been discovered to date that have been subdivided into four lenses (W1, W2, E1, and E2). These vary from 0.5m to about 25m in horizontal width and with the highest base metal grades situated at or near the base of the massive sulphide lenses. The high-grade Cu-Pb-Zn sulphides are typically finely laminated and are overlain and in sharp contact with massive pyrite (Scully, 1988).

The high-grade sulphides typically include 45% to 60% pyrite, 15% sphalerite, 3% galena, and 4% chalcopyrite. There are also minor amounts of tetrahedrite, tennantite, arsenopyrite, magnetite, and barite. Laminations are typically 2 mm to 5 cm in thickness and are compositionally defined (Scully, 1988).

The West Lens is the most prominent massive sulphide lens discovered to date having been traced by drilling over a 300m strike length and to a vertical depth of 825m. Notably, it also is the highest grade of all lenses based on current and historic drilling and the most intense footwall alteration. The West Lens, especially along its eastern edge, includes fold repetitions and structurally stacked mineralisation in the hanging wall (previously interpreted as W2 Lens). Re-logging, which is in progress, has been completed and lithogeochemical data are supporting the updated interpretation. It is also likely that additional holes will need to be drilled to finalise the updated interpretation. As well, the West Lens lies directly on either felsic volcanic in the upper part of the zone and sedimentary rocks, in the lower part of the zone. This suggests that, the massive sulphides were deposited, both on top of the felsic volcanic rocks and in local, likely structurally-controlled sub-basins, with local structures, likely controlling mineralising hydrothermal fluid flow.

The East Lens (E1 and E2 Lenses) can be traced over a strike-length of close to 550m and to a maximum vertical depth of about 550m below the surface with the bulk of the zone above 400m. The QFP unit occurring below the East Lens, may have modified the orientation of the lens resulting in a locally shallower dip and may have propagated a fault or dislocation between the East and West Lenses. In addition, the QFP and a massive feldspar quartz porphyry unit appears to partially truncate the East Lens at its eastern extremity, occurring as a felsic dome intruding the upper rhyolite. The continuation of the favourable stratigraphy hosting the West and East Lenses, beyond the eastern limits of this intrusive cut-off, is unknown.

Longitudinal sections for the massive sulphide lenses, that are subject to the resource estimate, are depicted in Figure 7.6 and Figure 7.7.

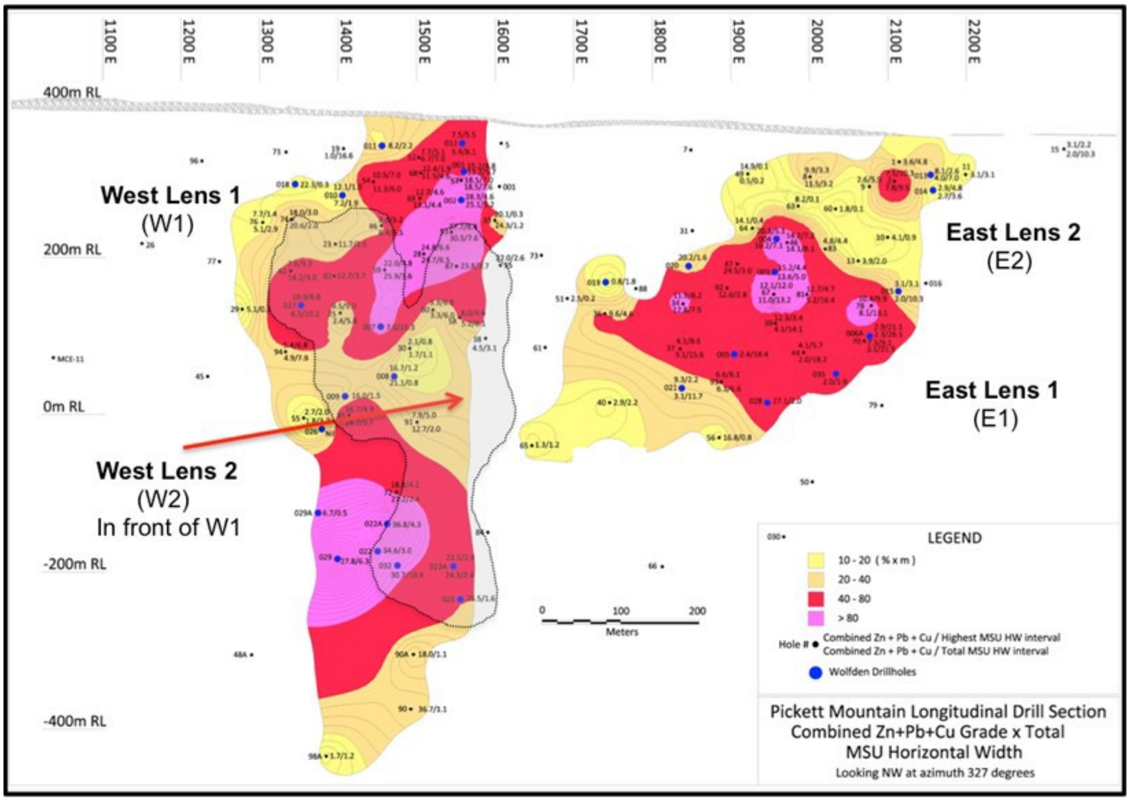


Figure 7.6 Longitudinal Section of the W1, E1, and E2 Massive Sulphide Lenses

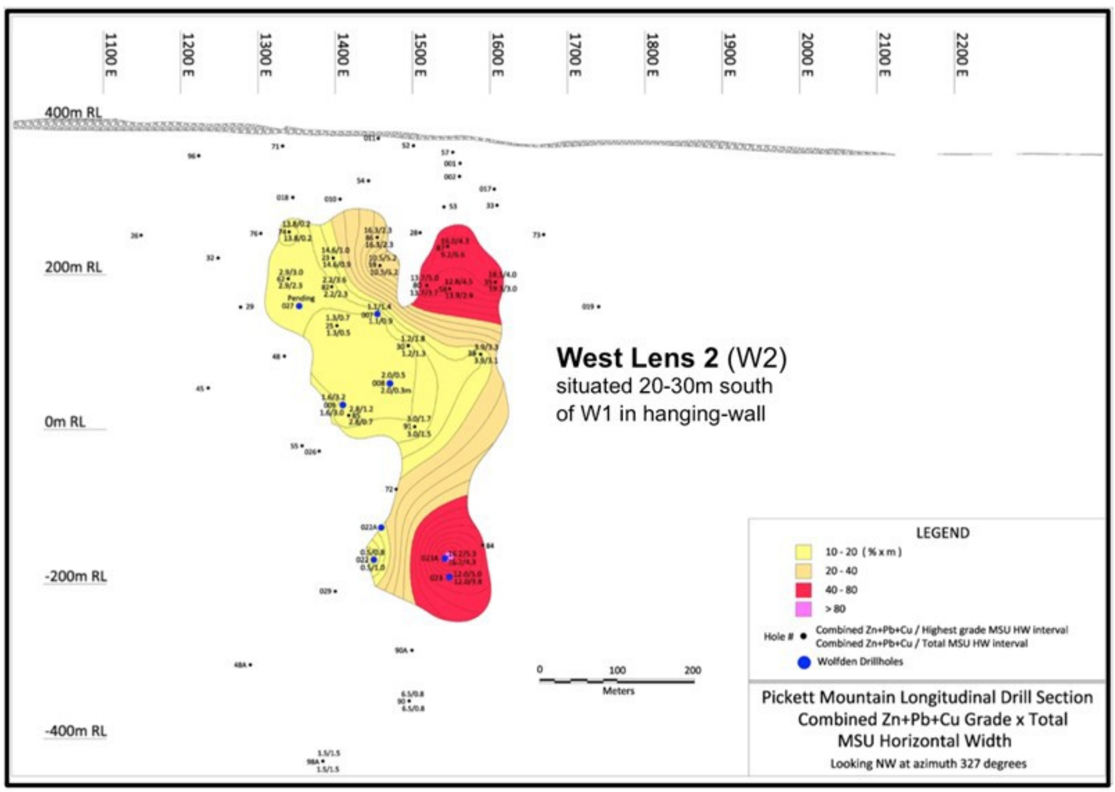


Figure 7.7 Longitudinal Section of the W2 Massive Sulphide Lens

The Footwall Zone (FWZ) occurs at the contact between the QFP and/or sediments and FWBX felsic units about 150m north of the trend of the West and East Lenses. First identified in the 2018 drill results, about 700m below the surface, it now appears that some of the historical drilling also intersected this zone. Notable intercepts from the 2018 and 2019 drill programs include 10.0% Zn, 5.0% Pb, 1.1% Cu, 396.9 g/t Ag, and 0.4 g/t Au (24.7% ZnEq) over 7.10m (PM-18-31) as well as 4.4% Zn, 2.2% Pb, 0.5% Cu, 62.7 g/t Ag, and 0.3 g/t Au (9.0% ZnEq) across 9.1m (PM-19-31a). The FWZ is generally narrow but can locally contain very high-grades. In a few drill holes, the FWZ consists of discrete zones of semi-massive to massive sulphide (as seen in PM-18-31 and PM-19-31A), while in others, it occurs as heavy disseminations, stringers, and bands of sulphide mineralisation. Stringer mineralisation along this time break, some 170m from the discovery hole, suggests potential for additional massive sulphide mineralisation between the discovery hole and the stringer zone. The FWZ is open both up and down plunge, and to the east, as depicted in the footwall longitudinal section. Further drilling to test extensions is warranted. This drill testing can be cost-effectively accomplished by extending a couple of the previous holes.

Table 7.1 tabulates the most significant intersections obtained from the 4 massive sulphide lenses based on historic drilling (Getty and Chevron) and by recent drilling completed by Wolfden.

Table 7.1
Significant Drill Intercepts from Historic Drilling and Wolfden Drilling (PM Series of Holes)

Pickett Mountain Massive Sulphide Zone Comprehensive Drill Results - Historical and Recent

Section	Zone	Hole #	From (m)	To (m)	Length (m)	Long HW (m)	Zn (%)	Pb (%)	Cu (%)	Ag (%)	Au (%)	Cu + Pb + Zn (%)	(Cu + Pb + Zn) * HW (%m)
1250E	MS-W1	29	279.04	279.65	0.45	0.28	3.50	1.30	0.26	8.57	0.34	5.06	1.40
1300E	MS-W1	76	162.89	168.30	5.41	2.89	1.67	0.57	2.88	38.24	0.36	5.12	14.79
1350E	MS-W1	55	457.60	462.77	5.18	3.71	0.99	0.33	0.56	8.96	0.18	1.88	6.99
1350E	MS-W1	62	212.59	219.91	7.31	3.00	8.59	4.55	1.09	76.88	0.87	14.23	42.69
1350E	MS-W1	74	158.20	161.54	3.34	1.98	13.53	5.11	1.95	167.80	0.98	20.58	40.75
1350E	MS-W1	94	316.43	332.25	15.82	7.89	2.71	1.08	1.09	44.55	0.73	4.89	38.61
1350E	MS-W1	PM-18-018	110.10	110.60	0.49	0.29	11.70	8.31	2.25	100.00	0.46	22.26	6.48
1350E	MS-W1	PM-18-027	254.99	279.40	13.20	10.21	6.47	2.72	1.41	96.92	0.56	10.60	108.28
1350E	MS-W1	PM-18-029A	612.46	613.42	0.95	0.70						Pending	Pending
1400E	MS-W1	23	197.92	201.32	3.40	2.45	6.32	3.22	2.12	65.37	0.71	11.66	28.61
1400E	MS-W1	25	314.55	321.47	6.92	5.78	1.35	0.33	0.72	10.74	0.24	2.39	13.83
1400E	MS-W1	82	254.20	259.10	4.90	3.66	7.22	2.85	2.64	115.21	0.95	12.71	46.53
1400E	MS-W1	85	398.50	411.21	12.71	5.70	8.81	4.04	1.88	92.74	1.02	14.72	83.88
1400E	MS-W1	98A	925.92	927.32	1.40	1.16	1.58	0.01	0.10	1.30	0.10	1.69	1.96
1400E	MS-W1	PM-18-010	123.60	126.89	3.30	1.91	5.02	1.63	0.50	40.10	0.30	7.15	13.68
1400E	MS-W1	PM-18-026	492.49	497.20	4.71	3.21						Nil	Nil
1400E	MS-W1	PM-18-029	657.60	668.15	10.55	6.34	19.32	7.24	1.24	206.36	1.28	27.80	176.37
1450E	MS-W1	54	112.93	121.30	8.37	6.01	7.76	2.27	1.23	44.25	0.71	11.26	67.65
1450E	MS-W1	59	194.28	211.99	17.70	3.60	14.98	7.80	3.14	150.75	1.19	25.92	93.30
1450E	MS-W1	72	526.77	530.69	3.92	2.44	18.10	8.54	0.56	210.05	1.13	27.20	66.30
1450E	MS-W1	86	172.20	180.20	8.00	5.53	3.04	1.11	2.23	35.78	0.59	6.38	35.30
1450E	MS-W1	PM-18-007	279.70	311.19	31.49	15.32	4.41	1.65	0.97	60.54	0.61	7.03	107.69
1450E	MS-W1	PM-18-008	342.29	344.70	2.41	0.79	16.78	3.98	0.37	68.38	0.53	21.13	16.78
1450E	MS-W1	PM-18-009	380.90	384.40	3.50	1.50	10.58	4.14	1.26	85.15	0.59	15.97	23.96
1450E	MS-W1	PM-18-011	56.59	59.59	3.00	2.15	4.22	1.42	2.60	34.26	0.54	8.24	17.75
1450E	MS-W1	PM-18-022	662.20	666.91	4.71	3.01	23.83	9.88	0.88	262.59	1.52	34.58	104.25
1450E	MS-W1	PM-18-022A	639.40	645.30	5.90	4.25	23.95	11.84	0.95	324.08	1.35	36.73	156.13
1500E	MS-W1	28	200.75	210.82	10.06	6.53	15.91	7.41	1.42	181.06	1.83	24.74	161.57
1500E	MS-W1	30	342.35	344.00	1.62	1.13	0.90	0.40	0.36	22.00	0.27	1.67	1.89
1500E	MS-W1	52	54.03	68.00	13.79	6.96	4.03	1.71	0.92	33.75	0.50	6.66	46.39
1500E	MS-W1	68	64.77	85.64	20.87	4.55	7.80	2.59	1.13	44.75	0.52	11.51	52.44
1500E	MS-W1	69	91.28	121.60	30.32	4.37	8.40	3.55	1.16	107.32	0.95	13.11	57.34
1500E	MS-W1	80	283.46	293.06	9.61	6.04	1.61	1.09	0.58	14.74	0.44	3.27	19.76
1500E	MS-W1	90	812.43	814.34	1.91	1.10	25.21	10.66	0.87	140.53	0.85	36.75	40.27
1500E	MS-W1	91	432.51	435.12	2.62	1.95	8.26	3.00	1.45	70.65	1.45	12.72	24.81
1500E	MS-W1	90A	761.76	763.13	1.37	1.05	12.50	4.75	0.77	93.24	0.79	18.02	18.96
1500E	MS-W1	PM-18-023	722.00	724.40	2.40	1.60	20.39	3.75	1.39	107.14	1.05	25.53	40.82
1500E	MS-W1	PM-18-023A	686.90	690.19	3.29	2.42	15.83	7.78	0.70	167.87	0.93	24.30	58.78
1550E	MS-W1	53	157.89	171.79	13.90	7.58	18.61	10.25	1.63	229.89	1.62	30.49	230.94
1550E	MS-W1	57	81.40	95.98	14.58	7.64	11.06	5.91	1.54	145.75	0.92	18.51	141.36
1550E	MS-W1	58	278.07	292.61	14.54	8.15	2.97	1.30	0.94	78.25	0.45	5.21	42.48
1550E	MS-W1	87	214.97	220.70	5.73	3.71	15.53	6.02	2.25	191.42	0.88	23.80	88.19
1550E	MS-W1	PM-17-001	84.25	92.20	7.95	5.69	7.88	3.83	1.51	104.01	0.85	13.23	75.25
1550E	MS-W1	PM-17-002	109.80	119.69	9.90	5.19	16.31	7.09	1.73	185.61	1.42	25.13	130.40
1550E	MS-W1	PM-18-012	37.30	48.70	11.40	8.14	3.63	1.43	0.83	34.84	0.30	5.89	47.94
1600E	MS-W1	33	156.88	158.44	0.76	1.25	13.60	9.78	0.90	186.76	1.01	24.28	30.25
1650E	MS-E1	65	413.30	416.59	3.29	1.16	0.77	0.26	0.32	4.86	0.53	1.35	1.57
1750E	MS-E1	36	275.97	282.30	6.33	4.52	6.11	2.46	1.07	63.61	0.72	9.64	43.60
1750E	MS-E1	40	385.66	388.90	3.20	2.22	1.54	0.59	0.77	0.00	0.55	2.90	6.43
1750E	MS-E1	PM-18-019	235.90	238.80	2.90	1.83	0.40	0.08	0.28	3.88	0.05	0.76	1.39
1850E	MS-E1	34	245.04	259.10	14.06	7.53	8.68	3.28	0.82	78.59	0.99	12.78	96.23
1850E	MS-E1	37	320.03	341.99	21.96	15.62	1.64	0.69	0.72	51.18	0.73	3.05	47.71
1850E	MS-E1	PM-18-020	194.60	197.80	3.20	1.65	13.15	5.34	1.70	124.66	1.14	20.20	33.24
1850E	MS-E1	PM-18-021	350.00	371.00	21.00	11.68	1.99	0.69	0.34	15.02	0.30	3.02	35.25
1900E	MS-E1	47	181.19	187.60	6.40	3.02	17.09	6.42	1.02	128.91	1.40	24.53	74.20
1900E	MS-E1	49	67.21	67.57	0.30	0.22	0.21	0.17	0.12	0.17	0.14	0.50	0.11
1900E	MS-E1	56	396.85	398.80	1.95	0.81	12.00	4.08	0.67	96.38	1.30	16.76	13.53
1900E	MS-E1	64	118.06	130.82	12.77	0.35	8.24	3.71	1.36	83.84	0.78	13.32	4.72
1900E	MS-E1	92	225.31	229.50	4.19	2.82	8.41	3.29	0.87	76.64	0.81	12.57	35.48
1900E	MS-E1	93	330.04	343.70	13.66	6.62	3.94	1.46	0.85	72.97	1.12	6.25	41.38
1900E	MS-E1	PM-18-005	278.11	323.91	45.80	18.39	1.30	0.51	0.59	24.04	0.43	2.41	44.28
1950E	MS-E1	39	236.52	268.70	32.15	14.09	2.31	0.90	0.85	22.46	0.44	4.06	57.26
1950E	MS-E1	46	161.85	171.91	9.30	8.08	9.58	3.66	0.79	86.27	0.72	14.04	113.34
1950E	MS-E1	67	172.66	254.12	68.28	13.20	6.78	3.04	1.20	50.22	0.65	11.02	145.65
1950E	MS-E1	PM-18-003	192.89	202.60	9.71	4.99	9.27	3.39	0.99	58.78	0.73	13.64	68.10
1950E	MS-E1	PM-18-004	170.61	180.89	10.29	7.12	10.96	4.06	1.23	117.31	0.96	16.25	115.76
1950E	MS-E1	PM-18-028	390.89	394.30	3.41	2.04	19.14	7.37	0.60	151.04	1.16	27.12	55.22

TABLE 7.1

SIGNIFICANT DRILL INTERCEPTS FROM HISTORIC DRILLING AND WOLFDEN DRILLING (PM SERIES OF HOLES)

(CONTINUED)

Section	Zone	Hole #	From (m)	To (m)	Length (m)	Long HW (m)	Zn (%)	Pb (%)	Cu (%)	Ag (%)	Au (%)	Cu + Pb + Zn (%)	(Cu + Pb + Zn) * HW (%m)
2000E	MS-E1	8	88.09	91.74	2.74	3.25	7.27	2.62	1.63	60.61	1.47	11.53	37.46
2000E	MS-E1	44	292.46	318.21	25.69	18.19	0.94	0.52	0.53	8.50	0.41	1.99	36.24
2000E	MS-E1	63	95.71	95.90	0.20	0.04	5.30	2.40	0.54	30.17	0.45	8.24	0.35
2000E	MS-E1	81	230.12	255.60	25.01	16.44	3.16	1.22	0.78	32.11	0.59	5.16	84.81
2000E	MS-E1	83	204.37	209.85	5.48	4.41	3.14	1.24	0.43	29.45	0.33	4.81	21.23
2050E	MS-E2	9	78.03	85.95	7.62	5.88	2.08	0.28	0.30	20.16	0.75	2.66	15.63
2050E	MS-E2	13	183.79	186.69	2.74	2.14	2.62	0.54	0.72	57.90	0.57	3.88	8.30
2050E	MS-E1	60	94.49	96.31	1.82	0.13	0.79	0.28	0.75	5.14	0.17	1.82	0.24
2050E	MS-E2	70	263.05	304.80	35.35	21.53	1.76	0.58	0.69	20.40	0.49	3.03	65.22
2050E	MS-E2	78	229.51	252.56	23.05	13.10	5.01	2.00	1.10	48.88	0.62	8.12	106.32
2050E	MS-E2	PM-18-006A	255.70	308.10	52.40	26.09	1.33	0.48	0.50	18.63	0.30	2.31	60.28
2100E	MS-E2	1	40.84	47.55	6.58	4.90	2.08	0.73	0.72	29.36	0.20	3.54	17.36
2100E	MS-E2	2	64.16	76.96	12.65	9.50	5.02	1.87	0.91	49.63	0.63	7.80	74.08
2100E	MS-E2	10	168.86	170.08	1.07	1.02	2.49	0.67	0.93	48.48	0.20	4.09	4.18
2150E	MS-E2	PM-18-013	59.10	68.50	9.40	7.01	2.38	0.83	0.75	32.17	0.41	3.96	27.78
2150E	MS-E2	PM-18-014	86.70	91.70	5.00	3.61	1.70	0.55	0.47	24.66	0.28	2.72	9.84
2100E	MS-E2	PM-18-015	229.00	245.50	16.50	10.33	1.10	0.42	0.52	18.58	0.27	2.04	21.08
2200E	MS-E2	11	53.04	57.45	4.41	3.06	2.15	0.56	0.42	30.43	0.41	3.13	9.58
1350E	MS-W2	62	199.33	205.13	5.80	2.37	1.47	0.50	0.94	20.53	0.72	2.91	6.90
1350E	MS-W2	74	151.27	151.64	0.01	0.22	9.30	3.65	0.84	82.96	0.62	13.79	3.03
1350E	MS-W2	PM-18-027	242.80	246.00	3.20	1.32						Pending	Pending
1400E	MS-W2	23	192.02	193.32	1.30	0.92	8.08	5.01	1.58	101.95	0.87	14.66	13.51
1400E	MS-W2	25	308.46	309.07	0.61	0.50	0.60	0.23	0.50	14.23	0.33	1.33	0.67
1400E	MS-W2	82	242.83	245.97	3.15	2.32	0.94	0.32	0.91	26.52	0.46	2.17	5.03
1400E	MS-W2	85	382.21	383.87	1.34	0.72	1.75	0.85	0.23	7.54	0.24	2.83	2.03
1400E	MS-W2	PM-18-009	369.60	376.40	6.79	2.92	0.78	0.25	0.53	16.53	0.23	1.56	4.56
1450E	MS-W2	59	167.19	181.66	14.47	2.88	6.16	2.75	1.58	88.43	0.62	10.49	30.17
1450E	MS-W2	86	162.00	164.59	2.59	1.76	11.18	3.86	1.28	85.01	1.03	16.31	28.63
1450E	MS-W2	PM-18-007	251.60	253.39	1.80	0.85	0.63	0.15	0.34	9.03	0.14	1.12	0.96
1450E	MS-W2	PM-18-008	331.88	332.88	0.98	0.33	0.96	0.36	0.73	17.60	0.17	2.05	0.67
1450E	MS-W2	PM-18-022	656.80	658.30	1.50	0.96	0.00	0.00	0.54	28.92	0.58	0.54	0.52
1500E	MS-W2	30	309.67	311.63	1.95	1.31	0.39	0.08	0.74	0.00	0.17	1.22	1.60
1500E	MS-W2	80	217.63	223.87	6.24	3.73	8.15	3.90	1.64	110.32	1.00	13.68	51.01
1500E	MS-W2	91	413.00	415.10	2.10	1.51	1.00	0.37	1.65	9.25	0.38	3.02	4.56
1550E	MS-W2	58	212.63	218.45	5.82	2.93	8.08	3.31	2.40	123.54	1.12	13.79	40.43
1550E	MS-W2	87	151.27	163.37	12.10	6.63	5.54	2.35	1.28	56.94	0.56	9.17	60.81
1550E	MS-W2	PM-18-023	661.14	667.06	5.92	3.79	7.48	3.21	1.31	62.66	0.73	12.00	45.44
1550E	MS-W2	PM-18-023A	646.60	652.60	6.00	4.33	10.19	4.69	1.28	52.93	0.49	16.17	70.01
1600E	MS-W2	35	210.65	215.10	4.39	3.03	12.82	5.65	0.86	87.84	0.83	19.34	58.61
1600E	MS-W2	38	327.57	331.55	3.98	3.09	2.26	0.88	0.73	32.37	0.45	3.87	11.98

The historical drill results included in Table 7.1 were generated between 1979 to 1989 by Getty Mining Company and Chevron Resources. The historic drill core samples were cut in half using a diamond saw or core splitter and sent to Skyline Laboratories in Tucson, Arizona for analyses. Copper, lead, and zinc were analysed utilising atomic absorption spectrometry (AA) while gold and silver were analysed utilising the fire-assay technique. High-grade copper, lead, and zinc assays obtained by AA were checked routinely utilising wet chemistry techniques. Wolfden is not aware of theQA/QC programs undertaken with these results, if any. The historical data, which does include most of the drill core in storage, does not include the original assay certificates. The historical results were compiled by Wolfden utilising original drill logs, drill sections, working files and reports, and databases prepared by the former owners of the Property at that time and subsequently acquired by Wolfden. Wolfden has not independently verified the historic results, although some corrections (non-material) have been identified since the January 7, 2019 Mineral Resource report and are now included in an updated historical drill hole table on the Company's website, readers should also see the Company's AIF filed on SEDAR and dated April 28, 2020. Holes drilled by Wolfden begin with PM-17, PM-18, PM-19, and now PM-20.

8.0 Deposit Type

VMS deposits are the product of hydrothermal vents on the sea floor that form syngenetically with active volcanism and/or plutonism. They form at or just below the sea floor as a product of the discharge of high temperature, seawater-dominated hydrothermal fluid. There are six main elements typically present and are considered essential for the formation of VMS hydrothermal systems and their associated base metal deposits.

This deposit type is typically an accumulation of massive to semi-massive sulphides that are syngenetic, stratabound, and in part strataform. They usually consist of two parts: a concordant massive sulphide lens and an underlying discordant vein-type sulphide stringer or stock-work zone that is within a footwall alteration zone (Figure 8.1). In some cases, the stringer zone extends into the hangingwall as well, which appears to be the case at Picket Mountain. This continuation of the stringer zone into the hangingwall may indicate the continuation of the hydrothermal system and could represent additional exploration opportunities.

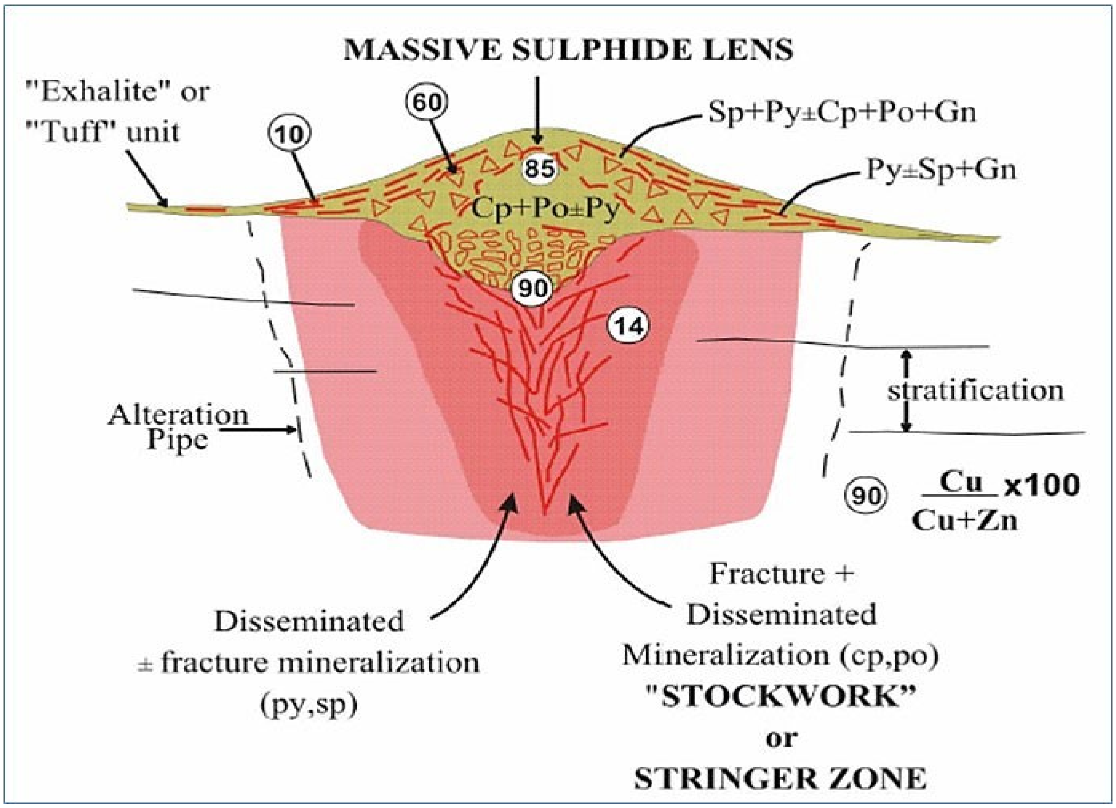


Figure 8.1 Idealised VMS Deposit Showing a Strataform Lens of Massive Sulphide Overlying a Discordant Stringer Sulphide Zone within an Envelope of Altered Rock (Alteration Pipe)

Base metal zonation is indicated by numbers in circles with the highest numbers being Cu-rich and the lower numbers more Zn-rich (Py = pyrite, Cp = chalcopyrite, Po = pyrrhotite, Sp = sphalerite, and Gn = galena (**Source**: Modified from Gibson, 2005).

VMS deposits are the product of hydrothermal vents on the sea floor that form syngenetically with active volcanism and/or plutonism. They form at or just below the sea floor as a product of the discharge of high temperature, seawater-dominated hydrothermal fluid, as illustrated in Figure 8.2. There are 6 main elements typically present and are considered essential for the formation of VMS hydrothermal systems and their associated base metal deposits (Gibson, et al., 2007):

1. A heat source is required to drive the hydrothermal system. This may be syn-volcanic, high level intrusions.

2. There is a high-temperature reaction zone that forms through the reaction of seawater with volcanic and sedimentary strata that result in the leaching of metals from these rocks.

3. There need to be deep penetrating syn-volcanic faults that allow the recharge and discharge of the metal-bearing hydrothermal fluid.

4. The interaction of the ascending high-temperature fluids and mixing with ambient seawater results in footwall and hanging wall alteration zones.

5. Massive sulphide deposits form at or near the seafloor due to interaction with the overlying cold seawater and the ascending hydrothermal fluids resulting in the precipitation of dissolved metals.

6. Distal products, usually exhalites, form due to the contribution of the hydrothermal system to background sedimentation.

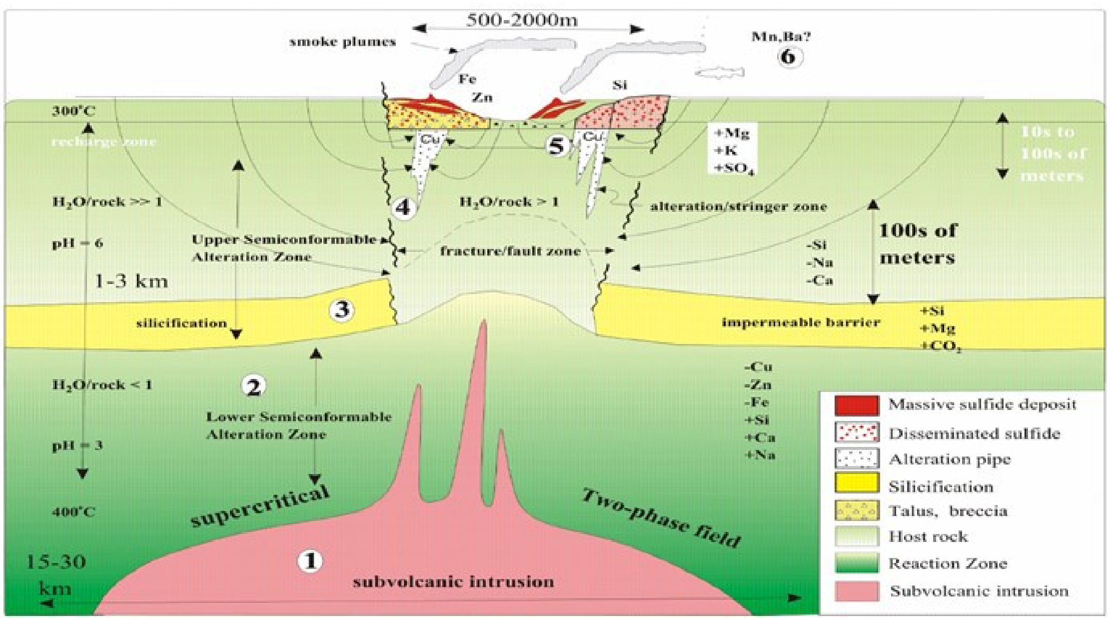


Figure 8.2 Schematic Illustrating the Relationship between Subvolcanic Intrusions, Subsea-Floor Alteration, Syn-Volcanic Faulting and the Generation of VMS Deposits

(**Source:** Modified after Galley, 1993 and Franklin, et al., 2005)

VMS deposits typically form in a diverse spectrum of volcanic-sedimentary environments that range from those dominated by flow, volcaniclastic, and or sedimentary rock types. Any of the three end members may be dominant, but what is characteristic for exploration purposes are the overall characteristics listed above.

9.0 Exploration

Since acquiring the Property in November 2017, Wolfden has completed an airborne geophysical survey (VTEM^{TM}), ground Time-Domain (TDEM) electromagnetic surveys, borehole electromagnetic surveys, ground induced polarization surveys (IP), as well as geological mapping. A summary of each component of the exploration program is presented in this section.

9.1 Airborne Geophysical Survey

During May 3 to May 24, 2018, Geotech Ltd. carried out a helicopter-borne geophysical survey over the Pickett Mountain project situated near Patten, Maine.

The geophysical surveys consisted of helicopter-borne electromagnetics (EM) using the versatile time-domain electromagnetic (VTEM^{TM}) plus a system with Full-Waveform processing. Measurements consisted of Vertical (Z) and In-line Horizontal (X) components of the EM fields using an induction coil and a horizontal magnetic gradiometer using two caesium magnetometers. Ancillary equipment included a GPS navigation system and a radar altimeter. A total of 2,853 line-kilometres of geophysical data, covering an area of 397 square kilometres, were acquired during the survey, as illustrated in Figure 9.1.

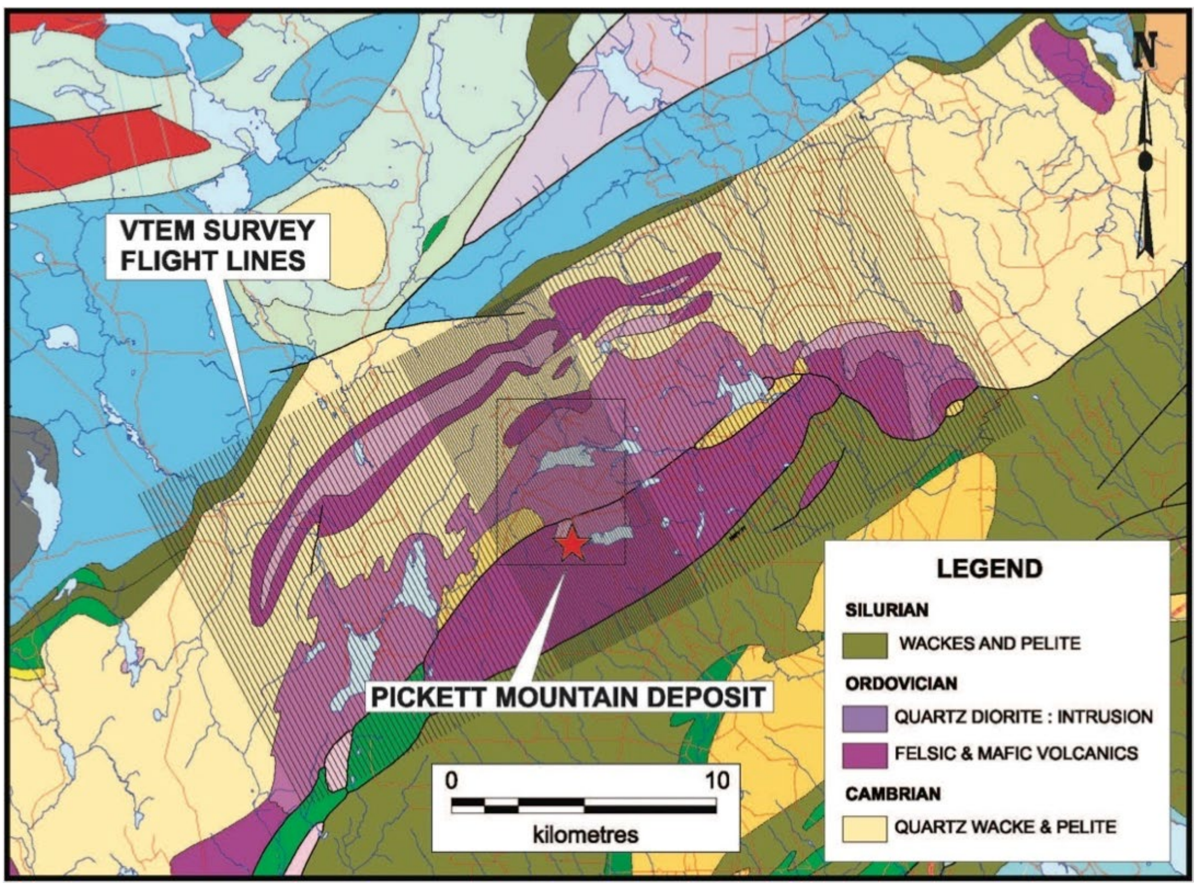


Figure 9.1 VTEM™ Airborne Geophysical Survey Coverage

Data QC/QA and preliminary data processing were carried out daily during the acquisition phase of the project. Preliminary and final data processing, including generation of final digital data and map products, were undertaken from the office of Geotech Ltd. in Aurora, Ontario.

Follow-up investigation is warranted where a clear indication of a bedrock conductor has been interpreted. The highest priority targets are reserved for those with a high conductance that can often indicate sulphide mineralisation. Lower priorities are assigned for lower conductance; however, it is important to note that lower conductance can be associated with economic mineralisation that is located deeper below the surface as well as less to non-conductive material, such as the dominant zinc-bearing mineral, sphalerite.

The VTEM™ survey delineated several EM anomalies across the Property, including prominent anomalies over the known Pickett Mountain deposit. According to calculated TAU values, most of the conductors defined by the survey correspond to low to moderate conductive targets. Additionally, most of the conductors delineated are associated with high magnetic gradient zones. Ground geophysical surveys were recommended to follow-up on the results of the airborne VTEM™ survey.

9.2 Ground Infinitem XL Time Domain Electromagnetic Survey

A ground TDEM survey was completed on the Pickett Mountain Property from April 16 to April 27, 2018 by Abitibi Geophysics, based out of Val d'Or, Quebec. The purpose of the survey was to establish an electromagnetic signature over the known Pickett Mountain massive sulphide deposit and to look for similar EM signatures in the locale of the known deposit or elsewhere on the Property that might be reflecting the presence of additional massive sulphide lenses or deposits.

A total of 18 lines were surveyed for total survey coverage of 21.4-line kilometres with readings being collected every 25m and 50m on the grid lines. The survey utilised the InfiniTEM XL configuration reading the X, Y, Z, B-field, and dB/dt components on lines spaced 100m and 200m apart, as illustrated on Figure 9.2.

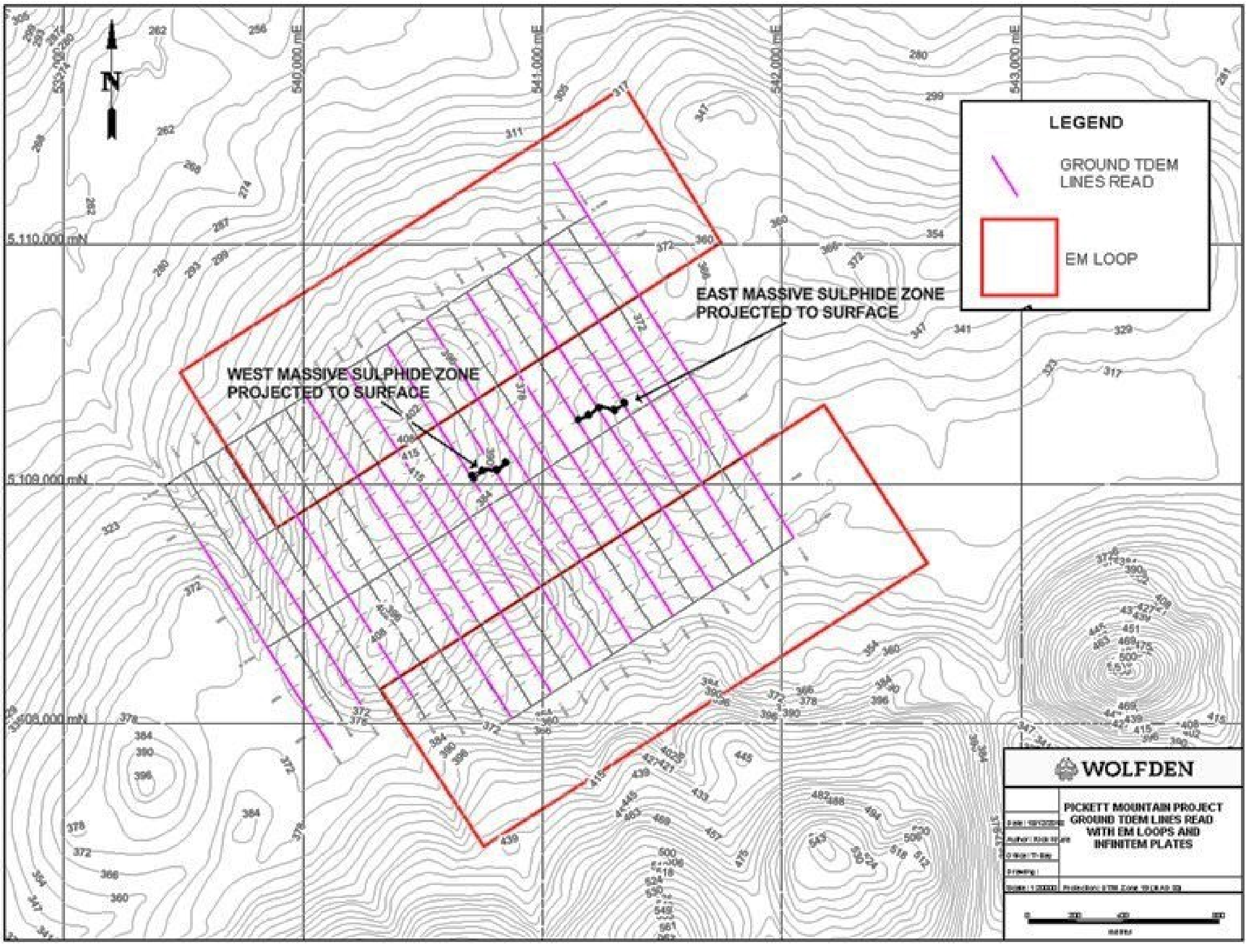


Figure 9.2 Ground InfiniTEM Time-Domain EM Survey Coverage

The surface probe utilised was the ARMIT 3 axis B-field and dB/dt sensor and the receiver used for the survey was the EMIT SMARTem 24. The survey employed 2 Tx Terrascope transmitters for a total of 36 kW.

The ground TDEM survey delineated several conductors or conductive plates, as illustrated on Figure 9.3. The conductive plates were modeled utilising the Maxwell™ software. Maxwell™ automates the handling of large data sets with inversion and forward modeling of conductive plate targets. Both the East and West Lenses of the Pickett Mountain massive sulphide deposit elicited prominent conductive responses and are reflected by coincident conductive plates. In addition to these conductors, 3 additional significant bedrock conductors were delineated by the survey.

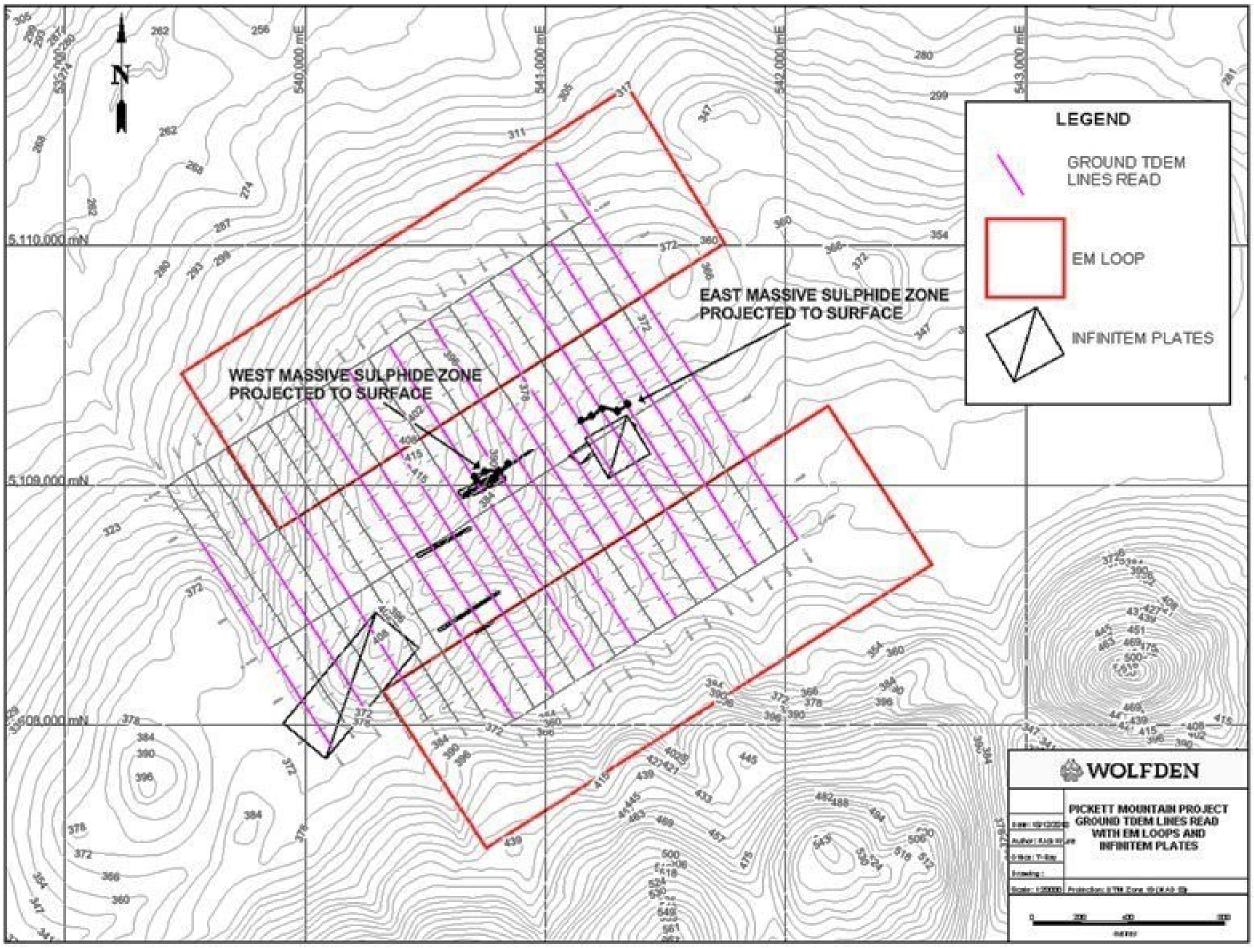


Figure 9.3 Conductors Defined by the Ground InfiniTEM Time-Domain EM Survey

9.3 Borehole Infinitem XL Time Domain Electromagnetic Survey

Borehole EM surveys were also completed by Abitibi Geophysics on the Pickett Mountain Property in 2018. The surveys were carried out in 2 phases; the first occurred in April 2018 and involved the surveying of 12 drill holes, while the second occurred in August 2018 and comprised the surveying of 3 drill holes, as illustrated on Figure 9.4. The purpose of the surveys was to help trace the depth and down-plunge extension of the known massive sulphide lenses, to detect and characterise deeply buried conductors potentially reflective of new massive sulphide mineralisation, and to identify additional targets for future exploration.

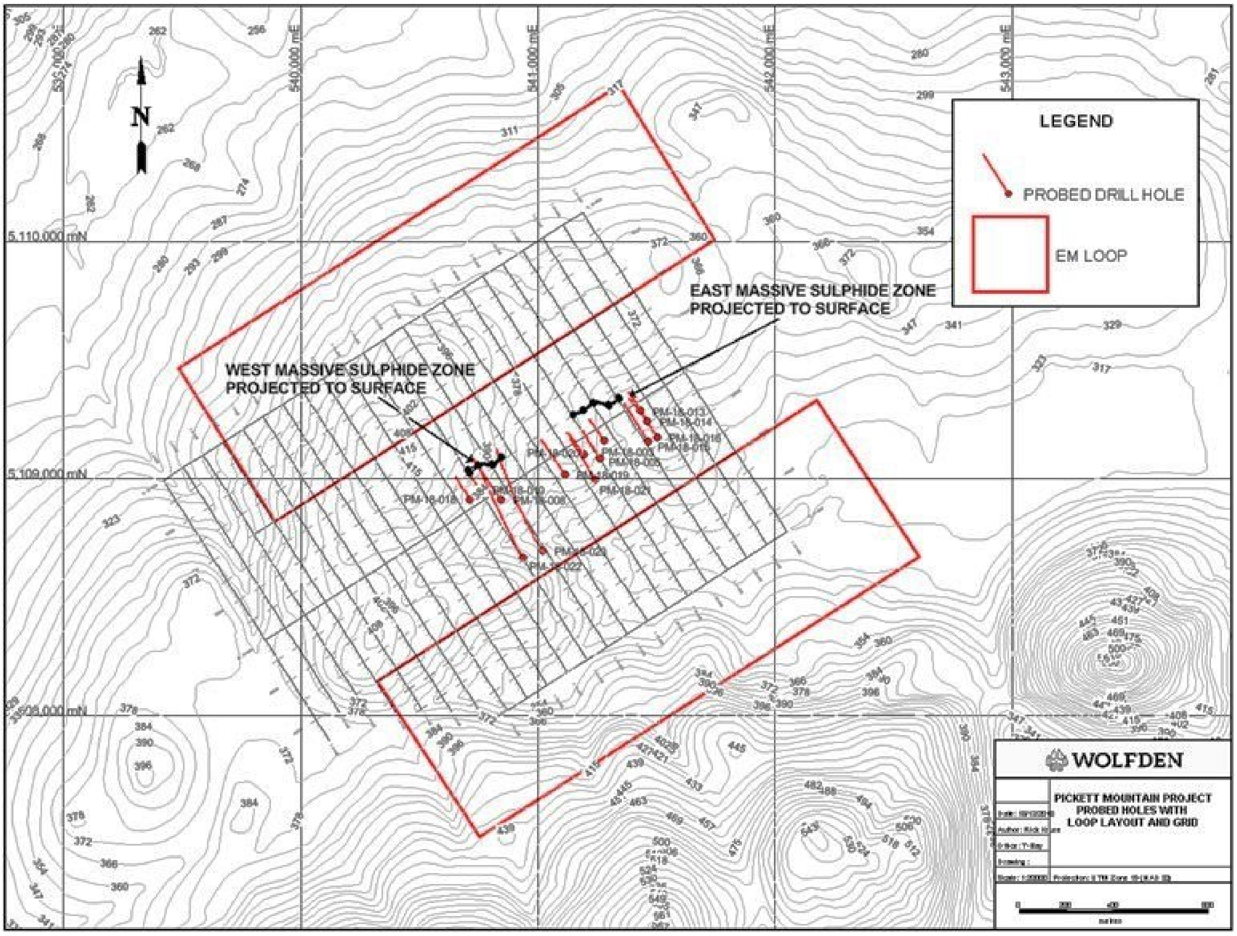


Figure 9.4 Holes Surveyed by InfiniTEM XL TDEM Survey

The borehole surveys utilised the InfiniTEM XL configuration and measured the secondary magnetic B-field as well as the axial (A) and orthogonal (U and V) components with the DigiAtlantis™ sensor. Reading intervals for the borehole surveys were at 5m and 10m intervals down-the-hole.

The surveys utilised 2 TX TerraScope 600V, 25A, 18 kW transmitters powered by a Voltmaster 12 kW generator. A DigiAtlantis™ receiver and probe was also employed during the survey.

Modeling of the electromagnetic data by the Maxwell™ software delineated several conductive plates, as shown in Figure 9.5. The known configuration of the West Lens is well reflected by conductive plates over much of its extent while the East Lens exhibits fewer conductive plates. The East Lens may have been subjected to more structural complexities, including folding or offsets. Notably, both the East and West Lenses show potential for expansion from their known extent as reflected by the location of the conductive plates.

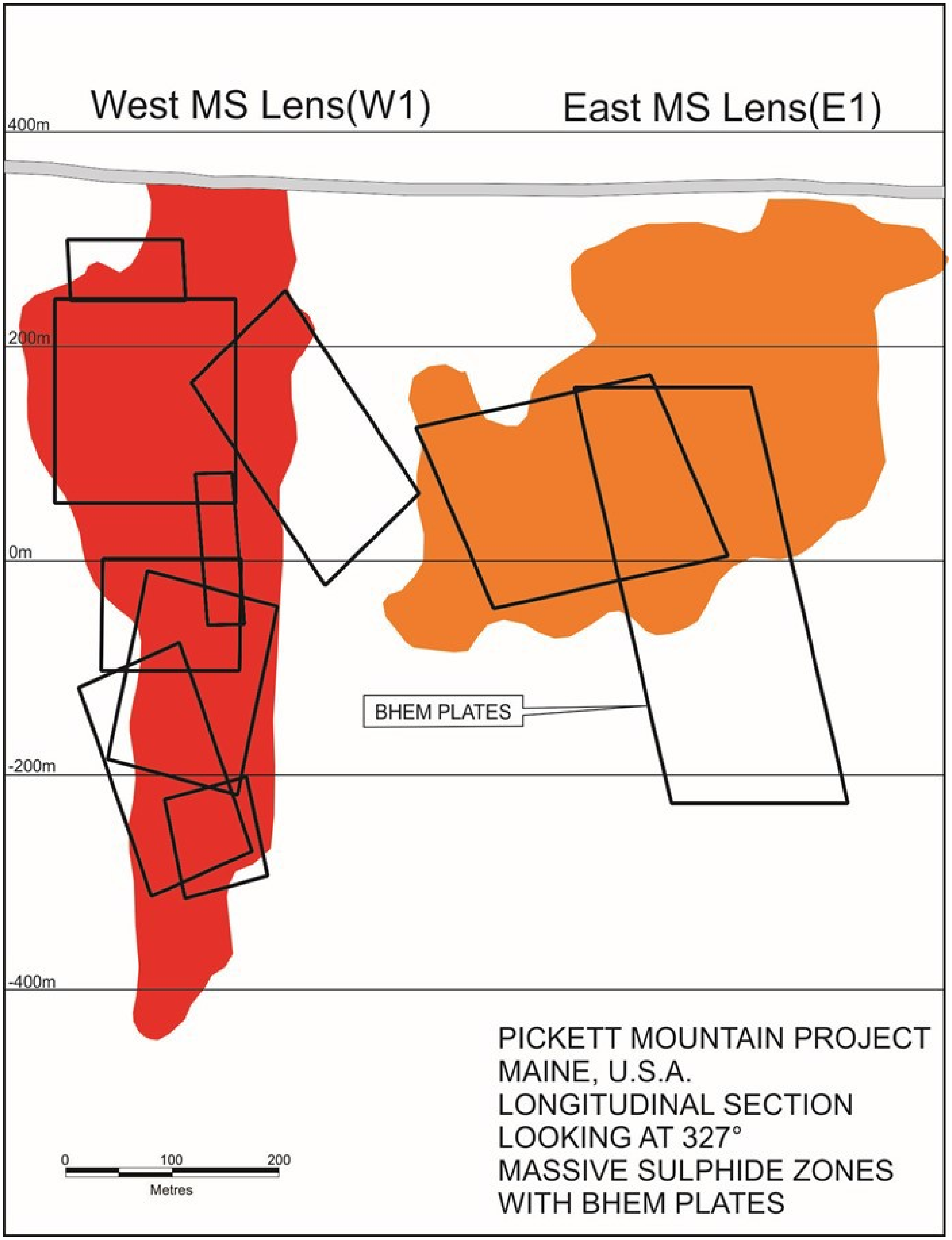


Figure 9.5 Conductive Plates Defined by the down-the-hole InfiniTEM XL TDEM Survey

9.4 OreVision Induced Polarization Survey (IP)

Abitibi Geophysics completed an OreVision Time Domain Resistivity/Induced Polarization survey on the Pickett Mountain Property during the period of March 24 to April 2, 2018. The purpose of the survey was to identify geophysical signatures over mineralised zones and to define and prioritise targets for future mineral exploration. In all, the survey totaled 24.75-line-kilometres, comprising the surveying of 15 grid lines spaced 100m apart.

The IP transmitter utilised was the IRIS Instrument TIPIX with a maximum output of 2.2 kW employing a Honda 2000 VA as a power supply. The receiver employed during the survey was an IRIS Elrec-PRO with integrated SwitchPRO featuring 10 input channels. Electrode spacing or “a” spacing was 50m and readings were taken from “n” = 1 to 20.

Detailed interpretation of the pseudo sections reveals a number of chargeability sources or anomalies (Figure 9.6). The strongest chargeable sources were delineated over the East and West Lenses of the Pickett Mountain deposit. Other chargeability anomalies are located primarily to the north of the Pickett Mountain deposit and immediately to the south of it. A prominent chargeability low occupies the southern portion of the survey grid and the northern anomaly may represent a lens in the footwall, 180m from the East Lens.

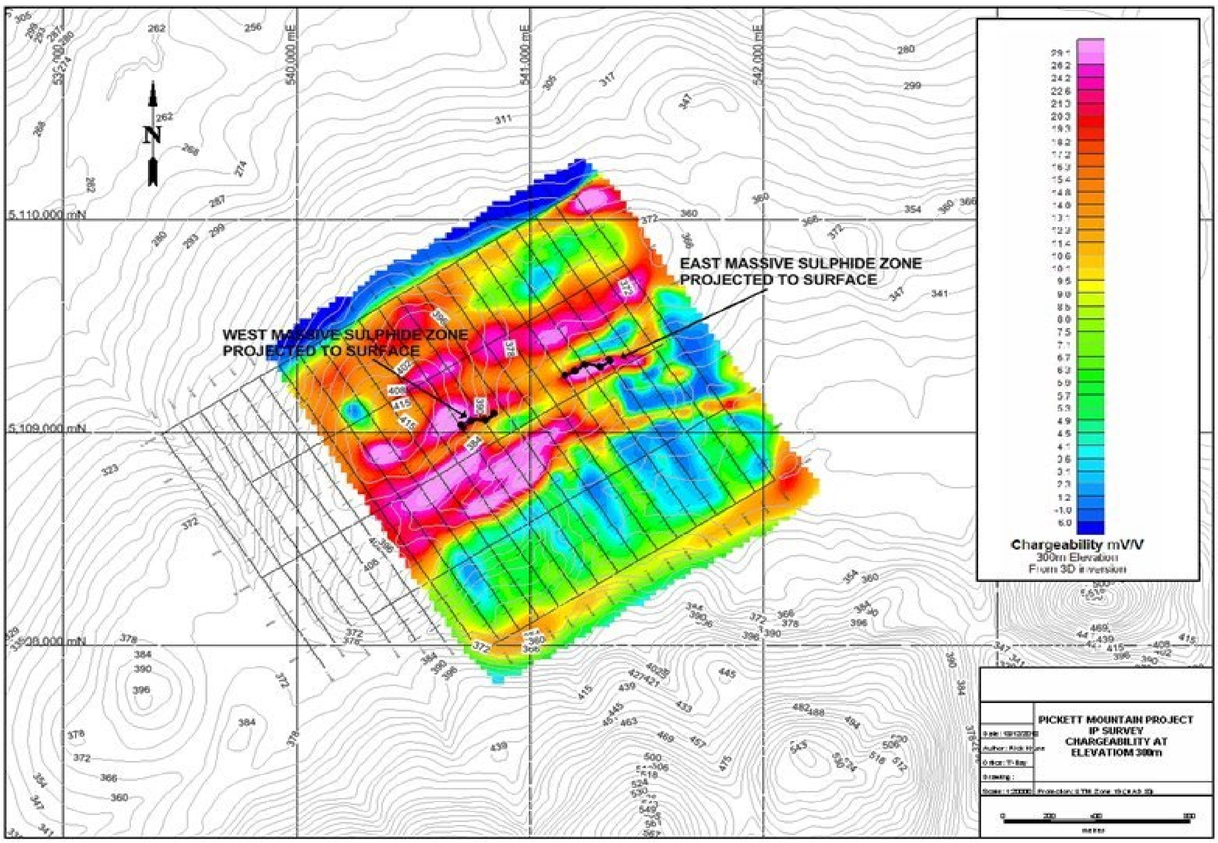


Figure 9.6 Chargeability Anomalies Defined by the OreVision IP Survey

Resistivity anomalies were also interpreted by studying the pseudo sections. As was the case with chargeability, both the East and West Lenses of the deposit were manifested by anomalous responses, in his case deep resistivity low trends, reflecting massive sulphide mineralisation (Figure 9.7). A broad area characterised by low resistivity located immediately to the south of the West Lens is thought to be reflecting the presence of sedimentary rocks.

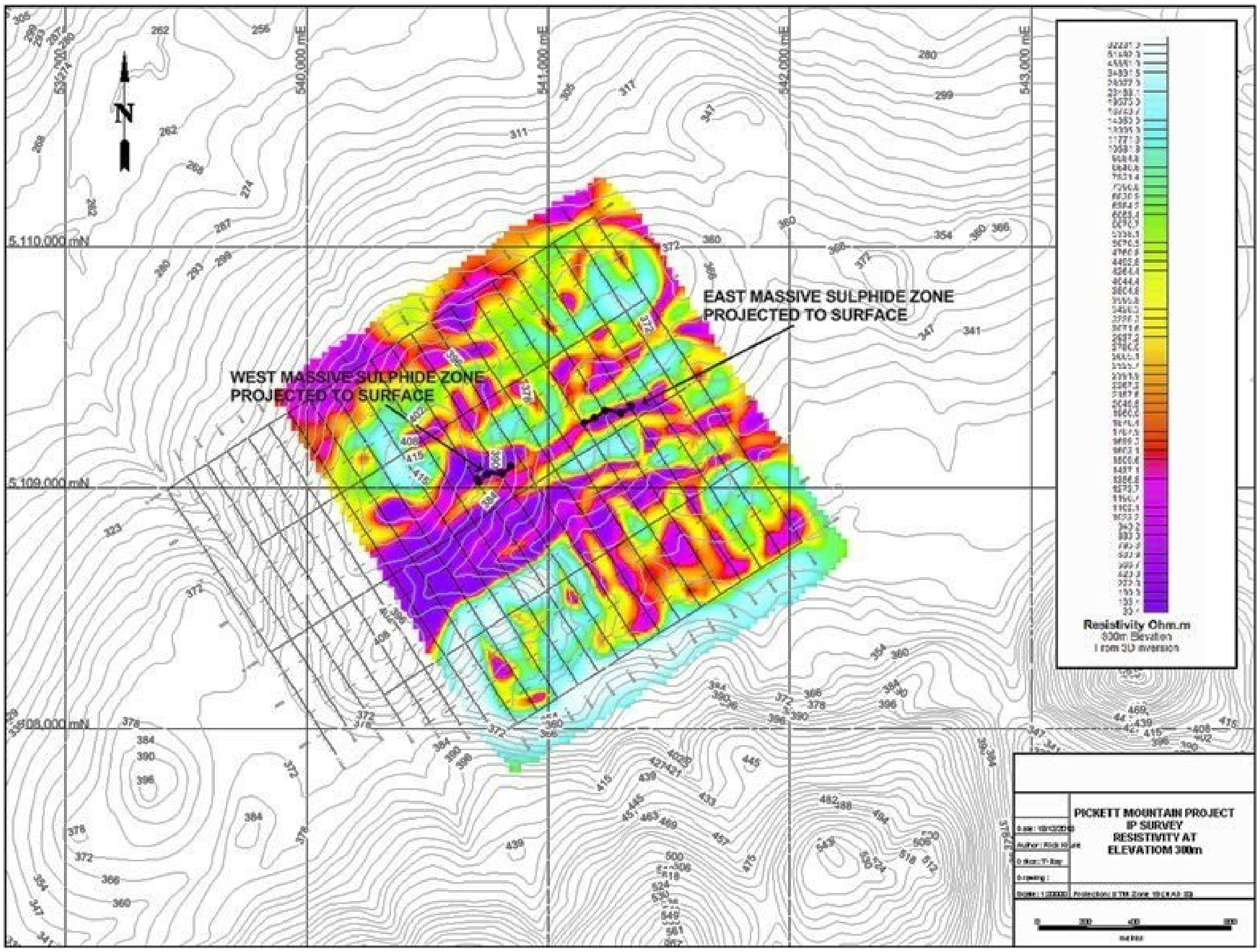


Figure 9.7 Resistivity Anomalies Defined by the OreVision IP Survey

9.5 Wolfden 2019 Exploration

The primary focus of the 2019 drill program (3,539m) sought to test the expansion potential of the Footwall Zone discovered in late 2018 in hole PM-031. The 2019 drill results completed on the West and East Lens were limited and considered to have no material impact on the 2019 mineral resource statement and, therefore, were not included in the updated 2020 Mineral Resource statement.

Drilling results yielded 7.1m at 24.7% ZnEq. The first wedge hole yielded 9.1m at 9.0% ZnEq. Additional deep drilling and other wedge holes were lost by the drillers and the program was terminated prematurely until a suitable crew could be assembled. Subsequent re-interpretation of the geology in this area has progressed and these ideas will be further tested in the next drill program. Similarly, deeper drilling to test the expansion potential of the West and East Lenses at depth, based on new structural interpretations, are priorities for the next drill program.

Other components of the 2019 program included geological mapping (see Section 7.0 – Geological Setting and Mineralisation) trenching, ground geophysical surveys, borehole EM surveys, whole rock geochemistry, and relogging of historic and Wolfden drill holes. Collectively, these surveys continue to suggest that the deposit and surrounding area holds potential for the expansion know mineralisation and the discovery of other massive sulphide lenses. Many of the historic drill holes in the area (off of the main horizon) contain broad intervals of highly anomalous Zn-Pb values within strongly altered volcanic rocks, similar to those of the Pickett Mountain deposit. An additional large-loop EM geophysical survey was completed during the fourth quarter that identified new drill targets along trend of the East and West Lenses that will be followed up in 2020 with additional ground surveys and diamond drilling.

Numerous quality targets were defined by Wolfden's VTEM airborne geophysical survey. Work on these targets included mapping, whole-rock geochemistry, and soil sampling that collectively, yielded compelling results. These targets have never been previously identified or drilled and Wolfden plans to be in a position to drill test these targets in 2020.

9.5.1 Borehole InfiniTEM XL Time Domain Electromagnetic Survey

Borehole EM surveys were completed by Abitibi Geophysics at Pickett Mountain the fall of 2019. In all, 13 drill holes were surveyed. The purpose of the survey was to assist in tracing the extent of massive sulphide mineralisation by the detection of conductors, situated along strike or down-plunge from known mineralisation. Such conductors may reflect the presence of extensions to known mineralisation, or new massive sulphide lenses.

Modeling of the electromagnetic data by Maxwell™ software delineated several conductive plates, as is illustrated on an older version of the longitudinal section below (Figure 9.8). Two conductors plates are located below the extent of massive sulphide at the East 1 Lens and are suggestive of expansion potential, below the 0m elevation (approximately 400m below surface). The upper-most plate was derived from a survey of hole G-066 and the lowermost plate was determined by a survey of hole G-137. These two conductive plates correlate well with the massive sulphide mineralisation in hole G-137A (5.54 ZnEq over 6.45m horizontal width) and supports the extension of mineralisation to hole G-056 (20.8% ZnEq over 0.8m horizontal width) a distance of approximately 175m, along an area where no holes were drilled. Additionally, a conductive plate, seen from hole G-048A, situated at depth associated with a high grade portion of the West Lens, suggests potential for additional mineralisation along trend to the west of the known deposit and in an area that has seen only wide spaced holes. Additional drilling is clearly warranted to test these targets and borehole surveying of the deepest hole, G-098A, so that some Wolfden can gain some insight into the depth extension of the West Lens.

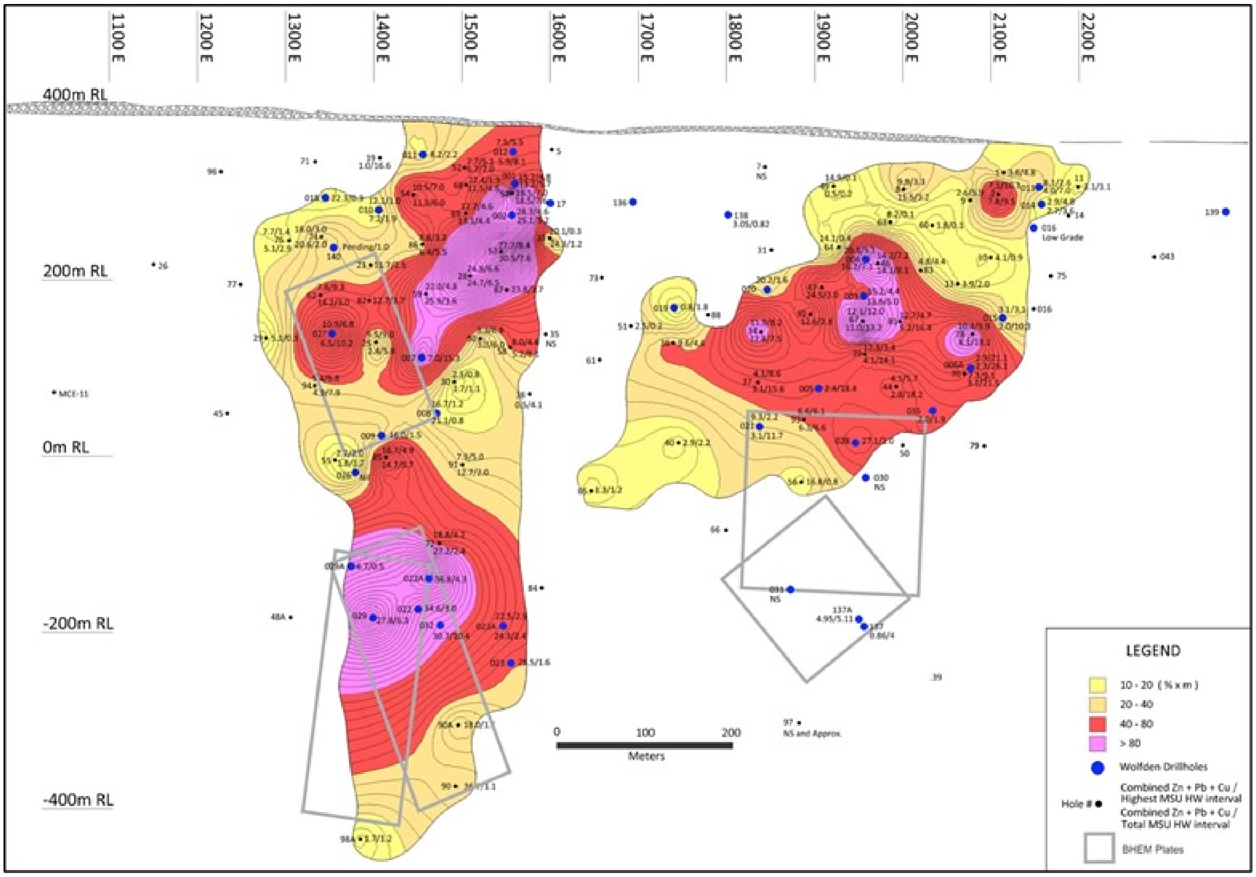


Figure 9.8 Longitudinal Section Depicting Borehole InfinitTEM XL Time Domain EM Survey, Conductive Plates

9.5.2 Ground InfintiTEM XL Time Domain Electromagnetic Survey

Ground TDEM surveys were also completed by Abitibi Geophysics at Pickett Mountain during October 2019. In all, 29 km of TDEM survey work was completed. The purpose of the survey was to detect conductors over untested stratigraphy, bearing similar electromagnetic signatures exhibited by the known East and West Lenses. The survey utilised the InfiniTEM XL configuration, reading the X, Y, Z, B-field, and dB/dT components, on lines spaced 100m apart.

The 2019 TDEM survey successfully delineated 4 target areas comprising linear sets of conductors (Figure 9.9). Target area A appears to be associated with the easterly extension of the main Pickett Mountain horizon and has been tested in part by 7 historic drill holes. Target areas B, C, and D are new target areas situated away from the known massive sulphide deposits, on portions of the property that have not been tested by drilling. All targets, based on preliminary interpretation, are classified as moderate in strength. Follow-up work is required and warranted to further characterise the nature and cause of these anomalies.

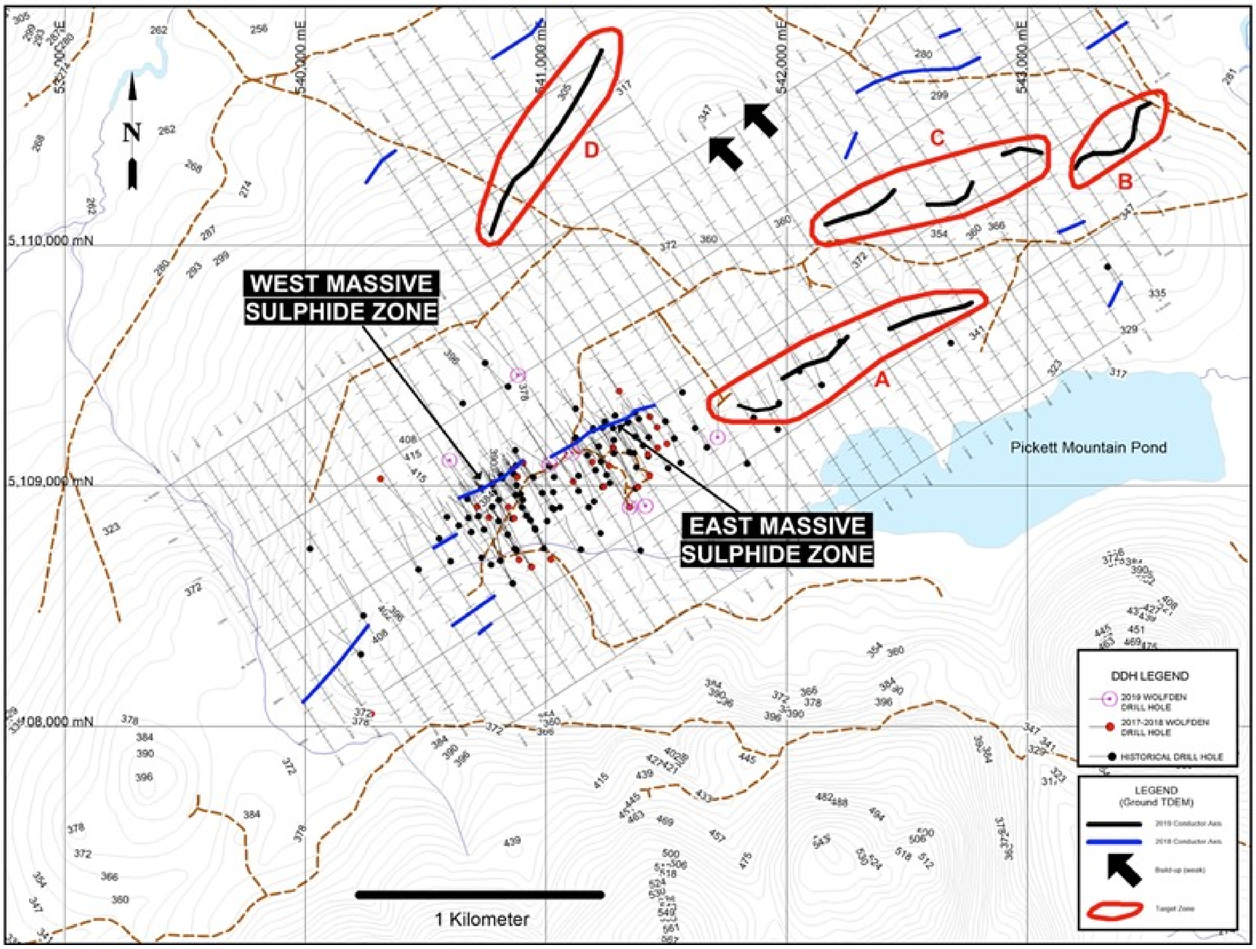


Figure 9.9 Ground InfiniTEM XL Time Domain EM Survey, Conductors

9.5.3 Gravity Survey

A gravity survey was also completed during 2019 by Great Lakes Exploration, based out of Menominee, Michigan. The purpose of the survey was to establish a density signature for the East and West Lenses and to outline areas of similar density elsewhere in the known deposit locale, in efforts to locate prospective targets for the discovery of additional massive sulphide lenses.

In all, 20 line-kilometres of gravity surveying was completed utilising a La Coste and Romberg Model D gravity meter. Stations were read at 25m and 50m intervals on grid lines spaced 100m apart. Station elevations were established with a high resolution Trimble RTK GPS system, with a base and rover and were processed for maximum accuracy.

Raw gravity data was processed in both Gravmaster® and Oasis Montage® software and both simple and complete bouguer gravity were calculated. Simple bouguer data (density of 2.67) was gridded to produce total, residual, and vertical derivative plots and line profiles were created from the total gravity.

The residual plot reveals gravity anomalies associated with the East and West Lenses, with amplitudes of 0.6 and 0.5 mGals, respectively. In all, 7 residual gravity anomalies were defined, including anomaly 2, a build-up target defined at the end of the survey line. Anomaly 2 yields an amplitude of +0.6 mGals and has only been partially established and will require additional gravity work to fully define its extent. This gravity anomaly may also be associated with a prominent conductor, located immediately to the northeast of it. The residual gravity anomalies defined by the survey are illustrated in Figure 9.10, below.

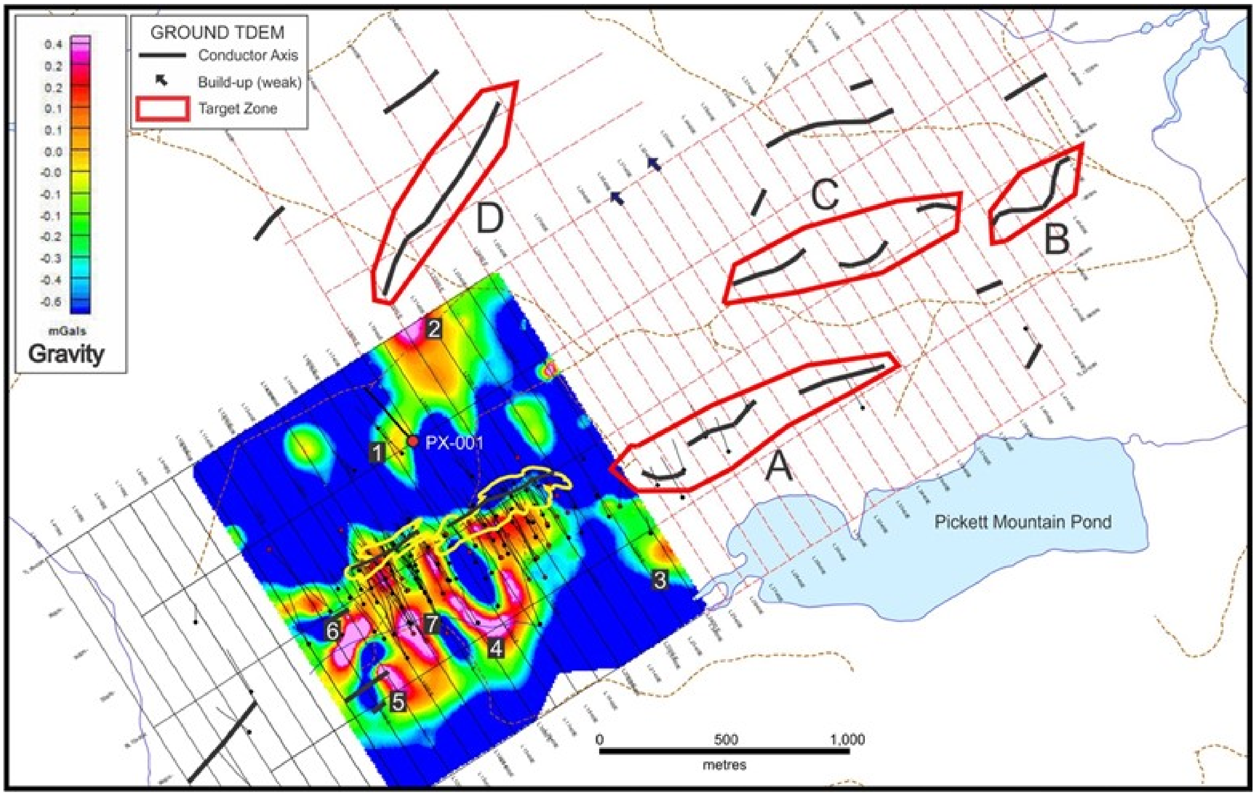


Figure 9.10 Residual Gravity Anomaly Map

9.5.4 Magnetics Survey

A ground magnetic survey was completed in 2019 at Pickett Mountain by GeoXplore Surveys Inc. of Bathurst, New Brunswick. The survey in total comprised 80 line kilometres-readings were collected at 25m intervals on grid lines spaced 100m apart. The magnetic survey employed a Scintrex Envi-Mag VLF instrument.

The total field magnetic map, shown in Figure 9.11, below, is useful to assist in determining rock types and is particularly helpful in areas of poor outcrop. Magnetic surveys can also assist in characterising alteration patterns (magnetite rich or magnetite poor) and for gleaning structural information.

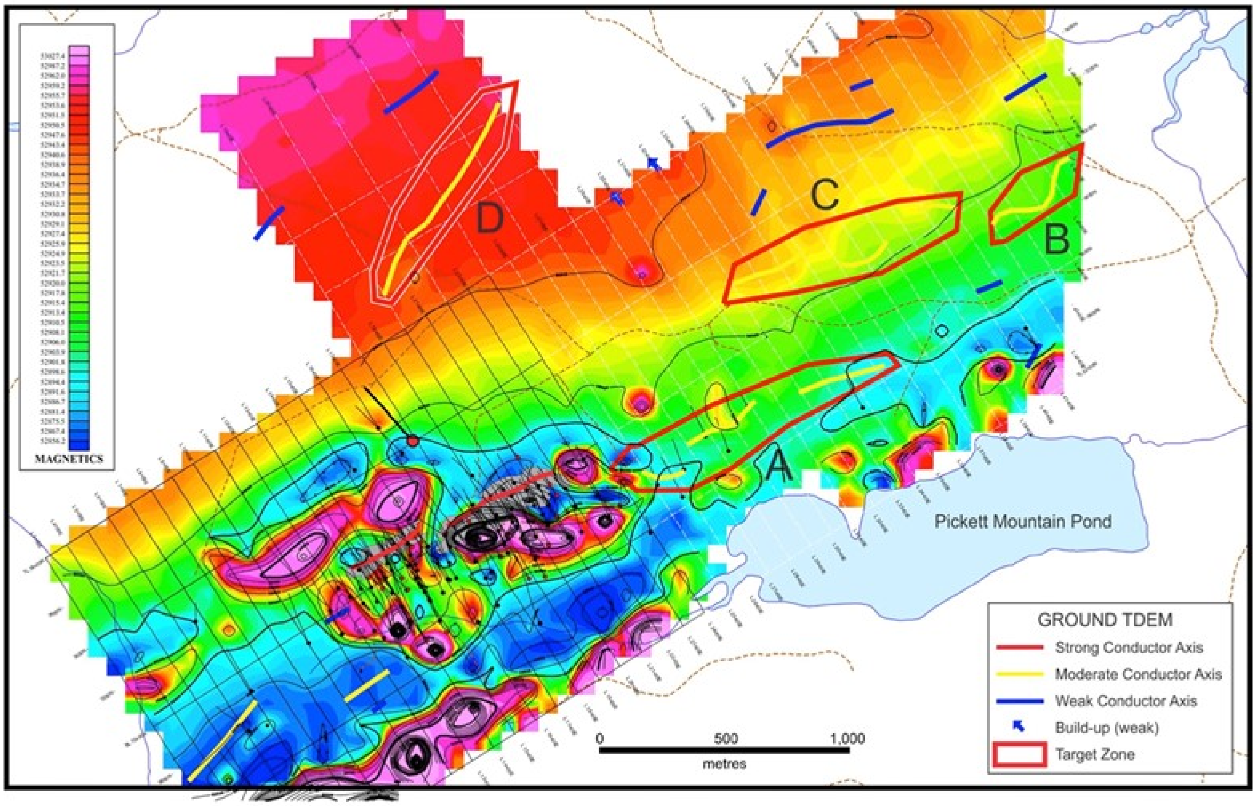


Figure 9.11 Total Field Magnetic Survey

The total field map shows a number of discrete magnetic highs flanking the East and West Lenses. The magnetic features situated to the south of the massive sulphide lenses are reflecting the presence of the mafic breccia and gabbro, while the magnetic high immediately to the north of the West Lens, is reflecting the presence of disseminated magnetite in footwall, felsic volcanic rocks.

The magnetic survey, in general, also outlined 2 contrasting domains of magnetic susceptibility. A disrupted domain of variable magnetic signatures is situated in the southern portion of the grid, reflecting various rock types. The northern domain comprises a magnetic high and is likely reflecting the presence of a more homogeneous sequence of rocks. Notably, conductors comprising target C, are located close to the interface of the 2 magnetic domains.

9.5.5 Whole Rock Geochemistry

In order to supplement the study of lithotypes and alteration patterns of enveloping rocks hosting the known massive sulphide deposit, whole rock geochemical analyses was completed on numerous samples, collected from outcrop and drill holes associated within the East and West Lenses.

Whole rock analyses were completed by Activation Labs utilising the 4B ICP OES whole rock package. This technique employs a lithium metaborate/tetraborate fusion. The resulting bead is rapidly digested in a weak nitric acid solution. The fusion ensures that the entire sample is dissolved. Whole rock data generated by this technique meets or exceeds the quality of data by conventional fusion XRF.

A primary use of WRA and trace element data is to help identify primary rock types to supplement visual core logging. Plots of relatively immobile elements including Ti versus Al, P versus Nb, and P versus Zr serve this purpose and have been particularly helpful at Pickett Mountain, in breaking out the various felsic volcanic, volcaniclastic, and intrusive lithotypes, that host mineralisation. The analysis clearly shows that the volcanic stratigraphic sequence hosting the Pickett Mountain deposit is bimodal in nature.

An additional use of the data is to investigate elements that may have been removed from the rocks in up-flow zones beneath the massive sulphides, versus those that have been added to the rocks by the same fluids. This helps to characterise styles of alteration characteristic of the VMS deposit employing a number of alteration indices. Once appropriate alteration indices have been established for a given massive sulphide deposit, such indices can be extrapolated out and utilised elsewhere within a volcanic belt.

At Pickett Mountain, sampling of felsic volcanic rocks north of the East and West Lenses (in the footwall), is characterised by widespread sodium depletion (<1% Na_20), due to destruction of the feldspars by ascending hot hydrothermal fluids. Sampling of trenches and drill core in the vicinity of drill hole PX-001, exhibit similar intensities of Na depletion as do footwall rocks immediately below the East and West Lenses. This augurs well for the existence of an additional potential mineralised horizon in and along trend from hole PX-001 and to the southwest of the West Lens, given the similarity in alteration patterns, as is illustrated in Figure 9.12.

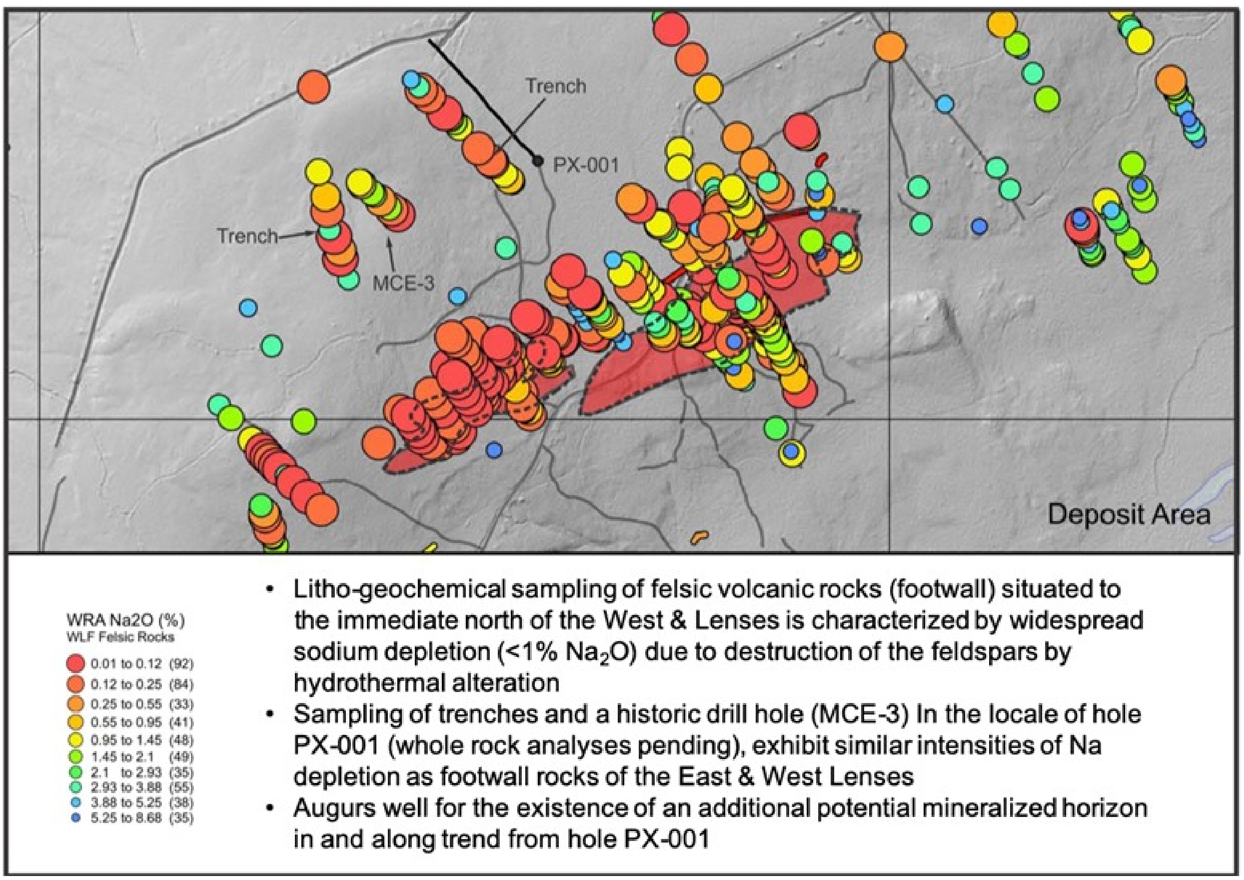


Figure 9.12 Whole Rock Geochemistry (Na_20) in the locale of the East and West Lenses

9.5.6 Soil Geochemistry Compilation

Getty Mines Ltd. completed detailed soil sampling in the locale of the East and West Lenses in the early 1980s. They utilised a grid for tie-in and analysed the soils samples for zinc, lead, copper, and silver, employing an aqua-regia digestion and utilising AA spectrometry for analyses.

The soil geochemistry map (Figure 9.13) clearly shows that both the East and West Lenses are reflected by strong, well defined soil anomalies (Zn + Pb + Cu). There is also a significant component of dispersion of such soil anomalies to the southeast of the massive sulphide lenses, likely due to glacial smearing of overburden in the down-ice direction. The ice direction is from the north-northwest to the south-southeast (170°), as evidenced by the presence of glacial striae observed in outcrops proximal to the East and West Lenses.

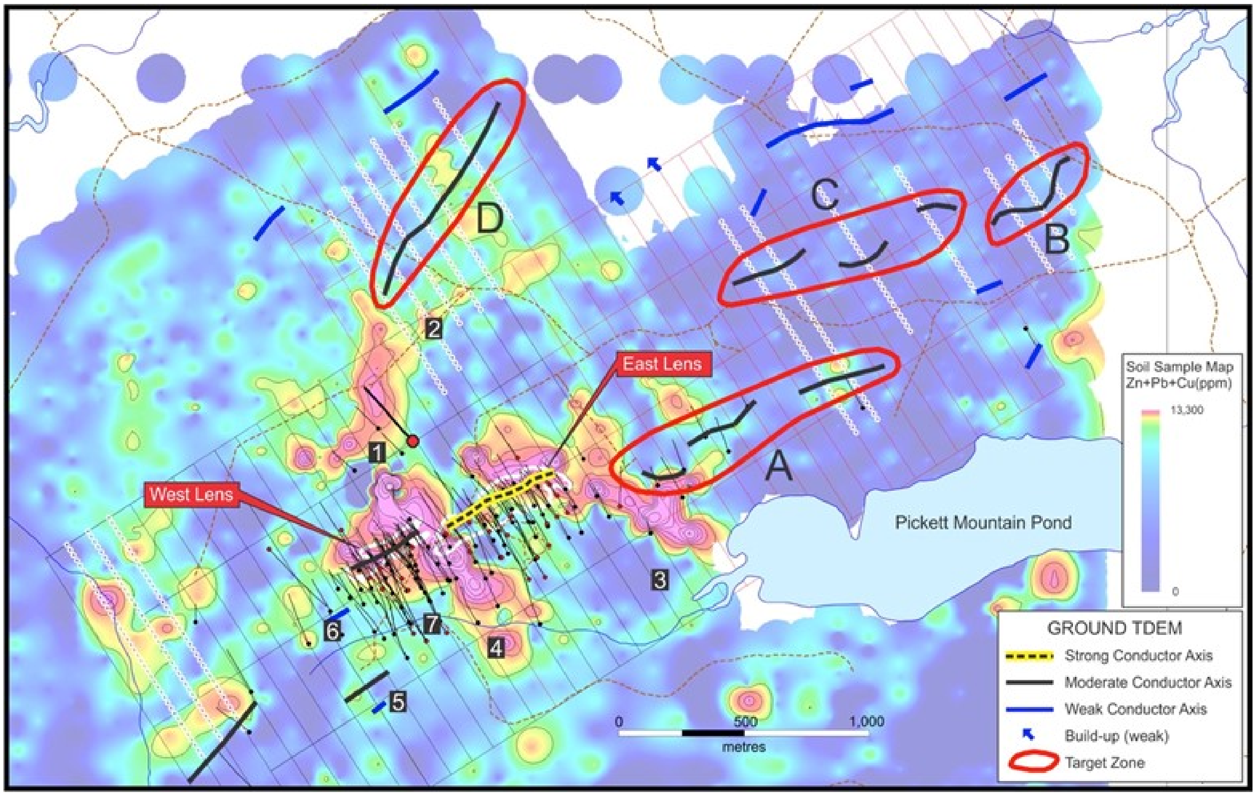


Figure 9.13 (Zn+Pb+Cu) Compilation Map

Other soil anomalies defined are compelling targets and warrant further investigation. In particular, a prominent anomaly located immediately to the west of the West Lens on the western fringe of the grid has not been tested by diamond drilling and may represent the southwestern extension of the main Pickett Mountain horizon. Strong soil anomalies persist immediately to the north and upslope from the East and West Lenses. The presence of such anomalies augurs well for the potential discovery of additional footwall lenses of massive sulphide in relation to the main mineralised horizon. Finally, a strong north-northeast trending soil anomaly situated 500m to the north of the East and West Lenses has seen minimal diamond drilling. The extension of this soil anomaly to the north-northeast is coincident with a prominent conductor defined by the ground TDEM survey (Target Zone D).

10.0 Drilling

10.1 Historic Drilling

Getty Mining and Chevron Resources completed historic diamond drilling programs at Pickett Mountain during the period of 1979 to 1985. The drilling was completed by Kennebec Drilling, based out of Bangor, Maine.

In all, a total of 113 drill holes were completed during this period, for a total meterage of 34,204m. All HQ-, NQ-, and BQ-sized equipment were utilised during these drilling programs. The drill holes were surveyed at the collar and down-the-hole using a Gyro instrument that measured the dip and azimuth every metre. In general, core recovery was averaging over 90%.

The drilling program was successful in that the first drill hole completed, intersected massive sulphide mineralisation. This result subsequently led to an extensive drilling campaign in efforts to determine the size and grade of the new discovery as well as the limits of mineralisation. Of the 113 historic drill holes completed by Getty Mining and Chevron Resources, 74 of them intersected massive sulphide mineralisation bearing significant Zn-Pb-Cu-Ag-Au values Table 7.1). Mineralisation from this drilling was traced over a 900m strike length and to a vertical depth of 750m.

The location, azimuth, and dip for all historic drill holes are summarised in Table 10.1, below. Additionally, the intersected widths and corresponding horizontal widths of all mineralised intercepts generated by the historic drilling, is documented on Table 7.1. It is notable that the general uniformity of grade for the mineral deposit is consistent, with no significant outliers in the assay results.

The data from most of these drill holes are utilised in the Mineral Resource estimate documented in this report. The historic drill core samples were cut in half using a diamond saw and sent to Skyline Laboratories in Tucson, Arizona for analyses. Copper, lead, and zinc were analysed by AA spectrometry while gold and silver were analysed utilising fire-assay techniques. High-grade copper, lead, and zinc assays, obtained by AA, were checked routinely using wet chemistry techniques.

The historical data includes most of the drill core in storage but does not include the original assay certificates. The historical results were compiled by Wolfden using original drill logs, drill sections, working files, reports, and databases prepared by the former owners of the Property at that time and subsequently acquired by Wolfden.

Table 10.1
Summary of Historic Diamond Drilling

Hole Number	Easting (UTM m)	Northing (UTM m)	Elevation (m)	Azimuth	Dip	Total Depth (m)
1	541370.81	5109305.38	353.02	330	-45	91.89
2	541385.58	5109278.11	350.62	328	-55	117.65
3	541836.67	5109092.73	324.88	344	-50	100.58
4	540874.33	5109082.93	374.24	360	-50	24.54
5	540873.19	5109147.93	374.33	150	-45	140.36
6	541123.38	5109319.83	365.35	330	-50	130.75
7	541122.99	5109201.08	365.47	330	-48	112.16
8	541280.06	5109239.92	356.41	334	-45	112.16
9	541341.19	5109263.01	353.34	340	-60	113.68
10	541427.34	5109201.69	348.61	334	-59	246.27
11	541495.30	5109268.47	345.16	329	-48	182.57
12	541620.00	5109239.60	336.38	330	-60	100.27
13	541370.76	5109203.18	350.17	330	-70	213.65
14	541534.18	5109196.63	350.76	337	-50	202.07
15	541567.89	5109388.96	347.31	330	-50	91.44
16	542222.75	5109598.30	336.15	330	-50	91.44
17	541372.95	5108988.45	339.67	330	-60	152.40
18	541034.17	5108902.76	353.99	330	-60	183.18
19	540731.63	5108990.83	385.09	330	-50	213.36
20	540603.01	5108685.53	380.00	330	-60	182.88
21	540018.86	5108738.69	381.62	0	-45	101.80
22	540870.35	5108737.85	359.93	330	-60	166.41
23	540776.62	5108911.88	377.16	329	-60	279.49
24	540776.62	5108911.88	377.16	330	-80	159.72
25	540841.22	5108797.24	364.85	330	-60	361.78
26	540557.09	5108780.22	387.70	330	-60	248.40
27	540241.63	5108459.88	394.85	330	-60	309.05
28	540873.13	5108962.11	372.48	330	-60	266.99
29	540690.59	5108808.31	378.70	330	-60	358.73
30	540918.18	5108874.56	362.68	330	-60	382.81
31	541173.09	5109126.88	363.27	330	-60	228.59
32	540732.62	5108700.34	372.98	330	-60	218.38
33	540947.81	5109035.96	365.70	330	-60	233.78
34	541206.72	5109064.11	359.35	330	-60	334.20
35	540984.57	5108970.01	360.09	330	-60	334.35
36	541135.80	5108977.89	353.54	330	-60	325.21
37	541234.91	5108994.81	351.16	330	-66	386.16
38	541024.66	5108906.73	354.99	330	-65	453.22
39	541283.48	5109135.14	355.68	330	-65	328.25
40	541178.09	5108910.90	343.64	330	-65	453.22
41	540877.61	5108731.32	359.81	330	-65	322.16
42	540955.90	5108818.48	354.28	330	-65	213.65
43	541669.01	5109158.94	335.36	330	-65	288.63
44	541366.42	5109135.30	348.11	330	-73	438.28
45	540771.49	5108671.72	370.71	330	-70	495.89
46	541282.58	5109189.47	354.58	330	-65	179.82
47	541220.87	5109162.47	360.45	330	-70	232.55
48	540811.17	5108688.33	367.64	335	-75	471.50
48A	540811.17	5108688.33	367.64	335	-75	806.77
49	541195.64	5109231.09	361.41	335	-58	92.35
50	541376.35	5109077.55	344.37	335	-77	524.23
51	541034.45	5109036.36	360.61	333	-70	249.01
52	540818.36	5109031.73	380.03	333	-62	151.78
53	540878.27	5109002.47	372.41	335	-60	197.20
54	540783.22	5108970.38	381.00	333	-50	161.54
55	540811.63	5108689.10	367.65	339	-60	599.54
56	541266.56	5109011.47	350.37	334	-73	453.83
57	540866.03	5109050.23	374.54	337	-61	132.28

Table 10.1
Summary of Historic Diamond Drilling
(continued)

Hole Number	Easting (UTM m)	Northing (UTM m)	Elevation (m)	Azimuth	Dip	Total Depth (m)
58	540904.10	5108941.50	368.42	338	-62	337.09
59	540776.92	5109003.32	382.87	335	-80	237.12
60	541288.58	5109292.07	356.59	274	-88.5	115.21
61	541031.16	5108973.11	358.11	335	-73	334.04
62	540719.45	5108872.77	379.53	335	-65	411.46
63	541245.53	5109266.18	358.90	336	-80	119.17
64	541196.86	5109239.11	361.03	301	-89	152.70
65	541060.01	5108911.26	352.35	335	-73	510.21
66	541200.47	5108933.74	345.62	340	-73	600.46
67	541234.24	5109240.72	358.78	207	-90	257.24
68	540813.23	5109043.25	380.72	337	-78	149.65
69	540813.30	5109042.80	380.59	335	-83	164.89
70	541424.32	5109121.70	347.08	335	-66	387.38
71	540672.74	5108944.46	389.61	334	-64.5	160.93
72	540953.60	5108823.49	354.68	329	-64.5	565.07
73	541013.59	5109081.09	365.29	301	-66.5	184.40
74	540720.44	5108873.57	379.43	336.5	-56.5	181.04
75	541560.37	5109094.72	344.30	339	-66.5	295.34
76	540678.28	5108866.87	384.40	337	-59	241.39
77	540636.86	5108835.05	386.43	337	-62.5	242.32
78	541403.82	5109167.83	348.06	338.5	-65	297.78
79	541508.48	5109073.72	341.26	334	-65	455.20
80	540899.87	5108911.40	367.25	335	-56	345.02
81	541347.38	5109138.78	349.80	333.5	-59	273.39
82	540803.61	5108853.57	369.85	331	-54	294.73
83	541348.29	5109139.39	349.71	339	-46	261.20
84	541005.67	5108853.87	352.60	338	-70	611.09
85	540859.54	5108862.82	367.14	331	-67	450.47
86	540803.13	5108940.86	377.78	337	-55	208.78
87	540895.03	5108965.40	370.21	338	-59	254.80
88	541135.56	5109040.81	361.28	337.5	-64	261.20
89	541226.32	5108802.15	338.03	336	-73	492.84
90	540992.25	5108739.72	347.20	337	-68	886.01
90A	540992.25	5108739.72	347.20	337	-68	791.53
91	540935.17	5108856.23	359.46	336	-67	602.13
92	541228.45	5109100.87	358.56	339	-65	260.59
93	541251.87	5109044.19	353.67	336.5	-70	355.38
94	540743.10	5108817.77	372.77	338	-67	373.97
95	540231.95	5108298.99	406.55	315	-60	263.64
96	540588.38	5108870.09	393.63	334.5	-50	138.07
97	541400.02	5108730.94	328.59	337	-69	953.06
98	540863.16	5108594.94	382.21	330	-75	749.16
98A	540863.16	5108594.94	382.21	330	-75	944.22
99	541145.16	5108737.09	336.66	325	-80	762.30
MCE-1	542683.15	5109593.93	337.60	330	-55	214.57
MCE-2	540842.26	5109411.65	376.68	315	-45	158.79
MCE-3	540747.18	5109510.09	383.26	315	-50	199.63
MCE-4	540656.83	5109340.40	394.31	310	-50	184.40
MCE-5	543334.79	5109910.38	339.53	150	-55	159.77
MCE-6	541863.34	5109282.79	331.46	340	-55	130.75
MCE-7	541967.88	5109344.40	328.85	340	-55	227.06
MCE-8	542055.06	5109477.16	331.40	335	-55	224.02
MCE-9	541963.69	5109234.86	324.62	338	-62	321.85
MCE-10	542146.93	5109419.86	327.16	340	-65	306.61
MCE-11	540469.52	5108651.52	383.52	335	-60	398.05
					Total	19459.17

10.2 Wolfden Drilling

Wolfden completed a drilling program comprising 38 drill holes totaling 15,451m during the period of December 2017 to December 2018. The drilling was completed by Downing Drilling Inc., based out of Duluth, Minnesota. Drilling performed in 2019 and 2020 was not included in the Mineral Resource estimate.

Both NQ- and HQ-sized equipment was utilised in the Wolfden drilling program. Drill holes were surveyed at the collar and down-the-hole using a Gyro instrument, every 30m down-the-hole. Core recovery was greater than 95%.

Most of the holes were drilled in the locale of the known Pickett Mountain deposit, largely directed at confirming the nature, grade, and extent of the massive sulphide deposit. The holes were intended largely as fill-in holes and a few were twinned holes to validate the historical drill findings obtained by Getty and Chevron during their earlier drilling campaigns. Step-out holes, along trend and down-plunge from the known mineralisation, were also completed by Wolfden, in efforts to determine the limits of massive sulphide mineralisation and to explore for additional massive sulphide lenses.

In general, the Wolfden drilling program was successful in that it did confirm and verify the nature, grade, and extent of the massive sulphide deposit in relation to the historic work. The infill component of Wolfden's drilling program largely demonstrated continuity of massive sulphide mineralisation in locales where there were significant gaps along strike and at depth, in the historic drilling. In particular, deeper drilling below the 400m level at the site of the West Lens (W1) was successful in intersecting high-grade base and precious metal mineralisation and is an instrumental component in the new NI 43-101 compatible Mineral Resource estimate, documented in this report.

The step-out component of Wolfden's drilling program also generated success with the discovery of a potential new massive sulphide lens located in the footwall, 180m to the north of the known massive sulphide deposit (E1-E2 Lens). The New Footwall Lens yielded an intercept of 4.1m at 38.2% ZnEq, including 16.6% Zn, 8.4% Pb, 1.9% Cu, 612.0 g/t Ag, and 0.5 g/t Au in drill hole PM-18-031. Further drilling to test the continuity of this new lens is clearly warranted.

Drill hole locations, azimuth, and inclination for Wolfden drill holes are included in Table 10.2. The intersected widths and corresponding horizontal widths of all mineralised intercepts obtained in the Wolfden drilling program are documented in Table 7.1.

The location of the historic drill holes and Wolfden drill holes are illustrated on Figure 10.1. Total metreage for both the historical and Wolfden drilling campaigns is 49,665, comprising 151 drill holes.

TABLE 10.2
SUMMARY OF WOLFDEN DIAMOND DRILLING

Hole Number	Easting (UTM m)	Northing (UTM m)	Elevation (m)	Azimuth	Dip	Total Depth (m)
PM-17-001	540881.9	5109037.7	372.7	327	-46	168
PM-17-002	540882.3	5109037.0	372.7	327	-61	180
PM-18-003	541278.9	5109158.7	356.4	327	-64	234
PM-18-004	541278.7	5109160.1	356.3	327	-50	378
PM-18-005	541262.7	5109084.4	355.2	327	-68	354
PM-18-006	541422.6	5109131.5	346.5	333	-63	33
PM-18-007	540843.9	5108911.0	372.3	327	-62	465
PM-18-008	540844.0	5108910.9	372.2	327	-69	495.5
PM-18-009	540867.7	5108865.1	366.5	316.5	-60.5	447
PM-18-010	540755.5	5108954.2	381.7	327	-55	198
PM-18-011	540778.5	5109012.0	382.9	327	-45	106.5
PM-18-012	540860.8	5109070.2	375.9	327	-45	99
PM-18-013	541431.9	5109286.8	349.1	327	-45	120
PM-18-014	541462.2	5109242.0	347.8	327	-45	171
PM-18-015	541463.6	5109158.1	350.3	327	-55	312
PM-18-016	541502.6	5109175.0	351.7	327	-55	312
PM-18-017	540905.1	5109095.1	369.8	327	-69	120
PM-18-018	540713.5	5108912.7	383.6	327	-55	192
PM-18-019	541113.1	5109018.3	358.8	327.0	-55.0	298
PM-18-020	541193.0	5109099.5	362.7	327.0	-63.0	237
PM-18-021	541241.3	5108996.6	351.0	327.0	-63.0	420
PM-18-022	540938.7	5108664.0	369.4	329.0	-58.0	759
PM-18-022A	540938.7	5108664.0	369.4	329.0	-58.0	699
PM-18-023	541020.3	5108694.6	349.2	327.0	-58.0	801
PM-18-023A	541020.3	5108694.6	349.2	327.0	-58.0	750.0
PM-18-024	540275.2	5108052.8	381.7	310.0	-70.0	597
PM-18-025	541304.7	5109393.7	357.9	329.5	-50.0	258
PM-18-026	540887.6	5108692.3	363.2	326.0	-57.0	585
PM-18-027	540762.4	5108868.5	374.3	321.5	-65.0	336
PM-18-028	541383.2	5108995.7	339.4	328.9	-60.0	708
PM-18-029	540940.9	5108662.1	369.3	325.5	-62	771
PM-18-029A	540940.9	5108662.1	369.3	325.5	-62	753
PM-18-030	541383.5	5108995.3	339.4	325.5	-74	651
PM-18-031	541349.7	5108910.0	336.9	326.3	-69.2	846
PM-18-032	540887.7	5108692.4	363.2	335	-60	705
PM-18-033	540513.0	5113880.0	313.9	340	-50	250
PM-18-034	540312.0	5109030.0	402.7	140	-55	522
PM-18-035	541430.0	5109044.0	339.0	331	-59	537

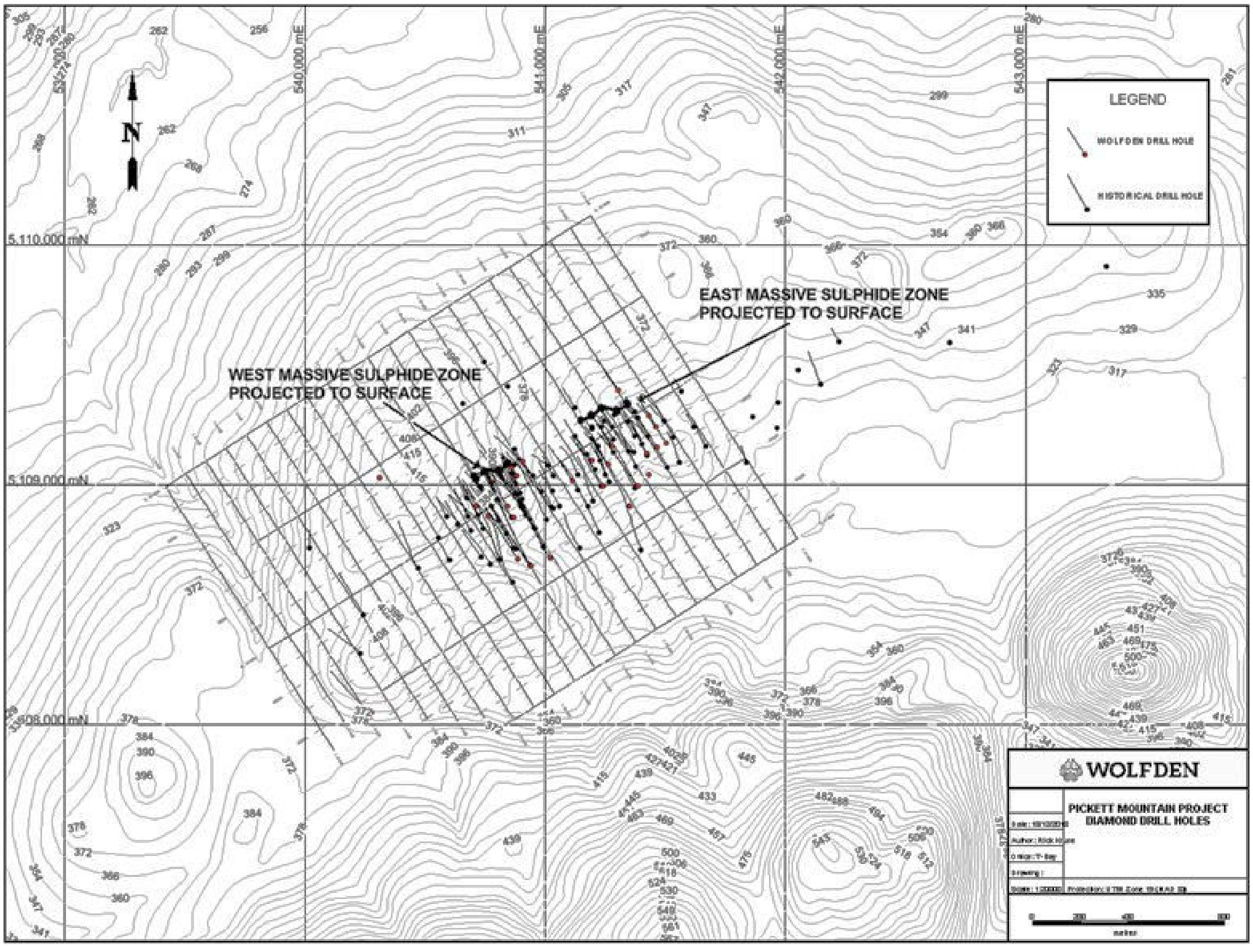


Figure 10.1 Location of Historical and Wolfden Drill Holes

10.2.1 Wolfden Drilling 2019

In 2019, Wolfden completed 3,530m of diamond drilling. Six holes were drilled from the surface and an additional 9 holes (both Wolfden and historic) were reamed out and/or extended for the purpose of completing down-hole EM surveys. The drilling was undertaken by Progressive Diamond Drilling Inc., based out of Sussex, New Brunswick and its United States subsidiary. Drill holes completed by Wolfden in 2019 are illustrated on the map below (Figure 10.2). One hole (PM-140) was drilled purely for the purposes of future metallurgical testing and as a result, has been placed in cold storage without assay until such time.

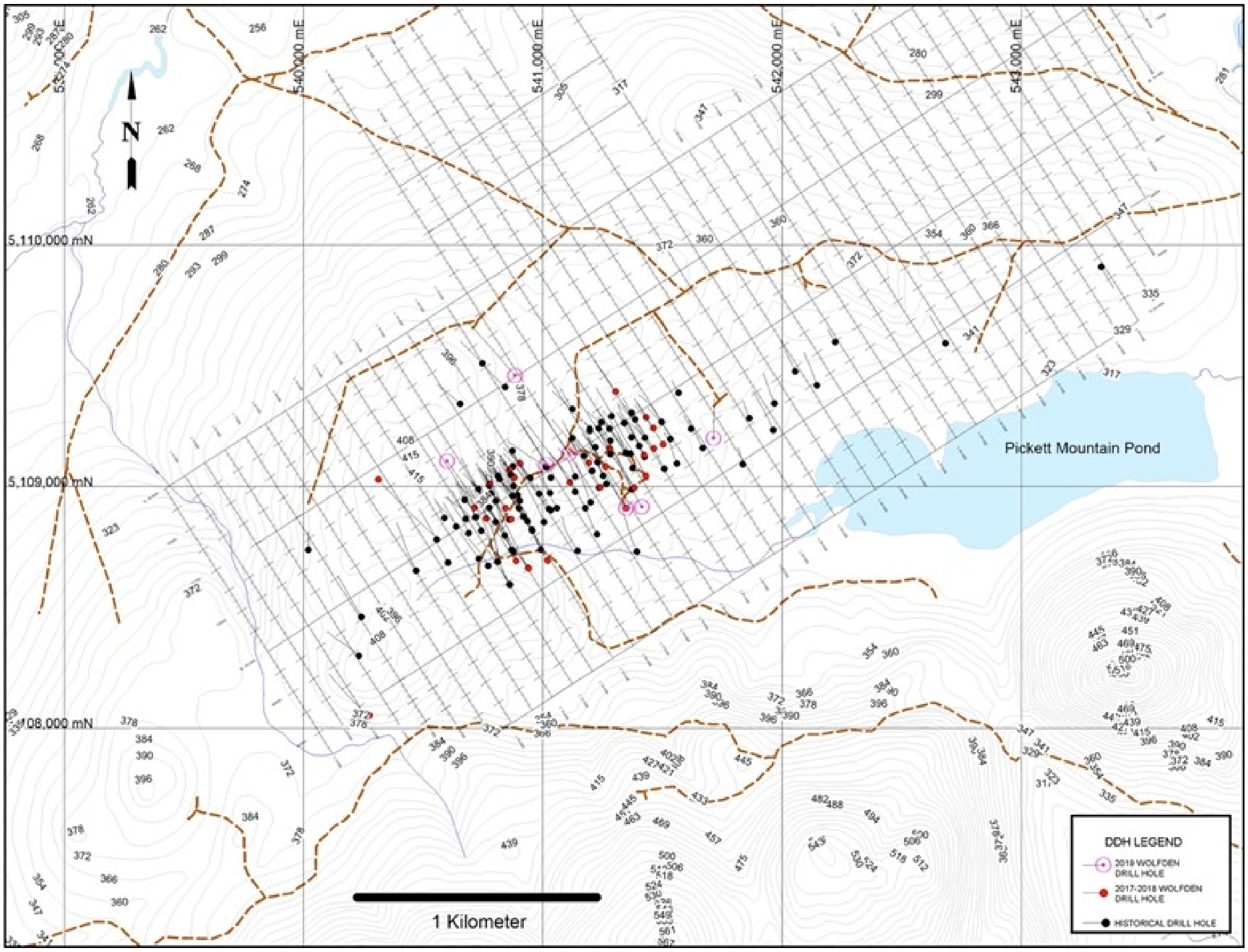


Figure 10.2 Location of All Drill Holes and Wolfden 2019 Drill Holes

As part of the 2019 drill program, 8 previous drill holes were reamed for down- hole geophysics (borehole electromagnetics – BHEM). All holes were successfully surveyed and the results confirmed this technique can be used to identify the East and West Lens extensions at depth that occur within a 100-200m distance of the hole. Although obvious depth potential exists with both lenses from 400m to 800m below surface and perhaps beyond, the next drill will prioritise near surface targets (conductors) with coincident soil and whole rock geochemical signatures. For more details on the 2019 drill results, please also refer to the Wolfden Resources Corporation Annual Information Form, filed on SEDAR April 28, 2020.

At depth on the East Lens, hole 137A, a wedge hole off of hole 137, targeting the deeper FWZ, unexpectedly intersected 9.6m of the East Lens at 6.7% ZnEq (3.0% Zn, 1.3% Pb, 0.5% Cu, 54.0 g/t Ag, and 0.3 g/t Au) in semi massive to massive sulphides that includes a higher-grade portion of 2.6m at 11.7% ZnEq (6.2% Zn, 2.7% Pb, 0.4% Cu, 109.4 g/t Ag, and 0.3 g/t Au). Hole 137 also intersected the East Lens yielding 3.5m at 2.3% ZnEq. A subsequent BHEM survey of hole 137A and another nearby historic hole, yielded strong build-up conductors, suggesting potential expansion of the East Lens at depths of 400 to 700 vertical metres.

Drill hole PX-001, the last hole completed in the 2019 exploration program, tested an historic drill hole with coincident gravity and soil anomalies, situated 500m to the north and parallel to the main horizon hosting the East Lens. The hole intersected a 207m felsic stringer zone with disseminated sulphide mineralisation and silica-sericite alteration. This rock type along with the sulphides and alteration is a typical marker for lithotypes underlying the East and West Lenses and could possibly be a significant indicator that another base metal rich sulphide lens is nearby. Unfortunately, the hole was stopped short of the planned target depth with core barrel jammed at the bottom of the hole. This area and its trend have been followed up with a gravity survey and warrant additional drilling in the next program (Table 10.3).

TABLE 10.3
SUMMARY TABLE OF WOLFDEN 2019 DRILL RESULTS

Hole No	From	To	Length (m)	Target	Results
PM-136	0	320	320	Shallow gap area between East & West Lenses & Footwall Zone	No West Lens intercept; FW Zone: 2.5 m of 0.68% Zn, 0.38% Pb, 0.14% Cu, 89 g/t Ag, 0.12 (1.9 m TW)
PM-19-031A	533	767	234	Footwall Zone intercept above PM-18-031	FW Zone: 4.2 m of 7.35%Zn, 3.65% Pb, 0.97% Cu, 105 g/t Ag, 0.59 g/t Au (2.1 m TW)
PM-137	0	882	882	East Lens and also Footwall Zone	East Lens: 8.0 m of 0.44% Zn, 0.20% Pb, 0.18% Cu, 11.2 g/t Ag, 0.06 g/t Au (4.4 m TW) FW Zone: 6.5 m of 1.36% Zn, 0.27% Pb, 0.04% Cu, 2 g/t Ag (2.8 m TW)
PM-137A	501	612	111	Footwall Zone - lost core barrel in bottom of hole - abandoned	East Lens: 9.6 m of 3.03% Zn, 1.34% Pb, 0.53% Cu, 54.0 g/t Ag, 0.30 g/t Au (5.1 m TW)
PM-18-028 ext	708	828	120	Footwall Zone	No significant mineralization
PM-18-029				West Lens, cleaned out to 753 m (EOH)	Not surveyed
PM-18-008	495	603	108	West Lens & Footwall Zone; to test and/or better define PEM build-up anomaly	Stringer sulphide zone; FW Zone not intersected
PM-18-035 ext	553	627	74	Footwall Zone	Footwall Zone not intersected
PM-18-021 ext	420	633	213	Footwall Zone updip from PM-19-031A	FW Zone: 18.0 m of 1.25% Zn, 0.64% Pb, 0.15% Cu, 78 g/t Ag (14.9 m TW)
PM-18-019 ext	298	496	198	Footwall Zone	Possible FW Zone intersected at sediment contact - no significant mineralization
G-048	806	807	1	Extension of West Lens to the southwest	BHEM picked up West Lens
PM-138	0	340	340	Shallow gap area between East & West Lenses & Footwall Zone	1.3 m of 1.81% Zn, 0.43% Pb, 0.81% Cu (0.82 m TW)
PM-139	0	250	250	Step out east of East Lens	No significant mineralization
PM-140	0	269	269	West Lens - hole for Met sample & infill	9.0 m of massive sulphide, assays pending
PX-001	0	410	410	Soil & gravity anomaly - rods dropped and hole abandoned with core barrel stuck	207 m of altered Felsic FW with dissem/stringer sulphides
Total metres			3530		

11.0 Sample Preparation, Analyses, and Security

At the core shed, the core boxes are laid out in order on benches, which can support up to five boxes. A geological technician measures the core and labels the box ends with UV-resistant plastic dymo tape. A geologist then logs and samples the core. A technician then collects magnetic susceptibility and conductivity measurements at 0.5m to 3.0m intervals, as determined by the geologist. All drill core is photographed wet and dry, after which some core may be placed on pallets and moved to outdoor storage.

Where base and/or precious metal minerals have been observed or are suspected to occur – intervals immediately above and below are marked in red wax for assay sampling by the geologist. Assay samples are generally 0.3m to 1.0m long and where warranted, intervals up to 3m have been routinely sampled. The breaks between samples are marked at changes in rock type or metal content in mineralisation, although some wall rock must be included in shorter intervals.

The date, drill hole number, and interval are recorded on the computer and in a pre-numbered tag book provided by the assay lab. Two tags, one large and one small, are placed under the core at the end of the sample interval.

The core for the sample interval is cut, piece by piece, in a core saw using diamond-impregnated steel blades. The core is cut parallel to the core axis, if possible, along the long axis of the intersection between the dominant structural fabric and the core. One-half of the core is returned to the core box, if possible, with the structural fabric at a counter-clockwise angle to the core axis. The other half of the core is placed in a sturdy, plastic, sample bag. After the last piece of cut core has been cut, the small sample tag is stapled in the core box at the end of the sample. The large sample tag is inserted into the sample bag, and the bag is sealed with a zip tie. The sample bag is added to a labeled rice bag, which is also zip tied, once it contains up to 25 kg of samples.

For assay samples, several digestions and ICP packages have been used in the past. Currently, the assay techniques are:

1. 1E3 Aqua Regia ICP(AQUAGEO): digestion by aqua regia and ICP-OES analysis of 38 elements;

2. 8-Peroxide ICP Sodium Peroxide Fusion ICP: reanalysis of over-grade zinc, lead, or copper by peroxide digestion and ICP-OES;

3. 8-Ag Ag-Fire Assay Gravimetric: reanalysis of over-grade silver by 30 gram fire assay and atomic absorption analysis; and

4. 1A2 Au-Fire Assay AA: analysis of gold by 30-gram fire assay with a gravimetric finish.

WRA samples are analysed by:

1. ME-MS61: 4-acid digestion and ICP-MS analysis of 43 elements.

Drill core is stored in 1.5 m-long wooden core boxes containing 3 rows of NQ core (4.76 cm diameter) or two rows of HQ core (6.35 cm diameter). A second core box is placed inverted on top, and the two are fiber-taped together for transportation. Usually, drill core is held at the drill until shift change, when it is taken to the driller's lay-down area and transferred to the drill foreman's truck. The foreman is met by the drill geologist at the core shed and the core is moved onto benches. In some circumstances, the core may be transported from the drill to the core shed by Wolfden employees.

The core remains in the locked core shed during core processing. Long intervals of unmineralised hanging wall rock are stacked on pallets, bundled with metal strapping and plastic wrapped, and moved to an unsecured outdoor core storage.

Mineralised core is sampled and with any other core of interest to the geologist, is moved to the locked indoor storage facility. Core samples are stored in the locked core shed until Wolfden's staff transports them to the assay laboratories sample prep lab. At present, the prep lab is Actlabs' facility in Fredericton, New Brunswick.

Actlabs is an independent, commercial assay laboratory that provides contract analytical services to Wolfden on the Pickett Mountain Project. They are ISO 17025 accredited and/or certified to 9001:2008.

It is the opinion of the authors of this technical report that sample preparation, security, and analytical procedures, currently employed, are adequate and meet industry standards.

12.0 Data Verification

12.1 Historical Data Verification

The Pickett Mountain Mineral Resource estimate documented in this technical report, in part, utilises historical drilling data generated between 1979 and 1989. Of the 111 historical drill holes on record, most of the cores from these holes are in two storage facilities owned and maintained by Huber Engineered Wood, at their production facility located in Easton, Maine. At the time of a site visit on September 26, 2017, it was observed that some core is on shelving and is easily accessible (see Figure 12.1).

Figure 12.1 Core Storage Facilities Maintained by Huber Engineered Woods at Their Plant Located in Easton, Maine at "Remote Warehouse 1" Where a Majority of the Core is Stored Piled on Shelving Units

Of the core in storage, most is stacked on pallets, wrapped in shrink wrap, and held together with binding straps. This prevented many of the holes being available for examination and re-sampling. However, some are stored on open shelves or racks. Subsequently, 4 holes (66-82-23, 66-82-28, 66-83-36, and 66-83-39) known to have massive sulphide were located, examined, and sampled (see Figure 12.2).

Figure 12.2 Core Boxes with Mineralised Sections Located at the Core Storage Facility

After the September 2017 visit to the Huber storage facility in Easton, Maine, the core has since been moved to a new secure storage facility located in Presque Isle, Maine. The new storage facility is operated and maintained by the Maine Geological Survey.

Data verification consisted of examining portions of these four holes. A random selection of medium- to high-grade intervals were selected and after cutting with a diamond saw, a total of 7 intervals of quartered core was sampled. Table 12.1 show the comparison of the re-assays with the original assays on record.

TABLE 12.1
COMPARISON OF CHECK SAMPLE ASSAYS WITH HISTORICAL RECORD
SIX DIGIT SAMPLE NUMBERS ARE THE VALIDATION SAMPLES
(AU AND AG IN OZ/T, CU, PB, AND ZN IN %)

Sample	*Au*	*Ag*	*Cu*	*Pb*	*Zn*
537556	0.007	0.875	0.41	0.27	2.93
13712	0.006	0.290	0.44	0.28	2.20
537557	0.005	0.758	0.31	0.21	0.48
13713	0.007	0.140	0.29	0.18	0.41
537558	0.025	3.238	3.82	3.76	8.23
13721	0.028	3.000	3.85	4.25	8.50
537559	0.016	2.392	1.06	2.02	5.60
4823	0.023	2.030	1.91	3.69	8.97
537560	0.040	11.435	1.11	14.00	31.00
2021	0.055	8.880	0.92	15.40	29.90
537561	0.020	2.975	1.18	4.98	12.70
13747	0.025	2.590	1.25	4.80	12.50
537562	0.024	3.676	1.33	6.48	14.70
2008	0.058	3.410	1.16	5.52	12.00

All validation sample values are in the same order of magnitude, as the historical values. While not the same, considering that less sample material was submitted due to the quartering of the core, all validation sample results are consistent with that of the historic numbers and confirm that they are a valid record of mineral grades.

In addition, the collars of 3 holes were located in the field where the casing had not been removed or destroyed. Figure 12.3 shows the collar of drill hole 66-46; the GPS coordinates (541190E, 5109225N) correspond within acceptable error with the calculated UTM equivalent to the original hole collar location (541195.6E, 5109231.1N) Maine State Plane coordinates. Two other collars were located (66-42 and 66-63) and similar correlations were found.

Figure 12.3 The Casing Located in the Field and Identified as Being the Collar for Hole 66-46

Based on the positive correlation of the assays obtained from check sampling of the historic drill core and for the hole collars found in the field, it is the opinion of authors, the QPs responsible for this report, that the information in the historical documents is reliable and is suitable for use for current and future studies, including Mineral Resource estimation.

12.2 Current Data Verification

Control charts showing standard, blank, duplicate, same-lab check, and outside-lab check sample results for each of Zn, Pb, Cu, Ag, and Au indicate good reliability for the assay data, except high blanks are common, indicating poor cleaning at the sample preparation lab.

12.2.1 Description of Control Charts

The standard, duplicate, same-lab check, and outside-lab check control charts plot the results of the control sample analysis against the reference values.

Standard Charts: The reference values for standard samples are the certified values taken from certificates published on-line. For the base metals, the selected certified values and errors (1σ) are those appropriate to the digestion: 4 acid for standards with certified values below the upper detection limit and sodium peroxide for those above.

The standard results for each lab report were evaluated against the certificate values at ±2σ and ±3σ; however, to save space, +10% and -10% lines have been substituted on the standard charts.

Blank Charts: The blank charts plot the blank sample results versus the results for the preceding sample (which assumes the samples were prepared in alphabetical order). Below-detection values were set to half of the detection limit.

Lines representing a US$2/t value for the metal are shown for reference and calculated as:

Element	US$ Price	US$2/t	Conversion	Grade
Zn	$1.20/lb	1.67 lb	0.0454%/lb	0.08%
Pb	$1.00/lb	2.00 lb	0.0454%/lb	0.09%
Cu	$2.50/lb	0.80 lb	0.0454%/lb	0.04%
Ag	$16/oz	0.125 oz	31.10 g/oz	3.89 gpt
Au	$1,200/oz	0.002 oz	31.10 g/oz	0.052 gpt

Lines representing the blank values versus previous sample values of 0.2%, 0.5%, and 1.0% are also shown on the charts for reference.

***Duplicate Charts*:** The charts plot analyses of the duplicate versus the previous sample in the sample batch, with below-detection values were set to half of the detection limit. Quarter-core duplicates were replaced by pulp duplicates part way through the project to conserve material for later analytical or metallurgical work. The +10% and -10% lines are shown on the charts for reference.

***Same-Lab Check Charts*:** The charts plot check versus the original Actlabs analyses for a batch of pulps selected from the drilling to date and submitted to Actlabs. Below-detection values were set to half of the detection limit. Lines representing +10% and -10% are shown on the charts for reference.

Outside-Lab Check Charts: The charts plot check versus original Actlabs analyses for a batch of pulps selected from the drilling to date and submitted to AGAT. Below-detection values were set to half of the detection limit. Lines representing +10% and -10% are shown on the charts for reference.

12.2.2 Quality Assurance/Quality Control (QA/QC) Evaluation

Zinc (Zn) (Figure 12.4): Most Zn standard analyses fall well within ±10% and ±3σ of the certified reference values. Two OREAS 623 (1.03 ± 0.04% Zn) samples returned -3.8σ and -3.3σ analyses, but the standard generally provides below certificate results.

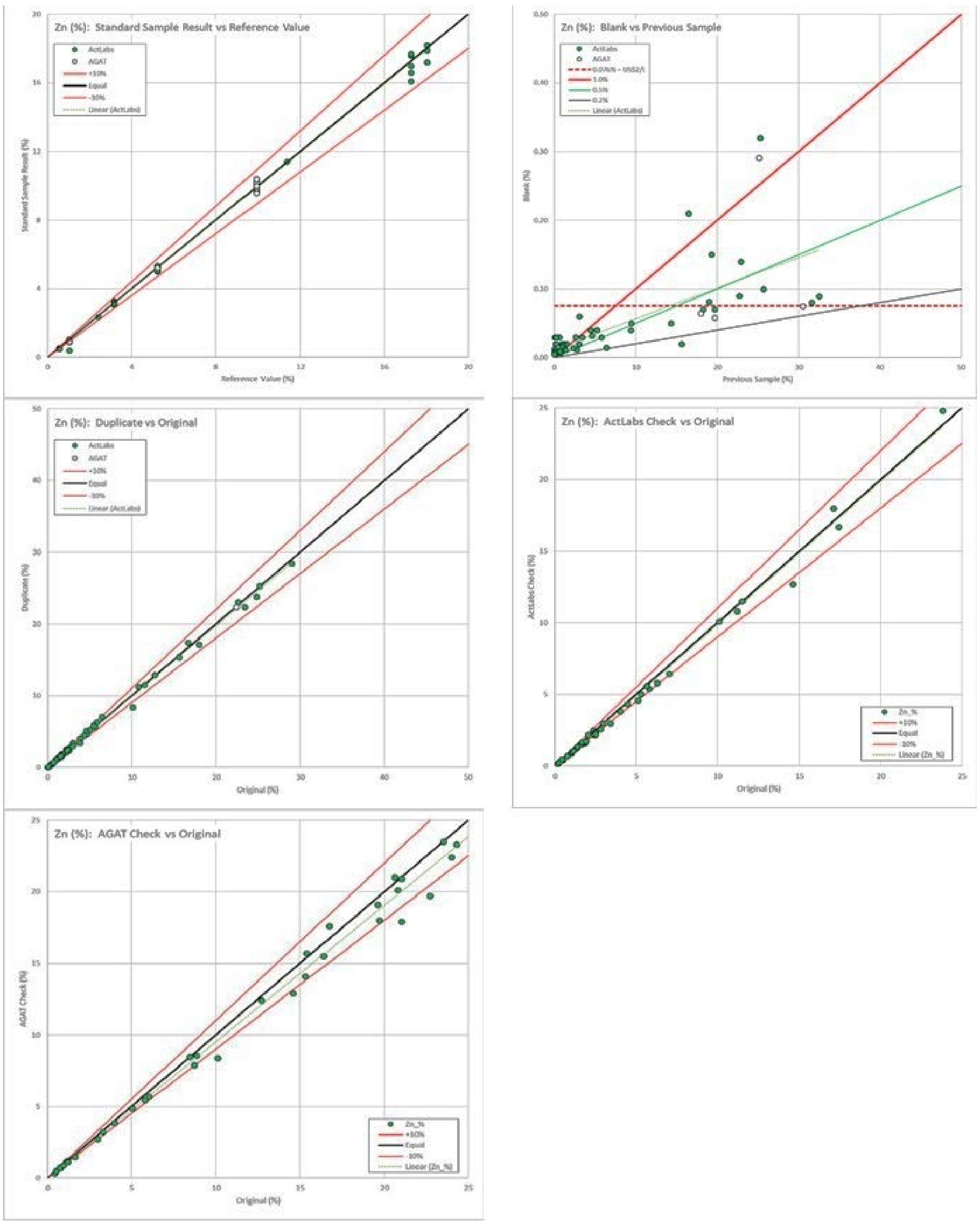


Figure 12.4 Zn QA/QC Control Charts

An OREAS 623 standard (E5407686, PM-18-031, Actlabs report A18-18236, finalised 2019-Jan-10) returned a value of 0.41% Zn, which is 60% or 15.4σ low. Neither this sample nor adjacent samples have values matching a standard and so a sample swap is unlikely.

Several blank samples returned significant Zn (up to 0.32%), several of which represent more than US$2/t. Three samples have results indicating >1% contamination from the preceding samples, indicating very poor cleaning practices at the samples preparation facility. A check analysis of the same pulp as one of these three samples reproduced the analysis at the outside lab.

The duplicate and check sample analyses suggest good reproducibility, although AGAT returned Zn values about 4% below those of Actlabs, on average.

Lead (Pb) (**Figure 12.5*****):*** The Pb control sample results share the problems of the Zn results, although less pronounced. Only one blank result corresponded to more than US$2/t in Pb values. AGAT returned Pb values about 5% below those of Actlabs, on average.

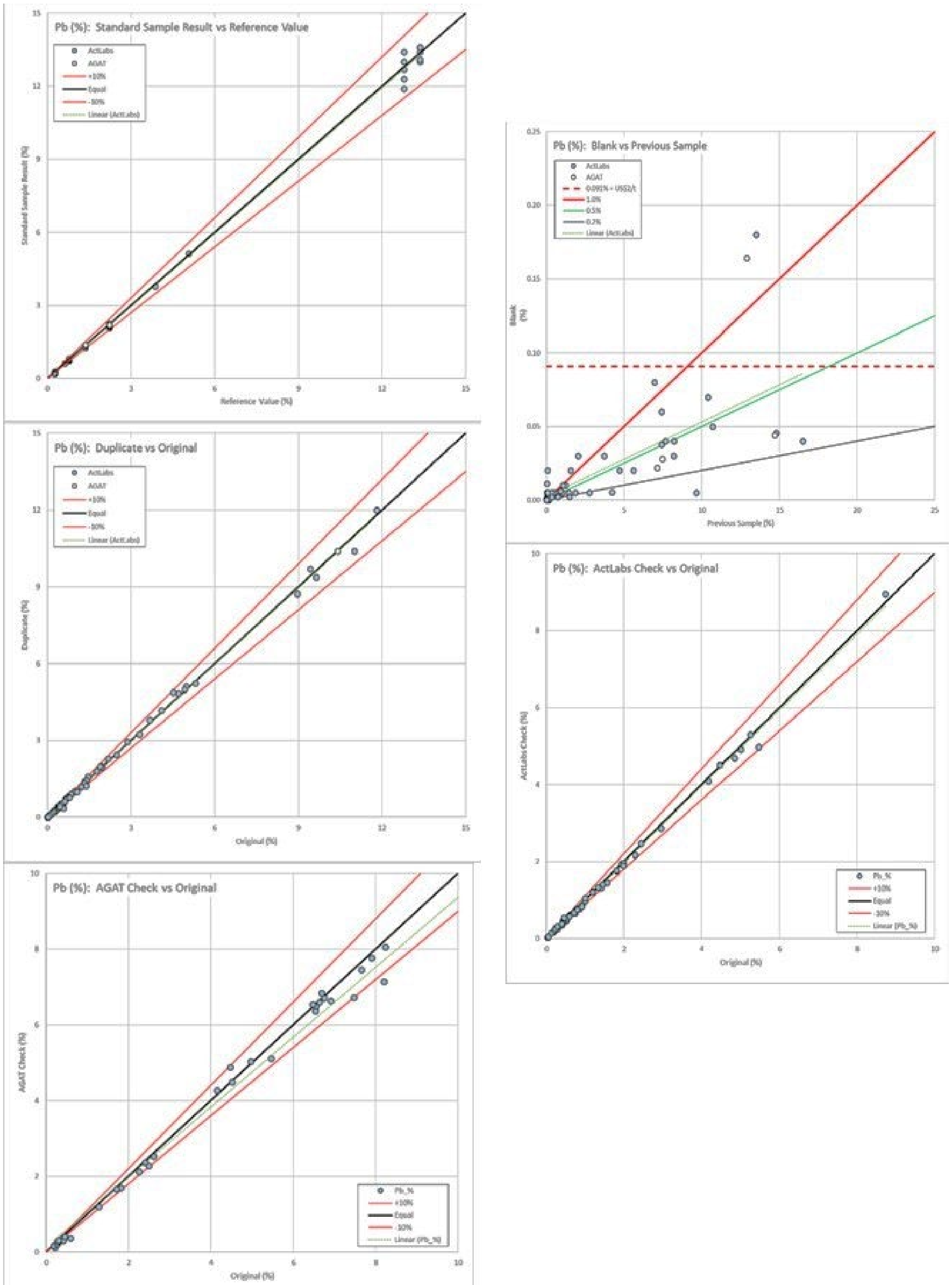


Figure 12.5 Pb QA/QC Control Charts

Copper (Cu) (**Figure 12.6*****):*** Again, the Cu control sample results share the problems of the Zn results, although less pronounced. No blank results corresponded to more than US$2/t in contained Cu, although a significant number of the blank sample results were above 1% of the previous sample results. AGAT returned Cu values similar to those of Actlabs.

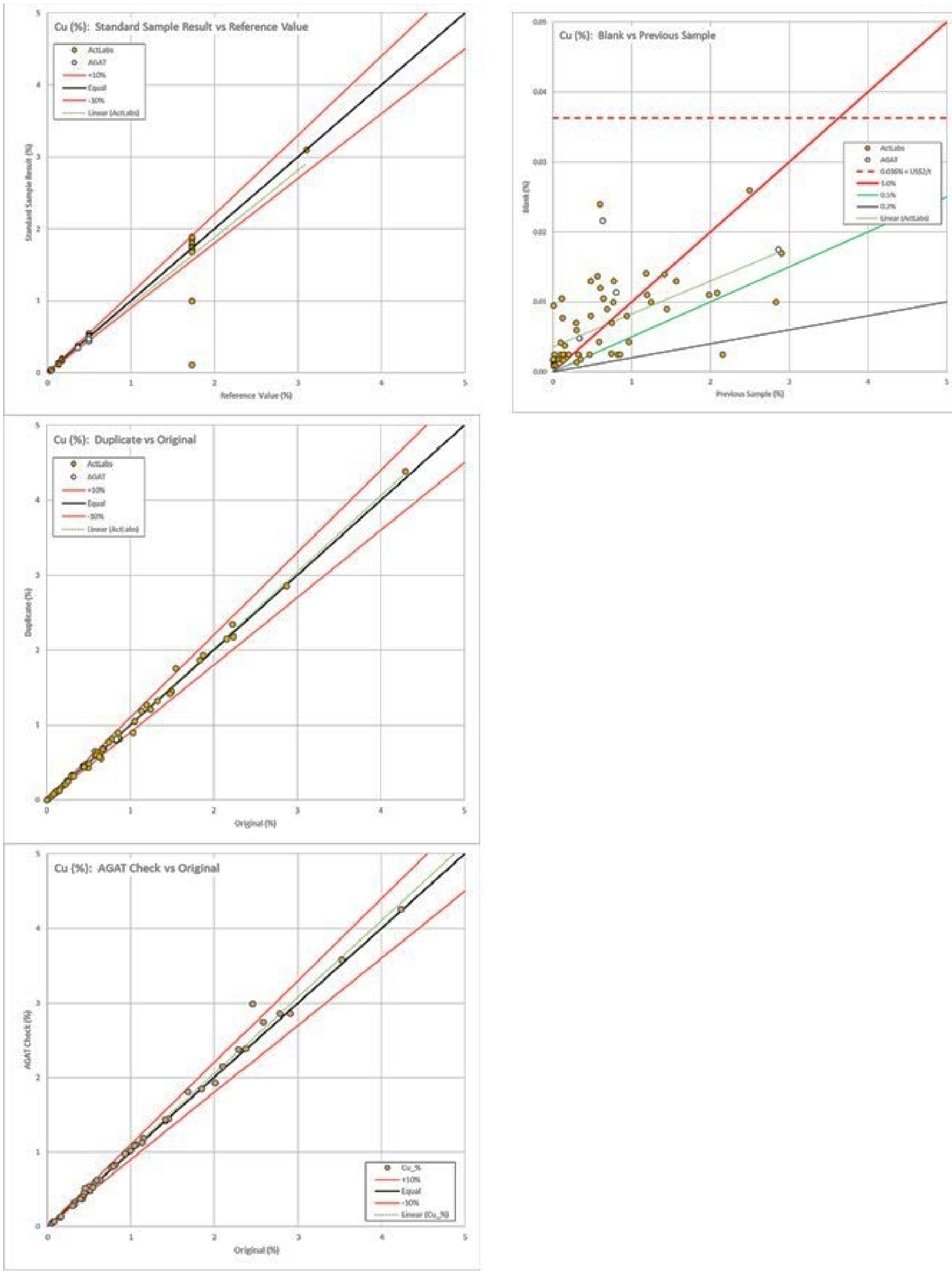


Figure 12.6 Cu QA/QC Control Charts

Silver (Ag) (Figure 12.7): Nugget effects are likely responsible for the increased scatter of the precious metal control charts relative to the base metal charts.

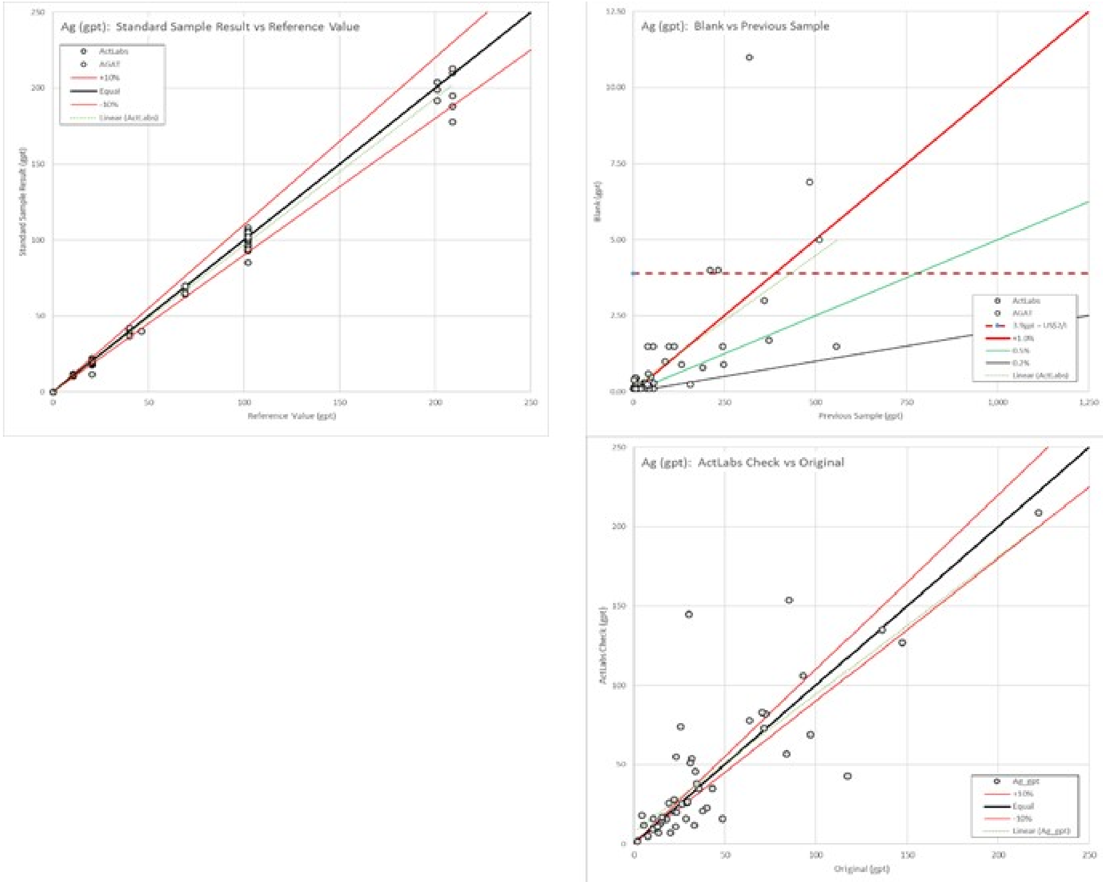


Figure 12.7 Ag QA/QC Control Charts

Both labs tend to be a bit low on high-grade Ag standards and a few blanks represent more than US$2/t in contained Ag. The duplicate Ag correlate fairly well, with the worst deviations from original values that are capped at the 100 gpt upper detection limit because no over-grade analyses were run. Oddly, the Actlabs check analyses show poorer correlation than those run at AGAT.

Gold (Au) (Figure 12.8): As with Ag, the higher-grade Au standards ran a bit low, on average. Blanks, duplicates, and both sets of check analyses show considerable scatter.

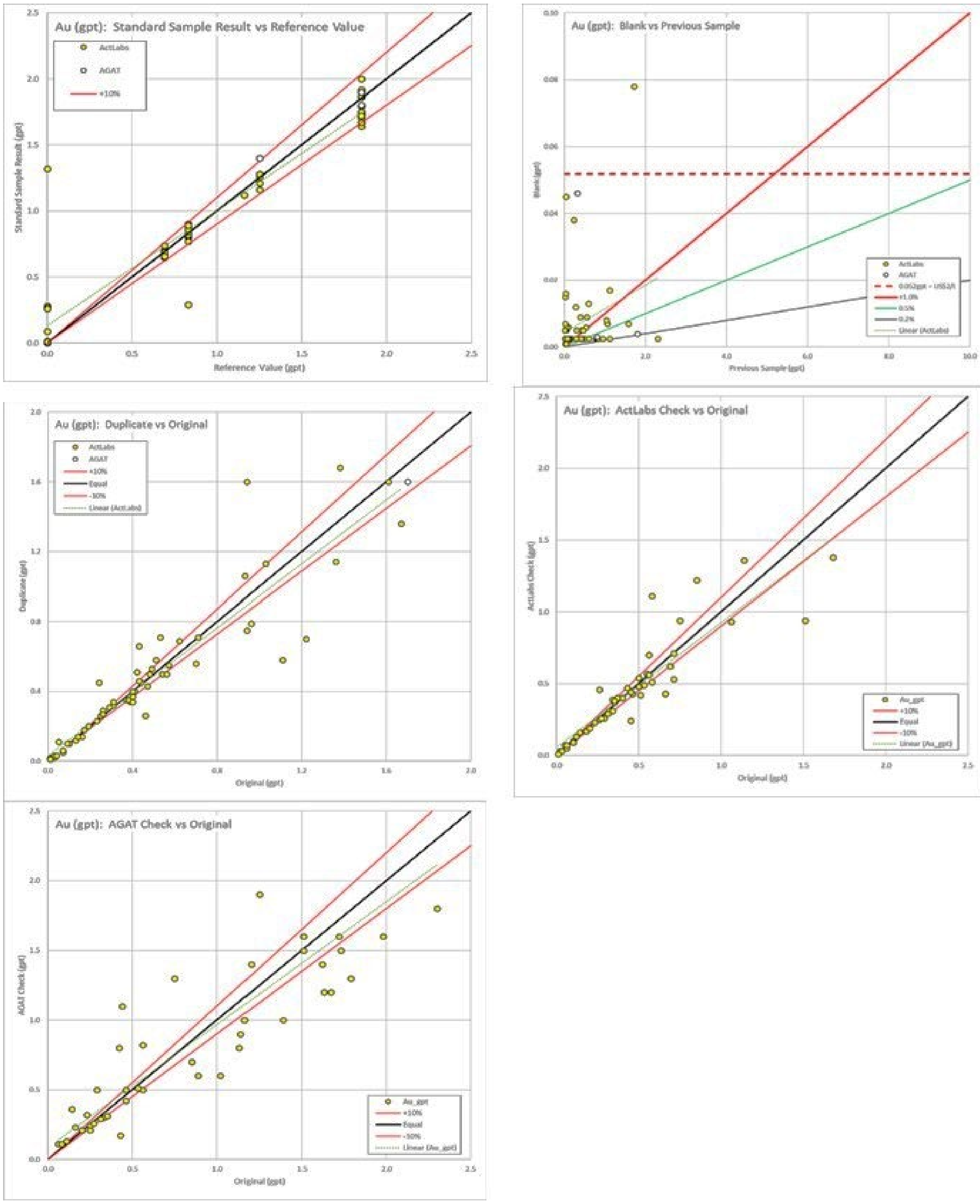


Figure 12.8 Au QA/QC Control Charts

The standards utilised in the drilling program include OREAS 133b, 134a, 134b, 630, 132b, 620, 621, 622, and 623.

12.2.3 Comparison of Original and AGAT Check Sample Composites

The check samples sent to AGAT include six mineralised intervals analysed at different times throughout the program. A comparison of the calculated composites for the original and check analyses (Table 12.2) indicates an about 5% lower ZnEq (and the related dollar value) for the check samples, with no trend through time.

Table 12.2
Original Versus Outside-Lab Check Composites

Hole-ID	From	To	Len	Samples	Lab	$/t	Zn eq	Zn %	Pb %	Cu %	Ag gpt	Au gpt	Date
PM-18-004	172.10	180.90	8.80	11	ActLabs	$629	23.40	12.64	4.68	1.36	133.45	1.23	2018-Feb-28
					AGAT	$596	22.16	11.85	4.67	1.41	123.82	0.98	2019-Jan-25
					Change	-$33	-1.24	-0.79	-0.01	0.05	-9.63	-0.25	
					Percent	- 3%	-5.3%	-6.3%	0.2	3	-7	-.3	

In general, the quality control analyses are more accurate and reproducible for the base metals (Zn, Cu, and Pb) than for the precious metals (Au and Ag). This reflects the lower abundances of the precious metals and analytical difficulties, particularly for Ag.

The analyses for standard samples are acceptable with the exception of analyses of standard OREAS 623 that commonly returned lower values. The cause of this is not known.

Many of the blank sample analyses indicate more than 0.5% contamination from preceding samples with some in excess of 1% contamination. This suggests poor cleaning procedures at the preparation laboratory.

The duplicate and check analyses correlate adequately. It is the opinion of the authors that the quality control results are generally good, and therefore, the analytical data is reliable.

13.0 Mineral Processing and Metallurgical Testing

The test work was originally performed at Lakefield Research in 1984 and 1988 and recently a scoping study was completed at Resource Development Inc. (RDi).

The metallurgical reports were reviewed and a summary of the results are presented in this section.

13.1 Getty Mining Company, 1984

Lakefield Research undertook metallurgical test work for the recovery of copper, lead, and zinc from five composite samples. Approximately 3,350 kg of one-half core, one-quarter core, and slices of core was sent to Lakefield.

The metallurgical test work included sample preparation, preparation of five composites, rougher and cleaner flotation tests, Bond's Work Index determination, settling tests, mineralogical examination of test products, gold and silver recovery from tailings and determination of acid producing potential of tailings.

13.1.1 Sample Characterisation

Five master composite (MC) samples were prepared from the fresh drill core splits. The head analyses of the composite samples are given in Table 13.1. The assays indicated some variation in copper (1.01% to 4.03%), lead (2.84% to 5.29%), zinc (7.23% to 17.1%), gold (0.52 g/t to 1.26 g/t) and silver (57.5 g/t to 142 g/t). Samples 2-5 were not assayed for arsenic.

TABLE 13.1
HEAD ANALYSES OF COMPOSITE SAMPLES

Sample	Assay							
Master Composite	% Cu	% Pb	% Zn	% Fe	% S	% As	Au, (g/t)	Ag, (g/t)
1	1.32	4.29	9.72	26.9	33.4	0.16	0.75	91.4
2	4.03	5.29	11.8	25.7	34.6	-	1.26	142.0
3	1.11	6.58	17.1	21.1	30.0	-	0.95	139.0
4	1.13	2.84	7.23	23.7	28.4	-	0.52	57.5
5	1.01	4.93	9.07	19.6	24.4	-	0.78	116.0

The iron and sulfur analyses of the composites indicated that the deposit is massive sulfide. The specific gravity of MC-1 of 4.19 confirmed it.

13.1.2 Bond's Ball Mill Work Index

Bond's ball mill work index was determined for master composite 1 at 200 mesh. It was determined to be 8.79.

13.1.3 Flotation Testing

Most of the flotation test work was performed on Master Composite 1. The other composites were tested at the end of the program to evaluate their response to the developed flowsheet.

Two alternative flowsheets were evaluated:

1. Copper-lead bulk flotation followed by Cu-Pb separation.

2. Selective copper and lead flotation.

The highlights of the test work are presented in the following sub-sections.

13.1.3.1 Cu-Pb Bulk Flotation

Tests were conducted to evaluate the response of the ore to the circuit involving flotation of a bulk Cu-Pb rougher concentrate followed by regrinding and cleaning and subsequently Cu-Pb separation.

Four different depressant schemes were evaluated to depress zinc and float copper and lead. Other variables investigated included fineness of grind, flotation time, and reagent amounts. The best overall results in the rougher flotation was approximately 90% Cu and Pb recovery with 25% Zn recovery. These results were obtained with Na_2SO_3 – NaCN – Na_2CO_3 and Ca $(OH)_2$ – $ZnSO_4$/NaCN and SO_2 depressant systems at a grind size of ±75% minus 400 mesh. The test data are given in Table 13.2 and Table 13.3.

TABLE 13.2

METALLURGICAL RESULTS FOR NA_2SO_3 – NACN – NA_2CO_3 DEPRESSANT SYSTEM

Test No.	Grind % -400 Mesh	Reagents, g/t				Flotation Time, Min	Cu-Pb Rougher Concentrate						
		Na_2SO_3	NaCN	Na_2CO_3	A325		Assay, %			Distribution, %			
							Cu	Pb	Zn	Wt.	Cu	Pb	Zn
1	84.3	2000	50	250	200	18	6.26	18.6	8.15	18.81	89.1	81.7	15.4
3	76.7	1000	50	250	40	8	3.23	10.7	7.02	37.45	90.4	90.8	25.9
8	42.0	1500	50	250	55	20	2.56	12.0	18.0	30.35	63.8	86.4	55.3

TABLE 13.3

METALLURGICAL RESULTS FOR $CA(OH)_2$– $ZNSO_4$/NACN – SO_2 DEPRESSANT SYSTEM

Test No.	Grind % -400 Mesh	Reagents, g/t				Flotation Time, Min	Cu-Pb Rougher Concentrate						
		$Ca(OH)_2$	$ZnSO_4$/ NaCN	SO_2	A325		Assay, %			Distribution, %			
							Cu	Pb	Zn	Wt.	Cu	Pb	Zn
2	70.2	1000	500	450	70	9	4.32	14.2	9.35	26.93	90.2	89.8	24.9
5	58.9	1000	500	500	55	9	4.31	14.7	10.3	25.20	84.7	87.9	25.6

Based on these results, the latter depressant system was selected for further testing, which included the effect of grind and reagent levels in the rougher flotation and to investigate the cleaner flotation response. The best Cu-Pb rougher flotation results were obtained at primary grind of 78.9% minus 400 mesh with 1,000 g/t lime, 500 g/t $ZnSO_4$/NaCN, 500 g/t SO_2, and 80 g/t A325. The rougher concentrate recovered 32.67% of weight, 91.6% of copper, 90.6% of lead, and 29.9% of zinc at a grade of 3.37% Cu, 11.5% Pb, and 8.83% Zn.

Upgrading of Cu and Pb in the cleaner flotation was poor with problems in depression of the pyrite and sphalerite. The Cu-Pb cleaner concentrate could be recovered in the ranges of:

Assays, %			Distribution, %		
Cu	Pb	Zn	Cu	Pb	Zn
9	30	7.5	62	61	7
7	28.5	7.8	64	75	9
6	26	8.0	73	85	12

The lower grade concentrates had reasonable recoveries but were contaminated with pyrite and sphalerite. The high-grade concentrates had low recoveries.

Cu-Pb separation was performed on the Cu-Pb cleaner concentrate from two different tests, namely test 38 and 41. The Cu-Pb concentrate was conditioned with activated carbon, dextrin, and SO_2 and a copper rougher concentrate was recovered. The concentrate was cleaned once.

The Cu-Pb concentrate was conditioned with activated carbon and SO_2. A copper concentrate was collected and cleaned once. The copper flotation tailing was conditioned with lime and $ZnSO_4$/NaCN and a Pb concentrate was recovered and cleaned. The test results, summarised in Table 13.4, indicated that copper was upgraded reasonably well but lead upgrading and recovery were poor.

TABLE 13.4
RESULTS OF CU-PB BULK FLOTATION AND SEPARATION

Test No.	Product	Assay, %			Distribution, %			
		Cu	Pb	Zn	Wt.	Cu	Pb	Zn
38	Cu Cleaner Conc.	23.9	5.38	3.46	3.31	63.3	4.3	1.2
	Cu Flotation Conc.	16.3	16.4	5.04	5.09	66.3	20.0	2.7
	Cu-Pb Separation Tailing (Pb Conc.)	0.70	25.6	9.42	10.50	5.9	64.2	10.3
	Cu-Pb Cleaner-Conc. (Separation Feed)	5.79	22.6	7.99	15.59	72.2	84.2	13.0
	Cu-Pb Rougher Conc.	2.79	9.55	9.87	41.45	92.5	94.5	42.6
41	Cu Cleaner Conc.	28.3	4.32	2.48	2.29	52.8	2.4	0.6
	Cu Flotation Conc.	21.8	13.3	3.52	3.31	58.1	10.8	1.2
	Pb Cleaner Conc.	0.22	47.1	9.53	3.40	0.6	39.3	3.4
	Pb Flotation Conc.	0.35	42.5	9.53	4.19	1.2	43.7	4.2
	Cu-Pb Sepn. Tail	0.97	21.0	8.48	5.25	4.1	27.0	4.6
	Cu-Pb Cleaner Conc. (Separation Feed)	6.17	26.1	7.54	12.75	63.4	81.5	10.0
	Cu-Pb Rougher Conc.	3.57	12.0	9.79	31.32	90.1	92.4	31.9

13.1.3.2 Selective Cu, Pb, and Zn Flotation

Based on the visual observation of the Cu-Pb bulk flotation, it was noted that Cu floated ahead of the Pb. Hence, a decision was made to investigate sequential Cu and Pb flotation. The addition of SO_2 to the grind resulted in production of a fairly selective copper rougher concentrate, which could be upgraded following regrind of rougher concentrate. Preliminary testing results are given in Table 13.5.

TABLE 13.5
PRELIMINARY SELECTIVE COPPER FLOTATION RESULTS

Test No.	Procedure	Product	Assay %			Distribution %			
			Cu	Pb	Zn	Wt.	Cu	Pb	Zn
26	Cu-SO_2 to Grind M200 Collector Regrind Ro. Conc. Pb-Na_2CO_3, NaCN, A325 Collector Regrind Ro. Conc.	Cu Cl. Conc.	23.3	2.90	3.07	3.28	63.0	2.4	1.1
		Cu Ro. Conc.	6.64	3.90	7.78	15.29	83.7	12.8	12.6
		Pb Cl. Conc.	0.13	45.3	9.41	5.21	0.6	60.0	5.2
		Pb Ro. Conc.	0.22	14.2	11.2	20.82	3.8	75.2	24.7

Optimisation of the copper flotation resulted in primary grind of 60% to 80% minus 400 mesh, M203 as primary collector and reduction in dosage of A325. This improved selectivity toward lead and higher copper recovery. Overall, from the batch tests, a copper concentrate assaying 22% to 24% Cu with a recovery of 55% to 65% copper was obtained.

Copper rougher tailing and first-cleaner tailing were combined as feed to the lead circuit. The test results, given in Table 13.6, indicated that lead concentrate can be upgraded to 45% to 48% Pb, but after that lead losses were higher. Overall lead recovery for a 45% to 48% Pb concentrate was about 65% and zinc loss to the concentrate was about 5%.

TABLE 13.6
LEAD FLOTATION RESULTS

Test No.	Grind % -400M	Lime g/t		$ZnSO_4$ NaCN, g/t		A325 g/t	Pb Cl. Conc. Pb Ro. Conc.						
							Assay, %			Distribution, %			
		Ro.	Cl.	Ro.	Cl.		Cu	Pb	Zn	Wt.	Cu	Pb	Zn
44	70.2	1800	450	500	550	83	0.19 0.51	56.9 14.1	6.67 11.20	2.58 22.76	0.4 9.3	40.5 79.5	2.0 26.5
45	78.9	2400	450	500	375	87	0.28 0.38	58.1 17.2	6.20 10.80	2.02 18.37	0.5 5.8	29.1 78.4	1.2 20.3
47	78.9	2000	265	600	625	83	0.15 0.24	46.6 14.7	8.11 11.5	5.55 21.54	0.7 4.4	63.1 77.3	4.6 25.6
48	58.9	1600	270	600	625	93	0.19 0.22	45.1 14.7	9.93 13.50	5.81 20.27	0.9 3.7	64.2 74.9	6.0 29.1

A few batch tests were performed for zinc recovery because the feed for the zinc circuit could only be produced in locked cycle tests involving recirculation of Cu and Pb cleaner tailing. The tests indicated that a final zinc concentrate of 55% Zn at 62% recovery could be obtained in a batch test.

13.1.4 Locked Cycle Flotation Tests

Locked cycle tests were performed using the selective flotation procedure to determine the effect of recirculation of cleaner products on the grades and recoveries of Cu, Pb, and Zn.

Several LCT tests were performed using slight variations of the flowsheet given in Figure 13.1. The test conditions are given in Table 13.7 and the best results are presented in Table 13.8. The impurity analyses are given in Table 13.9. The results indicate the following:

- Copper concentrate, assaying 23.1% Cu, 3.64% Pb, 2.85% Zn, 1.90 g/t Au, and 493 g/t Ag, recovered 77.4% of copper in the feed.
- The lead concentrate, assaying 50.9% Pb, 7.85% Zn, 0.34% Cu, 2.02 g/t Au, and 494 g/t Ag, recovered 77.5% of lead in the feed.
- The zinc concentrate, assaying 53% Zn, 1.57% Pb, 0.86% Cu, 0.46% g/t Au, and 51.6 g/t Ag recovered 87.7% of the zinc in the feed.
- The concentrate did not contain detrimental elements to trigger penalties.

TABLE 13.7
LOCKED-CYCLE TEST CONDITIONS

Process Parameters	Reagents, g/t			Grind P_{80}, µm	Flotation Time, min	pH
	SO_2	M200	A325			
COPPER CIRCUIT						
Primary Grind	1000	-	-	37	-	-
Conditioner	-	25	10	-	1	6.3
Rougher Float	-	-	-	-	9	-
Regrind	340	-	-	21	-	-
1st Cleaner	-	12.5	5.0	-	6	5.5
2nd Cleaner	-	-	-	-	4	5.0
3rd Cleaner	-	-	-	-	3	4.5
4th Cleaner	-	-	-	-	2	4.0

Table 13.7
Locked-cycle Test Conditions
(continued)

Process Parameters	Reagents, g/t				Flotation Time, min	pH
	Lime	$ZnSO_4$/NaCN (ratio)	A 325	MIBC		
LEAD CIRCUIT						
Conditioner	2125	600 (2:1)	45	-	1	9.5
Rougher Float	-	-	-	-	9	9.5
Regrind	235	525	-	14	-	-
1st Cleaner	-	-	33	-	6	9.5
2nd Cleaner	-	-	-	-	5	9.5
3rd Cleaner	-	-	-	-	4	9.6
4th Cleaner	-	-	-	-	3	9.5

Table 13.7
Locked-cycle Test Conditions
(continued)

Process Parameters	Reagents, g/t				Grind P_{80}, µm	Flotation Time, min	pH
	Lime	$CuSO_4$	A343	M200			
ZINC CIRCUIT							
Conditioner	1750	1000	40	10	-	-	11.9
Rougher Float	-	-	-	-	-	9	12.0
Regrind	1250	150	-	-	22	-	12.0
1st Cleaner	-	-	10	5		10	
2nd Cleaner	-	-	-	-	-	6	12.1
3rd Cleaner	-	-	-	-	-	5	12.3
4th Cleaner	-	-	-	-	-	4	12.4
Final Tailing	-	-	-	-	29	-	-

Table 13.8
Locked-cycle Test Results (Test 49)

Product	Assay %, g/t					Distribution, %					
	Cu	Pb	Zn	Au	Ag	Wt.	Cu	Pb	Zn	Au	Ag
Cu Cl. Conc.	23.1	3.64	2.85	1.90	493	4.26	77.4	3.8	1.2	13.3	27.3
Pb Cl. Conc.	0.34	50.9	7.85	2.02	494	6.16	1.6	77.5	4.8	20.4	39.6
Zn Cl. Conc.	0.86	1.57	53.0	0.46	51.6	16.63	11.2	6.5	87.7	12.5	11.1
Zn Comb. Tail	0.17	0.68	0.87	0.45	23.2	72.95	9.8	12.2	6.3	53.8	22.0
Cal. Head	1.27	4.05	10.05	0.61	76.9	100.0	100.0	100.0	100.0	100.0	100.0

Table 13.9
Impurity Analyses of Concentrates

Element, %	Cu Conc.	Pb Conc.	Zn Conc.
Fe	30.6	13.7	9.04
S	36.8	26.4	33.4
As	0.85	0.13	0.038
Sb	0.47	0.076	0.013
Bi	0.014	0.096	0.007
Cd	0.01	0.024	0.015
Hg	0.0011	0.0018	0.0029
SiO_2	0.40	0.09	0.77
Al_2O_3	0.15	0.029	0.24
MgO	0.036	0.007	0.049
CaO	0.004	0.034	0.11

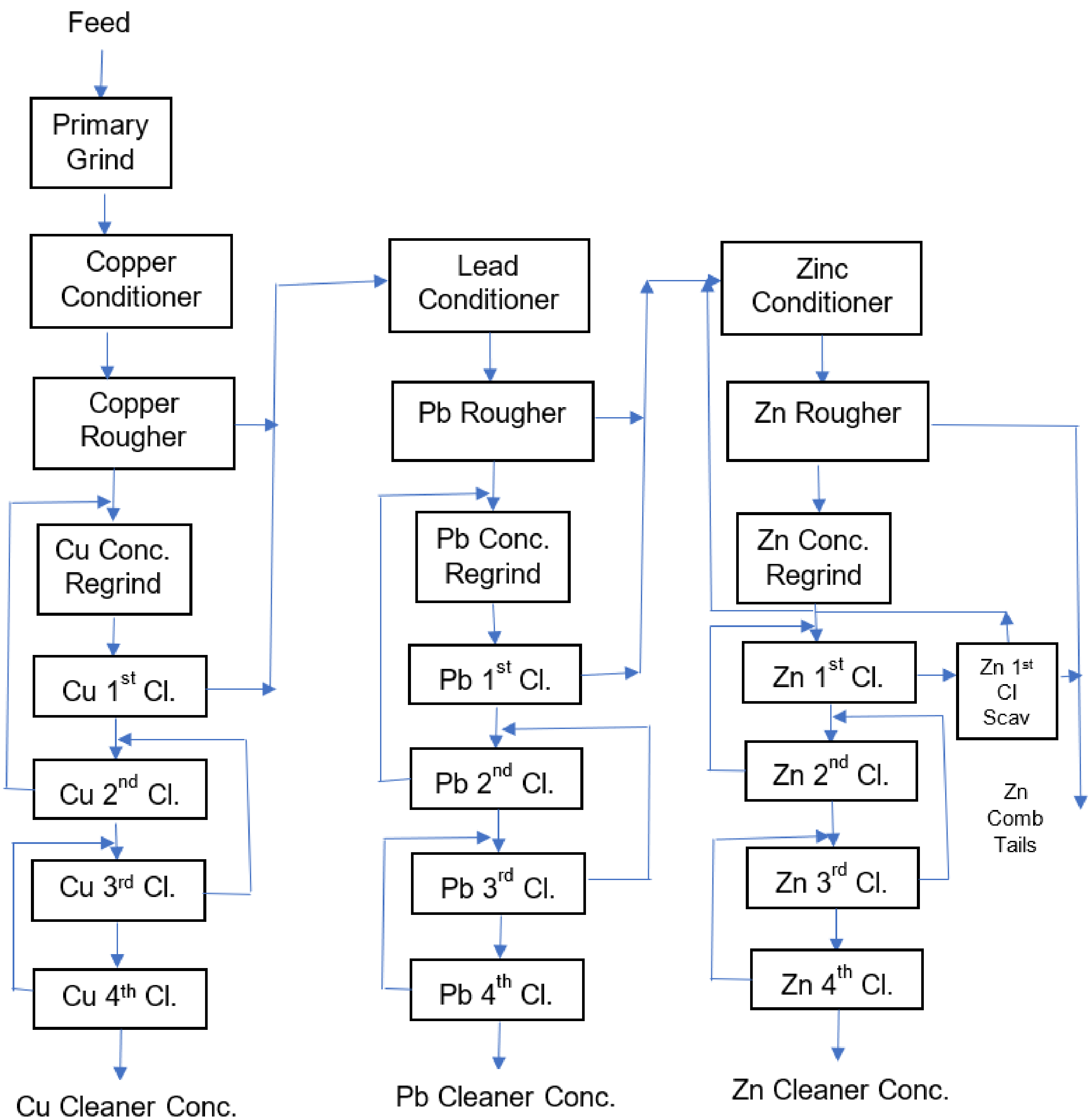


Figure 13.1 Locked-cycle Test Flowsheet

13.1.5 Miscellaneous Tests

Open-circuit sequential flotation tests were performed on Composites 2 to 5. The results were similar to those obtained for Composite 1 except for Composite 5 which gave inferior copper recovery and copper concentrate from Composite 3 assayed over 1% As which will have to be blended with other concentrates to avoid penalty for impurities.

The aging of samples at ambient atmosphere resulted in poor copper and lead recovery as compared to samples stored in the freezer.

The tailing settled to ±63% solids and was characterised to be strong acid producer.

13.2 Chevron Resources, 1988

Lakefield Research undertook scoping level metallurgical test work for Chevron Resources using Composite 1 stored in the freezer from 1984 studies.

The primary objective of the study was to determine if bulk flotation flowsheet for treatment of the ore could be utilised for recovery of Cu, Pb and Zn products. Two flowsheets were examined in the test work:

1. Cu-Pb bulk flotation flowsheet followed by Cu-Pb cleaning, Cu-Pb separation, and zinc flotation and upgrading.

2. A Cu-Pb-Zn bulk flotation followed by Cu-Pb and zinc separation, Cu-Pb upgrading, and separation and zinc upgrading.

Only open-circuit flotation tests were performed. The projected metallurgical results for the two flowsheets using batch test data are given in Table 13.10.

TABLE 13.10
PROJECTED METALLURGICAL RESULTS FROM BATCH TESTS USING DIFFERENT FLOWSHEETS

Flowsheet	Product	Assay %			Distribution, %			
		Cu	Pb	Zn	Wt.	Cu	Pb	Zn
Cu-Pb Bulk Flotation	Cu Cl. Conc.	28.10	2.5	3.0	3.53	80.0	2.3	1.1
	Pb Cl. Conc.	1.64	53.2	10.5	5.49	7.2	75.0	5.8
	Zn Cl. Conc.	0.42	1.0	56.0	15.71	5.4	3.9	88.0
	Zn Comb. Tailing	0.12	0.97	0.68	75.27	7.4	18.8	5.1
	Cal. Head	1.24	3.90	10.00	100	100	100	100
Cu-Pb-Zn Bulk Flotation	Cu Cl. Conc.	29.00	2.0	2.5	3.21	75.0	1.6	0.8
	Pb Cl. Conc.	1.50	55.0	10.1	5.53	6.7	78.0	5.6
	Zn Cl. Conc.	0.50	1.6	55.5	16.10	6.5	6.6	88.5
	Zn Comb. Tailing	0.19	0.71	0.57	75.16	11.8	13.8	5.1
	Cal. Head	1.24	3.90	10.10	100	100	100	100

The conclusions of this study were as follows:

1. The Mount Chase (Pickett Mountain) material belongs to a group of refractory finely disseminated Cu-Pb-Zn massive sulfide ores from which the separation of the individual minerals and production of separate Cu, Pb, and Zn concentrates represents significant mineral processing problems.

2. Irrespective of the process flowsheet selected, large quantities of depressants would be needed to produce marketable-grade concentrates of Cu, Pb, and Zn.

3. The rate of Cu-Pb flotation in either of the two flotation schemes evaluated was relatively low and significant losses of copper and lead occurred in the cleaner tailings.

13.3 Wolfden Resource Corporation, 2019

Resource Development Inc. (RDi) undertook scoping level metallurgical test work for Wolfden Resource Corp. in 2019 with the primary objective of determining metal recoveries and flotation concentrate grades for the mineralised material from the Pickett Mountain deposit. The program investigated both sequential and differential flotation schemes to separate Cu, Pb, and Zn from the polymetallic ore and produce final concentrates. The concentrates were analysed for impurities. In addition, flotation tailings were characterised for environmental purposes.

13.3.1 Sample Preparation and Characterisation

RDi received 37 samples weighing approximately 0.6 to 2 kg each for the test work. A master composite sample was created utilising each individual sample. The head analyses of the master composite are given in Table 13.11. The master composite assayed 1.2% Cu, 3.4% Pb, 8.6% Zn, 27.4% S_{Total}, 0.7 g/mt Au, and 95 g/mt Ag. The sample was a massive sulfide but approximately 21% of the sulfur was present as sulfate sulfur. Hence, **the sample may have been oxidised**.

TABLE 13.11
HEAD ANALYSES OF COMPOSITE SAMPLES INCLUDING ICP

Element	Master Composite
Au, g/mt	0.704
Ag, g/mt	95
Sulfide S %	21.7
Sulfate S %	5.69
Total S %	27.4
%	
Al	3.00
Ca	0.41
Cu	1.21
Fe	21.1
K	1.00
Mg	0.97
Na	0.27
Pb	3.37
Ti	0.05
Zn	8.56
ppm	
As	953
Ba	1060
Bi	81
Cd	246
Co	33
Cr	141
Mn	365
Mo	<1
Ni	26
Sr	17
V	25
W	518

13.3.2 Bond's Ball Mill Work Index

Bond's ball mill work index was determined for the master composite sample at a closed size of 100 mesh (150 microns). The BW_i of 10.96 kwh/st indicated that the sample will be considered soft to medium hardness.

13.3.3 Rougher Flotation Testing

The rougher flotation test for both schemes, namely bulk-flotation of Pb/Cu and sequential flotation utilised similar promotors and depressants but different pulp pHs. Aero 5100 promoter was used to collect the copper due to a good selectively against iron sulfides and SIPX/Aero 3418A was used to recover lead. After the copper/lead flotation, both approaches utilised the same process to collect the zinc and remaining sulfides. The zinc was activated with copper sulfate and then collected with SIPX. The remaining sulfides left in the sample were then floated with a combination of PAX/AP404 to attempt to create a benign tail. The primary grind was varied between P_{80} 200 mesh and P_{80} 325 mesh and various amounts of depressants were added to determine if the separation was more sensitive to liberation size or reagent dosages. The differential flotation data is summarised in Table 13.12, while the sequential flotation data is summarised in Table 13.13.

TABLE 13.12
DIFFERENTIAL FLOTATION RESULTS

Product	Recovery %						Product Grade				
	Wt.	Au	Ag	Pb	Zn	Cu	Au (g/mt)	Ag (g/mt)	Pb (%)	Zn (%)	Cu (%)
FT-1 (200 mesh, Standard Depressants)											
Cu/Pb Ro Conc	36.0	76.0	84.2	89.0	55.9	89.0	1.7	215	7.8	13.3	3.1
Zn Ro Conc	27.8	20.6	12.4	8.5	42.7	7.4	0.6	40.8	1.0	13.2	0.3
Pyrite Ro Conc	2.7	1.3	1.1	0.7	0.5	1.1	0.4	37.5	0.8	1.6	0.5
Combined Ro Conc	66.5	98.0	97.7	98.2	99.1	97.6	1.2	135	4.6	12.8	1.8
Ro Tail	33.5	2.0	2.3	1.8	0.9	2.4	<0.10	6.2	0.17	0.22	0.09
Calc. Feed	100	100	100	100	100	100	0.82	92	3.14	8.58	1.25
FT-4 (200 mesh, 2X Depressants)											
Cu/Pb Ro Conc	24.9	80.5	81.8	81.0	38.7	90.1	2.4	339	5.9	11.0	4.6
Zn Ro Conc	17.9	13.1	9.4	9.9	59.3	4.7	0.5	54.2	1.0	23.3	0.3
Pyrite Ro Conc	2.2	2.7	1.2	1.1	0.4	0.9	0.9	53.4	0.9	1.3	0.5
Combined Ro Conc	45.0	96.3	92.4	92.1	98.4	95.6	1.6	212	3.7	15.4	2.7
Ro Tail	55.0	3.7	7.6	7.9	1.6	4.4	<0.10	14.2	0.26	0.2	0.10
Calc. Feed	100	100	100	100	100	100	0.74	103	1.80	7.03	1.26

Table 13.13
Sequential Flotation Results

Product	Recovery %						Product Grade				
	Wt.	Au	Ag	Pb	Zn	Cu	Au (g/mt)	Ag (g/mt)	Pb (%)	Zn (%)	Cu (%)
FT-2 (200 mesh, Standard Depressants)											
Cu Ro Conc	9.5	42.9	44.7	13.9	6.4	78.8	3.0	427	4.1	5.7	9.8
Pb Ro Conc	10.8	16.3	29.9	63.0	18.8	8.0	1.0	251	16.3	14.7	0.9
Zn Ro Conc	32.2	30.5	18.7	17.4	73.2	8.0	0.6	52.5	1.5	19.2	0.3
Pyrite Ro Conc	8.0	7.3	2.8	2.0	0.6	1.9	0.6	31.8	0.7	0.6	0.3
Combined Ro Conc	60.4	97.0	96.1	96.2	98.9	96.7	1.1	144	4.4	13.8	1.9
Ro Tail	39.6	3.0	3.9	3.8	1.1	3.3	<0.10	8.8	0.27	0.22	0.10
Calc. Feed	100	100	100	100	100	100	0.66	90	2.79	8.43	1.18
FT-3 (200 mesh, 1.5X Depressants)											
Cu Ro Conc	10.4	47.3	48.6	24.4	9.0	81.5	3.2	486	4.3	6.5	9.5
Pb Ro Conc	12.2	17.7	25.6	40.6	19.2	5.4	1.0	219	6.1	11.8	0.5
Zn Ro Conc	34.2	29.6	21.0	26.4	69.8	8.7	0.6	64.3	1.4	15.3	0.3
Pyrite Ro Conc	4.7	2.7	2.6	2.8	0.6	1.5	0.4	58.2	1.1	1.0	0.4
Combined Ro Conc	61.5	97.3	97.8	94.1	98.7	97.2	1.1	166	2.8	12.0	1.9
Ro Tail	38.5	2.7	2.2	5.9	1.3	2.8	<0.10	6.0	0.28	0.26	0.09
Calc. Feed	100	100	100	100	100	100	0.71	104	1.83	7.50	1.22
FT-5 (325 mesh, 1.5X Depressants)											
Cu Ro Conc	10.6	41.4	52.0	19.0	6.9	85.7	4.5	492	3.1	5.2	9.9
Pb Ro Conc	12.6	24.5	30.6	48.8	19.2	5.3	2.3	244	6.7	12.1	0.5
Zn Ro Conc	33.0	30.4	12.8	22.3	71.8	5.3	1.1	39.0	1.2	17.4	0.2
Pyrite Ro Conc	2.9	1.9	1.0	1.6	0.4	0.7	0.8	35.1	1.0	1.2	0.3
Combined Ro Conc	59.2	98.2	96.4	91.8	98.4	97.0	1.9	164	2.7	13.3	2.0
Ro Tail	40.8	1.8	3.6	8.2	1.6	3.0	0.41	9.0	0.35	0.31	0.09
Calc. Feed	100	100	100	100	100	100	1.16	101	1.74	7.97	1.23

The scoping level rougher flotation test results indicate the following:

- The differential and sequential approaches floated nearly all of the metals contained in the sample. Each individual metal recovery ranged from approximately 91% to 99% once all concentrates were combined.

- Differential flotation testing recovered 81% to 90% of the lead and copper into the first rougher concentrate, but 39% to 56% of the zinc was also recovered during this stage that decreased the zinc recovery to the zinc concentrate to below 60%. Additional depressants decreased the amount of zinc reporting to the copper/lead concentrate, but not to an acceptable level.

- Sequential flotation testing recovered as much as 86% of the copper, 63% of the lead, and 73% of the zinc into their respective concentrates. Approximately 14% to 24% of the lead reported to the copper concentrate and 19% of the zinc reported to the lead concentrate.

- A finer grind to 325 mesh improved copper recovery to the copper concentrate. Increased depressant additions had a negative effect on lead recovery and little effect on zinc recovery.

- The majority of the gold and silver present in the sample reported to the copper concentrate during sequential flotation (41% to 52%). Approximately 20% of the gold and 30% of the silver reported to the lead concentrate, while the rest of the precious metals were collected in the zinc concentrate.

- The rougher flotation tailings still contain approximately 2.5% sulfide sulfur after the pyrite flotation stage.

13.3.4 Cleaner Flotation Testing

Based on the scoping level rougher flotation test results, the sequential flotation scheme was selected for cleaner flotation testing.

A series of rougher flotation tests were completed with the composite sample to produce rougher copper, lead, and zinc concentrates for cleaner flotation testing. A sequential flotation process was utilised at a primary grind of P_{80} 325 mesh with standard depressant additions. A summary of the flotation results is given in Table 13.14.

TABLE 13.14
ROUGHER FLOTATION RESULTS - CONCENTRATE PRODUCTION

Product	Recovery %						Product Grade				
	Wt.	Au	Ag	Pb	Zn	Cu	Au (g/mt)	Ag (g/mt)	Pb (%)	Zn (%)	Cu (%)
FT-Production (325 mesh, Standard Depressants)											
Cu Ro Conc	9.6	29.0	49.6	18.2	6.0	84.7	2.8	505	3.01	5.35	11.6
Pb Ro Conc	14.5	34.3	33.4	55.0	29.3	8.1	2.2	225	6.03	17.3	0.73
Zn Ro Conc	23.9	23.2	11.3	17.4	62.7	4.0	0.9	46	1.16	22.4	0.22
Pyrite Ro Conc	11.7	11.3	2.9	3.6	0.9	1.4	0.9	24	0.49	0.63	0.16
Combined Ro Conc	59.8	97.8	97.1	94.2	98.9	98.2	1.52	159	2.51	14.2	2.16
Ro Tail	40.2	2.2	2.9	5.8	1.1	1.8	0.3	7	0.23	0.24	0.06
Calc. Feed	100	100	100	100	100	100	0.93	98	1.59	8.86	1.32

The rougher flotation results were similar to the previous results observed in test FT2 with more lead reporting to the copper concentrate and more zinc reporting to the lead concentrate. Cleaner flotation tails from the copper circuit would be sent to the lead circuit and the lead cleaner tails would be sent to the zinc circuit in commercial operations. This can be simulated in locked-cycle tests, which should be undertaken with freshly drilled samples in the next phase of the study.

Representative splits of the rougher concentrates were then used for the subsequent cleaner testing. Testing was completed with and without regrind of the rougher concentrates. The cleaner flotation data are summarised in Table 13.15 to Table 13.17.

Table 13.15
Copper Cleaner Results

Product	Cumulative Recovery %						Product Grade				
	Wt.	Au	Ag	Pb	Zn	Cu	Au (g/mt)	Ag (g/mt)	Pb (%)	Zn (%)	Cu (%)
FT-6 (No Regrind)											
Cu CL3 Conc	24.8	27.2	42.9	35.1	24.2	52.9	8.78	851	4.28	5.43	23.6
Cu CL2 Conc	36.1	48.1	55.6	49.8	32.8	65.8	10.66	757	4.16	5.05	20.2
Cu CL1 Conc	53.8	69.1	73.6	71.1	50.0	81.3	10.24	670	3.98	5.15	16.7
Calc Feed	100	100	100	100	100	100	7.98	491	3.02	5.55	11.0
FT-7 (7.5 minute Regrind)											
Cu CL3 Conc	16.1	22.1	38.9	8.2	6.7	41.0	5.62	1170	1.49	2.21	27.1
Cu CL2 Conc	23.5	35.3	51.2	14.3	11.6	54.6	6.19	1059	1.79	2.63	24.8
Cu CL1 Conc	33.0	49.6	63.0	25.5	21.6	65.5	6.19	927	2.28	3.48	21.2
Calc Feed	100	100	100	100	100	100	4.11	485	2.94	5.32	10.6

Table 13.16
Lead Cleaner Results

Product	Cumulative Recovery %						Product Grade				
	Wt.	Au	Ag	Pb	Zn	Cu	Au (g/mt)	Ag (g/mt)	Pb (%)	Zn (%)	Cu (%)
FT-6 (No Regrind)											
Pb CL3 Conc	18.0	35.5	39.3	19.3	15.0	21.1	2.95	500	6.63	14.8	0.82
Pb CL2 Conc	28.1	54.1	52.9	29.4	25.1	32.0	2.87	430	6.48	15.8	0.79
Pb CL1 Conc	41.5	73.0	67.2	45.1	37.8	47.2	2.64	371	6.73	16.1	0.79
Calc Feed	100	100	100	100	100	100	1.50	229	6.19	17.7	0.70
FT-7 (5 minutes Regrind)											
Pb CL3 Conc	7.3	15.1	21.9	9.8	3.3	10.0	2.76	677	9.08	8.00	1.01
Pb CL2 Conc	14.7	26.8	36.8	16.2	9.1	18.6	2.45	575	7.56	11.1	0.94
Pb CL1 Conc	28.7	49.0	54.9	30.1	21.1	33.8	2.29	439	7.17	13.2	0.88
Calc Feed	100	100	100	100	100	100	1.34	229	6.83	17.9	0.74

Table 13.17
Zinc Cleaner Results

Product	Cumulative Recovery %						Product Grade				
	Wt.	Au	Ag	Pb	Zn	Cu	Au (g/mt)	Ag (g/mt)	Pb (%)	Zn (%)	Cu (%)
FT-6 (No Regrind)											
Zn CL3 Conc	27.2	39.5	23.8	17.6	73.7	14.7	1.44	39	0.70	56.1	0.13
Zn CL2 Conc	34.1	42.9	33.1	27.6	83.6	21.1	1.25	43	0.87	50.7	0.15
Zn CL1 Conc	43.9	49.7	47.1	43.5	90.8	34.5	1.12	48	1.07	42.8	0.19
Calc Feed	100	100	100	100	100	100	0.99	45	1.08	20.7	0.24
FT-7 (10 minutes Regrind)											
Zn CL3 Conc	20.7	10.0	17.2	10.7	60.6	15.4	0.27	35	0.61	62.3	0.18
Zn CL2 Conc	26.1	14.9	22.6	16.0	68.2	19.9	0.32	36	0.72	55.7	0.18
Zn CL1 Conc	39.5	32.9	38.2	32.8	79.2	34.8	0.47	41	0.98	42.8	0.21
Calc Feed	100	100	100	100	100	100	0.56	42	1.18	21.3	0.24

The open-circuit cleaner flotation results indicate the following:

- The copper cleaner circuit achieved a grade of 27.1% Cu after three stages of cleaners with one stage of regrind. Copper recovery was low at 41.0% in the open-circuit test. The recovery increased to 52.9% without regrind, but the concentrate grade was reduced to 23.6% Cu. The regrind stage also helped to reduce the lead and zinc grades to 1.49% Pb and 2.21% Zn as opposed to 4.28% Pb and 5.43% Zn without regrind.

- The lead cleaner circuit did not recover a significant amount of the lead or upgrade the concentrate. The maximum lead grade was 9.08% Pb with only 9.8% recovery. In addition, the zinc grade was high at 8.00% Zn. The poor metallurgical performance could potentially be due to aging of the samples. Additional tests are needed to optimise the lead circuit with fresh drill core.

- The zinc cleaner circuit achieved a grade of 62.3% Zn and recovery or 60.6% after three stages of cleaners with one stage of regrind. The recovery increased to 73.7% without regrind, while the concentrate grade was reduced to 56.1% Zn. The regrind stage had little effect on the copper and lead grades. The copper grade was 0.18% Cu while the lead grade was 0.61% Pb with regrind.

13.3.5 Vacuum Filtration Test

Vacuum filtration testing was completed with the rougher flotation tails generated during flotation testing at P_{80} 200 mesh and P_{80} 325 mesh. Portions of each flotation tail sample were slurried to approximately 40% solids and placed in a vacuum filtration apparatus. The vacuum was initiated, and the slurry was allowed to filter until a cake was formed with no visible moisture on the top of the cake. Data was collected to determine the form time and cake moisture and thickness to determine the filtration rate. Additional tests were completed that included multiple dry times and varied cake thicknesses. The dry time is the additional time that the vacuum is engaged after the cake has been formed to determine if additional moisture can be removed from the cake. The filter data is summarised in Table 13.18.

TABLE 13.18
VACUUM FILTRATION DATA OF ROUGHER FLOTATION TAILS

Sample	Dry Time (min)	Cake Thickness (mm)	% Solids of Filter Cake	Filtration Rate (Dry lb./ft^2/hr.)
FT-4 Tails (200 mesh)	0	6.4	74.4	190.7
FT-4 Tails (200 mesh)	1	6.4	75.6	81.8
FT-4 Tails (200 mesh)	1	3.2	74.9	54.7
FT-5 Tails (325 mesh)	0	7.9	66.9	157.5
FT-5 Tails (325 mesh)	1	7.9	68.6	80.8
FT-5 Tails (325 mesh)	1	4.0	67.9	57.4

The flotation tails filtration results indicate the following:

- A maximum percent solid of 75.6% was achieved during testing with the 200-mesh sample, while a percent solid of 68.6% was achieved with the 325-mesh sample.

- A filtration rate of approximately 190 dry lb./ft^2/hr. was achieved with vacuum filtration at 200 mesh and approximately 160 dry lb./ft^2/hr. at 325 mesh.

13.3.6 Smelter Penalty Analyses

The copper and zinc concentrates were analysed for smelter penalty elements. The analyses, given in Table 13.19, indicate that the copper concentrate could be penalised for arsenic and antimony.

TABLE 13.19
METAL ANALYSIS OF CLEANED CONCENTRATE SAMPLES

Element	Third Stage Cleaner Cu Concentrate (FT6)	Third Stage Cleaner Zn Concentrate (FT7)
Au, g/mt	8.78	0.27
Ag, g/mt	851	35
%		
Al	7.80	3.25
Ca	0.04	0.07
Cu	23.6	0.18
Fe	26.6	2.48
K	<0.01	<0.01
Mg	0.05	0.04
Na	<0.01	<0.01
Pb	4.27	0.61
Ti	<0.01	<0.01
Zn	5.43	62.3
ppm		
As	9340	90
Ba	71	48
Bi	<2	<2
Cd	159	1740
Co	23	3
Cr	<1	<1
Hg	10.2	25.2
Mn	37	63
Mo	15	3
Ni	<5	<5
Sb	5190	30
Se	6	<5
Sr	4	3
Te	88	39
V	<1	<1
W	<10	<10

In 1984, Getty contracted A.H. Ross & Associates to complete a metallurgical test work program at Lakefield Research of Canada Limited (Lakefield). Lakefield developed an ore treatment process and established information on likely product composition, plant tailings, and water characteristics. Based on the Lakefield work, a process flowsheet and material balance were determined (Bosch and Grimes, 1984).

A composite sample was submitted for study based on three locked-cycle flotation tests. It is not known how representative this sample was to the various types and styles of mineralisation and the mineral deposit as a whole. The grade of the composite sample, the head grade for the study, was (Bosch and Grimes, 1984):

Copper	–	1.32%
Lead	–	4.29%
Zinc	–	9.72%
Gold	–	0.022 opt
Silver	–	2.66 opt

The sample was subjected to conventional grinding involving primary crushing, followed by grinding with a rod mill, followed by further grinding in a ball mill, with final output being 80% minus 400 mesh. The output was reclassified using a cyclone with oversize going back to the grinding circuit. The cyclone slurry, with about 33% solids, was passed directly to the flotation circuit. It was found that a sequential flotation of the Cu, Pb, and Zn minerals was better than a bulk Cu-Pb flotation (Bosch and Grimes, 1984). It is not known to what extent there are any processing factors or deleterious elements that could have a significant effect on potential economic extraction.

The flotation test resulted in the following recoveries (Bosch and Grimes, 1984):

	Cu Con.	Pb Con.	Zn Con.
Copper	**77.4%**	1.6%	11.2%
Lead	3.8%	**77.5%**	6.5%
Zinc	1.2%	4.8%	**87.7%**
Gold	13.3%	20.4%	12.5%
Silver	27.3%	39.6%	11.1%

It should be noted that the above mineral processing and metallurgical test work comprises historical work and requires verification and updating, given that technology in this field has improved in the last 35 years.

13.4 Conclusions

The scoping level test work has indicated that a sequential flotation process will produce marketable-grade copper, lead, and zinc concentrates. Arsenic and antimony levels were high in copper concentrates produced in open-cycle and locked-cycle tests. **Additional geo-metallurgical test work will provide additional information on the impurities in the marketable-grade copper concentrate to determine if penalties needs to be paid**. In addition, blending of ores from different areas in the mine will keep impurities (As/Sb) below penalty levels.

14.0 Mineral Resource Estimate

14.1 Introduction

The authors were retained by AMPL to update the Pickett Mountain Project Deposit Resources in accordance with the guidelines of NI 43-101 and CIM standards. This resource estimate was undertaken by Mr. Finley Bakker, P. Geo. of Campbell River, British Columbia. The Mineral Resource used in this Preliminary Economic Assessment is an update from the A-Z Mining Professionals Ltd. report titled "National Instrument 43-101 Technical Report, Pickett Mountain Project Resource Estimation Report," Penobscot County, Maine, USA, located at: 68.468°W Longitude 46.134°N Latitude and effective January 7, 2019 and is based on the drill results received during the 2018 drilling program as well as the historical drill results.

The resource estimate, as used in this study and reported in a press release on September 14, 2020, utilises the 7% cut-off grade (or an NSR value of $139/t) rather than the previous 9% cut-off grade ($178/t NSR) portion as reported in the base case of the January 7, 2019 statement. No other changes were made. The exploration and drilling results completed on the property in 2019 were limited and, therefore, were considered to have no material impact on the updated mineral resource statement.

14.2 Database

The database used data initially verified in "National Instrument 43-101 Technical Report – Pickett Mountain Project," Penobscot County, Maine, USA, located at: 68.468°W Longitude 46.134°N Latitude – Prepared for Wolfden Resources Corporation, by Mr. Alan Aubut, P.Geo., April 02, 2018.

The database was updated and verified by Mr. Jerry Grant, P.Geo. and included the results from 2018 diamond drill program, a total of 34 holes or intersections. Only a few holes drilled after November 15, 2018 (after hole PM-18-029A) were not included in the resource estimate.

A total of 148 diamond drill holes made up the database and included 2,550 samples. Of these samples, approximately 940 samples were used in the resource calculation. These samples constituted 104 intervals over 4 lens codes (wireframes).

14.3 Data Verification

As due diligence had been undertaken in a previous NI 43-101 Technical Report, titled "National Instrument 43-101 Technical Report Pickett Mountain Project," Penobscot County Maine, USA, located at: 68.468°W Longitude 46.134°N Latitude – Prepared for Wolfden Resources Corporation by Mr. Alan Aubut, P.Geo., April 02, 2018, its data was assumed correct. The authors had no reason not to rely on the information that they may have referenced. However, spot checks of historical data were still undertaken. No serious errors or omissions were found.

Information added to the database was verified by Mr. Don Hoy, P.Geo., Mr. Andre Labonte, and Mr. Jerry Grant, P.Geo., and was further confirmed by the authors.

Sample analyses were undertaken by ACTIVATION LABORATORIES LTD., 41 Bittern Street, Ancaster, Ontario, Canada, L9G 4V5 Telephone: +1.905.648-9611 or +1.888.228.5227; Fax: +1.905.648.9613; Activation Laboratories Ltd. (Act labs) is ISO 17025 accredited and/or certified to 9001: 2008.

- Code 1A2-Au – Fire Assay AA
- Code 1E-Ag Aqua Regia ICP (AQUAGEO)
- Code 8-Peroxide ICP Sodium Peroxide Fusion ICP
- Check assays were undertaken at Act Labs in Kamloops.
- Code 1A2-Kamloops Au – Fire Assay AA
- Code 1A3-Ag-Kamloops Au – Ag-Fire Assay Gravimetric (QOP Fire Assay Thunder Bay)
- Code 1E3-Kamloops Aqua Regia ICP (AQUAGEO)
- Code 8-Peroxide ICP-Kamloops Sodium Peroxide Fusion ICP

Analyte Symbol	Au	Ag	Cu	Pb	Zn	Au	Ag
Unit Symbol	ppb	ppm	%	%	%	g/tonne	g/tonne
Lower Limit	5	0.2	0.005	0.01	0.01	0.03	3
Method Code	FA-AA	AR-ICP	FUS-Na2O2	FUS-Na2O2	FUS-Na2O2	FA-GRA	FA-GRA

The location of the recent diamond drill holes was verified by Jerry Grant, as he was on site during the drill program. All data was collected in NAD-83 format.

14.4 Domain Interpolation

The various domains were interpreted based on mineralogy, lithology, and grade. It was felt by the authors that the 3D Block Model would resolve any grade continuity issues as part of the interpolation.

Wireframe models were created by Andre Labonte using Gems™ 3D modeling software. These wireframes were then imported into Hexagon™/MineSight™ and validated.

It was ensured that there was no overlapping of lenses. Any overlaps, based on block size, were trimmed using MineSight™ software to negate any double reporting of tonnes (Figure 14.1, Figure 14.2, and Figure 14.3).

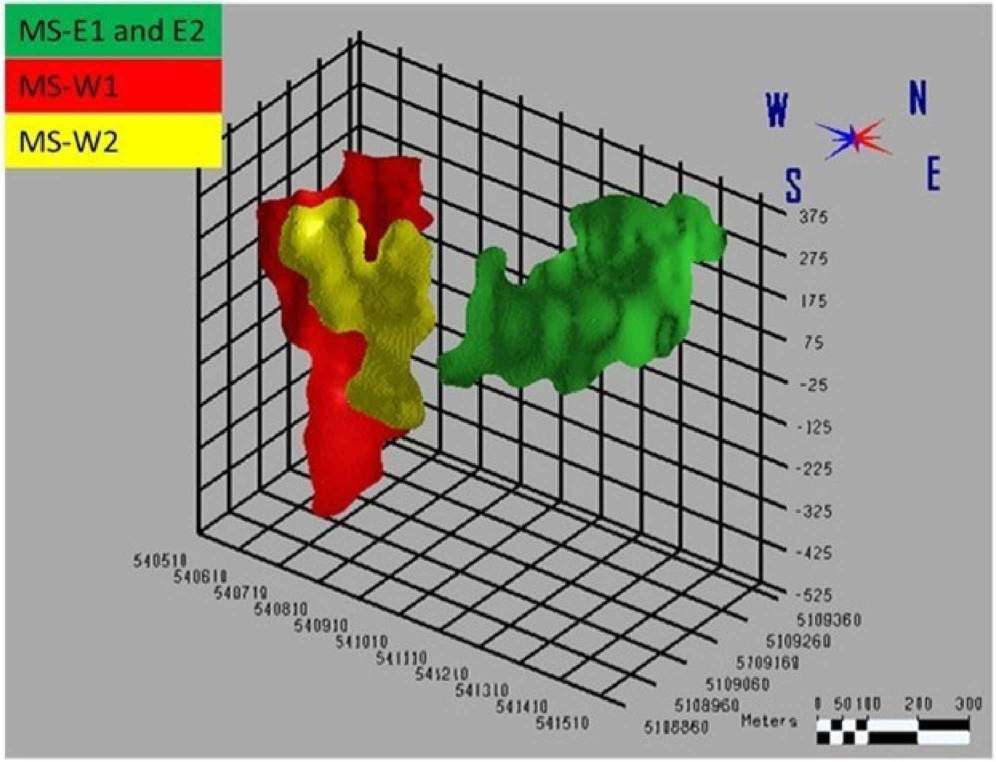


Figure 14.1 Wireframes

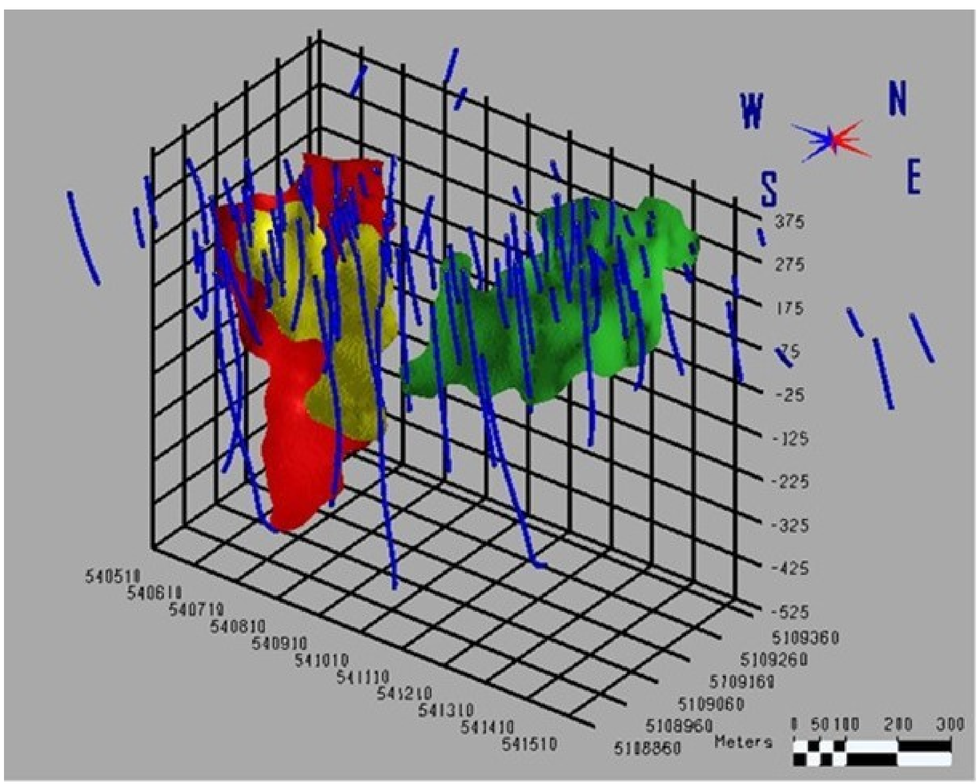


Figure 14.2 Pre-2018 Diamond Drilling

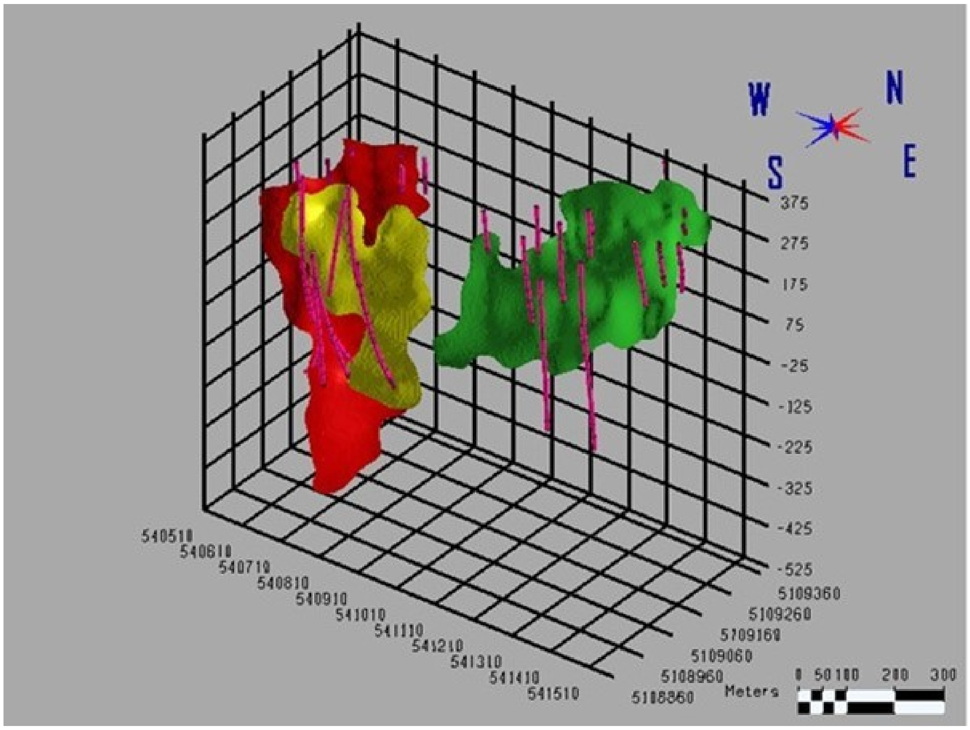


Figure 14.3 2017/2018 Showing 2018 Diamond Drilling

14.5 Lens/Rock Code

The original work centered on 4 lens codes. However, two of these were combined after density was applied, such that only three wireframe lenses were used in the estimate.

- MS-E1 and MS-E2
- MS-W1
- MS-W2

14.6 Composites

Two sets of composites were created for calculating purposes.

1. **Composite 1** – Composited metal grades over the entire lens code. This was done to ensure that the entire interval used in the calculation could be put into an Excel™ spreadsheet and compared to the block model. Calculations were limited by lens code (LENS). These intervals are reported in Table 14.5, below.

2. **Composite 2** – Composited metal grades were limited to lens code but assigned a maximum of 1m. Calculations were limited by lens code (LENS). These composites were used in the grade interpolation.

14.7 Grade Capping

Grade-capping was not performed based on histograms and the QP's experience. As well, observations by Wolfden and J. Grant, during a review of the massive sulphide intercepts, note that no obvious secondary enrichment was observed, with massive sulphide grades generally correlating with the strength of the footwall alteration, or, local structural sub-basins where more focused hydrothermal activity would be expected. Overall, it was noted that the uniformity of grade was reasonably consistent and well correlated with geological controls and without any significant outliers in the assay results.

14.8 Variography

A series of variograms were created using Hexagon™ program, MSDA; this allowed a series of 3D variograms to be created based on lens code.

These 3D variograms were imported into MineSight™.

Surprisingly, the zinc variogram shows the weakest continuity with the weakest orientation of only 21m. As such, 25m was chosen as the distance from the diamond drill hole for the Indicated Resource (Figure 14.4 , Figure 14.5, and Figure 14.6).

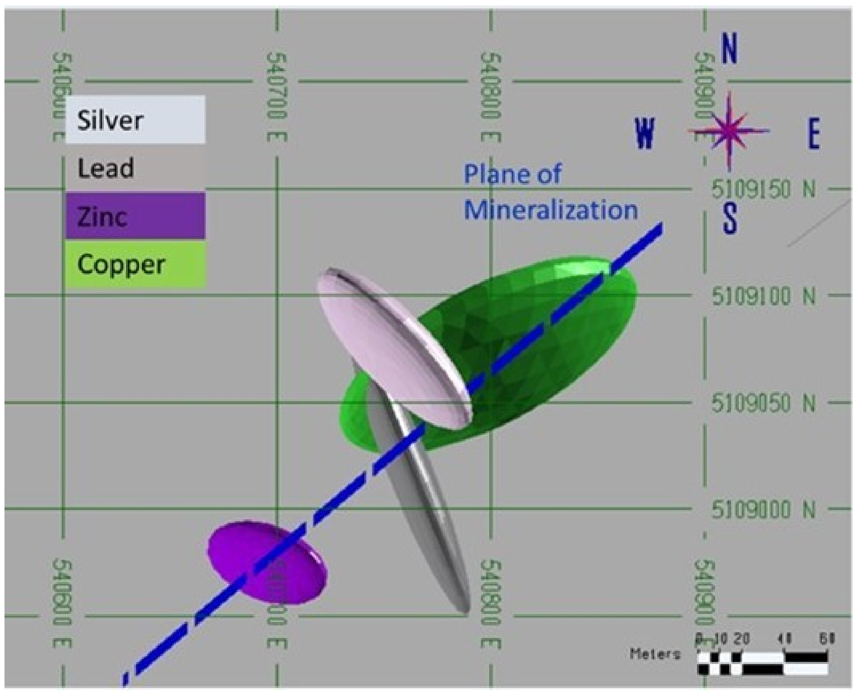


Figure 14.4 3D Variogram of Metals Looking Down

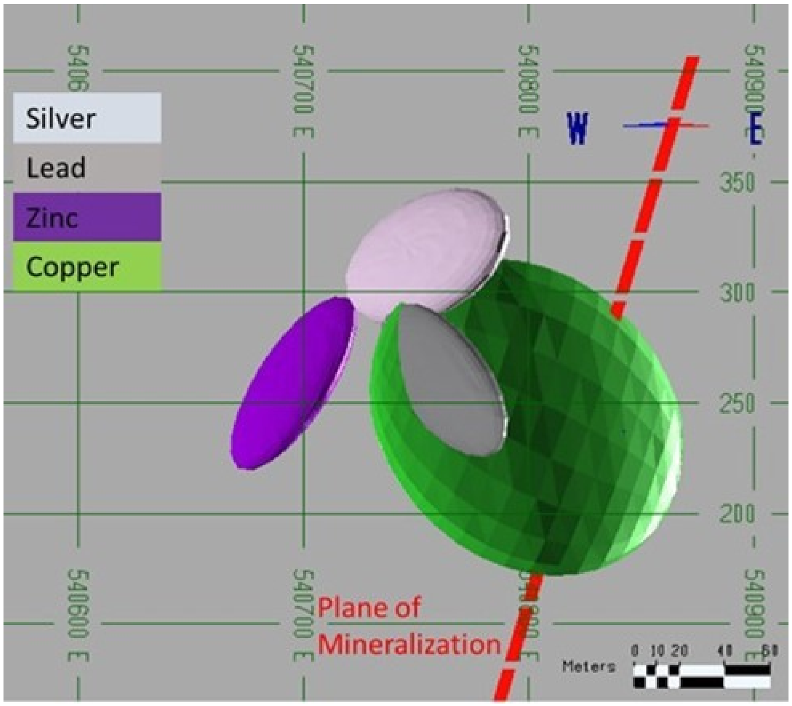


Figure 14.5 3D Variogram of Metals Looking North

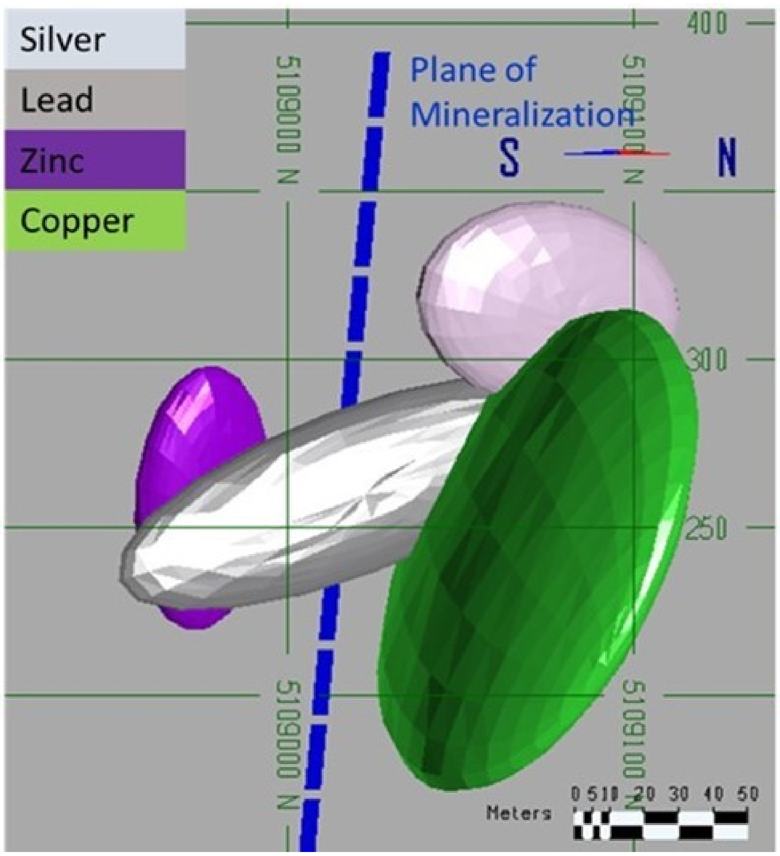


Figure 14.6 3D Variogram of Metals Looking West

14.8.1 Discussion of Variograms

Several discrete populations are evident. Pb and Ag seem to have a strong correlation. Cu has the greatest continuity and Zn shows a somewhat strange orientation. The apparent weakest correlation of zinc may be due to it having the greatest extremes in grade.

It should be noted that variograms are used to predict that the grade will continue over a certain length. The continuity of the lens is predicted by geological modeling. Hence, Zn, which has the greatest fluctuations in grade, shows the smallest continuity over distance.

However, it was noted that while composites of 1m were used in the model, sample intervals were often as much as 3m, meaning that when composites were created, there were now 3 identical composites down the length of the diamond drill core. This would/could obviously create variograms that, in reality, were only mimicking diamond drill holes and had less to do with the integrity of the data.

As a result, not much credence was given to the variograms and the authors were, therefore, more comfortable with an IDW^2 model.

It should be noted that the geologists working on the 2018 drill program were very confident in intersecting the lenses at distances of 50m to 70m when targeting an intersection within the model envelope. This was, in itself, a practical assessment of the variography, as opposed to interpretation of the data.

14.9 Bulk Density

The specific gravities used in the model were updated based on a total of 253 measured specific gravities within the mineralised lenses. These densities were loaded into the model based on sample ID. Averages were then calculated for each of the 4 mineralised zones. Where no information was present, average densities were loaded. The average for the 4 zones and waste was:

- Waste = 2.95
- MS-E1 = 4.06
- MS-E2 = 4.14
- MS-W1 = 3.89
- MS-W2 = 4.04

In the case of MS-E1 and MS-E2, the average densities per lens type was still applied in the assay file, in spite of the geometry being combined.

14.10 Block Model

A 3D block model was built with the following parameters.

```
(452)   3 = TYPE OF COORDINATES= Metric

(455)   1 = TYPE OF MINE MODEL = "3-D"

(476)     = TYPE OF  PROJECT   = METL

(21) XMIN = EASTING AT WEST MATRIX LIMIT    540510.00
(22) XMAX = EASTING AT EAST MATRIX LIMIT    541575.00
(23)  DX  = MATRIX BLOCK WIDTH ON EASTING        5.00
(24)  NX  = NO. OF MINE MODEL COLUMNS E-W      213.00

(25) YMIN = NORTHING AT SOUTH MATRIX LIMIT 5108860.00
(26) YMAX = NORTHING AT NORTH MATRIX LIMIT 5109390.00
(27)  DY  = MATRIX BLOCK WIDTH ON NORTHING       5.00
(28)  NY  = NO. OF MINE MODEL  ROWS  N-S       106.00

(29) ZMIN = TOE ELEVATION OF  LOWEST LEVEL    -525.00
(30) ZMAX = CREST  ELEV.  OF HIGHEST LEVEL     380.00
(31)  DZ  = MATRIX ELEVATION BLOCK HEIGHT        5.00
(32)  NZ  = NO. OF MINE MODEL BENCHES          181.00
```

14.11 Resources Classification

The resources were based on the distance from a diamond drill hole and by a ZnEq cut-off. The mineralised wireframes were created based on mineralogy. The Indicated Resource was based on a cut-off of 9% ZnEq and a maximum distance of 25m to a drill hole. The 25m was based on maximum continuity indicated by the variograms. Because of the relatively small distance used to calculate Indicated, a simple on- hole minimum was used. Using a two-hole minimum was not recommended. Visual inspection of the wireframes suggests a much stronger correlation between assays than the variograms suggest.

For the Inferred Resource, a minimum of 25 metres and a maximum of 200m were used (Figure 14.7, Figure 14.8, and Figure 14.9).

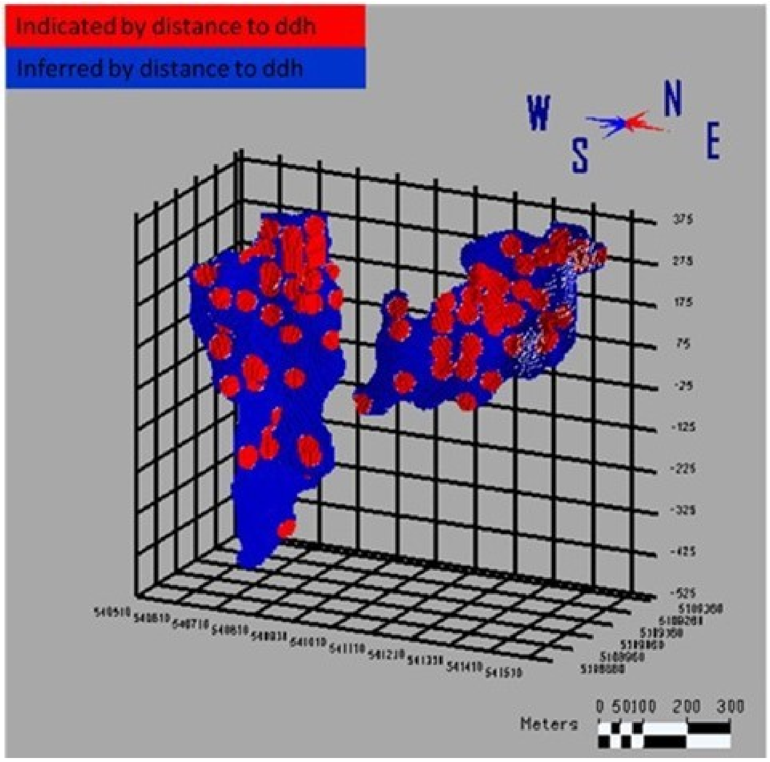


Figure 14.7 Mineralisation, as Defined from Distance to Diamond Drill Hole

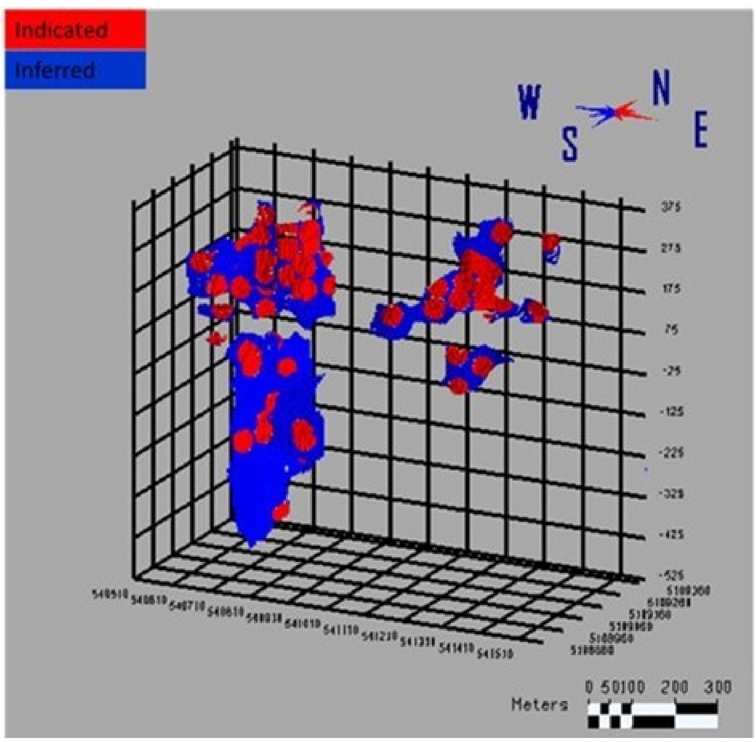


Figure 14.8 Area in Blue Shows Inferred Resource, Red Shows Indicated Resource; Both Categories Trimmed to 9% ZnEq

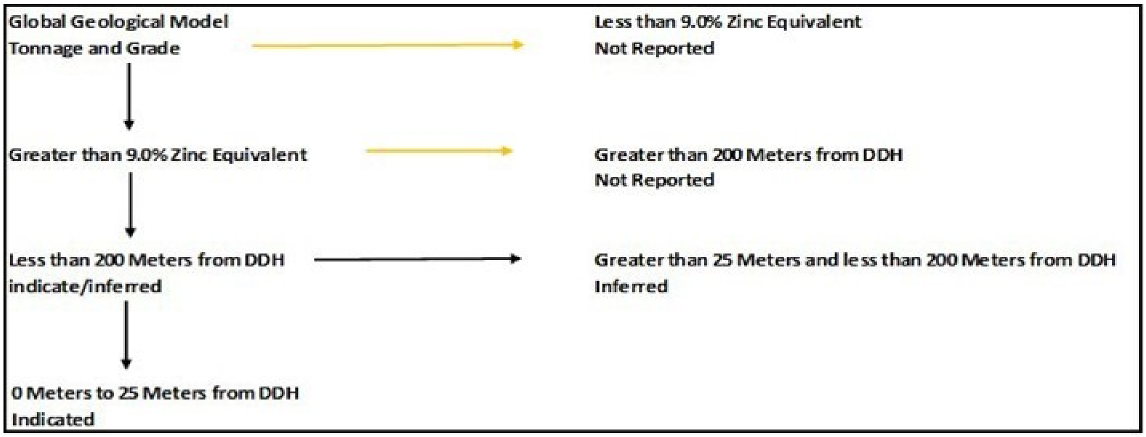


Figure 14.9 Flow Chart of Resource Classification

The Indicated material was calculated as being a maximum of 25m from a diamond drill hole and meeting a single hole minimum. In addition, it had to meet a minimum grade of 9% ZnEq.

A no hole minimum and a minimum of 25m and a maximum distance of 200m were used for Inferred. In addition, it had to meet a minimum grade of 9% ZnEq.

14.12 Resources Estimate

A number of potential cut-off grades in ZnEq were calculated. Results are given in Table 14.1, Table 14.2, Table 14.3 and Table 14.4. The tonnage and grade are robust over the intervals chosen. However, in January 2019, a 9% ZnEq was chosen as the cut-off grade for the resources in order to be able to compare the updated resource to the historical estimates performed by Getty and Chevron in the 1980s (Figure 14.10 and Figure 14.11). For the purposes of this PEA, the cut-off grade was reduced to 7% ZnEq. This, in the opinion of the authors, was deemed acceptable and did not require an entire recalculation of the resource for the following reasons:

1. An indicated and inferred resource had already been calculated at 7% Zinc Equivalent although a 9% cut-off was ultimately used. It was felt that by lowering the cut-off, the only impact on grade shells would be to make them more robust, not less

2. Different metal prices were used in the PEA as well as different recoveries. None the less these metal prices were still robust and similar enough that the resource did not need to be recalculated

3. The resource reported and used in the PEA at a 7% cut-off was slightly lower than that reported in the January 2019 report. This resulted in a slightly more conservative tonnage with the contained metal seeing an overall reduction of approximately 5% contained metal. This is due to removing some of the outliers from the blocks used for the PEA.

TABLE 14.1
MINERAL RESOURCE SUMMARY JANUARY 2019 WITH 9% ZNEQ CUT-OFF

Category	Tonnes	% Zn	% Pb	% Cu	g/t Ag	g/t Au	Density	% ZnEq
Indicated	2,050,000	9.88	3.93	1.38	101.58	0.92	3.99	19.32
Inferred	2,030,000	10.98	4.35	1.2	111.45	0.92	4	20.61

Table 14.2
Mineral Resource Statement – Updated September 2020 with 7% ZnEq Cut-off

Category	Tonnes	% Zn	% Pb	% Cu	g/t Ag	g/t Au	Density	% ZnEq
Indicated	2,177,000	9.25	3.68	1.32	96.4	0.9	3.98	18.23
Inferred	2,294,000	9.79	3.88	1.15	101.1	0.9	3.99	18.62

Table 14.3
Sensitivity of Indicated Resource to Cut-off Grades – January 7, 2019
(Based on < 25m from Diamond Drill Holes)

% ZnEq Cut-off Grade	Tonnes	% Zn	% Pb	% Cu	g/t Ag	g/t Au	Density	% ZnEq
3% ZnEq	3,970,000	6.03	2.38	1.02	65.39	0.68	4.02	12.39
5% ZnEq	2,820,000	7.89	3.12	1.21	83.61	0.81	4	15.79
7% ZnEq	2,320,000	9.11	3.62	1.32	95.04	0.88	3.98	17.99
9% ZnEq	2,050,000	9.88	3.93	1.38	101.58	0.92	3.99	19.32
11% ZnEq	1,770,000	10.77	4.29	1.41	109.32	0.96	4	20.79

Table 14.4
Sensitivity of Inferred Resource to Cut-off Grades – January 7, 2019
(Based on > 25m and < 200m from Diamond Drill Holes)

% ZnEq Cut-off Grade	Tonnes	% Zn	% Pb	% Cu	g/t Ag	g/t Au	Density	% ZnEq
3% ZnEq	4,020,000	6.59	2.58	0.94	69.91	0.68	4.03	13.03
5% ZnEq	2,980,000	8.35	3.29	1.06	87.12	0.79	4.01	16.14
7% ZnEq	2,450,000	9.67	3.83	1.15	99.99	0.86	4	18.43
9% ZnEq	2,030,000	10.98	4.35	1.2	111.45	0.92	4	20.61
11% ZnEq	1,740,000	12.06	4.77	1.24	121.42	0.97	4	22.39

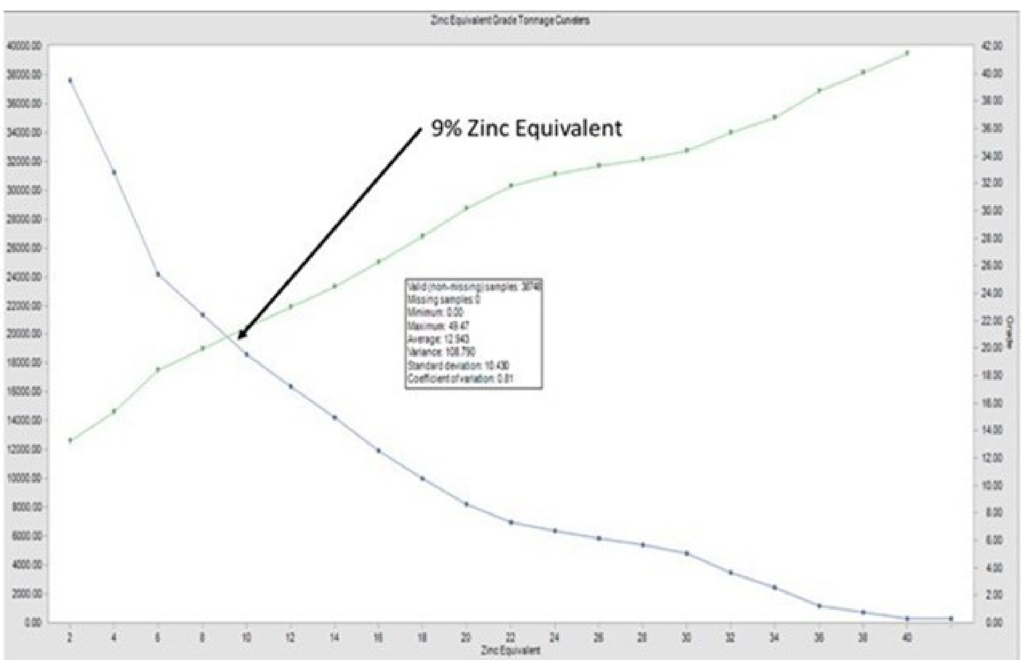


Figure 14.10 ZnEq Tonnage Curve Limited to Mineralised Lenses

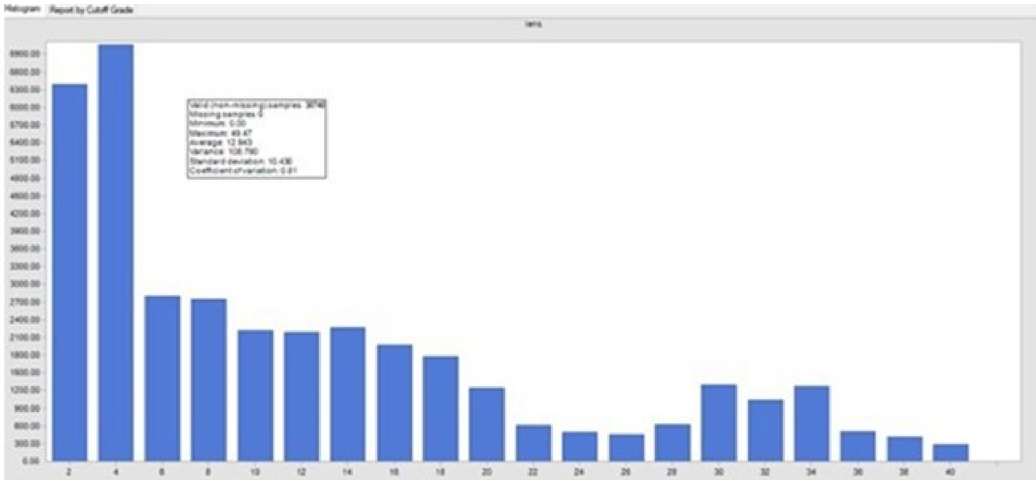


Figure 14.11 ZnEq Histogram Limited to Mineralised Lenses

The histogram above also indicates that a cut-off between 6% to 9% ZnEq would ensure that a bell curve would capture the bulk of the tonnes. It does not, however, show a sharp inflection point, which would ease the determination of a cut-off (Table 14.5).

TABLE 14.5
ZNEQ CUT-OFFS LIMITED TO MINERALISED LENSES

Zinc Equivalent Cutoffs						
Class	Cutoff >=	Cutoff <	Samples	Average	Metal (units)	%Total
1	0.0	5.12	382	2.634	1006.2	9.1179
2	5.12	9.65	148	6.7652	1001.2	9.0732
3	9.65	13.7	86	11.712	1007.2	9.1273
4	13.7	16.98	66	15.209	1003.8	9.0963
5	16.98	20.6	54	18.82	1016.3	9.2094
6	20.6	24.65	44	22.732	1000.2	9.0638
7	24.65	28.5	38	26.574	1009.8	9.1508
8	28.5	33.04	33	30.773	1015.5	9.2025
9	33.04	36.71	29	34.451	999.09	9.0536
10	36.71	42.5	27	38.663	1043.9	9.4596
11	42.5	51.91	20	46.6	932.0	8.4457
Total:			927	11.904	11035.1	100.0

Table 14.5 also shows that while there are a considerable number of assays that fall below the 9% ZnEq, the total contained metal units do not exhibit much variation between cut-offs. Nonetheless, based on visual examination of the model, it was the opinion of the authors that a 9% ZnEq cut-off was appropriate until such time that detailed mining options become available and additional infill diamond drilling and associated geological interpretations are carried out. As noted above, more detailed mining options became available and using data available for the PEA, the ZnEq cut-off was revised to 7%.

It is apparent that there appears to be a natural break in ZnEq grades at approximately 9% (Figure 14.12 and Figure 14.13).

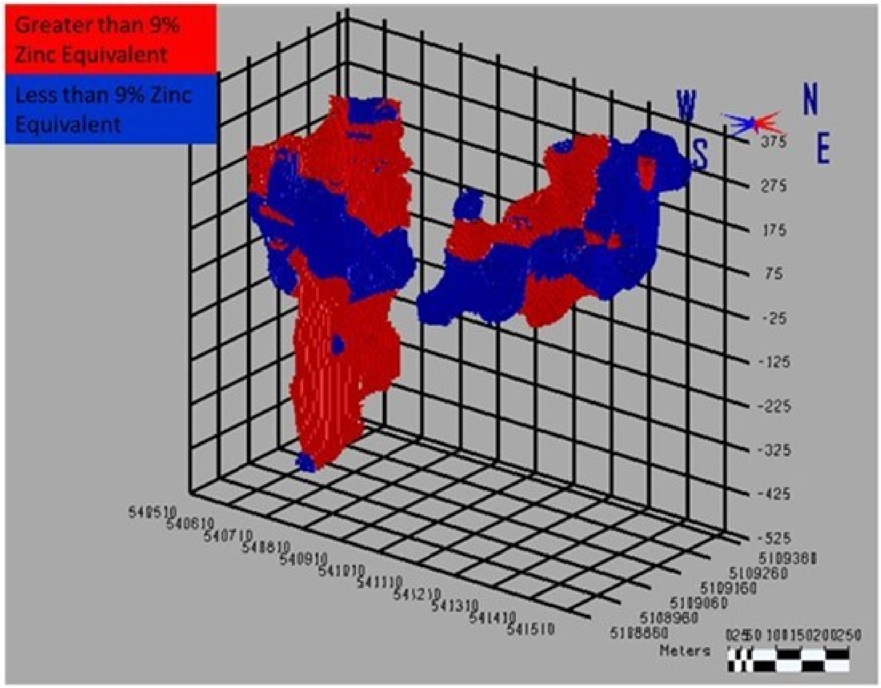


Figure 14.12 Indicated (Red) and Inferred (Blue) Domains Superimposed on the Entire Geological Model

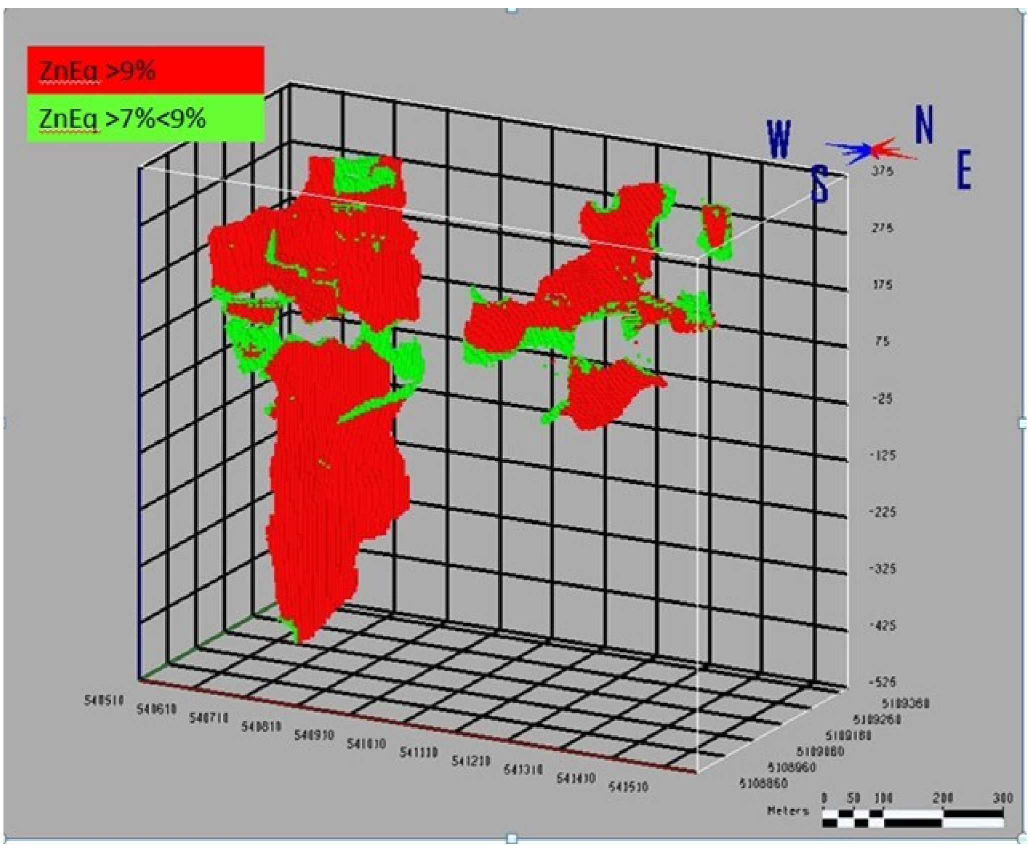


Figure 14.13 Showing Location of Material > 7% ZnEq < 9% ZnEq

14.13 Calculation of Cut-off

Both a Gross ZnEq cut-off and an NSR were calculated. A generic NSR was calculated using the parameters in Table 14.6.

TABLE 14.6
MINERAL RESOURCE METAL PRICES

Assumptions	$US/pound/oz	
Zn	$ 1.20	/pound
Cu	$ 2.50	/pound
Pb	$ 1.00	/pound
Ag	$ 16.00	/troy ounce
Au	$ 1,200.00	/troy ounce

A 9% ZnEq was then calculated based on values in Table 14.6 and lacking any recent metallurgical testing, an assumed conservative similar recovery of all metals (75%) was used. Historical metallurgical testing indicated 88% for Zn, 78% for Pb, and 77% for Cu.

Assuming an overall recovery (milling and smelting) of 75%, a 9% ZnEq equates to a $178 NSR cut-off. This is not meant as a precise number but more as an aid in determining cut-off (Table 14.7).

TABLE 14.7
CALCULATION OF NSR/ZNEQ CUT-OFF

US$/Pound	$ 1.20
9 % zinc = pounds/tonne	198
75 % recovery = pounds zinc	149
value in US$	$ 178

The 9% ZnEq was based on both economics as well as a tonnage grade curve and indicated that a significant portion of the tonnes and grade would be captured at this cut-off (Table 14.8, Table 14.9, Table 14.10, and Table 14.11).

This same calculation using a 7% ZnEq resulted in a $139 NSR.

TABLE 14.8
INDICATED RESOURCES

Indicated Mineral Resources									
	ZnEq Cutoff	Tonnes	%Zn	%Cu	%Pb	g/t Au	g/t Ag	Den	%ZnEq
LENS E1 + E2	9%	890,000	8.27	1.10	3.24	0.87	75.64	4.07	16.00
LENS WEST 1	9%	990,000	11.60	1.60	4.60	0.99	128.39	3.92	22.68
LENS WEST 2	9%	170,000	8.30	1.51	3.59	0.75	81.41	4.03	17.11
9% Zinc Equiva	9%	2,050,000	9.88	1.38	3.93	0.92	101.58	3.99	19.32

Tonnes may not add due to rounding
The resource for the Pickett Mountain Project zinc deposit was estimated based on metal prices of US$1.20/lb Zn, $2.50/lb Cu, $1.00/lb Pb, $16.00/oz Ag, and $1,200/oz/Au, and equates to an NSR cut-off of $178/tonne or a 9% ZnEq cut-off based on the above metal prices. An average recovery of 75% for all metals for underground mining and milling was utilised to report the resource

TABLE 14.9
INFERRED RESOURCES

Inferred Mineral Resources									
	ZnEq Cutoff	Tonnes	%Zn	%Cu	%Pb	g/t Au	g/t Ag	Den	%ZnEq
LENS E1 + E2	9%	670,000	6.87	1.00	2.69	0.82	67.44	4.07	13.68
LENS WEST 1	9%	1,120,000	14.23	1.28	5.58	1.04	146.32	3.95	25.87
LENS WEST 2	9%	240,000	7.26	1.42	3.20	0.68	71.05	4.01	15.27
9% Zinc Equiva>=	9%	2,030,000	10.98	1.20	4.35	0.92	111.45	4.00	20.61

Tonnes may not add due to rounding
The resource for the Pickett Mountain Project zinc deposit was estimated based on metal prices of US $1.20/lb Zn, $2.50/lb Cu, $1.00/lb Pb, $16.00/oz Ag, and $1,200/oz/Au, this equates to an NSR cut-off of $178/tonne or a 9% ZnEq cut-off based on the above metal prices. An average recovery of 75% for all metals for underground mining and milling was utilised to report the resource.

TABLE 14.10
SHOWING REPRESENTATIVE RECOVERED VALUE OF INDICATED RESOURCE

Indicated Mineral Resource						
	Price Assumption				$US/tonne	Approx.NSR
	$US/Pound	$US/gm	%US/%	input file	metal	% payable
Zn	1.2		26.45	9.88	196	51%
Cu	2.5		55.12	1.38	57	15%
Pb	1		22.05	3.93	65	17%
Ag	16	0.51		101.59	39	10%
Au	1200	38.58		0.92	27	7%
					383	100%

TABLE 14.11
SHOWING REPRESENTATIVE RECOVERED VALUE OF INFERRED RESOURCE

Inferred Mineral Resource						
	Price Assumption				$US/tonne	Approx.NSR
	$US/Pound	$US/gm	%US/%	input file	metal	% payable
Zn	1.2		26.45	10.98	218	53%
Cu	2.5		55.12	1.20	50	12%
Pb	1		22.05	4.35	72	18%
Ag	16	0.51		111.45	43	11%
Au	1200	38.58		0.92	27	7%
					409	100%

NSR values given are only an estimate using an overall 75% recovery of all metals. This is less than the historical metallurgical work undertaken in 1984 and is considered to be conservative. While a recovery of 75% for gold may be optimistic, on average it only contributes 7% to the value of the Mineral Resource, so any discrepancies would be minor (Table 14.12).

TABLE 14.12
METALLURGICAL RECOVERIES (1984 BY GETTY AT LAKEFIELD)

The flotation test resulted in the following recoveries (Bosch and Grimes, 1984):

		Cu Con.	Pb Con.	Zn Con
Copper	-	**77.4%**	1.6%	11.2%
Lead	-	3.8%	**77.5%**	6.5%
Zinc	-	1.2%	4.8%	**87.7%**
Gold	-	13.3%	20.4%	12.5%
Silver	-	27.3%	39.6%	11.1%

14.14 Confirmation of Estimate

The accuracy of the model reporting was verified in four ways:

1. Pitres (Hexagon™/MineSight™) was used to calculate the resource tonnage and grade.

2. UG1res (Hexagon™/MineSight™) was then used to compare the resource estimate to the Pitres resource tonnage and grade. As expected, the numbers were identical.

3. The 3D block model data was exported into Excel™ and the resource tonnage and grade results independently verified the data, as reported by Hexagon™/MineSight™, was correct. The global tonnage was within 3 tonnes.

4. Andre Labonte calculated a gross tonnage and grade in Gems™ and this was compared to MineSight™. Using a 0.5% Zn cut-off, his global tonnage was within 1% of MineSight™ and the variance for global contained metals ranged from 3% for base metals to a maximum of 5% for precious metals.

14.15 Discussion of Results

A historical resource estimate was undertaken using the "Contour Plotting System" for Getty in 1983. This historical resource does not use the classification terms "Inferred Mineral Resource," "Indicated Mineral Resource," and "Measured Mineral Resource" that have the meanings ascribed to them by the Canadian Institute of Mining, Metallurgy and Petroleum, as the CIM Definition Standards on Mineral Resources and Mineral Reserves adopted by CIM Council, as amended. The authors have not done sufficient work to classify this historical estimate as a current Mineral Resource and Wolfden is not treating this historical estimate as a current Mineral Resource and they are included in this section for illustrative purposes only and should not be disclosed out of context. Using an average density factor of 8.25 cubic feet per ton, the estimated resource was 3.15 million tons with an average grade of 9.66% Zn, 4.30% Pb, 1.24% Cu, 0.029 opt Au, and 2.96 opt Ag (Laverty, 1983; Riddell, 1983). The conversion from imperial to metric is given below in Table 14.13.

TABLE 14.13

HISTORICAL 1983 RESOURCE

Historical Resource Estimate							
	Tons	%Zn	%Cu	%Pb	oz/t Au	oz/t Ag	Tonnage Factor
Getty 1983 Imperial	3,150,000	9.66	1.24	4.30	0.03	2.96	8.25
	Tonnes	%Zn	%Cu	%Pb	g/t ?Au	g/t Ag	Density
Getty 1983 Metric	2,857,050	9.66	1.24	4.30	0.99	102.16	3.89

This historical resource does not use the classification terms "Inferred Mineral Resource," "Indicated Mineral Resource," and "Measured Mineral Resource" that have the meanings ascribed to them by the Canadian Institute of Mining, Metallurgy and Petroleum, as the CIM Definition Standards on Mineral Resources and Mineral Reserves adopted by CIM Council, as amended (Table 14.14).

TABLE 14.14

MINERAL RESOURCE SUMMARY – JANUARY 7, 2019

Total Resources	Tonnes	% Zn	% Cu	% Pb	g/t Au	g/t Ag	Density	% ZnEq
Indicated Mineral Resource	2,050,000	9.88	1.38	3.93	0.92	101.58	3.99	19.32
Inferred Mineral Resource	2,030,000	10.98	1.20	4.35	0.92	111.45	4.00	20.61

While a direct comparison of tonnage and grade is not possible, it is apparent that regardless of the methodology employed to calculate the resource, the system is robust enough to support multiple methods of calculating resources. The apparent increase in tonnage is at least partially due to the 2018 diamond drill program that extended the deposits to depth. The historical resource was noted to occur only to a depth of 400 metres. The updated estimate goes to a vertical depth of +800m in the West Lens (Table 14.15 and Table 14.16).

TABLE 14.15

LIST OF SIGNIFICANT INTERSECTIONS USED TO CALCULATE THE MINERAL RESOURCE

HOLE-ID	EAST	NORTH	ELEV.	LNGTH	ZN	CU	PB	AU	AG	DEN
1	541,355	5,109,333	322	6.71	2.04	0.71	0.72	0.19	28.80	4.14
2	541,362	5,109,314	295	12.80	4.96	0.90	1.85	0.62	49.02	4.14
8	541,246	5,109,299	299	3.65	5.46	1.22	1.97	1.10	45.49	4.06
9	541,326	5,109,306	286	7.92	2.00	0.29	0.27	0.73	19.39	4.14
10	541,373	5,109,289	218	1.22	2.18	0.81	0.59	0.17	42.51	4.14
11	541,476	5,109,300	305	4.41	2.14	0.42	0.56	0.41	30.38	4.14
13	541,327	5,109,280	189	2.90	2.49	0.68	0.52	0.54	54.86	4.14
23	540,719	5,109,002	218	1.30	8.07	1.58	5.00	0.86	101.95	4.04
23	540,716	5,109,006	212	3.40	6.31	2.12	3.22	0.71	65.29	3.89
25	540,745	5,108,964	127	0.61	0.60	0.50	0.23	0.33	14.09	4.04
25	540,742	5,108,970	122	6.92	1.35	0.72	0.33	0.23	10.75	3.89
28	540,813	5,109,059	202	10.07	15.89	1.42	7.40	1.84	180.93	3.89
29	540,623	5,108,925	135	0.61	2.58	0.19	0.96	0.25	6.32	3.89
30	540,809	5,109,035	81	1.65	0.89	0.35	0.40	0.27	21.60	3.89
33	540,892	5,109,111	241	1.56	6.66	0.44	4.77	0.50	90.77	3.89
34	541,138	5,109,166	140	14.06	8.67	0.82	3.27	0.99	78.52	4.06
35	540,920	5,109,074	187	4.45	12.65	0.85	5.58	0.82	86.61	4.04
36	541,054	5,109,112	125	6.33	6.11	1.07	2.46	0.72	63.58	4.06
37	541,140	5,109,154	78	21.96	1.64	0.72	0.69	0.74	51.27	4.06
38	540,911	5,109,057	90	3.98	2.26	0.73	0.88	0.44	32.34	4.04
39	541,235	5,109,229	126	32.18	2.31	0.85	0.89	0.41	22.43	4.06
40	541,082	5,109,080	11	3.24	1.52	0.76	0.58	-	-	4.06
44	541,282	5,109,238	79	25.75	0.94	0.53	0.51	0.22	8.49	4.06
46	541,236	5,109,259	212	10.06	8.85	0.73	3.38	0.66	79.71	4.06
47	541,184	5,109,221	190	6.41	17.08	1.02	6.42	1.40	128.89	4.06
49	541,178	5,109,263	305	0.36	0.18	0.10	0.14	0.11	0.14	4.06
52	540,805	5,109,058	327	13.97	3.97	0.91	1.69	0.49	33.27	3.89
53	540,841	5,109,078	231	13.90	18.60	1.63	10.21	1.62	229.78	3.89
54	540,748	5,109,040	294	8.37	7.76	1.23	2.27	0.71	44.23	3.89
55	540,714	5,108,931	- 9	5.17	0.99	0.56	0.33	0.18	8.97	3.89
56	541,222	5,109,117	- 30	1.95	12.00	0.67	4.08	1.30	96.41	4.06
57	540,848	5,109,091	298	14.58	11.05	1.54	5.91	0.92	145.72	3.89
58	540,868	5,109,041	181	5.82	8.09	2.40	3.31	1.12	123.63	4.04
58	540,855	5,109,075	122	14.54	2.97	0.94	1.30	0.44	78.28	3.89
59	540,766	5,109,034	212	14.47	6.16	1.58	2.75	0.62	88.43	4.04
59	540,764	5,109,039	184	17.71	14.99	3.14	7.80	1.19	150.73	3.89
60	541,285	5,109,293	262	1.82	0.79	0.75	0.28	0.17	5.14	4.06
62	540,682	5,108,947	195	5.80	1.47	0.94	0.50	0.72	20.52	4.04
62	540,680	5,108,952	183	7.32	8.59	1.09	4.55	0.87	76.67	3.89
63	541,237	5,109,282	265	0.19	5.30	0.54	2.40	0.45	30.17	4.06
64	541,193	5,109,240	238	12.76	8.24	1.36	3.71	0.78	83.84	4.06
65	540,996	5,109,027	- 40	3.29	0.77	0.32	0.26	0.53	4.86	4.06
67	541,230	5,109,234	146	81.46	5.68	1.00	2.55	0.54	42.09	4.06
68	540,807	5,109,058	308	20.87	7.79	1.13	2.59	0.52	44.74	3.89
69	540,808	5,109,056	276	30.32	8.40	1.16	3.55	0.95	107.33	3.89
70	541,368	5,109,231	92	41.75	1.49	0.58	0.49	0.41	17.27	4.14
72	540,793	5,109,031	- 101	3.92	18.08	0.56	8.53	1.13	209.86	3.89
74	540,680	5,108,954	248	3.34	13.55	1.95	5.12	0.98	167.98	3.89
76	540,643	5,108,945	243	5.41	1.66	2.88	0.57	0.36	38.22	3.89

Table 14.15
List of Significant Intersections Used to Calculate the Mineral Resource
(continued)

HOLE-ID	EAST	NORTH	ELEV.	LNGTH	ZN	CU	PB	AU	AG	DEN
78	541,356	5,109,270	136	23.05	5.01	1.10	2.00	0.62	48.85	4.14
80	540,845	5,109,024	186	6.24	8.15	1.64	3.90	1.00	110.32	4.04
80	540,826	5,109,060	132	9.60	1.61	0.58	1.09	0.44	14.74	3.89
81	541,272	5,109,252	150	25.48	3.10	0.77	1.20	0.51	31.53	4.06
82	540,724	5,108,986	182	3.14	0.94	0.91	0.32	0.46	26.51	4.04
82	540,719	5,108,994	173	4.90	7.23	2.64	2.85	0.95	115.16	3.89
83	541,285	5,109,275	208	5.48	3.13	0.43	1.24	0.33	29.39	4.06
85	540,759	5,108,978	18	1.66	1.42	0.19	0.69	0.20	6.13	4.04
85	540,752	5,108,984	- 2	12.71	8.81	1.88	4.04	1.02	92.78	3.89
86	540,765	5,109,031	247	2.59	11.19	1.28	3.86	1.03	85.12	4.04
86	540,761	5,109,039	237	8.00	3.04	2.24	1.11	0.57	35.79	3.89
87	540,865	5,109,042	236	12.10	5.53	1.28	2.35	0.56	56.91	4.04
87	540,852	5,109,074	187	5.73	15.50	2.25	6.00	0.87	191.16	3.89
90	540,805	5,109,047	- 377	1.91	25.27	0.88	10.21	0.86	140.84	3.89
91	540,832	5,109,023	- 1	2.10	1.00	1.65	0.37	0.38	9.26	4.04
91	540,823	5,109,033	- 15	2.61	8.28	1.46	3.00	1.45	70.80	3.89
92	541,187	5,109,196	158	4.19	8.40	0.87	3.29	0.81	76.59	4.06
93	541,201	5,109,157	41	13.66	3.94	0.85	1.46	1.11	72.91	4.06
94	540,683	5,108,941	79	15.82	2.72	1.09	1.09	0.73	44.61	3.89
90A	540,808	5,109,049	- 312	1.37	12.50	0.77	4.75	0.79	93.24	3.89
98A	540,733	5,108,954	- 446	1.40	1.58	0.10	0.01	0.10	1.30	3.89
PM-17-001	540,853	5,109,090	309	7.95	8.06	1.56	3.92	0.87	105.89	3.56
PM-17-002	540,854	5,109,086	273	9.89	16.31	1.73	7.09	1.42	185.64	3.89
PM-18-003	541,229	5,109,235	180	9.71	9.51	1.06	3.45	0.77	60.82	4.07
PM-18-004	541,218	5,109,257	223	10.28	11.40	1.28	4.20	1.00	120.25	4.06
PM-18-005	541,203	5,109,178	76	45.80	1.30	0.59	0.51	0.43	24.03	4.06
PM-18-006A	541,368	5,109,244	96	52.40	1.33	0.50	0.48	0.30	18.63	4.14
PM-18-007	540,778	5,109,011	149	1.79	0.63	0.34	0.15	0.14	9.00	4.04
PM-18-007	540,767	5,109,028	112	31.49	4.62	1.01	1.73	0.63	63.50	4.15
PM-18-008	540,794	5,109,012	60	1.00	0.94	0.72	0.35	0.17	17.25	4.04
PM-18-008	540,792	5,109,016	49	2.41	16.79	0.37	3.99	0.53	68.43	3.89
PM-18-009	540,747	5,108,975	32	6.80	0.79	0.54	0.25	0.24	16.99	4.35
PM-18-009	540,744	5,108,978	24	3.50	10.38	1.28	4.07	0.59	84.28	4.32
PM-18-010	540,720	5,109,012	279	3.29	5.35	0.51	1.74	0.31	41.77	3.42
PM-18-011	540,756	5,109,047	342	3.00	4.22	2.60	1.42	0.54	34.26	3.89
PM-18-012	540,844	5,109,097	345	11.40	3.84	0.87	1.51	0.31	36.35	3.27
PM-18-013	541,406	5,109,325	305	9.40	2.41	0.76	0.84	0.42	32.21	4.16
PM-18-014	541,430	5,109,295	285	5.00	1.70	0.47	0.55	0.28	24.65	4.14
PM-18-015	541,391	5,109,274	155	16.50	1.10	0.53	0.42	0.27	18.59	4.14
PM-18-018	540,679	5,108,961	293	0.50	11.70	2.25	8.31	0.46	100.00	3.89
PM-18-019	541,038	5,109,137	168	2.90	0.40	0.28	0.08	0.05	3.88	4.06
PM-18-020	541,141	5,109,173	188	3.20	13.32	1.75	5.37	1.14	126.36	3.94
PM-18-021	541,149	5,109,142	32	21.00	2.09	0.35	0.72	0.31	15.47	4.22
PM-18-022	540,770	5,109,011	- 171	1.50	-	0.54	-	0.58	29.00	3.47
PM-18-022	540,768	5,109,015	- 176	4.71	23.85	0.88	9.86	1.53	262.52	4.44
PM-18-022A	540,775	5,109,023	- 143	5.90	23.95	0.95	10.21	1.35	324.08	3.89
PM-18-023	540,855	5,109,057	- 193	5.92	7.41	1.38	3.20	0.74	63.39	3.97
PM-18-023	540,841	5,109,094	- 238	2.40	20.76	1.37	3.83	1.07	108.07	4.13
PM-18-023A	540,846	5,109,058	- 171	6.00	10.20	1.28	4.69	0.49	52.95	4.04
PM-18-023A	540,836	5,109,084	- 197	3.29	15.83	0.70	7.78	0.93	167.93	3.89
PM-18-027	540,691	5,108,958	139	24.41	3.50	0.76	1.47	0.30	52.41	3.89
PM-18-028	541,259	5,109,171	14	3.41	19.15	0.60	7.37	1.16	151.11	4.06
PM-18-029	540,739	5,108,969	- 186	10.55	19.32	1.24	7.24	1.28	206.37	3.89

TABLE 14.16
LIST OF DIAMOND DRILL COLLARS FROM 2018 DIAMOND DRILLING

HOLE-ID	EAST	NORTH	ELEV	AZ	DIP	DEPTH
PM-17-001	540,887	5,109,039	372	327	- 46	168
PM-17-002	540,887	5,109,039	372	327	- 61	180
PM-18-003	541,280	5,109,160	356	327	- 64	234
PM-18-004	541,280	5,109,160	356	327	- 50	204
PM-18-005	541,264	5,109,082	355	327	- 68	354
PM-18-006	541,424	5,109,130	348	333	- 63	33
PM-18-006A	541,424	5,109,130	348	333	- 63	336
PM-18-007	540,845	5,108,913	372	327	- 62	465
PM-18-008	540,845	5,108,913	372	327	- 69	496
PM-18-009	540,869	5,108,866	367	313	- 61	447
PM-18-010	540,760	5,108,953	382	327	- 55	198
PM-18-011	540,778	5,109,012	383	327	- 45	107
PM-18-012	540,861	5,109,071	375	327	- 45	99
PM-18-013	541,431	5,109,287	349	327	- 45	120
PM-18-014	541,464	5,109,241	348	327	- 45	171
PM-18-015	541,465	5,109,157	348	327	- 55	312
PM-18-016	541,501	5,109,176	349	327	- 55	312
PM-18-017	540,906	5,109,092	372	327	- 69	120
PM-18-018	540,714	5,108,908	383	327	- 55	192
PM-18-019	541,114	5,109,021	360	327	- 55	298
PM-18-020	541,193	5,109,097	361	327	- 63	237
PM-18-021	541,233	5,108,999	352	327	- 63	420
PM-18-022	540,939	5,108,666	362	329	- 58	759
PM-18-022A	540,939	5,108,666	362	329	- 58	699
PM-18-023	541,020	5,108,714	350	327	- 58	801
PM-18-023A	541,020	5,108,714	350	327	- 58	750
PM-18-025	541,304	5,109,390	365	327	- 50	258
PM-18-027	540,759	5,108,870	382	322	- 65	336
PM-18-028	541,380	5,109,002	346	326	- 60	498
PM-18-029	540,939	5,108,668	370	326	- 62	771
PM-18-006	541,424	5,109,130	348	333	- 62	33
PM-18-026	540,889	5,108,694	370	326	- 57	585
PM-18-029A	540,939	5,108,668	370	326	- 62	714
PM-18-030	541,380	5,109,002	346	326	- 74	651
PM-18-031	541,348	5,108,907	340	326	- 70	537

It is the opinion of the authors that continued expansion and infill diamond drilling will have significant potential to expand and certainly upgrade the resource.

Figure 14.14 and Figure 14.15 show the ZnEq grade shells for the lenses of the deposits. Additional drilling and modeling may allow material that is less than 9% ZnEq to be upgraded. In addition, metallurgical work and developing mining costs for the deposit may allow the inclusion of lower grade material in the resource. This was undertaken as part of the PEA.

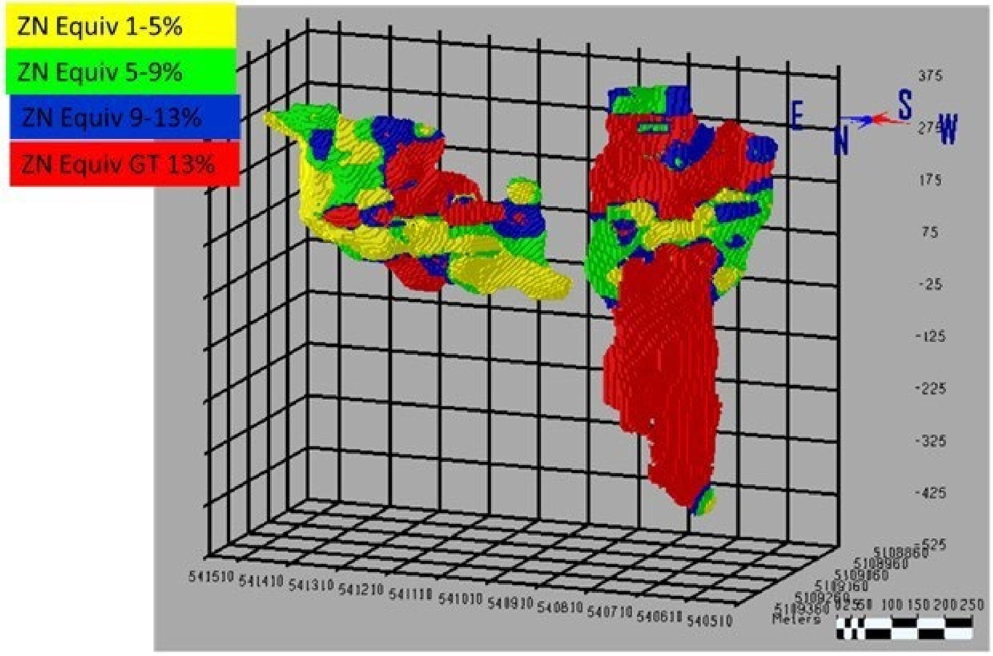


Figure 14.14 Grade Shells of Lenses – Footwall View Looking Southeast

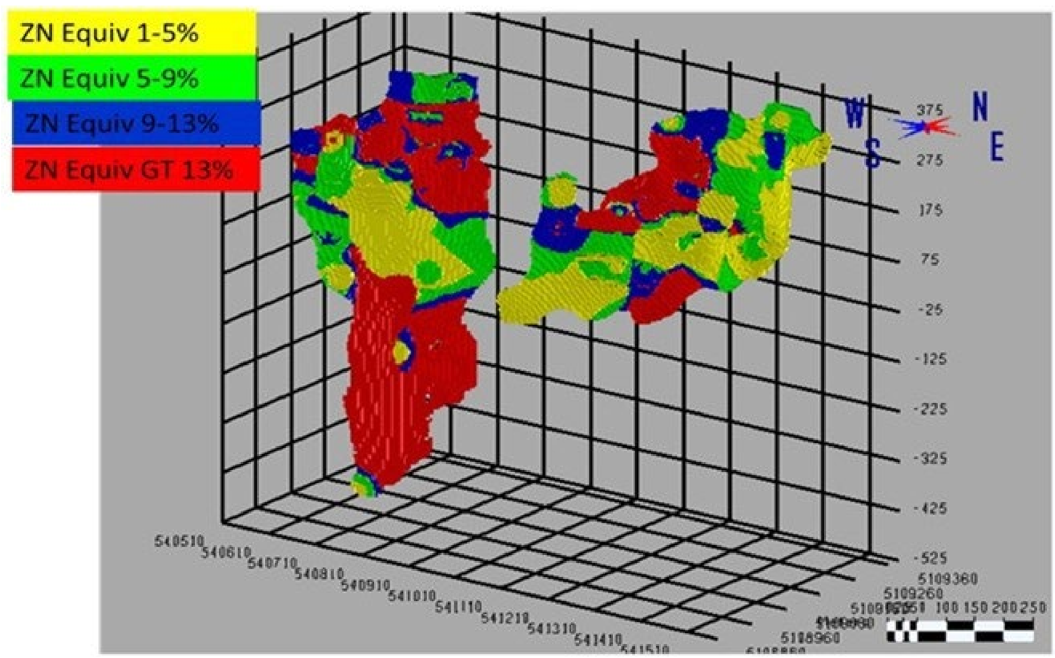


Figure 14.15 Grade Shells of Lenses – Hanging Wall View Looking Northwest

A series of grade shells were created for the West and East mineralised zones (lenses). This was undertaken to confirm the continuity of the mineralised zones as well as for future diamond drill and exploration targets.

To the knowledge of the authors, there are no known environmental, legal, title, taxation, socio-economic, marketing, or political factors that could materially affect the Mineral Resource estimate.

In conclusion, it is the opinion of the authors that the Pickett Mountain deposit, as currently defined to a depth of 875m, has significant infill and expansion opportunities. The local exploration target expansion range is 6 to 10 million tonnes grading 12% to 20% ZnEq, based on the current geological model, without the addition of other lenses. This target size is derived from the interpretation of the drilling, geological structure, geology, and surface sampling carried out on the Property to date. The potential quantity and grade of the target is conceptual in nature. There has been insufficient exploration of this target to define a Mineral Resource and it is uncertain if further local exploration will result in this target being delineated as a Mineral Resource.

14.15.1 Mineral Resources Used in the Mine Plan

The mineral resource used in the PEA includes Indicated and Inferred Resources and is an update from the January 7, 2019 Mineral Resource statement. The estimate uses a 7% cut-off grade (or an NSR value of $139/t) rather than the previous 9% cut-off grade ($178/t NSR). The same methodology used in the 2019 estimate was applied to the updated estimate where the metal prices were not updated (to those used in the PEA financial model) and no additional information was either included or excluded even though infill drill results since 2019 is expected to upgrade the mineral resource and could potentially lead to an increase.

The resource estimates used in the mine plan are only those contained within the main zones of the mineralised zones and have had fringe outliers removed from the estimates. They are presented in Table 14.17.

TABLE 14.17
UPDATED MINERAL RESOURCES USED IN MINE PLAN

Category	Tonnes	Zn %	Pb %	Cu %	Ag g/t	Au g/t	Density	ZnEq %
Indicated	2,177,000	9.25	3.68	1.32	96.4	0.9	3.98	18.23
Inferred	2,294,000	9.79	3.88	1.15	101.1	0.9	3.99	18.62

The mineral resources were estimated using the metal prices of US$1.20/lb Zn, $2.50/lb Cu, $1.00/lb Pb, $16.00/oz Ag, and $1,200/oz/Au, using a 7% cutoff grade that equates to an NSR cut-off of $139/tonne at the same metal prices. An average recovery of 75% for all metals was assumed. A 10% mining dilution at zero grade was only added to the financial model which also used different metal prices.

15.0 Mineral Reserve Estimates

There has not yet been any Mineral Reserve estimation done.

16.0 Mining Methods

The Pickett Mountain Project is a polymetallic deposit containing zinc, lead, copper, and some precious metals, namely silver and gold, and is separated into two main zones; the West Zone and the East Zone. The focus of mining will be zinc. The West Zone is narrow but high-grade while the East Zone is broader and has a larger volume but a large portion of it is currently sub-economic.

The zones are near surface making the mine accessible by decline and mobile equipment and allowing the project a shorter time to get into steady-state production.

Initially, mining will commence in the upper part of the West Zone (Zone 1), then the higher-grade Lower West Zone (Zone 3), and main East Zone (Zone 2). A borehole hoist will be developed to increase productivity and minimise the mobile fleet. The overall mine plan is to have three main zones operating to ensure a steady feed to the processing plant (Figure 16.1).

Figure 16.1 Pickett Mountain Mine Design Longitudinal Section Looking North

16.1 General

The roadway for the portal entrance is to be at a grade that water will drain from the entrance.

The excavation requires 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.

Final excavation of the overburden shall be secured from falling into the working area while the portal entrance structure is erected. Length of the entrance canopy will be determined by the depth of overburden and the topography of the selected location.

Roadbed material should be at least 300 mm in depth. A ditch should be established on the same side as the pipelines.

16.1.1 Portal Excavation and Decline Drive

The mine will be accessed from the surface by a decline collared in the hillside. An excavation will be cut into the hillside to create a large free face in which to collar the portal. The floor of the cut and the first two rounds of the decline will be designed to be upgrade at a sufficient grade to allow water to drain away from the portal. The decline will be driven at a nominal 15% grade and access the upper sections of the ore zone at 100m intervals. A 50m crown pillar will be left in place. The nominal surface elevation is approximately 400m asl (above sea level). In order to better define the underground levels, 5000 has been added to the surface elevation, making surface nominally 5400 in the mine nomenclature.

The 4.0m × 4.5m decline will proceed down to the 5150 level (250m below the surface) where the bottom of the first mining block will be established. The 5150 level will also serve as the main haulage access for the lens of the East Zone. 2×2×2m safety bays will be driven every 100m where required. Remuck bays will be established at 250m distances along the ramp and electrical substations will be established where required. A main ventilation raise will be driven internal to the ramp at 100m intervals to allow for exhaust ventilation as the ramp proceeds down. The raise will be offset at each elevation, screened and bolted and equipped as an emergency manway for a future second egress.

As mining begins in the upper West Zone, the ramp will proceed to the bottom of the mine at the 4550 level. Below the 5150 level, mining horizons will be established at 125m intervals for Alimak mining, down to the 4650 level. The East Zone will be developed separately with a ramp down from the 5150 level or as take-offs from the main ramp system above this elevation. A typical cross section is shown. Safety bays shall be cut 2×2×2m 300 mm above the finished grade of the ramp on the same side of the opening as the compressed air line (Figure 16.2).

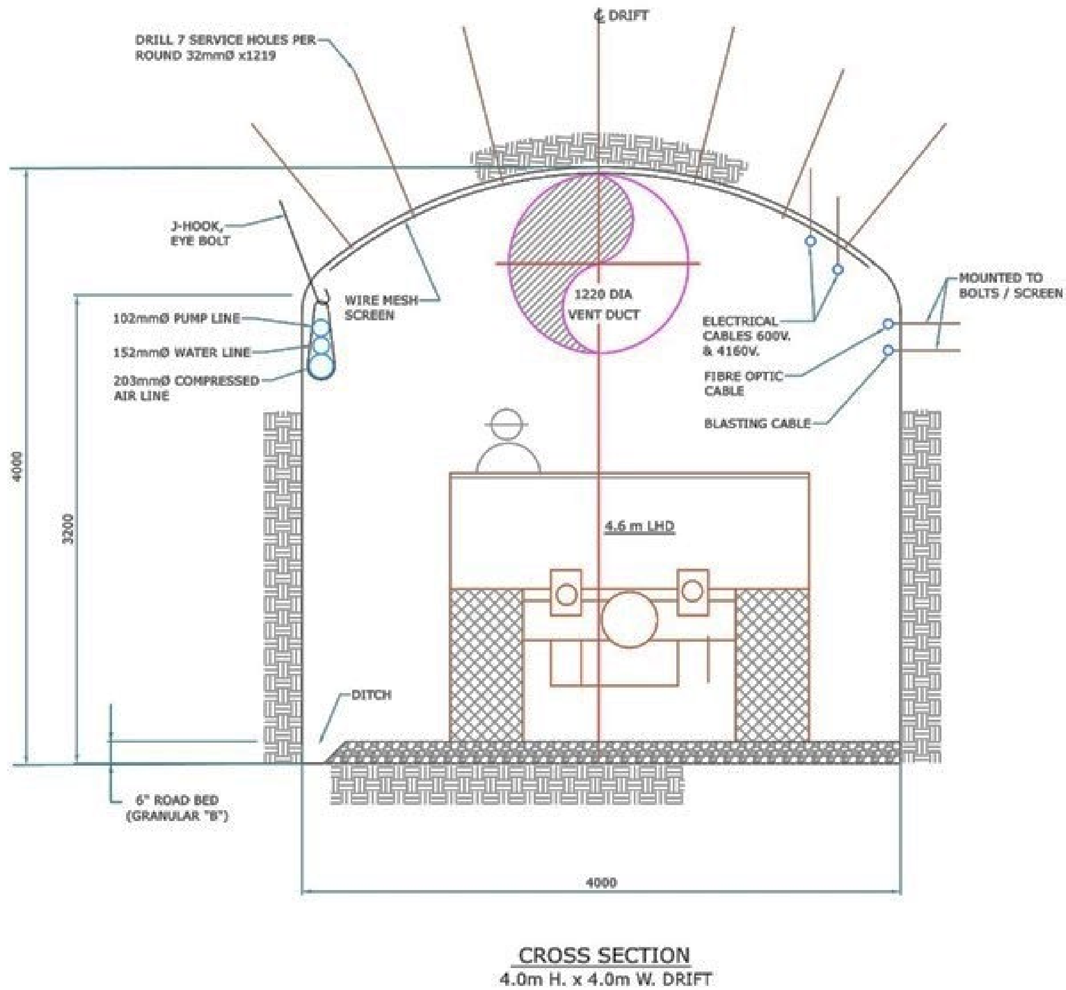


Figure 16.2 Typical Ramp Cross Section

16.2 Underground Mine Design

On each level, the mining areas would be accessed from the main ramp by a 4.0m × 4.0m wide access drift. The proposed mining method in the West Zone is Alimak mining. This method greatly reduces the lateral development required while allowing the narrow zones to be mined cost effectively. The stopes will be a maximum of 20m wide with a 2.4m × 3.0m Alimak raised driven up the centre of the stope. This will allow for cable bolting of the hanging wall of the stope for support and allow wider stopes to be mined. A backfill distribution system will be set up at 100m vertical intervals to fill the stopes with cemented rock fill. Below the 5150 level, all mining in the West Zone will be done utilising the Alimak mining method at 125m vertical intervals.

Underground development, including excavation of ramps, accesses, and haulage drifts, will employ diesel powered, rubber tired 2 boom electric/hydraulic drill jumbos, load-haul-dump (LHD) units, 30 tonne haul trucks, bolters, and scissor lifts with work platforms. Mining will utilise diesel powered, track and rubber tired mobile equipment including an ITH drill, ANFO loading units, LHDs, and haul trucks.

The East Zone will be a combination of both longhole and Alimak mining. The main core of the East Zone is quite wide allowing for longhole mining to be effective. As the zone approaches surface it gets much narrower making it more amenable to Alimak mining.

16.3 Geotechnical Considerations

For the purposes of this study, the geotechnical design has been based on conservative past practices at other operations. 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.

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.

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

16.4 Mine Access and Level Development

16.4.1 Main Access

The heading size will be 4.0m high ×4.5m wide and declines will be driven at a 1:7 (-15%) grade. The main decline will be situated in the hanging wall and will be set up as a downcast airway ensuring fresh air is flowing over the truck motors as they climb the haulage way. In the lower sections of the East Zone, the ramp will transition to be located in the footwall.

All personnel, equipment, and materials will be transported into and from the mine via this main ramp. All ore and rock (as required only) will be transported from the underground in diesel powered haul trucks operating in the underground drifts and the main ramp until the borehole hoist is established and then the material flow will be focused on moving resources and rock to passes for hoisting to surface.

16.4.2 Level Access and Development

On each level, the mining areas would be accessed from the main ramp by a 4.0m high by 4.0m wide access drift driven in the footwall for Long hole stopes and in the hanging wall for Alimak stopes both parallel to the strike of the ore zones. The wider access drift would allow for truck haulage of material from the production stopes. The proposed mining methods are stoping utilising Alimaks (horizontal ring drilling) and longhole blasting (vertical ring drilling). Stoping will take place in panels, which are nominally 15m-20m wide (along strike and depending on the stope stability based on Mathew's Stability Graph and extend lengthwise up dip over vertical intervals of approximately 50m for the long hole stopes and 100m-125m for the Alimak stopes.

Underground development, including excavation of ramps, accesses, and haulage drifts, will employ diesel powered, rubber tired 2 boom electric/hydraulic drill jumbos, 5 cubic meter load-haul-dump (LHD) units, 30 tonne haul trucks, bolters, and scissor lifts with work platforms. Mining will utilise diesel powered rubber tired mobile equipment including an In-The Hole drill unit as well as a single boom drill with extension rods drill, ANFO loading units, LHDs, and haul trucks.

16.5 Rock Handling

Initial ramp development will be done utilising 30 tonne haulage trucks to bring the muck to surface. As the borehole hoisting system is established, rock handling will transition to hoisting of both ore and waste.

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, backfill plant, maintenance shop, explosives magazines, mine rescue station, power substation, compressor station, small warehousing facility, laydown yard, and a water storage pond.

16.6.1 Electrical Distribution

Primary electrical power for the mine would be provided from the main surface substation connected to the outside powerline. Maine has multiple power suppliers within 11 km to the property. Emera Maine has indicated capacity to supply power for the project within the existing infrastructure.

The powerline would be connected to a surface substation located near to the mine portal. Power from the main substation would feed the main underground power line, a 500 mcm cable, installed in the main access ramp from the surface. This powerline would feed portable substations located on levels central to the working areas. Portable power centres would supply loads on the nearby levels and transform power down to 4,160V and 600V, as required.

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

The main underground mine electrical feed will consist of a 4,160V, 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. Equipment underground will be powered by 750 KVA portable substations located in the electrical substation openings. The substations will step power down to 600V for mining equipment and 120V for smaller, electrical powered equipment.

Table 16.1 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 Hrs/Day	Consumption Per Unit (kW)	Total Installed	Total Monthly
Production Hoist	1	67%	16	626	626	201,322
Development Jumbo	2	50%	12	180	360	64,800
Bolter/Screening Jumbo	2	50%	12	90	180	32,400
Longhole Drill-ITH	1	90%	20	37	37	19,980
Diamond Drill	1	50%	12	23	23	4,140
Ventilation Booster Fans						
X/C Ventilation	12	75%	16.5	22	264	98,010
Ramp/Level Development	2	75%	16.5	93	186	69,053
Main Ventilation Fans						
Underground Exhaust	2	100%	24	300	600	432,000
Surface Intake	2	100%	24	300	600	432,000
Pumps						
Main Dewatering Sumps	2	50%	12	44	88	15,840
Miscellaneous pumps	6	75%	18	150	900	364,500
Compressors						
Compressor 1	1	50%	12	186	186	33,480
Compressor 2	1	50%	12	186	186	33,480
Lighting and Miscellaneous	1	80%	19	70	70	31,920
Total Monthly Power Consumption (kwh)						**1,832,924**

16.6.1.1 Electrical Cabling

The electrical cabling shall be hung from messenger cable that will be installed on the opposite side 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 the roof anchors.

16.6.2 Compressed Air

Compressed air would be supplied by 2 compressors in enclosures located in the warehouse, backfill, and a compressor building, near the ramp portal. They would provide approximately 23.8 cm per minute at a minimum pressure 8.3 bar (120 psi) to the underground mine. Each compressor would operate at half capacity to ensure one compressor could provide mine requirements when the second compressor is being repaired or maintained. The compressors would supply the main compressed air pipeline located in the main access ramp from the surface.

Compressed air consumption is presented in Table 16.2.

TABLE 16.2
WATER AND AIR USAGE

Unit	Quantity	Utilization	Service Water Consumption			Compressed Air	
			Per Unit (l/min)	Total (l/min)	Yearly Consumption	Per Unit (cu.m/min	Total (cu.m/min
Development Jumbo	2	70%	120	240	84672		
Development Bolter	2	80%	60	120	48384		
Stopers & Jacklegs	8	50%	60	480	120960	5	40
Alimak Units-Double Drive	3	80%			0	10	30
ANFO Loader	2				0	15	30
Long Hole Drill-S36	1	90%	38	38	17237	16	16
Long Hole Drill-ITH	1	90%	200	200	90720	25	25
Shotcrete Machine	1	100%	8	8	4032	24	24
Air Tools (1 lot)	1	100%			0	40	40
Misc. Water Usage (1 lot)	1	100%	20	20	10080		
Water Sprays-Mucking	1	100%	40	40	20160	20	20
Gland Water, etc. (1 lot)	1	100%	10	10	5040		
Total Consumption					**401285**		**225**

16.6.3 Service Water

The underground mine would require approximately 401,000m^3 of service water per year for use in drilling, dust suppression, etc. This water will be supplied from 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 802.11N (Wi-Fi) 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 Dewatering

The average underground dewatering requirement during production will be approximately 1,420m^3 per day. This quantity is based on the mine water balance as follows:

- 1,160m^3 per day will be produced by underground production mining and development activities;
- 60m^3 per day will decant from backfilled stopes; and
- 200m^3 per day net groundwater inflow to the mine (estimated).

During the pre-production and initial year of production periods, water will be pumped directly from level sumps to the surface because mining will be near the surface.

The long-term mine dewatering 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. Main collection sumps would be located on the 5000 and 4600 Levels. Each main sump would be comprised of 2 sets of dirty water and clear water sumps. Dirty water sumps would be subdivided by removable timber baffle walls into 3 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 16.3).

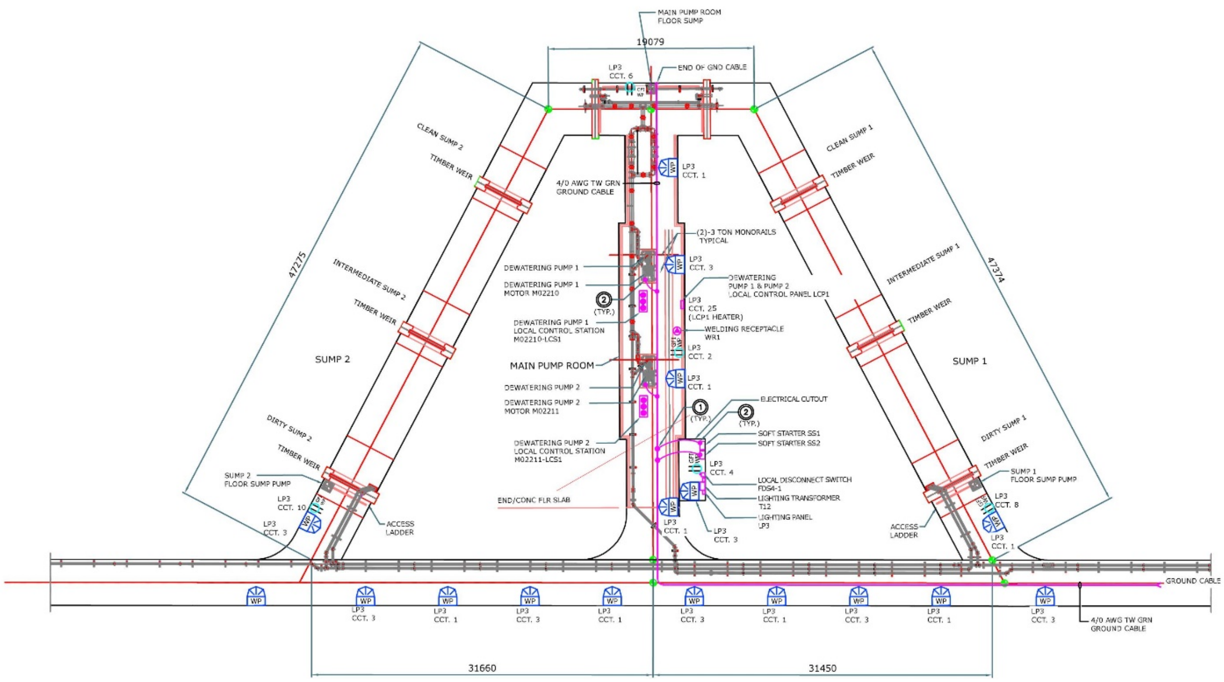


Figure 16.3 Typical Sump Arrangement

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 discharge. Water would be pumped to a surface holding pond for underground process water or discharged to the water treatment facility on the surface.

Development crews shall proceed with caution regarding inflows of water. Crew leads shall apprise engineering of flow rates after each round ensuring relevant data has been collected regarding the amount of water that has been pumped to keep the heading dry and any other data worth noting regarding safety of the workplace and dewatering. To keep records of the amount of water pumped out of the mine, a flow meter on the main discharge shall be installed.

16.6.6 Breakdown Maintenance Shop

A small breakdown shop will be set up during the pre-production period in an abandoned re-muck off ramp. This shop will be used until a permanent breakdown shop is located lower in the mine. The mobile equipment breakdown maintenance shops would be used to perform all breakdown maintenance on mobile mining equipment. Major equipment preventative maintenance work and other major repairs would be performed in a surface shop located near the portal.

The permanent breakdown shop would be constructed near the 5150 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, 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. 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.

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 16.4).

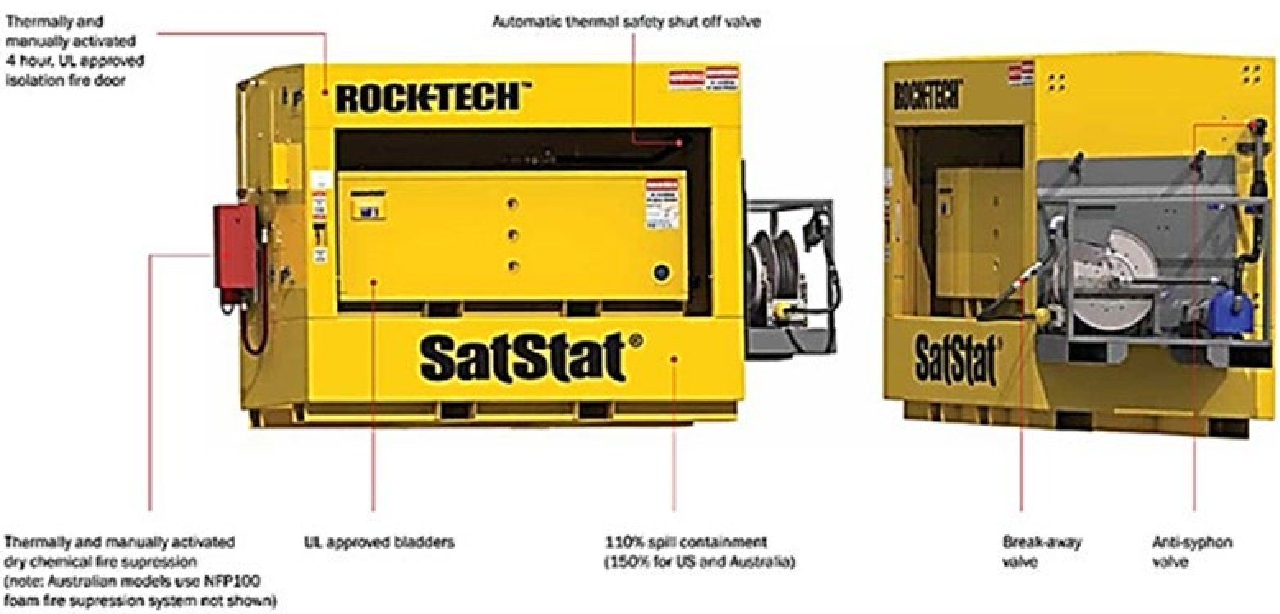


Figure 16.4 Fail Safe Fuel and Lubrication Systems

16.6.8 Refuge Stations

Main refuge stations would be located approximately every 100m vertical intervals on the 5250, 5150, 5025, 4900, 4775, and 4650 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 16.5).

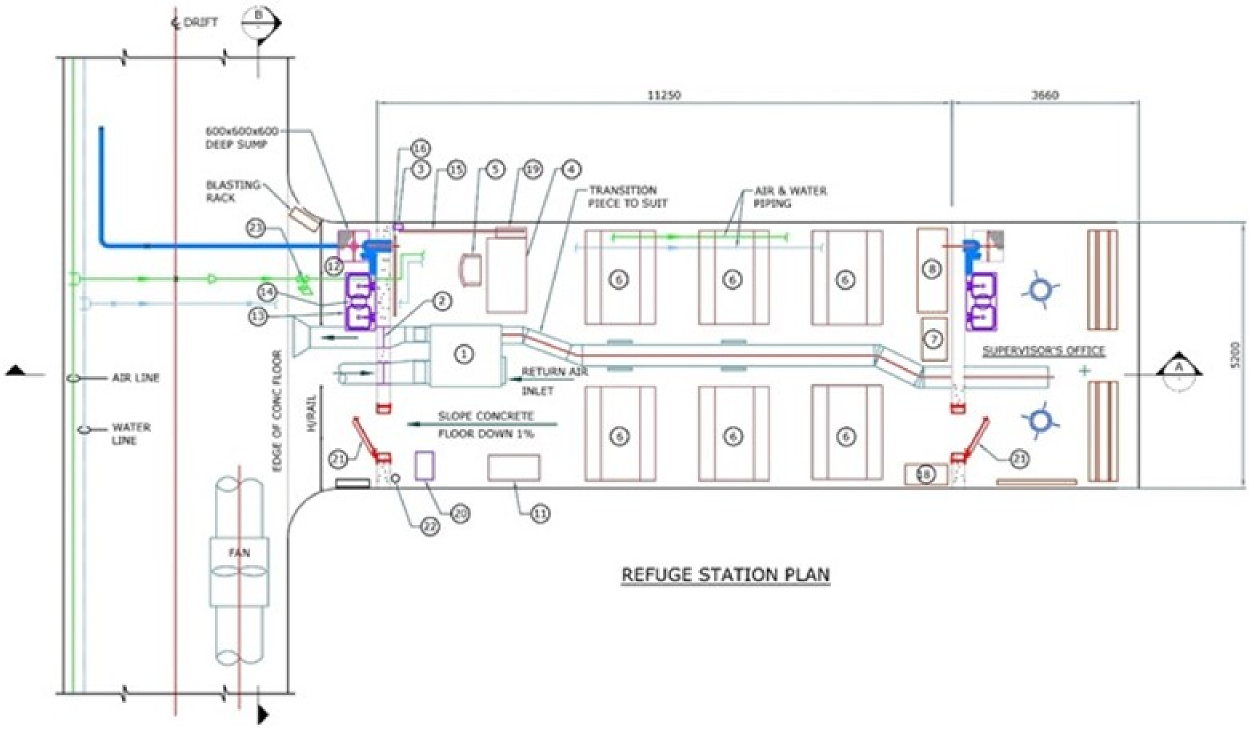


Figure 16.5 Standard Refuge Station

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.

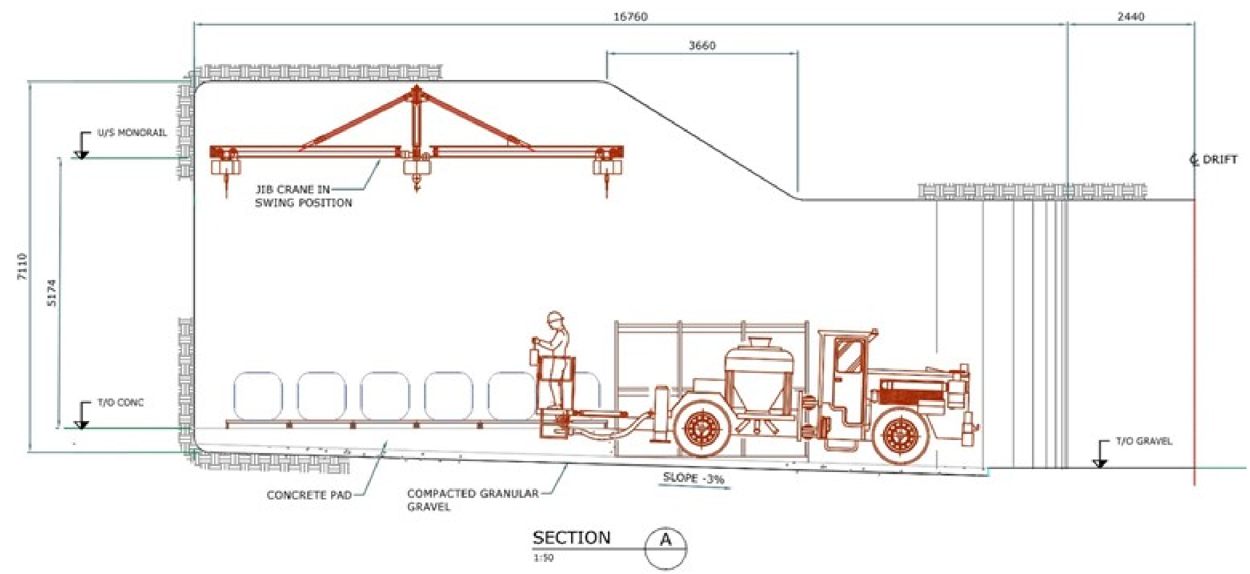


Figure 16.6 Section Through a Standard Explosives Storage Area

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 work place 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 main maintenance shop, backfill plant; explosives magazines, laydown yard, mine rescue station, water storage pond, power substation, and compressor 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 500m 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.

A laydown yard would be constructed near the ramp portal 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 property. The mine rescue station would be equipped with all 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

Two mining methods will be employed at Pickett Mountain. The West Zone is narrow and high-grade and will utilise an Alimak mining methodology. Alimak stopes will be 100m high in the upper part of the West Zone and increase to 125m high below the 5150 level. In the wider areas of the East Zone, below 5145 elevation, a longhole mining method will be employed utilising 6-inch blastholes with stoping at 50m vertical spacing. Both methods will employ a primary/secondary mining sequence.

Mining horizons would be developed on each main level (5350, 5250, 5150, 5025, 4900, 4775, 4650, and 4600 levels) Each Alimak vein stope would be 20m along strike with 1 drawpoint per stope in the centre, from the footwall drift.

An undercut over the full width and length, on the lower main level of the potentially economic mineralisation block, would be developed. An Alimak raise would be driven in the centre of a stope from the undercut to the level above the stope. The raise would be screened over its entire length to facilitate drillers working in the raise. Cable bolts would be installed into the hanging wall of the stope, from the Alimak platform in the raise, to support the hanging wall. The Alimak installation would be left in the raise and a longhole ring drill installed on the work platform of the Alimak. The longhole drill would drill 70 mm horizontal drill holes (approximately 8.5m in length) parallel to the footwall and hanging wall of the potentially economic mineralisation. Drill holes would be loaded with ANFO and Nonel detonators and blasted in horizontal slices into the undercut below. Access to stope raises to allow workers to perform drilling and blasting functions on the Alimak, would be from the level above the stope. Broken potentially economic mineralisation would be mucked from the undercut by LHDs and transported to truck loading stations. The final 10% of the stope will require mucking by remote controlled LHDs.

Stope undercut sills would be developed to full width of the zone to be mined by 3.5m high. The openings would be drilled with 2 boom E/H jumbos and mucked with 3.0m^3 bucket LHDs. Ground support would consist of 1.5m resin rebar and screen.

Alimak drilling raises would be developed 3m × 2.4m using stopers and the walls of the raise supported by resin rebar and welded wire mesh screen.

Stope mucking would utilise 3.0 m^3 bucket LHDs mucking in the drawpoints.

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.

Mined out stopes would be backfilled with cemented (primary stopes) and uncemented (secondary stopes) rockfill.

16.8 Dilution and Extraction

Expected dilution and mining recovery for the proposed Alimak vein stope mining method would be approximately 10% and 85%, respectively, with these factors included in the mineable resources.

With the Alimak mining method, a pilot raise will be developed and supported with 1.8m long resin grouted rebar on a 1.2m by 1.2m pattern and welded wire mesh screen (1.2m by 2.4m sheet with 5.6 mm wire thickness, 100 mm × 100 mm apertures) on the backs and 1.2m rebar and screen on the walls. Screen sheets will be installed with a 0.2m overlap.

The length of 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.

The following geotechnical design criterion has been used for the stopes at the Pickett Mountain deposit. The ore thickness ranges from 3.0m-5.0m wide with an average of approximately 3.5m. Stope widths are planned at 20m along strike and stope lengths up dip of approximately 100m-125m based on the level spacing and orebody dip of approximately 80°. Stope lengths have been established using an allowable hydraulic radius.

To ensure hanging wall stability while mining of a stope, the pilot raise would also have fully grouted (cement) 8.5m twin-strand bulbed, 15.2 mm cable bolts on a 2.5m pattern installed into the hanging wall of the stope in a narrow fan pattern. Cables should be tensioned and installed with plates.

In the middle and lower sections of the East Zone, longhole mining in a primary/secondary sequence will be utilised. In order to minimise development, the levels have been spaced at 50m intervals and the drilling has been sized at 6-inch blastholes. A central cross-cut 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.

Primary stopes will be filled with cemented rockfill while secondary stopes will be filled with unconsolidated fill except on the 5150 level where a sill will need to be established to allow for mining to come up underneath this original mining horizon. In this case, the bottom 10m of the stope will require good quality cemented rockfill.

More geotechnical drilling will be conducted at the Pickett Mountain deposit 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 4,180,000 tonnes at a grade of 8.56% Zn, 1.11% Cu 3.4% Pb .79g/t Au, and 88.8 g/t Ag. The tonnes and grade include an average dilution of 10% overall and a recovery of 85% of the potential resource. This PEA relies on Indicated Mineral Resources (approximately 48.7% of the total resource tons) 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 PEA to 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

Longholes in the Alimak stopes will be drilled off using a longhole drill boom incorporated into the Alimak platform with pressurised stabiliser arms, which can be extended to the walls of the pilot raise to maintain drill setups. The drill would use 1.5m long extension steel and button bits to drill the long holes of 64 mm diameter. The same drilling unit would be used to drill cable bolt holes.

A track mounted ITH drill capable of drilling up to 6-inch holes will be required to drill off the longhole stopes.

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. Alimak stopes will be blasted from the bottom up, initiating 4-5 rings per blast. During this process, the raise climber will access the raise only from the top cut. Due to the length of the stopes, it will take approximately 2 weeks to blast the entire stope. Longhole stopes will be taken in three lifts, the first two (5-7m and 10-15m) to create sufficient void to blast the bulk of the stope in the final blast (25m-30m).

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 change 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 on site.

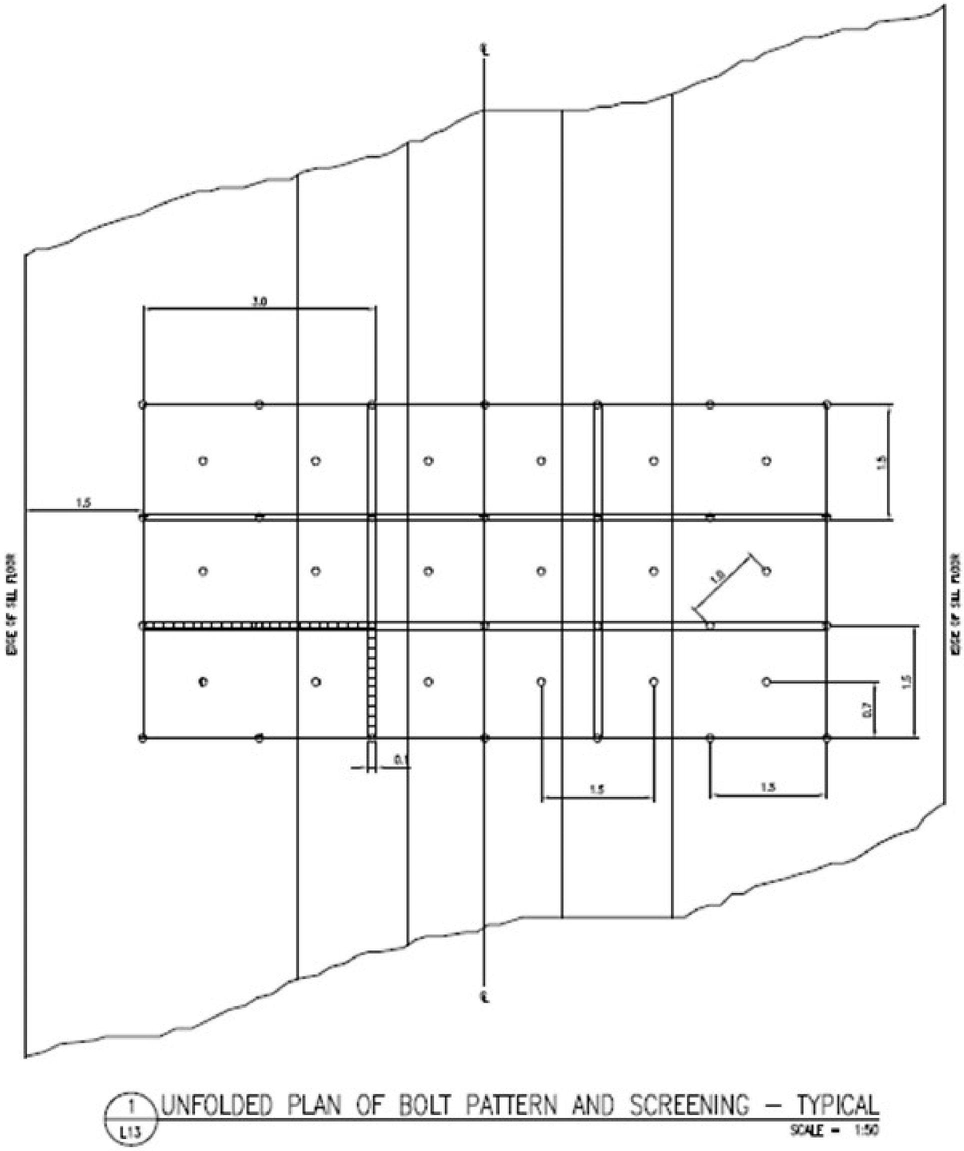
Provisional rock support shall be as follows:

- 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 Pickett Mountain Project:

  - 1.8m length × 20 mm diameter rebar bolts installed with resin on a 1.2m × 1.2m 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

6.0m cement grouted cable bolts installed in areas greater than a 5.0m diameter span (Figure 16.7,



- Figure 16.8, Figure 16.9, and Figure 16.10).

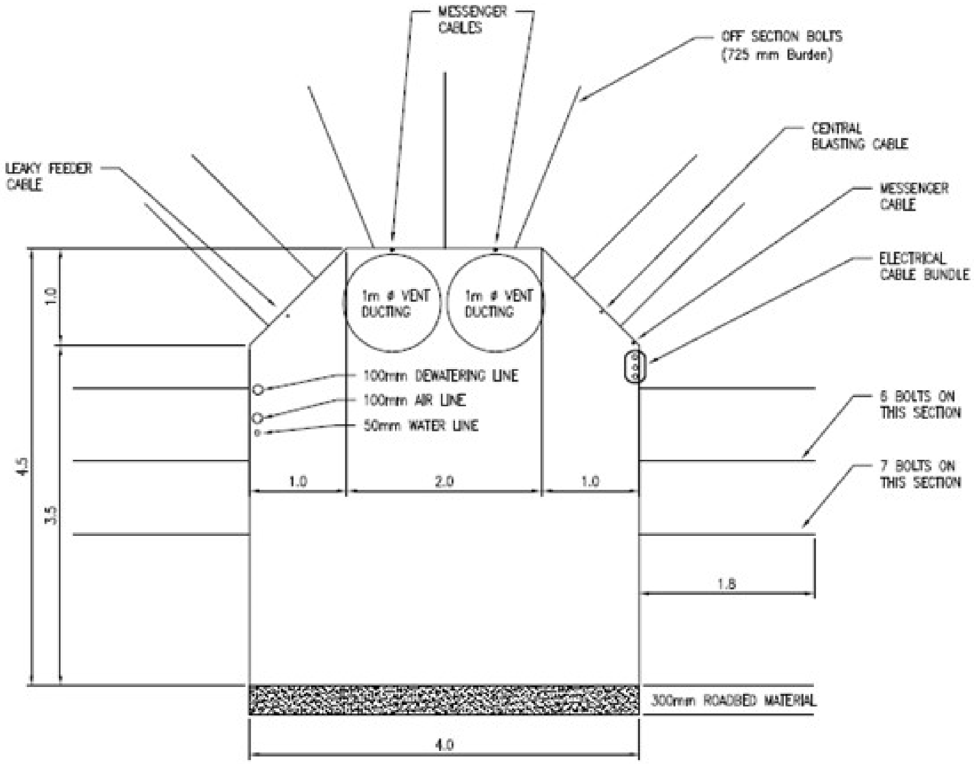


Figure 16.7 Section of a Typical Drift

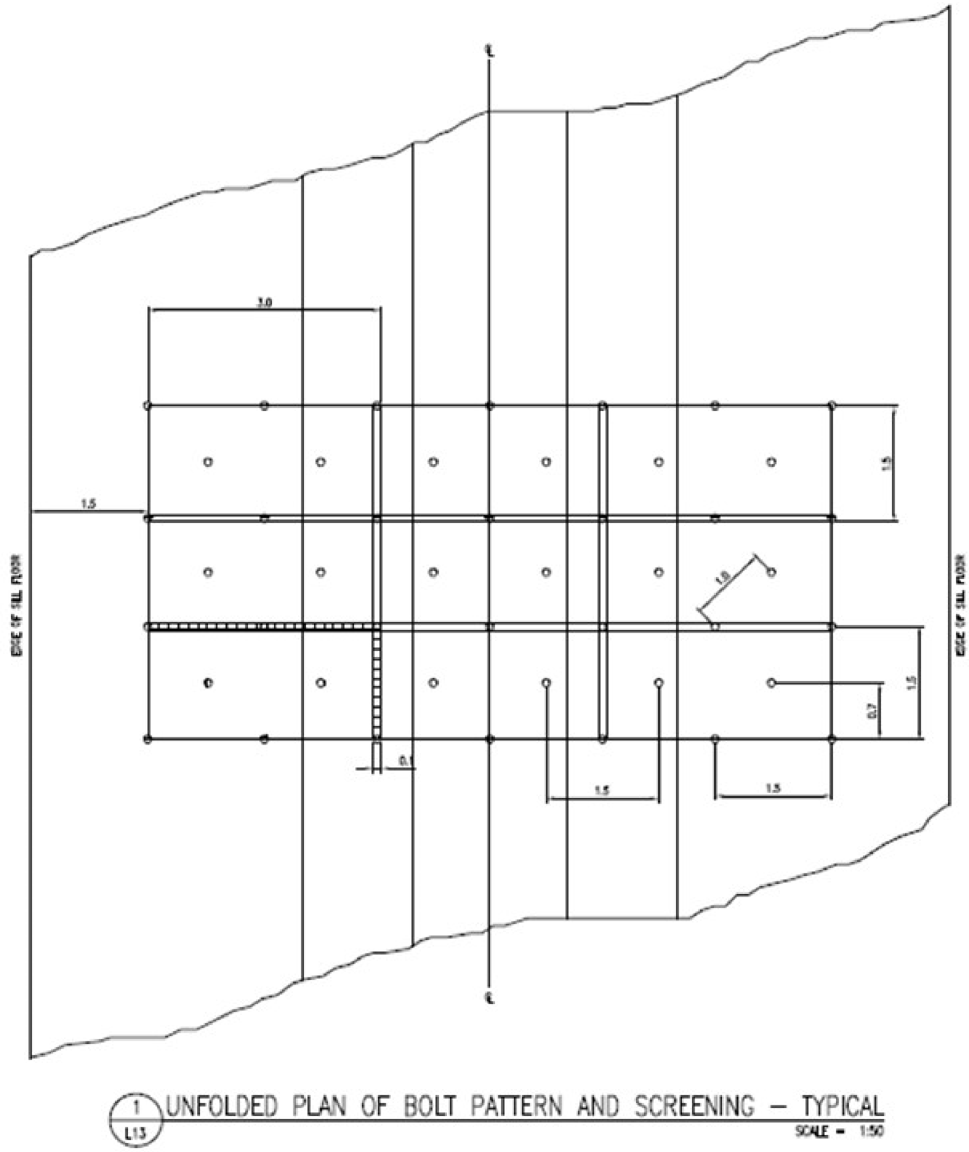


Figure 16.8 Nominal Bolting and Screening Pattern

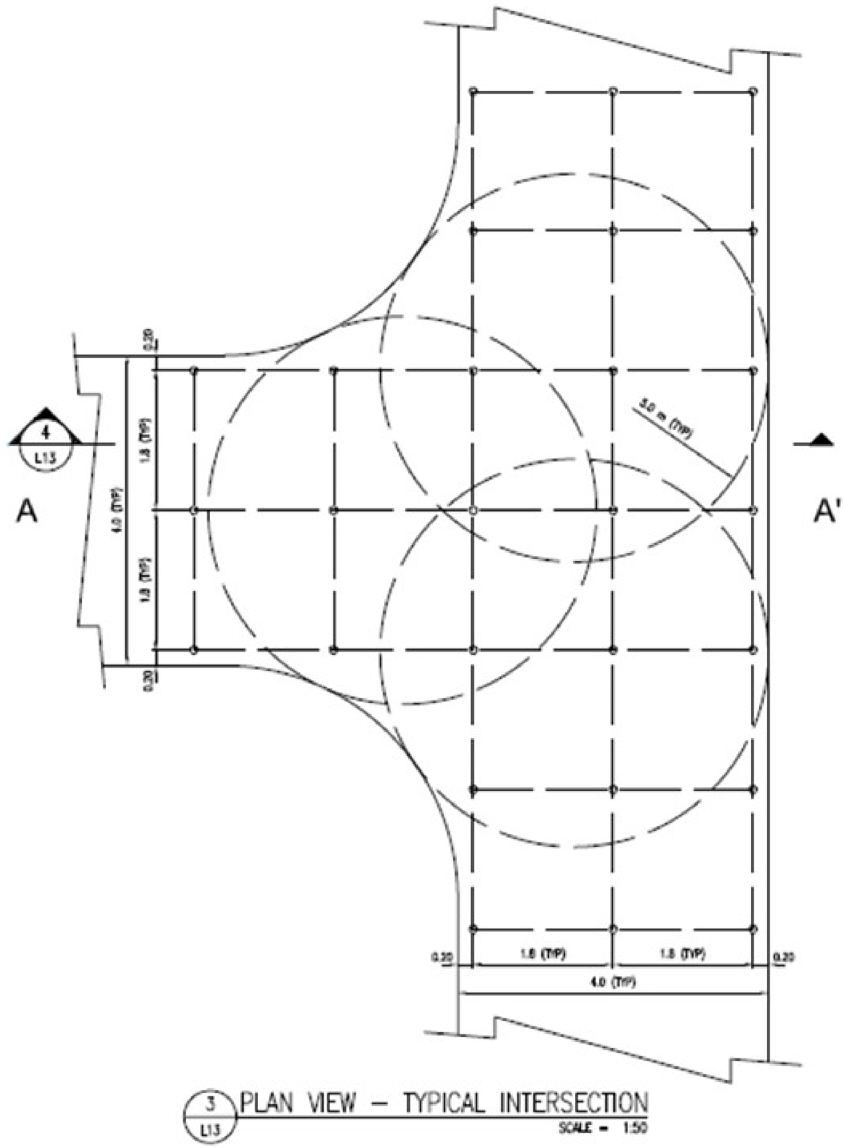


Figure 16.9 Plan of Cable Bolting Pattern in Intersections

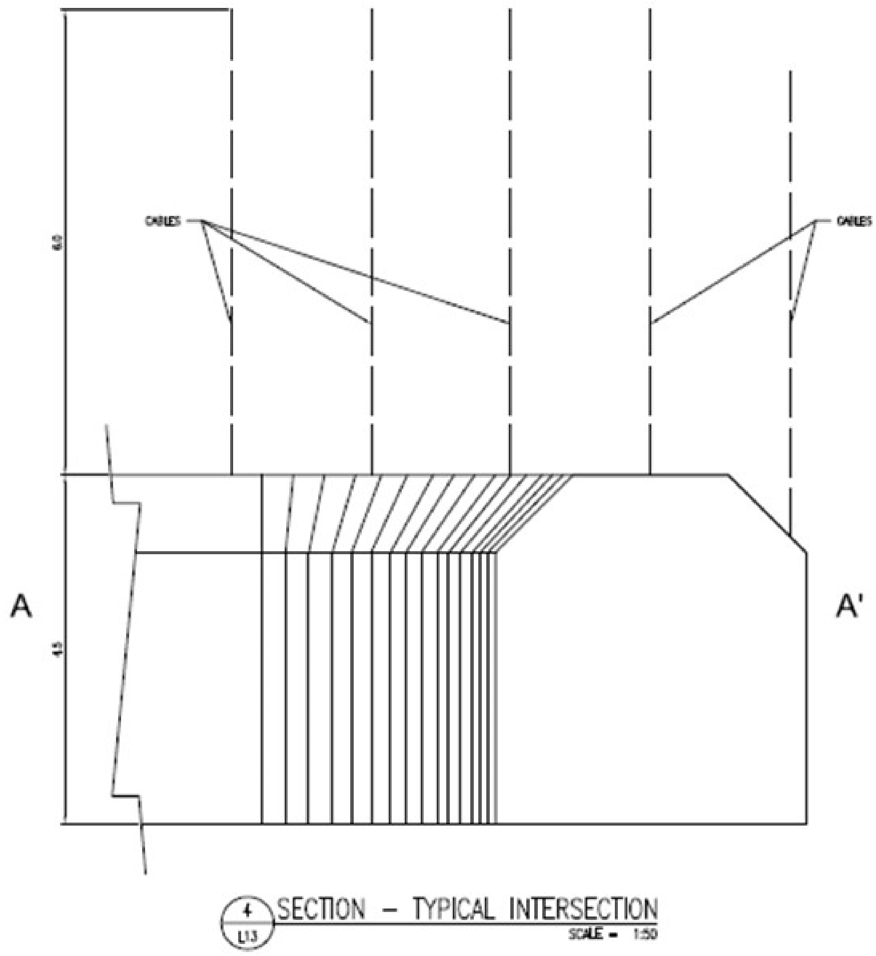


Figure 16.10 Section Showing Cable Bolts in an Intersection

16.9.4 Mucking

Mucking of the production stopes will be done utilising 5.0m^3 remote capable LHD units. Until the borehole hoist is in operation, initial production will be hauled to the surface with 30 tonne haulage trucks. Once the ore pass system and borehole hoisting system has been constructed, all mucking will be done with the LHD units and report to the ore pass on the level.

Borehole hoisting is the use of hoisting plant to hoist rock, and materials or both, vertically through a borehole, between a mine level and surface or between two levels of an underground mine.

The borehole or, in this case, the 8 foot by 10 foot raise, is part of the overall hoisting system that comprises: the hoisting plant (including the hoist, power supply, ropes, and sheaves); the headworks at the top of the raise (including the dumping and/or off-loading arrangements); the raise including the rope guide system; the 10 tonne capacity skip for carrying the hoisted material; the counter weight on the opposite side; and the loading pockets at the mid-level and at the bottom of the raise.

The hoisting system will be configured for primary production hoisting, while still providing a ventilation function Figure 16.11.

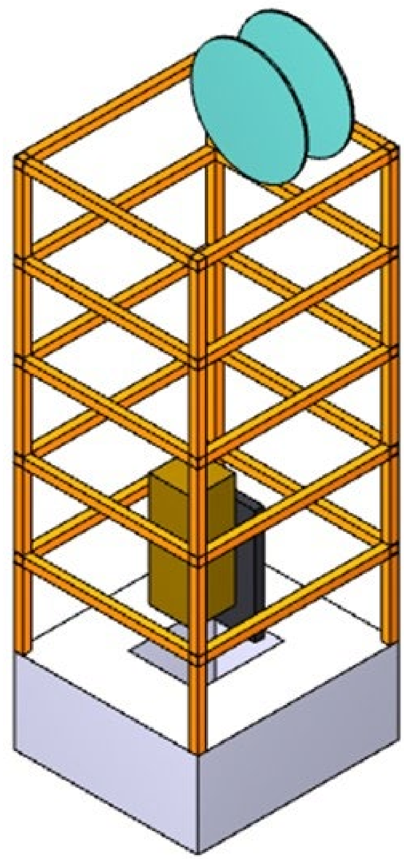


Figure 16.11 Borehole Hoist Showing Skip and Counterweight

16.10 Mining Equipment

The mine development group will require two 2 boom electric/hydraulic jumbos, two 5m^3 bucket LHDs, two 30-tonne truck c/w electric retarder, 2 scissor lift trucks, two electric/hydraulic scissor screener bolting units, one ANFO loading unit, 3 double drive Alimak units, handheld drills, and 2 light utility vehicles.

The mine production group, including stope preparation, will require 2 longhole drill units for the Alimaks, 2 cable inserting units and grout pumps, two 5m^3 LHDs c/w remote capability, two 30-tonne trucks with ejector style boxes, 1 ITH drill capable of 6-inch drilling, 1 ANFO loading unit, 1 scissor lift, and 2 light utility vehicles.

The mine services and construction group will require 1 scissor lift truck, 2 utility boom trucks, 2 personnel carriers, 1 grader, 1 light service vehicle and a 3.0m^3 LHD forklift unit.

The maintenance group will require 3 light utility vehicles, one equipped with a man basket lift attachment. Warehousing will require 1 front-end loader. The mine staff, engineering, and geology will require 3 light utility vehicles. Table 16.3 presents the mine equipment fleet.

TABLE 16.3
MINE EQUIPMENT FLEET

Equipment	Units	Development	Production	Services	Maintenance	Staff	Total
Electric/Hydraulic 2 Boom Jumbo	each	2					2
Scissor Screener Bolter	each	2					2
5 cu.m. LHD (c/w Remote System)	each	2	2				4
Haulage Trucks 30 tonnes(c/w Electric Retarder)	each	2					2
Haulage Trucks 30 tonnes (c/w ejector box)	each		2				2
Scissor-Lift Truck	each	2	1	1			4
ANFO Loader	each	1	1				2
Longhole Drill Rig	each		1				1
Cable Bolt Unit	each						0
Utility Boom Truck	each			2			2
3.0 cu.m. LHD	each			1			1
Light Service Vehicle	each	2	2	1	3	3	11
Man Carrier	each			2			2
Grader	each			1			1
Double drive Alimak units	each	3					
Handheld Drills (jackleg/stoper)	each	24		4			28

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

All stopes will be backfilled with rock backfill, cemented and uncemented, to fill the voids and prevent caving. The rock backfill will consist of development waste and mined waste rock from a surface quarry. The fill will be delivered at a rate of approximately 850 t/d in order to keep up with the voids created by mining.

16.11.1 Underground Distribution System

The fill would be delivered to the top of the stopes by the boreholes and/or Alimak raises. Fill loading chutes and slurry distribution systems will be spaced approximately 100m vertically throughout the mine at the approximate centroid of the mining area. Slurry would be pumped from the backfill plant located by the mine portal through a series of 6 inch holes to the required fill distribution level.

Each distribution system will feed off a main backfill raise from surface. Two backfill systems will be required, one for the West Zone and one for the East Zone. Slurry will be pumped to a tank at the operating system location and sprayed into the muck as the backfill truck moves back and forth underneath the loading chute. For secondary stoping, all fill will be uncemented unless a pillar needs to be established to allow mining to proceed beneath the secondary stope. Wherever possible, waste development will be used as fill.

Fill fences, constructed at the stope entrances, would consist of a muck pile jammed to the back of the crosscut. Backfill would be delivered to the top of the stope, or a borehole, by a 30-tonne haulage truck equipped with an ejector box. Truck dump bumpers will be installed at each filling location.

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 cm per second (100 cubic feet per minute) 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 cm per second (50 cfm) may be allowed for this equipment. The horsepower rating of the underground equipment was determined and utilisation factors were applied to estimate the total amount of air required (see Table 16.4, below). A model of the Pickett Mountain Mine main ventilation circuit was created in VNETPC™ from the mine design and standard frictional resistance values. Headings that would only be supplied with air via auxiliary ventilation were not included in the model.

The mining operation to support the mining equipment fleet would require ventilation air volumes of approximately 120-130m^3 per second (250,000-275,000 cfm). The ventilation system would consist of a push-pull system utilising ventilation raises and the main access ramp (Figure 16.12).

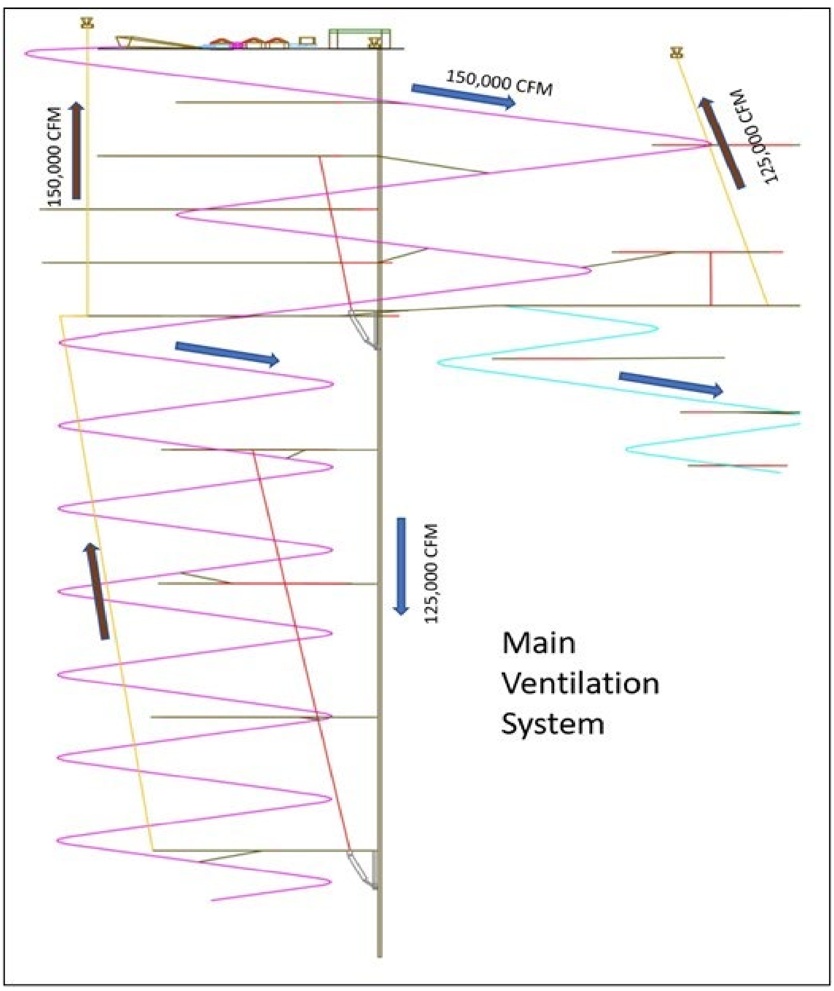


Figure 16.12 Main Ventilation System Showing 150,000 CFM Down Ramp and 125,000 CFM Down the Borehole

Two 2.7m by 3.3m ventilation raises would be developed from surface to the bottom of the mine in legs and be located at the furthermost end of the levels. An additional ventilation raise of the same size would be developed internal to the ramp as the ramp development progresses downwards. This raise would provide the main exhaust for the ramp during development and would be reversed to provide additional fresh air and a secondary escapeway once the main ventilation system is operational. One main exhaust raise would be located at the extremities of both the West Zone and the East Zone to provide ventilation for production mining.

The main air intake would be down the ramp, which would then exhaust out on each mining level, as required. 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 footwall drift on the level above the stope. Air would travel in the main footwall drift to the exhaust raise and to the surface in the raise. If required, low pressure fans would be connected to the ramp near the portal to assist air movement from the surface.

Ventilation doors and fan installations will be constructed at the portal to accommodate passage of men and equipment to and from the mine. The intake raise, internal to the main ramp, will also have a manway installed in it to provide a second means of egress.

The intake raise will be equipped with high volume, low head fans and a mine air heating unit fired by propane. The intake fans would be two 182 cm, 1,000RPM and 150 HP at approximately 1 kPa operating pressure and supply 130m^3/s (275,000 cfm) of fresh air.

The exhaust fans installations will be located on the first level below surface at the top of the two exhaust ventilation raises (west and east exhaust raises). An offset from the lower raise will be required at this level to accommodate the fans and bulkhead structure. The 2 exhaust raise ventilation fans will be 152 cm, 1,500 RPM, 150 HP fans with a capacity of approximately 65m^3/s (140,000 cfm) with maximum operating head pressure capacity of approximately 2-2.5 kPa. Flow into the exhaust raises will be controlled by the use of 75-100 HP fans and ventilation regulators and bulkheads. The portal fans will be two 100 HP fans moving 65m^3/s (140,000 cfm) of air, at approximately 2-2.5 kPa operating pressure. The central ventilation raise internal to the ramp will be similarly equipped. Fans will be variable speed to facilitate adjusting of air volume delivery to working areas, as required.

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 development will use 150 HP fans. Other lateral development will use a combination of 75 HP and 100 HP fans depending on the heading length. Development headings are sized to accommodate large ducting (122 mm), to reduce head losses.

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

TABLE 16.4
VENTILATION REQUIREMENTS

Units	Quantity	Engine kW	Engine Hp	Total Installed Hp	Utilization	Total Ventilation CFM (000)
Electric/Hydraulic 2 Boom Jumbo	2	110	148	295	25%	74
Scissor Screener Bolter	2	110	148	295	25%	74
5 cu.m. LHD (c/w Remote System)	4	235	315	1261	50%	630
Haulage Trucks 30 tonnes	4	310	416	1663	60%	998
Scissor-Lift Truck	4	110	148	590	25%	148
ANFO Loader	2	110	148	295	30%	89
Utility Boom Truck	2	110	148	295	50%	148
3.0 cu.m. LHD	1	75	101	101	50%	50
Light Service Vehicle	11	55	74	811	30%	243
Man Carrier	2	110	148	295	20%	59
Grader	1	110	148	148	50%	74
TOTAL				**6048**		**2586**

16.13 Development and Production Schedules

Mine production will be 1,200 t/d or 432,000 t/year. Development is scheduled to meet stope mining requirements, on each yearly basis. Priority will be given to developing the down ramp in order to establish the first mining block on the 5150 level then progress down to the bottom of the West Zone to establish mining from the 4650 level upwards.

16.13.1 Productivities

Development crews in waste headings will generally have multiple headings available for advancing at any time. For development scheduling, each crew is scheduled to advance 1.5 rounds (3.5m length) per day, of 4.0m by 4.5m or 4m by 4m headings, for a total of 1,835m of advance per year (not including safety bays, slashing, cut outs, etc.).

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

- Each stope pilot raise produces approximately 70 tonnes per round, 1.5 rounds per day. There will be one pilot raise being developed at any time for a total of 100-140 t/d.
- Each stope can blast and muck the equivalent of 250-300 t/shift, requiring 2 available stopes mining at a time.
- To meet daily production will require 2 stopes in mining; 1 stope in pilot raise development, 1 stope in access development, 1 stope being backfilled, and 1 being readied for backfilling. Therefore, a total of 6-7 stopes at any time is required.
- Development has been scheduled so it is well ahead of the mining requirements and mining takes place on more than one level simultaneously.

16.13.2 Underground Mine Development Schedule

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

The 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 20% additional development factor was applied to all metres to account for safety bays, small storage areas, and other cut outs required.

16.13.3 Mine Production Schedule

The mine production schedule is based on mining 1,200 t/d of potentially economic mineralisation, for 360 days per year. Table 16.5 and Table 16.6 presents the development schedule for life of mine.

TABLE 16.5

LIFE OF MINE LATERAL DEVELOPMENT SCHEDULE

				Year -2	Year -1	Year 1	Year 2	Year 3	Year 4	Year 5	Year 6
Access Development	Borehole Hoist	Dump Access	200			100		100			
		ancilliary	30			15		15			
	Main Access	East Decline	1368			660	500	208			
		Main Decline	5950	1,439	1,352	1,500	1,500	159			
		Ancillary	1098	216	203	324	300	55	-	-	-
		Vent Accesses	355	70	70	70	70	75			
	Exploration		600						300	300	
Longhole Mining	5355	45 Bottom			60					223	355
	5255	145			60					224	353
	5155	245			120	316	320	200	157	321	125
	5105	295				155	291				
	5055	345					245	123	144		
	5005	395								250	143
Alimak Mining	5350	50			485						
	5250	150			640	255					
	5150	250			680	275	63				
	5025	375					191	150	400	123	
	4900	500					190	150	321	123	
	4775	625						300	297	50	
	4650	750						300	216	221	
	4600	800									
	Level Development Totals		-	1,725	3,670	3,670	3,670	1,835	1,835	1,835	976

TABLE 16.6

LIFE OF MINE VERTICAL DEVELOPMENT SCHEDULE

	Year -2	Year -1	Year 1	Year 2	Year 3	Year 4	Year 5	Year 6	Year 7	Year 8	Year 9
Vertical Development											
West Exhaust				300	477						
East Exhaust							456				
Decline Exhaust	202	218	218	162							
West Backfill System			275			350	175				
East Backfill System					200	225					
Borehole Hoisting System											
Alimak			400		450						
400 Pocket			72								
750 Pocket					80						
Mineralised Development m		400	1,250	1,550	1,000	1,000	1,000	1,000	1,000	1,000	800
Total Vertical Development m	202	618	2,215	2,012	2,207	1,575	1,631	1,000	1,000	1,000	800

The production schedule is derived from scheduling all the stopes, which meet the 7% ZnEq cut-off grade. At the end of Year -1 initial stoping started on the 5150 level working up to the 5350 level, as this minimises pre-production development, while accessing more than one year of production. Table 16.7 presents the summary production schedule.

TABLE 16.7
LIFE OF MINE PRODUCTION SCHEDULE

	Year -1	Year 1	Year 2	Year 3	Year 4	Year 5	Year 6	Year 7	Year 8	Year 9	Year 10	Totals
Tonnage	64800	367200	432000	432000	432000	432000	432000	432000	432000	432000	292094	**4180093**
Zn %	8.43	8.43	8.43	11.79	11.46	11.46	9.04	6.43	6.52	5.93	4.93	**8.56**
Cu %	1.52	1.52	1.52	1.01	1.02	1.02	1.03	0.92	0.97	1.08	0.98	**1.11**
Pb %	3.47	3.47	3.47	4.64	4.53	4.53	3.56	2.48	2.53	2.36	1.89	**3.40**
Au g/t	0.80	0.80	0.80	0.92	0.90	0.90	0.78	0.76	0.71	0.65	0.59	**0.79**
Ag g/t	94.34	94.34	94.34	118.82	116.62	116.62	92.17	65.88	67.44	59.08	50.19	**88.80**

16.14 Mine Surface Infrastructure

Surface facilities will generally be centred near the portal or processing plant.

Surface support facilities will include explosives magazines, mine supervision, geology, engineering and administration offices and mine change house, power substation, warehouse and laydown yard, and water collection ponds.

16.14.1 Explosives Magazines

The explosives magazine would be located 500m 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

The backfill plant would be capable of supplying cement slurry to the underground to meet the total daily backfill requirements to support production underground. The following criteria are used in the plant capacity design:

- Yearly Mining Rate: 432,000 tonnes
- Mine Operating Days: 360 days
- Daily Mining Rate: 1,200 tonnes
- Backfilling Capacity: 900 t/d
- Backfilling Placement Time: 18 hours per day
- Backfilling Rate per Hour: Approximately 35 tonnes per hour
- Backfill Mixtures 70% Solids: Approximately4% cement by weight and uncemented

A small surface quarry will be required to augment the development waste and supply sufficient fill.

A warehouse, backfill, and compressor building will be constructed from shipping containers stacked two high for walls and a roof placed on top of the container walls. The building will have dimensions of 10m by 36m. The containers will provide space for warehouse storage of smaller items. Larger items will be stored at the main warehouse near the processing plant. Items would be stored on a combination of pallet (large or bulk items) and shelved (smaller items) storage systems. Valuable items would be placed in a locked storage area.

A mine laydown yard will be constructed near the portal 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 cold storage building will store equipment requiring protection from the elements but will not require heating. Smaller supplies and components requiring heated storage would be placed in the warehouse at the mill.

All underground mine water would be sent to the water treatment facility and reused or discharged. All mine process water will be obtained by gravity clarification of mine effluent water in a 3-stage settling pond system. It is assumed that this system is of sufficient capacity to produce clear enough overflow that can be used for the underground equipment and to a lesser degree, the backfill plant. Additional steps, such as the use of flocculants, might have to be considered should the clarity of the recycled mine water not be suitable for use in meeting the site process water demands.

Mine process water will be transferred from the settling ponds to the underground mine workings by a surface pump house feeding the main water distribution piping system through a water line located in a dedicated mine service raise.

A drilled well will be used to meet all of the potable water demand at the mine.

A fully equipped mine rescue station is required on the property and will be incorporated into the shipping containers near to the warehouse, compressors, and backfill 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 ore reserve estimations. There will be a separate exploration group to undertake the exploration work on the property and 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 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, surface storage pads, concentrator feed belts, and concentrate and tailings handling locations.

16.16 Underground Personnel

The underground workforce is anticipated to be initially contracted then move to an owner operated workforce in year 3 of operation. Significant training will be required throughout the entire project life due to lack of local experience. 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 shows the anticipated manpower compliment in the mine.

TABLE 16.8
UNDERGROUND MINE AND MAINTENANCE MANPOWER COMPLEMENT

Positions	Total Complement
Alimak Stoping	
Driller	4
Blaster	2
LHD Operator	2
Longhole Stoping	
Driller	4
Blaster	2
LHD Operator	2
Lateral Development	16
Alimak Raise Development	10
Total Mine Development and Stoping Manpower	**42**

Position	Total Complement
Serviceman	8
Grader Operator	4
Construction/Services/Backfill Leader	1
Construction /Services/Backfill Helper	2
Lamproom/Dryman	4
Hoistman	4
Total Services Complement	**23**

Position	Total Complement
Leadhand Mechanic	4
Mobile Mechanic	4
Mechanic	4
Mechanics Helper	4
Electrician	2
Electrician Helper	2
Stationary Mechanic	2
Total Mine Maintenance Department Manpower	**22**

Position	Complement
Mine Manager	1
Mine Superintendent	1
Mill Superintendent	1
Technical Services Superintendent	1
Senior Engineer	1
Accountant	1
Eng/Geo technicians	2
Purchasing/Warehouse Manager	1
Environmental Coordinator	1
Medical Contract	1
Security Guard	4
Site Services	1
Total Complement	**16**

17.0 Recovery Methods

The conceptual process flowsheet and the process design criteria were developed based on the metallurgical results discussed in Section 13.0.

17.1 Conceptual Process Flowsheet

SGS test work in 1984 and RDi test work in 2018 indicated that sequential flotation process was a better choice over the bulk Cu/Pb flotation process. The conceptual process flowsheet, given in Figure 17.1, consists of three stage crushing to produce feed of P_{80} of 0.5 inch for the milling circuit. The potentially economic mineralised material is ground to P_{80} of 37 microns in a ball mill with SO_2. The ground product is conditioned with M200 and A325 at natural pH and rougher copper concentrate floated. The rougher concentrate is reground to P_{80} of 21 microns and the concentrate is cleaned two to four times to produce a plus 25% Cu concentrate.

The rougher copper tailing and first-cleaner tailing are combined and sent to the lead flotation circuit. Lime, $ZnSO_4$/NaCN are added to condition the potentially economic mineralised material to pH 9.5 and A325 collector is used to float lead minerals. The lead rougher concentrate is reground to P_{80} of 14 microns and subjected to two to four stages of cleaner flotation to produce marketable Pb concentrate.

The Pb rougher and first-cleaner tailings are combined and conditioned with lime and copper sulfate and zinc rougher concentrate floated using A343 collector and M200 frother. The zinc concentrate is reground to 22 microns and cleaned in two to four stages of flotation to produce a marketable zinc concentrate.

The zinc rougher and first-cleaner flotation tailings are combined and constitute final tailing at this time.

RDi's open-circuit test work indicates that two to three cleaner flotation stages would be sufficient for producing marketable-grade Cu, Pb, and Zn concentrates using slightly different reagents than used in the SGS study. However, the Capex is determined for four stages of cleaners for each mineral.

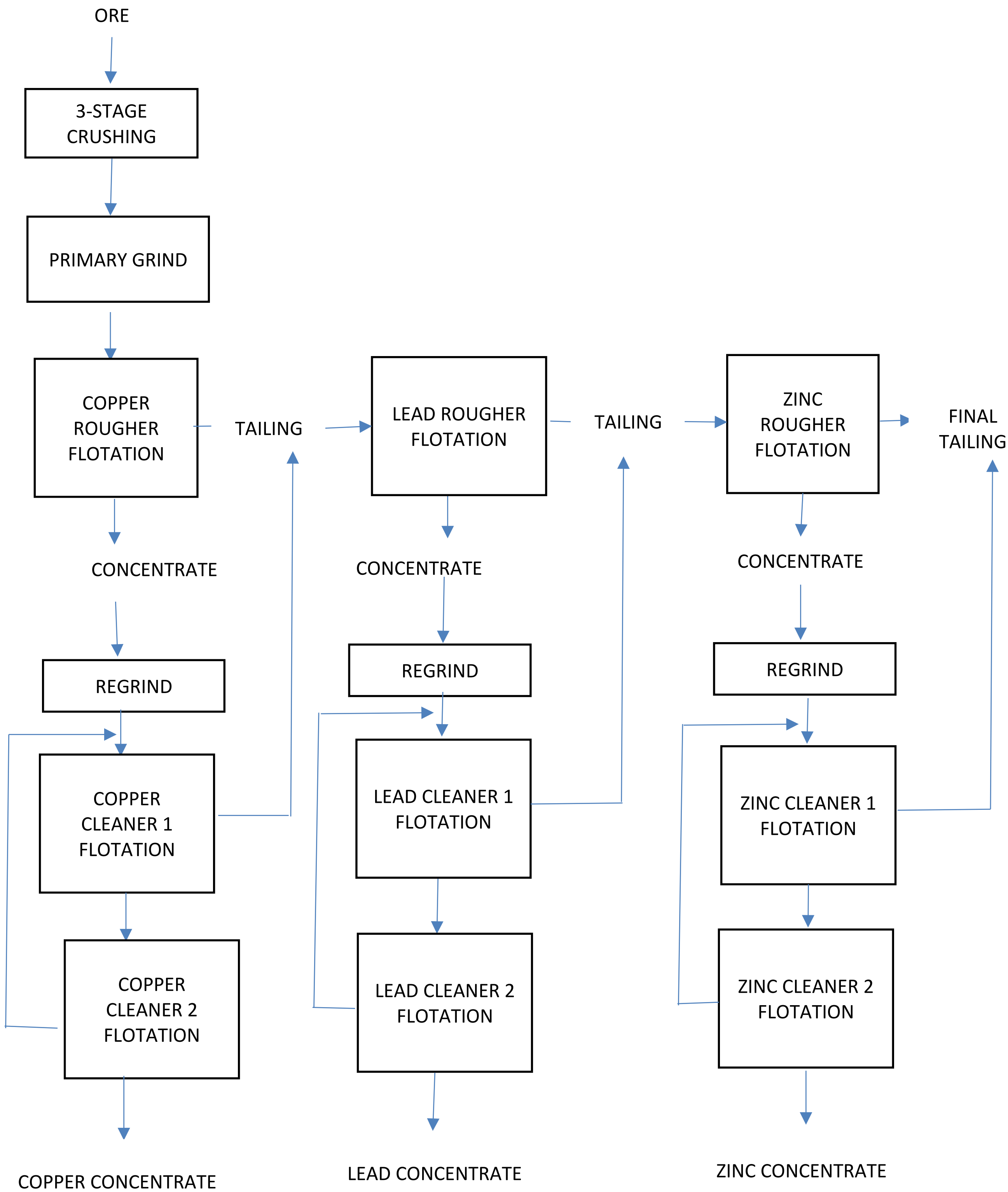


Figure 17.1 Conceptual Process Flowsheet

17.2 Process Design Criteria

The plant is designed to process 1,300 mtpd with an overall availability of 92%. The design criteria were developed using the locked-cycle flotation test No. 46, discussed in Section 13.0. The design criteria are given in Table 17.1.

TABLE 17.1
DESIGN CRITERIA FOR PLANT PROCESSING 1,300 MTPD

No.	Parameter	Units	
1.	Plant Tonnage	dmtpd dmtph	1300 54.2
2.	Plant Availability	%	92
3.	Design Plant Tonnage	dmtph	59.0
4.	Crushing Circuit Feed, F_{80}	Inches	4
5.	Crushing Circuit Product, P_{80}	inches	0.5
6.	Bond's Ball Mill Work Index	Kwh/st	10.94
7.	Flotation Feed, P_{80} microns		37
8.	Cu Rougher Conc. Regrind Product, P_{80}	Microns	21
9.	Pb Rougher Conc. Regrind Product, P_{80}	Microns	14
10.	Zn Rougher Conc. Regrind Product P_{80}	Microns	22
11.	Cu Flotation Circuit Rougher Cleaner 1 Cleaner 2 Cleaner 3 Cleaner 4	 Min Min Min Min Min	 20 15 10 7.5 5.0
12.	Pb Flotation Circuit Rougher Cleaner 1 Cleaner 2 Cleaner 3 Cleaner 4	 Min Min Min Min min	 20 15 10 7.5 5.0
13,	Zn Flotation Circuit Rougher Cleaner 1 Cleaner 2 Cleaner 3 Cleaner 4	 Min Min Min Min Min	 20 20 15 10 7.5

17.3 Reagents

The consumption of reagents in the milling circuit was estimated from the locked-cycle test. The consumption of each reagent per tonne of ore is given in Table 17.2.

TABLE 17.2
ESTIMATED REAGENT CONSUMPTION PER TONNE OF ORE

Reagent	Kg/t
SO_2	1.34
Lime	5.5
$ZnSO_4$	1.6
NaCN	0.8
A325	0.15
M200	0.063
$CuSO_4$	1.15
A343	0.05
Flocculant	0.10

17.4 Process Make-up Water

The water balance was estimated based on the following assumptions:

- Open-circuit scoping flotation test data was used to estimate the tonnage of copper, lead ,and zinc concentrates.
- The remaining tailing will also be thickened and filtered since there will not be tailing pond at site.
- All the filtered products will have 20% moisture. The remaining water will be collected and recycled as process water to the plant.

The estimated overall water is given in Table 17.3. It assumes no evaporation loss. The overall water balance indicates that the plant will need approximately 325m^3/day of fresh water since the amount is lost to the products from the plant. However, if pressure filters are installed, the water requirement would be less.

TABLE 17.3
CONCENTRATOR WATER BALANCE

Product	Solids		Water t/d or m^3/d	Comments
	%	t/d		
Plant Feed (Flotation Feed)	30	1,300	3,033.3	Need Per Day
	80	20.15	5.04	Lost in Concentrate
Pb Conc.	80	13.78	3.45	Lost in Concentrate
Zn Conc.	80	64.35	16.09	Lost in Concentrate
Tailings	80	1,201.72	300.43	Lost in Concentrate
Process Water Recycle	0	0	2,708.29	Amount Recovered
Makeup Process Water	0	0	325.01	Makeup Per Day

17.5 Material Balance (Product Production)

Several assumptions were made in order to calculate the material balance for the conceptual process flowsheet. They were as follows:

- Limited scoping level test work has been completed to date. Only open-circuit test data was available for obtaining material balance.

- An assumption was made that 95% of the metal of interest recovered in the rougher flotation will report to the final concentrate.

- Reliable data for lead flotation was not available. Hence, assumed that the lead concentrate will assay 35% Pb based on historical data and the lead recovery in the concentrate will be ±70%.

The material balance is presented in Table 17.4. Additional open-cycle and locked-cycle tests need to be undertaken in order to refine/confirm the material balance for the conceptual process flowsheet.

TABLE 17.4
MATERIAL BALANCE FOR THE PROCESSING PLANT

Product	Recovery %						Grade				
	Wt.	Cu	Pb	Zn	Au	Ag	Cu	Pb	Zn	Au	Ag
Plant Feed	100	100	100	100	100	100	1.32	1.59	8.86	0.93	98.0
Cu Rougher Concentrate (open circuit)	9.6	84.7	18.2	6.0	29.0	49.6	11.6	3.01	5.35	2.8	505.0
Cu Final Concentrate (cleaner 2)	4.3	80.5	4.8	1.3	28.6	46.5	24.8	1.79	2.63	6.19	1059.0
Pb Circuit Feed	95.7	19.5	95.2	98.7	71.4	53.5	0.26	1.58	9.14	0.69	54.8
Pb Rougher Concentrate	13.9	8.0	(74.0)	9.1	33.1	31.9	0.73	(8.05)	6.00)	2.2	225
Pb Final Concentrate (Cl3 Conc)	3.2	2.4	(70.3)	1.5	9.5	22.1	1.01	(35.0)	(4.00)	2.76	677
Zinc Circuit Feed	92.5	17.1	24.9	97.2	61.9	31.4	0.24	0.4	9.31	0.62	33.3
Zinc Rougher Conc.	37.2	6.3	16.7	94.2	36.1	17.4	0.22	(0.4)	22.4	0.9	46
Zinc Final Conc. (Cl 2 conc.)	14.2	1.9	6.4	89.5	4.9	5.2	0.18	0.72	55.7	0.32	36
Final Tailing	78.3	15.2	18.5	7.7	57.0	26.2	0.26	0.38	0.87	0.68	33
Note: The numbers in brackets were assumed based on past experience. Additional test work will be needed to confirm that the assumed grade and recovery can be achieved											

The material balance projected Cu, Pb, and Zn recoveries in their respective concentrate to be 80.5%, 70.3%, and 89.5%. The concentrate grades were 24.8% Cu, 35% Pb, and 55.7% Zn, respectively.

The historical locked cycle recoveries and grades for Cu, Pb, and Zn in their respective concentrates were 77.4% at 23.1% Cu, 77.5% at 50.9% Pb, and 87.7% at 53% Zn, respectively.

The results are similar in the two cases except for the Pb concentrate. The samples tested at RDi had been aged, and hence, effected lead metallurgy. Higher lead recovery and concentrate grade are to be expected for fresh core samples. The lead recovery and concentrate grade are assumed to be the same as obtained in the locked-cycle test.

18.0 Infrastructure

A total of $19.5 million will be required to supply and install the infrastructure for the Pickett Mountain Project.

18.1 Existing Infrastructure

The Pickett Mountain Project is a green fields mining project in the midst of a logged area that has access roads used by the foresters to reach timber lots. The right-of-way has been established and it requires upgrading to meet safety standards for higher volumes of traffic that will occur with the advent of construction and operation of the mine.

18.2 Access Road

The access road from the paved highway (Hwy 11) is approximately 5 miles (8 Km) to the mine site. There are $1.5 million set aside to upgrade the road ensuring safe reliable access year round (Figure 18.1).

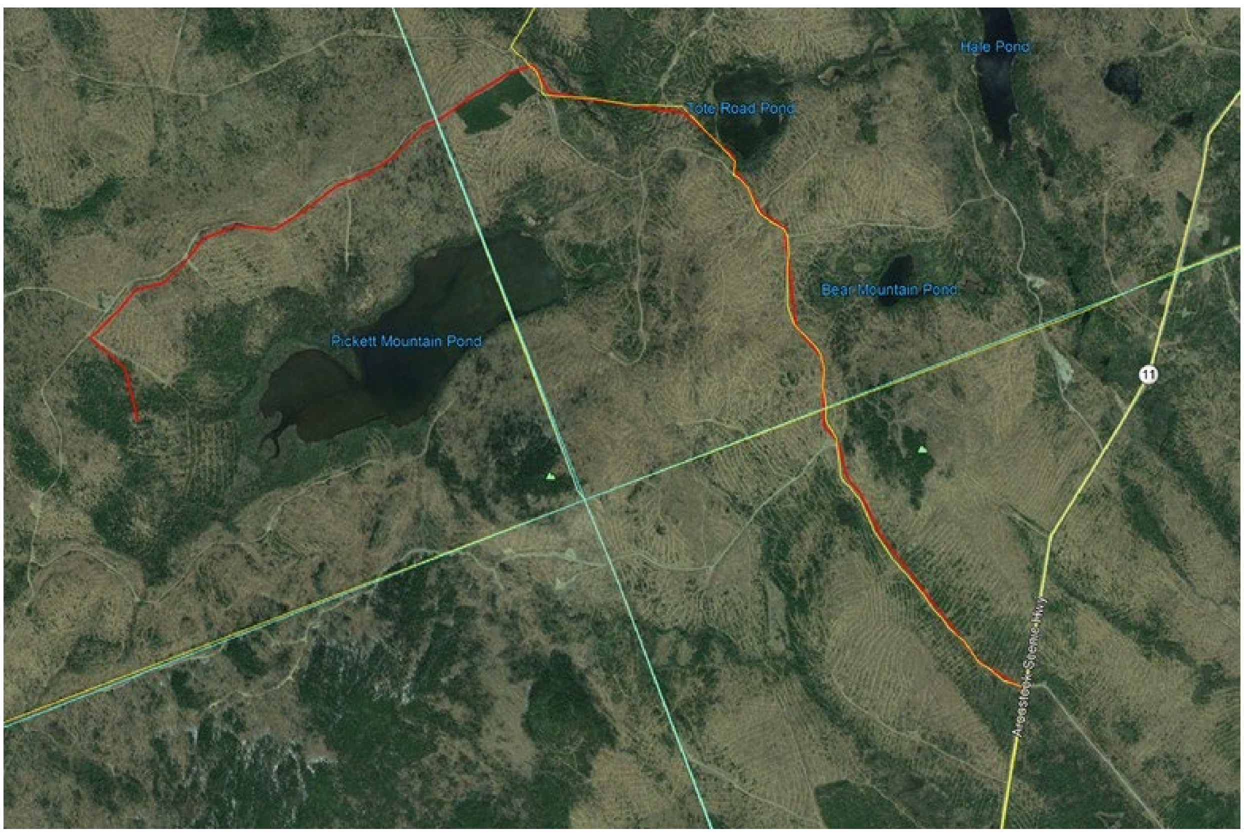


Figure 18.1 Potential Road Access Upgrade

18.3 Site Pad

Based on the size of the operation, the lay of the land, and considering the environmental and social sensitivity of any construction of mines, it has been determined to utilise a 1,500 foot by 500 foot (460m by 150m) pad as the main construction and operation area. The project requires accommodation for the building and construction phase as well as have the room necessary to safely operate the various components of the process.

Figure 18.2 shows the westerly most configuration of the main mining infrastructure pad (see Figure 18.4, below, for overall site plan).

Figure 18.2 Potential Pad Design

18.3.1 Main Pad Preparation

Maine Drilling and Blasting was contacted to get budgeting pricing on the preparation of the main pad. There is a $65,000 mobilisation fee plus $3.45 per cubic yard to drill and blast the material necessary to level out the area. The volume required to make this pad including its sub-grade is approximately 500,000 cubic yards at a cost of $1.79 million. This is only the figure for drilling and blasting and not the figure for removing the material, crushing it to size, and then placing it to grade.

18.3.2 Pad Construction

The pad placement will consist of various grades of cleaned crushed stone covering the entire area. The thickness is 15 feet to accommodate underground piping, electrical utilidors, and other infrastructure that needs protection from mine equipment or staging areas. The cost is estimated at $2.71 million at a $5 per ton rate.

18.3.2.1 Parking

An additional $50,000 has been allocated for parking at the mine site. Parking will be designed to accommodate roughly 75 employees in order to manage contractors during construction. Parking will be outdoors and will be of simple construction specifically, levelled, and compacted till base with a compacted ¾ inch minus gravel cover.

18.4 Main Power to Site

Emera Power was recently purchased by Enmax of Calgary, Canada to form the new corporation, Versant Power (https://www.emeramaine.com/). Previously, Emera had indicated that the price of delivering 6 MVA to the Mine site would be $7.0 million.

18.4.1 Power Distribution

Power distribution on site was quoted as $2.0 million. This allowed for step down from the supplied Enmax line to 4160V distribution on site. Distribution will be via overhead lines on the surface with some underground tech cable near buildings and leading to underground workings.

18.5 Potable Water System

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.

The water needs to be drawn from an authorised site by the State of Maine to a suitable tank and from the tank, distributed after being treated for organics, total dissolved solids, as well as metal ions.

The price of the system is approximately $250,000.

18.6 Sewage System

The two main factors that dictate the size and complexity of a commercial septic system are the maximum amount of wastewater that the buildings could produce daily, and soil/site conditions. Some domestic sewage systems today may require a separation of grey water and black water to ensure proper operation of the overall process in turning the waste products into environmentally safe materials. The assumption here is to forgo this issue and design a system for 2,500 gallons per day.

The price of the system is approximately $250,000.

18.7 Buildings Earthworks

Once the main area has been established, the construction contractors will require excavation of the various buildings, including the mill, the electrical substation, offices, and other outer buildings, including warehouse bunks.

The price of the system is approximately $750,000.

18.8 Quarry

Several quarry and road construction contractors were contacted and the budget quote received for crushed rock was $15 per ton FOB plant plus $12 per ton delivered to site for a total price of $27 per ton.

The alternative option is to construct a quarry on site and operate it at approximately $5 per ton and utilise the crusher system to produce the products needed for surface and underground. This latter idea has been shown inside the Tailings Area in Figure 18.4. This location being conceptual will require evaluation of depth to water table and material quality prior to excavating.

18.9 Fuel and 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. Monies allocated for these items are $25,000.

18.10 Propane

Propane needs special attention as the contents of the tank are under pressure and protection of a potential explosion is needed. An extra $10,000 has been allocated to accommodate a berm around the tank making the total cost of containment $35,000.

18.11 Change House (Dry Facility)

The building is sized for worker and staff personnel (underground and surface) to change out of their street clothes for work gear. The facility will have a street clothes locker on one side, a shower/washroom facility in the middle, and a work gear PPE on the other side. Monies allocated are $750,000.

18.12 Materials Pads

Whatever the pads are holding, whether it be potentially economic mineralisation, waste rock, acid generating waste rock or backfill, each material storage pad will require construction and lining. Lining of acid generating or potentially acid generating pads will require a minimum of 2 feet this of clay or silt covered by a geosynthetic liner and a protective layer. Non-acid generating pad simply require a clay or till base and a working layer.

18.12.1 Rock Dump Construction (Clean)

Berms and drainage systems containing water and preventing seepage are designed to handle two and one half years of development.

The pad size is 300×35×15m and will cost $125,000 to construct.

18.12.2 Rock Dump (Acid Generating)

The acid generating rock pad requires significantly more attention to detail regarding drainage, collection, and potentially pumping to a holding pond for further treatment. Lining the pad with clay and a geosynthetic liner then a gradational covering over the membrane ensures no leakage to the environment. The effluent will be directed to a holding pond for further treatment.

The pad costs are designated at $400,000 and 150m square by 2m thick.

18.12.3 Ore Pad and Temporary Stockpile

The stockpiles of ore and development will have a similar design as the acid generating pad and will similarly cost $400,000.

18.13 Office Maintenance Complex

The offices and maintenance complex are constructed of dry-storage shipping containers to speed construction and to minimise foundation work. The compressor room will be in one of the units and the stand-by generator in another. Finally, there are office and maintenance areas throughout the containers.

Costs set aside for these buildings are $750,000.

18.14 Surface Explosives Storage

Surface storage of explosives are set at $25,000 and will be rented from the explosives supplier until underground storages can be established.

18.15 Accommodations

No onsite accommodations have been considered in this study. Employees and contractors will commute to the site.

18.16 Water Management System

Water management will be a series of collection ditches and pond used to collect impacted water from around the property outside of the tailings facility. Water drawn from the tailings facility will be recirculated 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 reused in 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. There will be no single point discharge of any water to into any natural creeks, streams or bodies of water.

18.17 Water Treatment Facility

Collected water will be mixed using a series of frac tanks with hydroxide and Metclear (metal precipitating agents) to form a metal precipitate. The metal precipitate will be captured by an ultrafiltration membrane (filter) for disposal and the decant solution will return to the frac tanks to be reused in the process. The ultrafiltration permeate (clear water) will either be discharged (if it meets State environmental standards) or flow through a reverse osmosis unit, if required (TBD). Reverse osmosis permeate will be either discharged or reused as mine process water. Reverse osmosis concentrate will flow to the storage tank for decant and solids removal. This report considers a Build Own Operate option from the supplier and includes the reverse osmosis unit as a conservative approach. The water treatment facility will be owned before the operation is closed and may be used in reclamation and longer term standby post closure treatment, if ever required. The block diagram below (Figure 18.3) shows the anticipated design of the water treatment facility.

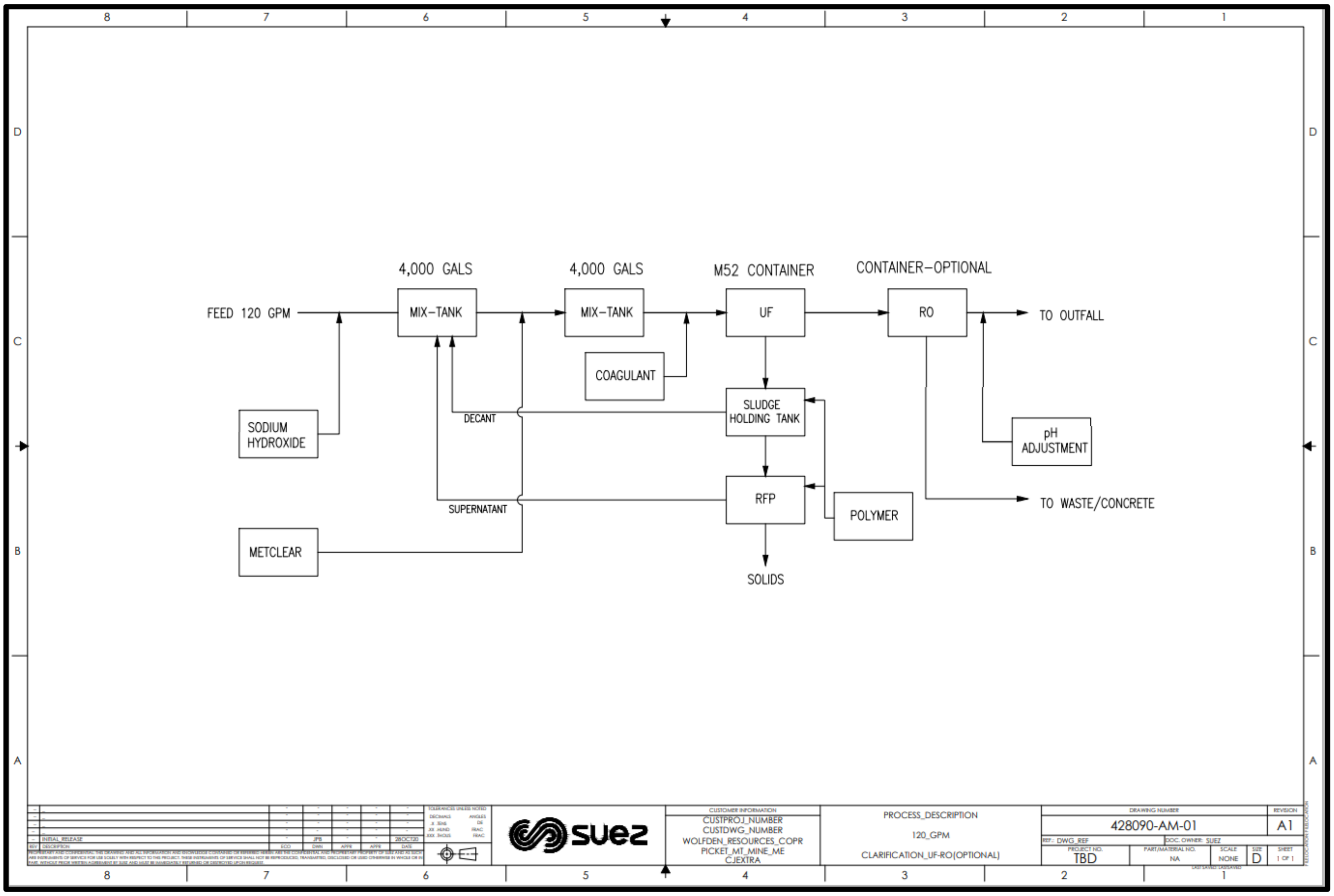


Figure 18.3 Water Treatment Plant Block Diagram

18.18 Garbage

Domestic garbage will be collected and taken to the nearest approved landfill site and managed by a local contract.

18.19 Standby Power Plant

An estimate of 3 MVA will be required to ensure safety of the operation during a power disruption. The standard method of providing this power is through diesel generators.

18.20 Communications

There will be a requirement for high speed intranet on site and potentially connecting to the outside internet. Pricing is to be determined but covered under contingency at this time.

18.21 Security

Security of all industrial sites are required and fencing is required around all hazardous points as well as gates for entry onto the property (Figure 18.4).

Figure 18.4 Plan Showing General Arrangement of Surface Infrastructure

18.22 Tailings Management Facility

Tailings will be dewatered by thickening followed by pressure filtration to produce a filter cake consistency tailings that will be deposited in a filter stack (Dry Stack) Tailings Management Facility (TSF) in compliance with State Regulations (Maine, 2017). The regulations allow for amendments, if tailings alternatives may be demonstrated to be equivalent; however, a filter cake TSF is well suited for this project over alternative methods because:

- Reduced footprint allows for full containment of tailings within limited space available;
- Soil-like tailings consistency allows for steep deposition slopes that are ideal for the hillside topography of the TSF site;
- Lack of natural containment mean hydraulically placed tailings would require significant containment dams;
- Water recovered in the mill obviates the need for large tailings ponds for mill make-up water, especially during winter months when significant amount of water lost to ice;
- The mill process rate allows for filtered tailings to be handled and deposited using minimal resources;
- The facility is readily expandable by stacking higher or expanding the footprint without significant dam construction;
- Amenable to progressive reclamation and immediate closure as the filtered tailings do not require drying prior to installation of closure cover; and
- Reduced risk of containment loss result of low water content tailings.

The main risks to successfully developing a filter cake TMF that will need to be carefully managed are:

- Off-spec tailings that are not sufficiently dewatered by filtration will require additional rework, drying, or re-processing before they can be deposited; and
- The design deposited dry density may not be achievable during the winter months and may require temporary storage until spring when the thawed tailings may be compacted.

18.22.1 Operating Data

Approximately 4.1 Mt/4.5 M short tons (M-st) of mineralised material will be mined and processed over 9 years (approximately 1.2 ktpd/1.32 kstpd) to produce three separate concentrate streams for copper, lead, and zinc. Approximately 20% of the mineralised material will be shipped offsite as concentrate, with the remaining 80% or 3.3 Mt/3.5 M-st of tailings stored on site.

Tailings will be dewatered by thickening to approximately 60-63 wt% solids followed by pressure filtration to approximately 83-85% wt% solids and having between 15%-17% moisture calculated as process engineering (Mine Paste, 2020). Based on a specific gravity of 3.25 and a void ratio of 0.6, SLR estimated the filtered tailings density to be 2.0 tonnes per cubic metre (2.1 t/m^3)/125 pounds per cubic foot (pcf) with a corresponding TMF storage capacity of 1.6 million m^3.

18.22.2 Design Criteria

Key Project design criteria, assumed for the TMF, are summarised as follows:

- **Life of Mine** (Commercial Production): 9 years
- **Mined Tonnes**: 4,100,000 tonnes (4,520,000 tons)
- **Tailings Produced**: 3,280,000 tonnes (3,620,000 tons)
- No Underground Backfill
- **Dam Classification**: Significant (CDA, 2013)
- **Environmental Design Flood**: 500-year 24-hour storm (198 mm/7.8 inches)
- **Inflow Design Flood**: 1,000-year 24-hour storm (217 mm/8.5 in)
- **Contact Water Containment**: composite liner system with solution collection system
- **Contact Water Drainage**: maximum 0.3m/1 ft head above containment system liner
- **Seepage Control Measures**: collection pond with water reused by mine or treated and discharged
- **Closure Cover**: composite liner system with drainage layer and soil cover for vegetation growth

The contact water containment/drainage and closure cover requirements are per state regulations (Maine, 2017).

The Project constraints for the TMF are:

- **Target Maximum Height of TMF**: less than the height of trees 7m/22 ft
- **Wetlands Setback Distance**: 23m/75 ft minimum
- **Re-zoning Boundary Setback Distance**: 122m/400 ft minimum

The maximum height of the TMF was targeted to not exceed the height of trees by Wolfden to minimise the visibility of the TMF to the general public from the surrounding areas. However, this constraint is considered flexible with the design criteria and setback distance constraints taking precedence.

18.22.3 TMF Conceptual Design

The TMF is comprised of the following design components:

- Containment system, consisting of a composite liner system, to minimise the seepage to the environment;

- Perimeter berms to provide containment of the tailings;

- Collection Pond to store excess contact water;

- Collection Ditches to convey contact water to the Collection Pond; and

- Surface Water Ditches to convey fresh, or non-contact water, around the TSF.

The design and operation of a filter cake disposal facility is dependent on tailings to the specified consistency, *i.e.*, filtering to near optimum moisture content to allow for placement and compaction.

Additional rework of the tailings may be necessary to achieve the optimum moisture content and design dry density.

If adequate and consistent filtering cannot be achieved, the system may not work.

A site plan, showing the location of the TSF and appurtenant structures, is shown in Figure 18.5.

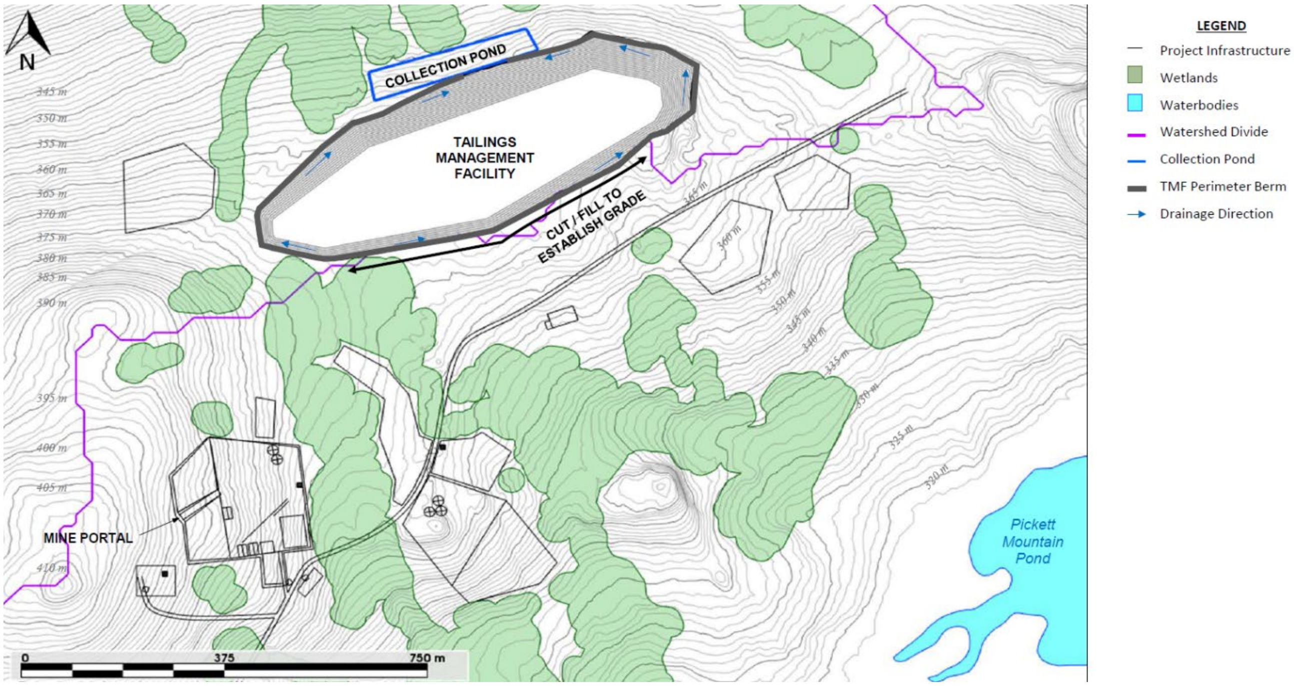


Figure 18.5 Plan Showing General Arrangement of Tailings Management Facility

18.22.4 Base Grade and Containment System

The base or foundation of the TMF will generally follow the natural topography of the ground surface, sloping from the topographic divide downwards to the Collection Pond in the north. The topographic ridge features two small crowns (approximately 2m/6 ft tall) that will need to be regraded to provide positive internal drainage to the Collection Pond on the north side of the TSF.

A containment system is required by State regulation (Maine, 2017) consisting of a composite liner and drainage layer. Contact water collected above the composite liner will gravity drain to the Collection Ditches and ultimately the Collection Pond. Contact water will be recirculated to the mill. Minimal contact water in excess of the mill requirement will be directed to the water treatment plant.

The containment system is comprised of the following components, from bottom to top:

- 0.6m/2 ft minimum thick low permeability soil fill (permeability less than 1×10^{-6} centimetres per second (cm/s));
- 1.5 mm/60 mil High Density Polyethylene (HDPE) geomembrane; and
- 0.6m/2 ft minimum thick Drainage Collection Layer.

The containment system is required to also ensure that the contact water head does not exceed 0.3m/1 ft above the HDPE liner. To satisfy this condition, a series of perforated, corrugated polyethylene (PCPE) drainage pipes will be installed within the free draining Drainage Collection Layer.

18.22.5 Perimeter Berm and Ditch

A 1m to 3m (3 ft to 9 ft) high perimeter berm will be provided along the toe of the TMF. The berm will be used for anchoring the geomembrane liner and for creating a Collection Ditch for contact water collection along the tailings perimeter. The height of the perimeter berm will be a function of the length of the tailings slope. The south side of the TSF is situated on flat ground and will have a minimum berm height of 1m (3 feet), while the north side of the TSF is situated on a slope and will have a maximum height of 3m (10 feet).

Filtered tailings will be placed up to the perimeter berm, maintaining a minimum 1m (3 ft) deep ditch between the filtered tailings and berm slopes. The grade of the ditch will follow the natural topography, sloping from north to south. Grading will be required on the topographic divide through a grading plan that ensures a minimum 1% slope that drains toward the east side then turning and draining downhill south to the Collection Pond. Borrow from the cut of the two crowns can be used to regrade the lower areas in between.

The perimeter berm will be notched at the low point in the TMF to allow drainage from the TMF to the Collection Pond.

18.22.6 Collection Pond

Contact water from precipitation and tailings seepage will drain into the perimeter Collection Ditch system, which ultimately drains to the Collection Pond on the north side of the TMF.

The Collection Pond was sized to contain a total of 43,000m^3, which is the run-off from an Environmental Design Flood (EDF) in addition to the maximum operating level. SLR sized the Collection Pond based on the following:

- SLR assumed a maximum operating pond volume of 5,000m^3 (18,000 cf) for a facility of this size and with progressive reclamation potentially reducing the quantity of contact water generated. The filtered tailings are relatively dry with an estimated gravimetric moisture content of about 18% which SLR expects to result in very little free water to drain out. Rainfall is expected to be the main source of contact water, and some infiltration is expected to report as tailings seepage.

- The EDF criterion for the Project is defined as the 500-yr 24-hr. event required by (Maine, 2017). SLR calculated an EDF volume of 38,000m^3 to be stored in the Collection Pond based on a the lined TMF footprint area of 215,000m^2, a 500-yr 24-hr. event of 198 mm and a run-off factor of 90%.

SLR assumed that the containment berms for the Collection Pond will be constructed by placing and compacting soils excavated from within the Collection Pond footprint. The Collection Pond will be constructed with a similar containment system as the TMF to prevent solution seeping into the groundwater, and is comprised of the following components, from bottom to top:

- 0.6m/2 ft minimum thick low permeability soil fill (permeability less than 1×10^{-6} cm/s)

- 1.5 mm/60 mil HDPE geomembrane

A spillway equipped with a rip rap protected channel and energy dissipation downstream of the south side of the pond will prevent overtopping and will be sized to safely pass the IDF event, defined as the 1,000-year 24-hour event. The water discharged by the spillway will drain overland north to Pleasant Lake. In the event of such a major storm, rain water would be pumped directly into the mine in order to prevent usage of the spillway.

18.22.7 Closure

At closure, the TMF is required to be fully covered with a geomembrane liner on the outer slopes and vegetated with similar plant species found naturally in the area. In the active phase of closure, water treatment and environmental monitoring will be required until it is demonstrated that the site conditions meet pre-mining conditions. As the site reaches passive phase of closure, no additional treatment or monitoring would be expected.

A closure cover, consisting of a composite liner equivalent to the basal liner, is required to be constructed over the TMF (Maine, 2017).

The closure cover liner system is comprised of the following components, from bottom to top:

- 0.6m/(2 ft thick low permeability soil fill (permeability less than 1×10^{-6} cm/s)
- 1.5 mm/60 mil HDPE geomembrane liner
- 0.3m/1 ft thick Drainage layer
- 0.3m/1 ft thick random soil for a root penetration layer,
- 0.15m/0.5 ft thick topsoil vegetation growth medium, revegetated with small grasses and shrubs

At closure, the contact water collected in the Collection Pond will continue to require treatment until water quality meets regulatory requirements. Once the water quality requirements have been met, the geomembrane may be removed and the Collection Pond breached to prevent future accumulation of water with the area covered with soil and vegetated.

Selective areas of the TMF can be progressively reclaimed during the operations phase by installing the closure cover system. Reclamation will be limited by the areas of the cell that are final and do not tie into an adjacent cell (*i.e.*, the internal boundaries of the cells should not be reclaimed). Progressive reclamation will allow the closure costs to be spread out into the operating period, allowing the closure construction to be reduced in cost and duration in addition to providing an opportunity to evaluate the closure cover performance.

19.0 Market Studies and Contracts

19.1 Market Studies

No market studies were conducted at this time. The pricing used for the various commodities was market consensus pricing, as provided by Wolfden.

The metals price deck used is in Table 19.1.

TABLE 19.1
METAL PRICE DECK

Commodity	Consensus Pricing	
Zinc	$	1.15
Copper	$	3.00
Lead	$	1.00
Gold	$	1,500.00
Silver	$	18.00

19.2 Contracts

The following contracts will be part of the construction and/or operation of the Pickett Mountain project:

- Zinc, lead, and copper concentrate offtake agreements are estimated with input from major smelters including a large, diversified resource conglomerate and commodity trader, for life of mine feed at International Benchmark terms.

- Pickett Mountain's initial underground construction, ramp up, and operation for up to 3 years will be conducted by mining contractors.

- Site infrastructure construction will be performed, in majority, by locally sourced contractors.

- Water treatment activities will be performed by specialised supplier and operator.

- Pickett Mountain concentrate material will be transported to the nearest deep water port via local logistics contractor.

- Construction, operations, supplies, and consumables contracts have not been established; however, estimates by service providers used in this document have been supplied by potential candidates.

No contracts currently exist for construction or operation of the project. The contracts listed above have submitted budgetary quotations and estimates for input to the study.

20.0 Environmental Studies, Permitting, and Potential Impacts

The proposed mine project is located on a 528.2-acre parcel within a 7,145-acre tract of privately owned land owned by Wolfden Mt. Chase LLC, a wholly owned subsidiary of Wolfden Resources Corporation. The proposed mine project is within Maine's unorganised territory and subject to regulatory approval of both the Maine Land Use Planning Commission (LUPC) and the Maine Department of Environmental Protection (MEDEP).

This item is organised as follows:

- ***Section 20.1, Regulatory Framework*** discusses the roles of both LUPC and the MEDEP in the permitting process for the Pickett Mountain Mine Project;

- ***Section 20.2 Permit Status*** summarises the status of the project with respect to permitting;

- ***Section 20.3 Environmental Studies and Issues*** provides a summary of the results of environmental studies completed, identifies additional studies that are expected to be required, and discusses known environmental issues that could materially impact the project's ability to extract the mineral resources or mineral reserves;

- ***Section 20.4 Social and Community Requirements*** provides a discussion of potential social or community related requirements and plans for the project and the status of negotiations or agreements with local communities; and

- ***Section 20.5 Mine Closure*** provides a discussion of mine closure including remediation and reclamation requirements and costs.

20.1 Regulatory Framework

20.1.1 Land Use Planning Commission (LUPC)

Within the regulatory framework of the State of Maine, the LUPC was commissioned and charged by the Maine Legislature with developing a comprehensive plan to extend principles of sound planning, zoning and development to the unorganised and de-organised townships of the State. Through its Comprehensive Land Use Plan (CLUP) the LUPC reviews, approves, and administers development projects ensuring that proposed projects meet the broad and specific goals of the Commission and objectives of the legislature. Specifically the legislatures purpose was "*to preserve public health, safety and general welfare; to support and encourage Maine's natural resource-based economy and strong environmental protections; to encourage appropriate residential, recreational, commercial and industrial land uses; to honor the rights and participation of residents and property owners in the unorganised and de-organised areas while recognising the unique value of these lands and waters to the State; to prevent residential, recreational, commercial and industrial uses detrimental to the long-term health, use and value of these areas and to Maine's natural resource-based economy; to discourage the intermixing of incompatible industrial, commercial, residential and recreational activities; to prevent the development in these areas of substandard structures or structures located unduly proximate to waters or roads; to prevent the despoliation, pollution and detrimental uses of the water in these areas; and to conserve ecological and natural values.*"

The purpose of LUPC's oversight in the context of metallic mineral mining projects is to certify that the project meets the goals and the objectives established in the CLUP and that the property can be rezoned to allow the project to move forward with a Mining Permit Application under the MEDEP Chapter 200 rules. The area proposed for the project is currently zoned as a general management subdistrict. The proposed project is a major planned development that must be conducted within a Planned Development (D-PD) subdistrict as required by the LUPC for metallic mineral mine projects, consistent with standards for said subdistricts and within the intent and provisions of 12 M.R.S.A. Chapter 206A. Under Chapter 685-B, Development Review and Approval, a permit is not required for metallic mineral mining projects that are reviewed under the Maine Metallic Mineral Mining Act. This project will require review and permitting by the MEDEP under its Chapter 200 rules for Metallic Mineral Exploration, Advanced Exploration and Mining, since all metallic mineral mining activity within a D-PD district is permitted through the MEDEP. The LUPC must certify to the MEDEP that the proposed development is an allowed use and that the proposed development meets applicable land use standards established by the LUPC and not otherwise considered as part of the MEDEP's review.

Pursuant to 12 M.R.S.A., Section 685-A(8-A), no change in a district boundary shall be approved by the Commission unless there is substantial evidence that:

- The change would be consistent with the standards for D-PD Development Subdistrict boundaries in effect at the time; the Comprehensive Land Use Plan; and the purpose, intent and provisions of 12 M.R.S.A. Chapter 206-A; and

- The change in districting will have no undue adverse impact on existing uses or resources or a new district designation is more appropriate for the protection and management of existing uses and resources within the affected area.

20.1.2 Maine Department of Environmental Protection (MEDEP)

The MEDEP is charged with permitting and regulation of metallic mineral mining projects under its Chapter 200 rules. The Maine Metallic Mineral Mining Act (Act) provides the framework for all metallic mining activity within the state. The current statute became effective on June 1, 2014. The Maine Board of Environmental Protection, on January 5, 2017, unanimously voted to repeal the existing Chapter 200 rules and adopt new rules. In June 2017, an Amendment Bill LD-820 to the Act, "To Protect Maine's Clean Water and Taxpayers from Mining Pollution" became law, which provides additional provisions and restrictions. The Chapter 200 Rules under the MEDEP were repealed and replaced with the current rules on December 28, 2017.

The intent of the 2014 statue was to streamline the existing permitting system and incorporate many of the permitting requirements under one regulatory agency, the MEDEP. Under the Act, permits that were previously required under other state law are no longer required in that those provisions are covered directly in the new metallic mining permit program.

The rules and recently enacted legislation effectively require best-in-class environmental protection technologies and practices, as well as financial assurance provisions for site closure.

Pickett Mountain is the first project in Maine to apply for permits under this regulatory scheme.

20.2 Mine Permitting Stages and Status

Wolfden initiated discussions with the Maine LUPC in 2019 to formulate a plan and a schedule to submit a Petition to Rezone a portion of its land (528.2 acres) from a General Management subdistrict to a Planned Development (D-PD) subdistrict. A draft petition was submitted to the LUPC on January 27, 2020.

The petition was submitted with the understanding that certain specific field studies would be initiated in the Spring of 2020 and provided when completed. The Petition is listed as LUPC Zoning Permit 779 – Wolfden Mt. Chase, LLC.

These studies included reconnaissance level and detailed mapping (delineation) of vernal pools and wetlands as well as Phase 0 archaeological survey. Results of these studies have been submitted to the LUPC together with a revised application on June 12, 2017 that addressed comments and requests for additional information provided to Wolfden in letters from the LUPC dated March 6, 2020, April 15, 2020, and May 27, 2020. Pursuant to additional communication by electronic mail and telephone, Wolfden has also agreed to conduct a soil suitability study, by a Maine certified soil scientist, where the mine development facilities are proposed. The soils field work was conducted in September/October 2020. Other requested information, including this PEA, have resulted from this correspondence between LUPC and Wolfden. As part of the application review process, LUPC may make additional information requests. Once all additional information requests have been addressed, Wolfden would provide a final petition. On July 27, 2020, LUPC accepted Wolfden's petition as complete for processing and review. However, subsequent to this date, LUPC has requested clarification and additional information for several areas of the petition. Wolfden is actively addressing these additional requests.

The petition contains information on 21 specific information questions concerning the project and is supported by 15 Exhibits and one Appendix to specifically address LUPC Chapter 12 Requirements for Mining and Level C Mineral Exploration Activities.

The petition provides information on project location, description, site conditions, and current use of the property, local public and community services, surrounding land uses, and anticipated impacts, consistency of the project with the CLUP including development goals, natural and cultural resources, as well as financial capacity and corporate standing of the applicant. Key areas of assessment include potential for project impacts on public services, local recreational resources, scenic resources, rare or special plant communities, and wildlife habitats including wetlands, lakes and ponds, and suitability of site soils for intended uses.

The LUPC will follow the following general steps in its review:

1. Review to determine if petition is complete and acceptable for processing – Completed
2. Petition will be reviewed by Commission staff once accepted for processing
3. Initiate Public Hearing Process once Commission review is complete
4. Closure of the hearing record
5. Commission deliberation and decision

On May 28, 2020, the LUPC posted the following petition review schedule (Figure 20.1):

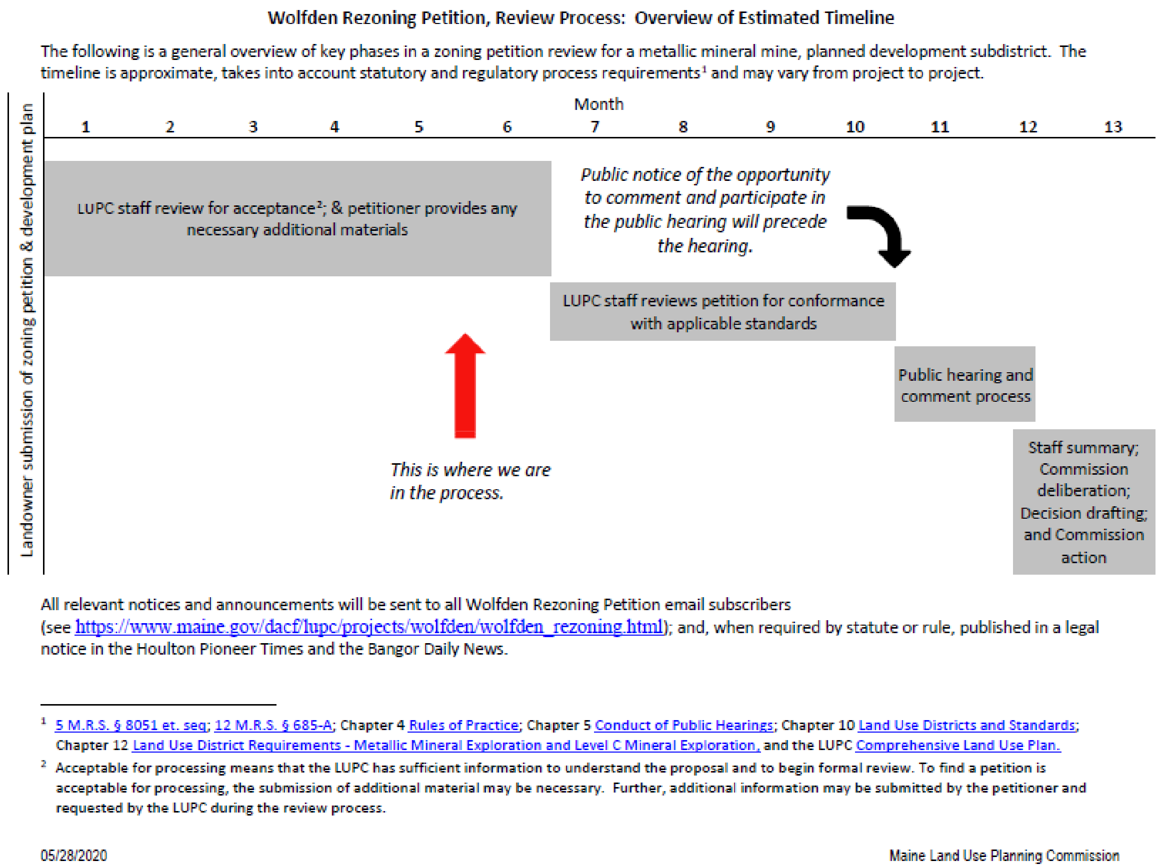


Figure 20.1 LUPC Posted Following Petition Review Schedule

The application process with the MEDEP under the Chapter 200 rules has not been formally initiated. This project will be the first project under the new MEDEP mining regulations to seek a permit. While the mining regulations have not detailed any specific requirements that need to be meet, there has been no precedent set for how the regulations will be applied. Under the current regulatory framework, Wolfden can pursue a concurrent application process rather than a sequential process to reduce permitting timeframes. This, of course, is at the applicant's sole risk. Wolfden may prepare a Baseline Characterisation and Quality Assurance Plan (Baseline Work Plan) for submittal to MEDEP to allow it to proceed with required two years of baseline water quality monitoring while the LUPC deliberates on the Rezoning Petition.

The MEDEP Chapter 200 regulations will require the following submittals and notification steps and public involvement as part of the mine application process:

PRE-APPLICATION PHASE

- Pre-Application Meeting (including required Pre-application information and reports)
- Publication and Notice of Baseline Work Plan
- Preparation of Environmental Impact Assessment Scoping Document

APPLICATION CONTENTS

- Baseline Site Characterisation Report
- Mining Operation Plan
- Engineering Report
- Quality Assurance Plan
- Environmental Impact Assessment
- Alternatives Analysis
- Mine Plan
- Monitoring Plan
- Contingency Plan.
- Financial Assurance

APPLICATION PHASE

- Advanced Notice of Intent to File
- Notice of Intent to File Applications
- Adjudicatory Hearings
- Intervenor Status
- Assistance Grants for Municipal and County Intervenors (Applicant Supported)
- Access to the Site by Intervenors
- Public Information Website (Applicant Financially Responsible)

The Chapter 200 rules also specify a lengthy list of criteria for permit approval and the term of the mining permit has conditions. Once approved a permit will remain in effect for the life of the mining operation and reclamation period so long as mining commences with 4 years of issuance. The MEDEP will conduct annual reviews of the mining operations and assess compliance with the permit terms. If the MEDEP determines at any time that the Permittee is in noncompliance with the Act, rules, permit, or order, and determines that the violation is causing an imminent and substantial endangerment, the Commissioner may issue an order requiring that the Permittee cease mining for metallic metals, and cease the removal of metallic metals from the site until the compliance issues are corrected to the Department's satisfaction.

If the Permittee has not conducted mining activities covered by the mining permit within 4 years after the effective date of the mining permit, the MEDEP may terminate or request surrender of the mining permit, after public notice. The Permittee may request MEDEP approval of an extension of time to conduct mining activities covered by the mining permit prior to Department ordered termination or surrender the permit. The Department may approve an extension of time to commence construction of mining facilities or conduct mining activities if the Permittee demonstrates that the mining operations are expected to commence within a reasonable time period.

The Chapter 200 rules have specific requirements for the type of mining that can be conducted and the manner in which mining residuals such as tails are managed. Specifically, the rules only allow underground mining methods and require tailings disposal as dry stacked tailings in lined facilities to be closed with a final cover of equal hydraulic performance. Technically the project can meet these two requirements. The other technical requirements of the Chapter 200 rules are too numerous to summarise here. Review of the many requirements for mine design, mine operation, mine closure, water collection and treatment, and reclamation and environmental monitoring, did not identify technical or operational requirements that could not be met by a well-designed and responsibly managed project.

20.3 Environmental Studies and Impact Assessments

Both desktop and field environmental studies have been conducted or are in progress in conjunction with the LUPC submittal.

Desktop studies generally included compiling and analysing existing information on the following areas and topics:

- Surficial and bedrock geology and hydrogeology
- Site soils and suitability for intended uses
- Location of significant mapped sand and gravel aquifers and groundwater resources
- Lake survey maps and fish management programs
- Recreational resources within 3 miles of the Site
- Viewshed and visibility of Site within 3 miles (Impacts to scenic resources)
- Noise
- Correspondence with the Maine Natural Areas Program for rare and exemplary botanical features in proximity to the Pickett Mountain Project. Based on current information and reconnaissance, rare and exemplary botanical features (rare, threatened, or endangered plants) have not been observed or are not known or expected to exist in the area proposed for rezoning.
- Correspondence with the Maine Department of Inland Fish and Wildlife (IF&W) for potential for significant wildlife, endangered, threatened of special concern species within the project area. The IF&W reported it has not mapped any significant wildlife habitats within the project area. The IF&W did identify Great Blue Heron colonies as possible species of concern, if they are found to be present on the site and noted the special protection afforded to eight species of bats and concern for habitat protection.
- Correspondence with the Maine State Historic Preservation Office identified a stone tool archeological habitation site near the headwater of Pickett Pond (Site 147.001).

Synopsis - Based on initial desktop studies and subsequent detailed field studies (wetlands/vernal pools and archaeological resources) provided in the LUPC petition, the applicant concluded the project would not have an undue or unacceptable impact on the types of natural resources that are under the jurisdiction and purview of the LUPC. The petition, overall, concluded the proposed project would meet the goals and specific requirements of the LUPC and could be completed in a manner that would fit harmoniously within the surrounding area. There are certain aspects of the project that the LUPC has requested more detailed studies or additional information. These include but not limited more information on soil suitability, noise, scenic, and economic impact. Field studies provided or that are underway in 2020 are described below.

Field Studies have been conducted in the following areas:

Archaeology - A Phase 0 Archaeological Assessment was conducted in June 2020 in accordance with direction from Maine State Historic Preservation Office. The assessment included background research and a field inspection. Background research considered various 19th and 20th Century maps of the area, contemporary topographic and bedrock/surficial geological maps, and review of MHPC site files associated with previously identified site 147.001. These resources confirmed the potential presence of tool-stone geological resources near the project area, possibly including chert and fine-grained volcanics. A more detailed Phase 1 survey could be recommended when Wolfden continues to the next phase of background study test work and permitting for the MEDEP.

Wetlands and Vernal Pools - Vernal pools were identified and mapped over a two-week period during amphibian breeding season in May/June 2020. Egg mass counts (wood frog and spotted salamander) did not indicate that the vernal pools identified would be considered significant vernal pools under State of Maine criteria. Wetland delineation and reconnaissance was completed over the 528-acre parcel in June 2020, and included identification of flowing and potentially intermittent streams. Several large and smaller isolated wetlands were mapped. Large contiguous upland areas are also present. The location and sizes of areas required to construct and operate the mining project were conceptually laid out by others. Site designs and layouts layouts have been located such that no wetlands, vernal pools, or stream channels, including a 75-foot setback on each, would be impacted by site development.

The following additional field studies are currently underway to provide additional details and clarity for the LUPC petition:

Soils - A survey to characterise the suitability of soils for the intended uses on the property is underway and will be completed in the fall of 2020. This work will be conducted by a Maine certified soil scientist with support from a Maine licensed civil engineer. Collectively, an assessment of soil suitability will be conducted to respond to an information request by the LUPC. The primary areas of interest will be areas proposed for septic systems, subsurface re-infiltration of treated water (PDPs), the plant site (ore pads, rock stockpiles, and building locations), roads, and the tailings management facility. If soils are found to be unsuitable, alternative engineering and construction approaches will be evaluated and presented to overcome these limitations.

Noise Study – At the request of the LUPC, additional noise modeling studies are being conducted to supplement initial calculations and to address information requests submitted to the LUPC. The initial calculations in the petition suggested that noise will be within permissible levels required under both LUPC and MEDEP rules and guidance.

The following additional field studies will need to be provided during the MEDEP Baseline Study:

Aquatic Species - A qualified professional aquatic biologist will need to conduct a survey of aquatic and semi-aquatic species present along the inlet and outlet streams to Pickett Pond and the West Branch of the Mattawamkeag River to the inlet of Pleasant Lake. Biological sampling is discussed below. The aquatic survey will include documentation of observations concerning presence, absence, and relative abundance of aquatic species including but not limited to invertebrates, fish, amphibians, reptiles (turtles), and freshwater mussels.

The aquatic species survey will also include fish populations in Pickett Pond, Pleasant Lake, and Grass Pond. The fish population survey will be conducted by manual methods or electrofishing (removal), if necessary. The objective will be to survey the types of different species within the shallow ponds, potential for spring holes to support the colder water species and gauge the importance of feeder streams as nursery habitats for the cold-water species. Based on findings of the initial surveys, analyses of fish tissue may be required.

Aquatic Invertebrates - Aquatic invertebrate (macroinvertebrate) species will be assessed using the biological sampling methods and procedures presented in the revised MEDEP guidance for Biological Sampling and Analysis of Maine Rivers and Streams (MEDEP, 2002). Based on the initial aquatic survey, locations for rock baskets or rock bags (depending on water depth and flow) will be selected within inlet and outlet streams to Pickett Mountain Pond and the inlet stream to Pleasant Lake. The biological sampling will consist of one event conducted between July 1 and September 30, 2021, when low streamflow conditions occur, and the biological community is under greatest stress.

Flora - A botanical meander survey will be conducted within the area proposed for development for Rare, Threatened, and Endangered (RTE) species by a qualified botanist. In consultation with MNAP, the botanist shall develop a target list of plant species and exemplary botanical features as the primary focus for the survey.

Terrestrial Fauna - A terrestrial fauna survey to determine species present, distribution, and abundance including the existence of RTE species and presence or absence of significant wildlife habitats. The fauna survey will include habitat assessment for Great Blue Heron and Rusty Blackbird, as state listed species of concern. These surveys will need to be completed prior to June 15th, 2021, the close date for these species.

A species assessment encompassing a meander survey of the area proposed for development for individual species and or suitable habitat for the species identified will be conducted by a wildlife biologist. Habitat assessments will also include a desktop study of preferred prey species habitat (snowshoe hare) for the Canadian Lynx.

Ambient Water Quality Monitoring - Baseline water quality monitoring will include at least 2 years of data collected over 24 months to generate the information on surface water and sediment chemical characteristics, stream discharge and flows, groundwater quality, and groundwater elevations. This work will require collection of surface water and sediment samples from streams and ponds for chemical analysis. This work will require installation and sampling of both overburden and bedrock groundwater monitoring wells to establish baseline groundwater quality and characterise groundwater hydrogeologic conditions. These data will be used to support development of a numerical groundwater flow model that is a required element in the baseline report.

Site Visibility and Scenic Impacts - Additional assessment of scenic impacts is expected under the MEDEP application. Initial viewshed models and assessment of site visibility indicate that scenic impacts should be limited to the immediate hill tops, which are largely within Wolfden owned land. The geomorphology and the site topography, extent of typical tree height and forest cover is favourable providing a visual screen of the project from most vantage points within a three-mile radius of the site. The site will not be visible from any for miles from any State park or publicly maintained trails or sites.

Noise Modeling – Additional noise modeling studies may be needed to supplement the Noise Study (discussed above) provided to LUPC.

20.4 Social and Community Requirements

There are no specific social or community requirements. The LUPC petition included a required assessment of "Socioeconomic Impacts of Proposed Mining Upon Immediate Area, Adjacent Communities County and State." This assessment developed data on the local economy and workforce, locally provided markets and services including:

- Housing;
- Education;
- Solid waste disposal;
- Emergency services (fire and ambulance);
- Healthcare and medical;
- Public safety, and
- Power and utilities.

The analysis indicated there is adequate capacity to provide these services to the extent needed for the proposed mine project and for anticipated employees who are largely expected to be employed and trained from the existing labour pool.

The demographics of the current population and employment statistics included the following towns and communities (Table 20.1):

TABLE 20.1
DEMOGRAPHICS OF THE CURRENT POPULATION AND EMPLOYMENT STATISTICS

Communities in Proximity to the Site	
Within 20 Miles	Within 20-30 Miles
Crystal	Central Aroostook
Dyer Brook	Glenwood
Island Falls	Hammond
Moro, Merrill	Haynesville
Mount Chase	Hodgdon
Oakfield	Houlton
Patten	Linneas
Sherman	Littleton
Smyrna	Ludlow
South Aroostook	Masardis
Staceyville	Medway
	Monticello
	New Limerick
	North Penobscott
	Orient
	Oxbow
	Reed

The regional public schools that served these communities (Table 20.2):

TABLE 20.2
REGIONAL PUBLIC SCHOOLS

School Year	SAU Name	School Name	Number of Students
2019	RSU 29/MSAD 29	Houlton Elementary School	423
2019	RSU 29/MSAD 29	Houlton High School	364
2019	RSU 29/MSAD 29	Houlton Junior High School	298
2019	RSU 29/MSAD 29	Houlton Southside School	319
2019	RSU 50	So Aroostook School	358
2019	RSU 70/MSAD 70	Hodgdon Middle/High School	191
2019	RSU 70/MSAD 70	Mill Pond School	284
2019	RSU 89	Katahdin Elementary School	145
2019	RSU 89	Katahdin Middle/High School	181
Total			2,563

Wolfden correspondence with the principal towns in the closest proximity to the Site resulted in these communities providing concurrence letters that the project would not pose an undue burden on services provided by these communities and that other purchased services (solid waste, communications, power) had adequate capacity.

20.5 Mine Closure

The mine (Dry Tailings Facility, Potentially Economic Mineral, Staging Area, Interim Waste Rock Storage Facility, Surface Water Management Facility) will be constructed in a manner to capture contaminated leachate and surface water run off for collection, treatment, and management.

At Pickett Mountain, there will be 3 classes of structures.

Class 1 is a permanently fixed structure that will remain post-closure of the property. Specifically, this will be the dry stacked Tailings Management Facility. The final design of the final cover system will be designed to meet the performance requirements of the MEDEP Chapter 200 rules, providing a performance of equal to or better than the liner system, so that leachate will not be generated once closed.

Class 2 is a non-permanent structure that is deemed acceptable to decommission and remove only after the site has been deemed ready for rezoning back to a General Management (M-GN) Subdistrict. Specifically, this will be the water management and water treatment facilities including all drainage and water collection structures.

Class 3 is a non-permanent structure that is decommissioned and removed as soon as production operations cease. Specifically, this includes all buildings on site that are not related to water collection and treatment, potentially economic mineralisation and waste rock storage pads, and non-essential roadways.

Upon completion of mining and processing of material from the Pickett Mountain mineral deposit, all Class 3 structures will immediately be decommissioned and sold, or, to the extent practical, demolished and deconstructed to allow inert materials to be placed in remaining open underground workings (raises and drifts). The land surface will then be contoured and smoothed to reasonably match the original landscaping. This closure work will be conducted under an approved erosion and sedimentation control plan and reclamation plan under the MEDEP Permit. Material from the overburden storage areas (original soils stripped prior to mine construction) will be placed on top of the regraded surface as final soil cover to support natural growth of vegetation. Openings to surface from underground that are non-essential will be plugged and capped with engineered concrete or steel plugs to ensure future access cannot happen, either purposefully or not. All precipitation that contacts these locations will continue to be collected and monitored for water quality and treated before being discharged. After removal of all Class 3 structures, it is anticipated that water quality of run-off being collected and treated will already begin to improve.

Class 1 structures will remain in place into perpetuity. Concurrently with the placement of tailings on the Tailings Management Facility (TMF), the facility will be reclaimed through progressive capping and revegetating. Therefore, the final reclamation will be to cover the TMF with an engineered composite cap derived from soil material components provided from local borrow sources. After it is capped and contoured to support precipitation drainage, the TMF will be covered with a final vegetative soil layer using the remaining material from the overburden storage areas. This will support regrowth of natural vegetation and long-term, permanent erosion control. Precipitation that falls on the TMF will drain off around the perimeter of the facility. The restoration design will include appropriately sized and constructed drainage features to handle storm events, consistent with DEP's stormwater management requirements. With all the Class 1 and Class 3 structures being closed or removed, the remaining site features will not adversely impact the water quality of run-off that is being collected and treated prior to discharge. After roughly 1-year post-complete closure, it is anticipated that the drainage water from the site will be back to historical quality and no longer require treatment. After this has been confirmed, Wolfden will decommission and remove the water management facility. The water management facility will be excavated and inert material (demolition debris) placed underground and the area recontoured. A final engineered plug will be placed in the portal area to completely and permanently block access to any underground workings.

Once final reclamation work is completed, continued post-closure monitoring of surface water and groundwater will take place for a duration that is specified in the DEP mining permit. Within the first year, samples will be taken frequently, following the sampling requirements established for operating the property. The frequency of monitoring and duration will be established statistically based on water quality trends and data.

The property will then be rezoned. Land use restrictions and deed covenants will be instituted over land occupied by the tailings facility to ensure that no industrial or commercial activity occurs over that portion of the site post closure.

Beneficial re-use of the property will include timber harvesting, as it occurs presently outside the tailings facility footprint. Also, the portal will be closed in a manner that will allow entry underground to bats, providing valuable habitat. Recreational uses will be allowed on the property, including hunting, hiking, ATV riding, etc. Restriction would be placed on the tailings facility in order to protect that area from damage by off road vehicles. In order to ensure protection of the tailings facility area, a series of permanent signs will be posted around the perimeter restricting access to authorised personnel only. In addition, if any future transfer of land ownership were to take place, the deed would restrict the use of heavy equipment or any small vehicles and recreational vehicles within the tailings area, to ensure that damage to the tailings cover is mitigated.

The Financial Assurance Trust fund required by MEDEP is included as capital cost in year -2. Costs include cost to cap and close the remaining open TMF facility cell (only one of five cells would open at any time), inspection and maintenance of the TMF facility, facility-wide post-closure monitoring including groundwater monitoring wells, surface water monitoring locations, resampling of baseline soil and sediment locations to document post closure conditions, and allow comparison of pre-mining and post mining chemical data for these media. A worse case failure scenario includes construction of a new Water

Treatment Facility and a groundwater pump and treatment system, as well as site monitoring for 100 years post closure. The total cost for the Financial Assurance Trust used in the PEA is $13,684,557. This considers a present value from future costs at a discount rate of 2% based on standard federal rates and does not include any salvage value for assets at closure. As the project progresses through its life, this financial assurance trust fund is anticipated to be reviewed and may be modified to reflect the ongoing and anticipated remediation requirements of the project during operations based on the results of the sequential closure of each cell of the tailings facility.

The reclamation and closure of the five TMF cells will occur as periodic events during the mine operation. It assumed that a new tailings cell will be opened every 2 years and the existing cell will be capped and closed. Cost for these elements were developed by SLR and are listed under TMF capital costs in Section 21.1.10 and are carried as a separate, stand alone cost in the cashflow model. The cost for the closure of the last remaining TMF cell is $1,226,081 (SLR). The cost for building demolition, remaining mine backfill, and site grading/re-vegetation was developed by AMPL, and is estimated at $2,200,000, $600,000, and $978,393, respectively. The total reclamation and closure cost (approximately $5,000,000) for these elements will be paid for through salvage income following the cessation of mining. Salvage income is expected to be in the range of $5,200,000 or 15% of the construction cost. The income for the salvage component of mine closure and reclamation has not been included as part of the financial analysis in this PEA, but should be factored in in the next phase – preliminary feasibility study.

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. Much of the pricing used was from supplier quoted prices obtained within the last 3 years from other projects. Capital expenditure 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 2020 constant US 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;
- 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 construction and installation costs.

Mine infrastructure costs for facilities, such as maintenance shops, mine dewatering, refuge stations, etc., were developed based on the conceptual plans and general arrangements presented earlier. Wherever possible, equipment and material quotes and contractor 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 local US or Canadian suppliers.

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

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

Where possible, direct costs are based on actual takeoffs:

- Earthwork/site work;
- Concrete;
- Structural steel;
- Buildings and architectural;
- Electrical;
- Instrumentation and controls; and
- Piping.

Commodity pricing for earthwork, concrete, steel, architectural, and piping were based on local Maine costs (in some cases escalated to 2020 costs from past costs). Labour rates and equipment usage rates used throughout the estimate were provided by mining contractors and other sources. Commodity prices were from local suppliers, where available.

Pricing was obtained from local contractors for rock excavation and transport during the pre-production stage.

Labour rates generally reflect U.S. contractor rates as well as industry rates obtained from a University of Virginia Tech 2016 Study. The mine labour costs are based on four types of estimates:

- Owner operator expected labour rates.
- 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 rates, developed and 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 cover all the costs associated with temporary construction facilities and services, construction support, freight, vendor representatives, spare parts, initial fills and inventory, Owner's costs, 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 were either provided by the vendor or estimated based on weights and typically include containerised and break-bulk shipping, and each are respectively divided into ocean freight and inland freight. For imported equipment, the cost of freight and export packing, ex-works to the nearest port, is included with the cost of the equipment. Freight insurance is included in the Owner's cost.

Vendor Representatives – The requirement for the vendor representatives to supervise the installation of equipment or to conduct a checkout of the equipment prior to start-up of the equipment, as deemed necessary for equipment guarantees or warranties, has been included in the estimate. Typically, the cost for this item is inclusive of salary and travel.

Taxes and Duties – Taxes and duties have been excluded.

Engineering, Procurement, and Construction Management (EPCM) – EPCM has been calculated based on the Pickett Mountain Project team managing development and construction, and using consultants where deemed appropriate by Wolfden.

Capital Cost Qualifications and Exclusions – All surface 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 2020 constant US Dollars.

21.1.4 Underground Mining

Underground capital cost estimates are based on quote pricing from suppliers, consultants, and contractors, provided with reasonable detail 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.

Most mobile mining equipment is leased by Wolfden. Alimaks, hand-held drills, and light service vehicles will be purchased by Wolfden.

The mine pre-production capital expenditures are estimated to total $54.0 million including a 20% contingency. The breakdown of the mine capital expenditures is presented in Table 21.1.

TABLE 21.1
BREAKDOWN OF MINE CAPITAL EXPENDITURES

Component	Expenditures ($millions)		Total Expenditures ($ millions)
	Year -2	Year -1	
Mine Development	6.75	14.69	21.44
Underground Infrastructure	3.21	6.95	10.16
Surface Infrastructure	10.00	10.11	20.11
Mining Equipment	0.99	0.99	1.98
Contingency	-	-	10.74
			64.42

The initial capital expenditure for the underground mine will include the collaring of an access portal and development of a decline down to an elevation of 300m below the surface. From the ramp, production levels will be established on the 5350, 5250, and 5150 levels. Stopes will be developed for production mining with the excavation of Alimak raises in ore from one level up to the overlying level.

The mine development will also include the development of a ventilation raise, 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 for the Pickett Mountain Mine is expected to be 21 months and will be conducted simultaneously with plant construction, infrastructure investments, and mine site preparatory work.

The mine development capital expenditures and underground mine infrastructure capital expenditure estimates are shown in Table 21.2 and Table 21.3, respectively. Capital Expenditures for equipment purchases and leasing total $20.7 million dollars including a 20% contingency.

TABLE 21.2
MINE DEVELOPMENT CAPITAL EXPENDITURES ESTIMATES

		Year -2	Year -1	Total Units (m)	Pre-production	Total Pre-Production
Main Access	East Decline					
	Main Decline	1439	1352	2791	$ 3,703	$ 10,333,739
	Ancillary	216	203	419	$ 3,703	$ 1,550,061
	Vent Accesses	70	70	140	$ 3,703	$ 518,353
East Zone	5355 Level	0	60	60	$ 3,703	$ 222,151
	5255 Level	0	60	60	$ 3,703	$ 222,151
	5155 Level	0	120	120	$ 3,703	$ 444,303
West Zone	5350 Level	0	485	485	$ 3,703	$ 1,795,723
	5250 Level	0	640	640	$ 3,703	$ 2,369,614
	5150 Level	0	680	680	$ 3,703	$ 2,517,715
Decline Exhaust Raise		202	218	420	$ 1,783	$ 749,058
						$ 20,722,869
Contingency					$ 0	$ 4,144,574
Total Pre-production Development Expenditures						**$ 24,867,443**

TABLE 21.3
UNDERGROUND MINE INFRASTRUCTURE CAPITAL EXPENDITURE ESTIMATES

Component	Quantity	Units	Unit Cost (US)	Total Cost (US)	Year -2	Year -1
SURFACE INFRASTRUCTURE						
Mine Portal	1	L.S.	$ 350,000	$ 350,000	$ 350,000	
Surface Intake Vent Fan Installation	1	L.S.	$ 650,000	$ 650,000	$ 650,000	
Mine Air Heaters	1	L.S.	$ 80,000	$ 80,000	$ 80,000	
Explosives Magazines (Supplier Provided)	1	L.S.	$ 25,000	$ 25,000	$ 25,000	
Compressors	3	L.S.	$ 258,462	$ 775,385	$ 258,462	$ 258,462
Cemented Backfill Plant	1	L.S.	$ 1,500,000	$ 1,500,000		$ 1,500,000
Mine Rescue and Fire Fighting Equipment	1	L.S.	$ 255,274	$ 255,274	$ 255,274	
Hoisting plant system	1	L.S.	$ 4,500,000	$ 4,500,000		$ 2,250,000
Total Surface Infrastructure				$ 8,135,659	$ 1,618,736	$ 4,008,462
Mob, Setup & Demob	1	L.S.	$ 100,000	$ 100,000	$ 100,000	
UNDERGROUND SUPPORT SERVICES FACILITIES						
Exhaust Ventilation Fans Installations	1	L.S.	$ 750,000	$ 750,000		$ 750,000
Maintenance Breakdown Shop	1	L.S.	$ 250,000	$ 250,000		
Fuelling Station (Marcotte)	10	L.S.	$ 69,231	$ 692,308	$ 138,462	$ 138,462
Explosives & Detonators Magazines Construction & Equipping	6	L.S.	$ 66,154	$ 396,923	$ 66,154	$ 132,308
Main Storage Area Construction & Equipping	1	L.S.	$ 36,000	$ 36,000		$ 36,000
Main Dewatering Sump Construction & Equipping	2	L.S.	$ 88,462	$ 176,923		$ 88,462
Refuge Station Construction & Equipping	7	L.S.	$ 107,692	$ 753,846	$ 107,692	$ 107,692
Portable Toilets	7	L.S.	$ 5,000	$ 35,000	$ 5,000	$ 5,000
Total Underground Support Services Facilities				$ 3,091,000	$ 317,308	$ 1,257,923
MINE SERVICES						
Portable Substations	25	each	$ 95,942	$ 2,398,558	$ 575,654	$ 575,654
Mine Communication	1	L.S.	$ 192,308	$ 192,308		$ 192,308
Computers, Peripherals & Software	1	L.S.	$ 84,615	$ 84,615	$ 42,308	$ 42,308
Engineering & Geology Equipment	1	L.S.	$ 33,846	$ 33,846	$ 33,846	
Backfill Distribution System	8	L.S.	$ 350,000	$ 2,800,000	$ 350,000	$ 700,000
Underground Booster Fans & Auxilliary Ventilation	1	L.S.	$ 338,000	$ 338,000	$ 169,000	$ 169,000
Mine Lamps	75	each	$ 200	$ 15,000	$ 7,500	$ 7,500
Total Mine Services				$ 5,862,327	$ 1,178,308	$ 1,686,769
TOTAL MINE INFRASTRUCTURE EXPENDITURES				$ 17,188,986	$ 3,214,351	$ 6,953,154

21.1.5 Processing Plant

The following methodology was used to develop the capital cost for the processing plant treating 1,300 dmtpd of ore and 92% plant availability:

1. Major equipment was sized based on available metallurgical data. The list of equipment along with the cost are provided in Table 21.4.

2. Cleaner flotation cells for second to fourth stage were not sized because two or three stages of cleaners may be sufficient. However, estimated cost was provided for the flotation cells.

3. Feed bins, conditioning tanks and other miscellaneous equipment were costed at 25% of the major equipment cost.

4. Construction, EPCM, installation etc. were factored for this study.

5. Process water treatment plant, which may or may not be required, was estimated at $1 million.

TABLE 21.4
MAJOR EQUIPMENT LIST FOR MILLING CIRCUIT

No.	Equipment	HP	No. of units	Cost/unit $	Total Cost $
1.	Grizzly Feeder	1	1	15,000	15,000
2.	20 x 30 Jaw Crusher	100	1	268,000	268,000
3.	3 ft. Cone Crusher	200	1	375,000	375,000
4.	2.5 for Short Head Cone	100	1	375,000	375,000
5.	4 ft. x 8 ft. Vibrating Double Deck Screen	10	1	52,500	52,500
6.	30 in. x 100 ft. Conveyor 30 HP each	90	3	72,000	216,000
7.	12 in. x 5 ft. Vibrating Feeder W/Bin	-	1	15,000	15,000
8.	14 ft. diameter x 22 ft. long Ball Mill	1500	1	1,000,000	1,000,000
9.	10 in. Cyclones	-	8	3,500	28,000
10.	Sump/Pump (1400 gpm)	44	1	45,000	45,000
11.	Rougher Flotation Cells 40 HP each, 500 ft^3	600	15	70,000	1,050,000
12.	Pumps (1400 gpm) 44 HP each	132	3	30,000	90,000
13.	Cleaner 1 Flotation Cells 20 HP, 150 $ft.^3$	300	15	45,000	675,000
14.	Cleaners 2 to 4 Cells				1,000,000
15.	Regrind Mills 100 HP	300	3	350,000	1,050,000
16.	Thickener 20 ft. diameter 3 HP each	9	3	50,000	150,000
17.	1000 gpm Plate & Frame Filter, 10 HP	30	3	230,000	690,000
18.	300 cfm Vacuum Pump, 1 HP	3	3	27,000	81,000
19.	Reagents Tanks & Pumps	-	-	-	824,500
				Sub-total	8,000,000
20.	Miscellaneous Equipment (25% of sub-total)				2,000,000
				TOTAL	**10,000,000**

The total plant cost, given in Table 21.5, was estimated to be approximately US$36 million. The cost estimate includes a contingency of $5.26 million (20%).

TABLE 21.5
PLANT CAPITAL COST

No.	Item	Cost $
1.	Equipment	10,000,000
2.	Installation Labor (@ 70%)	7,000,000
3.	Concrete (@ 10%)	1,000,000
4.	Piping (@ 30%)	3,000,000
5.	Structural Steel (@ 10%)	1,000,000
6.	Insulation (@ 3%)	300,000
7.	Instrumentation (@ 7%)	700,000
8.	Electrical (@ 12%)	1,200,000
9.	Coating & Sealants (@ 1%)	100,000
10.	Mill Building	1,000,000
11.	Process Water Treatment	1,000,000
	Sub-total	26,300,000
12.	EPCM (@ 15% of sub-total)	3,945,000
13.	Contingency (@ 20% of sub-total)	5,260,000
	Total	**35,505,000**

21.1.6 Surface Infrastructure

Total pre-production capital expenditures for project infrastructure and surface department are estimated to be approximately $21.8 million, including a 20% contingency. The breakdown of expenditures is presented in Table 21.6. Major expenditure components are for access road upgrading, power supply and distribution, site preparation, waste rock and ore storage pads, shops equipping, water supply and treatment, and mobile equipment.

TABLE 21.6
INFRASTRUCTURE AND SUPPORT SERVICES CAPITAL EXPENDITURES

Component		Cost
Main Access Road		$1,500,000
Main operations pad prep		$1,790,000
Pad construction		$2,708,333
Versant Power Transmittion		$5,600,000
Power distribution		$2,000,000
Potable Water System		$250,000
Parking		$50,000
Sewage System		$250,000
Buldings earthworks		$750,000
Maintenance Shop		350000
Fuel storage		$25,000
Propane		$35,000
Dry facility		$750,000
Waste dump Construction (Clean)		$125,000
Waste dump Construction (Acid Generating)		$400,000
Ore Pad/Temp Stockpile		$400,000
Office buildings		$750,000
Effluent pond		$750,000
Cap and powder magazines		
		$18,483,333
EPCM	8%	$ 1,478,667
First Fills		$75,000
Spare Parts		$75,000
Contingency	20%	$ 4,022,400
Total Infrastructure Expenditures		**$24,134,400**

21.1.7 Project Indirects and Owner's Costs

Project Indirects and Owner's Costs are estimated at US$6.72 million over the 2-year pre-production period. Owner's costs also include all equivalent General and Administration 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 US$153.7 million. The total expenditures include EPCM, contractor overheads and a 20% contingency on all estimated expenditures. A summary of project pre-production capital expenditures is presented in Table 21.7. A working capital allowance of US$11.6 million is estimated to be required.

TABLE 21.7
PROJECT PRE-PRODUCTION CAPITAL EXPENDITURES

Cost Component	Year -2	Year -1
	Expenditure	
Underground Development	$ 6,745,987	$ 14,689,701
Mine Facilities and Equipment	$ 3,214,351	$ 6,953,154
Mine Equipment Leasing and Remanufacturing	$ 987,430	$ 987,430
Infrastructure	$ 10,000,000	$ 10,112,000
Surface Mobile Equipment		$ 1,000,000
Tailings Management Facility		$ 2,001,495
Build and equip mill	$ 9,000,000	$ 25,581,000
Owners Indirects	$ 2,217,000	$ 4,116,000
Reclamation and Closure	$ 13,684,557	
Contingency @ 20%	$ 9,169,865	$ 15,392,956
Working Capital		$ 11,524,000
Sub-total Capital Expenditures	**$ 55,019,190**	**$ 92,357,736**
Total Capital Expenditures	**$147,376,925**	

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.

21.1.9 Working Capital

Working Capital has been estimated at US$11.5 million based on 4 months of the estimated operating costs for the year.

21.1.10 Sustaining Capital

Sustaining capital is estimated at US$100 million for the life of the mine and consists of continuing underground development, expansion and construction of mine facilities and equipment, equipment leasing and replacement, construction of the tailings storage facility, and staged closure costs of the tailings storage facility. Sustaining capital requirements are presented in Table 21.8.

TABLE 21.8
PROJECT SUSTAINING CAPITAL REQUIREMENTS

	Year 1	Year 2	Year 3	Year 4	Year 5	Year 6	Year 7	Year 8	Year 9
Underground Development	$15,533,815	$15,236,152	$9,381,831	$8,453,491	$8,535,734	$4,734,586	$1,468,628	$1,468,628	$1,174,903
Mine Facilities and Equipment	$4,417,577	$1,794,115	$809,788						
Mine Equipment Leasing and Remanufacturing	$3,175,078	$3,175,078	$3,175,078	$2,187,648	$2,187,648		$1,376,923		
Tailings Storage Facility	$247,860	$1,226,081	$1,629,618	$1,226,081	$1,629,618	$1,226,081	$1,629,618	$1,226,081	$1,629,618
	$23,374,330	$21,431,426	$14,996,316	$11,867,220	$12,353,001	$5,960,667	$4,475,169	$2,694,709	$2,804,521
Total Sustaining Capital	**$99,957,359**								

The tailings storage facility is planned to be constructed in phases, with five roughly equal-sized cells constructed sequentially and progressively reclaimed after each cell is completed. The construction and reclamation costs have been split evenly in 5 phases for this cost estimate; however, realistically, these costs would be greater for initial construction and final closure, and less during the intervening years of operation.

21.1.11 Financial Assurance Trust – Reclamation and Closure Costs

The Financial Assurance Trust fund, required by MEDEP, is included as capital cost in Year (-2). The total cost for the Financial Assurance Trust used in the PEA is $13,684,557. This considers a present value from future costs at a discount rate of 2% based on standard federal rates and does not include any salvage value for assets at closure.

The reclamation and closure costs include the cost for the closure of the last remaining TMF cell is $1,226,081 (SLR). As outlined in Section 21.0, the cost for the closure of all other TMF cells is included in the TMF capital cost line item. The cost for building demolition, remaining mine backfill, and site grading/re-vegetation was developed by AMPL, and is estimated at $2,200,000, $600,000, and $978,393, respectively. The total reclamation and closure cost (approximately $5,000,000) for these elements will be paid for through salvage income following the cessation of mining. Salvage income is expected to be in the range of $5,200,000. The income for the salvage component of mine closure and reclamation has not been included as part of the financial analysis in this PEA; therefore ,the cost for reclamation, which will be funded by the revenue stream, was not included as a line item cost. The salvage revenue should be factored and reflected in the next technical study – preliminary feasibility study.

21.2 Operating Cost Estimates

21.2.1 Basis for Estimates

Operating costs are based on U.S. 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 State of Maine.

Critical operating cost components are based on the following costs:

- The diesel fuel price is assumed to be US$0.80/litre.
- The electrical power cost is assumed to be US$0.085 per kWh.

Labour costs for the operating period are based on the manpower schedules presented for each department and the associated labour costs. The costs include a burden component of approximately

30%. Labour rates are based on local rates where available and/or 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 2020 US 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 $47.73 per tonne of potentially economic mineralisation (presented in Table 21.9).

TABLE 21.9
MINE OPERATING COSTS

	Longhole Stoping		Alimak Stoping
Component	**Cost**		**Cost**
Stope Development	$ 3.55		Incl. in Dev
Cable Bolting			$ 1.53
Longhole Drilling Operating Costs	$ 0.70		$ 0.25
Longhole Blasting	$ 1.09		$ 0.46
Stope Mucking	$ 2.80		$ 2.80
Longhole Drilling Manpower	$ 6.33		$ 10.57
Services Equipment	$ 2.07		$ 2.60
Heating Costs	$ 0.84		$ 0.84
Electrical Power	$ 4.02		$ 4.02
Backfill	$ 12.00		$ 12.00
Services Manpower	$ 13.16		$ 13.16
Total Mining Cost per Tonne	**$ 46.55**		**$ 48.22**

Mines services and overheads costs include all other non-direct stoping costs for the Picket Mountain Mine. 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 average mining cost is $47.73 based on an approximate 30/70 split for longhole mining and Alimak mining.

The mining costs are based on costs provided by U.S.-based underground contractors and from Infomine™ data supplied by Virginia Tech University regarding mining costs in the U.S.

21.2.3 Water Treatment Facility

The total costs for the water treatment facility are included in the operating costs as it is anticipated to operate via a Build Own Operate contract with the supplier. This type of arrangement ensures that the supplier has full control over the construction and operation of the plant in order to manage costs and ensure the water quality is achieved to the levels agreed upon (background or better quality). The supplier then charges the project owner a flat rate per day to operate the plant. The cost to operate the water treatment plant is $1.74/t using this contract basis.

21.2.4 Processing Plant and Tailings Management

The operating costs for the processing plant and the tailings management facility are detailed in Table 21.10, below.

TABLE 21.10
BREAKDOWN OF PROCESSING COSTS

Component	Cost
Equipment Operation	$ 8.00
Supplies	$ 12.00
Labour	$ 5.00
Administration	$ 3.00
Sundry Items	$ 2.00
Filter Plant Operation	$ 1.25
Dry Stack Tailings Placement	$ 1.30
Total	**$ 32.55**

21.2.5 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. Administration operating costs include costs and taxes for maintaining the property in good standing, land taxes, and resource usage fees (water, etc.).

The total yearly G&A costs are estimated to be approximately US$3.3 million (presented in Table 21.11), of which approximately US$2.17 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 US$7.69 per tonne of potentially economic mineralisation processed. However, the life-of-mine G&A cost would be US$7.95 per tonne as a result of the partial final year of operations and fixed costs to maintain production.

TABLE 21.11

GENERAL AND ADMINISTRATIVE OPERATING COST COMPONENTS

Component	Annual Cost ($US)
Salaries & Overhead	$ 2,166,000
Training	$ 10,000
Safety Equipment	$ 5,000
Medical, Health & Safety	$ 50,000
Government Relations	$ 20,000
Power	$ 40,000
Travel & Accommodations	$ 20,000
Marketing	$ 25,000
Legal and Accounting	$ 30,000
Consultants & Vendors	$ 500,000
Shipping, Courier and light freight	$ 30,000
Communications	$ 25,000
Office Supplies	$ 15,000
Computer Supplies	$ 20,000
Light Vehicles Operation	$ 25,000
Roads and Yards Maintenance	$ 30,000
Insurance	$ 100,000
Human Resources	$ 30,000
Bank Costs	$ 10,000
Surface ITC	$ 50,000
Buildings Maintenance	$ 5,000
Electrical Distribution Repair	$ 5,000
Water Supply & Water Treatment	$ 50,000
Office Equipment Leases	$ 12,000
Security Supplies	$ 5,000
Cleaning contract	$ 20,000
Dues & Subscriptions	$ 5,000
PR	$ 20,000
TOTAL G&A COSTS	**$ 3,323,000**

The mine management and administration roster and costs have been estimated in Table 21.12. A total of 16 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 is US$5.01 per tonne of potentially economic mineralisation processed.

TABLE 21.12
G&A MANPOWER COSTS

Position	Complement	Annual Salary	Fringe Benefits	Total Cost
		($US)	35%	($US)
Mine Manager	1	$ 200,000	35%	$ 270,000
Mine Superintendent	1	$ 175,000	35%	$ 236,000
Mill Superintendent	1	$ 160,000	35%	$ 216,000
Technical Services Superintendent	1	$ 160,000	35%	$ 216,000
Senior Engineer	1	$ 135,000	35%	$ 182,000
Accountant	1	$ 75,000	35%	$ 101,000
Eng/Geo technicians	2	$ 90,000	35%	$ 243,000
Purchasing/Warehouse Manager	1	$ 140,000	35%	$ 189,000
Environmental Coordinator	1	$ 80,000	35%	$ 108,000
Medical Contract	1	$ 60,000	35%	$ 81,000
Security Guard	4	$ 45,000	35%	$ 243,000
Site Services	1	$ 60,000	35%	$ 81,000
Total	**16**			**$ 2,166,000**

21.2.6 Concentrate Transport Charges

Transportation charges of $40.00 per tonne of concentrate have been included in the cash flow model and are based on $4 handling on site, $12/tonne to transport to port, $4 to handle at port, and $20 to transport by ship to smelter.

21.2.7 Project Total Operating Costs

The estimated total average operating cost (excluding smelting and refining) for the Pickett Mountain Mine is approximately $93.08 per tonne. Table 21.13 presents a summary table of life-of-mine average operating costs for each department on a cost per tonne of potentially economic mineralisation basis.

TABLE 21.13
PROJECT TOTAL OPERATING COSTS

Department	Cost
Underground Mining	$ 47.73
Processing	$ 31.25
Dry Stack Placement of Tailings	$ 1.30
Surface Services	$ 2.63
General and Administration	$ 7.95
Environmental and Sustainable Development	$ 2.21
Total Cost	**$ 93.08**

21.2.8 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 plan, operating costs, and capital expenditures incorporating expected long-term metal prices based on industry consensus pricing as of September 2020. (Table 22.1).

TABLE 22.1
COMMODITY PRICING

Commodity	Consensus Pricing
Zinc	$ 1.15
Copper	$ 3.00
Lead	$ 1.00
Gold	$ 1,500.00
Silver	$ 18.00

A summary of the expected parameters used for the financial analysis is presented in Table 22.2.

TABLE 22.2
CASHFLOW MODEL INPUT PARAMETERS

Undiluted Mineral Resources ~50/50 Indicate & Inferred	4,471,000 tonnes at grades of 9.51 % zinc, 1.23% copper, 3.77% lead, .88 g/t gold and 98.67 g/t silver	Average Mill Head Grade:	Payability	Average Long Term Pricing
Estimated Mining Dilution	10% at 0 grade			
Projected Mining Recovery	85%			
Zinc %		8.56	0.85	$ 1.15
Copper %		1.11	0.95	$ 3.00
Lead %		3.40	0.95	$ 1.00
Gold g/t		88.80	0.95	$ 1,500
Silver g/t		0.79	0.93	$ 18.00
Pre Production Capital, incl Working Capital	$ US 147.4 million			
Total Sustaining Capital	$ US 100 million			
Financial Assurance Trust: Reclamation & Closure	$ US 13.7 million			
Royalties	None			
Estimated Operating Costs ($/Tonne)	$ US 93.08 /tonne			
Life of Mine	9.7 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 also excluded from the cash flows.

Capital expenditures, as shown in the capital section, would be incurred over a two-year period, which is reflected in the discounted cash flow calculations. The cash flows include sustaining capital and capital expenditures contingency of approximately 20%.

Revenue is based on payments for the various metals produced and less the costs for metal sales and shipping and include the deductions that the refiner makes.

The expected cash flow analysis used the metal prices indicated above. The discounted cashflow analysis has been based on 2020 Constant US Dollar values.

The potentially mineable underground resource is estimated to be 4.2 million tonnes at a grade of 8.56% Zn, 1.11% Cu, 3.40% Pb, .79 grams Au/tonne, and 88.8 grams Ag per tonne. This PEA relies on Indicated Mineral Resources (approximately 48.7% 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. Metallurgical recoveries and capital and operating cost estimates are to at least PEA level of accuracy. Therefore, there is no guarantee that the economic projections contained in this PEA would be realised.

22.1 Taxes

Federal and state corporation and mining taxes, including allowed deductions for tax purposes, were included in the cashflow model.

The U.S. federal corporation tax rate is 21% on operating income after deducting capital expenditures.

The State of Maine Mining Excise Tax is levied on mining projects and applied as presented below.

The excise tax due on each mine site shall be the greater of the following:

1. **Tax on facilities and equipment**. The value of facilities and equipment multiplied by 0.005; or

2. **Tax on gross proceeds.** The gross proceeds multiplied by:

    A. If net proceeds are greater than zero, the greater of the following:

        (1) 0.009; or

        (2) A number determined by subtracting from 0.045 the quotient obtained by dividing:
            (a) Gross proceeds, by
            (b) Net proceeds multiplied by 100.

    B. If net proceeds are equal to or less than zero, then 0.009.

In the case of the Pickett Mountain Project, the applicable excise tax calculation is 2.2 tax on gross proceeds calculation 2.

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.

TABLE 22.3
EXPECTED PROJECT RETURNS

	Pre-Tax	After Tax
Undiscounted Net Revenue	626.6 million	626.6 million
Undiscounted Total Cash Flow	462.5 million	390.8 million
NPV (5%)	305.2 million	255.5 million
NPV (8%)	238.1 million	198.3 million
IRR	40%	37%
Payback Period	2.4 Years	2.4 Years

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

22.3 Sensitivity Analysis

Sensitivity analyses were performed for metal prices, capital expenditures, operating costs, mined grades, smelter charges, and recoveries with ranges up to 20% positive and negative variations. The Project is most sensitive to changes in metals prices and reasonably sensitive to changes in all the other variables. The results of the sensitivity analysis at ±20% are presented in Table 22.4 and Table 22.5.

TABLE 22.4
NPV 8% DISCOUNT SENSITIVITY ANALYSIS

Parameter	Pre-Tax NPV 8% ($M)								
	-20%	-15%	-10%	-5%	0%	5%	10%	15%	20%
Mined Grade	$104	$137	$171	$204	$238	$272	$306	$339	$373
Recovery	$106	$139	$172	$205	$238	$271	$304	$337	$369
Smelter Charges	$271	$262	$254	$246	$238	$230	$222	$214	$205
Metal Price	$74	$115	$156	$197	$238	$279	$320	$361	$402
Operating Costs	$285	$273	$261	$250	$238	$226	$215	$203	$191
Capital Costs	$276	$267	$257	$248	$238	$228	$219	$209	$200

Table 22.5
Pre-tax IRR Sensitivity Analysis

Parameter	Pre-Tax IRR (%)								
	-20%	-15%	-10%	-5%	0%	5%	10%	15%	20%
Sensitivity	23	28	32	36	40	44	48	51	55
Mined Grade	24	28	32	36	40	43	47	50	54
Recovery	43	42	42	41	40	39	38	37	36
Smelter Charges	19	25	30	35	40	44	49	53	57
Metal Price	45	44	42	41	40	38	37	36	34
Operating Costs	23	28	32	36	40	44	48	51	55
Capital Costs	52	48	45	42	40	37	35	33	31

The NPV and IRR sensitivities to variations in key parameters are depicted graphically in Figure 22.1 and Figure 22.2. The IRR is most sensitive to variations in metal prices and mined grades and less sensitive to capital and operating costs.

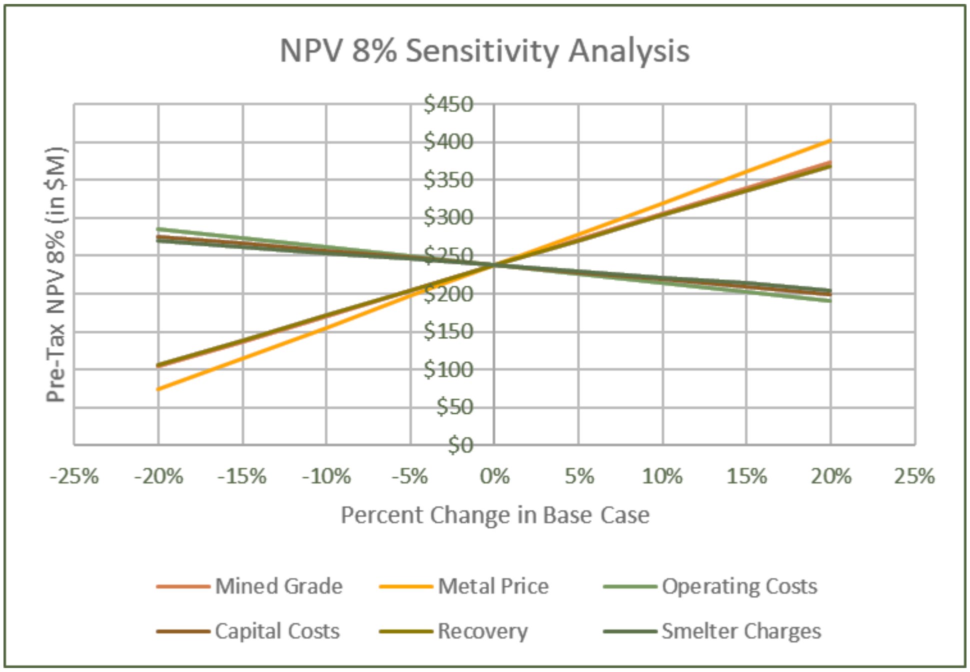


Figure 22.1 NPV at 8% Discount Sensitivity Analysis

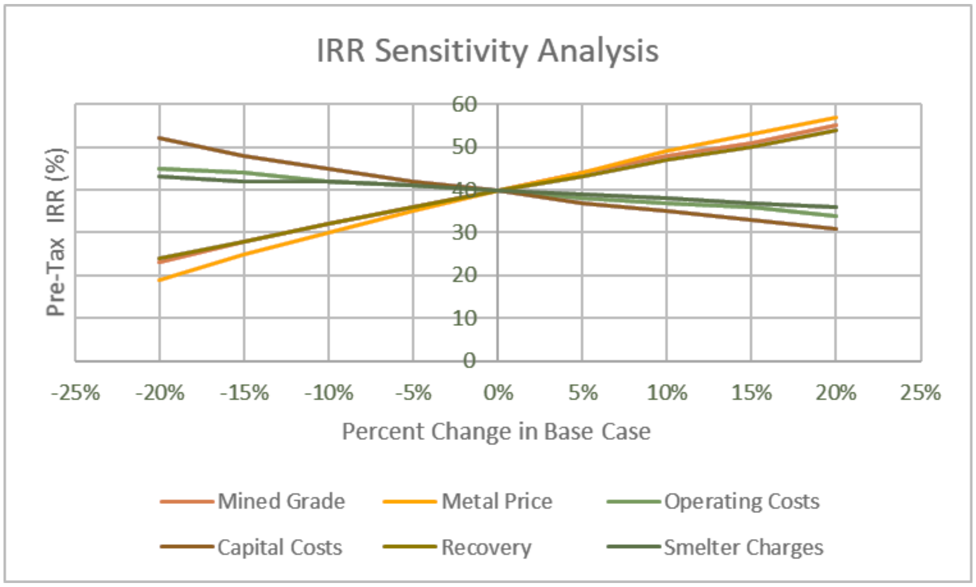


Figure 22.2 IRR Sensitivity Analysis

23.0 Adjacent Properties

There are no adjacent properties to the Pickett Mountain Project included in this study.

24.0 Other Relevant Data

There is no other relevant data.

25.0 Interpretation and Conclusions

The Pickett Mountain deposit is a typical volcanogenic massive sulphide deposit, with upper quartile grades. The deposit lies near the top of a variably altered, generally fragmental, felsic volcanic sequence with evidence of at least two periods of hydrothermal activity and base metal deposition. Two lenses of massive sulphide, West and East Lenses, comprise the Mineral Resource. The two sulphide lenses are separated by a short strike length zone of Z-folding, which results in a 50m to 80m horizontal displacement where fold-repetition of the massive sulphide lenses have been observed.

In addition to the East and West Lenses, a high-grade lens of footwall massive sulphide mineralisation (FW Zone) was discovered, approximately 150mstratigraphically below the East Lens. Interpretation of geology indicates that this FW Zone can traced along strike and is open both along strike and up plunge. More testing of this horizon is required.

This PEA has identified a diluted mineral resource of 4.2 million tonnes at 8.56% Zn, 1.11% Cu, 3.4% Pb, 0.79 g/t Au, and 88.8 g/t Ag. This resource is comprised of 50% Indicated Resources and 50% Inferred 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 deposit would be mined by underground mining methods with metals extracted in a processing plant custom built for the purpose. The mine site infrastructure facilities would be minimised but include a processing plant, small surface shop, warehouse, office complex, water treatment facility, dry stack tailings facility, and transformer and power distribution. Water for the project is assumed for this study to be provided from a well(s) near to the Project, initially, then mainly recycled within the project site.

The mine would operate at 432,000 tonnes per annum and produce $1.36 billion in cash flow during the life of the mine.

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

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 deposit to production.

2. The proposed project would be considered a small- to medium-sized underground mining operation, which can be developed for production at a reasonable cost in a near-term horizon, provided regulatory approval and permits are acquired.

3. The mined grade of potentially economic mineralisation is an important variable for the success of the operation as are operating costs. Operating management efforts during mine production must be focused on these parameters.

4. The scoping level test work has indicated that a sequential flotation process will produce marketable-grade copper, lead, and zinc concentrates. Arsenic and antimony levels were high in copper concentrates produced in open-cycle and locked-cycle tests. Additional geo-metallurgical test work will provide additional information on the impurities in the marketable-grade copper concentrate to determine if penalties need to be paid. In addition, blending of ores from different areas in the mine will keep impurities (As/Sb) below penalty levels.

5. The following conclusions can be drawn based on the historical and current scoping level metallurgical studies:

   - The sequential flotation process is a process of choice for recovering marketable-grade concentrates of copper, lead, and zinc that include in each, quantities of previous metals.

   - Blending of material into the mill and/or final copper concentrate may be required to maintain low levels of arsenic and antimony, below the penalty limits for the concentrate.

   - Metal recoveries of 78% to 88% for copper, lead, and zinc are expected in the selected flowsheet while maintaining high quality concentrates.

   - Further testing is needed to optimise metal recoveries and reagent quantities in order to maximise revenue and reduce Capex and Opex for the milling circuit.

The Project will be required to first obtain, from the Maine LUPC, approval of a rezoning petition that will allow for mining in this unorganised township. The petition was submitted and has been accepted by the LUPC as complete for its review. Based on initial soil and wetland field surveys in addition to desktop studies as described in the petition, it was concluded by Wood that the preliminary designs of the proposed project would have no undue impact on the natural resources and could be completed in a manner that would fit harmoniously within the surrounding area, and therefore, could satisfy the goals and specific requirements of the LUPC.

The socio-economic analysis completed for the petition indicated there is adequate local capacity to provide municipal services and a sufficient labor pool to be employed and trained as employees. Wolfden correspondence and presentations with the towns proximal to the project resulted in concurrence letters from these towns in support of the project and that it would not pose an undue burden on municipal services provided by these communities and that other purchased services (solid waste, communications, power) had adequate capacity.

Following a rezoning approval, Wolfden will need to obtain a Maine Metallic Mining Permit from the MEDEP under the Maine Chapter 200 rules. The Chapter 200 rules with respect to metallic mines, only allow for underground mining methods and require tailings disposal as dry stacked tailings, in lined facilities, to be closed with a final cover of equal hydraulic performance. It is technically and financially feasible for the project to meet these two requirements. Review of the requirements for mine design, mine operation, mine closure, water collection and treatment, and reclamation and environmental monitoring, did not identify technical or operational requirements that could not be met by a well-designed and responsibly managed project. The future Baseline Characterisation Studies needed to support an MEDEP Permit application are being developed and discussed with the MEDEP.

As part of Chapter 200 rules, the MEDEP will require that a financial assurance trust fund is established, prior to the issuance of a mining permit. In accordance with Section 17 of the Chapter 200 rules, the project will need to continuously maintain a financial assurance trust, as a condition of the mining permit, as determined by MEDEP that addresses all concerns related to closure, reclamation, post-closure maintenance and monitoring, and potential corrective actions that the mining operation and any associated waste material could pose an unreasonable threat to public health and safety or the environment. The financial assurance trust must include sufficient funds for the following:

a) The cost to investigate all possible releases of contaminants at the site, monitor all aspects of the mining operation, close the mining operation in accordance with the closure plan, conduct treatment activities as necessary for all fluids and wastes generated by the mining operation and those post closure for a minimum of 100 years, implement remedial activities for all possible releases and maintenance of structures and waste units as if these units have released contaminants to the groundwater and surface water, conduct corrective actions for potential environmental impacts to groundwater and surface water resources as identified in the environmental impact assessment and conduct all other necessary activities at the mine site in accordance with the environmental protection, reclamation and closure plan; and

b) The cost to respond to a worst-case catastrophic mining event or failure, including, but not limited to, the cost of restoring, repairing, and remediating any damage to public facilities or services, to private property, or to the environment resulting from the event or failure.

A filter cake TMF sized to contain the projected life of mine filtered tailings within the siting constraints identified can be constructed. Contact water from the TMF can be collected in an adequately sized pond constructed at the base of the TMF on the south side of the facility. The TMF would have a maximum elevation of 380m, which is approximately the elevation of the tree tops, as measured from the ground surface at the topographic divide at the south side of the facility. There is potential for expansion of the TMF using the land to the south and there is flexibility for phased construction and progressive reclamation of the TMF.

26.0 Recommendations

It is recommended that infill drilling should continue with the aim of upgrading the Inferred Resources to Indicated Resources and to better define the fold zone between the East and West Lenses. As well, follow-up drilling of the FW Lens and step-out drilling around the East and West Lens and regional drill targets is required in order to determine if additional resources can be delineated.

Metallurgical test work should be undertaken to optimise the process parameters for the proposed process flowsheet, including confirmation that the process flowsheet is capable of processing variable ores from the mine, determining if the metal recoveries can be improved, and determine whether the variable ores respond to mechanical sorting technologies.

Perform sufficient test work to size equipment for the planned throughput. Some test work may be required at the vendor's facility (*i.e.*, regrind mill sizing, thickener size, etc.).

Perform a detailed rock mechanics analysis for stope geometry and mine design including oriented core geotechnical drilling.

Continue to advance the project toward production by undertaking an advanced exploration program in parallel with finalising the project design and capital requirements. The goal of the Advanced Exploration Program will be to confirm resources with the objective of converting Mineral Resources to Mineral Reserves.

Complete a geochemical characterisation of simulated processed tailings for possible metal leaching and acid rock drainage (MI/ARD) potential.

It is recommended that Wolfden proceeds with the rezoning and mine permitting process, using contacts it has established with LUPC and MEDEP and engage with them in a proactive and collaborative fashion. Aligning with the State of Maine on how to meet legislative requirements for financial assurance trust fund should be prioritised. Future feasibility studies and designs should seek to avoid and minimise impacts to environmental, natural, cultural, scenic, and recreational resources to the extent possible. While the current mine development plan for treated water is subsurface infiltration, going forward Wolfden should consider other alternatives to provide greater flexibility and redundancy for the management of re-infiltration of treated water, if soil site conditions warrant. These alternatives should, at a minimum, consider spray irrigation, which is an established method for treated waters in the State of Maine and which could be designed to operate year-round.

Further studies are recommended to advance the tailings facility design including geotechnical and hydrogeological investigations including laboratory testing to confirm site conditions, identify any potential geologic hazards, characterise foundations and groundwater conditions, and identify suitable borrow sources for construction fill. Tailings characterisation testing is recommended to better define the geochemical, physical, settling, and filtration properties to validate the TMF design criteria. 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. A grading plan should be developed that optimises the cut/fill balance for the TMF base grade. 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 Pre-Feasibility Study or Feasibility Study. The cost to complete a Pre-Feasibility or Feasibility Study for the Pickett Mountain Project is estimated to be between US$3 million to US$5 million.

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Wolfden Mt. Chase LLC, Revised Petition to Rezone Portion of Township 6, Range 6 Penobscot County, Maine for Development of an Underground Metallic Mineral Deposit; Submitted to: Maine Land Use Planning Commission, June 30, 2020.

Wolfden Resources Corporation, April 28, 2020, Annual Information Form for the year ended December 31, 2019, Sedar filing.

CERTIFICATE OF QUALIFICATIONS

I, Finley J. Bakker, P. Geo., as a Professional Geoscientist and Consultant, residing at 4798 Andy Road, Campbell River, British Columbia, V9H 1C6, Canada, do hereby certify that:

1) I graduated with a BSc. Honours in Geology from McMaster University in 1979.

2) I am a member of the Association of Professional Engineers and Geoscientists of British Columbia APEGBC (1991) (Registration No. 18639).

3) I have worked as a geologist for a total of 41 years since my graduation from university.

4) I have read the definition of "qualified person," as 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.

5) This certificate applies to the technical report titled "Preliminary Economic Assessment Pickett Mountain Project" Penobscot County, Maine, USA, located at: 68.468°W Longitude 46.134°N Latitude (the "**Technical Report"**), and it is effective September 14, 2020 with a filing date of October 29, 2020.

6) My relevant experience for the purpose of the Technical Report is:

   a) Chief Geologist at four mines;
   b) I have held the positions of Senior Resource Geologist, Exploration Manager, and Superintendent of Technical Services;
   c) I have undertaken Resource calculations for 40 years;
   d) I have worked on a number of deposit types including VMS, skarn, epigenetic, and porphyry deposits and have specifically worked at and on Massive Sulphide deposits for 26 years;
   e) I have been involved with commodities including copper, lead, zinc, gold, silver, REE, tungsten and iron, and molybdenum; and
   f) I have used MineSight/Compass/Hexagon software used in calculating the Mineral Resource for 32 years.

7) I am responsible for the preparation of Sections 1.0 and 14.0 of the technical report titled ""Preliminary Economic Assessment Pickett Mountain Project," Penobscot County, Maine, USA"

8) I have had prior involvement with the property that is the subject of the Technical Report and have not completed a personal inspection of the property that is the subject of the Technical Report.

9) At 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 of the tests in Section 1.5 of NI 43-101. I have read NI 43-101 and Form 43-101F1 and the Technical Report, and the Technical Report has been prepared in compliance with that instrument and form.

11) I consent to the filing of the Technical Report with any stock exchange and other regulatory authority and any publication by them for regulatory purposes, including electronic publication in the public Company files on their websites accessible by the public, of the Technical Report.

Dated this 29th day of October 2020

Finley J. Bakker, P.Geo.

CERTIFICATE OF QUALIFICATIONS

I, Jerome Walton Grant, M.Sc., P. Geo, a consulting geologist with residence and business address at Box 627, Durham, Ontario, N0G 1R0, Canada do hereby certify that:

1) I have practiced my profession as a geologist in the private sector since 1981 in the gold and base- metal sectors of the mineral exploration industry.
2) I completed Geological Engineering, Mineral Exploration Option degree at Queen's University, Kingston, Ontario in 1985 and a Master of Science degree in geology at Queen's University in 1995.
3) I am a Professional Geoscientist – a Practicing Member of the Association of Professional Geoscientists of Ontario (APGO).
4) Over my career I have conducted GIS compilation, geological mapping, surface sampling and diamond drill programs at numerous locations in Canada and beyond, including VMS exploration at Kidd Creek and Hackett River.
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, experience and affiliation with a professional association, I am a "qualified person" for the purposes of NI 43-101.
6) This certificate applies to the technical report titled This certificate applies to the technical report titled "Preliminary Economic Assessment Pickett Mountain Project" Penobscot County, Maine, USA, located at: 68.468°W Longitude 46.134°N Latitude (the "**Technical Report"**), and it is effective September 14, 2020 with a filing date of October 29, 2020.
7) I am responsible for the preparation of Sections 8.0 to 13.0 of the Technical Report.
8) 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.
9) I conducted a personal inspection and worked on the Pickett Mountain Property for 6 weeks ending November 29, 2018, mapping and compiling geological, geochemical, geophysical, and the drill hole database.
10) I have had no prior involvement with the property that is the subject of the Technical Report.
11) I am not aware of any material fact or material change with respect to the subject matter of the Technical Report that is not reflected in the Technical Report, the omission to disclose which makes the Technical Report misleading.
12) I am independent of the issuer applying all of the tests in Section 1.5 of NI 43-101. There were no circumstances that were or could be seen to interfere with my judgement in preparing the Technical Report.
13) I have read NI 43-101, Form 43-101F1 and the Technical Report, and the Technical Report has been prepared in compliance with that instrument and that form.
14) I consent to the filing of the Technical Report with any stock exchange and other regulatory authority and any publication by them for regulatory purposes, including electronic publication in the public company files on their websites accessible by the public, of the Technical Report.

Dated this 29th day of October 2020

Jerry Grant, M.Sc., P. Geo.

CERTIFICATE OF QUALIFICATIONS

I, Brian LeBlanc, B.Sc., P. Eng., residing at 781 Community Hall Road, Thunder Bay, Ontario, Canada, do hereby certify that:

1) I am President and a Principal of A - Z Mining Professionals Limited.
2) This certificate applies to the technical report titled "Preliminary Economic Assessment Pickett Mountain Project" Penobscot County, Maine, USA, located at: 68.468°W Longitude 46.134°N Latitude (the "**Technical Report"**), and it is effective September 14, 2020 with a filing date of October 29, 2020.
3) I am a graduate of the Haileybury School of Mines as a Mining Technician (1981). I have also obtained a Bachelor of Science degree in Mining Engineering from Michigan Technological University (1986).
4) I am licensed by the Professional Engineers Ontario (License No. 90427972).
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) 13 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, 16.0, and 18.0 to 26.0. I co-authored and am responsible for Sections 16.0 and 21.0 to 26.0 of the Technical Report.
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 in order to make the Technical Report not misleading.
10) I am independent of the issuer applying all of 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.

Dated this 29th day of October 2020

Brian LeBlanc, B.Sc., P. Eng.

CERTIFICATE OF QUALIFICATIONS

I, Eric Hinton, residing at 27 Claremont Drive, Niverville, Manitoba R0A 0A2, Canada, do hereby certify that:

1) I am a Professional Mining Engineer.
2) This certificate applies to the technical report titled “Preliminary Economic Assessment Pickett Mountain Project” Penobscot County, Maine, USA, located at: 68.468°W Longitude 46.134°N Latitude (the “**Technical Report”**), and it is effective September 14, 2020 with a filing date of October 29, 2020.
3) I am a graduate of Queen’s University at Kingston in 1988 with a Bachelor of Science in Mining Engineering.
4) I am licensed by the Association of Professional Engineers and Geoscientists of the Province of Manitoba (License No. 33054).
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) Since 1988, I have been working in the mining industry as a mining engineer, mining researcher and mine consultant (32 years).
    b) I have worked in and consulted on base metal mines that were bulk tonnage operations as well as narrow vein ventures for 15 years.
7) I assisted in preparation of the Technical Report and Peer Review for Sections 1.0, 16.0, 18.0, 19.0, and 21.0 to 26.0 of the Technical Report. I co-authored and am responsible for Sections 16.0, 18.0, 21.0, and 22.0 of the Technical Report.
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 in order 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.

Dated this 29th day of October 2020

Eric Hinton, P. Eng.

CERTIFICATE OF QUALIFICATIONS

I, Ron C. deGagne, P. Geo. residing at 99 Dow Drive, Copper Cliff, Ontario, P0M 1N0, Canada, do hereby certify that:

1) I am a geoscience professional employed by Environmental Applications Group (EAG) Inc.
2) This certificate applies to the technical report titled "Preliminary Economic Assessment Pickett Mountain Project" Penobscot County, Maine, USA, located at: 68.468°W Longitude 46.134°N Latitude (the "**Technical Report"**), and it is effective September 14, 2020 with a filing date of October 29, 2020.
3) I am a graduate of Sir Sandford Fleming College, Ontario with a technology diploma in Earth Science (1981) with continuous geoscience working experience since 1995. I am a geoscientist currently licensed by the Professional Geoscientists of Ontario (License No 0557).
4) 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 fulfill the requirements to be a "Qualified Person" for the purposes of NI 43-101.
5) My relevant experience for the purpose of the Technical Report is:
   c) Environmental Technologist (Sudbury), Abandoned Mine Assessments, DST Consulting Engineers (1995-2002)
   d) Senior Geologist, (Sudbury) Wood Environment & Infrastructure Solutions Inc., Mine Waste Geochemistry and Abandoned Mine Assessments (2002-2020)
   e) Senior Geoscientist (Sudbury), Environmental Applications Group (EAG) Inc., Mine Waste Geochemistry (2020 - Present)
6) I have not visited the Property that is the subject of this Technical Report.
7) I am responsible for reviewing Section 20.0 and co-authoring Section 26.0 of this Technical Report.
8) I am independent of the Vendor and the Property.
9) I have not had prior involvement with the Project that is the subject of this Technical Report.
10) I have read NI 43-101 and Form 43-101F. Section 20.0 of this Technical Report has been prepared in compliance therewith.
11) As of the effective date of this Technical Report, to the best of my knowledge, information and belief, Section 20.0 of the Technical Report contains all scientific and technical information that is required to be disclosed to make the Technical Report not misleading.

Dated this 29th day of October 2020

Ron C. deGagne, P. Geo.

CERTIFICATE OF QUALIFICATIONS

I, Frank Palkovits, residing at 26 Windsor Crescent, Sudbury, ON, do hereby certify that:

1) I am a Mining Engineer, P.Eng.
2) This certificate applies to the technical report titled "Preliminary Economic Assessment Pickett Mountain Project" Penobscot County, Maine, USA, located at: 68.468°W Longitude 46.134°N Latitude (the "**Technical Report"**), and it is effective September 14, 2020 with a filing date of October 29, 2020.
3) I am a graduate of Laurentian University (1988)
4) I am licensed by the Professional Engineers Ontario BY NI 43-101 (License No. 90276379).
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: 40 years industry experience with 20 years in operating mining companies with increasing roles of responsibility up to and including Chief Engineer, and 20 years in consulting and EPCM projects in mine backfill and tailings management. I have worked on numerous polymetallic mines having many similar characteristics with this project.
   a) 40 years
7) I completed Section 18.22 of the Technical Report and contributed to Section 21.1.10 (Sustaining Capital for Tailings Expansion), and Section 21.2.7 (Project Total Operating Costs).
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 in order 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.

Dated this 29th day of October 2020

Frank Palkovits

CERTIFICATE OF QUALIFICATIONS

I, Deepak Malhotra, Ph.D., of Lakewood, Colorado, do hereby certify that:

1) I am currently employed as President of Pro Solv, LLC with an office at 15450 W. Asbury Avenue, Lakewood, Colorado, 80228.
2) This certificate applies to the technical report titled "Preliminary Economic Assessment Pickett Mountain Project" Penobscot County, Maine, USA, located at: 68.468°W Longitude 46.134°N Latitude (the "**Technical Report"**), and it is effective September 14, 2020 with a filing date of October 29, 2020.
3) I am a graduate of Colorado School of Mines in Colorado, USA (Masters of Metallurgical Engineering in 1973 and Ph. D. in Mineral Economics in 1978). I am a registered member in a good standing of the Association of Society of Mining and Metallurgical Engineers (SME) and a member of the Canadian Institute of Mining and Metallurgy (CIM). I have 48 years of experience in the area of metallurgy and mineral economics.
4) I have read the definition of "qualified person" set out in National Instrument 43-101 (NI43-101) 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 fulfill the requirements to be a "qualified person" for the purposes of NI 43-101.
5) I am responsible for Sections 13.0, 17.0 and parts of Sections 1.0, 19.0, 21.0, 25.0, 26.0, and 27.0 of the Technical Report.
6) I have not completed a personal inspection of the Property that is the subject of the Technical Report.
7) As of the effective date of the Technical Report and the date of this certificate, to the best of knowledge, information and belief, this Technical Report contains all scientific and technical information that is required to be disclosed to make the Technical Report not misleading.
8) I am independent of the Issuer and related companies applying all of the tests in Section 1.5 of the NI 43-101.
9) I have not had prior involvement with the property that is the subject of the Technical Report.
10) I have read NI 43-101, Form 43-101F1 and the Technical Report, and the Technical Report has been prepared in compliance therewith.

Dated this 29th day of October 2020

Deepak Malhotra, Ph. D.

CERTIFICATE OF QUALIFICATIONS

I, David Ritchie, M.Eng., of Brampton, Ontario, Canada, do hereby certify that:

1) I am currently employed as Managing Principal of SLR Consulting (Canada) Ltd. with offices at 55 University Avenue, Suite 501, Toronto, Ontario, Canada, M5J 2H7
2) This certificate applies to the technical report titled "Preliminary Economic Assessment Pickett Mountain Project" Penobscot County, Maine, USA, located at: 68.468°W Longitude 46.134°N Latitude (the "**Technical Report"**), and it is effective September 14, 2020 with a filing date of October 29, 2020.
3) I am a graduate of Ryerson Polytechnic University, Toronto, Canada (Bachelor of Engineering (Civil) in 1995) and University of Western Ontario (Master of Engineering (Geotechnical) in 2000). I am a registered member in a good standing of the Professional Engineers Ontario. I have 25 years of experience in the area of tailings management.
4) I have read the definition of "qualified person" set out in National Instrument 43-101 (NI43-101) 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 fulfill the requirements to be a "qualified person" for the purposes of NI 43-101.
5) I am responsible for Section 18.22, part of Section 18.22.1 and Section 18.22.2 through 18.22.7 and parts of Sections 1.10, and 21.1.10 and 26.0 of the Technical Report.
6) I have not completed a personal inspection of the Property that is the subject of the Technical Report.
7) As of the effective date of the Technical Report and the date of this certificate, to the best of knowledge, information and belief, this Technical Report contains all scientific and technical information that is required to be disclosed to make the Technical Report not misleading.
8) I am independent of the Issuer and related companies.
9) I had limited involvement with conceptual tailings management options in 2019 for the property that is the subject of the Technical Report.
10) I have read NI 43-101, Form 43-101F1 and the Technical Report, and the Technical Report has been prepared in compliance therewith.

Dated this 29th day of October 2020

David Ritchie, M.Eng.