NI 43-101 Technical Report

MINERAL RESOURCE ESTIMATE FOR THE
PINE POINT LEAD-ZINC PROJECT

Hay River, Northwest Territories, Canada

Prepared for:

Osisko Metals Inc.

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By qualified persons:

Jeffrey Cassoff, P. Eng. .... *BBA Inc.*
Colin Hardie, P. Eng...................... *BBA Inc.*
Pierre-Luc Richard, P. Geo............ *BBA Inc.*

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This amended and restated technical report is effective as of the 20th day of December 2018.

"Original signed and sealed on file"	*January 25, 2019*
Jeffrey Cassoff, P. Eng. BBA Inc.	Date

"Original signed and sealed on file"	*January 25, 2019*
Colin Hardie, P. Eng. BBA Inc.	Date

"Original signed and sealed on file"	*January 25, 2019*
Pierre-Luc Richard, P. Geo. BBA Inc.	Date

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TABLE OF ABBREVIATIONS	
Abbreviation	**Description**
3D	Three dimensional
a	Annum (year)
AA	Atomic absorption
AACE	American Association of Cost Engineers
ADR	Adsorption-desorption-recovery
AES	Atomic emission spectrometry
Ag	Silver
Al	Aluminum
ALS	ALS Minerals
Au	Gold
B	Billion
BBA	BBA Inc.
BLK	Blank
BMC	Bathurst Mining Camp
BV	Bureau Veritas Commodities Canada Ltd.
BWi	Bond work index
C	Carbon
Ca	Calcium
$Ca(OH)_2$	Calcium hydroxide
CAD or $	Canadian dollar (examples of use: CAD2.5M / $2.5M)
CaO	Lime
Cd	Cadmium
Ce	Cerium
CIM	Canadian Institute of Mining, Metallurgy and Petroleum
CO_2	Carbon dioxide
CoA	Certificate of authorization
conc.	Concentrate
CRM	Certified reference material
Cu	Copper
CV	Coefficient of variation
DDH	Diamond drillhole
DGPS	Differential Global Positioning Systems
DL	Detection limit
DUP	Duplicate
EA	Environmental assessment
EIA	Environmental Impact Assessment
EIS	Environmental Impact Statement

TABLE OF ABBREVIATIONS	
Abbreviation	**Description**
EOH	End of hole
ESA	Environmental Site Assessment
et al.	et alla (and others)
Fe	Iron
FS	Feasibility study
FW	Foot wall
GEMS	Geovia GEMS software
GNTW	Government of Northwest Territories
HQ	HQ- Caliber drillhole
HW	Hanging wall
ICP-MS	Inductively Coupled Plasma Mass Spectrometry
ID^2	Inverse distance square
IP	Induced Polarization
K	Potassium
K_{80}	80% passing – Particle size
KNA	Kriging neighbourhood analysis
LOM	Life of mine
M	Million
m.a.s.l.	Metres above sea level
Ma	Mega anum (Million years)
MAG	Magnetic
Mg	Magnesium
Mn	Manganese
Mpa	Mega pascals
MRE	Mineral Resource Estimate
MS	Massive sulphide
MVT	Mississippi Valley-Type
Na	Sodium
NaCN	Sodium cyanide
NaOH	Sodium hydroxide
Na_2O	Sodium Oxide
Ni	Nickel
NN	Nearest neighbour
No.	Number
NQ	NQ- Caliber drillhole
NSA	Not sufficient assay
NSR	Net smelter return

TABLE OF ABBREVIATIONS	
Abbreviation	**Description**
NTS	National topographic system
O_2	Oxygen
ORE	Ore Research & Exportation Pty Ltd.
OREAS	Ore Research & Exportation Pty Ltd. Assay Standards
Osisko Metals	Osisko Metals Incorporated
P_{80}	80% passing - Product size
Pb	Lead
PEA	Preliminary economic assessment
pH	Potential of hydrogen
PhD	Doctor of philosophy
PPMC	Pine Point Mining Camp
PPML	Pine Point Mining Limited
PYSTR	Pyrite stringer
QA/QC	Quality Assurance / Quality Control
QP	Qualified person
R^2	Coefficient of determination
RF	Revenue Factor
RQD	Rock Quality Designation
RWi	Rod work index
S	Sulphur
SD	Standard deviation
S.U.	Standard Unit
SEDAR	System for electronic document analysis and retrieval
SG	Specific gravity
SiO_2	Silicon dioxide / silica
SO_2	Sulphur dioxide
SO_4	Sulphate
SRM	Standard Reference Materials
Std	Standard S.U.
Ti	Titanium
TiO_2	Titanium dioxide
U	Uranium
USD or US$	United States dollar (examples of use: USD2.5M / US$2.5M)
UTM	Universal Transverse Mercator
vs.	Versus
Zn	Zinc

TABLE OF ABBREVIATIONS – UNITS OF MEASURE	
Unit	**Description**
Imperial	
ac	acre
deg. or °	angular degree
B	Billion
Btu	British thermal units
ft^2	square feet
ft^2/d	square feet per day
ft^3	cubic feet
ft^3/h	cubic feet per hour
cfm	cubic feet per minute
d	day (24 hours)
°F	Degrees Fahrenheit
Ø	diameter
ft	feet (12 inches)
ft/d	feet per day
ft/s	feet per second
ft/s^2	feet per second squared
gal	gallon
gpm	gallons (US) per minute
gal/h	gallons per hour
ha	Hectare
hp	horsepower
h	hour (60 minutes)
in. or "	inch
in. Hg	inches of mercury
in. WC	inches Water Column
in^2	square inch
K	Thousand (000)
k	Kips
k/ft^2	kips per square foot
lb	pound
lb/ft^3	pounds per cubic foot
lb/gal	pounds per gallon
lb/h	pounds per hour
lb/min	pounds per minute
lb/lb	pounds per pound
lb/t	pounds per tonne

TABLE OF ABBREVIATIONS – UNITS OF MEASURE	
Unit	**Description**
Imperial	
mi.	miles
mph	miles per hour
M	Million
MBtu	Million British thermal units
Mgal/d	Million gallons per day
mesh	US Mesh
min	minute (60 seconds)
mil	one thousandth of an inch
oz	Troy ounce
oz/t	Troy ounces per tonne
oz/y	Troy ounces per year
ppm	parts per million
%	Percent
%solids	Percent solids by weight
psf	pounds per square foot
psi	pounds per square inch
rpm	revolutions per minute
s	second
st	short ton (2,000 lbs)
SG	specific gravity
V	Volt
Wk	Week
wt%	weight percent
yd.	yard (36 inches)
y	year (365 days)

TABLE OF ABBREVIATIONS – UNITS OF MEASURE	
Unit	**Description**
Metric	
deg. or °	angular degree
m^3	cubic metre
d	day (24 hours)
°C	Degrees Celsius
Ø	diameter
$/t	Dollars per metric tonne
G	Giga
g	gram
g/t	grams per (metric) tonne
h	hour (60 minutes)
kg	kilogram
kg/t	kilograms per tonne
km	kilometres
km^2	square kilometre
kt	kilotonne
L	litre
m	metre
mg	milligram
Ml	millilitre
µm	micron
mm	millimetre
M	Million
Mt	Million metric tonne
ppm	parts per million
%	percent
SG	specific gravity
m^2	square metre
mm^2	square millimetres
K	Thousand (000)
t	tonne (1,000 kg) (metric ton)
tpa	tonnes per annum
tpd	tonnes per day
tpy	tonnes per year
W	Watt
wt%	weight percent
y	year (365 days)

1. SUMMARY

1.1 Introduction

In July 2018, BBA Inc. ("BBA") was contracted by Robin Adair, Vice President Exploration of Osisko Metals Inc. ("Osisko Metals" or the "issuer" or the "Company"), to prepare a new Mineral Resource Estimate (the "2018 MRE") for the Pine Point project (the "Project") and a supporting Technical Report in compliance with National Instrument 43-101 ("NI 43-101") and Form 43-101F1. The western boundary of the Project is located 42 kilometers east of the town of Hay River, Northwest Territories, Canada.

Osisko Metals is a mineral exploration company focused on the acquisition, exploration, and development of base metal resource projects in Canada. The TSXV symbol is OM and the headquarters are located in Montréal, Québec. BBA is an independent engineering and consulting firm with ten offices across Canada.

The Pine Point Mining Camp ("PPMC") was discovered in 1898 and exploited from 1964 to 1987 by Cominco Ltd. (Pine Point Mines Ltd. ("PPML")). During this period, approximately 69.4 Mst (64.3 Mt) of material grading 7.0% Zn and 3.1% Pb was extracted from approximately 50 open-pits and two underground mines.

This Technical Report, prepared by Pierre-Luc Richard, P. Geo., Jeffery Cassoff, P. Eng., and Colin Hardie, P. Eng., all from BBA Inc., provides an update on the Project, an updated resource estimate and supersedes all previous reports. The previous technical report was prepared for Darnley Bay Resources Limited and issued on February 20, 2017 (Siega and Gann, 2017). The current Technical Report is based on recent drilling by Osisko Metals, reviews of the historical work on the Project and all data obtained since the completion of the 2017 report. BBA also consulted other sources of information, primarily government databases, for assessment reports and the status of mining titles.

The authors believe the information used to prepare the Technical Report and to formulate its conclusions and recommendations is valid and appropriate considering the status of the Project and the purpose for which the report is prepared. The technical data are considered appropriate for producing a resource estimate for the Project. The authors, by virtue of their technical review of the project's exploration potential, affirm that the work program and recommendations presented in the report are in accordance with NI 43-101 and CIM Definition Standards for Mineral Resources and Mineral Reserves ("CIM Definition Standards").

1.2 Property Description, Location and Ownership

Osisko Metals' Pine Point mineral leases and claims are located approximately 800 km north of Edmonton, Alberta near the south shore of Great Slave Lake. The western boundary of the Project is located 42 km east of the town of Hay River, within the Mackenzie Mining Division of the Northwest Territories (NWT) of Canada.

The Project is composed of mineral leases and claims that cover a 65 km strike length in the PPMC and range between 42 km and 110 km from Hay River, NWT. The Project is situated about 10 km south of the Great Slave Lake and lies about 60 m above the lake level, which is at an elevation of 156 m above sea level (masl). Geographic coordinates are from 114° to 115° 15' West longitude and from 61° 0' to 61° 45' North latitude.

The mineral leases are situated north of the Territorial Highways 5 and 6 that connect Hay River, the former Pine Point town site to the east, and Highway 6 which continues eastward towards the hamlet of Fort Resolution. Highway 5 continues southward towards the Town of Fort Smith. These all-weather year-round highways parallel the southern boundary of the Osisko Metals mineral leases and claims (Figure 1-1).

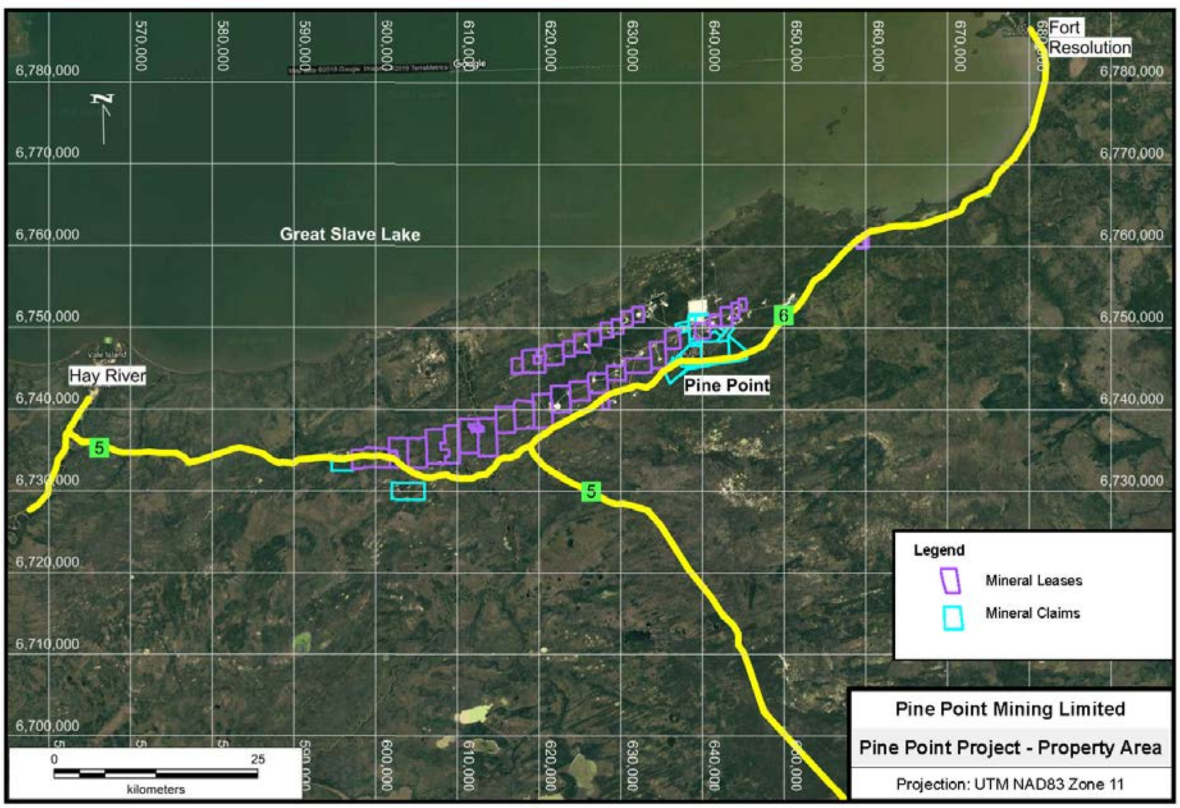

Figure 1-1: Osisko Metals Pine Point Leases and Claims

The Pine Point mineral deposits are covered by 40 mineral leases and 22 mineral claims in total. The mineral leases and claims are held by Pine Point Mining Limited a 100% owned subsidiary of Osisko Metals Incorporated. As of the effective date of this report, the leases are in good standing.

Osisko Metals, through Pine Point Mining Limited, has a 100% interest in the mineral leases and claims discussed in this report, subject to a 3% Net Smelter Return ("NSR") payable to Karst Investments LLC ("Karst"). There is an option agreement to purchase half of the royalty for USD3.5M. The Project is not subject to any other royalties, back-in rights, payments, or other agreements or encumbrances other than the territorial royalty (calculated as a tax but called a royalty).

1.3 Geology, Mineralization and Exploration Model

The Project is located on the eastern margin of the Western Canada Sedimentary Basin. The geology of the Project exhibits all the geological, mineralogical and geochemical attributes of Mississippi Valley-Type ("MVT"), carbonate hosted, base metal deposits (Leach et. al 2010).

The zinc-lead mineralization at Pine Point is hosted in a dolomitized carbonate barrier reef complex that transects the entire project area. Mineralization consists of sphalerite (ZnS), galena (PbS). As with most MVT deposits globally, there is no known spatial or temporal relation to igneous rocks.

The project area is underlain by the extensive southwest-trending sequence of Devonian Carbonate lithologies of middle-Devonian age. The sequence includes a large barrier reef complex that hosts mineralization. The carbonate sequence dips gently to the southwest and extends for up to 650 km into northern Alberta. In the project area, the individual Pine Point deposits are widely distributed across 65 km of a southwest trending belt of rocks that covers approximately 1,600 km^2. Most historically mined deposits contained between 0.20 Mt and 2 Mt of mineralized material, although the largest, the X-15 Deposit, contained nearly 18 Mt (Sangster, 1990).

The Middle Devonian sedimentary sequence at Pine Point is reasonably well understood because of the extensive diamond drilling campaigns carried out by Cominco Ltd. over almost 40 years.

The Pine Point deposits are aligned along the North, Main and Southern mineralized trends. These structural trends are within and parallel to the Pine Point barrier reef complex. They are described as follows:

- **Main Trend.** Mineralization is deposited within a zone, which is interpreted to have a subtle structural control allowing for the lateral continuity and alignment of the mineralized bodies.
- **North Trend.** Mineralization exhibits similar controls to mineralization where the zone extends with strong regularity for at least 20 km to the southwest.

- **South Trend.** Mineralization appears similarly aligned along a sub-parallel controlling zone but is less well defined and contains less intense dolomite alteration that is limited by restricted carbonate facies.

There is considerable exploration potential along strike to the west and at different stratigraphic levels in the sequence.

1.4 Status of Exploration and Drilling

The focus of activities at Pine Point in the period from January 2017 to time of this report has been on evaluating and defining the non-compliant historical resources left by Cominco Ltd.. An extensive drill program has been focussed on achieving a 30 m drill spacing as well as confirmation of historical drill data through hole twinning and relogging/assaying historical core. At the time of writing, drilling is on-going.

A DGPS survey has been underway since the summer of 2018 to locate and determine the absolute positions of approximately 4,000 drillholes among the roughly 18,000 historic drillholes in the Pine Point project area.

Ground gravity surveys were carried out in 2017 and consisted of 132-line km and 3,151 survey stations. These surveys were designed to locate areas of excess mass that could be caused by mineralization of significant concentrations. Test lines over known unmined deposits were conducted as a benchmark.

In 2017, 25 holes totaling 2,276 m were drilled testing areas outside existing resources. Some of the holes targeted areas of mineralization intersected by Cominco Ltd. in the past and others were directed at gravity features generated by the ground surveys. Only one hole, N38-17-PP-001 returned significant results intersecting 3.15 m of 2.73% Lead and 5.87% Zinc.

An in-fill drilling program was still underway at the time of writing and results are pending. As of December 6, 2018, 579 drillholes totalling 40,102 m of core were not incorporated into the drillhole database. The objective of the ongoing program is to continue to upgrade the Inferred Mineral Resource to the Indicated category by decreasing drill spacing to 30 m from the current average drill spacing of 40 m to 60 m. The Company expects to drill 920 additional holes totalling approximately 53,000 m in the remainder of 2018 and in 2019.

1.5 Data Verification

For the purpose of this MRE, BBA performed a basic validation on the entire database. Pierre-Luc Richard of BBA visited the Pine Point project from August 9 to August 12, 2018. The site visit included a visual inspection of historical core and core drilling in progress, a field tour, and discussions of the current geological interpretations with geologists and engineers of Osisko Metals.

BBA reviewed several sections of mineralized core while visiting the Project. All core boxes were labelled and properly stored either inside or outside. Sample tags were present in the boxes and it was possible to validate sample numbers and confirm the presence of mineralization in witness half-core samples from the mineralized zones.

Drilling was underway during BBA's site visit, which provided an opportunity for Osisko Metal personnel to explain the entire path of the drill core, from the drill rig to the logging and sampling facility and finally to the laboratory.

BBA was granted access to the original assay certificates for all holes drilled from Osisko Metals (2017-2018). Assays of Zn and Pb were verified for all holes. The assays recorded in the database were compared to the original certificates from the different laboratories and no significant discrepancies were detected.

After visiting the site, BBA resource geologist, Pierre-Luc Richard and Vice President of Exploration at Osisko Metals Inc., Robin Adair, proposed to take inventory of the available core and pilot a re-sampling procedure of the historical Cominco Ltd. drillholes. The objective of the following exercise was to re-assay previously drilled and logged holes within current resource definition drilling targets. The results of these assays would be compared to historic Cominco Ltd. assays available and verified against reference material. A selection of unsampled historical holes with likelihood to host mineralization were also identified. This preliminary exercise sought to quantitatively define the feasibility of further re-sampling work in the core yard considering: the physical state of the core in storage, accessibility to core, volume of preserved material, contamination of samples, and preservation of relevant labels/tags.

The results of this preliminary program confirmed that historical core can be re-sampled and compared with modern analytical results. Four sampled holes in the historical database were confirmed. An additional six unsampled historical holes proved to contain significant enough grades to propose a larger scale assaying program of all unsampled historical intercepts that are identified within the current mineralized model. Currently, all historical unsampled intervals within the model are attributed a grade of 0% Pb and 0% Zn.

Re-sampling drillholes in the Cominco Ltd. core yard has proven to be possible with minimal contamination. A larger scale program would be possible to recover core. Although there must be a mutual understanding that some holes are partially disturbed or completely sampled. This resource is of utmost importance in North Trend resources or other deposits where capital can be saved by validating Cominco Ltd. holes.

BBA is of the opinion that the drilling protocols in place are adequate. The database for the Pine Point project is of good overall quality. Minor variations have been noted during the validation process but have no material impact on the 2018 MRE. In the QP's opinion, the Pine Point database is appropriate to be used for the estimation of Mineral Resources.

1.6 Mineral Processing and Metallurgical Testing

Historically, Pine Point Mines Ltd. (Cominco Ltd.) mined and concentrated over 69.4 Mst (64.3 Mt) of mineralized material from many deposits over a period of 23 years with only slight variations to the process being required to achieve economic recoveries.

Since the closure of operations at Pine Point in 1987, metallurgical test programs were conducted on samples from across the Pine Point District including R-190, O-556, Z-155 and N-204 deposits by Tamerlane Ventures. Heavy media separation and flotation testwork indicated that standard zinc and lead flotation preceded by dense media separation will likely yield good recoveries.

In 2018, Osisko Metals initiated testwork to investigate the potential of mineral sorting technology. Preliminary results indicate that material from Pine Point is well suited for sensor-based sorting and this technology could potentially be used to pre-concentrate the material prior to grinding and flotation. Overall metallurgical recoveries of 83.1% zinc and 87.8% lead were estimated from past testwork and the 2018 mineral sorting program.

An extensive review of the historical operating metallurgy and recent testwork is presented in Chapter 13

1.7 Mineral Resource Estimate

The 2018 Pine Point Deposit Mineral Resource Estimate (effective November 14, 2018) was prepared by Pierre-Luc Richard, P. Geo., using all available information. The drillhole database used for the 2018 MRE includes 18,542 surface drillholes of which 6,880 intercepted mineralization. This includes Osisko Metals' 318 infill drillholes (23,751 m) with the remainder comprised of Cominco Ltd.'s historical drillholes, the use of which was validated by a drillhole collar survey and a partial core resampling program.

Based on data density, search ellipse criteria, drillhole density and interpolation parameters, the total Inferred Mineral Resource for the Pine Point deposit is estimated at 38.4 Mt with an average grade of 4.58% Zn and 1.85% Pb (6.58% Zn Eq) based on using a Zn equivalent cut-off grade varying from 1.7% to 2.0% depending on pit location and their metallurgical parameters (Table 1-1).

The estimate follows CIM Definition Standards. The Inferred Mineral Resource Estimate presented is constrained within pit shells developed during the pit optimization analysis phases. All deposits for which no pit was generated were removed from the resource estimate.

Table 1-1: Pit-constrained Inferred Mineral Resource Estimate

Area	Tonnage (Mt)	ZnEq (%)	Zn (%)	Pb (%)	Strip Ratio
Central Zone	4.80	7.69	5.84	1.72	11.70
East Mill Zone	5.50	5.16	3.76	1.30	5.70
North Zone	13.10	6.27	4.26	1.87	5.30
West Zone	6.40	10.09	6.30	3.53	14.50
N-204 Zone	8.60	4.74	3.61	1.02	5.40
Total	**38.40**	**6.58**	**4.58**	**1.85**	**7.70**

Notes to Table 1-1:

1. The independent qualified person for the 2018 MRE, as defined by NI 43-101 guidelines, is Pierre-Luc Richard, P. Geo., of BBA Inc. The effective date of the estimate is November 14, 2018.
2. These mineral resources are not mineral reserves as they do not have demonstrated economic viability. The quantity and grade of reported Inferred resources in this MRE are uncertain in nature and there has been insufficient exploration to define these Inferred resources as Indicated or Measured; however, it is reasonably expected that the majority of Inferred Mineral Resources could be upgraded to Indicated Mineral Resources with continued exploration.
3. Resources are presented as undiluted and in situ for an open-pit scenario and are considered to have reasonable prospects for economic extraction. The constraining pit shells were developed using pit slopes of 50 degrees.
4. The MRE was prepared using GEOVIA GEMS 6.8.2 and is based on 18,542 surface drillholes, of which 6,880 intercepted mineralization, and a total of 31,120 assays. The drillhole database includes Osisko Metals infill drilling of 23,751 metres in 318 drillholes and also incorporates Cominco Ltd.'s historical drillholes, the use of which was validated by a drillhole collar survey and a partial core resampling program. The cut-off date for drillhole assays was September 12, 2018.
5. The estimate encompasses 243 zinc-lead-bearing zones each defined by individual wireframes with a minimum true thickness of 2.5 m. A value of zero grade was applied in cases of core not assayed.
6. High-grade capping was done on the composited assay data and established on a per zone basis for zinc and lead. Capping grades vary from 10% to 35% Zn and 5% to 40% Pb.
7. Density values were calculated based on the formula established and used by Cominco Ltd. during their operational period between 1964 and 1987. Density values were calculated from the density of dolomite, adjusted by the amount of sphalerite, galena, and marcasite/pyrite as determined by metal assays. A porosity of 5% was assumed. Waste material was assigned the density of porous dolomite.
8. Grade model resource estimation was calculated from drillhole data using an Ordinary Kriging interpolation method in a block model using blocks measuring 10 m x 10 m x 5 m (vertical) in size.
9. Zinc equivalency percentages are calculated using metal prices, forecasted metal recoveries, concentrate grades, transport costs, smelter payable metals and charges.
10. The estimate is reported using a Zn Equivalent ("ZnEq") cut-off varying from 1.70% to 2.00%. Variations take into consideration trucking distances from the open pits to the mill and metallurgical parameters for each area. The cut-off grade was calculated using the following parameters (amongst others): zinc price = USD1.10/lb; lead price = USD0.90/lb; CAD:USD exchange rate = 1.31. The cut-off grade will be re-evaluated in light of future prevailing market conditions and costs.
11. The MRE presented herein is categorized as an Inferred resource. The Inferred mineral resource category is only defined within the areas where drill spacing is less than 100 m and shows reasonable geological and grade continuity.
12. The pit optimization to develop the resource constraining pit shells was done using Hexagon's MineSight Version 15.10.
13. Calculations used metric units (metre, tonne). Metal contents are presented in percent or pounds. Metric tonnages were rounded and any discrepancies in total amounts are due to rounding errors.
14. CIM definitions and guidelines for Mineral Resource Estimates have been followed.
15. The author is not aware of any known environmental, permitting, legal, title-related, taxation, socio-political or marketing issues, or any other relevant issues not reported in this Technical Report, that could materially affect the Mineral Resource Estimate.

1.8 Interpretation and Conclusions

1.8.1 Mineral Resource Estimate

The objective of BBA's mandate was to produce a Mineral Resource Estimate for the Pine Point Lead-Zinc project and a supporting NI 43-101 Technical Report. This Report and the 2018 MRE herein meet this objective. The mineral resource estimation parameters and geological interpretation for the Pine Point project were established by BBA. Historical operating data, past metallurgical testwork and recent metallurgical testing was also reviewed.

The Pine Point pit constrained Inferred MRE is 38.4Mt grading 4.58% Zn and 1.85% Pb (6.58% ZnEq) containing approximately 3.9 billion pounds of zinc and 1.6 billion pounds of lead.

1.8.2 Exploration Potential

Following an overall review of all pertinent information, including the MRE, BBA concluded the following:

- The exploration potential remains high at the property scale, justifying compilation and target generation programs;
- The Pine Point project hosts a significant amount of mineralized intercepts that merit follow-up work;
- The potential is high for adding additional resources to Pine Point project by drilling lateral extensions of numerous of the currently identified zones;
- It is likely that drilling additional holes therefore improving the current drill spacing would translate into upgrading Inferred resources to the Indicated category;
- A sampling program of the historical core currently stored on the property is likely to improve the grade of the MRE presented in this Report.

1.9 Recommendations

Based on the results of the 2018 MRE, BBA recommends additional exploration/delineation drilling, further geological interpretation, metallurgical testing and hydrogeological studies to gain a better understanding of the Project.

Osisko Metals has allocated $28.6M for additional drilling and project development activities. Details of the program are provided in Chapter 26. BBA is of the opinion that the recommended work program and proposed expenditures are appropriate and well thought out and that the proposed budget reasonably reflects the type and scope of the contemplated activities.

2. INTRODUCTION

The Pine Point Project (the "Project") is a lead-zinc exploration project located in the Northwest Territories, on the south shore of Great Slave Lake, approximately 42 km east of the town of Hay River. Osisko Metals Incorporated ("Osisko Metals" or the "Company") is the sole owner of the Project.

In July 2018, Osisko Metals commissioned BBA Inc. ("BBA") to lead and perform the update of the Mineral Resource Estimate ("MRE") in accordance with the guidelines of the Canadian Securities Administrators National Instrument 43-101 ("NI 43-101") and Form 43-101 F1.

BBA (www.bba.ca) is an independent engineering consulting firm headquartered in Mont-Saint-Hilaire, Québec with its mining group based in downtown Montréal and in Val-d'Or, Québec. The firm's expertise is recognized in the fields of energy, mining and metals, biofuels and oil and gas. BBA is supported by a network of offices across Canada to serve its clients and carry out mandates at the local, national and international levels.

2.1 Scope of Study

The following Technical Report (the "Report") presents the results of the Mineral Resource Estimate for the Pine Point Project. As of the date of this Report, Osisko Metals is a Canadian publicly traded company listed on the TSX Venture Exchange ("TSXV") under the trading symbol OM with its head office located at:

Suite 300, 1100, Avenue des Canadiens-de-Montréal
Montréal, Québec, H3B 2S2
Phone: (514) 861-4441

This Report, titled "Pine Point Lead-Zinc Project – Mineral Resource Estimate", was prepared by Qualified Persons ("QPs") following the guidelines of the NI 43-101, and in conformity with the guidelines of the Canadian Institute of Mining, Metallurgy and Petroleum ("CIM") Standards on Mineral Resources and Reserves.

2.2 Report Responsibility and Qualified Persons

The following individuals, by virtue of their education, experience and professional association, are considered QPs as defined in the NI 43-101, and are members in good standing of appropriate professional institutions.

- Jeffrey Cassoff, P. Eng. — BBA Inc.
- Colin Hardie, P. Eng. — BBA Inc.
- Pierre-Luc Richard, P. Geo. — BBA Inc.

The preceding QPs have contributed to the writing of this Report and have provided QP certificates, included at the beginning of this Report. The information contained in the certificates outlines the sections in this Report for which each QP is responsible. Each QP has also contributed figures, tables and portions of Chapters 1 (Summary), 25 (Interpretation and Conclusions), and 26 (Recommendations). Table 2-1 outlines the responsibilities for the various sections of the Report and the name of the corresponding Qualified Person.

Table 2-1: Qualified Persons and areas of report responsibility

Chapter	Description	Qualified Person	Company	Comments and exceptions
1.	Executive Summary	P.-L. Richard	BBA	All Chapter 1
2.	Introduction	P.-L. Richard	BBA	All Chapter 2
3.	Reliance on other Experts	P.-L. Richard	BBA	All Chapter 3
4.	Project Property Description and Location	C. Hardie	BBA	All Chapter 4
5.	Accessibility, Climate, Local Resource, Infrastructure and Physiography	P.-L. Richard	BBA	All Chapter 5
6.	History	P.-L. Richard	BBA	All Chapter 6
7.	Geological Setting and Mineralization	P.-L. Richard	BBA	All Chapter 7
8.	Deposit Types	P.-L. Richard	BBA	All Chapter 8
9.	Exploration	P.-L. Richard	BBA	All Chapter 9
10.	Drilling	P.-L. Richard	BBA	All Chapter 10
11.	Sample Preparation, Analyses and Security	P.-L. Richard	BBA	All Chapter 11
12.	Data Verification	P.-L. Richard	BBA	All Chapter 12
13.	Mineral Processing and Metallurgical Testing	C. Hardie	BBA	All Chapter 13
14.	Mineral Resource Estimate	P.-L. Richard	BBA	All Chapter 14 except Section 14.15
		J. Cassoff	BBA	Section 14.15
15.	Mineral Reserve Estimate	P.-L. Richard	BBA	Not required for a resource estimate
16.	Mining Methods	P.-L. Richard	BBA	Not required for a resource estimate
17.	Recovery Methods	P.-L. Richard	BBA	Not required for a resource estimate
18.	Project Infrastructure	P.-L. Richard	BBA	Not required for a resource estimate
19.	Market Studies and Contracts	P.-L. Richard	BBA	Not required for a resource estimate
20.	Environmental Studies, Permitting, and Social or Community Impact	P.-L. Richard	BBA	Not required for a resource estimate
21.	Capital and Operating Costs	P.-L. Richard	BBA	Not required for a resource estimate

Chapter	Description	Qualified Person	Company	Comments and exceptions
22.	Economic Analysis	P.-L. Richard	BBA	Not required for a resource estimate
23.	Adjacent Properties	P.-L. Richard	BBA	All Chapter 23
24.	Other Relevant Data and Information	P.-L. Richard	BBA	All Chapter 24
25.	Interpretation and Conclusions	P.-L. Richard	BBA	All Chapter 25
26.	Recommendations	P.-L. Richard	BBA	All Chapter 26
27.	References	P.-L. Richard	BBA	All Chapter 27

2.3 Effective Dates and Declaration

This Report is in support of the Osisko Metals press release, dated December 6, 2018, entitled "Osisko Metals releases Pine Point in-pit Inferred Resource: 38,400,000 tonnes grading 6.58% ZnEq". The overall effective date of the report is December 20, 2018. The Report has a number of close-out dates for information:

- Drill Database close-out date: September 12, 2018;
- Metallurgical testwork close-out date: November 6, 2018;
- Effective date of the mineral resource: November 14, 2018;
- Mineral Lease and Claim Status: December 20, 2018.

This Report was prepared as National Instrument 43-101 Technical Report for Osisko Metals by Qualified Persons from BBA Inc. collectively the "Report Authors".

The quality of information, conclusions, and estimates contained herein is consistent with the level of effort involved in the Report Authors' services, based on: i) information available at the time of preparation; ii) data supplied by outside sources; and iii) the assumptions, conditions, and qualifications set forth in this Report. This Report is intended for use by Osisko Metals subject to terms and conditions of its respective contracts with the Report Authors. Except for the purposes legislated under Canadian provincial and territorial securities law, any other uses of this Report by any third party is at that party's sole risk.

It should be understood that the mineral resources presented in this Report are estimates of the size and grade of the deposits. The estimates are based on a certain number of drillholes and samples, and on assumptions and parameters currently available. The level of confidence in the estimates depends upon a number of uncertainties. These uncertainties include, but are not limited to: future changes in metal prices and/or production costs, differences in size, grade and recovery rates from those expected, and changes in Project parameters. In addition, there is no assurance that the Project implementation will be carried out.

As of the effective date of this Report, the QPs are not aware of any known litigation potentially affecting the Project. The QPs did not verify the legality or terms of any underlying agreement(s) that may exist concerning the Project ownership, permits, off-take agreements, license agreements, royalties or other agreement(s) between Osisko Metals and any third parties.

BBA is not an insider, associate or an affiliate of Osisko Metals and neither BBA nor any affiliate has acted as Advisor to Osisko Metals, its subsidiaries or its affiliates, in connection with this Project. The results of the technical review by BBA are not dependent on any prior agreements concerning the conclusions to be reached, nor are there any undisclosed understandings concerning any future business dealings. The QPs are being paid fees for this work in accordance with the normal professional consulting practice.

The opinions contained herein are based on information collected throughout the course of investigations by the QPs, which in turn reflects various technical and economic conditions at the time of writing. Given the nature of the mining business, these conditions can change significantly over relatively short periods of time. Consequently, actual results can be significantly more or less favourable.

2.4 Sources of Information

This Report is based in part on internal company reports, maps, published government reports, company letters and memoranda, and public information, as listed in Chapter 27 “References” of this Report. Section from reports authored by others may have been directly quoted or summarized in the report and are so indicated, where appropriate.

This MRE has been completed using available information contained in, but not limited to, the following reports, documents and discussions:

- Technical discussions with Osisko Metals direction and personnel;
- QPs’ personal inspection of the Pine Point project site, including drill core and facilities;
- Cominco Ltd.’s historical drillhole database;
- Review of exploration data collected by Osisko Metals;
- Agreements, technical data and internal technical documents supplied by Osisko Metals;
- Internal unpublished reports from Osisko Metals;
- Additional information from public domain sources (SEDAR, etc.).

The QPs believe that the basic assumptions contained in the information above are factual and accurate, and that the interpretations are reasonable. The QPs have relied on this data and have no reason to believe that any material facts have been withheld or doubt the reliability of the information used to evaluate the mineral resources presented herein. The authors have sourced the information for this Report from the collection of documents listed in Chapter 27 (References).

2.5 Site Visit

Pierre-Luc Richard visited the Pine Point property from August 9 to August 11, 2018, as part of the current mandate. The purpose of the visit was to review the Pine Point Project with the PPML Geology team. The visit included an overview of the general geological conditions, a tour of the core storage facility, and visual inspections of select mineralized drill core samples. Mr. Richard also examined drill collars in the field and reviewed several core intervals. An independent resampling program, as well as a review of assaying, QA/QC and drillhole procedures was also completed.

Colin Hardie and Jeffrey Cassoff did not visit the Pine Point property.

2.6 Currency, Units of Measure, and Calculations

Unless otherwise specified or noted, the units used in this Report are metric. Every effort has been made to clearly display the appropriate units being used throughout this Report.

- Currency is in Canadian dollars ("CAD" or "$"), unless otherwise stated;
- A Canadian dollar (CAD) to United States dollar (USD) exchange rate of CAD 1.31 for USD 1.00 was used;
- Grid coordinates for the block model are given in the UTM NAD 83 and latitude/longitude system; maps are either in UTM coordinates or latitude/longitude system;

This Report may include technical information that required subsequent calculations to derive subtotals, totals and weighted averages. Such calculations inherently involve a degree of rounding and consequently introduce a margin of error. Where these occur, the QPs consider them immaterial.

2.7 Acknowledgment

The Report Authors would like to acknowledge the general support provided by Osisko Metals personnel during this assignment. Their collaboration is greatly appreciated. The Project also benefitted from the inputs of the following specific individuals:

- Jeff Hussey, President & CEO – Osisko Metals
- Robin Adair, Vice President Exploration – Osisko Metals
- Stanley Clemmer, Chief Geologist – Pine Point Mining Limited
- Guy Desharnais, Director Of Mineral Resource Evaluation – Osisko Gold Royalties
- Killian Charles, Vice President Corporate Development – Osisko Metals
- Christian Laroche, Director of Metallurgy – Osisko Gold Royalties
- Paul Johnson, Director of Open Pit Project Evaluation – Osisko Gold Royalties

- Andrée Drolet, Director of Environment – Osisko Mining
- Mayana Kissova, Director of Tailings and Water Management – Osisko Gold Royalties
- Charlotte Athurion, Geologist – BBA
- Manon Dussault, Project Assistant – BBA

Their commitment, contributions and team work are gratefully acknowledged and appreciated.

3. RELIANCE ON OTHER EXPERTS

3.1 Introduction

The Qualified Persons ("QPs") have relied upon reports, information sources and opinions provided by Osisko Metals and outside experts related to the Project's mineral rights, 3rd party agreements, surface rights, property agreements, royalties, and environmental status.

As of the date of this Report, Osisko Metals indicate that there are no known litigations potentially affecting the Pine Point Project.

A draft copy of the Report has been reviewed for factual errors by Osisko Metals. Any changes made as a result of these reviews did not involve any alteration to the conclusions made. Hence, the statements and opinions expressed in this document are given in good faith and in the belief that such statements and opinions are neither false nor misleading at the date of this Report.

3.2 Mineral Tenure and Surface Rights

Osisko Metals supplied information about mining titles, options agreements, royalty agreements, environmental liabilities and permits. The QPs from BBA consulted the North West Territories Government's Mining Recorder's office online claim management system via: https://www.maps.geomatics.gov.nt.ca/Html5Viewer_PROD/index.html?viewer=NWT_MTV for the latest status regarding ownership and mining titles. Although the QPs have reviewed the option agreements and available claim status documents, they are not qualified to express any legal opinion with respect to the property titles, current ownership or possible litigations. A description of such agreements, the property, and ownership thereof, is provided for general information purposes only. In this regard, the QPs have relied on information supplied by Osisko Metals and the work of experts they understand to be appropriately qualified.

This information is used in Chapter 4 of the Report. The information is also used in support of the Mineral Resource Estimate in Chapter 14.

3.3 Environmental Studies, Permitting, and Social or Community Impact

Colin Hardie, QP, relied upon information with respect to the project's environmental status, permits and, Social and Community Impact as provided by Andrée Drolet, Environmental Director, Osisko Mining. This information is used in Chapter 4 of the Report.

4. PROPERTY DESCRIPTION AND LOCATION

4.1 Mineral Tenure

Osisko Metals through its 100% owned subsidiary, Pine Point Mining Limited ("Pine Point" or "PPML") controls a semi-contiguous group of 40 mineral leases and 22 mineral claims in the Pine Point District, near the south shore of Great Slave Lake in the Northwest Territories of Canada (Table 4-1). The total area of the Project is 22,213.48 hectares. This is current as of December 20, 2018. Detailed lists of the PPML mineral leases and mineral claims are shown in Table 4-2 and Table 4-3 respectively. Currently, there are no adjacent mineral claims or leases to the project as the surrounding areas have been withdrawn from staking.

Table 4-1: Mineral leases and mineral claims

Title Type	Number	Area (ha)
Mineral Leases	40	17,548.20
Mineral Claims	22	4,665.28
Total	**62**	**22,213.48**

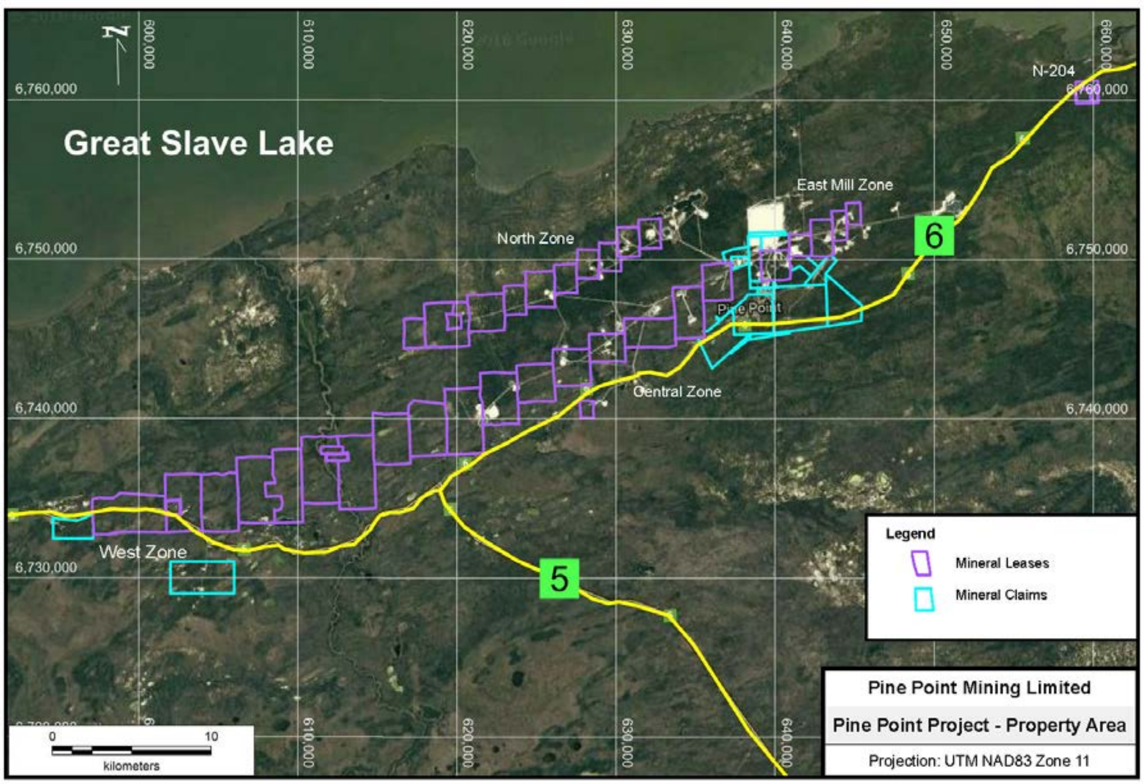

Figure 4-1: Pine Point Mining Limited Mineral Claims and Mineral Leases

4.2 Royalties and Encumbrances

The Pine Point project is subject to a 3% Net Smelter Return ("NSR") payable to Karst Investments LLC ("Karst"). On July 27, 2017, an option agreement was signed with Karst whereby Pine Point Mining may purchase a 50% undivided interest in the 3% NSR for US$3.5M. Pine Point Mining is required to make prepayment of USD75,000 on each anniversary date of the signing of the agreement until commercial production is achieved. Upon reaching commercial production, Pine Point Mining may exercise its option by paying Karst USD3,000,000 minus all prepayments. Following the excise of the option, the Pine Point project will be subject to a 1.5% NSR.

The Project is not subject to any other royalties, back-in rights, payments, or other agreements or encumbrances other than the territorial royalty (calculated as a tax but called a royalty).

4.3 Surface Leases

Pine Point has two surface leases in the R-190 deposit area that were acquired in 2010 to cover the proposed mine site and a settling pond envisioned in the Tamerlane 2007 feasibility study. Details of the leases are listed in Table 4-4 and due to devolution of government services between the government of Northwest Territories and the Federal government, two separate annual lease payments are due.

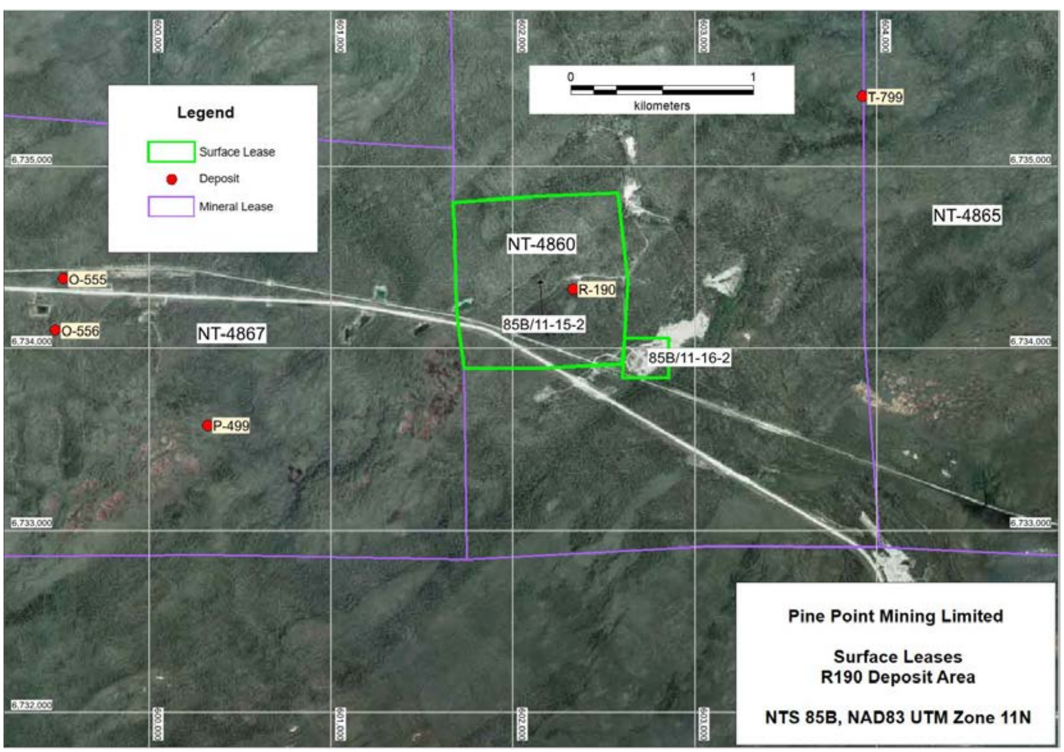

Figure 4-2: Pine Point Mining Limited Surface Leases

4.4 Permitting

PPML currently holds a ***Type-A Land Use Permit for Mineral Exploration*** for all of the leases and claims in the Pine Point area. The permit was issued on July 20, 2017, with an expiry date of July 19, 2022.

PPML also currently holds both a ***Type-A Land Use Permit and Type-B Water License for Mineral Exploration*** for confirmation drilling on some of the leases and claims in the Pine Point area. The permit and licence were issued on June 20, 2018, with an expiry date of September 19, 2020.

4.5 Environmental Liabilities

Part of the Pine Point project is a brownfield exploration and development project located on a previously disturbed mine site. Cominco Ltd. operated the Pine Point mine, producing 64 million tonnes of material from 50 open pits between 1964 and 1988. The Project includes historical open pits, waste rock stockpiles, tailings management facility, city water ponds, historically impacted areas including the plant site, haulage and service roads, the footprint of the former townsite of Pine Point and an airstrip.

As described above, most of the titles owned by PPML are mineral leases and mineral claims and no environmental liability is linked to these types of titles. However, environmental liabilities can be linked to surface leases, but in the case of the two surface leases owned by PPML, there are no existing liabilities within the lease’s boundaries.

4.6 Environmental Studies, Permitting, and Social or Community Impact

4.6.1 Introduction

The Pine Point project is located within the Taiga Plains Mid-Boreal ecoregion, south of the Great Slave Lake (ECG, 2009). Topography is gently undulating with three major hill systems. Characterization of the environment includes a cold boreal climate, wet conditions in low-lying poorly drained areas and a broad scale vegetation (Golder, 2018).

Part of the Pine Point project is a brownfield exploration and development project located on a previously disturbed mine site. Cominco Ltd. operated the Pine Point mine, producing approximately 64 million tonnes of material from 50 open pits between 1964 and 1988. The Project area includes historical open pits, waste rock stockpiles, tailings management facility, city water ponds, historically impacted areas including the plant site, haulage and service roads, the footprint of the former townsite of Pine Point and an airstrip.

4.6.2 Environmental Studies

The Pine Point project benefits from considerable previous baseline work and studies (published and unpublished) by Cominco Ltd., Tamerlane, Darnley Bay, Avalon, Government Agencies, Pine Point Limited and others. These studies will all be used to support future permitting activities. It is typical for the government agencies to request substantial site-specific back-up data during the permitting process, and that data gathering process is underway. Past environmental and/or baseline studies conducted across the Pine Point District include:

- Waste rock characterization;
- Bathymetry of existing mine pits;
- Dust fall monitoring;
- Air quality monitoring;
- Noise monitoring;
- Vegetation surveys;
- Wildlife surveys, including species at risk;
- Groundwater quality monitoring;
- Surface water quality monitoring;
- Aquatic resources surveys;
- Wetland characterizations;
- Heritage resources surveys;
- Soil surveys;
- Streamflow studies;
- Traditional Knowledge and Socioeconomic studies;
- Demographic data gathering.

4.6.3 Permitting

PPML currently holds a ***Type-A Land Use Permit for Mineral Exploration*** for all of the leases and claims in the Pine Point area. The permit was issued on July 20, 2017, with an expiry date of July 19, 2022.

PPML also currently holds both a ***Type-A Land Use Permit and Type-B Water License for Mineral Exploration*** for confirmation drilling on some of the leases and claims in the Pine Point area. The permit and licence were issued on June 20, 2018, with an expiry date of September 19, 2020.

4.6.4 Social and Community Impact

PPML is actively pursuing its engagement activities in the Pine Point District with local communities, First Nation and Metis groups. PPML has signed “Exploration Agreements” with the First Nation and Metis groups in the Pine Point area that cover the exploration and confirmation drilling programs.

There are three Aboriginal groups that have a potential interest in the Project: the Katlodeeche First Nation, the Deninu Kue First Nation, and the Northwest Territory Metis Nation. Information regarding the exploration activities has been shared with these groups as well as with the local municipal governments and the Northwest Territory government.

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Table 4-2: Details of mineral leases

LEASE No.	LEASE STATUS	ISSUE DATE	TERM EXPIRE	AREA HA	OWNER	LAND CLAIM
NT-4869	ACTIVE	2007-07-16	2028-07-15	251	Pine Point Mining Limited (100%)	AKAITCHO/NWTMN
NT-4866	ACTIVE	2007-05-09	2028-05-08	723	Pine Point Mining Limited (100%)	DEHCHO/NWTMN
NT-4860	ACTIVE	2007-05-09	2028-05-08	86.3	Pine Point Mining Limited (100%)	DEHCHO/NWTMN
NT-4864	ACTIVE	2007-05-09	2028-05-08	936	Pine Point Mining Limited (100%)	DEHCHO/NWTMN
NT-4868	ACTIVE	2007-07-16	2028-07-15	405	Pine Point Mining Limited (100%)	AKAITCHO/NWTMN
NT-4865	ACTIVE	2007-05-09	2028-05-08	837	Pine Point Mining Limited (100%)	DEHCHO/NWTMN
NT-4870	ACTIVE	2007-07-16	2028-07-15	522	Pine Point Mining Limited (100%)	AKAITCHO/NWTMN
NT-4862	ACTIVE	2007-05-09	2028-05-08	887	Pine Point Mining Limited (100%)	DEHCHO/NWTMN
NT-5242	ACTIVE	2011-08-25	2032-08-24	227	Pine Point Mining Limited (100%)	AKAITCHO/NWTMN
NT-5241	ACTIVE	2011-08-25	2032-08-24	249	Pine Point Mining Limited (100%)	AKAITCHO/NWTMN
NT-5240	ACTIVE	2011-08-25	2032-08-24	254	Pine Point Mining Limited (100%)	AKAITCHO/NWTMN
NT-5243	ACTIVE	2011-08-25	2032-08-24	251	Pine Point Mining Limited (100%)	AKAITCHO/NWTMN
NT-5239	ACTIVE	2011-08-25	2032-08-24	249	Pine Point Mining Limited (100%)	AKAITCHO/NWTMN
NT-5260	ACTIVE	2011-08-25	2032-08-24	722	Pine Point Mining Limited (100%)	NWTMN
NT-5249	ACTIVE	2011-08-25	2032-08-24	191	Pine Point Mining Limited (100%)	AKAITCHO/NWTMN
NT-5258	ACTIVE	2011-08-25	2032-08-24	938	Pine Point Mining Limited (100%)	NWTMN
NT-5255	ACTIVE	2011-08-25	2032-08-24	536	Pine Point Mining Limited (100%)	AKAITCHO/NWTMN
NT-5261	ACTIVE	2011-08-25	2032-08-24	89.2	Pine Point Mining Limited (100%)	AKAITCHO/NWTMN
NT-5252	ACTIVE	2011-08-25	2032-08-24	593	Pine Point Mining Limited (100%)	AKAITCHO/NWTMN
NT-5259	ACTIVE	2011-08-25	2032-08-24	786	Pine Point Mining Limited (100%)	NWTMN
NT-5251	ACTIVE	2011-08-25	2032-08-24	419	Pine Point Mining Limited (100%)	AKAITCHO/NWTMN
NT-5250	ACTIVE	2011-08-25	2032-08-24	336	Pine Point Mining Limited (100%)	AKAITCHO/NWTMN
NT-5248	ACTIVE	2011-08-25	2032-08-24	319	Pine Point Mining Limited (100%)	AKAITCHO/NWTMN
NT-5262	ACTIVE	2011-08-25	2032-08-24	165	Pine Point Mining Limited (100%)	AKAITCHO/NWTMN
NT-5247	ACTIVE	2011-08-25	2032-08-24	122	Pine Point Mining Limited (100%)	AKAITCHO/NWTMN

LEASE No.	LEASE STATUS	ISSUE DATE	TERM EXPIRE	AREA HA	OWNER	LAND CLAIM
NT-5257	ACTIVE	2011-08-25	2032-08-24	742	Pine Point Mining Limited (100%)	AKAITCHO/NWTMN
NT-5245	ACTIVE	2011-08-25	2032-08-24	64.8	Pine Point Mining Limited (100%)	AKAITCHO/NWTMN
NT-5246	ACTIVE	2011-08-25	2032-08-24	82	Pine Point Mining Limited (100%)	AKAITCHO/NWTMN
NT-5244	ACTIVE	2011-08-25	2032-08-24	39.4	Pine Point Mining Limited (100%)	AKAITCHO/NWTMN
NT-4858	ACTIVE	2007-05-09	2028-05-08	63.1	Pine Point Mining Limited (100%)	DEHCHO/NWTMN
NT-4859	ACTIVE	2007-05-09	2028-05-08	57.9	Pine Point Mining Limited (100%)	DEHCHO/NWTMN
NT-4861	ACTIVE	2007-05-09	2028-05-08	1,006	Pine Point Mining Limited (100%)	DEHCHO/NWTMN
NT-4863	ACTIVE	2007-05-09	2028-05-08	819	Pine Point Mining Limited (100%)	DEHCHO/NWTMN
NT-4867	ACTIVE	2007-05-09	2028-05-08	1,067	Pine Point Mining Limited (100%)	DEHCHO/NWTMN
NT-4871	ACTIVE	2007-07-16	2028-07-15	692	Pine Point Mining Limited (100%)	NWTMN
NT-4872	ACTIVE	2007-07-16	2028-07-15	241	Pine Point Mining Limited (100%)	NWTMN
NT-4873	ACTIVE	2007-07-16	2028-07-15	76.5	Pine Point Mining Limited (100%)	NWTMN
NT-5253	ACTIVE	2011-08-25	2032-08-24	569	Pine Point Mining Limited (100%)	AKAITCHO/NWTMN
NT-5254	ACTIVE	2011-08-25	2032-08-24	389	Pine Point Mining Limited (100%)	AKAITCHO/NWTMN
NT-5256	ACTIVE	2011-08-25	2032-08-24	546	Pine Point Mining Limited (100%)	AKAITCHO/NWTMN
			TOTAL	**17,548.2**		
			Number	**40**		

Table 4-3: Details of mineral claims

CLAIM No.	CLAIM STATUS	ISSUE DATE	ANNIVERSARY DATE	AREA HA	OWNER	CLAIM NAME	LAND CLAIM
M10296	ACTIVE	2017-02-16	2019-02-16	309	Pine Point Mining Limited (100%)	D8	DEHCHO/NWTMN
M10297	ACTIVE	2017-02-16	2019-02-16	800	Pine Point Mining Limited (100%)	D7	DEHCHO/NWTMN
M10298	ACTIVE	2017-02-16	2027-02-16	579	Pine Point Mining Limited (100%)	D6	AKAITCHO/NWTMN
M10299	ACTIVE	2017-02-22	2027-02-22	502	Pine Point Mining Limited (100%)	D9	AKAITCHO/NWTMN
M10300	ACTIVE	2017-02-22	2027-02-22	943	Pine Point Mining Limited (100%)	D10	AKAITCHO/NWTMN
M10301	ACTIVE	2017-02-22	2027-02-22	466	Pine Point Mining Limited (100%)	D11	AKAITCHO/NWTMN
M10303	ACTIVE	2017-02-22	2027-02-22	155	Pine Point Mining Limited (100%)	D12	AKAITCHO/NWTMN
M10302	ACTIVE	2017-02-22	2027-02-22	136	Pine Point Mining Limited (100%)	D13	AKAITCHO/NWTMN
M10191	ACTIVE	2018-06-22	2020-06-22	92.49	Pine Point Mining Limited (100%)	PPM 1	AKAITCHO/NWTMN
M10192	ACTIVE	2018-06-22	2020-06-22	32.54	Pine Point Mining Limited (100%)	PPM 2	AKAITCHO/NWTMN
M10426	ACTIVE	2018-06-22	2020-06-22	15.26	Pine Point Mining Limited (100%)	PPM 3	AKAITCHO/NWTMN
M10427	ACTIVE	2018-06-22	2020-06-22	4.59	Pine Point Mining Limited (100%)	PPM 4	AKAITCHO/NWTMN
M10653	ACTIVE	2018-06-22	2020-06-22	1.93	Pine Point Mining Limited (100%)	PPM 5	AKAITCHO/NWTMN
M10654	ACTIVE	2018-06-22	2020-06-22	35.2	Pine Point Mining Limited (100%)	PPM 6	AKAITCHO/NWTMN
M10658	ACTIVE	2018-06-22	2020-06-22	21.53	Pine Point Mining Limited (100%)	PPM 7	AKAITCHO/NWTMN
M10659	ACTIVE	2018-06-22	2020-06-22	6.94	Pine Point Mining Limited (100%)	PPM 8	AKAITCHO/NWTMN
M10660	ACTIVE	2018-06-22	2020-06-22	5.88	Pine Point Mining Limited (100%)	PPM 9	AKAITCHO/NWTMN
K15913	ACTIVE	2011-11-28	2023-11-28	167.48	Pine Point Mining Limited (100%)	D 1	AKAITCHO
K15914	ACTIVE	2011-11-28	2023-11-28	223.33	Pine Point Mining Limited (100%)	D 2	AKAITCHO
K15915	ACTIVE	2011-11-28	2023-11-28	67.09	Pine Point Mining Limited (100%)	D 3	AKAITCHO
K15916	ACTIVE	2013-03-27	2025-03-27	41.94	Pine Point Mining Limited (100%)	D 5	AKAITCHO/NWTMN
K15917	ACTIVE	2013-03-27	2025-03-27	59.08	Pine Point Mining Limited (100%)	D 4	AKAITCHO/NWTMN
			TOTAL	**4,665.28**			
			Number	**22**			

Table 4-4: Details of surface leases

REGISTRY NUMBER	LEASE NAME	LEASE NUMBER	ANNIVERSARY DATE	ANNUAL LEASE RENTAL	JURISDICTION
KM 42, NWT Highway No. 5	Settling Pond	85B/11-15-2	2029-June-30	$2,423.00	Northwest Territories Lands
KM 42, NWT Highway No. 5	Mine Site	85B/11-16-2	2029-June-30		Northwest Territories Lands
KM 42, NWT Highway No. 5	Settling Pond – Indenture	85B/11-18-2	2029-June-30	$300.00	Indigenous and Northern Affairs
KM 42, NWT Highway No. 5	Mine Site - Indenture	85B/11-19-2	2029-June-30		Indigenous and Northern Affairs

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

5.1 Accessibility

5.1.1 Road Access

The Pine Point project is accessible via paved highways from local communities, such as Hay River, NT, Fort Smith, NT, and Fort Resolution, NT, and larger cities, such as Edmonton, AB, and Yellowknife, NT. Access from Edmonton is via approximately 1,100 km of provincial and territorial highways (AB-44, AB-88, AB-58, AB-35, NT-2, NT-5 and NT-6); access from Yellowknife is via approximately 560 km of territorial highways (NT-1, NT-2, NT-5, NT-6).

The Property lays north of and roughly parallels territorial highways NT-5 and NT-6, beginning 42 km east from Hay River, NT, and extending to 110 km from Hay River. Access within the Property is via a network of paved roads and 101 km of all-season haul roads built to service the various open pit and underground mines that were worked by Cominco Ltd. between 1964 and 1988.

Off-road access of the Property varies according to the season. Parts of the Property become waterlogged during the summer due to poor drainage of muskeg swamps, but are accessible during winter months using snowmobiles, tracked vehicles, and other all-terrain vehicles.

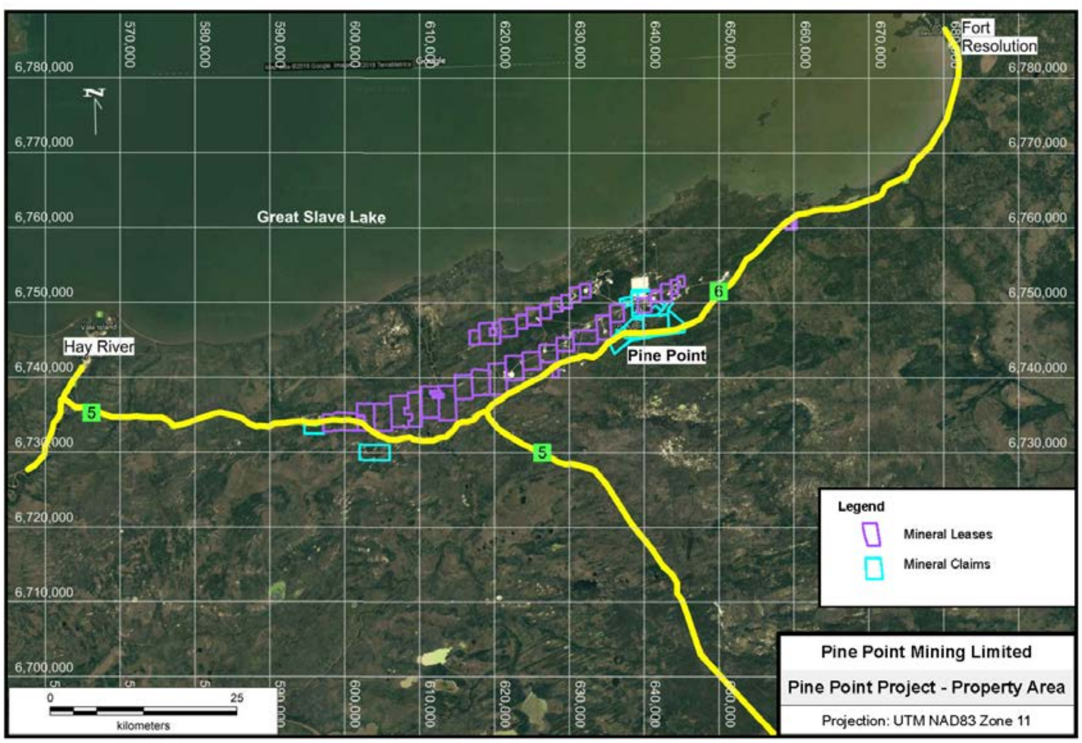


Figure 5-1: Location of the Pine Point Property

5.1.2 Climate

The Pine Point project area experiences a subarctic climate, characterized by short, cool summers and long, cold winters. The nearest permanent weather monitoring station maintained by Environment Canada (climat.meteo.gc.ca) is the Fort Resolution A station, approximately 84 km northeast of the Property. According to the available data collected at this weather station from 1980-2017 (2012 data was unavailable), the daily average temperature for January was -23.3°C, and the daily average temperature in July was 16.0°C. The record low during this period was -49.2°C, and the record high was 36.0°C.

Data collected from the Fort Resolution A weather station from 1930 to 2007 indicates that the total annual precipitation was 305.7 mm, with peak rainfall occurring during July (37.3 mm average), August (36.9 mm average), and September (37.6 mm average). Snowfall is light to moderate, with an annual average of 149.0 cm. Snow typically accumulates from October to April, with a peak snowfall occurring in November (45.5 cm average); during this period, snowpack averages 79 cm depth, with a maximum depth of approximately 104 cm. On average, the Property is frost-free for 95 days, though discontinuous permafrost exists in the area. Hours of sunlight vary from 19.3 hours at the summer solstice in June to 5.5 hours at the winter solstice in December.

The climatic conditions at the Property do not significantly impede the Project or hinder exploration or mining activities, beyond seasonal consideration for certain work (e.g., drilling muskeg swamps during winter freeze).

5.2 Local Resources and Infrastructure

5.2.1 Airports, Rail Terminals and Ports

The town of Hay River, with a population of approximately 3,700 residents, is located 42 km west of the western boundary of the Pine Point property along territorial highways NT-2, NT-5, and NT-6. Hay River, known as “the Hub of the North”, has all major services including an airport with scheduled service from Edmonton and Yellowknife, a rail terminal, and a port from which barge traffic traverses the Mackenzie and Slave Rivers.

The hamlet of Fort Resolution, NT, with a population of approximately 470 residents, is located 70 km northeast of the Pine Point project along territorial highway NT-6. Fort Resolution has a small airport with year-round access.

An airstrip suitable for small aircraft exists near the former Pine Point town site. It is currently unmaintained.

Rail service historically extended to the project area, but was discontinued after operations ceased. The tracks and bridges were dismantled, but the rail bed is still present and in good condition.

5.2.2 Local Work Force

According to the 2017 census prepared by the NWT Bureau of Statistics, the population of the town of Hay River was 3,734 people, nearly 8.4% of the total population of the Northwest Territories; 51% of the population is male, 49% is female, and 46% is Aboriginal. In 2016, 72.5% of the population participated in the labour force, with 40.3% employed in industries other than government, health, social services, education, or producing goods. A portion of the workforce is experienced in mining operations, as they are currently employed at diamond mines located elsewhere in the NWT.

The 2016 census by Statistics Canada of the hamlet of Fort Resolution gives a population of 470 people, with 57% of the residents aged 20-64, and an average age of 37.2 years. Male population accounts for 53% of the population, 47% is female, and 91% is Aboriginal. The participation rate of the population in the labour force was 58.7%, with 4% of the labour force employed in "mining, quarrying, and oil and gas extraction".

The 2016 census by Statistics Canada for the Town of Fort Smith listed a population of 2,542 people, with 61% of the residents aged 20-64, and an average age of 36.4 years. Additionally, the Aurora College has their largest campus in Fort Smith. The Thebacha Campus has more than 300 full time students and the college offers training in heavy equipment operations and supports the mining industry in the Northwest Territories.

5.2.3 Additional Support Services

Additional services within the town of Hay River include the H.H. Williams Memorial Hospital, an RCMP detachment, grocery stores, fuel stations, financial institutions, and hotels. Hay River has a Canada Post office, and additional shipping/freight services by several providers. Landline telephone, mobile service, high-speed internet, and satellite internet are available in town and the nearby vicinity.

Services in the hamlet of Fort Resolution include an RCMP station, a general store, and a motel. The cities of Yellowknife and Edmonton have all conventional services available expected for communities of their size.

Hydroelectric power generated by the Taltson Dam is supplied to the communities of Fort Smith, Hay River, Hay River Reserve, Fort Resolution, and Enterprise by the Northwest Territories Power Company ("NTPC"), with a substation located on the Property approximately 4 km northeast of the historic Pine Point town site. The Taltson Dam currently provides 18 MW and has the flow capacity to expand production to over 56 MW.

Pine Point Mining Limited, an Osisko Metals subsidiary, maintains an exploration office in the town of Hay River, which includes a large, well-equipped core logging area. Core is stored short-term at the exploration office, and long-term in a nearby fenced lot.

There is no concentrator currently present on the project, as Cominco Ltd. removed all buildings after operations ceased in 1988.

5.3 Physiography

A blanket of glacial debris produces a low-lying, hummocky terrain on the Property that slopes gently northward toward Great Slave Lake.

The Buffalo River flows northward through the project area and into Great Slave Lake. Other drainage by streams within the project area is virtually non-existent. With such poor drainage, the area is dominated by muskeg swamp and numerous small ponds. Several east-west trending sand and gravel ridges mark ancient shorelines of Great Slave Lake. The project area lies approximately 60 m (196 feet) above the level of Great Slave Lake, which itself is at an elevation of 156 m (512 feet) above sea level.

The Pine Point project area is part of the Boreal Plain terrestrial ecozone, it is predominantly muskeg swamp, with sparse to dense tree cover consisting predominantly of balsam poplar, black spruce, jack pine, tamarack, and white spruce.

6. HISTORY

6.1 Prior to 1920

Lead-zinc showings south of the Great Slave Lake were known to the local First Nations long before any mineral claims had been staked in the area. They had been using lead obtained from the showings to fashion musket balls. In 1920, evidences of galena smelting, ashes and blobs of lead were present around the mineralized outcrops (Dawson, 1963).

It was during the Klondike Gold Rush, when groups of prospectors bound to the Yukon passed through Fort Resolution, the local fur trader Ed Nagle started asking First Nations who traded at his post to bring any “shining stones” they might find.

During the summer of 1898, massive galena samples were brought by a group of Slavey First Nations. Later that summer, one of them led Nagle to the showing. The location of the showing was described as few miles inland from the Ile du Mort on the Great Slave Lake shore. Nagle staked eight claims over the showings and collected some galena samples. Nagle’s claims centered on what has become known as the historical P-32 deposit. In 1899, Nagle hired two prospectors to sink a 20-foot-deep shaft and collect samples at three different depths. Samples were sent to a lab in Vancouver, to the Department of Mines office in Ottawa, and an assay lab in Seattle. Since the assays did not reveal any silver or gold, Nagle allowed the claims to lapse after 3 years.

Dr. Robert Bell of the Geologic Survey of Canada visited the showings in 1899 and reported the mineralization as “occurring in Devonian limestones adjacent to numerous sinkholes.” Mineralization was described as galena crystals scattered in limestone over an area of several acres. At one place, where galena was mixed with blende (i.e. sphalerite), it was concentrated in “bunches” several feet in horizontal diameter. Dr. Bell stressed that many assays had confirmed that mineralization contained only traces of silver and that mining base metals in that location was not economical due to the remoteness of the area (Bell, 1900).

Showings were staked again in 1908 by John Erickson and named “Paragon claims”. Erickson did not work the claims but he kept them in good standing into the 1920s. In 1914, showings were over staked by British mining engineer Gwynn Gibbins; who did some limited work in the summer of 1914. Gibbins was later killed in World War I and his claims lapsed (Nagle and Zinovich, 1989).

Various field parties from the Geological Survey of Canada had visited the lead-zinc showings prior to 1920: Camsell in 1914, and Cameron in 1916 and 1917. Cameron identified the mineralized host-rock as coarse-crystalline vuggy dolomite with cavities occupied by curved rhombohedral dolomite crystals (‘saddle dolomite’). The geological age of Pine Point and Sulphur Point outcrops was determined to be middle Devonian, while the age of the outcrops along Hay River north of Alexandra Falls was established as Upper Devonian (Cameron, 1918). Geological map of the Mackenzie River Basin published in 1921, shows that the entire shore of the Great Slave Lake and the banks of all navigable rivers in the area had been mapped.

In 1920, James Mackintosh Bell, who had visited the Pine Point showings in 1900 with his uncle Dr. Robert Bell, partnered with Messrs. Paine, Weber & Company and Professor H.L. Smythe of Harvard University together forming the Boston Syndicate. In 1921, the Boston Syndicate sent geologist C.B. Dawson to Pine Point to re-stake the claims Gibbins had staked in 1914. In addition to claim staking, a new 25 ft shaft was sunk and several test pits were excavated on various showings. A new access trail to Pine Point was cut and a log cabin was built at the showings. A financial agreement was made with Erickson to secure Boston Syndicate's control over Paragon claims.

6.2 1920 to 1960

In 1927, W.M. Archibald, Manager of Mines with Consolidated Mining and Smelting Company (then CM&S, from 1966 known as Cominco Ltd.), sent a geologist, W.L. MacDonald, and Ted Nagle (son of the original claim staker Ed Nagle) to Pine Point in recognisance and to collect mineralized samples. This was the beginning of Cominco Ltd.'s involvement with Pine Point.

In the spring of 1928, J.M. Bell and C.B. Dawson, now representing Atlas Exploration Company, brought machinery to Pine Point for shaft sinking and other supplies to last a full season of work. Even before they arrived in Pine Point, a staking rush started: 16 claims surrounding Atlas claims had been staked by Cominco Ltd. At least four other groups staked claims in the Pine Point area. One of them, General Exploration Company, staked the area now known as the historical T-37 deposit, located east northeast of the North Trend, where galena mineralization was exposed in karst sinkholes. In 1929-1930, several test pits were excavated there (Meikle 1930a, 1930b). General Exploration Company staked or acquired interest in a total of 270 claims, but the T-37 showing received the most exploration.

In 1929, a joint-venture was formed by CM&S (Cominco Ltd.), Ventures Limited, and Atlas Exploration Group (J.M. Bell and C.B. Dawson). In 1930, joint venture partners formed Northern Lead and Zinc Company Limited with a controlling interest held by Cominco Ltd. Soon thereafter, the holdings of other companies in the area, including General Exploration Company Limited, were merged into the Northern Lead and Zinc Company property. Combined holdings at Pine Point now consisted of 403 licensed and 45 optioned mineral claims.

An extensive exploration program of churn drilling and shaft sinking was undertaken between 1929 and 1930. About 21,600 ft of churn drilling was completed. Additional diamond drilling conducted in the summer of 1930 and totalled 2,900 ft. By the end of the 1929-1930 exploration program, five occurrences of significant grades had been outlined, and two mineralized trends, three miles apart, had been identified. Numerous similarities between the Pine Point and MVT deposits in the Tri-State region of the Mississippi Valley were observed at this time (Bell, 1929).

The Great Depression and the realization that a mine would not be feasible without an adequate transportation route led to the temporary cessation of work. From 1930 until 1948, Cominco Ltd. carried out only enough work to maintain 104 claims in the area.

In 1947 a major exploration program, which required seven years of work (1948-1955) was initiated to test the stratigraphic control of the mineralization that was characteristic of classic Mississippi Valley-type deposits and to also test a speculated structural control by major Precambrian faults in the East Arm of Great Slave Lake that could be projected southwesterly beneath the younger rocks into the Pine Point area.

For this regional exploration program, Cominco Ltd. obtained a 500-square mile concession surrounding the area of known mineralization in 1948. A second concession was obtained the following year. A fence drilling program totalling over 60,045 m (197,000 ft) was completed between 1948 and 1953. The program successfully located a number of lead-zinc deposits several kilometers from the previous surface discoveries, all covered by overburden.

In order to obtain bulk mineralized samples, shafts were sunk into the historical N-42 and M-40 deposits in 1954. The historical N-42 shaft was sunk in the summer of 1954 to a depth of 98 ft. Mineralization was encountered at a depth of 35 ft with mineralization continuing to the shaft bottom. The M-40 shaft was sunk to a depth of 162 ft in 1954. A level was cut at the 145-foot level and lateral work totalling 661 ft were undertaken during the winter of 1954-1955. Heavy water inflow halted further work in both shafts (Silke 2009).

6.3 1960 to 2000

In 1961, under the “Roads to Resources” program, an agreement was reached between the Federal Government, Pine Point Mines Limited (subsidiary of Cominco Ltd. formed to finance Pine Point mine production) and Canadian National Railways whereby the Government undertook the construction of the railway to Great Slave Lake. Cominco Ltd. constructed a mine at Pine Point. The Northern Canada Power Commission agreed to build a 25,000 horsepower (approximately 18.6 megawatt) hydroelectric plant on Taltson River to supply power to Pine Point.

In 1963, a townsite was laid out in collaboration with the Department of Northern Affairs and Mineral Resources. Cominco Ltd. built 53 homes, two 50-men bunkhouses, a recreation hall, and water and sewage systems.

Shipments of high-grade material averaging 50% combined lead-zinc to Cominco Ltd.’s smelter in Trail started in 1964 and full mine production at a rate of 248,000 tons of concentrate per year, began in 1965.

During 1963-1964, a massive staking rush occurred as prospectors and companies sought claims adjoining the Pine Point property. Late in 1965, Pyramid Mining Co. Ltd. found a major deposit to the east of Pine Point's ground. In 1966, Pine Point Mines Ltd. acquired Pyramid's mineral claims in the area. The new deposit was developed into the X-15 pit, which eventually produced 17,474,260 tonnes of material at 2% lead and 6.2% zinc. Pyramid Mining also discovered W-17 deposit near X-15. Other discoveries made in 1966 include: A-55 deposit on the Buffalo River Exploration property, R-61 and S-65 deposits on the Coronet claims and YBM deposit on the Yellowknife Base Metals property (Thorpe, 1972). Pine Point Mines Ltd. purchased these properties (Gibbins et al., 1977).

In 1975, Western Mines (later known as Westmin Resources Ltd.), acquired claims west of Cominco Ltd.'s property, which is essentially west of the Buffalo River. Westmin then proceeded to conduct an extensive Induced Polarization ("IP") survey and drilling program from 1976 to 1981. The exploration program was referred to as "The Great Slave Reef ("GSR") Project". This project was a joint venture of Westmin, controlled by Boliden of Sweden, DuPont Exploration Canada and Phillip Brothers. Drilling programs conducted between 1975 and 1981 outlined seven additional lead-zinc deposits on the GSR property. Westmin drilled 885 holes totalling 154,816 m from 1975 to 1981.

Throughout the mine production, between 1964 and 1986, induced polarization geophysical surveys and grid diamond drilling were regarded as the two main exploration tools. Drilling IP anomalies discovered the majority of deposits at Pine Point. A total of 4,000 km of IP surveys have been conducted on Pine Point ground since the inception of surveying in 1964 to 1983 (Rhodes et al., 1984).

More than 10,000 drillholes totalling over 610,000 m had been drilled since 1948. Drillhole spacing varied across the property and most of the property had been tested by 100 m-300 m deep vertical holes drilled to the top of the Keg River Formation (E shale marker) on 915 m x 915 m or 915 m x 1,830 m grids. The mineralized trends on the eastern half of the property are completely covered by 300 m x 300 m and in many instances, by 150 m x 150 m grids of shallower drillholes (30 m-150 m deep). The individual deposits are drilled at spacings varying from 20 m to 35 m (Rhodes et al., 1984). At the time of shutdown, Cominco Ltd. had drilled 17,401 holes totalling 1,142,150 m in the period from 1930 to 1986.

A program of regional fence drilling designed to intersect the E shale, a prominent marker layer at the base of the favourable stratigraphy started in 1979 and a new geological model of the reef facies and associated karst, alteration and mineralization features was developed in later years. Greater focus was also placed on the discovery of new shallow high-grade tabular deposits along the North Trend.

Fifty-two of the 100 drill-defined deposits were brought into production in the Pine Point district. Total production from 1964 to 1988 was approximately 64,300,000 tonnes at an average grade of 3.10% Pb and 7.00% Zn. Annual mine production and reserves from 1964 to 1983 are tabulated in Table 6-1. All but two deposits were mined as open pits for the total of 50 open pits.

Size of the pits varied between 45,000 and 17,500,000 tonnes, but most of the pits were between 200,000 and 3,500,000 tonnes. There was only one pit, X-15, larger than 3,500,000 tonnes that remained in production for 12 years, from 1969 to 1979. The pits were from 500 ft to 2,800 ft wide, 100 ft to 300 ft deep, with 25-foot benches and a 45° slope (Silke, 2009).

Two deposits were mined as underground operations, the M-40 and Y-65. M-40 was in production from 1975 to 1977 and produced 350,870 tonnes of material grading 2.2% lead and 5.5% zinc. Y-65 deposit was mined in 1984 and 1985 and produced 148,770 tonnes of material grading 7.0% lead and 12.9% Zinc (Giroux and McCartney, 2004).

Table 6-1: Pine Point Mines production from 1964 to 1983. Compiled from Pine Point Mines Limited Annual Reports 1964-1983. (Rhodes et al, 1984, conversion tons to tonnes in Hannington, 2007).

Deposit	Production (tonnes)	Grade			Metal Produced		
		% Pb	% Zn	% Zn + Pb	Pb (tonnes)	Zn (tonnes)	Zn + Pb (tonnes)
NORTH TREND - listed from northeast to southwest							
X-17	44,910	1.5	6.3	7.8	674	2,829	7.8
T-37	358,960	2.1	6.3	8.4	7,538	22,614	8.4
X-51	1,203,980	2.2	6.7	8.9	26,488	80,667	8.9
Y-53	967,710	1.5	5.6	7.1	14,516	54,192	7.1
X-52	1,104,080	1.6	6.3	7.9	17,665	69,557	7.9
X-53	1,231,940	2.7	9.2	11.9	33,262	113,338	11.9
Z-53	380,520	1.4	5	6.4	5,327	19,026	6.4
A-55	1,550,830	3	7.6	10.6	46,525	117,863	10.6
Y-54	263,840	1.3	4	5.3	3,430	10,554	5.3
X-54/X-55	216,130	2.1	6.7	8.8	4,539	14,481	8.8
X-56/X-57	1,319,580	1.6	6.3	7.9	21,113	83,134	7.9
Z-57	827,870	1.1	4.2	5.3	9,107	34,771	5.3
Y-65	149,770	7	12.9	19.9	10,484	19,320	19.9
Y-60	512,490	2.1	7.3	9.4	10,762	37,412	9.4
Y-61	549,040	3.5	9.3	12.8	19,216	51,061	12.8
Z-64	913,470	1.4	5.1	6.5	12,789	46,587	6.5
A-70	2,289,360	4.5	10.4	14.9	103,021	238,093	14.9
MAIN TREND – listed from northeast to southwest							
P-24	496,640	3.5	7.6	11.1	17,382	37,745	11.1
L-30	262,170	1.1	2.8	3.9	2,884	7,341	3.9
O-28	1,483,870	2	3.7	5.7	29,677	54,903	5.7
P-29	476,120	1.6	3.3	4.9	7,618	15,712	4.9
N-31	505,200	1.6	4.1	5.7	8,083	20,713	5.7
P-31	604,760	2.2	3.6	5.8	13,305	21,771	5.8
P-32	694,980	3.2	3.5	6.7	22,239	24,324	6.7
O-32	375,970	2.8	6.4	9.2	10,527	24,062	9.2
N-32	1,862,070	3.4	8.4	11.8	63,310	156,414	11.8
L-37	3,417,550	1	3.4	4.4	34,176	116,197	4.4
N-38	1,182,110	4.9	7.4	12.3	57,923	87,476	12.3
N-42	2,959,680	5.3	9.5	14.8	156,863	281,170	14.8
O-42	2,742,720	8.8	11.6	20.4	241,359	318,156	20.4
P-41	196,140	2.1	8.3	10.4	4,119	16,280	10.4
M-40	350,870	2.2	5.5	7.7	7,719	19,298	7.7
J-44	1,282,230	5.9	9.8	15.7	75,652	125,659	15.7
I-46	389,870	5.1	4.2	9.3	19,883	16,375	9.3
M-52	455,260	3.5	7.6	11.1	15,934	34,600	11.1
K-53	468,900	3.7	9.3	13	17,349	43,608	13
K-57	1,564,540	6.5	5.2	11.7	101,695	81,356	11.7
K-62	1,001,590	3.6	4.8	8.4	36,057	48,076	8.4
I-65	194,510	3.8	11.1	14.9	7,391	21,591	14.9
M-64	178,460	4.9	8	12.9	8,745	14,277	12.9
R-61	1,034,540	1.6	5.2	6.8	16,553	53,796	6.8
J-69	854,770	1.2	5.2	6.4	10,257	44,448	6.4
K-77	511,120	6.4	6.4	12.8	32,712	32,712	12.8
N-81	2,699,950	7	14.1	21.1	188,997	380,693	21.1
SOUTH TREND - listed from northeast to southwest							
X-15	17,474,260	2	6.2	8.2	349,485	1,083,404	8.2
W-17	3,515,400	2	6.1	8.1	70,308	214,439	8.1
T-58	563,310	4.5	12.6	17.1	25,349	70,997	17.1
R-61	1,034,540	1.6	5.2	6.8	16,553	53,796	6.8
S-65	575,550	1.2	5.7	6.9	6,907	32,806	6.9
TOTAL	**64,294,130**	**3.10%**	**7.00%**	**10.10%**	**2,023,000**	**4,569,671**	**6,593,139**

As mining proceeded to the west and deposits were found at greater depths, mine development was hampered by the higher operational costs related to the increased stripping ratio, haulage distance to the mill, and increased groundwater pumping requirements. Low base metal prices, high power consumption related to mine dewatering, and the acquisition of the Red Dog deposit with nearly double the average grades and better mining characteristics, prompted Cominco Ltd. to close its Pine Point mining operation in 1986. Processing of stockpiled material continued till 1988. Reclamation of the mine site was completed in 1991 and included removal of the concentrator, townsite and railroad (Giroux and McCartney, 2004).

6.4 2000 to Today

By August 2001, all Cominco Ltd. and Westmin claims had been allowed to lapse and all mining leases had expired. Prospective parts of the district, including much of the Pine Point Mines production area, and the geologic trend to the west, in the former Westmin property were staked shortly after by Ross Burns on behalf of Kent-Burns Group (later Karst Investments LLC).

In 2004, the claims were optioned by Tamerlane Ventures Inc. and then in 2006 Tamerlane acquired a 100% interest in the claims, subject to a 3% NSR to Karst. Between 2005 and 2010, Tamerlane carried out several confirmation and exploration drilling programs.

In 2002 and 2003, Tamerlane initiated an extensive work program on the Pine Point project. This work consisted of compiling available Cominco Ltd. and Westmin digital and hard copy data for the extensive inventory of developed and undeveloped deposits, across the property. This was followed by some efforts at geologic interpretation, drill core reviewing, reported geological cross-section generation across important mineral deposits. Tamerlane's initial work also included a preliminary internal mining study where the Pine Point development economics, in 2002 terms, were analyzed.

Geological data for up to 18,200 drillholes from the Cominco Ltd. and the Westmin eras was entered into Gemcom. This data consisted of down-hole surveys, geology, hydrothermal alteration information, assay and geochemistry, principally for the area covering the historical Westmin claims and survey data and varied geological and diamond drilling data for the larger eastern portion of the property covering the historical Cominco Ltd. claims.

In addition to the drilling program, Tamerlane undertook an airborne magnetic and electromagnetic (AeroTEM© II time-domain) survey in 2005.

In addition to the former Cominco Ltd. claims, Tamerlane considered the historical Westmin property to hold significant exploration potential. Exploration and diamond drilling guided by IP was successful west of the Buffalo River. Nine deposits were outlined by Westmin between 1976 and the early 1980's. The nine deposits are named from west to east: O-555, O-556, P-499, R-190, T-799, V-46, W-19, X-25 and Z-155. The X-25 deposit is the largest with R-190 and Z-155 containing very high-grade massive sulphides.

In 2005, Tamerlane carried out diamond drilling on three Pine Point deposits. This exploration and confirmation drilling focused on the W-85 deposit in the North Trend, the G-03 deposit in the western part of the former Cominco Ltd. Pine Point mine property, and the R-190 deposit west of the Buffalo River where Tamerlane carried out considerable work.

Environmental baseline studies conducted in 2005 and 2006 by EBA Engineering Consultants Ltd. included: water quality and stream assessment, vegetation/ecosystem studies, rare plant survey, wildlife surveys, water quality sampling program. Assessment of the ground freezing technology and Desktop Evaluation of Natural Groundwater Flow Velocities were done in 2006.

In July 2008, Tamerlane published a mineral resource estimate and a mineral reserve estimate in a Technical Report by Collins et al. (2008). This historical estimation was done on a series of deposits that are part of the current MRE. That being said, Osisko Metals has completed additional work in this area with results reported by the Author in Chapters 9, 10 and 14. The current MRE is provided in Chapter 14 of this Report and supersedes the report by Collins et al. (2008). The Author of the current Technical Report has read the documents pertaining to the description of the different methods used in the historical evaluation of the resources/reserves. The Author, also acting as the QP, has not done sufficient work to classify the historical estimate as current mineral resources or mineral reserves. The Author and Osisko Metals are not treating this historical estimate as current mineral resources or mineral reserves as defined in NI 43-101 and such historical estimate should not be relied upon. The categories of the Collins et al. (2008) report were classified under previous definition standards and do not match the current (2014) classification as defined in NI 43-101. More specifically, the category “probable reserve” was classified under previous definition standards and does not match the current (2014) classification. There are currently no mineral reserves classified for the Project.

Historical mineral resource estimates for R-190, X25, G-O3, P-499, O556, Z155 deposits were estimated and reported in 2008 by Pincock, Allen & Holt (PAH) Collins et al. (2008) . Their work was reviewed by this Author and a summary is presented in Table 6-2 (Resource Estimate) and Table 6-3 (Reserve Estimate). Table 6-4 summarizes the Life of Mine cut-off grade inputs for the Tamerlane 2008 historical Reserve Estimate

The deposits are prismatic in type and followed generally accepted modelling procedures at the time. Collins et al. (2008) found no capping was required, models were completed using a geological boundary corresponding to approximately 1% Pb+Zn. Inverse distance square (“ID^2”) was used for estimation. This historical reserve estimate was given an appropriate dilution.

Table 6-2: Tamerlane 2008 historical Mineral Resource Estimate for deposits R-190, X25, G-O3, P-499, O-556, Z-155 (Collins et al., 2008)

Deposit Name	Category	Cutoff Grade, % Zn	Tonnes	Lead Grade, %	Zinc Grade, %
R-190	Measured	>1%	472,000	1.84%	2.68%
R-190	Indicated	>1%	1,144,000	1.18%	2.74%
R-190	TOTAL	>1%	1,615,000	1.37%	2.72%
P-499	Measured	>1%	43,000	0.84%	2.12%
P-499	Indicated	>1%	531,000	0.76%	1.87%
P-499	TOTAL	>1%	574,000	0.77%	1.89%
O-556	Measured	>1%	138,000	1.86%	1.88%
O-556	Indicated	>1%	664,000	1.32%	1.92%
O-556	TOTAL	>1%	802,000	1.41%	1.91%
X-25	Measured	>1%	2,195,000	0.98%	2.37%
X-25	Indicated	>1%	1,026,000	0.67%	2.46%
X-25	TOTAL	>1%	3,222,000	0.88%	2.40%
Z-155	Measured	>1%	113,000	0.80%	1.69%
Z-155	Indicated	>1%	292,000	0.96%	1.90%
Z-155	TOTAL	>1%	406,000	0.92%	1.84%
G-03	Measured	>1%	394,000	2.08%	2.14%
G-03	Indicated	>1%	1,018,000	1.24%	1.76%
G-03	TOTAL	>1%	1,412,000	1.47%	1.87%
TOTAL	**Measured**	>1%	**3,355,000**	**1.26%**	**2.34%**
TOTAL	**Indicated**	>1%	**4,675,000**	**1.04%**	**2.20%**
TOTAL	**Measured + Indicated**	>1%	**8,030,000**	**1.13%**	**2.26%**

Deposit Name	Category	Cutoff Grade, % Zn	Tonnes	Lead Grade, %	Zinc Grade, %
R-190	Inferred	>1%	319,000	1.08%	2.86%
P-499	Inferred	>1%	519,000	0.94%	2.62%
O-556	Inferred	>1%	627,000	0.88%	1.97%
X-25	Inferred	>1%	1,873,000	0.54%	2.52%
Z-155	Inferred	>1%	331,000	0.84%	2.18%
G-03	Inferred	>1%	459,000	1.52%	1.74%
TOTAL	**Inferred**	>1%	**4,128,000**	**0.82%**	**2.36%**

Table 6-3: Tamerlane 2008 historical Reserve Estimate for deposits R-190, X25, G-O3, P-499, O-556, Z-155 (Collins et al., 2008)

Deposit Name	Category	Cutoff Grade, % Zn	Tonnes	Lead Grade, %	Zinc Grade, %
R-190	Proven	3.5%	647,000	6.10%	12.47%
R-190	Probable	3.5%	357,000	3.79%	8.27%
R-190	TOTAL	3.5%	1,005,000	5.28%	10.98%
P-499	Proven	2.5%	183,000	4.07%	7.78%
P-499	Probable	2.5%	709,000	2.32%	5.38%
P-499	TOTAL	2.5%	892,000	2.68%	5.87%
O-556	Proven	2.0%	434,000	3.47%	3.92%
O-556	Probable	2.0%	596,000	2.11%	3.49%
O-556	TOTAL	2.0%	1,030,000	2.68%	3.67%
X-25	Proven	2.5%	1,625,000	2.61%	7.21%
X-25	Probable	2.5%	483,000	1.33%	5.11%
X-25	TOTAL	2.5%	2,108,000	2.32%	6.73%
Z-155	Proven	2.5%	426,000	3.54%	5.89%
Z-155	Probable	2.5%	349,000	1.74%	3.96%
Z-155	TOTAL	2.5%	775,000	2.73%	5.02%
G-03	Proven	2.0%	961,000	3.57%	6.09%
G-03	Probable	2.0%	1,019,000	2.52%	3.91%
G-03	TOTAL	2.0%	1,980,000	3.03%	4.97%
TOTAL	**Proven**		4,276,000	**3.60%**	**7.31%**
TOTAL	**Probable**		3,514,000	**2.30%**	**4.75%**
TOTAL	**Proven + Probable**		7,790,000	**3.01%**	**6.16%**

Table 6-4: Tamerlane summary of life of mine cut-off grade inputs for the Tamerlane 2008 historical Reserve Estimate (Collins et al., 2008)

Description	UNITS	Value
Ore Production rate	tonne/day	**2,800**
Recovered Metals		
Zinc	lbs (000s)	**836,605**
Lead	lbs (000s)	**435,818**
Estimated Total Operating Costs		
Mining	US$/ore tonne	$ 30.25
DMS-Process	US$/ore tonne	$ 10.43
Power - Diesel	US$/ore tonne	$ 9.18
G&A	US$/ore tonne	$ 1.40
Environmental-Service	US$/ore tonne	$ 0.46
Total LOM Unit Operating Cost Per DMS Feed Ore Tonne	US$/ore tonne	**$ 51.73**
Overall Process Recoveries (DMS + Flotation)		
Zinc (%)	%	93.6
Lead (%)	%	88.9
Flotation Concentration		
Zinc (%)	%	61.8%
Lead (%)	%	71.8%
Metal Prices[1]		
Zinc	US$/lb	$ 1.15
Lead	US$/lb	$ 0.96
Equivalent Zinc	US$/lb	$ 1.58
Credits		
Lead as a credit to zinc	US$/lb Zn	$ 0.43
Royalty (3%)	US$/lb Zn	$ 0.04
Forward Recoveries and Costs (from Tamerlane 7/30/2008 cash flow)		
Smelter Payables		
Zinc Percent	%	85%
Lead Percent	%	95%
Equivalent Zinc Percent	%	8124%
Freight (US$153.80/tonne wet concentrate)	US$/ore tonne	$ 20.50
Zinc Smelting (US$262.82/tonne concentrate)	US$/ore tonne	$ 23.15
Lead Smelting (US$203.15/tonne concentrate)	US$/ore tonne	$ 7.18
Subtotal Forward Costs	US$/ore tonne	**$ 50.84**
Total Costs per ore tonne	**US$/ore tonne**	**$ 102.57**

In 2010, Tamerlane drilled 23 confirmation holes totalling 1,433 m at the N-204 deposit.

In 2011, 1,821 m in nine drillholes were drilled at the R-190 deposit for geotechnical studies. Between 2005 and 2011, Tamerlane Ventures drilled a total of 11,726 m in 106 drillholes at the Pine Point property (Gann, 2016).

The same year, in March, a mineral resource estimate was completed on the N-204 deposit by Pincock, Allen, & Holt (Horlacher, 2011). This historical estimation was done on a deposit that is part of the current MRE. That being said, Osisko Metals has completed additional work in this area with results reported by the Author in Chapters 9, 10 and 14. The current MRE is provided in Chapter 14 of this Report and supersedes the report by Horlacher (2011). The Author of the current Technical Report has read the documents pertaining to the description of the different methods used in the historical evaluation of the resources/reserves. The Author, also acting as the QP, has not done sufficient work to classify the historical estimate as current mineral resources or mineral reserves. The Author and Osisko Metals are not treating this historical estimate as current mineral resources or mineral reserves as defined in NI 43-101 and such historical estimate should not be relied upon. The categories of the Horlacher (2011) report were classified under previous definition standards and do not match the current (2014) classification.

Horlacher (2011) reports that the mineralized grade zones were interpreted based on a grade of 0.9% combined Pb+Zn. High-grade values (outliers) for Zn, Pb and Fe were cut to 25%, 6.5%, and 26.8%, respectively, prior to compositing. ID^2 was used for estimation. The selected block cut-off grade was 1.1% zinc-equivalent. No dilution was applied. A summary is presented in Table 6-5 below.

Table 6-5: Tamerlane 2011 historical Mineral Resource Estimate for deposit N-204 (Horlacher, 2011)

Resource Class		Main Zone	Upper Zone	Total
Indicated*	Tonnes	8,464,393	701,545	9,165,938
	%Zn	3.51	2.25	3.42
	%Pb	0.94	0.60	0.92
	%Zn+Pb	4.46	2.84	4.33
Inferred*	Tonnes	1,768,495	644,833	2,413,328
	%Zn	3.28	2.4	3.04
	%Pb	0.89	0.76	0.86
	%Zn+Pb	4.17	3.16	3.90

In 2013, Tamerlane declared bankruptcy. However, limited work continued. Tamerlane proposed developing one underground mine at R-190 and nine open pits; eight of the nine open pits in the area were designated as “Cluster Pits”, and the ninth open pit was planned for the N-204 deposit. The nine proposed open pits were J-68, HZ, W-85, X-65, M-67, K-68, M-62, M-63 and O-53. The W-85 deposit was one of the two North Trend Cluster Pits on which Tamerlane generated Open Pit Mining Reserves, the other being the X-65 deposit. The 2013 proposal was based on previous work and economic studies completed in 2012 including the R-190 and N-204 deposits.

In 2014, Tamerlane modified their development plans to six open pit mining operations (six 'Cluster Pits') across the Pine Point Main Mineralized Trend (Siega and Gann, 2014). The six deposits listed by Tamerlane are: J-68, M-67, K-68, HZ, M-62/M-63 and O-53 prismatic deposit.

In March 2014, Tamerlane published a Technical Report by two independent consultants including a mineral resource estimate (Siega and Gann, 2014). This historical estimation was done on deposits that are part of the current MRE. That being said, Osisko Metals has completed additional work in this area with results reported by the Author in Chapters 9, 10 and 14. The current MRE is provided in Chapter 14 of this Report and supersedes the report by Siega and Gann (2014). The Author of the current Technical Report has read the documents pertaining to the description of the different methods used in the historical evaluation of the resources/reserves. The Author, also acting as the QP, has not done sufficient work to classify the historical estimate as current mineral resources or mineral reserves. The Author and Osisko Metals are not treating this historical estimate as current mineral resources or mineral reserves as defined in NI 43-101 and such historical estimate should not be relied upon. The categories of the Siega and Gann (2014) report were classified under previous definition standards and does not match the current (2014) classification. More specifically, the category "probable reserve" was classified under previous definition standards and does not match the current (2014) classification. There are currently no mineral reserves classified for the Project.

The J-68, K-68, X-65, W-85, M-67, M63/M62, O-53, and Hinge Zone deposits were estimated in Siega and Gann (2014). Geological boundaries for the deposits were modelled by developing a 1% Pb+Zn shell boundary. The deposits were developed by iterative grade shells. Capping was applied at the 95th percentile, no numerical value was provided in the Technical Report. ID^2 was used for estimation. Open pit mining costs, dense media separation costs, milling costs, and all ancillary costs were taken from Horlacher (2011).

The N-204 deposit historical mineral resource estimate was carried over from Horlacher (2011) with the 2014 study applying new pitshell parameters for the reserves statement.

Pitshell parameters are presented in Table 6-6. A summary of the historical resource is presented in Table 6-7 below.

Table 6-6: Tamerlane 2014 pitshell parameters for the J-68, K-68, X-65, W-85, M-67, M63/M62, O-53, and Hinge Zone historical Resource Estimate (Horlacher, 2011)

Item	Value
Mining Costs (Includes Site Overhead)($US)	
Ore	$2.75 per tonne
Waste	$2.00 per tonne
Loose Overburden	$1.50 per tonne ($3.30 per cubic metre)
Dilution (Weighted Average of Internal and External Dilution)	12% at 0.23% Zinc, 0.18% Lead
Mining Recovery	95%
Additional Waste Rock and Tailings Reclamation	$0.25 per tonne
Metal Prices	
Lead	$1.00 per pound
Zinc	$0.95 per pound
Selling Costs (Concentrate)	
Zinc	$0.26
Lead	$0.21
Processing Cost (EMS, Trucking and Milling)	$10.14 per tonne
Overall Recovery (DMS & Flotation)	
Zinc	93.6%
Lead	88.9%
Specific Gravity	
Ore	Defined in Block Model, Depends on Sulphide Content
Waste	2.7
Loose Overburden	2.2
Overall Pit Slope	52° (Including Haul Roads)
Nested Pits	20% to 100% of Base Case Metal Prices in 5% Increments
Mining Rate (Mill Feed & Waste)	6 Million Tonnes per Year
DMS Feed Rate	2.5 Million Tonnes per Year
Discount Rate	10%

Table 6-7: Tamerlane 2014 historical Mineral Reserve Estimate (Horlacher, 2011)

Deposit	Total Tonnes	Lead Grade%	Zinc Grade%	Combined Pb+Zn%	Category
CP Main Trend Deposits					
J-68	265,516	2.68	5.80	8.48	Proven Reserve
J-68	2,780	0.63	2.34	2.97	Probable Reserve
HZ	911,600	3.10	3.65	6.75	Proven Reserve
HZ	773,796	2.21	3.67	5.88	Probable Reserve
M-67	473,465	1.35	4.57	5.92	Proven Reserve
M-67	210,419	0.89	5.20	6.09	Probable Reserve
K-68	262,800	1.09	3.27	4.36	Proven Reserve
K-68	769,126	0.76	2.61	3.37	Probable Reserve
M-62/M-63	803,721	1.01	2.25	3.26	Proven Reserve
O-53	274,812	0.83	2.71	3.54	Probable Reserve
South Trend Deposit					
N-204	12,830,000	0.70	2.60	3.30	Probable Reserve

On December 20, 2016, Darnley Bay Resources Limited acquired the Pine Point assets from Tamerlane's receiver and subsequently changed its company name to Pine Point Mining Limited on August 8, 2017.

In February 2017, Darnley Bay published a Technical Report by two independent consultants including a mineral resource estimate (Siega and Gann, 2017). This historical estimation was done on deposits that are part of the current MRE. That being said, Osisko Metals has completed additional work in this area with results reported in Chapters 9, 10 and 14. The current MRE is provided in Chapter 14 of this Report and supersedes that of Siega and Gann (2017). The author has read the documents pertaining to the description of the different methods used in the historical evaluation of the resources. The author, also acting as the QP, has not done sufficient work to classify the historical estimate as current mineral resources or mineral reserves. The author and Osisko Metals are not treating this historical estimate as current mineral resources or mineral reserves as defined in NI 43-101 and such historical estimate should not be relied upon.

Siega and Gann (2017) report that the J-68, K-68, X-65, W-85, M-67, M63/M62, O-53, and Hinge Zone deposits were carried-over from (Siega and Gann, 2014), but classification was downgraded.

A summary of the historical resource is presented in Table 6-8 below.

Table 6-8: Darnley Bay 2017 historical Mineral Resource Estimate for deposits J-68, K-68, X-65, W-85, M-67, M63/M62, O-53, and Hinge Zone (Siega and Gann, 2017)

Deposit	Pb +Zn Cut-off Grade Pit	Tonnes	%Pb	%Zn
Hinge Zone Measured	1.50%	911,600	3.1	3.65
Hinge Zone Indicated	1.50%	773,796	2.21	3.67
J68 Measured	1.50%	265,516	2.68	5.8
J68 Indicated	1.50%	2,780	0.63	2.34
K68 Measured	1.50%	262,800	1.09	3.27
K68 Indicated	1.50%	769,126	0.76	2.61
X65 Measured	2.50%	2,510,448	1.45	3.65
W85 Measured	2.00%	2,326,514	2.82	4.58
W85 Indicated	2.00%	1,125,598	1.47	3.14
M67 Measured	2.50%	473,465	1.35	4.57
M67 Indicated	2.50%	210,419	0.89	5.2
-M63 Measured M62	1.50%	803,721	1.01	2.25
O53 Indicated	1.50%	274,812	0.83	2.71

Between February and August 2017, Darnley Bay conducted in-fill drilling program at W-85 (226.7 m in two drillholes), at L-65 and at the nearby K-66 deposit (5,756.1 m in 54 drillholes and 193 m in two drillholes respectively), and at K-60 (1,565 m in 17 drillholes). Regional exploration completed eight stratigraphic drillholes (846 m) along a haul road between deposits J-69 and K-77 in the summer of 2017.

In June 2017, Darnley Bay published a Technical Report by JDS Energy & Mining Inc. including a mineral resource estimate (Macdonald et al., 2017). This historical estimation was done on deposits that are part of the current MRE. That being said, Osisko Metals has completed additional work in this area with results reported by the Author in Chapters 9, 10 and 14. The current MRE is provided in Chapter 14 of this Report and supersedes the report by Macdonald et al. (2017). The Author of the current Technical Report has read the documents pertaining to the description of the different methods used in the historical evaluation of the resources. The Author, also acting as the QP, has not done sufficient work to classify the historical estimate as current mineral resources or mineral reserves. The Author and Osisko Metals are not treating this historical estimate as current mineral resources or mineral reserves as defined in NI 43-101 and such historical estimate should not be relied upon.

Macdonald et al. (2017) report that the R-190, X25, G-O3, P-499, O556, Z155 deposits historical mineral resource estimates were carried over from historical mineral reserves established by Collins et al, 2008. A summary is presented in Table 6-9.

The N-204 deposit historical mineral resource estimates was carried over from Horlacher (2011) with JDS applying new pitshell parameters (Table 6-12). A summary is presented in Table 6-10.

The Hinge Zone, J-68, K-68, X-65, W-85, M-67, M63/M62 and O-53 deposits, also referred to as "the cluster pits" historical mineral resource estimates were carried over from Siega and Gann (2017) with JDS applying new pitshell parameters (Table 6-13). A summary of the historical resource estimate is presented in Table Table 6-11 below.

Table 6-9: Darnley Bay 2017 historical Mineral Resource Estimate for deposits R-190, X25, G-O3, P-499, O-556, Z-155 (Macdonald et al., 2017)

Deposit	Category	Pb + Zn cut-off (%)	Tonnes	%Pb	%Zn
R-190	Measured	3.50%	647,308	6.1	12.47
	Indicated	3.50%	357,311	3.79	8.27
	TOTAL		1,004,619	5.28	10.98
P-499	Measured	2.50%	182,841	4.07	7.78
	Indicated	2.50%	708,811	2.32	5.38
	TOTAL		891,652	2.68	5.87
O-556	Measured	2%	433,860	3.47	3.92
	Indicated	2%	596,398	2.11	3.49
	TOTAL		1,030,258	2.68	3.67
X-25	Measured	2.50%	1,625,380	2.61	7.21
	Indicated	2.50%	482,933	1.33	5.11
	TOTAL		2,108,313	2.32	6.73
Z-155	Measured	2.50%	426,121	3.54	5.89
	Indicated	2.50%	349,297	1.74	3.96
	TOTAL		775,418	2.73	5.02
G-03	Measured	2%	960,804	3.57	6.09
	Indicated	2%	1,019,207	2.52	3.91
	TOTAL		1,980,011	3.03	4.97
TOTAL	**Measured + Indicated**		**7,790,271**	**3.01**	**6.16**

Table 6-10: Darnley Bay 2017 historical Mineral Resource Estimate for deposits N-204 (Macdonald et al., 2017)

Deposit	Tonnes	%Pb	%Zn
N-204			
Measured	6,321,000	0.81	3.04
Indicated	3,684,000	0.79	2.88
Total M+I	10,005,000	0.80	2.98
Inferred	3,527,000	0.78	2.89

Table 6-11: Darnley Bay 2017 historical Mineral Resource Estimate for the "cluster pits" (Macdonald et al., 2017)

Deposit	Tonnes	%Pb	%Zn
Hinge Zone (HZ)			
Measured	825,000	3.23	3.55
Indicated	724,000	2.22	3.61
Total M+I	1,549,000	2.76	3.58
J-68			
Measured	301,000	2.27	4.93
Indicated	3,000	0.56	2.09
Total M+I	304,000	2.25	4.90
K-68			
Measured	300,000	0.94	2.97
Indicated	890,000	0.76	2.70
Total M+I	1,193,000	0.81	2.77
X-65			
Indicated	5,430,000	0.80	2.41
Total M+I	5,430,000	0.80	2.41
Inferred	56,000	0.00	3.67
W-85			
Measured	2,703,000	2.34	3.84
Indicated	1,280,000	1.23	2.65
Total M+I	3,983,000	1.98	3.46
M-67			
Measured	880,000	0.92	3.33
Indicated	485,000	0.68	3.97
Total M+I	1,365,000	0.84	3.55
M-62/63			
Measured	352,000	1.42	2.13
Total M+I	352,000	1.42	2.13
O-53			
Indicated	81,000	0.91	2.90
Total M+I	81,000	0.91	2.90
Inferred	121,000	0.83	2.79
L-65			
Measured	374,000	0.52	2.22
Indicated	1,204,000	0.76	1.87
Total M+I	1,578,000	0.70	1.95

Table 6-12: Darnley Bay 2017 pitshell parameters for the N-204 historical Resource Estimate (Macdonald et al., 2017)

Parameter	Unit	Value
Revenue, Smelting & Refining		
Lead Price	US$/lb Pb	0.90
Zinc Price	US$/lb Zn	1.05
Exchange Rate	C$:US$	0.75
Royalty (3%)	% NSR	3.0%
OPEX Estimates		
OP Waste Mining Cost	C$/t waste mined	2.85
OP Mill Feed Mining Cost	C$/t feed mined	2.85
OP Mining Cost	C$/t processed	14.25
Processing Cost (DMS and Milling) (N-204)	C$/t processed	7.43
Ore haulage from DMS to Mill	C$/t processed	1.57
Additional Waste Rock and Tailings Reclamation	C$/t processed	0.25
G&A	C$/t processed	7.13
Total OPEX (excluding Mining) (N-204)	C$/t processed	16.38
Total OPEX (including Mining) (N-204)	C$/t processed	30.63
Recovery and Dilution		
OP External Mining Dilution	%	12%
OP Mining Recovery	%	95%
Metal Recovery (DMS & Flotation)		
Lead (N-204)	%	71.4%
Zinc (N-204)	%	78.0%
Other		
Overall Pit Slope Angles	degrees	52
Mill Production Rate	tpd	1,800
Smelter Terms and Offsite Costs		
Concentrate Grades		
Lead concentrate (N-204)	%	55.7
Lead concentrate moisture content	%	8.00
Zinc concentrate (N-204)	%	55.3
Zinc concentrate moisture content	%	8.00
Payables		
Lead	%	95.0
Zinc	%	85.0
Unit deductions		
Lead	%	3.0
Zinc	%	8.0
Smelting & Refining Costs		
Lead concentrate smelter cost (TC)	US/dmt	135.00
Zinc concentrate smelter cost (TC)	US/dmt	135.00
Freight & Marketing		
Lead Concentrate	US$/wmt	167
Zinc Concentrate	US$/wmt	154

Table 6-13: Darnley Bay 2017 pitshell parameters for the "cluster pits" historical Resource Estimate (Macdonald et al., 2017)

Parameter	Unit	Value
Revenue, Smelting & Refining		
Lead Price	US$/lb Pb	0.90
Zinc Price	US$/lb Zn	1.05
Exchange Rate	C$:US$	0.75
Royalty (3%)	% NSR	3.0%
OPEX Estimates		
OP Waste Mining Cost	C$/t waste mined	2.85
OP Mill Feed Mining Cost	C$/t feed mined	2.85
OP Mining Cost	C$/t processed	14.25
Processing Cost (DMS and Milling) (CP)	C$/t processed	9.20
Ore haulage from DMS to Mill	C$/t processed	1.57
Additional Waste Rock and Tailings Reclamation	C$/t processed	0.25
G&A	C$/t processed	7.13
Total OPEX (excluding Mining) (CP)	C$/t processed	18.15
Total OPEX (including Mining) (CP)	C$/t processed	32.40
Recovery and Dilution		
OP External Mining Dilution	%	12%
OP Mining Recovery	%	95%
Metal Recovery (DMS & Flotation)		
Lead (CP)	%	85.0%
Zinc (CP)	%	88.1%
Other		
Overall Pit Slope Angles	degrees	52
Mill Production Rate	tpd	1,800
Smelter Terms and Offsite Costs		
Concentrate Grades		
Lead concentrate (CP)	%	61.0
Lead concentrate moisture content	%	8.00
Zinc concentrate (CP)	%	62.0
Zinc concentrate moisture content	%	8.00
Payables		
Lead	%	95.0
Zinc	%	85.0
Unit deductions		
Lead	%	3.0
Zinc	%	8.0
Smelting & Refining Costs		
Lead concentrate smelter cost (TC)	US/DMT	135.00
Zinc concentrate smelter cost (TC)	US/DMT	135.00
Freight & Marketing		
Lead Concentrate	US$/WMT	167
Zinc Concentrate	US$/WMT	154

A confirmation and twinning drill program at the L-35, L-36 and L37 deposits were conducted in August-September 2017. This consisted of 29 drillholes that twinned historic drillholes drilled by Cominco Ltd. The Total length of drillholes drilled during the twinning program was 1,597 m. Core from two drillholes (126.14 m) was used for metallurgical tests.

A ground gravity geophysical survey was conducted in the area of the Pine Point townsite and the N-42 deposit in the fall of 2017. Quantec Geoscience Ltd. collected 1,819 stations of gravity measurements over the Townsite grid, 455 stations over N-42 grid and 430 stations over the mill site grid. Smaller grids with 50-100 stations were surveyed at L-65, W-85 and R-190 deposits. A number of gravity anomalies were identified.

In the fall 2017, Pine Point Mining Limited drilled 541.95 m in eight drillholes in the N-42 area and 788.5 m in eight drillholes in the Townsite area to test geophysical anomalies. A further 171 m were drilled in three drillholes in the M-40 and N-38 areas.

Between February and November 2017, Pine Point Mining Limited drilled a total of 11,696 m in 131 drillholes (Table 6-14).

Table 6-14: 2017 Darnley Bay Limited drilling summary

Deposit	# of holes	Metres	Comment	Program
EX Haul Road	8	846.00		Exploration/Strat
EX Townsite	8	788.50		Exploration/Gravity
K-60	17	1,565.00		
K-66	2	193.00		
L-65	54	5,756.10		
L-35	14	676.38		
L-36	14	870.66	2 met holes: 126.14 m	Twinning/Met
L-37	1	50.00		1,597.04
N-38	1	50.00		
M-40	2	121.00		
N-42	7	541.95		Exploration/Gravity
W-85	3	237.37	incl. 1 abandoned at 10.67 m	
Total	**131**	**11,695.96**		

In February 2018, Osisko Metals Incorporated acquired Pine Point Mining Limited and became sole owner of the Pine Point project. Between February 2018 and September 2018, Osisko Metals completed definition drilling totalling 23,751 m in 318 drillholes that are included in the Mineral Resource Estimate detailed in this Report.

7. GEOLOGICAL SETTING

7.1 Regional Geology

The Pine Point deposits are located on the southern shore of Great Slave Lake. They form a 70 km long southwest-northeast-trending belt between Hay River and Fort Resolution in southern Northwest Territories. The area lies on the eastern margin of what is defined regionally as the Western Canada Sedimentary Basin. The Pine Point deposits exhibit all the geological, mineralogical and geochemical attributes of Mississippi Valley-Type (“MVT”) base metal deposits (Leach et. al 2010).

Zinc-lead mineralization at Pine Point is hosted in a dolomitized carbonate barrier reef complex that transects the entire property. As with most MVT deposits globally, there is no known spatial or temporal relation to igneous rocks. The area south of Great Slave Lake is underlain by the extensive southwest-trending sequence of Devonian Carbonate lithologies of middle-Devonian age.

The carbonate sequence dips gently to the southwest and extends for up to 650 km into northern Alberta. These Middle Devonian carbonate lithologies are host to the Pine Point MVT mineralization. The individual Pine Point deposits are widely distributed across the 70 km southwest trending belt covering up to 1,600 km^2.

Most deposits contain between 0.20 Mt and 2 Mt of mineralized material, although the largest, the X-15 Deposit, had nearly 18 Mt (Sangster, 1990). Fifty of the 100 drill-defined deposits were developed in the Pine Point District by Cominco Ltd. (Hannigan, 2007). Total production from 1964 to 1988 is reported at 64.3 Mt at an average grade of 7.0% Zinc and 3% Lead (Cominco Ltd.).

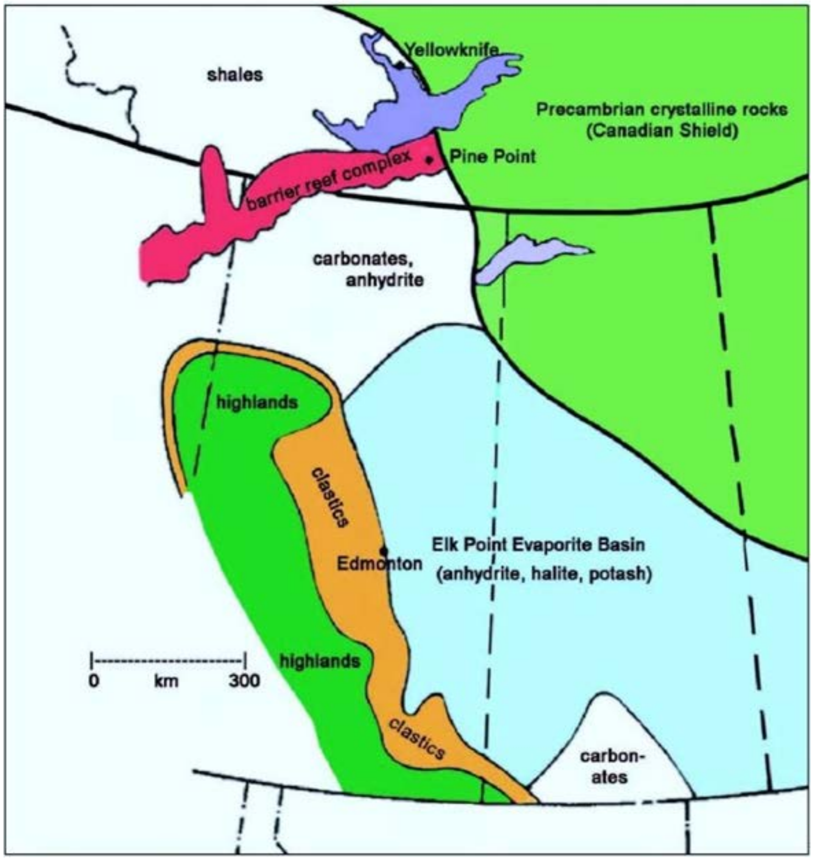


Figure 7-1: Regional geological setting (Gunn, 2016)

The Pine Point deposits are hosted by dolomitized carbonates and subordinate limestones that form part of an extensive carbonate reef and platform sequence. Wall-rock alteration consists of large scale dolomitization, which results in dissolution, recrystallization and local hydrothermal brecciation of the carbonate host lithologies. These properties combine to create ground preparation for sulphide deposition at Pine Point in a manner not dissimilar to other MVT districts.

In Middle Devonian time, a large carbonate reef complex, extending up to 1,000 km in length, developed along a southwest-northeast basement high in a marine environment extending across what is now southern NWT and extending across northern Alberta into British Colombia (Figure 7-1).

The carbonate reef complex developed in response to eustatic fluctuations and formed a classic barrier reef complex of Paleozoic age with the barrier forming a continuous boundary between the open ocean (north) and a shallow marine restricted environment (back-reef) to the south. The latter environment is dominated by a distinct suite of restricted shallow-water algal carbonate complexes and local evaporite sequences (Muskeg Formation) and tidal-flat sediments.

Regionally, the barrier reef complex attained a width of up to 10 km and accumulations at this basin margin exceeded some hundreds of metres in thickness. The southern restricted shallow-marine basin formed part of the extensive Elk Point Basin extending into northern Alberta, central Saskatchewan, Manitoba and south into North Dakota. Evaporite sequences are much more extensively developed in the central parts of the Elk Basin.

The open marine environment to the north developed into the extensive Mackenzie Basin. The barrier reef complex developed with remarkable continuity and facies consistency along the entire Mackenzie Basin margin. Sequence stratigraphy demonstrates the typical abrupt facies variations across the Pine Point Reef complex similar to carbonate reef complexes found throughout geological time and as observed in modern marine environments – Bahamas Bank, Great Barrier Reef, etc.

The Pine Point mineralized belt was extensively worked by Cominco Ltd. from the early 1960’s to 1988. In this period, considerable geological data was gathered. Geological description and interpretation of the Pine Point sedimentary sequence reflects the geological thinking and models of this time. A detailed summary of the Cominco Ltd. exploration effort over a 25-year period is documented by Rhodes in a compilation paper published in Economic Geology (1984). The Rhodes work includes a compilation of earlier researchers work at Pine Point, notably the contribution by H. Skall (1975). Reinterpretations of the Cominco Ltd. data followed, notably including significant work by Hannigan (2007).

The Great Slave Shear Zone is a large regional tectonic structure that transects the eastern part of the Pine Point district (Figure 7-2). This structural system can be traced with the aid of aeromagnetics and extends from within the Canadian Shield through the Pine Point area into northwestern Alberta and northwestern British Colombia. As a deep-sourced basement structure, it is a potential conduit for hydrothermal fluids and/or diagenetic fluids and may have played a part in fluid channelization during the dolomitization process, which affected the middle-Devonian carbonate sequence at Pine Point.

The northeast-southwest-trending Great Slave Shear Zone is interpreted to transect the eastern sector of the Pine Point Block between the X-15 and N-204 Deposits. It is of interest that a significant photo lineament is traceable at this locality but with a more north-northwest trend. This trend is largely orthogonal to the mapped Pine Point mineralized trends. The relationship between the two opposing trends agrees with the principles of fault transfer systems. The interpreted subtle structural controls to mineralization at Pine Point may be indirectly related to the Great Slave Shear Zone.

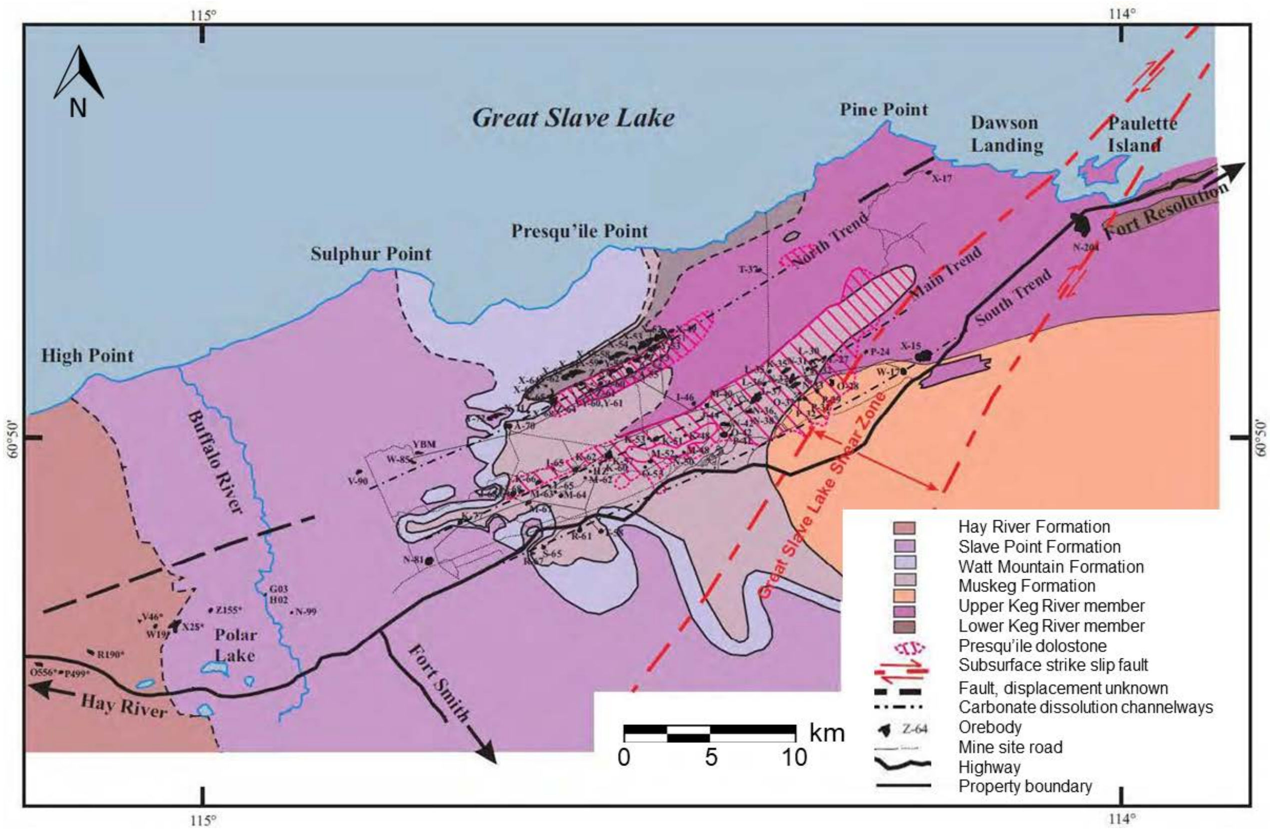


Figure 7-2: Pine Point property geology (Hannigan, 2007)

7.2 District Geology

The Pine Point district extends from the east to the settlement of Fort Resolution and continues to the west at shallow depths past the town of Hay River. The district is underlain by the Middle Devonian carbonate reef sequence that is the principal host to the Pine Point MVT mineralization. The Middle Devonian section is subsequently overlain by non-mineralized Middle and Upper Devonian sequences to the northwest and west where the Upper Devonian Sequence is represented by thick basinal shales and restricted carbonates dipping gently to the west District (Figure 7-2 and Figure 7-3). This upper sequence represents the extensive open marine environment, contrasting sharply with the sub-tidal, sabkha and shallow-marine environment represented by the Muskeg Formation representing the northern extent of the Elk Point Basin.

The current geological setting reflects post-Devonian uplift and westward tilting resulting in the exposure of the lower Pine Point Formation facies carbonates, marine calcarenites and grain stones to the east towards Fort Resolution (Figure 7-2 and Figure 7-3). The gently inclined stratigraphy containing the west-dipping mineralized sequence sub-crops for over 45 km before shallowly-plunging to the west under cover of the Upper Devonian basinal lithologies (Figure 7-2 and Figure 7-3). The belt is centered on the Pine Point Reef Complex. Sulphide mineralization occurrences are largely aligned along the Pine Point Reef Complex margin where there is a later increase in dolomite alteration both regional (diagenetic, ground preparation) and hydrothermal (mineralization-related).

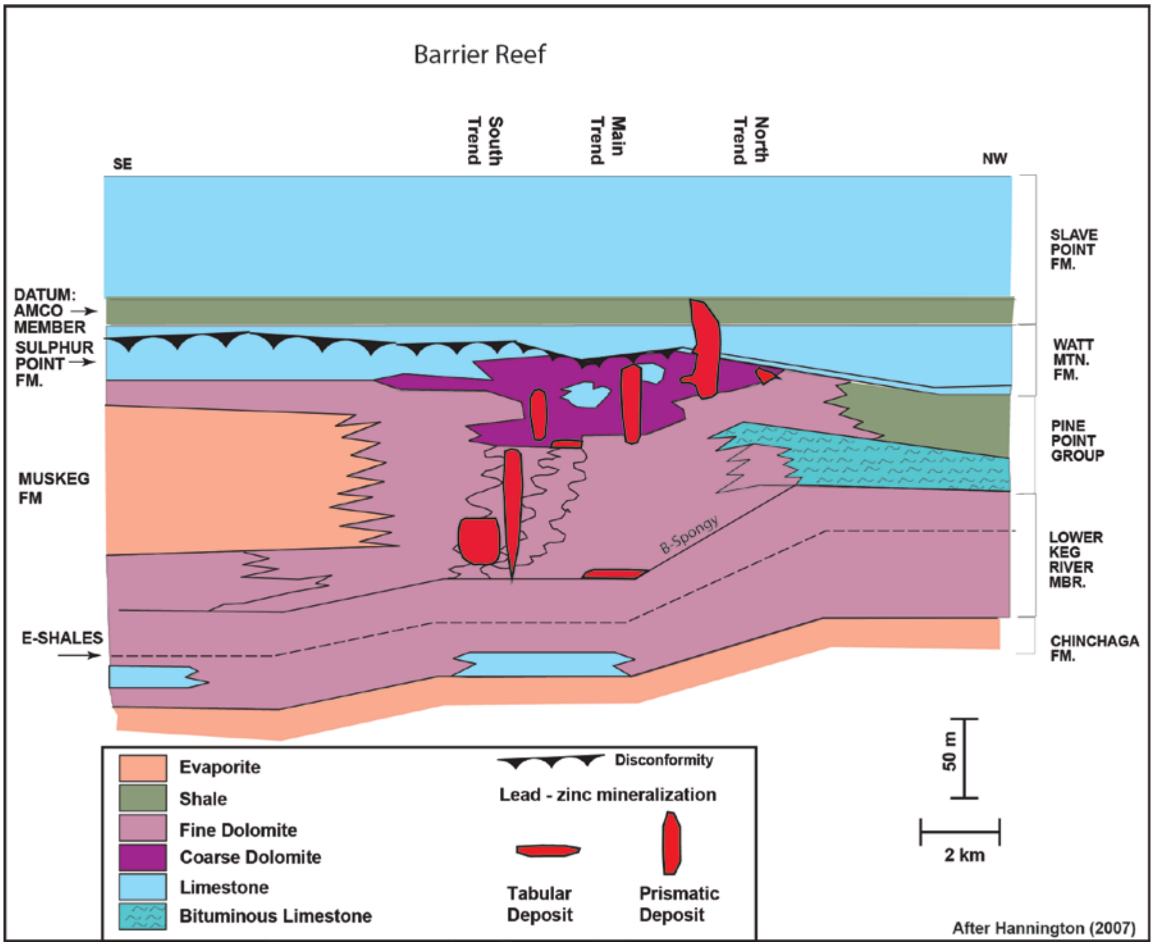


Figure 7-3: Geological section across the Pine Point barrier reef complex showing several Pine Point deposits and their stratigraphic setting within the Pine Point carbonate shelf-to-basin succession (Hannigan, 2007)

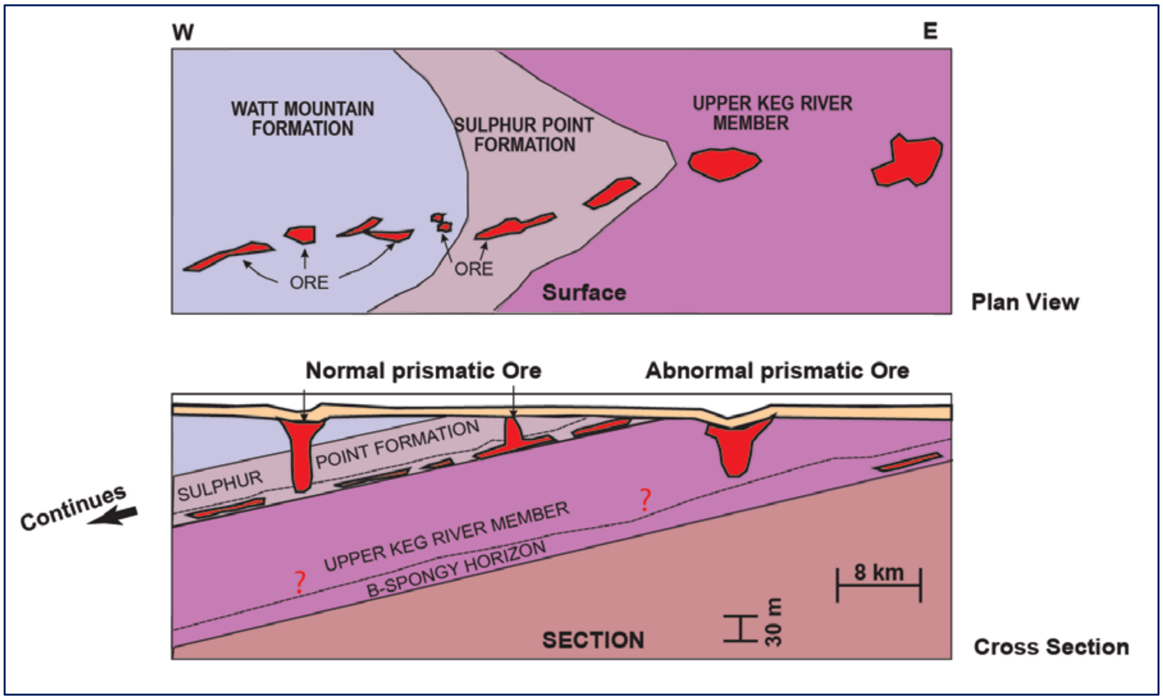


Figure 7-4: Schematic surface map and cross-section

The main Pine Point base metal mineralization extends along suitable carbonate host rocks central to the reef complex called the Sulphur Point Formation. The formation is characterized by a buildup of micrites, biomicrites, subordinate clean carbonaceous calcarenites, skeletal grainstones and algal carbonate-dominated sequences. These lithologies sub-crop beneath Pleistocene glacial deposits and post-glacial swamp deposits. There are three known mineralized trends: a Central "Main Pine Point Trend", and two subordinate mineralized trends – the well mineralized North Trend and the South Trend (Figure 7-2).

The orientation and form of these trends may reflect very subtle structural controls resulting in the parallel alignment of the mineralized bodies along the three known trends. Several southwest-trending lineaments in the underlying Precambrian basement are identified and are interpreted to extend to the southwest beneath the sedimentary rocks in the Pine Point district.

Structural elements likely represented by deep-seated features may extend into the younger overlying Devonian sedimentary sequences, which exhibit similar southwest-trending orientations. Such subtle structural elements feasibly exerted controls on the reef-margin orientation and on the development of the reef complex itself through the Devonian Period.

On both a regional and local scale, the Devonian lithologies, which host the MVT zinc-lead deposits, are un-deformed and form a 'layer-cake' west-dipping sedimentary sequence.

7.3 Property Geology

The Middle Devonian sedimentary sequence at Pine Point is reasonably well understood because of the extensive diamond drilling campaigns carried out by Cominco Ltd. over almost 40 years. Cominco Ltd. reports more than 10,000 diamond drillholes cored between 1948 and 1988 for an extraordinary 600,000 m of core.

The stratigraphy is well-defined, but lacks modern carbonate stratigraphy terminology and sequence stratigraphic principals. The bulk of the property is underlain by the prospective carbonate sequence. Underlying the western sector of the property is the younger Upper Devonian sequence dominated by the Slave Point Formation (Figure 7-2).

The host lithologies of the Sulphur Point and Pine Point formations are locally porous with extensive development of dissolution vugs, caverns and relict dissolution breccias in the clean carbonates, notably the Sulphur Point micrite complex. At one time, the dissolution cavities were interpreted to be related to meteoric karstification but are now interpreted as complex large-scale hydrothermal dissolution features occurring contemporaneously with the emplacement of the MVT mineralization (Qing, et al. 1994).

The importance of hydrothermal metal-bearing fluids in the formation of the extensive dissolution features was first alluded to in the 1970's (Skall, 1975). However, Cominco Ltd. workers appeared to continue favouring a meteoric karst origin for the extensive dissolution processes observed in the Pine Point host carbonates.

Independent of the dissolution processes the host lithologies are extensively dolomitized. The entire Sulphur Point Formation is pervasively altered with a regional 'grey' replacive dolomite overprint. This is subsequently overprinted with a coarser 'white' dolomite phase.

7.4 Stratigraphy

The carbonate stratigraphy of the Pine Point area is elucidated and is largely based on the work of Skall (1975) (Figure 7-3). The basic subdivisions are:

<u>Keg River Formation</u>. Basal, regionally extensive shallow-water bioclastic and biomicritic limestones comprising a carbonate platform sequence.

<u>Pine Point Formation</u>. An extensive accumulation of shallow-water grainstones, calcarenites and fine biocalcarenites with locally developed biohermal structural components and patch reef structure. The Pine Point Formation grainstones and calcarenites are very susceptible to diagenetic dolomitization processes with a resultant enhanced secondary porosity. The upper Pine Point stromataporoidal facies carbonates heralds the onset of extensive biohermal 'reef' development across the platform and the commencement of the Sulphur Point Formation reef build-up.

Sulphur Point Formation. An extensive carbonate package consisting of a combined micrite, biomicrite and algal carbonate build-ups with some biohermal structure. Considerable microbial buildup during the early stage Sulphur Point deposition yields to more stromataporoidal reef structure with stromataporoidal bioclastic debris accumulations interspersed with abundant micrite piles. The Sulphur Point Formation is a principal host to the Pine Point MVT mineralization. It contains the reef, reef front and some reef-slope facies carbonates.

Muskeg Formation. The formation constitutes the principal back-reef facies and is predominantly an evaporite dominated sequence containing anhydrite interbedded with fine shallow marine algal facies carbonates. In the back-reef environment, the Muskeg Formation is laterally equivalent to the lower stromatactis dominated microbial carbonate facies of the lowermost Sulphur Point Formation and locally the Muskeg Formation evaporitic sequence is also laterally equivalent to the upper sections of the Pine Point Formation grainstones and calcarenites indicative of a slight increase in wave energy of the depositional environment.

Fore-Reef Facies carbonates of the Buffalo River Formation and Windy Point Formation.

A brief period of subaerial exposure caused by a drop in sea-level resulted in the cessation of carbonate barrier development and the development of an unconformity – the sub-Watt Mountain unconformity (developed during 'sub-Watt Mountain emergence') that occurs throughout the northern part of the Western Canada Sedimentary Basin (Meijer Drees, 1988). This unconformity separates the Sulphur Point Formation from the Watt Mountain Formation.

Watt Mountain Formation. The Watt Mountain Formation consists of sub-tidal green, silty shales and very fine argillaceous limestones that are locally dolomitized. It exhibits very low porosity and permeability and may have acted as an aquitard that controlled the upward migration of dolomitizing and hydrothermal fluid flow including fluid movement related to the Pine Point mineralizing event or events.

Slave Point Formation, This formation consists of a mix of shallow water subtidal facies carbonates and deeper water, sub-basinal argillaceous limestones (Skall, 1975). The Slave Point Formation is divided into three facies types. The upper 'P-facies' consisting of shallow-water laminar facies carbonates intercalated with shallow, basinal carbonates. The middle O-Facies, is distinctly fossiliferous with abundant stromataporoidal debris. The O-facies is commonly mineralized in some of the North Zone Deposits, notably W85.

7.5 Alignment of Mineralization and Structural Controls on Mineralization

The Pine Point deposits are aligned along the North, Main and Southern trends as shown in Figure 7-5. These structural trends are parallel to the Pine Point barrier reef complex and described as follows:

- **Main Trend.** Mineralization is deposited within such a zone, which is interpreted to have a subtle structural control allowing for the lateral continuity and alignment of the mineralized bodies.
- **North Trend.** Mineralization exhibits similar controls to mineralization as described in the Main Trend. As a result, mineralization is present to varying degrees over 20 kilometres, with possible continuation in both directions.
- **South Trend.** Mineralization appears similarly aligned along a sub-parallel controlling zone but is less well defined and contains less intense dolomite alteration limited by restricted carbonate facies.

The Central Trend contains most of the developed deposits. There is considerable potential along strike to the west on the North Trend. Although mineralization is likely, it is overlain by younger Devonian, basinal shales and slope deposits.

In the Central Trend, and in particular the East Mill Zone, there is a clear deposit alignment with a superimposed southwest-north-easterly fabric. This fabric forms the basics of a new interpretation where the three local trends appear to be offset where ‘linked’ to a set of NE-SW-trending structural sets. Furthermore, this NE-SW trend also controls deposit distribution. The better mineralization ‘blow outs’ may coincide with points of intersection of the trends and structural sets.

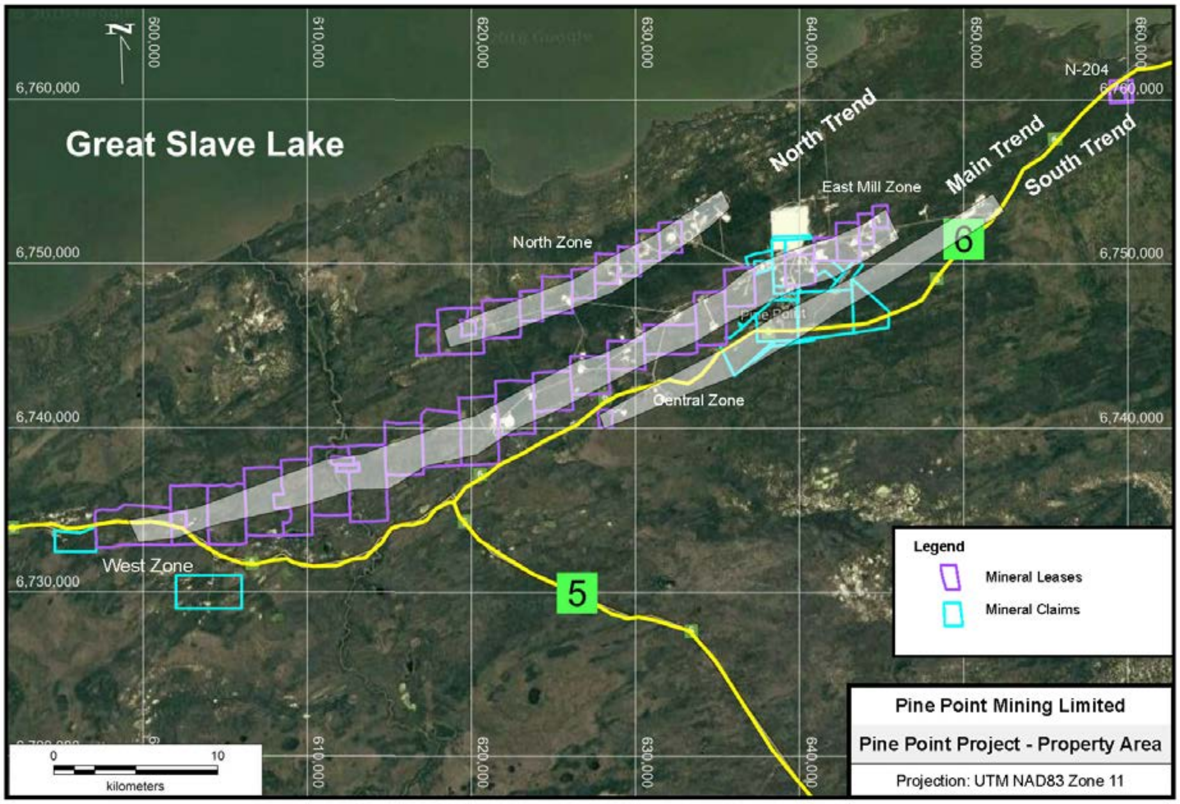


Figure 7-5: Mineralization trends

Recent drilling revealed a structural regime aligned along ENE-WSW trending structures in a number of areas. In some cases, there is an apparent down-throw of up to 65 metres on these structures. The northeast-southwest trending structural components are partly parallel to the basement geophysical signatures of the Great Slave Shear Zone. The higher-grade East Mill Zone mineralization is focused along structural segments, which trend parallel to the Great Slave Shear Zone.

8. DEPOSIT TYPES

The Pine Point deposits are carbonate-hosted lead-zinc sulphide deposits known as Mississippi Valley-Type (“MVT”). MVT deposits are a sub-type of a broader class of deposits called “sediment-hosted Pb-Zn” deposits (Leach et al., 2010).

Pine Point is probably the most famous and best-known Canadian example of MVT deposits. Other important Canadian deposits of the same type include Polaris, Nanisivik, Daniel’s Harbour, Gays River, Monarch-Kicking Horse, and Robb Lake.

The Pine Point deposits are distributed over an area of approximately 1,600 km^2 and define a MVT district comparable in size with the Tri-State MVT district in the USA (Leach et al., 2010).

The characteristic features of Mississippi Valley-Type mineralization (Sangster, 1996) that are present in the Pine Point District include:

- Development of platform carbonate sequences;
- Stratigraphic controls;
- Sphalerite-galena-pyrite mineralogy;
- Brittle fracturing;
- Evidence of dissolution of carbonate host rocks (expressed by slumping, collapse, and brecciation);
- Fluid inclusions containing dense saline aqueous fluids (dissolved salts are predominantly sodium and calcium chlorides).

8.1 Mineralization

Mineralization at Pine Point is relatively simple, consisting of zinc, lead and iron sulphides occurring in a dolomitized, carbonate barrier reef complex. Sulphides consist of sphalerite and galena with subordinate marcasite and pyrite. Mineralization occurs as both open-space fillings and replacement of dolomite. The following types are documented:

1. Fine replacive sphalerite. This style of mineralization is common and present in tabular deposit types and adjacent to more pervasive mineralization. It varies from massive to disseminated replacement sphalerite. The more massive replacement mineralization allows for the development of discrete banded colloform sphalerite masses (Figure 8-1).
2. Colloform sphalerite. Colloform mineralization develops with increased dissolution of the host carbonates with marginal replacement disseminations and fine filaments. The colloform mineralization is extensive at the core of tabular deposits (Figure 8-2). Well-developed colloform sphalerite decreases away from the core of the linear tabular deposits and is also present in the prismatic deposits.

3. Finely to coarsely-crystalline, replacive and open-space filling sphalerite mineralization. Sphalerite crystals up to 12 mm are recorded in clean dolomitized biomicrites (Figure 8-2 and Figure 8-3). There are limited and local amounts of associated marcasite.
4. Finely to coarsely-crystalline galena. Galena crystals typically define the last paragenetic stage of the depositional process and can be locally concentrated in galena-rich areas. Typically, however, galena is present in subordinate amounts compared to sphalerite (Figure 8-2).
5. Massive sulphides. Prismatic mineralization is generally massive, consisting of 100% carbonate-replacement sulphides with a complete grain size spectrum from microscopic to mega crystalline sphalerite and galena mineralization. Multiple fluid phases allow for re-dissolution/precipitation and the generation of internal ‘open-space’ colloform mineralization, often spectacular and unique to the Pine Point District.
6. Complex sulphide mineralization. Complex sulphide mineralization in Prismatic Deposits also consists of massive sulphide replacing polymictic carbonates consisting of multiple ‘fallen’ or collapsed blocks from the overlying strata of the upper sequence carbonates, e.g. large blocks consisting of Slave Point Formation and indeed the replacement of internal dissolution sediments developed during the dynamic formation of the chimney-like, vertically elongate massive sulphide deposits, which define typical Prismatic Deposits. There are local galena-rich zones, typically massive to coarsely-crystalline.
7. Heterogenous massive sulphides. Predominantly sphalerite-rich massive sulphides exhibiting characteristics of both replacive massive sphalerite but with a later more coarse-crystalline and slightly higher temperature sphalerite phase overgrowth. This appears to occur in several small prismatic deposits in the eastern portion of the Main Trend.
8. High temperature sulphides. Higher temperature, finely-crystalline, crypto-colloform ‘black’ sphalerite mineralization is typical of some North Zone prismatic deposits. This style is best developed in internal grey carbonate sediment with a ‘veines bleues’ dolomite association.
9. N-204 type mineralization. The N-204 mineralization is fine-grained, sphalerite dominated mineralization consisting of fine-replacive and minor fine to medium open space sphalerite crystals within moldic porosity.

Figure 8-1: Fine colloform and replacive sphalerite marginal to the coarse-crystalline mineralization at the East Mill Zone

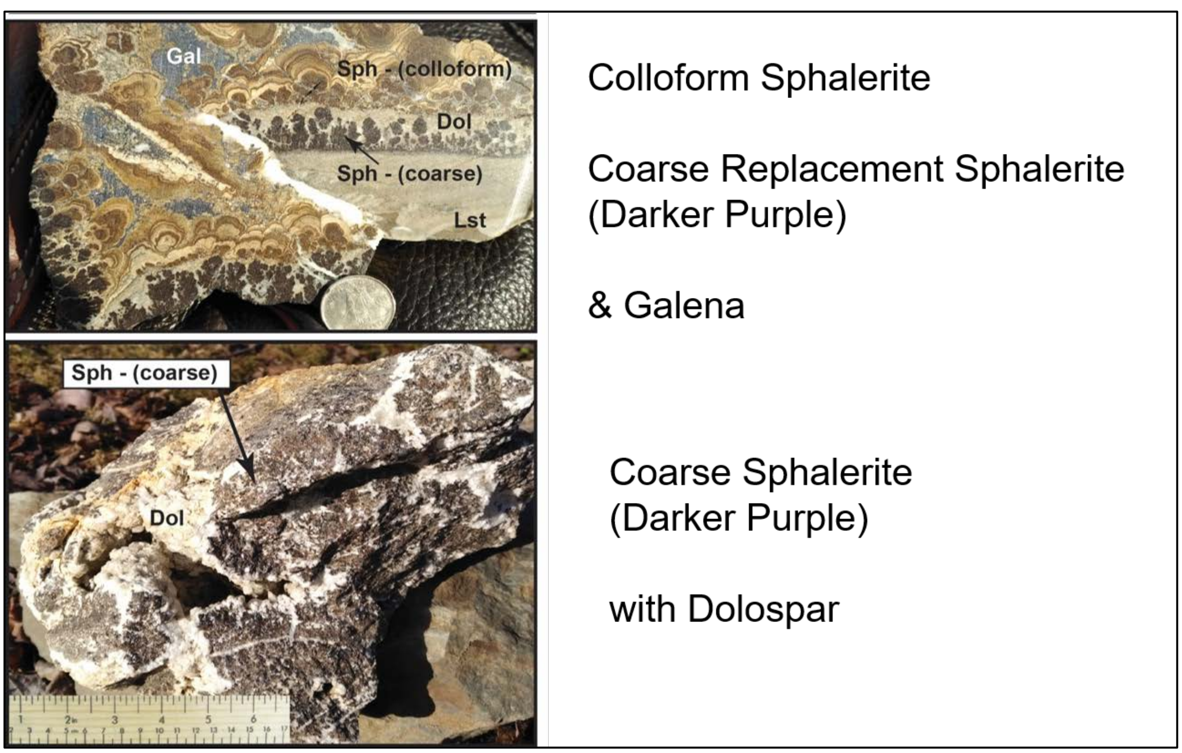


Figure 8-2: Examples of mineralization in hand specimen

Figure 8-3: Coarse-crystalline sphalerite-rich mineralization at the core of the L37 deposit. Sphalerite crystals up to 10 mm in white 'sparry' dolomite 'gangue'.

8.2 Deposit Types

Mineralization is hosted in three distinct deposit types:

1. **Prismatic Deposits** – Prismatic deposits demonstrate considerable vertical continuity but with limited lateral extent and are discordant. Mineralization is coarse to medium grained. These deposit types are further divided into normal and abnormal prismatic deposits based on their relative stratigraphic position in the sequence. They contributed greatly to the Pine Point historical production as they contained very high-grade base metal grades and thicknesses.
2. **Tabular Deposits** – Laterally continuous, semi-concordant mineralized zones mostly restricted to a specific carbonate horizon.
3. **N-204** – N-204 mineralization consists of fine crystalline mineralization deposited within the porosity of stratibound layers below the main sequence known as the B-spongy horizon where intense dissolution resulted in the development of a fine dolomite breccia. This type of deposit has to date only been identified at N-204.

8.2.1 Prismatic Deposits

Prismatic deposits are generally high-grade and can contain up to 50% combined Pb-Zn, depending on the amount of sulphide mineralization present. These deposits are vertically continuous for up to 60 m, and laterally restricted, generally 15-50 m, but can be up to 140 m wide. Metal zonation is present with a galena-rich core (Pb/Pb+Zn >0.3) enveloped by a more sphalerite-rich outer zone.

Prismatic deposits generally initiate at the Pine Point/Sulphur Point Formation facies transition and extend upwards through a dolomitized stratigraphic section, including upper sequences such as the Slave Point Formation. Abnormal prismatic deposits generally initiate within the lower stratigraphic sequences and extend upward.

These types of deposit generally contain blocks of overlying and adjacent stratigraphic material. Such blocks can exhibit angular and dissolution textures. Cavities and vugs are common and abundant internal sediment accumulations are observed to fill earlier open spaces. These sediments are the residues of intense hydrothermal dissolution consisting of insoluble carbonaceous debris, argillite components, and sulphide material. It is believed that these sediments are remnants of the intense hydrothermal dissolution process associated with sulphide emplacement.

8.2.2 Tabular Deposits

Tabular deposits develop in distinct biomicrite facies carbonate sequences in the lower Sulphur Point Formation. They are laterally continuous, following the general carbonate reef trend. They most likely following the increased porosity and permeability generated by hydrothermal fluid flow along distinct ‘channels’ or fluid pathways in suitable carbonate sequences. Other suitable host lithologies, e.g. the ‘B-Spongy’, are potential exploration targets.

Tabular deposits are vertically restricted relative to Prismatic Deposits and are generally zinc-rich (Pb/Pb+Zn) <0.2 with local lead-enrichment. They are lower-grade relative to prismatic deposits due to less massive and sporadic sulphide development. Zinc and lead grades decrease outwards from the core of hydrothermal fluid pathways (channels) and distally from interpreted local ‘feeder’ structures.

A related type of mineralization is found at N-204 where tabular-like mineralization occurs in a lower part of the barrier reef stratigraphy in a horizon named the “B-spongy horizon” (Cominco Ltd. terminology). Here, precipitation of fine-crystalline dolomite resulted in the preferred dissolution of macro fossil components resulting in the development of a distinct moldic porosity. Mineralization at the N-204 deposit is wholly confined to this dissolution horizon.

8.3 Mineralized Trends

The 100 or so known deposits are hosted by more than one carbonate horizon. However, the Sulphur Point Formation hosts most of the known deposits in the Pine Point District. The individual deposits are associated with the dolomitized carbonate strata along the barrier reef complex. Former deposits and remaining in situ mineralization are preferentially focussed along the three extensively dolomitized zones described in Chapter 7.

8.4 Gangue Mineralogy

Gangue mineralogy within the zones at Pine Point consists of dolomite, calcite, and lesser quantities of clay, pyro-bitumen, sulphur, and iron sulphides both marcasite and pyrite. Dolomite and calcite are the dominant gangue present and the other minerals/materials listed are minor but can be locally important.

Dolomite is present in all mineral zones and calcium carbonate as calcite crystals are almost always present. Calcium carbonate as limestone and silty limestone can also be present in prismatic deposits within the collapse breccia. Dolomite occurs as massive material, white vein dolomite, and less commonly as saddle dolomite with 2 mm to 5 mm crystals. Calcite crystals are post mineral, and occur commonly as 1 cm to 5 cm crystals but rare large 10 cm to 20 cm crystals do occur.

Clay minerals may be present in prismatic deposits where blocks of the overlying Watt Mountain, Windy Point, Buffalo River, or Slave Point Formations have collapsed into the mineralization. Additionally, recent karsting can allow glacial overburden and erratic boulders to become mixed in the mineralized zones of prismatic deposits.

Marcasite and less commonly pyrite are normally a minor component within the mineralization but typically occur above or below and separate from the mineralization. The marcasite is usually massive and pyrite crystals can locally develop.

Pyro-bitumen is rare in the Central and South Zone deposits but is common in the North Zone deposits and in the N-204 deposit. In the North Zone it appears to be late and occurs in voids and vugs. It can be present in concentrations of between 2% and 20% over several metres. Sulphur is very rare but has been observed in the North Zone with pyro-bitumen. Sulphur may also be introduced in blocks of Watt Mountain or Slave Point where sulphur is a common but minor component.

8.5 Nature of the Pine Point Sulphide Deposits

As described, the deposits consist of prismatic and tabular types that are aligned along distinct linear features at Pine Point. The linear dissolution zones appear to form a complex interconnected permeability network. The permeability networks prefer the more bioclastic facies carbonates represented by the Sulphur Point Formation micrites and biomicrites.

In general, the tabular mineralized zones are concordant with considerable lateral continuity and appear more restricted to the lower parts of the Sulphur Point Formation. The tabular bodies are elongated flat-lying bodies that are up to 12 m thick and broadly following distinct dissolution channels within the dolomitized sequences.

Prismatic deposits are chimney-like and vertically elongated bodies associated with more intense dissolution zones extending upwards from roots at the tabular horizons. Zones of intense dissolution coincide with areas of more intensive fracturing and jointing resulting in increased porosity and permeability of the host carbonates, which are subsequently mineralized.

The location of such increased dissolution zones may reflect subtle zones of structural weakness and/or distinct carbonate facies variations across the platform. Such a structural inference is augmented by the distinct alignment of the prismatic deposits and the apparent non-randomness of their distribution across the Pine Point District. Dissolution often resulted in sagging and collapse in the overlying strata, sometimes affecting up to 100 m of section. The collapse structures are often infilled with a heterogenous mix of locally derived sediments consisting of laminated sediments and irregular breccia components, exotic carbonate and mudstone blocks, etc.

The extensive carbonate dissolution forming the extensive ground preparation at Pine Point is locally irregular but forms along a set of linear trends (Figure 7-4). The tabular dissolution features are the most extensive and are locally 'studded' with prismatic deposits extending through the Sulphur Point Formation.

8.6 Stratigraphic Control

The most important characteristics of MVT deposits are that they are formed in platform carbonate sequences usually located at flanks of basins. The Pine Point District is hosted in a carbonate barrier reef at the flank of the Western Canadian Sedimentary Basin. Favourable stratigraphy in Pine Point encompasses various carbonate barrier lithofacies within the Pine Point and Sulphur Point formations that total 200 m in thickness of Middle Devonian stratigraphy. At the Lower boundary of the favourable stratigraphy is the contact with the Kegg River platform dolomites and E-shale marker horizon. The upper boundary is an unconformable contact with the Watt Mountain Formation shales. As such, Pine Point mineralization is stratabound on a district scale. On a deposit scale, prismatic mineralization is discordant and, sometimes, the mineralization extends above the Watt Mountain Formation and into the Slave Point Formation.

8.7 Alteration

The carbonate sequence at Pine Point exhibits extensive diagenetic dolomitization across the entire Pine Point Deposit area. This dolomitization of the carbonate sequence, in particular, the Sulphur Point Formation, results in ground preparation for the deposition of base metal mineralization and the propagation related fluids. A regional replacive secondary dolomitization event transected the sequence creating fluid pathways and conduits for hydrothermal Mg-rich and metal-rich fluids. Metal bearing solutions or brines moved laterally over several hundreds of kilometres in the favourable biomicrite carbonate horizons. Metal precipitation of the pregnant brines was likely augmented when the solutions interacted with elevated hydrocarbon components, themselves being constituents of the carbonate host rocks.

8.7.1 Fine to Medium Crystalline Dolomite

Fine-crystalline diagenetic dolomitization is pervasive and affects much of the Pine Point Formation and back-reef Muskeg Formation. Geochemistry and isotope analysis suggest that the dolomitizing fluids that caused the development of fine crystalline dolomites were similar to Middle Devonian seawater and that dolomitization occurred on or just below the sea floor (Qing, 1998a). Medium-crystalline dolomite formed soon after deposition of the Pine Point Formation carbonates by the lateral movement of Middle Devonian seawater derived from the Elk Point Basin. The restricted circulation caused evaporation of seawater from the back-reef, resulting in increased salinity and caused the precipitation of gypsum ($CaSO_4.2H_2O$), anhydrite ($CaSO_4$) and sodium salt (NaCl), thereby increasing the ratio of magnesium to calcium in the back-reef water column. As the magnesium-enriched water migrated from the back-reef basin through the barrier reef complex, the more porous carbonate lithologies were altered to dolomite (Skall, 1975; Krebs and Macqueen, 1984).

Medium-crystalline dolomite is the most abundant dolomite type in the Pine Point area. It is medium to deep brown in colour and consists of anhedral to subhedral dolomite crystals (150-250 µm) with well-defined crystal boundaries with planar extinction characteristics. Medium-crystalline dolomites display variable amounts of intra and inter-crystalline porosity (Qing, 1998). Sedimentary structures and fossils are preserved, although some fossil and biohermal structures are dissolved forming a distinct moldic porosity.

8.7.2 Coarse Crystalline Dolomite

Coarse-crystalline dolomitization affects a broad range of primary carbonate lithologies. It is estimated that 60-70% of the Sulphur Point Formation micrites and biomicrites were altered to coarse crystalline dolomite. Coarse crystalline dolomitization rarely extends above the disconformity into the overlying Watt Mountain and Slave Point Formations (Krebs and Macqueen, 1984). Crystal sizes range from 1 mm to 7 mm. In contrast to the fine-crystalline dolomite, the original fabrics of the precursor carbonates are destroyed or occur as relicts (Krebs and Macqueen, 1984). Qing (1991) describes this dolomite type as replacement, pre-mineralization dolomite.

8.7.3 White Vein Dolomite (White Matrix Breccia, Saddle Dolomite)

The most significant dolomite type that is observed across all of the Pine Point MVT Deposit is a white vein dolomite and white dolomite breccia complex (together – saddle dolomite). It formed after the sub-Watt Mountain Formation exposure during burial and the associated homogenization fluid inclusions temperatures indicate that this dolomitization process developed at temperatures exceeding maximum burial temperatures (Qing, 1991). A gradual decrease in Sr isotopes and homogenization temperatures with a corresponding increase on O isotopes eastwards along the Pine Point barrier complex led Qing (1991) to suggest a basin-scale movement of hydrothermal fluids up-dip from west to east along the Pine Point Barrier Complex. Two major tectonic events affected the Western Canada sedimentary basin:

1. Early burial occurring between the Late Devonian and Early Carboniferous.
2. Deep burial between the Late Jurassic and early Tertiary (Qing and Mountjoy, 2004).

This large-scale migration of hydrothermal fluids resulted in the widespread deposition of the Pine Point MVT mineralization. Metal deposition occurred contemporaneously and overlapped with the white dolomite (saddle dolomite) development. Extensive carbonate dissolution towards the final stages of deposition resulted in the development of radial-axial calcites (hydrothermal calcite). This phase resulted in the development of distinctive ‘blue calcite’ linings to hydrothermal cavities. The final stage of metal deposition coincided with a final hydrothermal alteration of the calcite to a ‘blue dolomite’ (‘veines bleues’ dolomite) and the latest deposition of sulphides – coarse-crystalline sphalerite and galena in the ‘veines-bleues’ dissolution cavities.

The White Vein Dolomite complex is intimately associated with the mineralization. It transects and replaces the fine-to-medium grain dolomite. The White Dolomite is the ‘hydrothermal’ dolomite at Pine Point. It has a complex age relationship with the sulphides ranging from pre, syn to post-mineralization (Krebs and Macqueen, 1984). The fluid inclusion homogenization temperatures of the saddle dolomites (90 to 100°C) support a ‘hydrothermal’ origin (Roedder, 1968). The measured initial melting temperatures of inclusions indicate that the fluid inclusions consist of a multi-component saline suite with dissolved NaCl, $CaCl_2$, KCl, and $MgCl_2$ corresponding with salinities ranging from 10 to 28 wt. % equivalent NaCl, three to eight times the salinity of seawater.

8.8 Hydrothermal Dolomitization as a Control

Pine Point mineralization is epigenetic, formed from warm, saline, aqueous solutions (similar to oil-field brines) that migrated out of the Western Canada Sedimentary basin, through aquifers, to the basin periphery and into the platform carbonate sequence. Movement of brines is facilitated by the topographic or gravity-driven fluid flow model (Garven, 1985). In this model, subsurface flow is driven away from an uplifted orogen by the hydraulic head produced by tectonic uplift and tends to be concentrated in permeable units of a foreland succession. This regional movement of hydrothermal brines resulted in the widespread hydrothermal dolomitization of barrier reef complex (stretching over 400 km from northeastern British Columbia to its erosional edge east of Pine Point).

The most fundamental control on the extent of hydrothermal alteration are lithology transitions, especially those that create dramatic changes in permeability of the rocks in carbonate platform sequences (Leach et al., 2010). In the Pine Point context, the most important are transitions between diagenetic dolostones and limestones and transitions between carbonates and shales. Chemical and mineralogical composition influenced how rocks were altered by hydrothermal fluids:

- Limestones were dissolved, dolomitized and disaggregated and eventually replaced by hydrothermal dolomite layer by layer;
- Diagenetic dolostones were hydrothermally altered on much smaller scale and locally: if altered, micritic dolomites were fractured, brecciated and delithified. Dolomitic grainstones were fractured, brecciated, flooded and recrystallized into coarse dolomite;
- Shales and argillaceous carbonate units were unaffected by hydrothermal fluids and acted as aquitards within a stratigraphic sequence.

In general, fluid migration was determined by transmissivity of various lithologies and facies within the barrier; porous reefal facies were more transmissive than massive micritic and fine-grained deep-water carbonates.

8.9 Hydrothermal Alteration Within the Main Trend

The lower part of the Sulphur Point Formation, is the host of most Main Trend, prismatic bodies and all Main Trend tabular bodies. It is bounded by two northward projections of the Muskeg formation, D3 breccia and C-horizon (Figure 8-4). Both marker horizons are dolomitic micrites interpreted as back-reef facies. The Sulphur Point Formation between the two marker horizons is dominated by bioclastic grainstones with biohermal mounds locally (stromatoporoidal boundstone, i.e. D2 facies), interpreted as a patch reef environment. The lower Part of the Sulphur Point Formation is intensely hydrothermally altered with hydrothermal fracturing, crackle breccia and recrystallized coarse grain dolomite. Pre-mineralization beige, medium grain replacement dolomite dominates the lower part of the Sulphur Point Formation. The upper part is dominated by post-mineralization, laminated replacement dolomite-calcite and disaggregated limestone replacement.

The lower micrite marker horizon is fractured, fragmented and delithified (edges of the fragments are softened, fragments are fused and deformed).

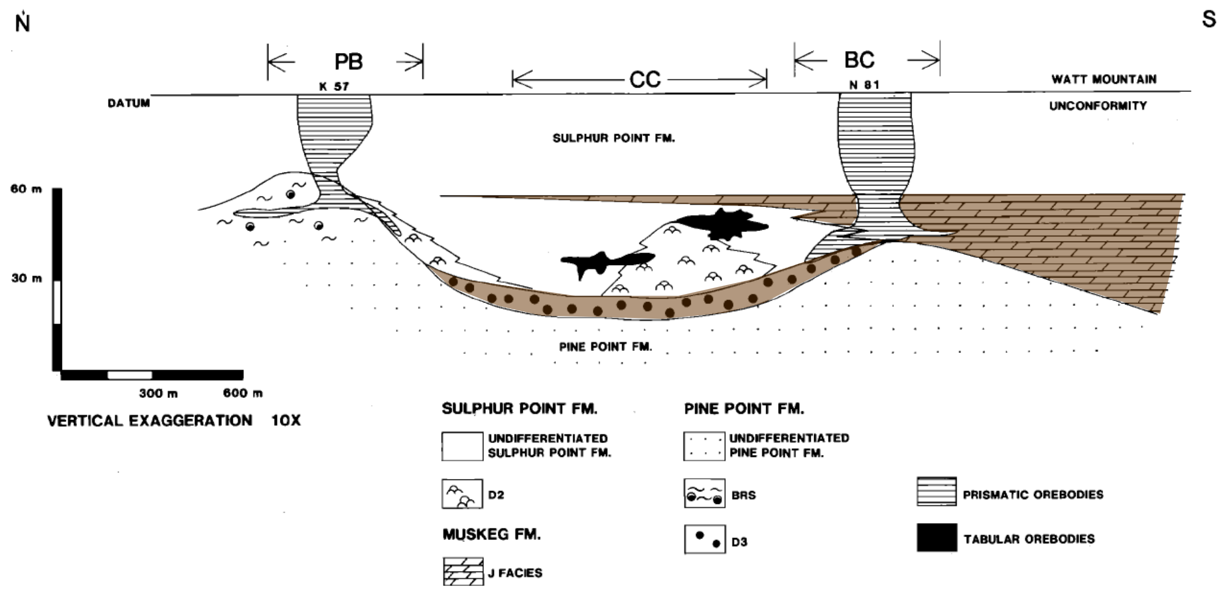


Figure 8-4: Idealized cross-section of the Main Trend showing location of sub-trends. Muskeg formation, C-horizon and D3 breccia are shaded in brown (after Rhodes at al., 1984).

The Zone between the northern limit of the Muskeg formation (known as the “J triple point” where Muskeg, Sulphur and Pine Point formations meet) and C-Horizon pinch out is known as the “CC sub-trend” of the Main trend (Rhodes et al., 1984).

8.10 Hydrothermal Alteration Within the North Trend

The flow of hydrothermal brines was controlled by shale or shaly formations which acted as aquitards (Figure 8-5). Hydrothermal alteration and mineralization occur within Windy Point formation biostromal facies which is sandwiched between Watt Mountain Formation and Buffalo River shales. Another zone of hydrothermal dolomite alteration and mineralization occurs below Buffalo River Formation shales within the Pine Point formation grainstones.

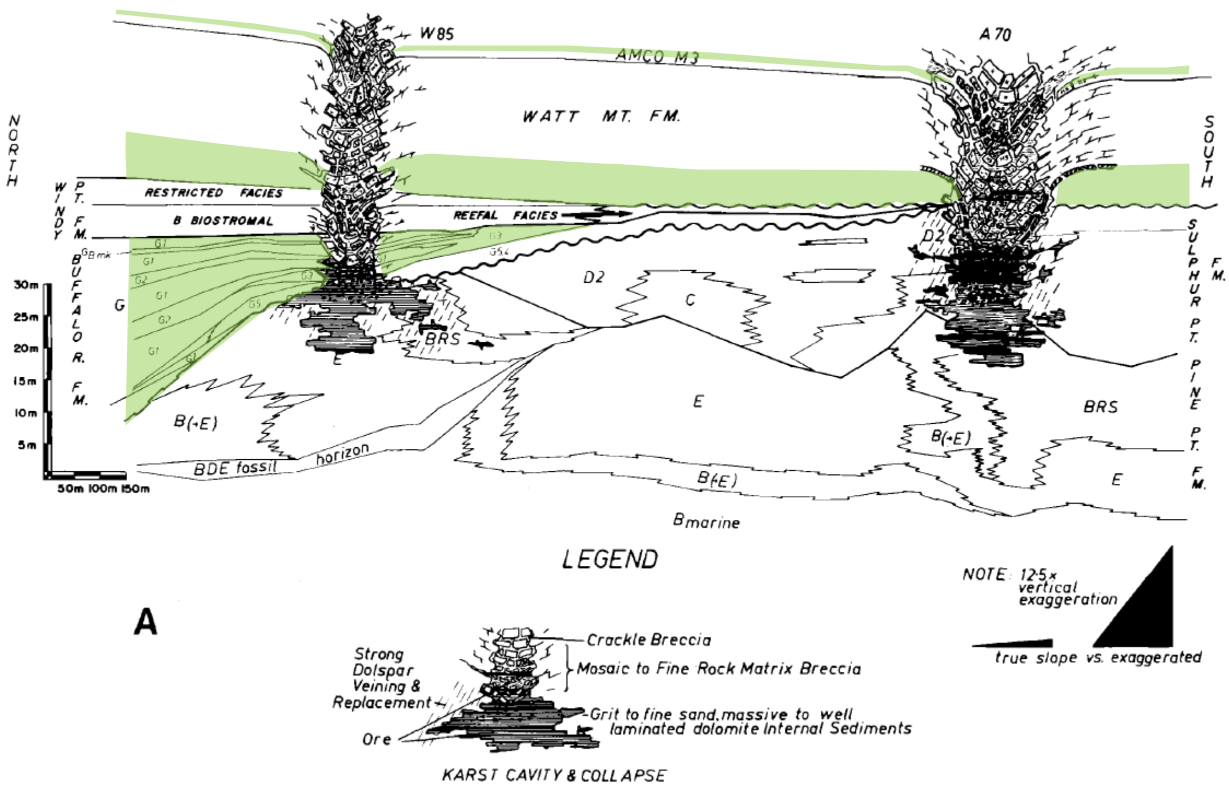


Figure 8-5: Composite section through North trend showing prismatic deposits. Shale units and formations are shaded in green (after Rhodes et al., 1984).

8.11 Hydrothermal Paleokarst

Hydrothermal activity was a regional phenomenon that lasted for many millions of years. The mineralization event was a relatively short phase, probably less than a million years. As a consequence, hydrothermal replacement dolomites are widespread throughout the barrier and mineralization is localized. Most beige, coarse hydrothermal dolomites predate mineralization, while laminated and disaggregated replacement dolomites and dolomite-calcite veins postdate mineralization.

Mineralization is closely related to and overlap with large scale dissolution, brecciation and precipitation of coarse sparry-dolomite (saddle dolomite or hydrothermal dolomite cement), Figure 8-6.

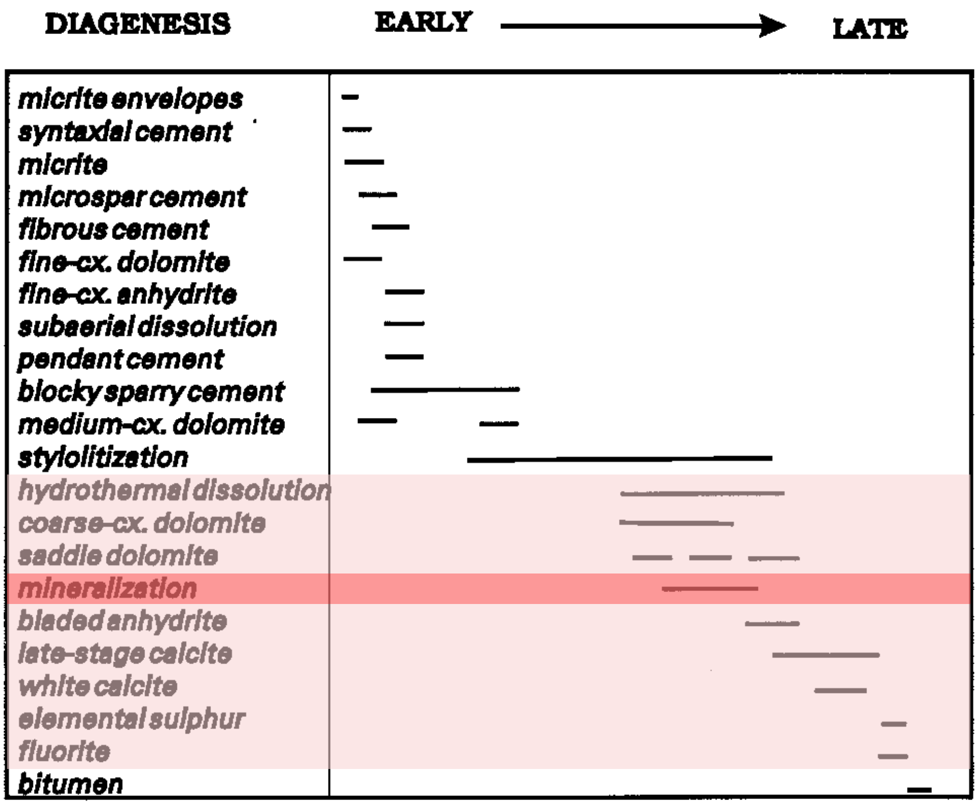


Figure 8-6: Paragenetic sequence at Pine Point (from Qing and Mountjoy, 1994). Hydrothermal processes and products are shaded in light red, mineralization phase is red.

9. EXPLORATION

The focus of activities at Pine Point in the period from January 2017 to time of this report has been on evaluating and defining the non-compliant historical resources left by Cominco Ltd. Exploration activities have been confined to a handful of exploration holes and limited-area ground gravity surveys.

9.1 Gravity

Ground gravity surveys were carried out in 2017 and consisted of 132-line kilometres and 3,151 survey stations. These surveys were designed to locate areas of excess mass that could be caused by mineralization of significant concentrations. Test lines over known unmined deposits were conducted as a benchmark. Limited drilling was unsuccessful in finding mineralization and the gravity features remain largely unexplained. The gravity survey was filed for assessment (Clemmer and Lesnikov 2018).

10. DRILLING

10.1 Drilling Methodology

10.1.1 Drillhole Selection

The process of selecting drillhole locations at Pine Point is currently a team effort between the Osisko Metals and Pine Point Geological Team and BBA. Drillhole proposals target previous drilling results across the property mainly using the extensive Cominco Ltd. database and concentrating on the many Cominco defined historical deposits.

In the late summer of 2018, BBA designed an infill program across the Pine Point Camp focusing on the large East Mill Zone, the Central Zone and the North Zone. Prior to the involvement of BBA drillholes were carefully selected by the Pine Point Geological Team based on location of historical deposits and related mineralization trends.

10.1.2 Drillhole Location/Set-up

The coordinate system in use is NAD83 Zone 11. Magnetic declination in the Pine Point region is: 16° 23.88' East.

On the Pine Point project, drill collar locations are pre-surveyed by Pine Point's surveyor using a Hemisphere S320 type Surveying Instrument. This is the most accurate method for drill collar locating in the field and it provides greater accuracy than any hand-held GPS method of locating drill collars.

A wooden stake or picket is hammered into the ground to mark the collar location. The stake is then inscribed with the predetermined drillhole #, the azimuth and anticipated depth of the hole. In the case of inclined drillholes, a separate set of clearly marked and inscribed wooden stakes/pickets mark the foresight and backsights for the alignment of the drill rig. These stakes are placed at a sufficient distance away from the collar location so as not to be disturbed by the drill contractors during their equipment installation and drill set-up. Foresights and backsights are accurately surveyed and installed by the Pine Point Surveyor.

The collar location is subsequently prepared to allow easy access of the drilling equipment as required. In many instances at Pine Point this involves brushing and some tree removal (the latter is kept to a minimum). As required, the geologists on site, visit and inspect the proposed collar location with the drill supervisor to confirm that each party is satisfied, and all health and safety criteria are met and there is sufficient space available to operate safely. In all cases care is taken to ensure that the drill platform and foot print of the area of operation is as small as possible.

10.1.3 Drillhole Orientation at Start-Up

Prior to the commencement of diamond drilling, the geologist on site, or the surveyor, will visit the drill site to confirm the alignment and set inclination of the drill rig. The drill contractor shall not initiate activity prior to having received the approval to commence from Pine Point staff on site.

The drillhole number will be confirmed with the driller or drill supervisor. Drillhole numbering will be assigned by the Pine Point Geologist, generally in a sequential manner. The drillhole number contains information on Project area, year drilled, Company Name and Hole #, as follows:

The numbering convention is as follows:

Area-Year(YY)-Company-Hole# (e.g. EM-18-PP-111). Drillhole numbers are consecutive for each individual area.

10.1.4 Drillhole Orientation during Operation

Once drilling commences the site geologist or the surveyor makes a daily trip to the drill site to maintain regular checks of the drilling progress, to inspect the drill site for any environmental or safety issues and to monitor the down-hole survey readings. The hole orientation is checked and monitored using a down hole surveying device as follows:

- First reading is taken 15 m past the end of the casing;
- Subsequent readings are taken at least every 30 m down the hole as the hole progresses;
- A final reading is taken at the end of the hole.

Readings are taken by the drill crew during operation with results recorded on special forms provided by the down hole survey instrument manufacturer. The readings include: i) operator name, ii) date and time, iii) depth of reading, iv) inclination and v) magnetic reading and vi) temperature.

The carbonate lithologies and mineralization at Pine Point do not contain significant magnetic minerals and as such magnetic readings do not vary significantly unless readings were collected too close to the drill rods (or casing). Any significant deviation in azimuth or inclination may require a repeat reading at the designated depth until a satisfactory reading is obtained or an explanation of the discrepancy has been determined.

At the end of each drillhole the site geologist, surveyor or field technician collects the down-hole survey data sheets from the drillers. The down hole survey data is added to the geological logging sheet in due course by the logging geologists. The original paper down hole survey data sheets are scanned and stored digitally and the paper copies filed in the Pine Point Mining office at Hay River.

10.1.5 Drillhole Coring

Drill cores are provided by the Drilling Contractor in either NQ (46 mm) or HQ size (60 mm). The core is collected in a standard drilling tube and the drillers carefully place the core into wooden core boxes or trays specially manufactured for this process and supplied to the drilling contractor by Pine Point Mining. The driller marks the depth in metres (m) after each run, usually every 3 m, sometimes at shorter intervals.

The drillhole is terminated by the Pine Point site geologist once the targeted depth is reached and the core at the drill site is reviewed with respect to target lithologies, alteration and mineralization. When the drilling is in an area where the expected shutdown depth is well known due to several adjacent historic holes, then the drill is shut down at the target depth without the intervention of the site geologist (this is the situation at several areas across the Pine Point Deposit, notably at the East Mill and Central areas where the holes are shut down on intersecting the upper part of the Pine Point Formation sandstones beneath the carbonate lithologies (host to the Pine Point mineralization).

Once the drillhole is terminated and the final down hole survey reading collected, the drill crew pull the rods for mobilization to the next drill site.

The drillhole is cemented and sealed utilizing a Van Ruth plug. This plug is placed at a depth where there is solid core for the plug and cemented therefore mitigating the possibility that any ground water will escape to surface. Casing may be left in the hole and cut to surface level if necessary. The casing is capped with a secure casing cap which is inscribed with the hole number and the total depth of the drillhole. Alternately, a special marker consisting of a 2 m-long piece of metal pipe is inserted into the hole and an aluminum marker plug, with the drillhole details punched onto it, is inserted into the marker tube.

The drill contractor is obligated to remove all equipment and garbage from the drill site and remove any contaminated snow, soil or overburden as per the NWT environmental guidelines. Once all of the equipment is removed and the site is cleaned, the site geologist may take a GPS reading of the drill collar location which is added to the drillhole database. However, the Pine Point Mining surveyor is tasked with the final collar pick-up and this is done as soon as possible after completion of the hole and removal of the drilling equipment. Holes drilled in mid-winter are generally surveyed in spring. Photographs of the site are also taken by the site geologist to record the condition of the drill site following completion of operations.

For winter drill sites a return visit to the site is required during the summer months to finalize clean up and review any remediation requirements. A completed drillhole inspection sheet is filled out and signed by the site geologist. The drill inspection form contains a sketch of the drill pad area that will allow an estimate of the disturbed area of the pad.

10.1.6 Drilling and Core Handling

Diamond drill cores are collected in up to 3-m lengths or runs in an NQ/HQ core barrel. The NQ core trays hold a nominal 4.5 m of core in three 1.5 m rows while HQ core trays hold a nominal 2.4 m of core in two (2) 1.2 m rows. Core breakage, faults and core tags will limit each tray to approximately 4+ (NQ) or 2+ m of core. Core is deposited into the wooden core trays at the drill rig by the driller's helper after completion of each drill run under the supervision of the driller. Core trays are numbered with a permanent marker by the driller's helper indicating the drillhole number and the sequential box number, beginning with box 1 after collaring the casing into bedrock. Numbering will be placed on the end piece of the core tray next to the first core placed in the row.

The driller's helper inserts a meterage tag (wooden block) at the down hole end of the last piece of core taken from the core tube. The block identifies the exact depth at the end of each drill run measured from the collar or stand pipe of the drill. Although the drill barrel is designed to take a 3-m run, often rock conditions or mechanical failures will dictate a run length.

The wooden depth markers are clearly marked in metres in clean and legible writing. Additional notations can be provided on additional wooden blocks indicating if bad ground, water conditions or cavities in the bedrock are encountered that result in core loss when encountered. Once the core tray is filled it is set aside, secured shut using wood screws and carefully stacked for transport to the logging facility in Hay River.

10.1.7 Receiving Core at the Hay River Warehouse

Securely-boxed drill core is transported daily to the core logging facility in Hay River. Care is exercised to ensure that the lids are securely attached to minimize core disturbance, breakage and loss during transport from the Pine Point site.

All core trays will be verified in the warehouse/logging facility, checking the wooden marker blocks before logging is initiated. If blocks do not correspond with the observed core, the shift driller and/or drill supervisor is consulted at the first available opportunity.

10.1.8 Core Logging

Logging of core is a collection of data that will be used in the short term to evaluate the success of an exploration program and in the long term (if success is inherent) to lead into resources and reserves estimations, pre-feasibility and feasibility studies. As such, observations made at the outset can have significant impact on the project going forward.

The Pine Point project has the definitive advantage of a large amount of historic drill data and knowledge with regards to lithologies, mineralization styles and controls. That said, it is necessary to bring this data into a modern classification system.

The detailed logging of core has several components, these being geological logging, geotechnical logging (lithology, structures, alteration and mineralization), sampling and photography. These components are described below.

At the outset a MS-Excel logging form was utilized for data collection before the data was imported into an Access database. In the fall of 2018 Geotic Logger software acquired by Osisko permitted direct logging onto this powerful program. The tabs are discussed below:

10.1.8.1 Header

The header sheet collects relevant data as it pertains to the drillhole including platform number (if applicable), drillhole number, locations, start and end dates, geologist, type of drilling and core sizes. The header sheet also contains the down-hole survey data collected by the drillers.

10.1.8.2 Consol

The Consol sheet is the main logging part where geological information is entered and collected. All geological characteristics including lithologies, structures, primary minerals, sulfide mineralization, sample intervals, sample numbers, etc. The main geological observations are described here.

A few specific notes are required regarding data input:

- Lithological descriptions and associated meterage are recorded for the main geological units.
- Individual intervals within the main lithological intervals are separated out and further described in terms of alteration, mineralization with or without associated sample intervals and sample numbers.
- Each interval separated out requires a description as to its affinities and a sample number breakdown if indeed it is being sampled for analysis
- The meterage sequence in the “From” to “To” columns need to be sequentially complete in order to facilitate later manipulation outside of the spreadsheet.

10.1.8.3 Geotech

As geotechnical determinations are a measure of natural characteristics of the rock, care in handling of core prior to geotechnical work is essential to prevent excessive mechanical breaking of the core.

The Geotech tab contains columns that are designed to calculate % RECOVERY as well as % RQD (Rock Quality Designation) once intervals are checked. The following sections describe procedures to ensure the correct and representative measurements are collected.

10.1.8.4 Core Fitting

The Geotechnician is responsible for realigning the core to achieve best fit along all natural and mechanical fractures. This assists in calculating core recovery, improving core photography results, and improving accuracy in the distance determinations during logging. Core is placed on standard sloped logging tables at the logging facility. A best fit for each piece of core is determined.

Core pieces that do not fit might be categorized as:

- Lost core areas
- Mechanical breakage or grinding in the core barrel (usually with rotation scarring)
- Misplaced core, or reversed runs.

Misplaced core occurs when a piece of core is put into the tray in the wrong location when the driller's helper is removing the core from the core barrel. Causes can include dropped core, during mechanical breaking of core to fit a row, or if there is a core blockage, and the driller's helper releases the retaining ring and removes the core from the other end of the barrel.

Any error in the placement of core is noted and immediately reported to the driller or drill supervisor as appropriate.

10.1.8.5 Core Recovery

Core recovery is collected from all drillholes. The core recovery is calculated by measurement in centimeters of core in the core tray divided by the centimeters claimed to be drilled on the meterage blocks. This number multiplied by 100 is recorded as percent recovery. Core recovery is recorded for each drill run. Specific areas of loss are noted if possible and marked by placement of a wooden marker and the estimated loss. 100% core recovery is ideal however, it is not always possible because of ground conditions or sometimes loss of drill core during the coring process e.g. grinding, etc.

10.1.8.6 Rock Quality Designation (RQD)

The rock quality designation is designed to give qualitative and quantitative information on the stability of rock surrounding and included in mineralized material. This information is used to determine the mineability and rock control procedures that will be required to extract the mineralized material.

RQD is a quantitative index of rock quality based on a core recovery procedure in which the core recovery is determined incorporating only those pieces of hard, solid core longer than twice the diameter of the core. For NQ core the nominal diameter is 5 cm, so the length index is 10cm. Shorter lengths of core are ignored. RQD is determined for each core run as these are the only definitively known distance markers. RQD is determined using the following formula:

RQD (%) =100 x the sum of the length of the core pieces equal to or longer than 10 cm length of the core run.

It is important to distinguish between mechanical breaks and natural breaks identified in the core.

RQD is valid for solid core only and should not be used for very poorly disaggregated materials such as highly weathered rock, clays or un-cemented aggregates.

10.1.9 Core Photography

All drill core is photographed. The object of core photos is to have a digital image record of sufficient detail to clearly see core features prior to destructive sampling procedures. This record can be used later to qualify rock quality features and to examine core images against geological logging if the core is unavailable for examination. The photos are also used as required during the construction of geological sections.

- Core is photographed following fitting, core recovery and rock quality designation. The camera is mounted on a special mobile cart with the camera set at an appropriate height to photograph the core as laid out on the logging tables;
- All depth marker blocks should be clean, legible and visible in the photograph. The 'From-to' for top and bottom core depth is clearly marked on the wooden core tray as well as the box number and drillhole name;
- The core is photographed dry and wet;
- Digital photographs are saved onto the appropriate drillhole folder for the project database. Additional close-up photographs may be taken of mineralized intersections, structural features or other items of note by the logging geologist during the logging process.

After the core is photographed, the core is assigned to a logging Geologist for geological logging and sample selection.

10.1.10 Sampling (Core Sample Selection)

Samples are broken at major rock code contacts to represent homogeneous units. The minimum sample interval in the hole will be not less than 50 cm. The maximum sample interval will not exceed 150 cm (20cm for density determination samples). No sample will cross a major rock boundary, alteration boundary or mineralization boundary.

Sampling intervals are determined by the geologist during logging and marked on the core boxes or on the core itself using colored lumber pencils with a line drawn at right angles to the core axis. Samples are numbered in consecutive order utilizing two-way sample tag books that are provided by the ALS Geochemistry. The sample sequence includes blank samples, duplicate samples and Standard Reference Materials (SRM's) that are inserted into the sample stream using sample numbers that are in sequence with the core samples.

Sample intervals, sample numbers and QC samples are noted in the drill log work sheet. A separate sheet for the sampling numbers is also produced. A copy of this sheet is later given to the sampler/cutter.

All sample tag books once filled are scanned and filed in the assay database for reference.

10.1.11 Core Sampling (Core Saw Splitting)

A Geotechnician trained in core cutting procedures executes the core cutting at the Hay River warehouse. The logging geologist has already clearly marked out all pertinent cores for cutting and sampling. The geologist also staples a paper sample tag containing a sample number corresponding with the required sample interval at the start of the sample interval. The logging geologist also staples a metal tag containing the sample number on to the box. This is a permanent sample reference which will remain on the wooden core tray. The Geotechnician removes the paper sample tag and places it inside of the plastic bag.

The core is sawn with a diamond saw and one half of the core sample is placed in a sample bag and the remaining half returned to the core box. The sample is taken consistently from the same half of the split core, using the red centerline drawn on the core as a reference. The cut core will be returned to the core box in the same position as it was removed so as not to rotate the core or reverse the down-hole direction of the core. If the above procedure is carefully followed the core remaining in the tray will retain its "fitted" appearance.

The sample tag number is also written on the outside of the sample bag using a permanent marker. The bag will then be closed using a zip tie and stored in sequence prior to sample dispatch preparation.

For quality assurance purposes "DUPLICATE" core samples are generated by cutting the ½ core sample in half to produce 2 x ¼ duplicate core samples. Care is taken to ensure that both ¼ core samples are virtually identical and thus representative (even though in a lesser amounts) to the original ½ core sample. One of the ¼ core intervals is placed in a sample bag for analysis and included with the sample batch for dispatch. The sample bags are prepared in the same manner as the original sample and immediately follow the original core sample with the corresponding sample number.

Sample bags are packed in large "rice" bags with 'full' weight not exceeding 22Kg and the rice bag is sealed with a numbered Security Tag seal which is only 'broken' or opened at the assay laboratory operated by ALS Geochemistry in Yellowknife.

A "STANDARD" sample consisting of material of known metal content and internationally recognized and verified is included in the sample sequence by the trained core sampler. Pine Point Mining includes a Standard after each 50 samples. Similarly, a "BLANK" is included in the sequence as part of the QA/QC process. Blank material is technically devoid of any metals. BLANKS and STANDARDS are stored in a designated secure area in the Hay River warehouse. There is never any written reference to the location of any control samples on sample bags, sample tags or dispatch documentation for the assay lab.

The range of sample numbers inside the bag is written on the 'rice' bag, along with the address of the analytical laboratory. For shipping purposes, the 'rice' bags are numbered sequentially and marked as per quantity (e.g. Bag 1 of 15, etc.).

The first bag in the sequence contains the Laboratory Sample Submission Form as well as a hard copy of the sample dispatch sheet and this bag is labeled "Laboratory Instructions Enclosed". The sealed rice bags are stored in the warehouse in Hay River until shipping to the laboratory is Yellowknife (which currently takes place every Thursday afternoon).

The lab is notified by email that the samples are en route and is instructed to notify the Pine Point Mining in Hay River office when the samples arrive at the prep lab in Yellowknife. A digital copy of the sample submission form as well as the sample dispatch list is emailed to the laboratory manager once the samples have left Hay River (Hay River Manitoulin Transport Depot).

10.1.12 Core Storage

Following sampling the core trays are labelled using a metal tag. The core tray metal tags are marked with the hole number, the tray number, and the From-To meterage. The final tray in a hole is marked with end of hole ("EOH").

The core trays are stored on pallets at PPM's permanent storage facility in Hay River. The core boxes are stored on the pallets in a crisscross manner to prevent tipping with the metal tags clearly visible.

10.2 2017 Drilling Program

In 2017, 25 holes totalling 2,276 m were drilled testing areas outside existing resources. Some of the holes targeted areas of mineralization intersected by Cominco in the past and others were directed at gravity features generated by the ground surveys. The holes are listed in Table 10-1.

Table 10-1: 2017 Drilling program

Hole-ID	Area	Easting	Northing	Elevation	Azimuth	Dip	Final Depth	Target
L37-17-PP-001	L37	640468.0	6750022.0	218.51	0	-90	50.00	Test tabular mineralization
N38-17-PP-001	N38	640471.0	6749496.0	220.42	0	-90	50.00	Between holes M-40-203 and 77-34-03 and 77-34-04
N42-17-PP-001	N42	639319.0	6749076.0	224.82	0	-90	74.00	Center of the gravity anomaly
N42-17-PP-002	N42	639483.0	6749072.0	221.96	260	-50	92.00	West of the gravity anomaly
N42-17-PP-003	N42	639280.0	6749034.0	224.42	135	-70	79.95	South end of the gravity anomaly
N42-17-PP-004	N42	639469.0	6748865.0	222.10	0	-90	77.00	east end of N42 pit above sp showing
N42-17-PP-005	N42	639343.0	6749012.0	222.74	0	-90	65.00	East side of the gravity anomaly to the south
N42-17-PP-006	N42	639479.0	6748327.0	227.05	0	-90	74.00	Between N42 and O42 pits
N42-17-PP-007	N42	638956.0	6748735.0	226.26	290	-50	80.00	Gravity feature
EX-17-DBL-001	EX	626566.0	6743274.0	212.15	0	-90	73.50	Test of tabular zone between L65 and K77
EX-17-DBL-002	EX	626866.0	6743396.0	212.25	0	-90	79.50	Test of tabular zone between L65 and K77
EX-17-DBL-003	EX	627028.0	6743485.0	212.26	0	-90	107.00	Test of tabular zone between L65 and K77
EX-17-DBL-004	EX	626190.0	6743115.0	211.61	0	-90	116.00	Test of tabular zone between L65 and K77
EX-17-DBL-005	EX	622719.0	6741363.0	209.05	160	-55	134.00	Test of tabular zone between L65 and K77
EX-17-DBL-006	EX	623913.0	6745007.0	210.21	0	-90	107.00	Test of tabular zone between L65 and K77
EX-17-DBL-007	EX	623643.0	6742034.0	209.51	160	-55	143.00	Test of tabular zone between L65 and K77
EX-17-DBL-008	EX	623581.0	6742305.0	206.57	0	-90	86.00	Test of tabular zone between L65 and K77
EX-17-PP-009	EX	639294.0	6746252.0	222.98	0	-90	135.50	Gravity feature
EX-17-PP-010	EX	639868.0	6746077.0	220.66	340	-60	95.00	Gravity feature
EX-17-PP-011	EX	639056.0	6745980.0	222.51	236	-65	92.00	Gravity feature
EX-17-PP-012	EX	637671.0	6747103.0	225.76	0	-90	80.00	Gravity feature
EX-17-PP-013	EX	639540.0	6747767.0	227.00	0	-90	95.00	Gravity feature

Hole-ID	Area	Easting	Northing	Elevation	Azimuth	Dip	Final Depth	Target
EX-17-PP-014	EX	638198.0	6746994.0	226.63	0	-90	86.00	Gravity feature
EX-17-PP-015	EX	640060.0	6746279.0	218.71	0	-90	86.00	Gravity feature
EX-17-PP-016	EX	638678.0	6746129.0	226.54	60	-55	119.00	Gravity feature

Only one hole, N38-17-PP-001 returned significant results intersecting 3.15 m of 2.73% Lead and 5.87% Zinc. The intersection suggests more tabular mineralization can be defined between the L37 pit and the M40 underground mine.

The other holes returned only minor values and were unsuccessful in finding any significant mineralization or explaining the gravity feature drilled.

10.2.1 Twinning Program

In order to confirm grades and widths of the mineralization reported in historic drillholes, PPML twinned 24 drillholes drilled by Cominco Ltd. in historical deposits L35 and L36 within the East Mill Zone. The area of investigation is within a relatively continuous tabular deposit with more localized sulphide concentrations. A total of 24 twin holes within a SW-NE direction were drilled in an area measuring approximately 2,750 m by 450 m. Coverage is uneven with 13 drillholes clustered in SE third of the drilled area and remaining 11 drillholes spread over NE half of the drilled area. Spacing between several holes within the SE cluster is about 60 m, which corresponds to the historical mine grid and drillholes drilled at 60 m (200 foot) centers. The gap between the SE cluster and NE drillholes is about 500 m, reflecting the distance between the two historical deposits.

The twinning program was conducted in June and July 2017. A total of 1,293.50 m in 24 vertical drillholes was drilled. A total of 689 samples (including QA/QC inserts) were sent to Bureau Veritas Vancouver lab for assaying. A total of 586 assay results were compared with 287 assay results from the twinned historic drillholes.

10.2.2 Compositing

Since the assay results in historic drillholes suggest one mineralized interval in most drillholes, the goal was to define a single interval with grade above 1% combined lead and zinc, Pb+Zn(%)>=1 in all drillholes. To remain consistent, all the assays (both the twin hole and the twinned hole assays) were composited using the same criteria. The criteria for the first pass composites:

- Continuous mineralized intervals with combined lead and zinc content equal or larger than 0.5% Pb+Zn(%)>=0.5.

Isolated mineralized intervals were joined in one composite if separated by less than 3.50 m.

By applying first pass criteria, single mineralized interval composites were calculated for 16 historic drillholes and 12 twin holes. For the remaining holes distance separating mineralized intervals was increased to up to 7 m. Second pass criteria were as follows:

- Continuous mineralized intervals with combined lead and zinc content equal or larger than 0.5% Pb+Zn(%)>=0.5

Isolated mineralized intervals were joined in one composite if separated by less than 7 m.

Composites created in the second pass meant grade dilution and longer mineralized intervals. In few cases, mineralized intervals had to be omitted because they were separated by more than 7 m from the main mineralized interval.

Final composites are shown in Table 10-2. Each historic drillhole is followed by the corresponding twin hole. First pass composites are labeled as Comp-1, second pass composited as Comp-2.

Table 10-2: Final Composites with Pb+Zn(%)>=0.5%

DDH	FROM	TO	WIDTH	Pb av	Zn av	Pb av+Zn av	
3807	9.75	31.09	21.34	0.53	2.31	2.84	Comp-1
L35-17-DBL-023	11.00	29.25	18.25	0.39	1.47	1.86	Comp-1
4038	18.90	34.14	15.24	0.27	0.96	1.23	Comp-2
L35-17-DBL-018	13.40	24.75	11.35	0.03	0.96	0.99	Comp-2
4095	9.14	12.80	3.66	1.58	5.33	6.91	Comp-1
L35-17-DBL-017	9.40	17.80	8.40	0.14	0.93	1.08	Comp-2
K-32-50	9.75	17.37	7.62	5.76	5.61	11.38	Comp-1
L35-17-DBL-015	10.50	19.05	8.55	9.65	9.16	18.81	Comp-1
K-32-71	9.14	17.07	7.93	0.38	2.94	3.32	Comp-1
L35-17-DBL-016	10.70	16.60	5.90	0.30	2.51	2.80	Comp-1
K-35-024	30.48	37.80	7.32	0.26	4.04	4.30	Comp-1
L35-17-DBL-021	25.20	26.55	1.35	0.22	2.14	2.36	Comp-1
K-35-073	24.69	38.10	13.41	0.64	3.51	4.16	Comp-1
L35-17-DBL-020	24.15	31.60	7.45	0.45	3.11	3.56	Comp-1
K-35-083	24.08	34.14	10.06	0.39	2.29	2.68	Comp-2
L35-17-DBL-022	30.30	31.60	1.30	3.24	6.66	9.90	Comp-1
K-35-114	28.96	35.66	6.70	0.08	1.39	1.47	Comp-1
L35-17-DBL-019	12.90	13.40	0.50	0.45	2.52	2.97	Comp-1
K-35-144	35.05	36.58	1.53	0.10	0.60	0.70	Comp-1
L35-17-DBL-024	31.00	32.90	1.90	0.12	1.33	1.46	Comp-1
K-35-205			0.00	0.00	0.00	0.00	Comp-1
L35-17-DBL-026	24.00	29.00	5.00	0.08	6.96	7.04	Comp-1
K-35-216	23.47	35.66	12.19	0.28	1.10	1.38	Comp-2
L35-17-DBL-025	23.38	31.90	8.52	0.30	1.46	1.76	Comp-2
L-35-046	31.39	47.85	16.46	1.10	3.23	4.33	Comp-1
L36-17-DBL-010	29.56	39.80	10.24	1.08	1.35	2.43	Comp-2
L-35-062	30.78	39.62	8.84	0.60	2.96	3.56	Comp-2
L36-17-DBL-009	31.20	50.20	19.00	0.28	2.87	3.16	Comp-2
L-35-087	28.96	45.72	16.76	1.54	2.53	4.07	Comp-1
L35-17-DBL-013	26.65	42.45	15.80	0.17	2.32	2.49	Comp-1
L-36-310	33.83	44.20	10.37	0.83	4.21	5.04	Comp-1
L36-17-DBL-003	34.00	45.50	11.50	1.06	5.56	6.63	Comp-1
L-36-313	26.82	33.83	7.01	0.69	10.40	11.09	Comp-1
L36-17-DBL-002	28.45	46.90	18.45	0.59	1.25	1.84	Comp-2
L-36-343	38.10	47.55	9.45	0.16	1.86	2.02	Comp-1
L36-17-DBL-004A	26.80	42.70	15.90	0.32	1.90	2.21	Comp-2
L-36-366	22.25	37.49	15.24	0.86	2.51	3.36	Comp-1
L36-17-DBL-005	24.30	36.90	12.60	0.24	1.19	1.43	Comp-2

It should be noted that the historic drillhole K-35-205 lacks assay data and is presumed unmineralized. Historic drillhole K-35-144 is only weakly mineralized (1.53 m @ 0.70% Pb+Zn).

Length of voids reported in several PPM twin holes was incorporated into the length of composites. In that way, voids effectively became 0% grade intervals and lead to certain grade dilution. A total of 8.55 m of voids was incorporated into the mineralized intervals. This represents 3.82% of the sum of all mineralized intervals in twin holes (223.90 m).

In the following two bar graphs, mineralization width or mineralization grade of historic holes and twin holes are shown side by side - historic drillholes are red, and corresponding twin drillholes are blue.

Variability of mineralization width and grade both within the historic drillhole population and twin hole population is evident. Also evident is variable difference in widths and grades in paired drillholes.

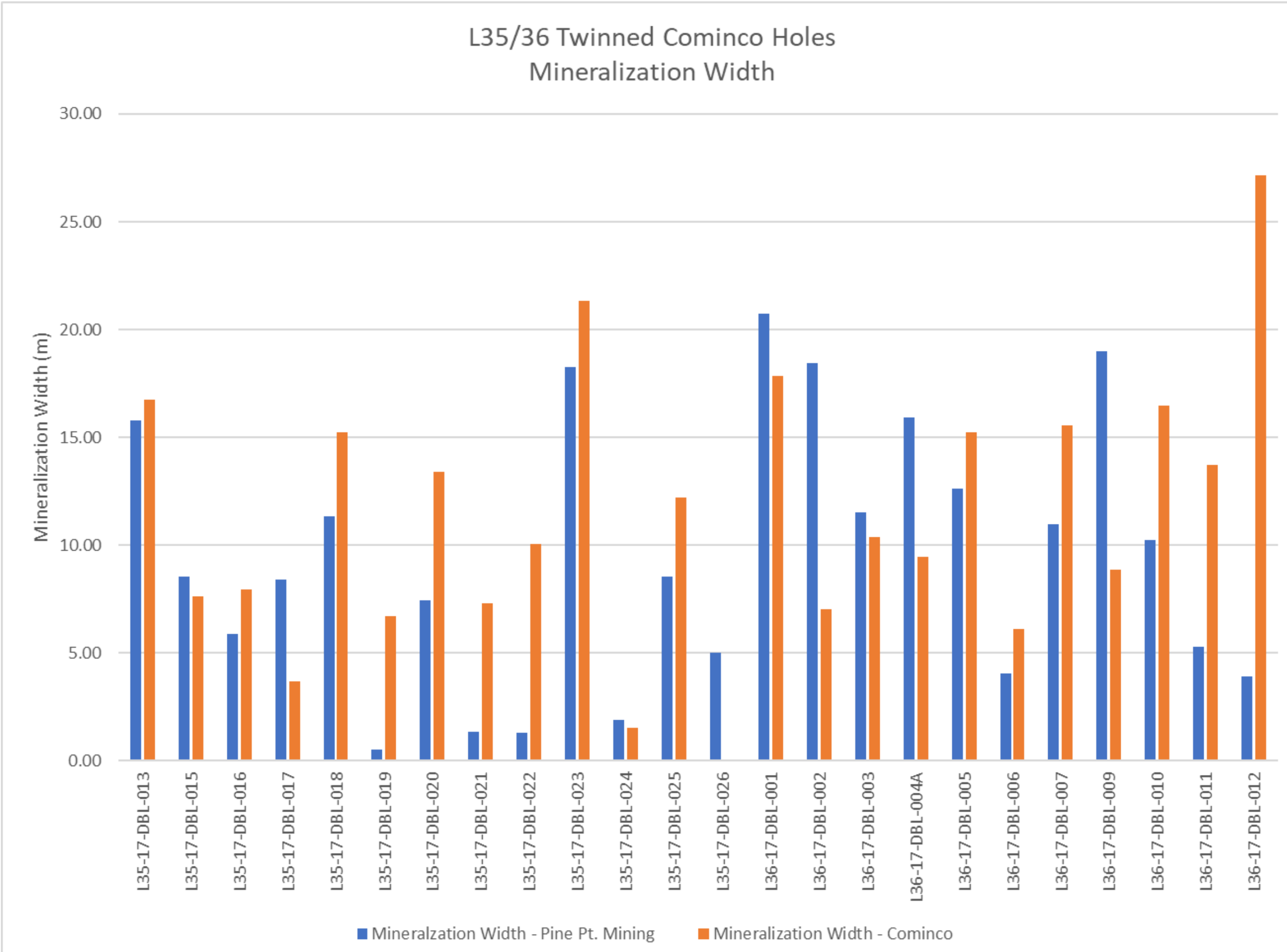


Figure 10-1: Twinned Cominco Ltd. holes (mineralization width)

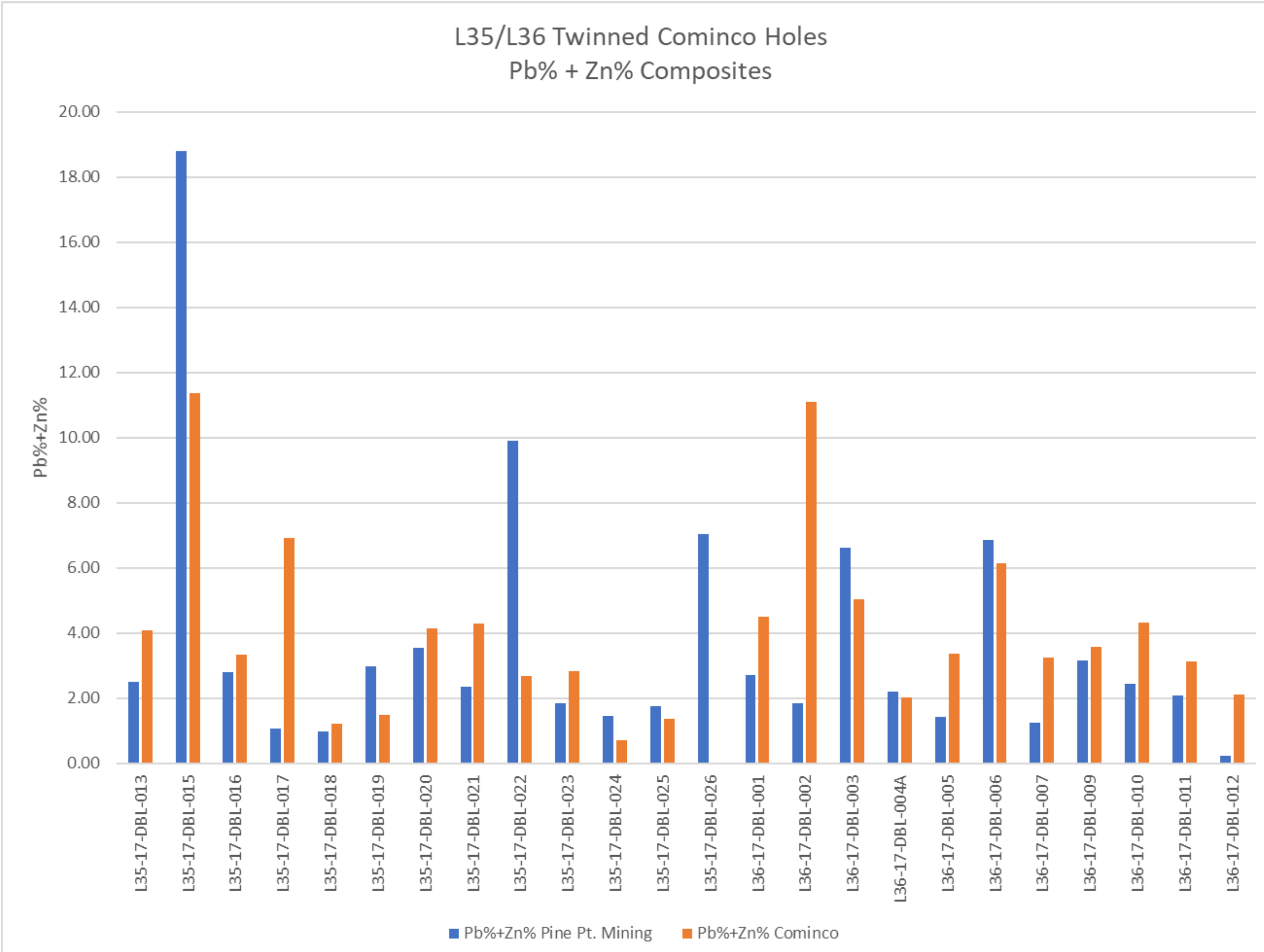


Figure 10-2: Twinned Cominco Ltd. holes (combined Pb+Zn grade)

10.2.3 Data Analysis

The following are descriptive statistics parameters for mineralization width, historic holes to the left, twin holes to the right:

Table 10-3: Descriptive statistics parameters for mineralization width

WIDTH_COM	
Mean	11.31
Standard Error	1.285
Median	10.215
Mode	15.24
Standard Deviation	6.2952
Sample Variance	39.629
Kurtosis	0.4515
Skewness	0.4637
Range	27.13
Minimum	0
Maximum	27.13
Sum	271.44
Count	24

WIDTH_PPM	
Mean	9.4546
Standard Error	1.2521
Median	8.535
Mode	#N/A
Standard Deviation	6.1339
Sample Variance	37.625
Kurtosis	-0.974
Skewness	0.3132
Range	20.25
Minimum	0.5
Maximum	20.75
Sum	226.91
Count	24

The mean for the historic holes is 19.6% higher than the mean for twin holes. Standard deviation and variance are relatively high for both data sets. Similar values for standard deviation indicate similar range within which the data is spread around the means in both data sets. Negative kurtosis in twin hole data set indicates broad data (platykurtic) distribution curve.

Following two tables list descriptive statistics parameters for mineralization grade (Pb%+Zn%):

Table 10-4: Descriptive statistics parameters for grade

Pb+Zn%_COM	
Mean	3.8741
Standard Error	0.5706
Median	3.3439
Mode	#N/A
Standard Deviation	2.7954
Sample Variance	7.8145
Kurtosis	2.5479
Skewness	1.4904
Range	11.375
Minimum	0
Maximum	11.375
Sum	92.978
Count	24

Pb+Zn%_PPM	
Mean	3.6617
Standard Error	0.8097
Median	2.3939
Mode	#N/A
Standard Deviation	3.9668
Sample Variance	15.736
Kurtosis	9.0799
Skewness	2.7896
Range	18.586
Minimum	0.2254
Maximum	18.812
Sum	87.881
Count	24

Means of both data sets are very similar (historic mean is 5.80% higher than the twin holes mean), and both data sets display high variability. This is especially true for twin holes data. High kurtosis and skewness values indicate data distribution which is not normal and asymmetrical.

To confirm that data distribution is not normal, normal probability plots were created for the differences in both mineralization width and mineralization grade data sets:

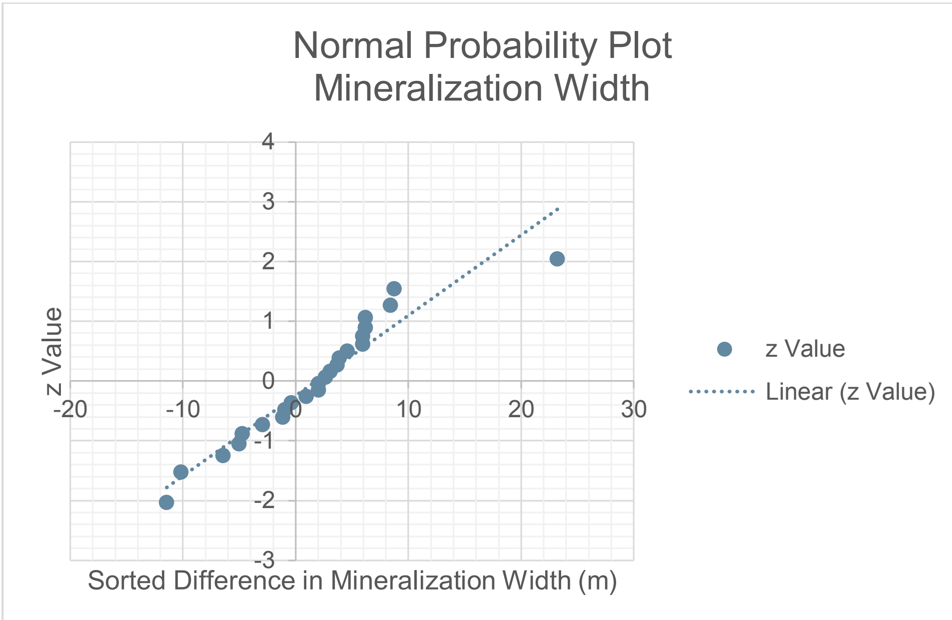


Figure 10-3: Normal probability plot mineralization width

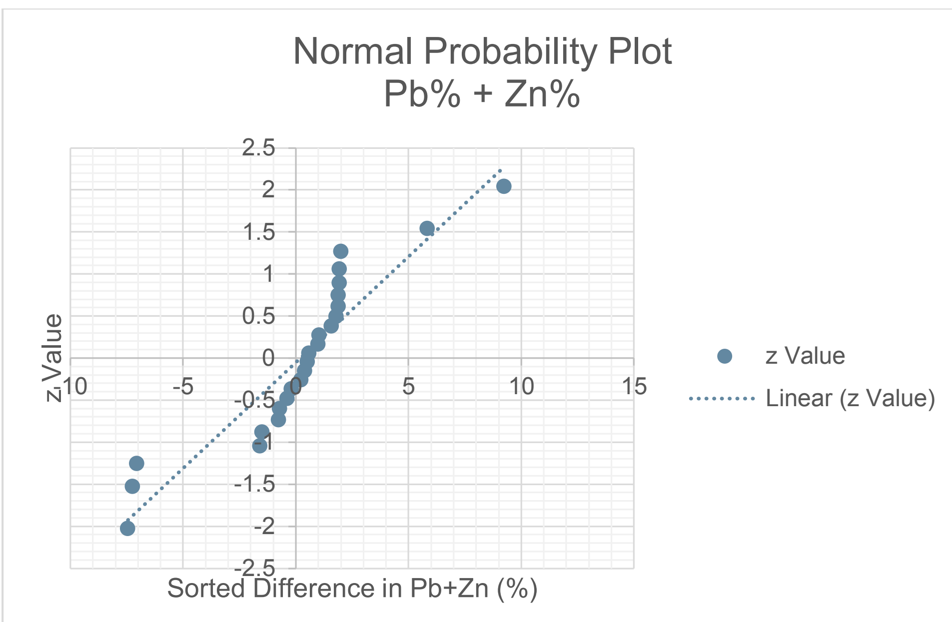


Figure 10-4: Normal probability plot

Points scattered far from the straight trendline indicate that both data sets are not normally distributed. This means that the assumption of data distribution normality has been violated and paired t-test cannot be utilized to analyse either of the two data sets. The option would be the non-parametric Wilcoxon (paired) signed rank test. Results of the Wilcoxon test suggest low probability that there is no difference between the mineralization width or grade in historic and twin holes.

At the usually used 5% level of significance there is no sufficient evidence to suggest that there is a difference between mineralization width in historic and twin holes. At the 20% level of significance there is a statistically significant evidence that the median difference is positive (that the mineralization width in historic holes is larger than mineralisation width in twin holes).

Results of the Wilcoxon test of the mineralization grade (Pb+Zn%) are similar. At the 20% level of significance, there is a statistically significant evidence that the median difference is positive (that the mineralization grade in historic holes is larger than the mineralization grade in twin holes).

The level of significance of 20% means acceptance of possible 1 in 5 random result and is not normally used in statistics. Signed rank test assumes symmetric distribution within the difference set. The skewness of the mineralization width difference set is 0.70. The skewness of the mineralization grade difference set is -0.29. This may have affected signed rank test results.

10.3 Conclusions

Twinning program has successfully confirmed mineralization in historic drillholes – mineralization >=1% Pb+Zn was intercepted in 23 out of 24 twin holes. Average grade in twin holes is roughly equal to the average grade in historic drillholes.

Major difference between the twin holes and historic holes is in mineralization width. Average mineralization width in historic holes is 19.6% higher than the average mineralization width in twin holes. Mineralization continuity in historic holes appears to be better than in twin holes (larger number of historic holes composited in first pass).

Twinning program has also demonstrated how significant is the variability of the tabular mineralization – grade and width of mineralization in both the historic and twin drillholes vary over a wide range of values. Tabular karst channels can be only weakly mineralized, mineralization can be discontinuous and variable.

There are 15 twin holes in which mineralization widths is at least 50% of the mineralization widths in the corresponding historic drillhole (or in case when mineralization width is larger in twin hole, historic width is at least 50% of the mineralization width in twin hole). These 15 holes represent 62.5% of the 24 twin holes.

There are also 15 twin holes in which mineralization grade is at least 50% of the mineralization grade in the corresponding historic drillhole (or in case when mineralization grade is larger in twin hole, historic grade is at least 50% of the mineralization grade in twin hole). These 15 twin holes represent again 62.5% of the 24 twin holes.

Finally, there are 12 twin holes in which both mineralization grade and widths are at least 50% of the mineralization grade and widths in historic holes (and vice versa, in case mineralization width or grade are larger in twin hole). These 12 twin holes are 50% of 24 twin holes drilled.

In other words, to get two holes with comparable mineralization widths, three historic holes had to be twinned. To get two holes with comparable mineralization grades, again three historic holes had to be twinned. Finally, to get two holes with comparable mineralization grade and width, four historic holes had to be twinned.

10.3.1 2018-2019 Drilling Program

An in-fill drilling program was still underway at the time of writing and results are pending. As of December 31, 2018, 605 drillholes totalling 41,379 m of in-fill drilling were completed, but not included in the MRE as results arrived after the September resource cut-off. They are not included in this MRE. The objective of this ongoing program is to upgrade the Inferred Mineral Resource to the Indicated category by decreasing drill spacing to 30 m from the current average drill spacing of 40 m to 60 m. The Company expects to drill 900 additional holes totalling approximately 49,000 m in the remainder of 2018 and in 2019 to meet this objective.

Figure 10-5 to Figure 10-10 show the location of both recent and historical drillholes throughout the property.

Figure 10-11 to Figure 10-16 show typical cross-section views throughout the property.

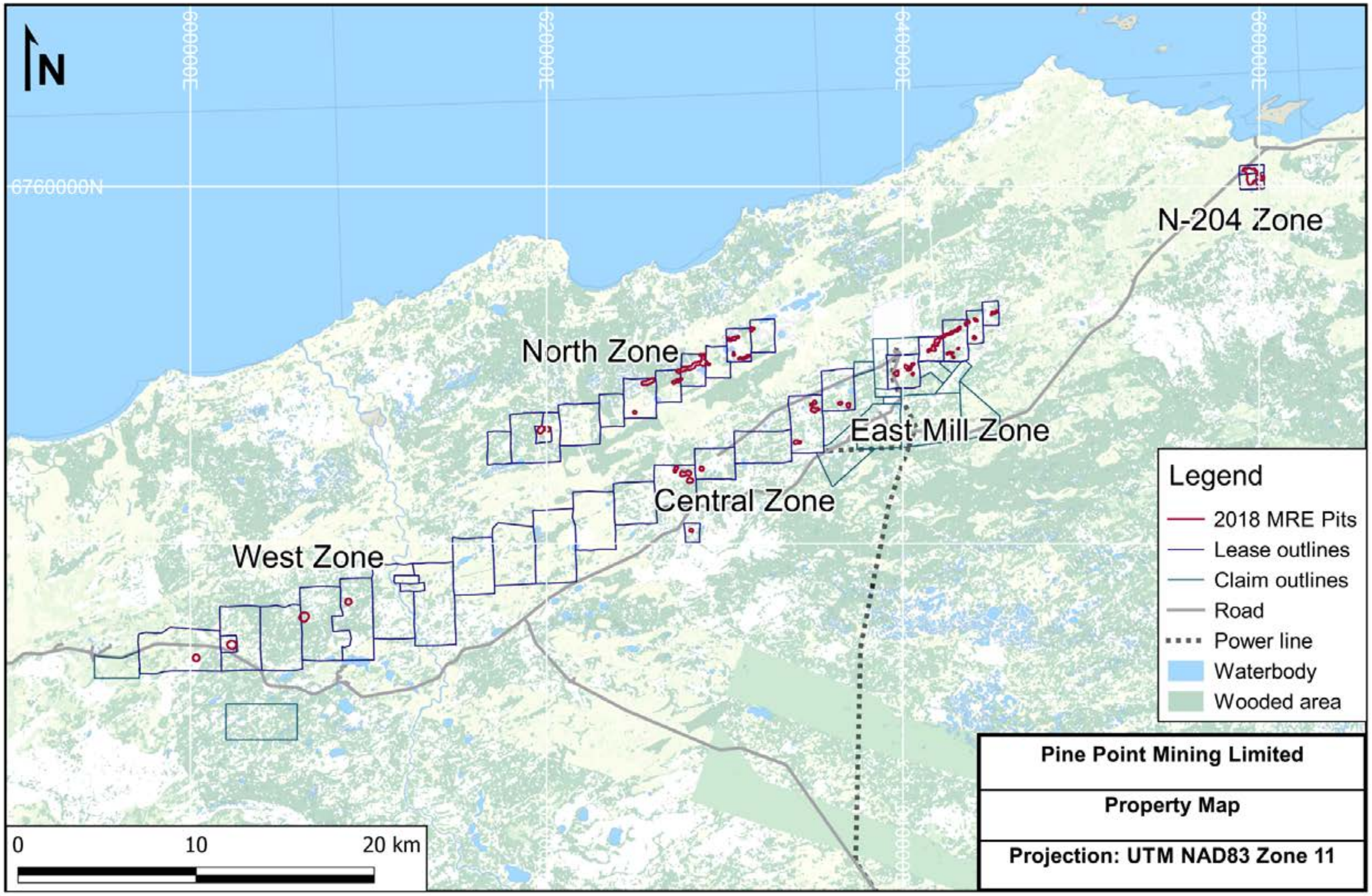


Figure 10-5: General view of the Property showing the location of the different zones

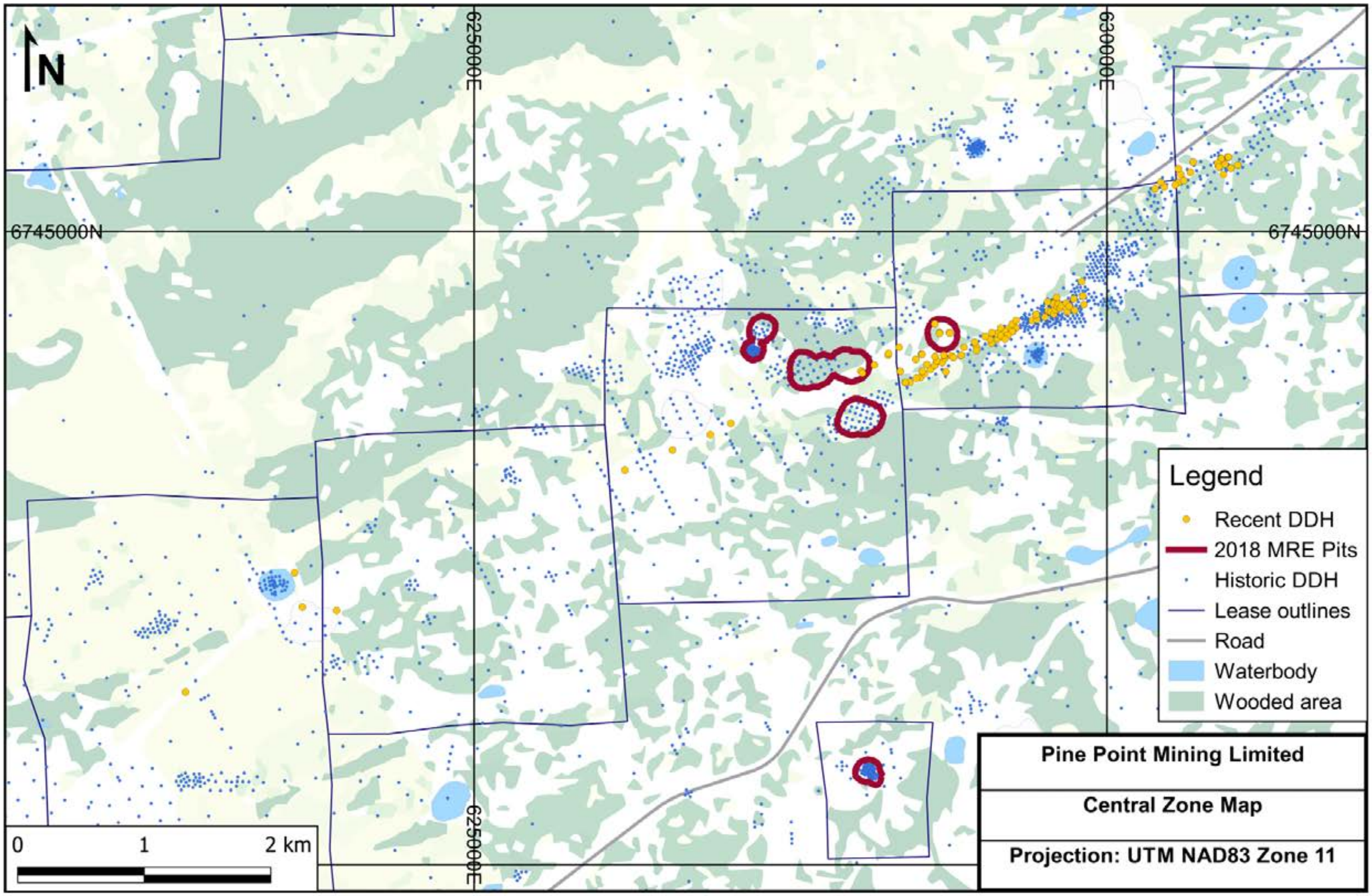


Figure 10-6: Central Zone Map showing the location of historical collars (blue) and recent holes from 2017-2018 (orange)

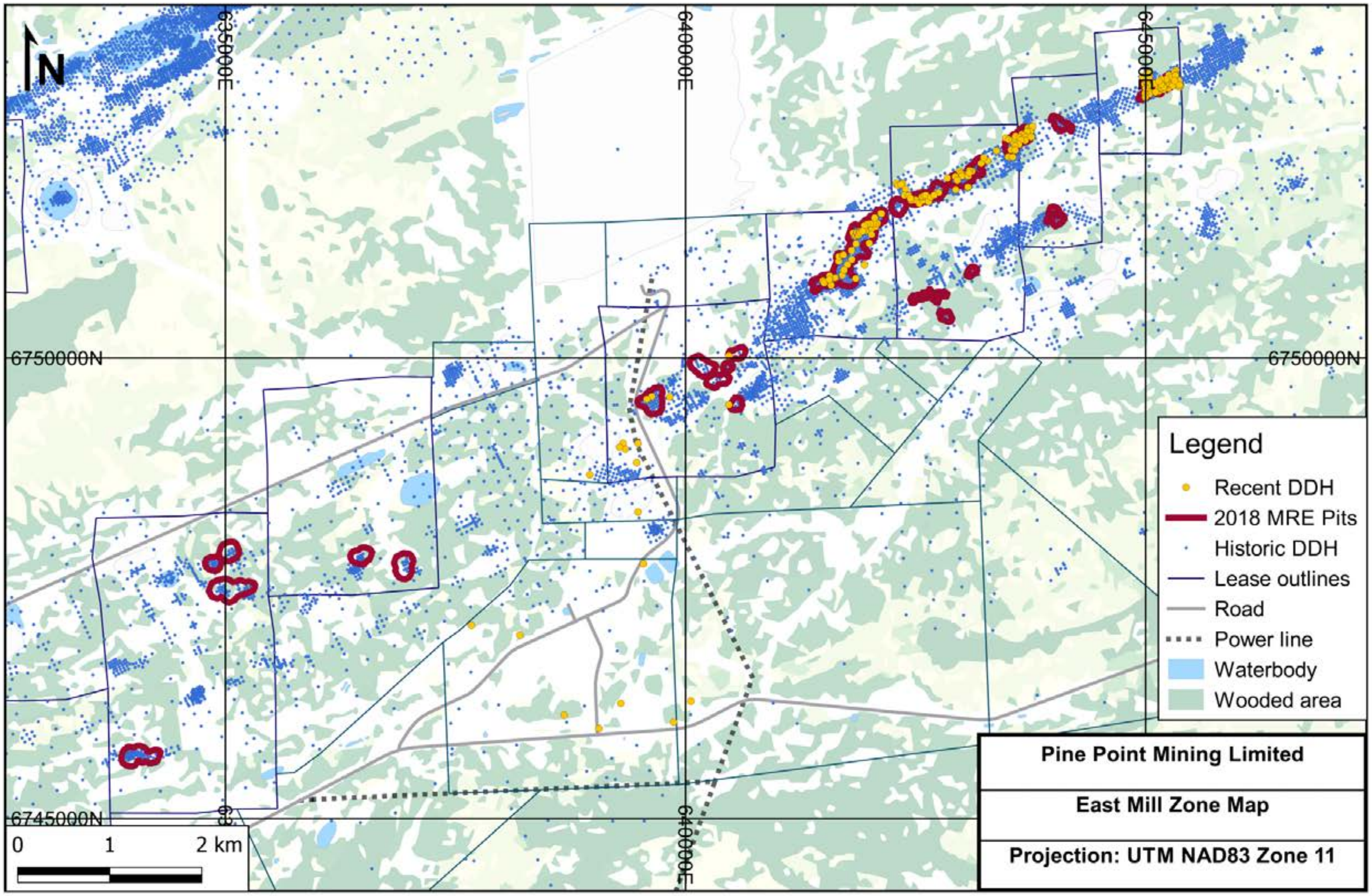


Figure 10-7: East Mill Zone Map showing the location of historical collars (blue) and recent holes from 2017-2018 (orange)

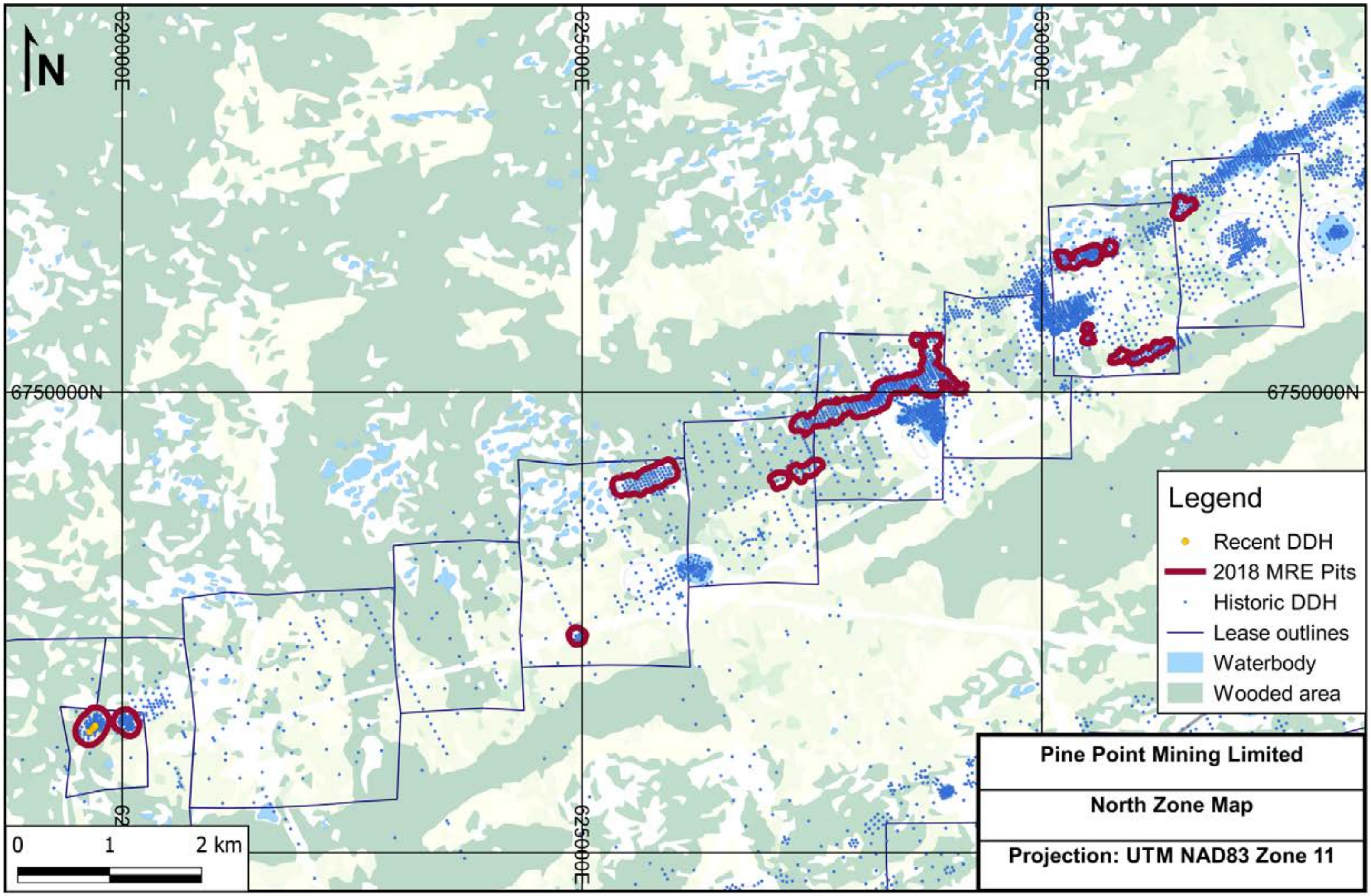


Figure 10-8: North Zone Map showing the location of historical collars (blue) and recent holes from 2017-2018 (orange)

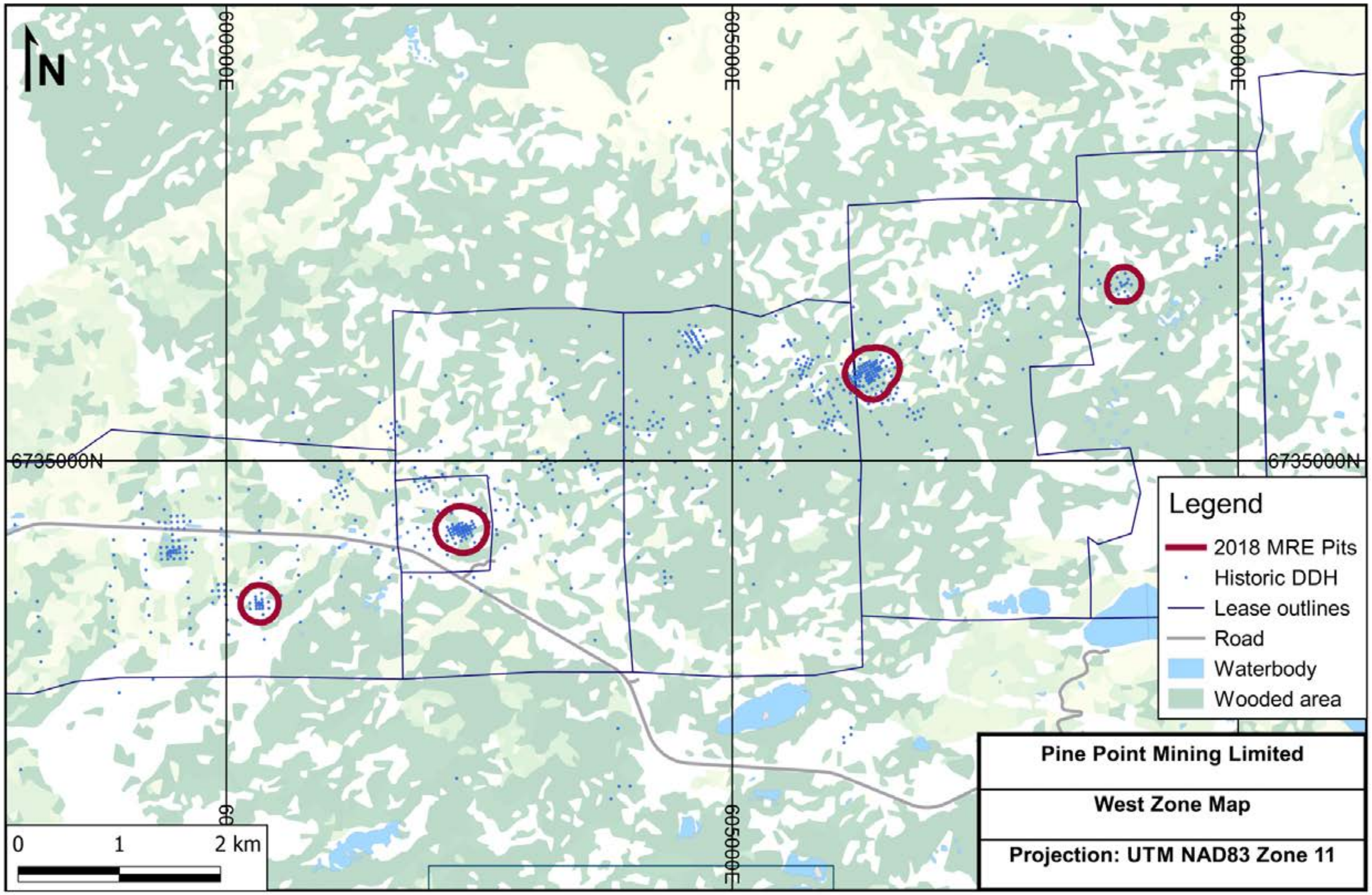


Figure 10-9: West Zone Map showing the location of historical collars (blue) and recent holes from 2017-2018 (none in this case)

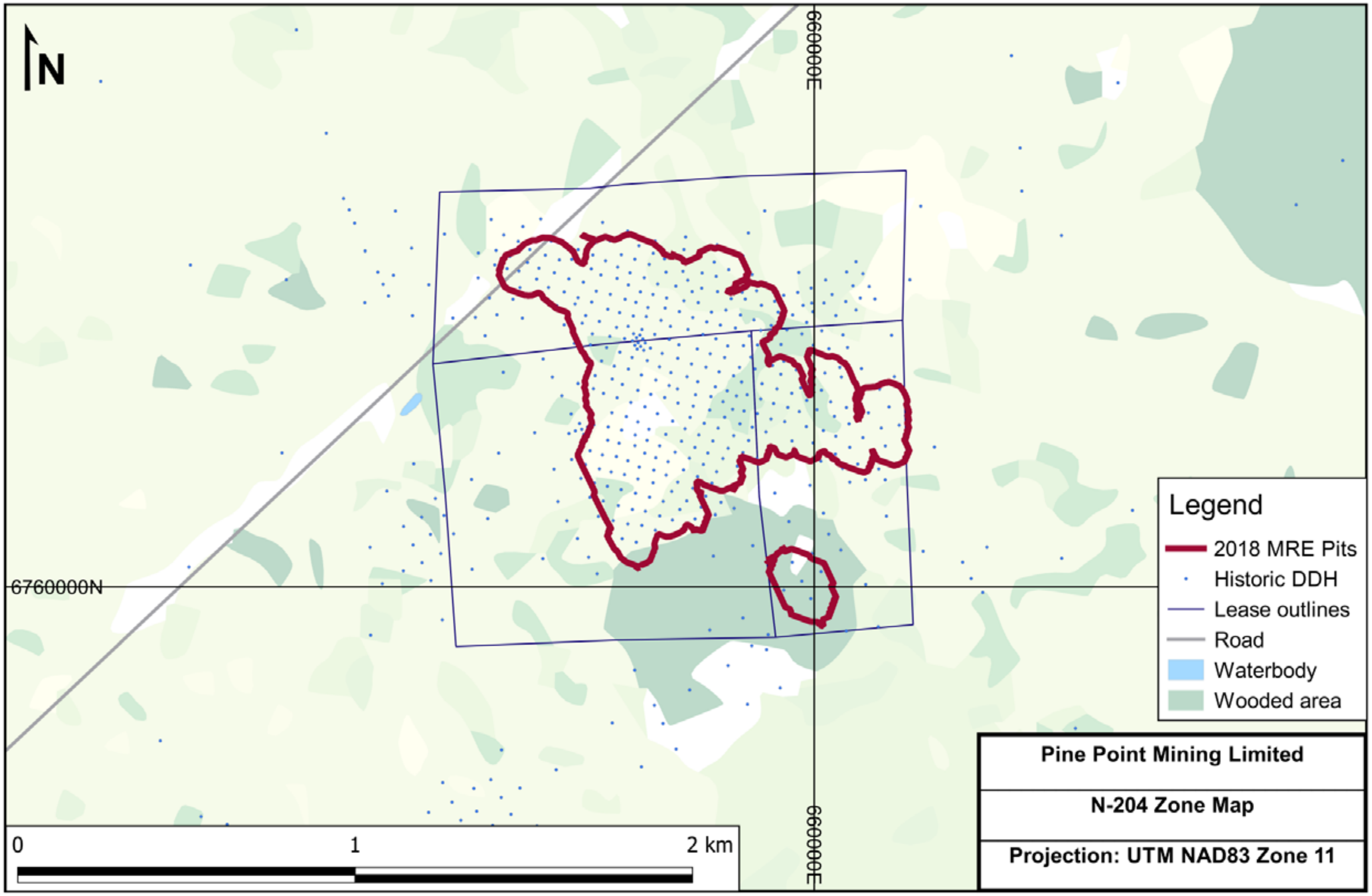


Figure 10-10: N-204 Zone Map showing the location of historical collars (blue) and recent holes from 2017-2018 (none in this case)

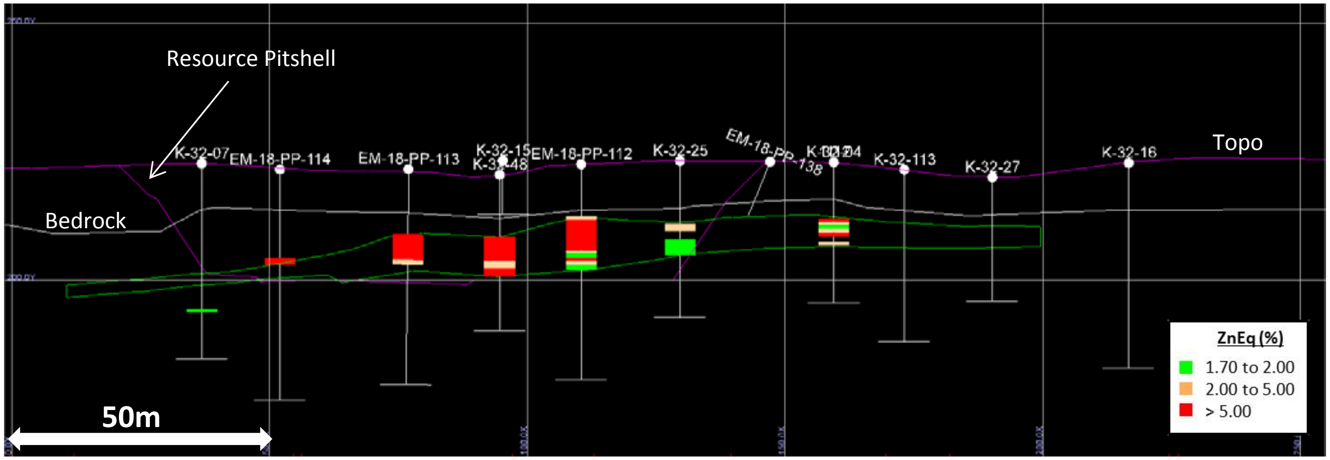


Figure 10-11: Cross-section view in the East Mill Area looking NW

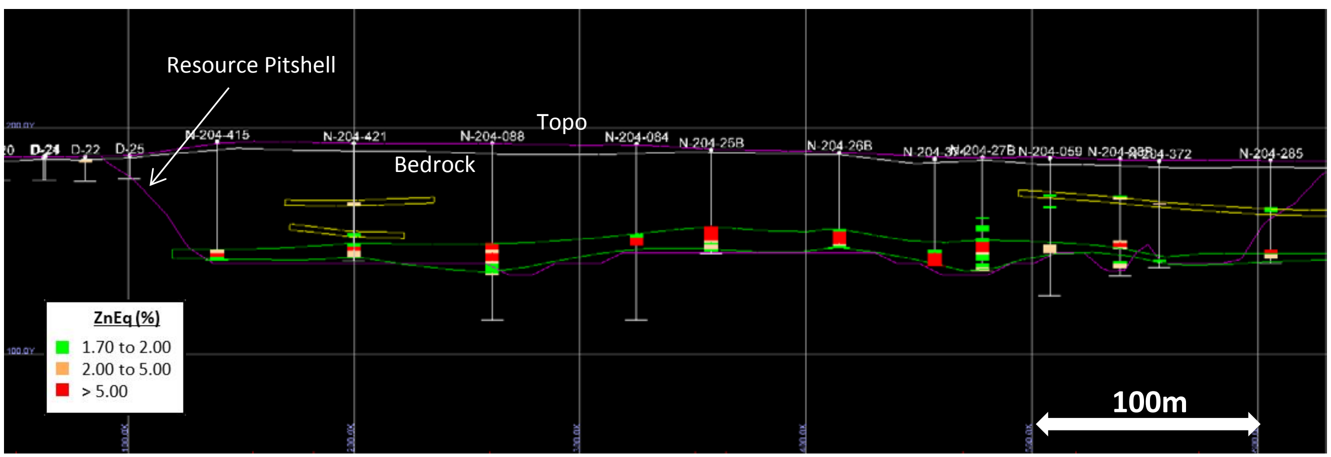


Figure 10-12: Cross-section view in the N-204 Area looking NW.
It should be noted that yellow zones to the left are pinch-outs from significantly larger zones showing geological continuity north of that section.

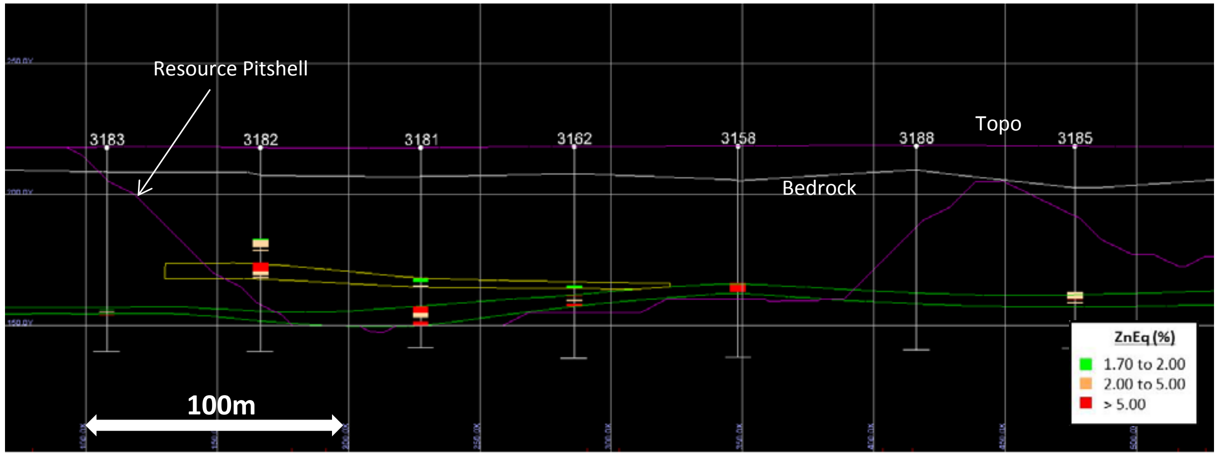


Figure 10-13: Cross-section view in the Central Area looking NW

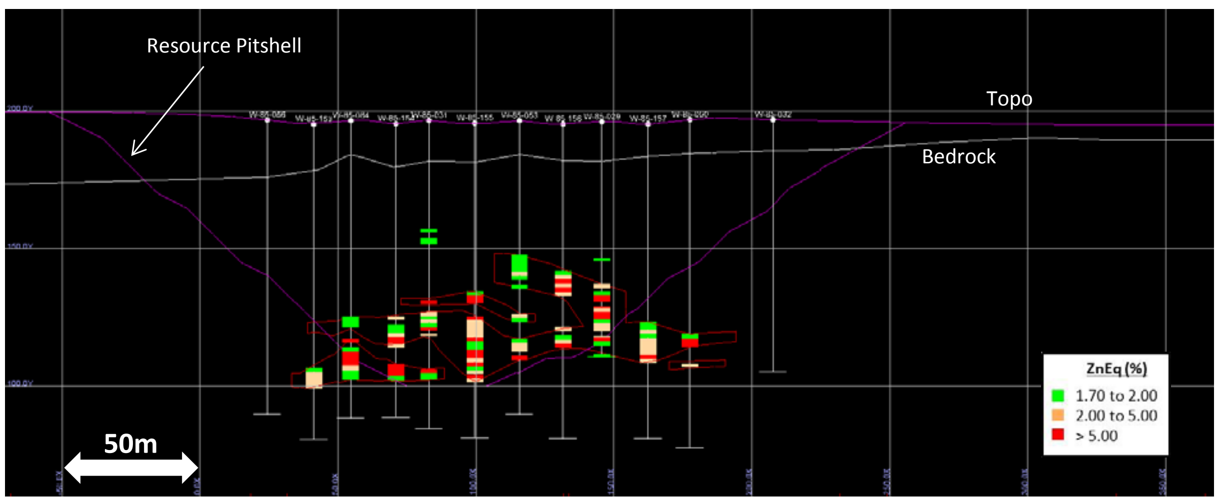


Figure 10-14: Cross-section view in the North Area looking NE (prismatic zone)

Figure 10-15: Cross-section view in the North Area looking NE (tabular zones)

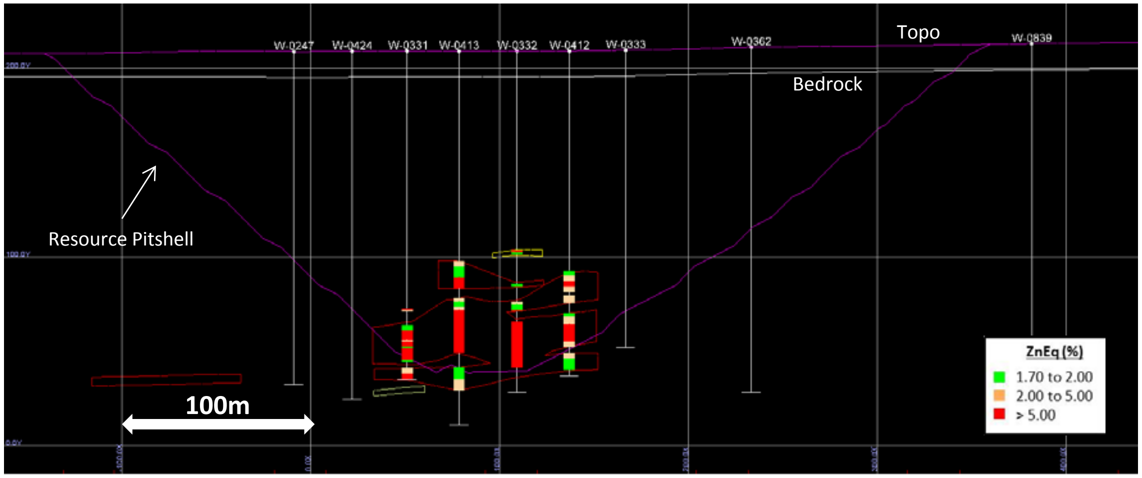


Figure 10-16: Cross-section view in the West Area looking NW (prismatic zone)

11. SAMPLE PREPARATION, ANALYSES, AND SECURITY

11.1 Historic Data

There is no information on the procedures that Cominco Ltd. used on the assay data collected historically. There is a digital assay file containing from and to intervals and assay data for lead, zinc and iron that was found on a computer drive given to Pine Point Mining Limited (“PPML”) in data obtained from Tamerlane Resources. No information on QA/QC procedures or recovery is available. Previous technical reports, including Siegel and Gann 2017, quote Mr. Ross Burns former Pine Point geologist, who indicates Cominco Ltd. used an XRF method for lead, zinc and iron assay.

There are two other data sets available from historic drilling. Westmin Resources drilled 855 holes in the period from 1975 to 1979 and Tamerlane Resources drilled 89 holes in the period from 2005 to 2012.

The Westmin data set includes hard copy drill logs and assay data in electronic form but there are no laboratory certificates or information on QA/QC procedures available.

The Tamerlane Resources data set is more complete with electronic drill logs that include information on lithology, alteration, core recovery, RQD, and density. Some laboratory certificates have been located. Siegal and Gann (2017) summarized QA/QC procedures as follows:

Split core samples from Tamerlane's drilling were shipped commercially to ALS Chemex in North Vancouver, B.C. There the samples were prepared and assayed. Most samples weighed between 2 kg and 5 kg as received. After oven-drying, samples were crushed to 70% <2 mm, then split and pulverized to 85% <75 um. Analyses were performed for Zn, Pb, and Fe by Chemex method AA-62, which involves a four-acid digestion and atomic-absorption analysis. Any Zn, Pb, or Fe values above 30% were re-determined by titration (methods ZN-VOL50, PB-VOL50 and FEVOL %), and reported to the nearest 0.01% Zn, Pb, or Fe.

ALS Chemex assay results were transmitted to CAM as secured .pdf files. ALS Chemex are a major assay lab, with ISO 9001:2000 and ISO 17025 certification of North American facilities. They routinely undertake QA/QC measures that are of world-standard class.

QA/QC procedures were strictly followed by company geologists and specified staff. For the 450 samples taken during the 2010 drilling season, 12% of the selected sample intervals were duplicated creating 55 repeated sample results. Standards and blanks were also inserted into the sample stream. All sampled core was stored in a secure facility, where access was controlled to qualified staff members. Company geologists controlled the samples right up to the delivery to the ALS Chemex sample reception, thus maintaining a strict chain-of-custody order. In the case where samples were shipped by unsupervised commercial carriers to ALS Chemex, the samples were shipped in sealed, tamper-proof containers.

No QA/QC data for the Tamerlane assays have been located.

11.2 Pine Point Mining Limited Data

Pine Point Mining Limited has been exploring the Pine Point property from February 2017 to date of this report. At the time of the data cut-off for this report, PPML completed infill drilling of 23,751 m in 318 drillholes. The results are incorporated in the resource estimate.

11.2.1 Assay Samples

In general, only mineralized intervals are sampled. To create representative and homogenous samples, sampling honours lithological contacts, i.e. no sample crossed a major lithological boundary, alteration boundary or mineralization boundary.

The sample length for the majority of intervals collected varies from 0.50 m to 1.5 m, and the recommended sample length is 1 m. Around 14% of the sample intervals are less than 0.5 m and taken in areas of high-grade mineralization to reflect sharp control boundaries. Approximately 6% of the sample intervals are over 1.5 m and are related to intervals of poor core recovery.

Two shoulder samples, each having a sample length of approximately 1 m, were collected from the non-mineralized core above and below the mineralized intervals.

The core was sawn in half with a diamond saw along its length. One half was put into a plastic sample bag and the other half was retained and kept in the core box for later reference. A sample assay tag was placed in the plastic sample bag and the bag tied off.

11.2.2 Density Samples

Density samples are collected from: a) unaltered, unmineralized lithologies; b) altered-unmineralized lithologies; and c) altered and mineralized units. The density samples have a sample length of 20 cm and depending on where they are taken should consist of full core (unmineralized core) or ¼ core (mineralized core already sampled for assaying). The frequency of samples in: a) should be 1-2 samples; b) 1 sample per 20 m; and c) one sample in intervals greater than 10 m minimum or 1 sample/every 10 m if greater than 10 m.

11.2.3 Lab Methods of Preparation, Processing and Analysis

Core samples were shipped to Bureau Veritas laboratory in Vancouver, BC, for analysis in 2017 and in 2018, samples were shipped to the ALS prep laboratory in Yellowknife and on to Vancouver for analysis. Both Bureau Vertias and ALS Chemex are certified and accredited laboratories.

11.2.3.1 2017 Sample Analysis Procedure – Bureau Veritas

The sample preparation followed the Bureau Veritas PRP70-250 procedure which includes: crushing and splitting until out of a 1 kg sub-sample, 70% or more passes through a 10 mesh (2 mm sieve size). A 250 g sub-sample split is then pulverized until 85% or more passes a 200 mesh (75 µm sieve size).

A 15 gram sub-split from the resulting pulp was then digested in 1:1:1 aqua regia mixture and subjected to ultra-trace Inductively Coupled Plasma Mass Spectrometry ("ICP-MS") analysis (code AQ251). Fifty-three (53) elements were analyzed. The ICP-MSP detection limit for Pb and Zn is 0.01 ppm, the Pb and Zn upper limit is 10,000 ppm (1.00%).

Assays with lead or zinc higher than 1.00% were assayed following method MA404: A 15 g pulp sub-split was subjected to 4 acid digestion followed by an Atomic Absorption Spectrometry (AAS) finish. The AAS detection limit for Pb and Zn is 0.01%, Pb upper limit is 20%, Zn upper limit is 30%.

Assays with lead values higher than 20% or zinc higher than 30% were analyzed by classical titration (method GC816 for zinc, GC817 for lead and GC818 for iron).

A total of 2,418 core samples were submitted to Bureau Veritas for analysis in 2017.

Specific gravity was determined according to method SPG03, i.e. utilizing a waxed core technique. A total of 657 core samples were submitted to Bureau Veritas for density measurement in 2017.

11.2.3.2 2018 Sample Analysis Procedure - ALS

Standard rock package PREP31a was used. Crush entire sample to 70% passing -2 mm, split off 250 g and pulverize split to better than 85% passing 75 microns.

A 0.25 g sample from the pulp was digested in a 4-acid leach and analyzed for 48 elements by ICP-MS under ALS procedure code ME-MS61. The ISP-MS detection limit for elements of interest are listed in Table 11-1.

Table 11-1: Detection limits for elements of interest

Elements	Lower limit	Upper limit
Lead	0.5 ppm	10,000 ppm
Zinc	2 ppm	10,000 ppm
Sulphur	0.01%	10%
Iron	0.01%	50%
Germanium	0.05 ppm	500 ppm
Gallium	0.05 ppm	10,000 ppm

Samples above detection for lead and zinc were submitted for assay by procedure Pb_OG62 for lead and Zn_OG62 for zinc. This has the same procedure as ME-MS61, but used an ICP-AES (atomic emission spectrometry) instead of an ICP-MS. This procedure has upper detection limits of 20% and 30% for Pb and Zn respectively. Samples above the OG62 detection limits were submitted for assay by procedure Pb_VOL70 for lead and Zn_VOL50 for zinc. A 4-acid dissolution was used on a 0.4 g sample and analysis was done by titration. Samples above detection for sulphur were analyzed by LECO furnace technique under code S-IR08.

11.2.4 Sample Shipping and Security

Individual cut samples were placed in poly bags with a unique bar coded assay tag and samples were place in rice bags that were closed with a security tag. Samples were then shipped via Manitoulin Transport from Hay River to the laboratory.

Results were received by email in secure PDF files and QA/QC data was evaluated before the samples were moved into a master database.

11.2.4.1 Chain of Custody

The following procedures are applied to ensure a safe and secure management of materials and data as it pertains to core samples at Pine Point:

- All core samples submitted for preparation and analysis to the laboratory (ALS) are secured in rice bags with numbered security zip ties are delivered directly to the transport company (Manitoulin) in Hay River by a member of the Pine Point Mining team;
- The sample shipment contains a sample submittal form as well as a sample dispatch list detailing the security tag#, rice bag# and samples contained in each rice bag;

- The sample submittal form and sample dispatch list are electronically transmitted to the ALS laboratory in Yellowknife once the shipment has left the Manitoulin Hay River warehouse. The Bill of Lading for the shipment is also emailed to the laboratory at this time;
- Hard copies of all sample submittal forms, sample dispatch lists, bill of lading, etc. are kept in binders in the Hay River warehouse for record keeping. All documentation is scanned and filed on the Pine Point server;
- Samples are sent to:
    - ALS Geochemistry – Yellowknife;
    - #8 – 3 Coronation Drive;
    - Yellowknife, NT X1A 0G5;
    - Canada.

11.3 Quality Assurance and Quality Control (QA/QC)

Canadian National Instrument 43-101 Standards of Disclosure for Mineral Projects, requires mining companies reporting results in Canada to follow CIM Best Practice Guidelines. The guidelines describe which items are required to be in the reports, but do not provide guidance for Quality Assurance and Quality Control ("QA/QC") programs.

QAQC programs have two components. Quality Assurance ("QA") deals with the prevention of problems using established procedures while Quality Control ("QC") aims to detect problems, assess them and take corrective actions. QA/QC programs are implemented, overseen and reported on by a Qualified Person as defined by NI-43-101.

QA programs should be rigorous, applied to all types and stages of data acquisition and include written protocols for: sample location, logging and core handling; sampling procedures; laboratories and analysis; data management and reporting.

QC programs are designed to assess the quality of analytical results for accuracy, precision and bias. This is accomplished through the regular submission of standards, blanks and duplicates with regular batches of samples submitted to the lab, and the submission of batches of samples to a second laboratory for check assays.

The materials conventionally used in mineral exploration QC programs include standards, blanks, duplicates and check assays. Definitions of these materials are presented hereunder:

- Standards are samples of known composition that are inserted into sample batches to independently test the accuracy of an analytical procedure. They are acquired from a known and trusted commercial source. Standards are selected to fit the grade distribution identified in the Pine Point mineralization;

- Blanks consist of material that is predetermined to be free of elements of economic interest to monitor for potential sample contamination during analytical procedures at the laboratory;
- Duplicate samples are submitted to assess both assay precision (repeatability) and to assess the homogeneity of mineralization. Duplicates can be submitted from all stages of sample preparation with the expectation that better precision is demonstrated by duplicates further along in the preparation process;
- Check Assays consist of a selection of original pulps that are submitted to a second analytical laboratory for the same analysis as at the primary laboratory. The purpose is to assess the assay accuracy of the primary laboratory relative to the secondary laboratory.

As per instrument NI 43-101, quality control samples were inserted into the sample batches sent to the laboratory. Inserts included duplicate samples, blank samples and standards.

A total of 152 blank samples, 148 CRM pulps, and 138 core duplicates were sent to Bureau Veritas as part of the 2017 QA/QC program and 100 blank samples, 101 CRM pulps, and 101 core duplicates were sent to ALS Chemex as part of the 2018 QA/QC program.

11.3.1 Duplicates

Duplicate samples are submitted to assess both assay precision (repeatability) and to assess the homogeneity of mineralization.

Several duplicates are used in the mineral industry these being core duplicates (1/2 core or ¼ core), coarse duplicates (rejects and preparation duplicates), pulp duplicates (2nd split of final pulp prior to analysis) and field duplicates (double samples collected in field – where applicable).

Pine Point Mining utilizes core duplicates with ½ of core being used for the primary analysis and ¼ core for the subsequent duplicate analysis, leaving ¼ core in the core box for record keeping. One duplicate sample was inserted for every 20 samples.

Poor recovery with abundant void space and nuggety sulphide material commonly results in large variations between duplicates and their original samples. Samples where a nugget of sphalerite or galena was recovered in one bag instead of the other can lead to a large relative error.

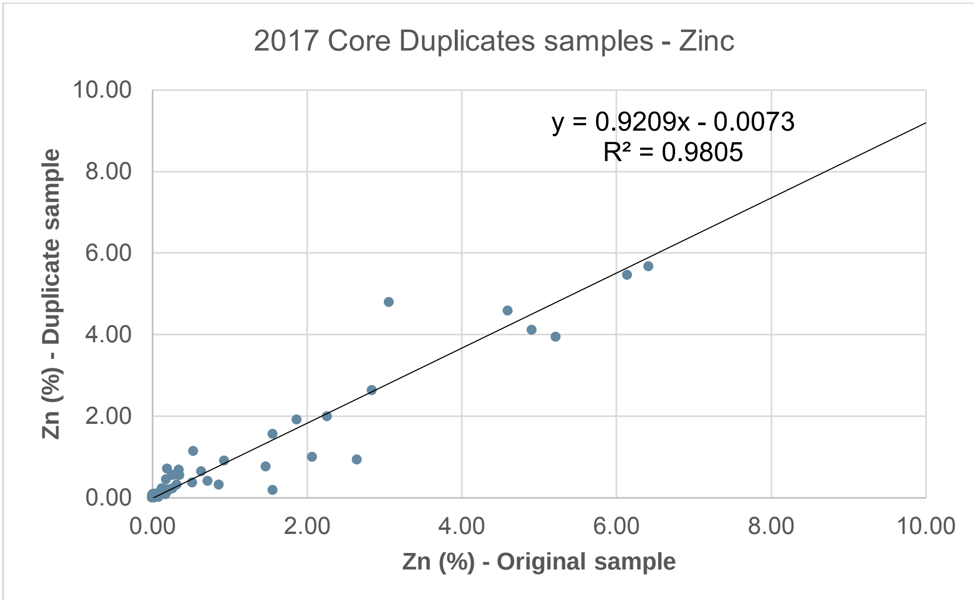


Figure 11-1: Linear graph of Zn duplicates for the 2017 drilling campaign

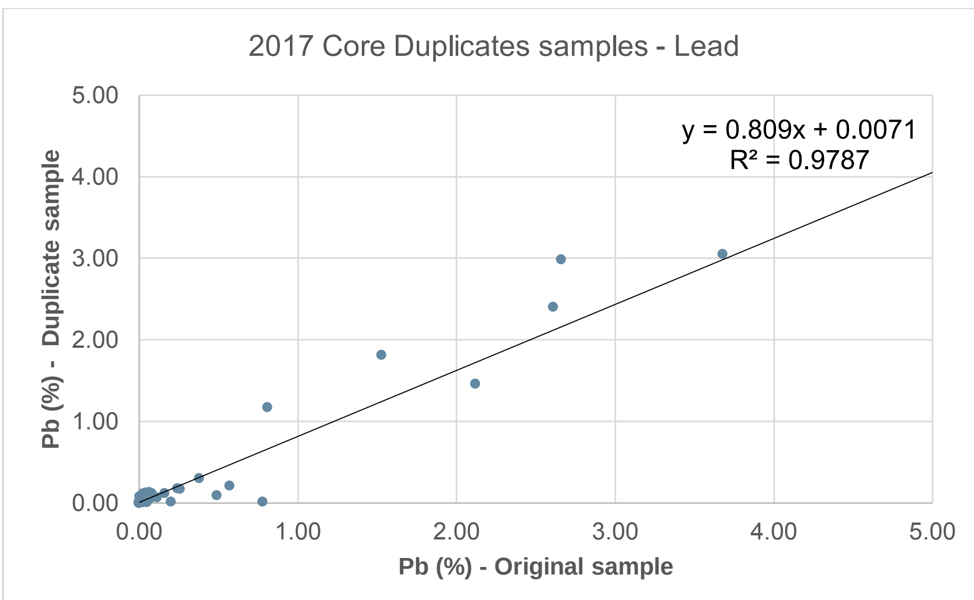


Figure 11-2: Linear graph of Pb duplicates for the 2017 drilling campaign

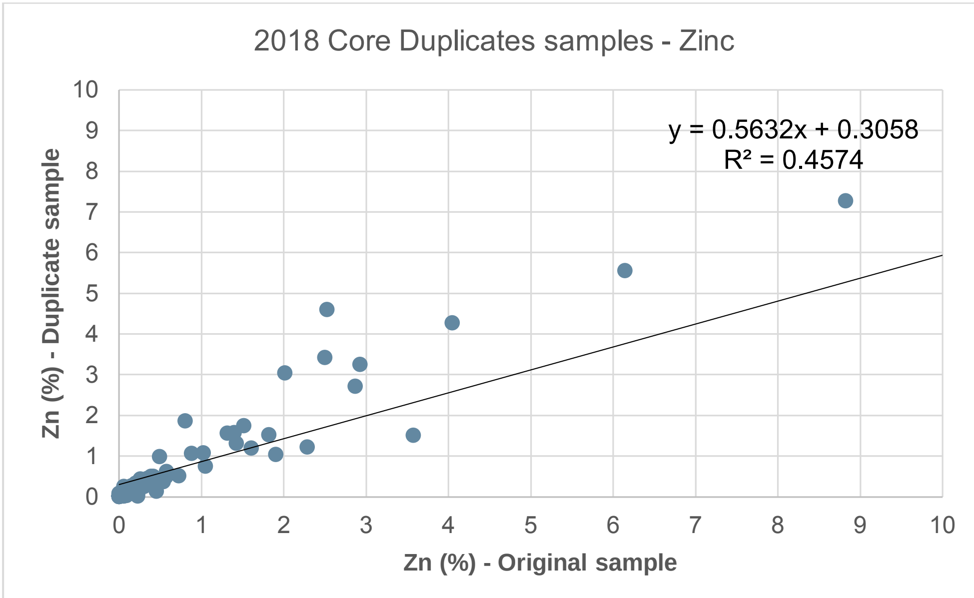


Figure 11-3: Linear graph of Zn duplicates for the 2018 drilling campaign

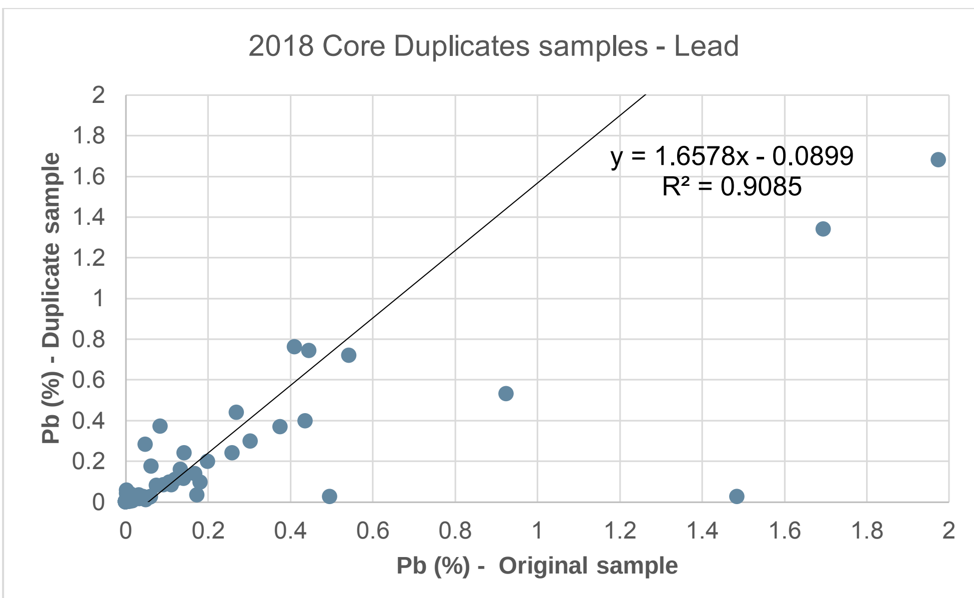


Figure 11-4: Linear graph of Pb duplicates for the 2018 drilling campaign

11.3.2 Blanks

Blanks are used to monitor for potential sample contamination that may take place during sample preparation and/or assaying procedures at the primary laboratory. There are three types of blanks commonly used in a QC programs, these being "Coarse Blanks", "Fine Blanks" and "Pulp Blanks".

Tested blank material, selected due to its depleted base metal geochemical signature, is used by Pine Point Mining. At the beginning of the 2017 diamond drilling program, ten samples of this material were analyzed at an ALS Minerals laboratory to assess their suitability. The initial material selected from local gravel was found to be elevated in lead. Subsequently, commercial bags of white marble were found to be a good blank material and have been used throughout the 2017 and 2018 programs.

One blank sample was inserted for every 20 samples.

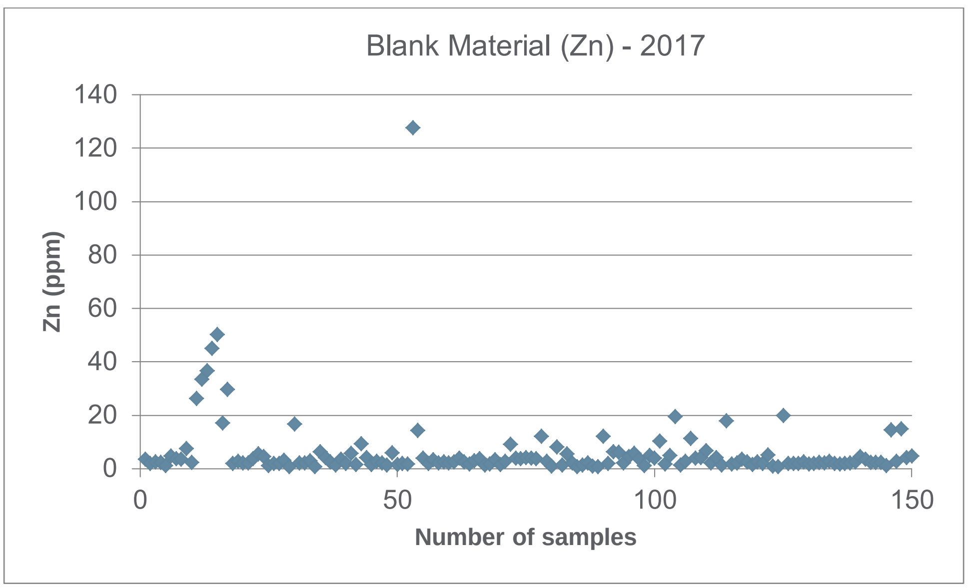


Figure 11-5: Results for the Zn blanks samples used during the 2017 drilling program

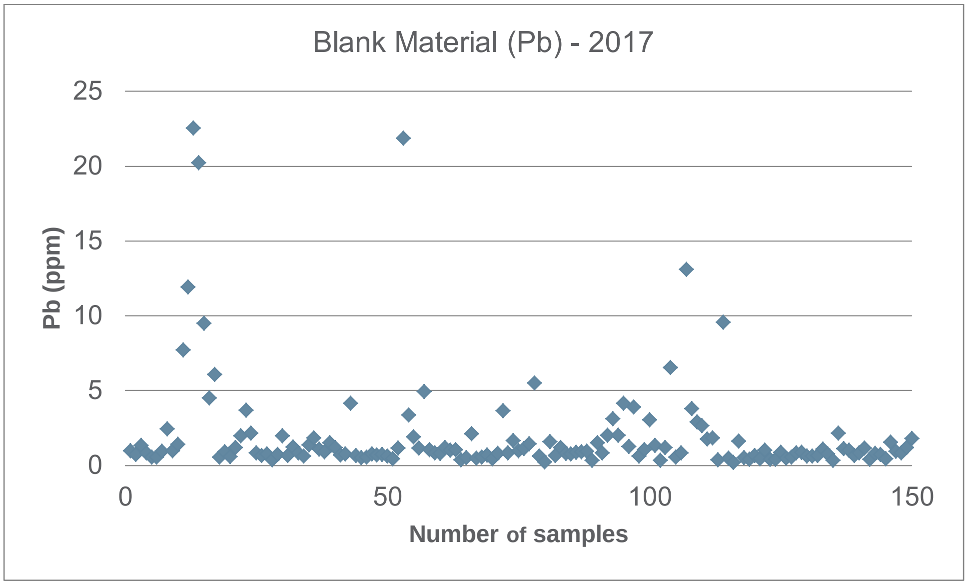


Figure 11-6: Results for the Pb blanks samples used during the 2017 drilling program

Generally, the blank indicates little contamination at the laboratory. There is one failure in zinc and the noise in the early samples is attributed to the discontinued contaminated blank used at the beginning of the program.

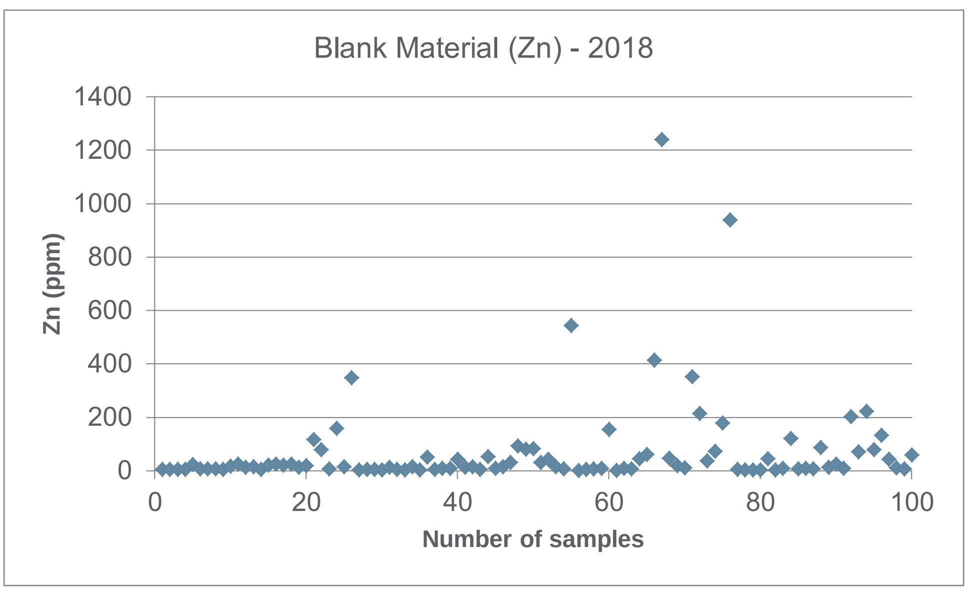


Figure 11-7: Results for the Zn blanks samples used during the 2018 drilling program

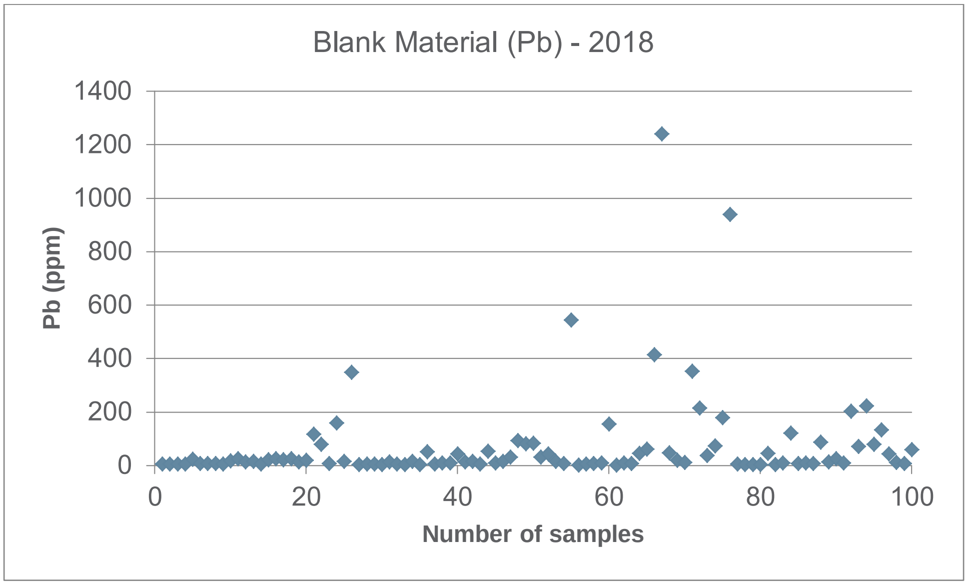


Figure 11-8: Results for the Pb blanks samples used during the 2018 drilling program

The blank samples at ALS Chemex showed more evidence of cross contamination than was seen in 2017 and it is occurring in the crushing stage or the blank is locally contaminated. Three extreme values were returned, one in particular for lead followed a high grade sample. The laboratory was called and procedures for cleaning the crusher circuit were re-emphasized in the preparation section; an improvement was seen after the 77th blank.

11.3.3 Certified Reference Materials (Standards)

A suite of commercially available Standard Reference Materials ("SRMs") are used at Pine Point (Table 11-2).

Table 11-2: Standard reference materials used at Pine Point

Standard#	Au (g/t)	Ag (g/t)	Pb (%)	Zn (%)	Mg (%)	As (ppm)	Cd (ppm)
ME-1301	0.437	26.1	0.188	0.797			
ME-1601	0.613	39.6	0.219	0.942			
OREAS 132b		60.7	3.86	5.25			
OREAS 133a		99.9	4.9	10.87			
OREAS 352		9.43	58.14	2.21	0.097	112	56

Commercially available Certified Reference Materials (CRMs) were used as standards during the 2017 and 2018 diamond drilling programs at Pine Point. The selection of the CRMs was based on anticipated lead and zinc grades ranging from low grade samples (<1% Zn) to average grade mineralized material (3% - 6% Zn) and higher grade samples (>10% Zn). In addition, CRMs were selected based on deposit type; in this case a carbonate-hosted MVT deposit.

One CRM was inserted for every 20 samples.

Table 11-3: Results for the Zn CRM used for the 2017 and 2018 drilling programs

CRM	Quantity inserted	Certified Zn value (%)	SD (%)	Lower limit (-2SD)	Upper limit (+2SD)	Number failed (upper limit)	% passing quality control	Number failed (lower limit)	% passing quality control
ME-1301	75	0.797	0.038	0.759	0.835	1	98.67	26	65.33
ME-1601	69	0.942	0.05	0.892	0.992	1	98.55	35	49.28
OREAS 132b	66	5.25	0.39	4.86	5.64	0	100.00	1	98.48
OREAS 133a	64	10.87	0.708	10.162	11.578	0	100.00	1	98.44
OREAS 352	15	2.21	0.112	2.098	2.322	0	100.00	0	100.00

Table 11-4: Results for the Pb CRM used for the 2017 and 2018 drilling programs

CRM	Quantity inserted	Certified Pb value (%)	SD (%)	Lower limit (-2SD)	Upper limit (+2SD)	Number failed (upper limit)	% passing quality control	Number failed (lower limit)	% passing quality control
ME-1301	75	0.188	0.01	0.178	0.198	7	90.67	1	98.67
ME-1601	69	0.219	0.012	0.207	0.231	3	95.65	2	97.10
OREAS 132b	66	3.86	0.132	3.728	3.992	0	100.00	4	93.94
OREAS 133a	64	4.9	0.324	4.576	5.224	0	100.00	1	98.44
OREAS 352	15	58.14	0.284	57.856	58.424	3	80.00	8	46.67

ME-1601 and ME-1301 results for Zn from the Bureau Veritas lab were under-reported (Figure 11-18 and Figure 11-19), but this issue was resolved by changing labs to ALS. This was likely due to how close these standards certified values are to the 1% ICP-MS upper detection limit.

OREAS-352 was discontinued after 2017 as the lead value was considered too high.

The 2017 CRM results are shown in Figure 11-9 to Figure 11-13.

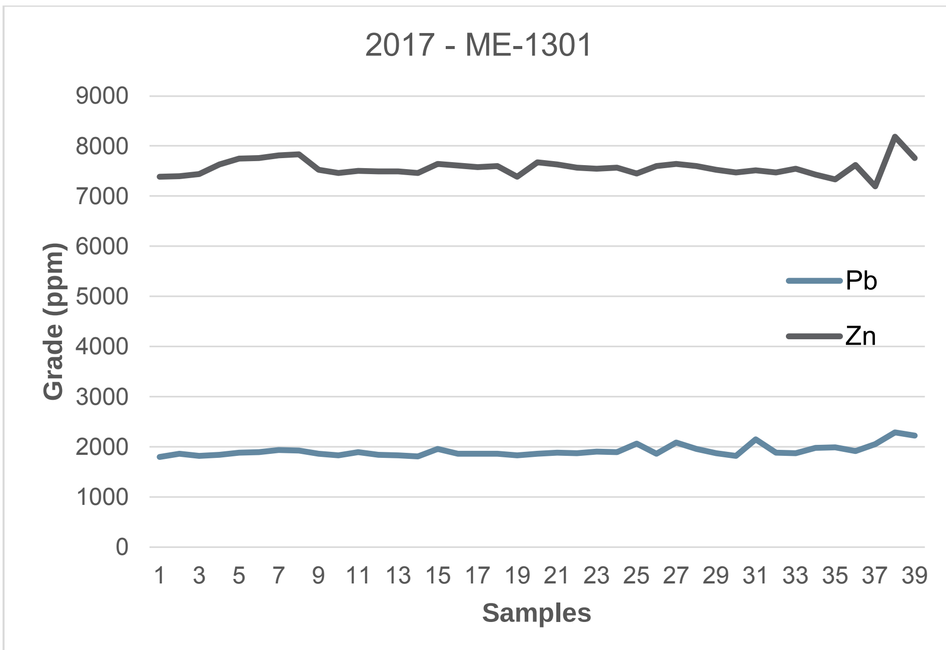


Figure 11-9: 2017 results of standard ME-1301

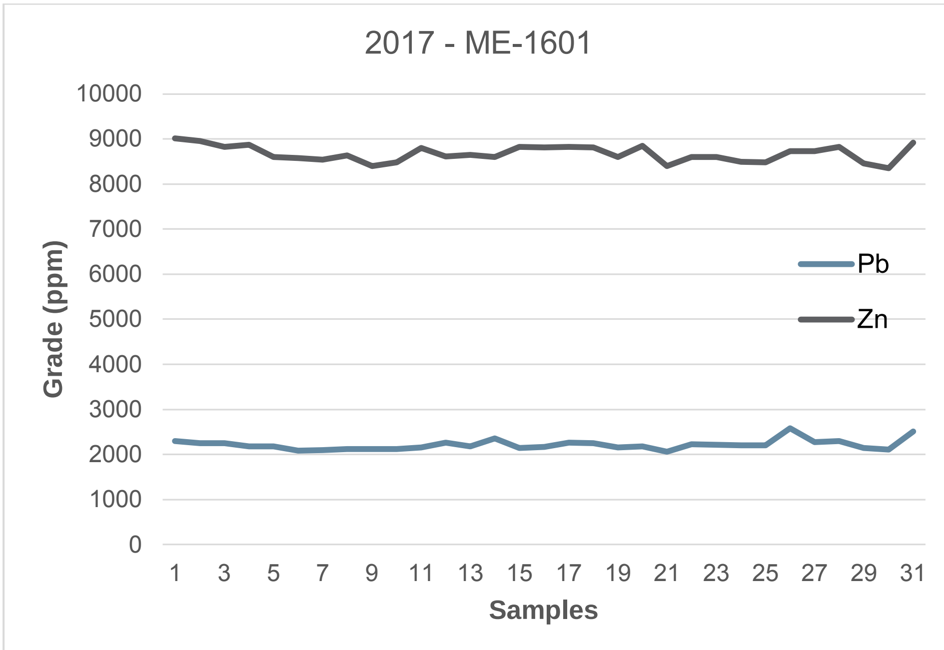


Figure 11-10: 2017 results of standard ME-1601

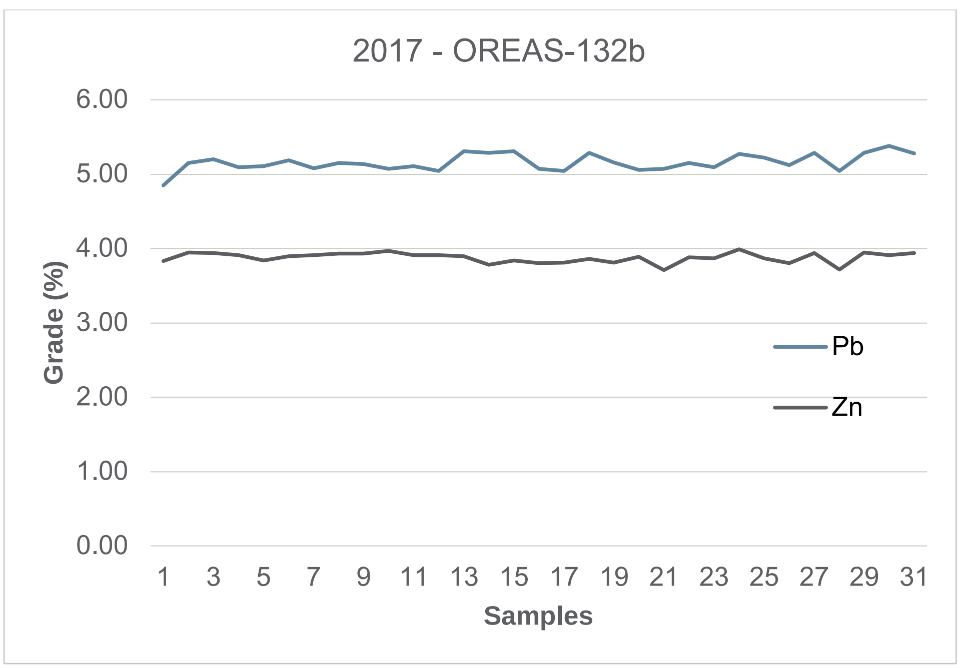


Figure 11-11: 2017 results of standard OREAS-132b

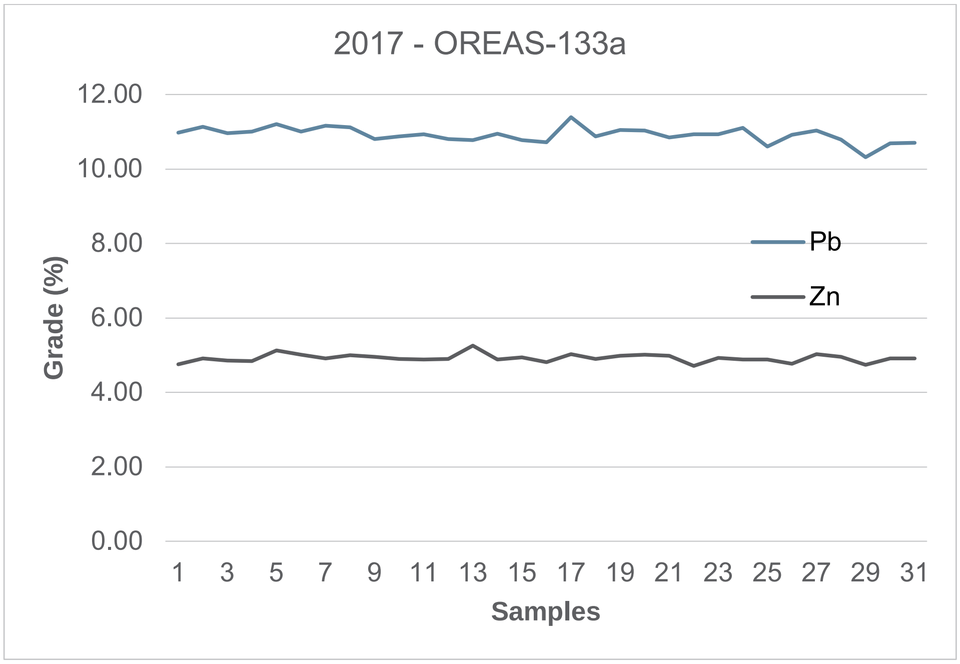


Figure 11-12: 2017 results of standard OREAS-133a

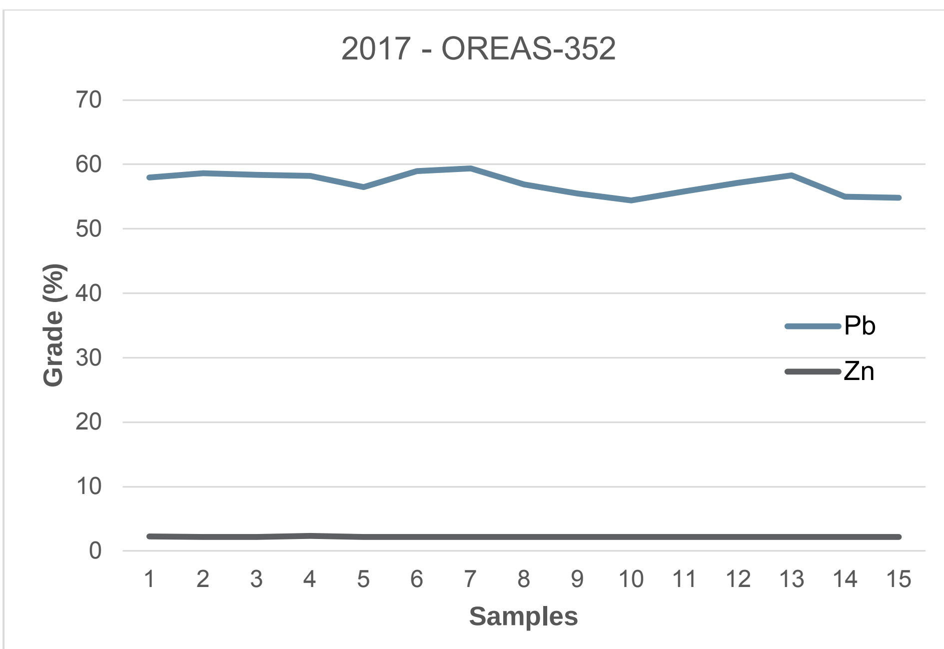


Figure 11-13: 2017 results of standard OREAS-352

The 2018 CRM results are shown in Figure 11-14 to Figure 11-17. No sample batches were rejected.

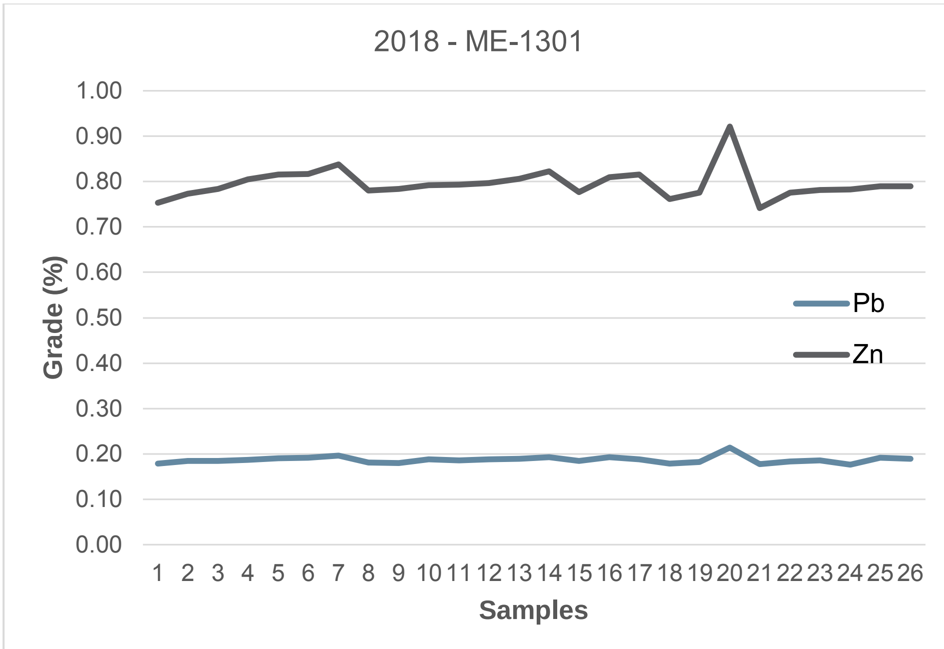


Figure 11-14: 2018 results of standard ME-1301

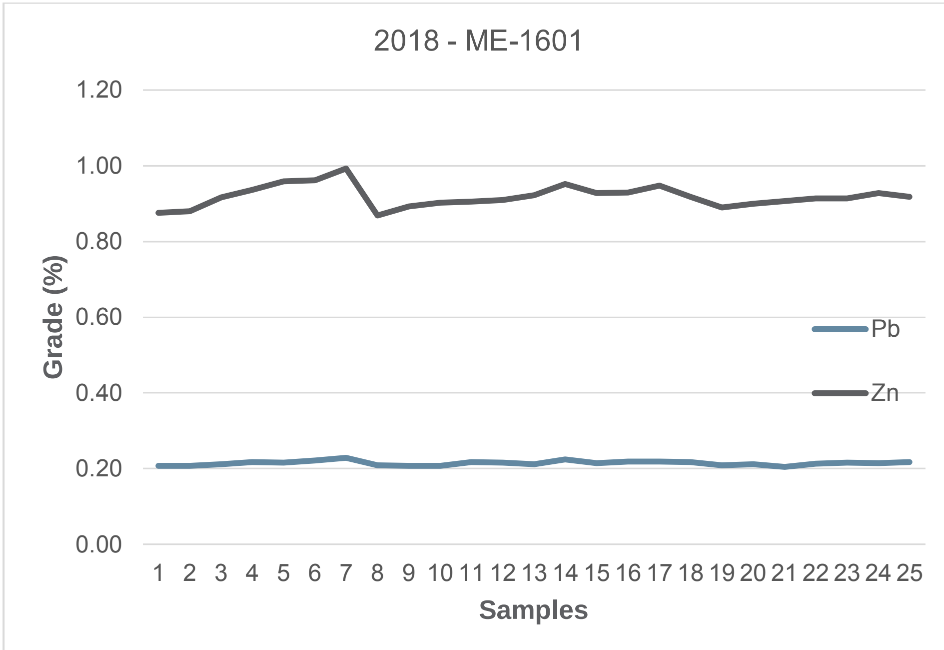


Figure 11-15: 2018 results of standard ME-1601

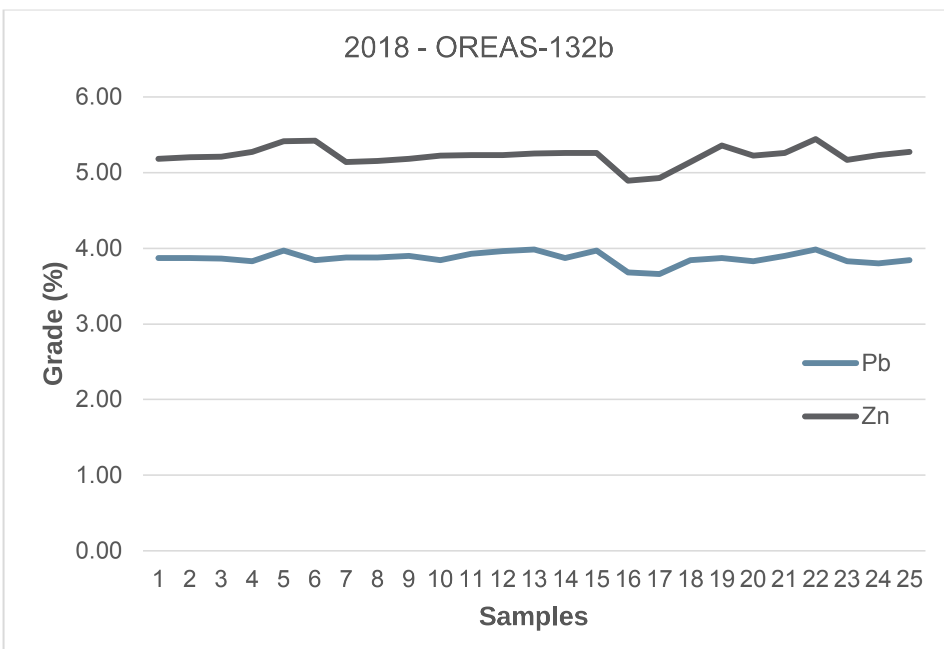


Figure 11-16: 2018 results of standard OREAS-132b

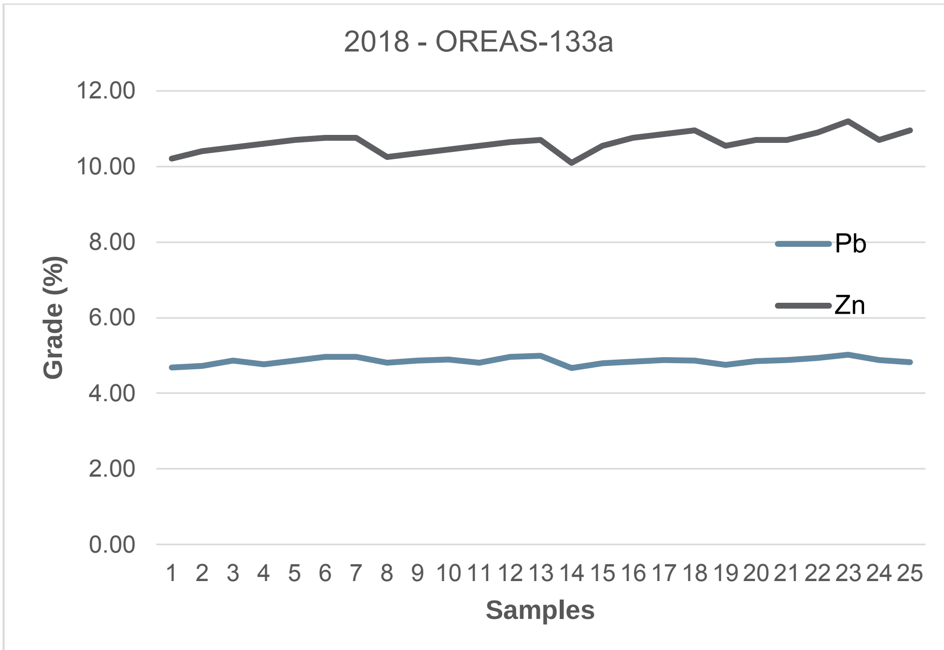


Figure 11-17: 2018 results of standard OREAS-133a

The Zn assays for 2017 and 2018 are plotted in Figure 11-18 and Figure 11-19 for ME-1301 and ME-1601 respectively. These plots show that the underreporting bias for these standards was corrected by changing labs to ALS.

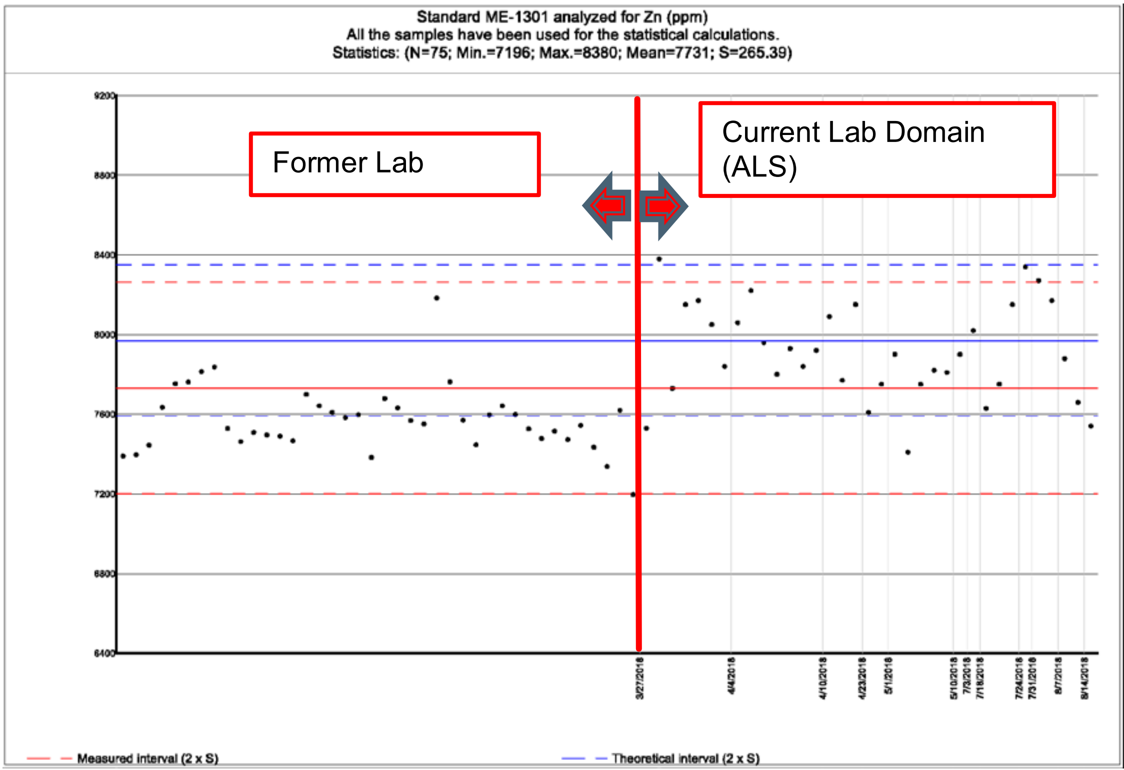


Figure 11-18: 2017 and 2018 Zn results for the ME-1301 standard

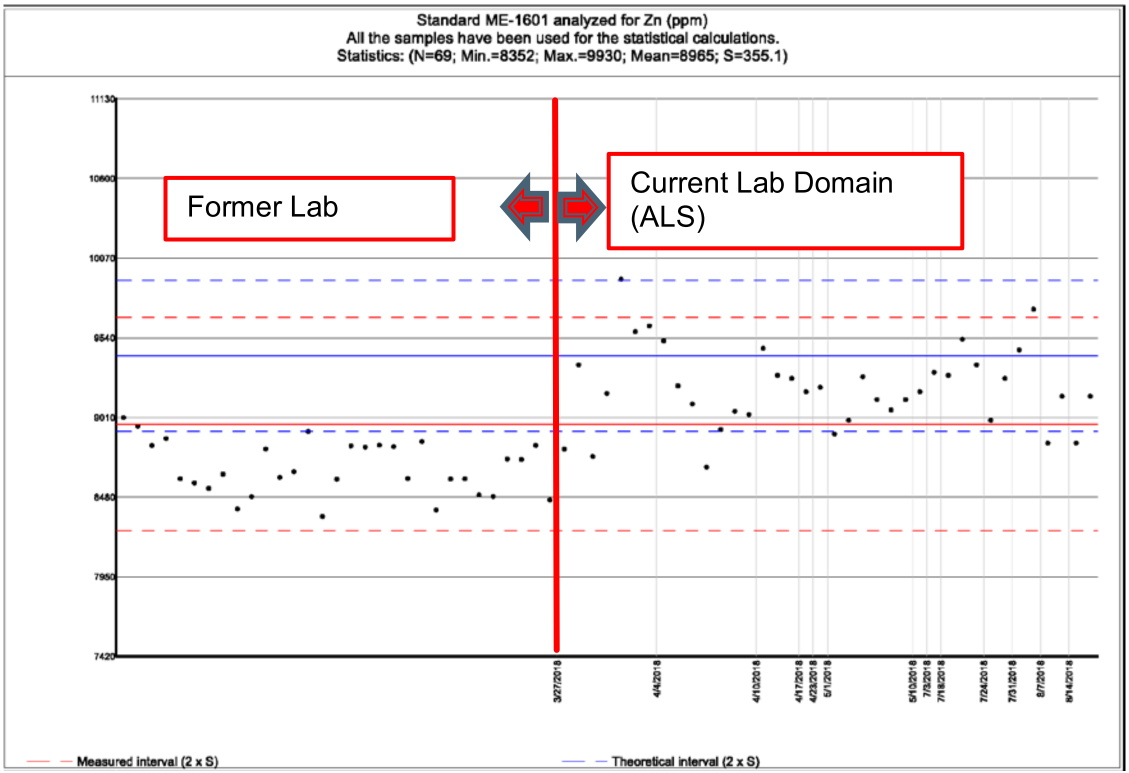


Figure 11-19: 2017 and 2018 Zn results for the ME-1601 standard

11.4 Assessment of Results

The following procedure assess the QA/QC program during the drill program to identify immediate issues. If they do in fact exist, they are immediately addressed:

- The Chief Geologist assesses the analytical results as they are received and communicate results to the Geological Team;
- The results will be assessed on a hole by hole basis first, then followed by incorporating results received for the program to date to evaluate results over time;
- Follow up action will be depended on the kind of failure observed or what type of other issues are observed. Some of the follow-up action may include:
  - Re-runs of the assays associated with the failed quality control material;
  - Changes that need to be made to the quality control materials (e.g. new blank if determined to be erratically mineralized);
  - Discussions with the lab over their procedures (e.g. to take more care cleaning equipment if contamination is detected);
  - Discussions with site sampling staff (e.g. if regular sample switches are occurring or standards are misidentified, numbering of samples).
- Quality control data is stored in a central Pine Point database at the field office in Hay River;
- A simple spreadsheet incorporating all analytical results and associated check assay results and evaluations is produced showing a summary sheet as well as sheets showing evaluations for each standard, blank, duplicate and check assay;
- The database/excel spreadsheet will include drillhole ID, sample number, certificate number for each line of quality control data as a link to the original data set and results obtained.

11.4.1 Conclusion

The QP reviewed the sample preparation, analytical and security procedures, as well as insertion rates and the performance of blanks, standards and duplicates for the 2017 and 2018 drilling programs, and concluded that the observed failure rates are within expected ranges and that no significant assay biases are present.

12. DATA VERIFICATION

The Mineral Resource estimate in this report is based on drill data from several eras of drilling at the Pine Point Mine Site that include the Cominco Ltd. era holes 1930 to 1986, Westmin resources era holes 1975 to 1984, and the current programs of Pine Point Mining Limited in 2017 and 2018.

For the purpose of this MRE, BBA performed a basic validation on the entire database. All data were provided by Osisko Metals in UTM NAD 83 Zone 11. The 2018 drilling program is still ongoing at the time of writing this report. The database close-out date for the resource estimate is September 12, 2018; data from 318 drillholes (23,751 m) incorporated in the resource estimate.

The Pine Point database contains 18,542 surface diamond drillholes. Of these 18,542 drillholes, a subset of 6,880 holes cut across the mineralized zones with a total of 31,120.

12.1 Site Visit

Pierre-Luc Richard of BBA visited the Pine Point Project from August 9 to August 12, 2018. The site visit included a visual inspection of historical core and core drilling in progress, a field tour (Figure 12-1) and discussions of the current geological interpretations with geologists and engineers of Osisko Metals.

Figure 12-1: Historical pits visited during the site visit

Selected drill collars in the field were also validated. The site visit also included a review of sampling and assays procedures, the QA/QC program, downhole survey methodologies, and the descriptions of lithologies, alteration and structures (Figure 12-2).

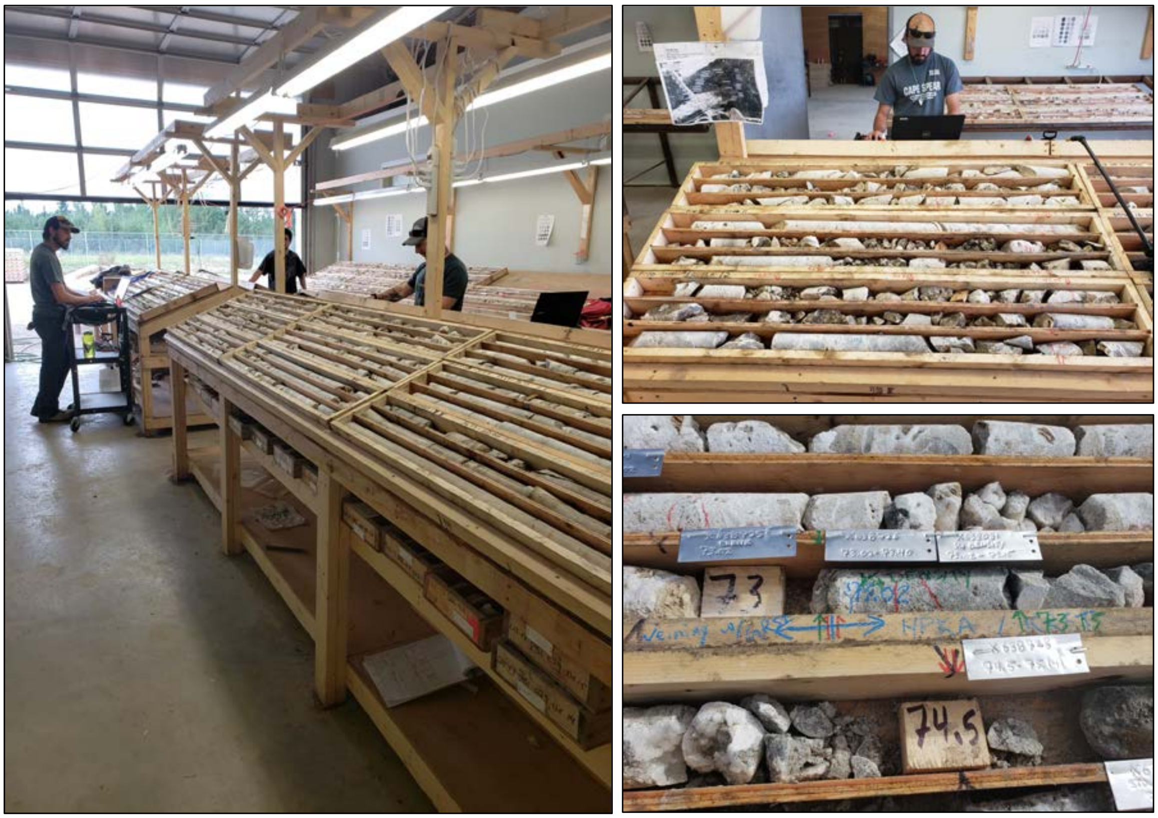


Figure 12-2: Core review in the core logging facility

12.2 Drilling and Sampling Procedure

Osisko Metals procedures are described in Chapters 10 and 11 of the current report. Discussions held with on-site geologists allowed to confirm said procedures were adequately applied.

BBA reviewed several sections of mineralized core while visiting the project. All core boxes were labelled and properly stored either inside or outside. Sample tags were present in the boxes and it was possible to validate sample numbers and confirm the presence of mineralization in witness half-core samples from the mineralized zones (Figure 12-2).

Drilling was underway during BBA's site visit (Figure 12-3 and Figure 12-4), which provided an opportunity for Osisko Metal personnel to explain the entire path of the drill core, from the drill rig to the logging and sampling facility and finally to the laboratory.

Figure 12-3: Drilling underway during the site visit and drill collar review

Figure 12-4: A) Core logging and sampling facility; B) sample preparation room; C) samples ready for shipment to the laboratory

12.3 Recent Drillhole Database

12.3.1 Assays

BBA was granted access to the original assay certificates for all holes drilled from Osisko Metals (2017-2018). Assays of Zn and Pb were verified for all holes. The assays recorded in the database were compared to the original certificates from the different laboratories and no significant discrepancies were detected.

12.3.2 Drillhole Location

For drilling conducted between 2017 and 2018 all drill collars have been surveyed using differential GPS equipment. Random field checks with hand held GPS units and comparison to plotted location on air photos confirmed holes are in their planned locations.

12.3.3 Down-hole Survey

For the 2017 and 2018 drilling program downhole orientation measurements are taken below the casing at the top of the hole, every 30 m within the hole and commonly at the bottom of the hole using a Reflex downhole survey instrument. Spurious measurements are removed from the database.

12.3.4 QA/QC

QA/QC reports were reviewed and did not yield issues. Some Out-of-range lengths in some assays were detected.

12.4 Historical Drillhole Database

The historical information used in this report was taken mainly from reports produced before the implementation of NI 43-101. In most cases, little or no information is available about sample preparation, analytical or security procedures. However, BBA assumes that exploration activities conducted by previous companies were in accordance with prevailing industry standards at the time. Basic cross-check routines between original logs and drillhole database typically done during a Data Verification process could not be performed due to the absence of historical hard copies. Validation to historical databases was however performed to validate the database used for the Mineral Resource Estimate.

In the absence of hard copies supporting historical databases, a considerable amount of energy was dedicated to validating historical drillholes as described below.

12.4.1 Drillhole Location

12.4.1.1 Summary of 2018 Pine Point Survey of Historical Drillholes

Sub-Arctic Geomatics out of Yellowknife was contracted to locate and determine the absolute positions of approximately 4,000 drillholes among the roughly 18,000 historic drillholes in the Pine Point mining area. The results of this survey will allow for better positioning of the latter through conversion of historical Cominco Ltd. survey coordinates to Nad83 (CSRS), Zone 11.

Original historical surveying by Cominco Ltd. of the drillholes was carried out over a number of years at the time of various drill programs across the mine site with the technology, and according to the practices, of the day. It was presumed that the drillholes were referenced to one or more localized "mine grids" allowing the drillholes to be positioned relative to each other and their respective origins, in a local coordinate system(s), but not in absolute terms to a geographic coordinate system.

Previous efforts have attempted to assign Universal Transverse Mercator ("UTM") coordinates to the historic drillholes through various combinations of best-fit transformations throughout the mining property. The process used to establish the transformation was largely unknown and therefore the quality of the resultant UTM coordinate locations is questionable.

In 2018, Sub-Arctic Geomatics established long-term, semi-permanent control points in the area and referenced it to the Nad83 (CSRS) and CGVD2013 horizontal and vertical reference frameworks respectively. All drillholes were referenced to the UTM map projection, Zone 11.

Field crews, using the previously-generated approximate best-fit UTM location information, navigated to the area and searched for evidence of a drillhole. A quality code was established and recorded to indicate the level of confidence in both the position and identification of the drillhole.

- In 29% of the cases, drillholes had posts with readable tags correctly identifying the drillhole;
- For 16% of the drillholes, a collar was located in the vicinity of the approximate positions but the post and/or tag identifying the drillhole was either lying nearby on the ground, missing, or illegible;
- In 43% of the cases, no drillhole was located at the approximate location although there was evidence of drill activity nearby.

When the drillhole was actually located it was possible to make a comparison between the observed UTM coordinate and the original best-fit UTM provided coordinate. Generally, within individual deposit areas, the differences between the observed and the provided coordinates were consistent and an average offset value was determined for each deposit area. The average offset value was applied to all drillholes that were not located in the field to improve the absolute positioning of these holes in the reference frameworks established by Sub-Arctic Geomatics.

12.4.1.2 Westmin Resources

Westmin Resources explored the Pine Point mineralized trend west of the Buffalo River from 1975 to 1984 drilling 885 holes. The core is no longer available and only hard copy logs, UTM collar coordinates and an assay database are currently available. Collar locations were verified by the Sub-Arctic field work in 2018. Future verification will use twin holes and confirmation drilling.

12.4.1.3 Confidence Levels and Drillhole Accuracy

- For 29% of the drillholes (1,384) located and identified in the field, accuracies are within 2-4 cm in both horizontal and vertical position;
- Positional accuracies for 16% of the drillholes (749) located in the field but not identified are estimated to be better than 1 m;
- Positional accuracies for 43% of the drillholes (2,084) not located are somewhat variable and dependent on deposit area. They are estimated to be 5 m horizontal and between -6 m to +8 m vertically;
- In 10% of the cases (476 drillholes), the ground has been disturbed (i.e. waste piles, roadways, clearings, etc.) so drillhole positioning could not be determined. In these cases, the elevation data will be accurate for that location;
- Location information of all new drillholes (145) surveyed will be accurate within 2-4 cm in both horizontal and vertical positions. This accounts for 3% of the holes surveyed.

12.4.1.4 Future Work

For those holes not found in the survey areas during the 2018 field season, and the remaining holes not yet surveyed (outside of the survey areas), it is assumed that their positional accuracies can be improved from their current 5 m horizontal and -6 m to +8 m vertical accuracy envelopes. How much improvement will depend on the amount and quality of information found on past survey techniques, the transformation processes and physical collar locations identified in the field.

Two areas of work are proposed:

1. If the information on past techniques is shared and deemed worthwhile, a new transformation can be undertaken to improve the existing approximate locations by adjusting the relative positions from the local coordinate system (mine grid) to absolute positions in the established geographic coordinate system.
2. Continue the 2018 field survey by expanding the survey to include a number of widely spaced lines transecting the mining property tying in all holes encountered to provide for a better, more complete network of points to be used for an even more refined transformation adjustment.

12.4.2 Logging and Re-sampling of Historical Cominco Ltd. Core

The Cominco Ltd. era core yard on the western flank of the historic mine office and maintenance complex was mapped and partly catalogued in the late summer of 2018. The core yard is a storage facility for nearly 1 million metres of core. The core is stored on 1,300 wooden pallets containing approximately 100 core boxes each. The pallets are in chronological order by row, however, some blocks are out of sequence. After visiting the site, BBA resource geologist, Pierre-Luc Richard and vice president of exploration at Osisko Metals Inc., Robin Adair, proposed to take inventory of the available core and pilot a re-sampling procedure of the historical Cominco Ltd. drillholes. The objective of the following exercise was to re-assay previously drilled and logged holes within current resource definition drilling targets. The results of these assays would be compared to historic Cominco Ltd.'s assays available and verified against reference material. A selection of unsampled historical holes with likelihood to host mineralization were also identified. This preliminary exercise sought to quantitatively define the feasibility of further re-sampling work in the core yard considering: the physical state of the core in storage, accessibility to core, volume of preserved material, contamination of samples, and preservation of relevant labels/tags.

12.4.2.1 Core Yard Surveying and Inventory

The entire core yard was surveyed with photogrammetry tied in to Differential GPS control points (Figure 12-5). This created an Orthomosaic photo map that was used to take inventory of core pallets. The second stage was to tag each pallet with a new identity generated from an air photo naming system that grouped rows to an alpha-numeric code. Tags were made of a weather resistant label paper and proved durable in a brief bench test. The pallets were photographed after being tagged using an SLR camera. The ultra-high-resolution camera was used to photograph the face of a core pallet in order to resolve the small metal tags with hole ID and box info. Most pallets fit entirely in one frame, however, some wider pallets may omit a column of tagged boxes. These photos are intended to be used to record which holes are on which pallet and with the reference map locate them on-site. Although these photos were taken, time and staff restriction did not yet allow recording each hole on the pallets photographed at the time this Report is being completed.

Figure 12-5: An orthomosaic air photo map of the Pine Point core yard (2 F250 trucks for scale centre-left)

12.4.2.2 Physical Preservation of the Drill Core

Through the years of vandalism and natural decay many pallets are partially disturbed. In the case of vandalism, most damage is isolated to the top 1 or 2 rows of any individual pallet, with the pallets closest to the access road the most vandalized. Some pallets have collapsed and tilted over (10%). Although some are tilted enough to spill boxes most are still recoverable. Some pallets were also ravaged by fungus and moisture. The extent of rot is rather difficult to estimate as the box face often appears to be good enough while the side boards are entirely rotten. Approximately 30% of the boxes are rotten to a point where extreme care is needed to extract boxes. This degradation is also highly variable and can affect one column with the adjacent column unaffected. In general, most of the boxes are recoverable, in some cases wax pen and markers are still legible. However, due to the storage method of stacking holes in columns, many drillholes will be missing a box or two.

The preservation of the original box tags varies; however, most boxes contain legible markings on the frame and can be read easily. See Figure 12-6 for an example of well preserved and poorly preserved core boxes.

Figure 12-6: A pallet of NQ core at the Pine Point core yard

12.4.2.3 Re-logging and Sampling

Once a list of required drillholes had been submitted by BBA, sampling work began on-site. A 2-3-men crew travelled to site regularly over the next week to search for the required holes and process them. Using the Geotic software on a remote laptop server, each recovered hole was logged and prepared for sampling. Sampling procedure was set by the drill runs in most cases. Samples were taken at each half run. A 5 ft run would be sampled as 2 x 2.5 ft samples. In holes where the sampling markers were still visible, samples were taken along Cominco Ltd.'s sampling intervals. Samples were taken over any intervals that were previously split, where-by half was left in the box. If the hole was unsampled by Cominco, the hole was searched for disseminated mineralization. When disseminated Sphalerite or Galena was encountered, it was sampled at half drill run intervals. Once samples were marked and tagged, photographs of the core were taken wet and dry with a 16 Mp cellphone camera. The samples were then bagged and packed in the truck. No duplicates were made as it was not possible to split and mix the samples in the field effectively. There were cases where some holes were entirely sampled leaving nothing but empty boxes with drilling tags at spaced intervals.

Table 12-1 presents the results from the preliminary re-sampling program conducted as part of this study.

Table 12-1: Preliminary re-sampling program results

Hole ID	From (m)	To (m)	Length (m)	Historical sampling		Re-sampling	
				Pb (%)	Zn (%)	Pb (%)	Zn (%)
L27-129	9.14	17.98	8.84	0.09	0.03	0.18	0.04
W85-196	28.96	38.10	9.14	0.04	0.05	0.00	0.02
W85-196	55.78	81.53	25.76	1.91	5.80	1.83	4.92
X51-422	21.34	30.48	9.14	0.92	1.57	1.04	1.55
L36-545	34.44	41.15	6.71	Unsampled	Unsampled	0.11	0.18
M63-02	65.53	74.68	9.14	Unsampled	Unsampled	0.03	3.12
M63-101	65.53	74.68	9.14	Unsampled	Unsampled	0.00	0.08
N81-149	78.33	81.38	3.05	Unsampled	Unsampled	0.01	0.01
Y61-61	47.24	49.68	2.44	Unsampled	Unsampled	0.01	0.19
4408	18.90	44.20	25.30	Unsampled	Unsampled	0.46	1.62

12.4.2.4 Conclusion on the Re-sampling Program

The results of this preliminary program confirmed that historical core can be re-sampled and compared with modern analytical results. Four holes with grade in the historical database were confirmed. An additional six unsampled historical holes proved to contain significant enough grades to propose a larger scale assaying program of all unsampled historical intercepts that are identified within the current mineralized model. Currently, all historical unsampled intervals within the model are attributed a grade of 0% Pb and 0% Zn.

Re-sampling drillholes in the Cominco Ltd. core yard has proven to be possible with minimal contamination. A larger scale program would be possible to recover core. Although there must be a mutual understanding that some holes are partially disturbed or completely sampled. If an inventory were to be completed on the digital photos taken of the pallets, it would be possible to determine which drillholes are available to be examined. This resource is of utmost importance in North Trend resources or other deposits where capital can be saved by validating Cominco Ltd. holes.

12.5 Conclusion

BBA is of the opinion that the drilling protocols in place are adequate. The database for the Pine Point Project is of good overall quality. Minor variations have been noted during the validation process but have no material impact on the 2018 MRE. In the QP's opinion, the Pine Point database is appropriate to be used for the estimation of Mineral Resources.

13. MINERAL PROCESSING AND METALLURGICAL TESTING

13.1 Introduction

This chapter summarizes and provides documentation for the metallurgical and process design work that has been performed on the Pine Point project through to December 2018. This includes a review of the operating history, a review of historical metallurgical testwork used to support various studies, an analysis of the current testwork program results as well as recommendations for future testing.

13.2 Geology

Pine Point's lead-zinc deposits are classified as Mississippi Valley-Type. Economic minerals consist of sphalerite and galena, while gangue minerals consist of marcasite, pyrite, grey and white vein dolomite and calcite (Fish, 1981).

13.3 Historic Operating Data

13.3.1 Historical Operations

Mine production commenced in early 1965 with high grade material (50% combined lead-zinc) being shipped directly to Cominco Ltd.'s Trail smelter in British Columbia. The Pine Point concentrator came on stream in November 1965 operating at a design capacity of 5,000 tpd processing material that graded 2.4% lead and 6.0% zinc. The acquisition of the Pyramid Mining Company's claims by Cominco Ltd. in 1966, enabled integration of a sizeable deposit known as X-15 located to the east of Pine Point's land package, this acquisition necessitated an expansion of an additional 3,000 tpd to the concentrator, which was commissioned in December, 1968. This expansion (the Sphinx circuit) was constructed with its grinding and primary flotation circuits independent of the two original flotation circuits. In 1973, the daily capacity of the plant was further expanded to 11,000 tpd through modifications in the crushing plant and additions to the flotation circuit.

The principle objective of the concentrator was to separate the galena (lead sulphide) and sphalerite (zinc sulphide) from the iron sulphides and waste rock to produce high quality lead and zinc concentrates.

The basic concentrator flow sheet consisted of primary and secondary crushing, grinding (rod and ball mills), separate lead and zinc flotation circuits, dewatering, tailings disposal and load-out of lead and zinc concentrates. The flowsheet also includes a zinc concentrate leaching circuit to remove carbonates to meet certain customer concentrate specifications. See Figure 13-1 for a high-level description of a block flow diagram of the historical Pine Point concentrator.

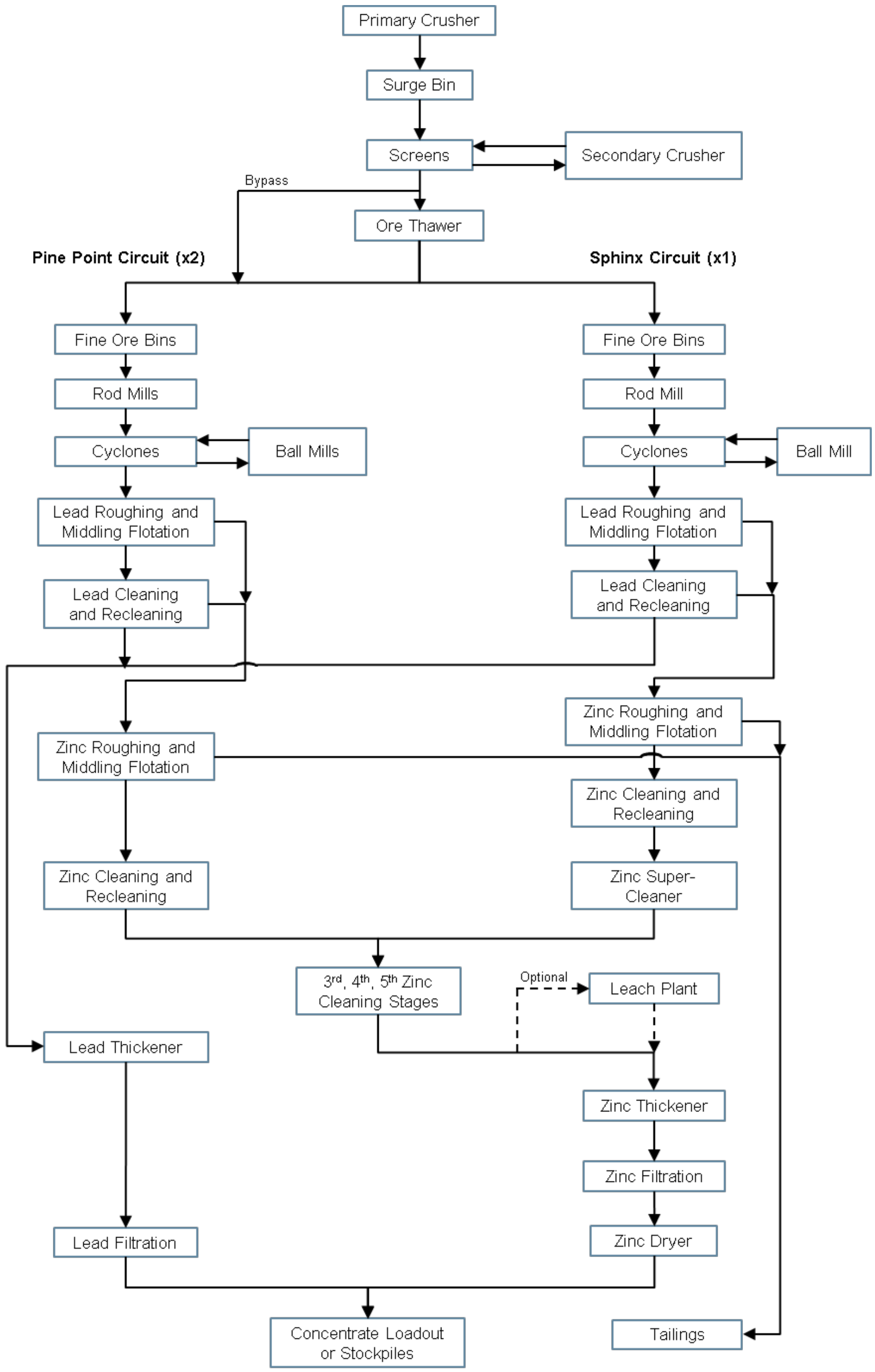


Figure 13-1: Historical process plant block diagram (Pickett, 1978)

Ore stockpiles containing up to 10 to 12 days of production (over 100,000 tonnes) of mineralized material were constructed at the concentrator adjacent to the primary crusher to enhance grade control of the feed. This also permitted material with adverse milling characteristics to be blended with more amenable material to dilute their effect on the flotation circuit.

Primary crushing was carried out in a 42 x 65 gyratory crusher, which reduced the size of the material to -5 inch. The material was then crushed to minus ¾ in an open-circuit secondary crushing system consisting of two 7 ft shorthead cone crushers. At various points in the crushing circuit heating, was provided to reduce the risk of material freezing during the winter months.

The two original Pine Point grinding circuits consisted of two conventional open-circuit rod mills (9 ft x 12 ft) followed by grinding in two closed-circuit ball mills (10 ft x 16 ft) – cyclone classifier combination arrangements. The Sphinx circuit was essentially built with the same configuration as the other circuits however the ball mill was slightly larger due to the harder material from the X-15 deposit. Flotation feed particle size distribution target was 30% -200 mesh.

Lead concentrates were produced by flotation in roughing, middling and cleaning (two stages) cells, followed by dewatering. Tailings from the lead cleaning cells was treated in the zinc circuit using roughing, middling and cleaning cells. The Pine Point zinc circuit had two stages of cleaning while the Sphinx arrangement involved three stages. Both zinc cleaner concentrates were combined for three more stages of cleaning in order to meet strict MgO and CaO penalty restrictions.

Reagents used in the flotation process included NaCN (zinc depressant), MIBC (Frother), Lime (pH modifier), CX31, Na_2S0_3 (zinc depressant) and Copper Sulphate (zinc promoter). In 1973, a continuous acid leach plant (sulphuric acid to react with carbonates) was constructed and added to the circuit following the final combined zinc cleaning stage as an optional step to produce zinc concentrates having a 0.2% MgO (maximum) for export to certain zinc smelters (Cormode, 1977).

Both Lead and zinc concentrates were dewatered by thickeners and rotating drum filters. The lead concentrate containing 5% to 6% moisture was ready for shipment following the filtration step. The zinc concentrates however required an additional drying step to reach a moisture content of approximately 4% for shipment.

According to numerous references, both the lead and zinc concentrate products were of very high quality.

13.3.2 Production History

The Pine Point Mines Ltd. (Cominco Ltd.) concentrator operated between 1964 and 1987 treating approximately 69.4 Mst (64.3 Mt) of material with a head grade of 3% Pb and 7.1% Zn. In its last full year of production (1986) the mill treated 3.5 Mt of material (3.98% Pb and 9.6% Zn) and in turn produced 163,000 short tons of lead concentrate (77.1% Pb) and 533,000 short tons of zinc concentrate (59.5% Zn). The average daily throughput for 1986 was 9,589 tpd, the lead recovery was 91.7% and the zinc recovery averaged 94.9%. Historical references indicate that operating time for the concentrator was consistently between 93% and 94%. Concentrator personnel totalled 119 consisting of 97 hourly employees and 22 staff. Table 13-1 presents a summary of annual material, lead concentrate and zinc concentrate production (Silke, 2009). It should be noted that in the early years some data related to concentrate grades and metal recoveries are missing.

Table 13-1: Pine Point production history (Silke, 2009)

Year	Feed			Lead Concentrate			Zinc Concentrate		
	Short tons	Pb (%)	Zn (%)	Short tons	Grade (Pb %)	Pb Rec. (%)	Short tons	Grade (Zn %)	Zn Rec. (%)
1964	75,000	4.3	7.6	4,000	-	-	-	-	-
1965	1,458,000	4.9	10.5	79,000	-	-	-	-	-
1966	1,521,000	4.7	9.7	83,000	-	-	-	-	-
1967	2,138,000	3.5	6.6	87,000	-	-	-	-	-
1968	3,605,000	3.2	7.4	137,000	75	89.1	431,000	-	-
1969	3,860,000	3.0	7.1	135,000	-	-	-	57	-
1970	3,892,000	2.6	6.5	118,000	-	-	-	-	-
1971	3,810,000	2.7	6.2	119,000	-	-	-	-	-
1972	3,896,000	2.9	6	130,000	-	-	-	55.6	-
1973	4,135,000	2.5	5.3	123,000	-	-	-	56.7	-
1974	3,905,000	2.4	4.9	104,000	78.2	86.8	301,000	57.9	91.1
1975	3,773,000	1.7	5.3	72,000	74.4	83.5	323,000	57.4	92.7
1976	3,443,000	2.1	5.3	85,000	73.5	86.4	290,000	56.6	90.0
1977	3,290,000	2.6	5.9	100,000	76.5	89.4	302,000	58.5	91.0
1978	3,291,000	1.9	5.5	74,000	73.7	87.2	288,000	57.3	91.2
1979	3,626,000	1.9	5.5	82,000	76	90.5	315,000	57.7	91.1
1980	3,636,000	2.0	4.8	86,000	77.1	91.2	274,000	58.4	91.7
1981	2,445,000	2.9	7.3	85,000	76.5	91.7	287,000	57.3	92.1
1982	985,000	2.7	8.2	32,000	73.8	88.8	130,000	56.9	91.6
1983	2,512,000	2.3	7.6	68,000	75.2	88.5	303,000	58.7	93.2
1984	2,356,000	3.0	8.2	83,000	74.7	87.7	300,000	59.2	91.9
1985	3,271,000	4.1	8.7	164,000	73.9	90.4	458,000	57.5	92.5
1986	3,514,000	3.9	9.6	163,000	77.1	91.7	533,000	59.5	94.0
1987	979,000	33	9.7	37,000	78.4	89.8	152,000	59.3	94.9
Total / Average	**69,416,000**	**3.0**	**7.1**	**2,250,000**	**75.6**	**88.8**	**4,687,000**	**57.7**	**92.1**

"-" = data not available

13.3.3 Historical Metallurgy

Lead and zinc grade/recovery curves were created based on the years for which operating data (1974 to 1987) was available and are summarized in Figure 13-2 and Figure 13-3 respectively. For this period the average historical lead recovery in the concentrator was 88.8% at a concentrate grade of 75.6% Pb while the average zinc recovery was 92.1% at a concentrate grade of 57.7% Zn.

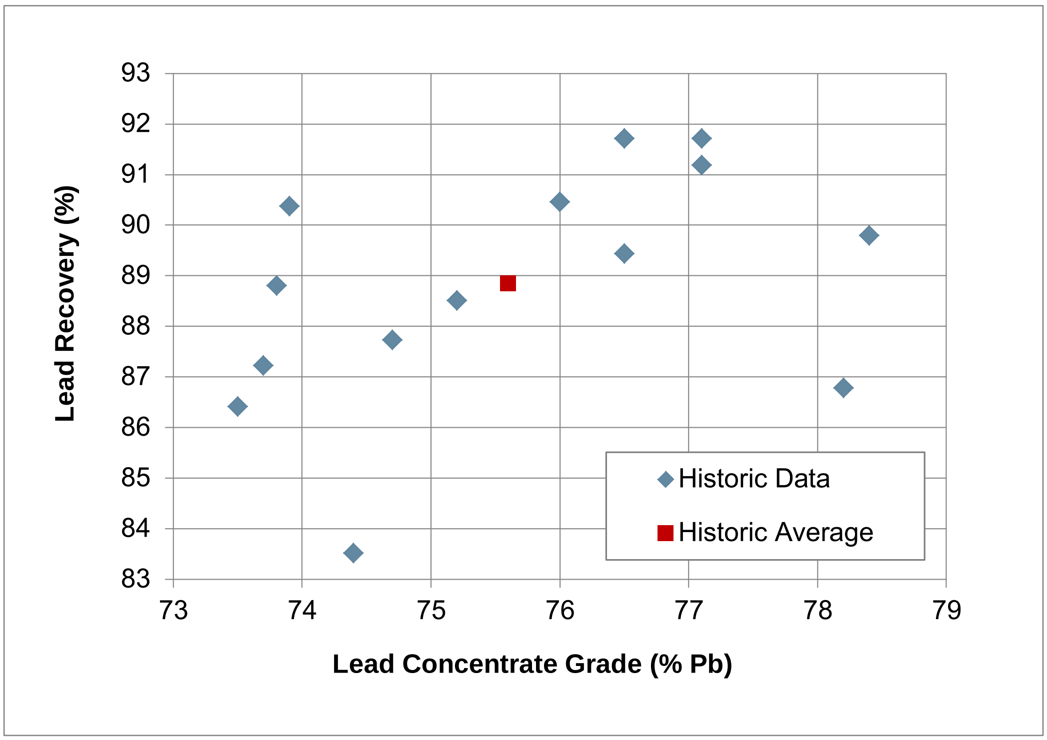


Figure 13-2: Historical lead recovery and concentrate grade (1974 to 1987)

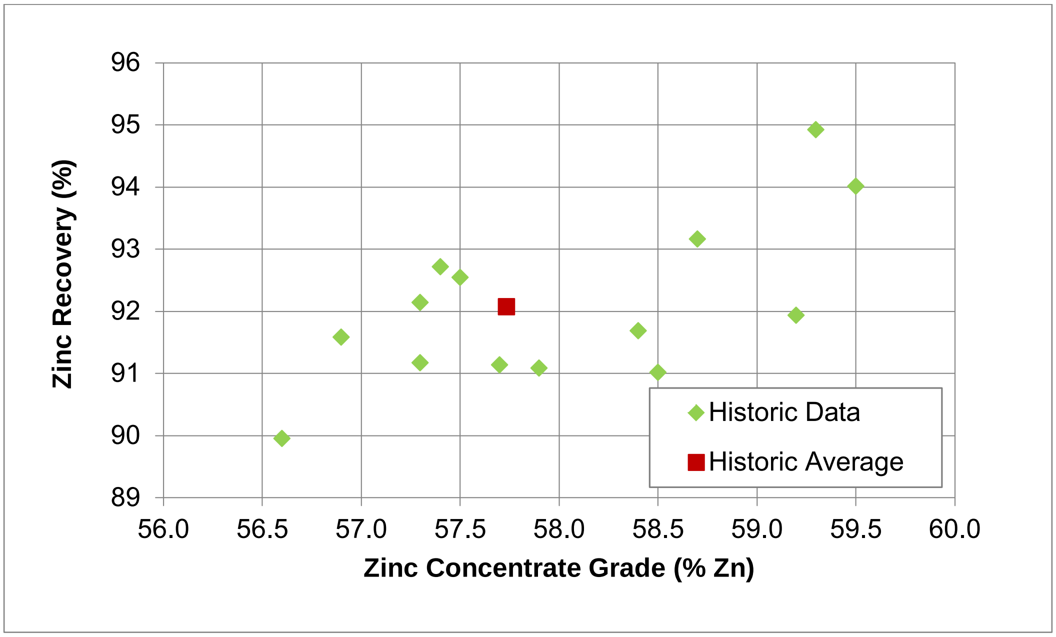


Figure 13-3: Historical zinc recovery and concentrate grade (1974 to 1987)

It is worth noting that during operations concentrate grades and/or metal recoveries were higher when processing higher grade material. For the operating period where data is available, a plot of head grade versus metal recovery was developed (Figure 13-4). Higher grades of both lead and zinc in the feed had a small but statistically significant impact on overall metal recovery to concentrate. No relationship between head grade and concentrate grade for either lead or zinc was observed.

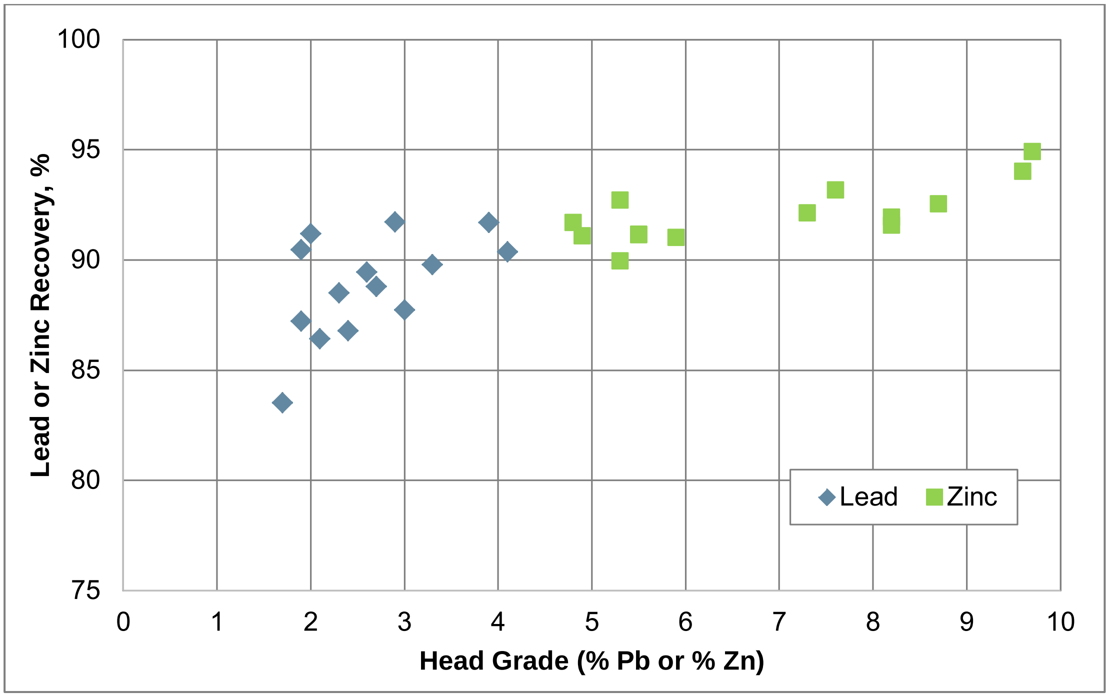


Figure 13-4: Historical head grade versus recovery relationship (1974 to 1987)

13.4 Past Metallurgical Testwork

13.4.1 Reference Reports

A number of historical metallurgical test programs were conducted on samples from multiple deposits in the Pine Point District to support the various publically issued technical reports produced by Tamerlane Ventures and Darnley Bay Resources.

The following sections are based on the results summarized in these past reports and the author's own observations. For further information on the metallurgical testing related to the Pine Point Project, please consult the following technical reports:

- Pincock, Allen & Holt, "NI 43-101 Technical Report Update Pine Point Project Northwest Territories, Canada", July 30, 2008;
- MineTech International Ltd., "Technical Report on the R-190, X-25, P-499, O-556, Z-155, and G-03 Deposits of the Pine Point Project", April 2, 2012;
- Siega, Albert & Paul Gann, "NI 43-101 Summary Technical Report Update of the Pine Point Mine Development Project Northwest Territories, Canada", March 14, 2014;
- JDS Energy and Mining Inc., "NI 43-101 Preliminary Economic Assessment Technical Report on the Pine Point Zinc Project, Northwest Territories, Canada", June 1, 2017.

13.4.2 Tamerlane Ventures (2004 to 2016)

Over a period of 12 years, Tamerlane Ventures, through its wholly owned- subsidiary Pine Point Holding Corp engaged numerous consultants and metallurgical laboratories to perform metallurgical studies and testwork for a number of deposits within their mineral claim holdings. In 2014, Tamerlane had plans to develop an underground mine based on the R-190 deposit and nine open pit mines based on eight deposits in the centrally located “Cluster Pit” (“CP”) area as well as the N-204 deposit area on the eastern extremity of the district.

The metallurgical characteristics of the N-204 deposit were found to be different when compared to the CP deposits due to the finer grained nature of the mineralization. The finer liberation size would require more grinding. The Tamerlane metallurgical testwork program is described below and it is broken down into three phases based on the deposits.

13.4.2.1 Phase 1 (R-109)

13.4.2.1.1 Dense Media Separation

Initially the objective of the metallurgical program for the project was to produce a Direct Shipment Material (“DSM”) product with a plus 45% combined lead and zinc grade. It was envisioned that DSM production would come from a relatively simple flowsheet including crushing to liberate the sulphides from the non-sulphides followed by screening to remove the crushed fines.

Dense Media Separation (“DMS”) would then be used to reject coarse non-sulphide gangue material and produce a final marketable DSM product. This approach was tested and while good lead and zinc products were recovered into a DSM product, they did not achieve the desired +45% combined lead plus zinc grade (Continental, 2006).

A second phase of testing (SGS 2007) was then performed to evaluate a finer particle size and secondary upgrading by gravity followed by coarse particle flotation. Diamond drill core from the R-190 deposit was used to develop three composite samples with different head grades (high, medium and low) for laboratory tests using heavy liquid separation (“HLS”). HLS is a laboratory version of DMS and is designed to simulate the best result that can be achieved by DMS. The HLS results for the high-grade composites showed that only a minimal upgrade with a minor weight rejection to the float product. This result indicated that high grade material may by-pass the DMS stage and report directly to the grind circuit. The medium and low-grade sample results, however, indicated that 40% to 60% weight of the ROM could be rejected with only a small loss of lead and zinc rich minerals. Thus, the DMS process was projected to save significant operating costs by rejecting a significant amount of the ROM material prior to grinding and flotation. Additional testing on the medium grade R-190 composite attempted to optimize the crushing circuit product size and employ gravity separation or coarse particle flotation on the “fines” fraction to further upgrade the R-190 material to DSM quality. Although the material was upgraded significantly, these process options failed to improve the concentrate product grade.

13.4.2.1.2 Flotation

Since attempts to produce a direct shipping material product from the Pine Point deposits by gravity mineral separation methods were unsuccessful, flotation became the focus of the metallurgical program. The objective was to develop a process flowsheet that included a DMS (or HLS for laboratory simulation) pre-concentration step followed by conventional grinding and selective lead, zinc flotation.

Published information on the reagent scheme and flowsheet details used by the 1978 Cominco Ltd., Pine Point Mill guided the development of the test program. Since the Cominco Ltd. mill relied heavily on the use of sodium cyanide, that is a compound with potential environmental impacts that would be more difficult to mitigate under present-day regulatory requirements, an array of alternative depressants were evaluated.

Batch flotation tests on the R-190 material were performed to determine the grind (particle liberation size) and reagent scheme required to produce marketable lead and zinc concentrates. Development testing was followed by laboratory locked-cycle tests (“LCT”) to confirm the batch flotation test flowsheet, concentrate grades and overall metal recoveries in the concentrates.

Table 13-2: Locked cycle flotation test results for O-556 / Z-155

Test No.	Product	WT (%)	Assay		Recovery	
			Pb (%)	Zn (%)	Pb (%)	Zn (%)
5	Pb Cleaner Concentrate	9.4	60.7	3.35	92.3	2.4
	Zn Cleaner Concentrate	20.2	0.91	61.9	3	94.2
	Zn Combined Tailings	70.4	0.42	0.64	4.7	3.4
	Head (Calculated)	100	6.19	13.2	100	100

13.4.2.2 Phase 2 (O-556 and Z-155)

The second phase of process development work was performed on samples from two deposits adjacent to the R-190 deposit, the O-556 and the Z-155 deposits. All available mineralized diamond drill core intersections from each deposit were combined into a composite sample for each deposit.

13.4.2.2.1 Comminution and Liberation Results

One Bond ball mill work index test was performed on a composite of R-190 (see G&T, 2011). A work index of 7.4 kWh/t was recorded, indicating a soft material. G&T Metallurgical Services Ltd. (2011) also performed a Particle Mineral Analysis ("PMA") using QEMSCAN technology, tests were performed on flotation feed ground to a sizing of 70 µm P_{80}. At this grind size, both galena and sphalerite had estimated liberations of approximately 70% and 80%, respectively. The unliberated galena and sphalerite were mainly in binary formats with pyrite and non-sulphide gangue. This grind size would improve the rougher concentrate grade.

13.4.2.2.2 Dense Media Separation

Small-scale heavy liquid tests were performed to assess the amenability of the two materials to DMS. Following this initial step, which confirmed that the DMS process worked, a large-scale HLS test was performed on a 300 kg composite of both O-556 and Z-155 material.

The results of this bulk DMS test are shown in Table 13-3. The bulk DMS test confirmed that the Pine Point materials are amenable to pre-concentration by HLS. More than 66% of the material was rejected as a light fraction using a liquid with a specific gravity cut point of 2.9 that resulted in very little loss of metal value.

Table 13-3: DMS test results for O-556 / Z-155

Product	Wt (%)	Assay		Recovery	
		Pb (%)	Zn (%)	Pb (%)	Zn (%)
-1/2 inch +14 mesh 2.9 SG Floats	66.38	0.058	0.21	2.8	3.1
-1/2 inch +14 mesh 2.9 SG Sinks	8.18	10.25	35	60.3	63.5
-14 mesh Fines	25.45	2.02	5.93	37	33.5
Combined Sinks and Fines	33.62	4.02	13	97.2	96.9
Calculated Head	100	1.39	4.51	100	100

Source: SGS (2008)

13.4.2.2.3 Flotation

Flotation flowsheet design and reagent scheme development continued with the HLS pre-concentrated composite sample. During this testing program, the following parameters were investigated:

- Fineness of grind;
- Various lead collectors;
- Various zinc depressants;
- Different lead circuit flowsheet configurations.

During the course of this program, it was discovered that some of the O-556 deposit material contained a bituminous oil which had the undesired effect of activating the flotation of fine clay slimes. Considerable work was devoted to developing a depressant scheme involving the use of the P92 and SQ6 flotation reagents to alleviate the negative effect of bitumen-activated slimes.

At the conclusion of the development test program a number of locked cycle tests were conducted based on the optimized parameters. The mass balance generated from the average two locked cycle tests is presented in Table 13-4.

Table 13-4: Locked cycle flotation test results for O-556 / Z-155

Test No.	Product	WT (%)	Assay		Recovery	
			Pb (%)	Zn (%)	Pb (%)	Zn (%)
22	Pb Cleaner Concentrate	4.5	72.27	1.37	90.1	0.5
	Zn Cleaner Concentrate	18.7	1.05	64.07	5.4	96.3
	Zn Combined Tailings	76.7	0.21	0.52	4.4	3.2
	Head (Calculated)	100	3.62	12.47	100	100
23	Pb Cleaner Concentrate	4.3	73.76	1.62	91.1	0.6
	Zn Cleaner Concentrate	19.4	0.94	59.14	5.3	96.6
	Zn Combined Tailings	76.3	0.16	0.44	3.6	2.8
	Head (Calculated)	100	3.45	11.85	100	100
Average 22 & 23	Pb Cleaner Concentrate	4.39	73.02	1.5	90.7	0.6
	Zn Cleaner Concentrate	19.03	0.99	61.61	5.3	96.4
	Zn Combined Tailings	76.58	0.18	0.48	4	3
	Head (Calculated)	100	3.54	12.16	100	100

Source: SGS (2008) and Pincock, Allen & Holt (2008)

The proposed general reagent scheme and dosages for the lead and zinc circuits (SGS 2008 and Pincock, Allen & Holt 2008) are shown below in Table 13-5.

Table 13-5: Proposed general reagent scheme

Process step	Flotation reagents (g/t of mill feed)							
	Lime	P92[1]	SQ6[2]	Na_2S	PAX	3418A	$CuSO_4$	3894
Grinding	1,000	1,000	250	800				
Pb Roughing					14	16		
Pb Cleaning (2 stages)	300	450	100		6	2		
Pb Circuit total	**1,300**	**1,450**	**350**	**800**	**20**	**18**		

Process step	Flotation reagents (g/t of mill feed)							
	Lime	P92[1]	SQ6[2]	Na_2S	PAX	3418A	$CuSO_4$	3894
Zn Conditioning	1,200				20		1,200	10
Zn Roughing					7			4
Zn Cleaning (3 stages)					5			2
Zn Circuit total	**1,200**		**400**		**32**		**1,200**	**16**
Overall total	**2,500**	**1,450**	**750**	**800**	**52**	**18**	**1,200**	**16**

(1) P92 is composed of $ZnS0_4$ (66%), $Na_2S_20_5$ (17%) and Oxalix Acid (17%)

(2) SQ6 is composed of Acrboxymethyl Cellulose (CMC) (39%), Suspendol PKK (Cognis) (39%) and Ethylenediamine Tetra Acetic Acid (22%)

The lead and zinc concentrates produced were analyzed for trace deleterious elements that might constitute smelter impurities. The analysis showed the concentrates were of good quality and should not be subject to smelter penalties.

13.4.2.3 Phase 3 (N-204)

The third phase of process development work (DMS and Flotation) was performed in 2011 on samples from the N-204 deposit which is located to the northeast of the Central Zone area. The metallurgical characteristics of the N-204 deposit differ from the Central and East Mill Zone areas in that it is finer grained. The head grade assay for the sample is shown in Table 13-6.

Table 13-6: N-204 Composite head assays

Pb (%)	Zn (%)	Fe (%)	Cu (%)	S (%)
0.92	3.67	0.93	<0.001	2.21

Source: SGS (2011)

13.4.2.3.1 Dense Media Separation

A large-scale HLS test was completed on ~100 kg of -13 mm + 20 mesh material at a specific gravity of 2.7. Between 97% and 98% of the Pb and Zn were recovered to the sinks. It was noted that a large proportion of the mass reported to the sinks resulting in a low upgrading of the feed. A decision was made to re-pass the approximately 80 kg of 2.7 sinks at a higher specific gravity of 2.8, which produced a better result. An overall summary of the results is presented in Table 13-7.

Table 13-7: DMS Test results for N-204

Product	Wt (%)	Assay		Recovery	
		Pb (%)	Zn (%)	Pb (%)	Zn (%)
SG 2.8 Sinks	22.1	3.54	12.1	82.4	83.5
SG 2.8 Floats	54	0.19	0.65	12.2	11
SG 2.7 Floats	21.2	0.1	0.33	2.5	2.2
-20 Mesh Fines	2.7	0.9	4.02	2.9	3.4
Calculated Head	100	0.84	3.2	100	100
Flotation Feed (Average)	24.8	2.9	11.2	85.3	86.9
HLS Rejects	75.2	0.16	0.56	14.7	13.1

Source: SGS (2011)

13.4.2.3.2 Flotation

A limited flotation testing program was conducted on a Master Composite N-204 (HLS sinks). The flotation testing consisted of a series of open circuit batch cleaner tests followed by a single locked cycle test. The results for the locked cycle test are shown in Table 13-8.

Table 13-8: Locked cycle flotation test results for N-204

Product	Wt (%)	Assay		Recovery (%)	
		Pb (%)	Zn (%)	Pb	Zn
Pb Feed	100	2.65	10.9	100	100
Pre-Float	1.65	5.81	9.2	3.6	1.4
Pb Concentrate	3.98	54.3	10.7	81.6	3.9
Zn Concentrate	17.23	1.04	55.7	6.8	88.3
Zn 1st Cleaner Tails	7.5	1.77	7.2	5	5
Zn Rougher Tails	69.65	0.11	0.23	3	1.5

Source: G&T (2011)

13.4.2.3.3 Tamerlane Flowsheet

The proposed Tamerlane Ventures process plant flowsheet was based on the previously described testwork and contained many common elements to the historical Cominco Ltd. operation flowsheet.

Tamerlane proposed to begin mining the high-grade R-190 deposit as an underground operation followed by open pit mining of the cluster pits. The metallurgical testwork program demonstrated that standard zinc and lead flotation preceded by dense media separation (DMS) would yield recoveries, above 90 percent and good concentrate grades. The proposed Tamerlane flowsheet is described in detail within the 2014 NI 43-101 summary report by Siega and Gann.

To reduce transportation and processing costs, Tamerlane proposed that the material be first processed at two crushing/DMS pre-concentration plants. One DMS plant was to be located near by the Cluster Pits and the other near to the N-204 deposit. The DMS concentrate from both plants would then be trucked to a centralised grinding/flotation plant (1,800 tpd capacity) at the R-190 mine site. It was expected that the pre-DMS feed would average above 2 % Pb and 5 % Zn. The post- DMS concentrate (sinks) was expected to average between 18% and 20% (Pb+Zn). Float rejects from the DMS (waste rock) was proposed to be utilized as construction material on site to build roads, possibly sold as aggregates, or used as pit backfill and/or reintroduced back into the mined out stopes.

The proposed grinding/flotation plant consisted of secondary and tertiary crushing; a grinding circuit consisting of a ball mill in closed-circuit with cyclones; pre-float flash flotation cell to scalp bitumen activated slimes, sequential lead and zinc flotation circuits each incorporating three stages of cleaning; concentrate dewatering circuits using thickeners and pressure filters; concentrate storage, load-out, and transportation systems; and tailings dewatering using a thickener and vacuum filters together with a slurry containment pond for excess tailings. The flotation reagent scheme did not use cyanide due to environmental protection commitments. The filtered tailings were planned to be back-hauled to assist in reclaiming the mined pits along with the DMS reject and mine waste material.

13.4.3 Darnley Bay Resources (2017)

13.4.3.1 Overview

Based on the authors review, no metallurgical testwork was performed during the period under which the Pine Point properties were owned by Darnley Bay. In 2017, Darnley Bay issued a PEA that was based on open pit mining and a process plant flowsheet similar to that proposed by the previous owner Tamerlane Ventures. The Process plant consisted of three-stage crushing and screening, Dense Media Separation, ball milling, pre-flotation to remove bitumen-activated slimes, lead flotation followed by zinc flotation, both using conventional and column cells, producing separate lead and zinc concentrates. Dewatering of the concentrates was performed by thickeners and drum filters.

The PEA used the previously summarized Tamerlane testwork and historical metallurgical performance of the Pine point operations as the basis for their predictions of Lead and Zinc recovery and concentrate grades (Table 13-9 and Table 13-10). It was assumed that the testwork performed on R-190 material could be used to predict the overall metallurgy of the Cluster Pit Zone material due to geological similarities. The recovery and grade predictions for N-204 material were shown separately due to its differing mineralogy.

Table 13-9: Metallurgical projections (R-190)

	Grade		Recovery	
	Pb (%)	Zn (%)	Pb (%)	Zn (%)
Feed	2.31	4.71		
DMS Stage Concentrate	6.91	13.95	94.4	93.5
Flotation Concentrate	70.0	61.9	90.0	94.2
Overall Recovery	70.0	61.9	85.0	88.1

Table 13-10: Metallurgical projections (N-204)

	Grade		Recovery	
	Pb (%)	Zn (%)	Pb (%)	Zn (%)
Feed	0.7	2.6		
DMS Stage Concentrate	2.47	9.32	87.1	88.4
Flotation Concentrate	55.7	55.3	82.0	88.2
Overall Recovery	55.7	55.3	71.4	78.0

13.5 Metallurgical Testwork - Osisko Metals (2018)

In mid-2018, Osisko Metals initiated a small-scale metallurgical testing program to determine whether sensor-based sorting technology could be used as an alternative to Dense Media Separation to upgrade the feed material prior to grinding and flotation as previously proposed by Tamerlane and Darnley Bay.

13.5.1 Sample Source

Approximately 800 kg of material was gathered from mineralized core available from the summer 2017 and winter 2018 drill programs. Four composite samples (approximately 200 kg each) were created as follows:

- High Grade ("HG") material: approximately 15% Zn and 6% Pb;
- Medium Grade ("MG") material: approximately 10-15% Zn and 3% Pb;
- Average (LOM) material: approximately 5% Zn and 2% Pb;
- Low Grade ("LG") material: approximately 2% Zn and 1% Pb.

13.5.2 Mineral Sorting

13.5.2.1 Introduction

Testwork on the previously described composite samples was performed by Steinert, a globally recognized supplier of mineral sorting technology at their Walton Kentucky US test facility. A variety of sensors can be utilized either individually or in a combination of different sensors to ensure the efficient sorting of minerals. The Pine Point samples were processed using X-ray transmission ("XRT") as the primary detection technology that enables materials to be recognized and separated based on their specific atomic density. One particular advantage of XRT sorting is that the particles do not need to be cleaned/washed, which is typically necessary when using other surface detection sensors such as cameras or lasers.

Prior to performing the mineral sorting tests, handpicked rock samples where scanned separately to define the x-ray absorption patterns of each rock type. This density information was then used to develop scatterplots which became the basis of development for the algorithm that was applied by the sorting computer to determine if the particle being analyzed would be rejected or accepted.

13.5.2.2 Results and Interpretation

Ten mineral sorting tests were performed using a five-step approach on various blends of the four samples provided. Samples were screened to remove fine material and sorting was performed on the -60 mm/+10 mm fraction. XRT scans conducted on the samples showed good differentiation between the poly-metallic sulphide mineralization and the host waste rock. Test results are summarized in Table 13-11:

Table 13-11: Mineral sorting testwork summary (-60 mm/+10 mm)

Test	Sample	Head grade		Concentrate			Upgrade ratio		Recovery (%)	
		Pb (%)	Zn (%)	Wt (%)	Pb (%)	Zn (%)	Pb	Zn	Pb	Zn
1	MG-LG	1.0	5.0	41%	2.4	11.2	2.4	2.3	98%	93%
2	LOM-MG	1.1	6.2	45%	2.4	12.6	2.1	2.1	97%	93%
3	MG-HG	3.3	11.4	60%	5.5	18.6	1.7	1.6	100%	99%
4	LG	0.5	2.0	32%	1.5	4.8	3.0	2.4	95%	77%
5	MG	1.3	7.3	51%	2.5	13.6	1.9	1.9	98%	96%
6	LOM-LG	0.7	3.2	34%	1.9	8.0	2.8	2.5	95%	86%
7	HG-LG	2.4	8.8	49%	4.9	17.2	2.0	1.9	99%	96%
8	LOM	1.1	4.6	44%	2.5	9.5	2.2	2.1	97%	91%

Test	Sample	Head grade		Concentrate			Upgrade ratio		Recovery (%)	
		Pb (%)	Zn (%)	Wt (%)	Pb (%)	Zn (%)	Pb	Zn	Pb	Zn
9	HG-LOM	2.5	11.3	63%	3.9	17.4	1.6	1.5	99%	97%
10	HG	4.8	16.3	70%	6.8	22.8	1.4	1.4	100%	98%
Average		**1.9**	**7.6**	**49%**	**3.4**	**13.6**	**2.1**	**2.0**	**98%**	**93%**

Source: Steinert (2018)

The following three graphs (Figure 13-5, Figure 13-6 and Figure 13-7) show the impact of Zn and Pb feed grade on concentrate mass pull, zinc recovery and lead recovery.

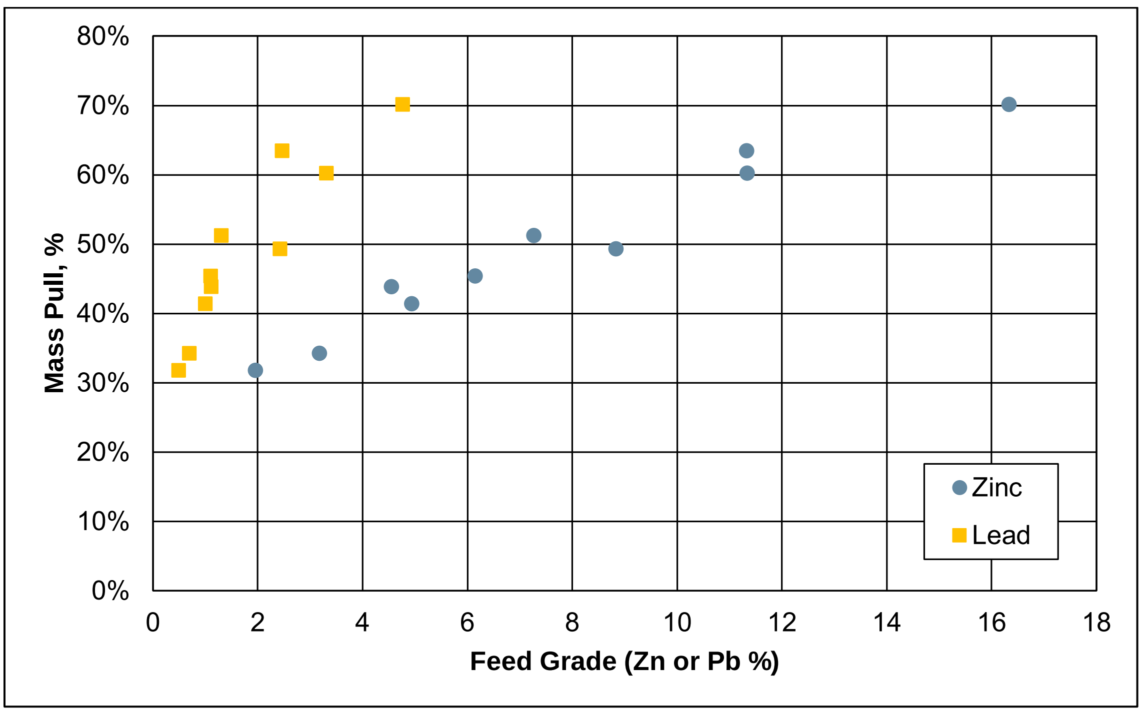


Figure 13-5: Feed grade versus mass pull to concentrate

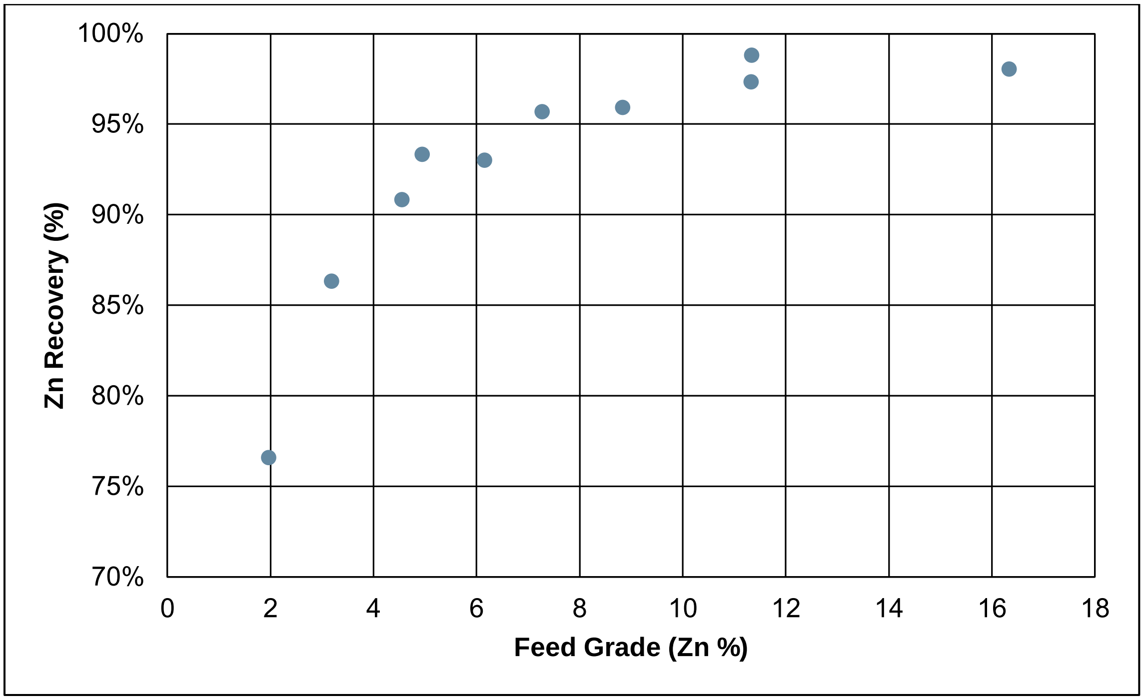


Figure 13-6: Zn feed grade versus recovery

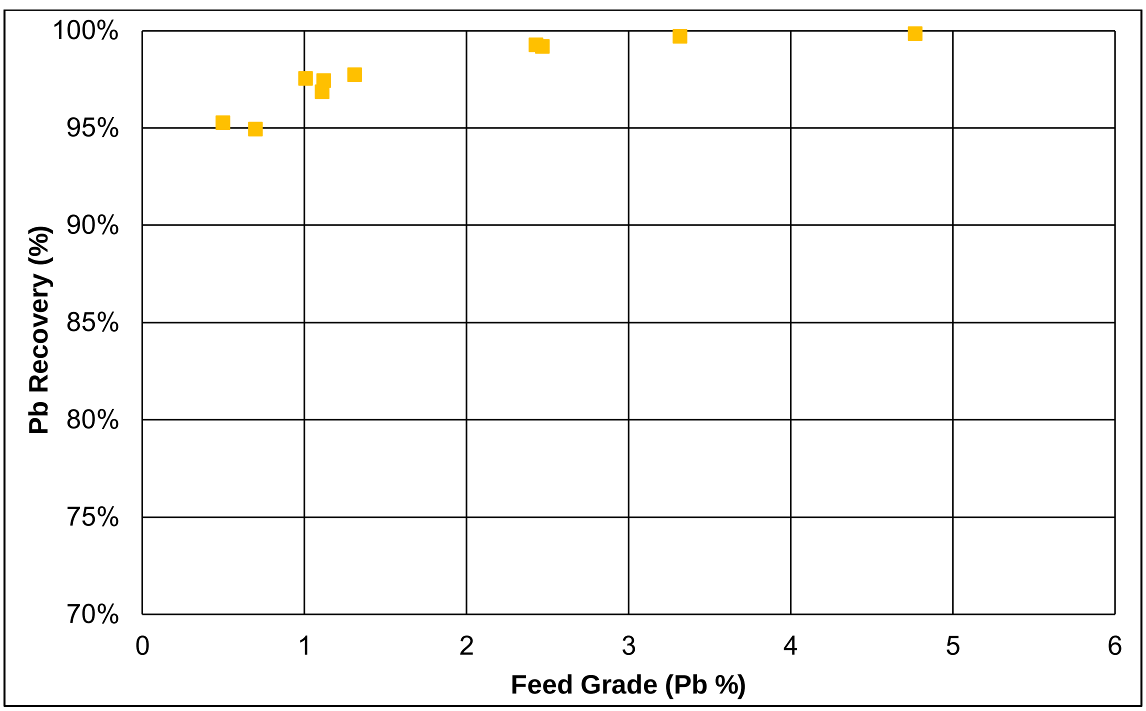


Figure 13-7: Pb feed grade versus recovery

Overall the mineral sorting test program indicated that the Pine Point material is well suited for sensor-based sorting. Based on an average of the ten samples tested, mineral sorting was able to increase the grade of both Pb and Zn by approximately two times, achieve recoveries of 98% for Pb and 93% for Zinc while rejecting approximately 50% of the mass to a waste product. In general, metal recovery to concentrate generally increased with feed grade. Pb recovery was higher than Zn recovery for all sorting runs.

13.5.3 Flotation

As of the date of this report, flotation testwork based on the products from the mineral sorting testwork has been initiated but the final results have not yet been received.

13.6 Recovery and Grade Projections – Mineral Resource Estimate

As part of the Osisko Metals 2018 Pine Point Mineral Resource Estimate, pit shells were developed to constrain the Mineral Resource Estimate based on economic and metallurgical parameters. Preliminary processing recovery and grade projections that were estimated for the Central and East Mill Zone areas (Table 13-12) and the N-204 deposit (Table 13-13). Mineral sorting pre-concentration recoveries of Zn and Pb are based on the recovery versus head grade relationships developed from the recent 2018 Mineral Sorting Test Program (Section 13.5.2) while the flotation Zn and Pb recoveries are based on Tamerlane testwork and historical production data. To estimate the overall recovery of Zn and Pb, a head grade of 3.4% Zn and 1.1% Pb was used, while conservative, final concentrate grades of 55% Zn and 55% Pb were assumed. It should be noted that these are preliminary estimates and additional confirmation testwork is planned.

Table 13-12: Recovery projections (Central Zone)

	Recovery (%)	
	Pb	Zn
Pre-concentration (Mineral Sorting)	97.6	88.3
Flotation Concentrate	90.0	94.2
Overall Recovery	87.8	83.1

Table 13-13: Metallurgical projections (N-204)

	Recovery (%)	
	Pb	Zn
Pre-concentration (Mineral Sorting)	97.6	88.3
Flotation Concentrate	82.0	88.2
Overall Recovery	80.0	77.8

13.7 Recommendations for Future Work

Colin Hardie, QP, recommends the following testwork and studies to validate the metallurgical characteristics and variability of the major geological zones (Central, East Mill, North Trend, West Zones and the N-204 deposit) of the Pine Point District as well as to determine the most suitable process flowsheet:

- Pre-concentration
  - Complete ongoing mineral sorting and dense media testing programs;
  - Process technology selection trade off study.
- Crushing and grinding
  - Comminution testwork to assess deposit variability and to gather data for mill sizing and power calculation purposes.
- Flotation
  - Investigate the impact of grind size on recovery;
  - Reagent evaluation;
  - Validate impact of slimes and bitumen;
  - Flowsheet definition.
- Concentrate characterization
  - Validate the quality of the Pb and Zn concentrates for potential impurities which could lead to smelter penalties.
- Dewatering testwork
  - Gather sedimentation and filtration data of the concentrate and tailings for equipment sizing purposes.

14. MINERAL RESOURCE ESTIMATE

BBA was retained by Osisko Metals to update the Mineral Resource Estimate ("MRE") for the Pine Point project (the "Project"), incorporates historical drilling data and recent drilling programs. Drillhole information up to September 12, 2018 was considered for this estimate.

The Pine Point Mining Camp ("PPMC"), recently acquired by Osisko Metals, was discovered in 1898 and exploited from 1964 to 1987. During this period, around 64 Mt of ore grading 7.0% Zn and 3.1% Pb was extracted from approximately 50 open-pits and two underground deposits.

14.1 Methodology

The herein MRE covers the whole Pine Point project with a strike length of 63 km and a width of approximately 8 km, down to a vertical depth of 200 m below surface. Figure 14-1 shows the Pine Point project along with the naming for the different areas.

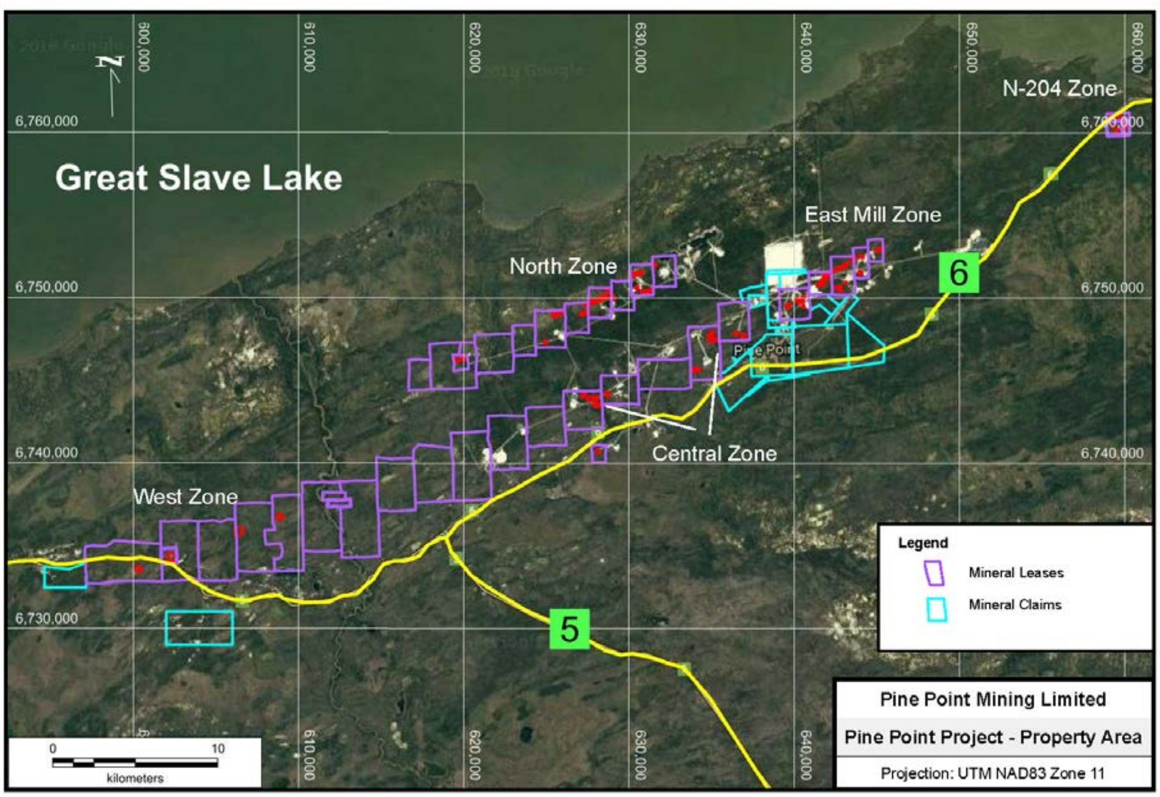


Figure 14-1: Overall plan view for the Pine Point project

Leapfrog Geo™ v.4.3.1 was used for the modelling of 243 mineralized zones and for the generation of the drillhole intercepts for each solid. GEOVIA GEMS v.6.8.2 was used for the compositing, the 3D block modelling and for the interpolation. Statistical studies were conducted using Excel and Snowden Supervisor v. 8.9.0.2.

The methodology for the estimation of the mineral resources involved the following steps:

- Database verification;
- 3D modelling of the mineralized zones;
- Drillhole intercept and composite generation for each mineralized zone;
- Basic statistics
- Capping;
- Geostatistical analysis including variography;
- Block modelling and grade interpolation;
- Block model validation;
- Resource classification;
- Cut-off grade calculation and pit shell optimization;
- Preparation of the mineral resource statement.

14.2 Resource Database

The resource database for the Project, as of September 12, 2018, consisted of 18,542 surface drillholes with a cumulative length of 1,314,033 m (Figure 14-2). The average length of a drillhole is 70 m. Of these 18,542 drillholes, a subset of 6,880 holes cut across the mineralized zones with a total of 31,120 assays. The drillhole database includes Osisko Metals infill drilling of 23,751 m in 318 drillholes and also incorporates Cominco Ltd.'s historical drillholes, the use of which was validated by a drillhole collar survey and a partial core re-sampling program. The database was validated as part of the current mandate.

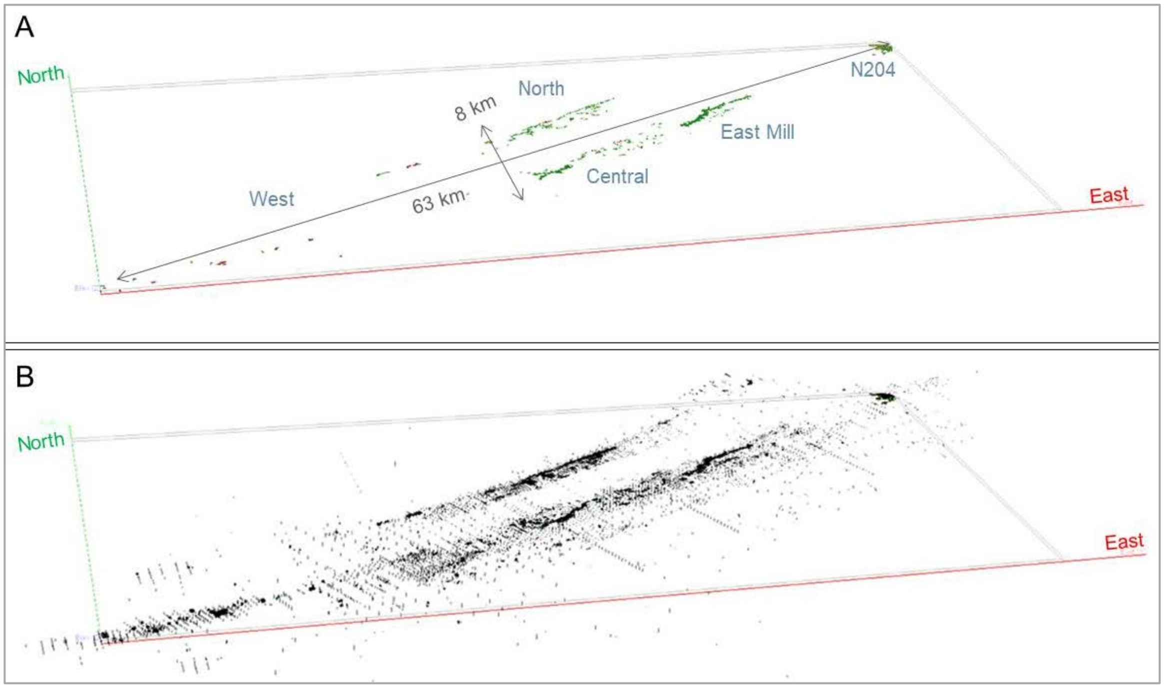


Figure 14-2: Overall plan view of the 3D Model of the mineralized zones (A) and of the drillholes included in this resource estimate (B)

14.3 Geological Model

Geological wireframes were constructed by BBA in Leapfrog Geo™. The model comprises 243 zinc-lead-bearing zones snapped to drillholes, which have a minimum thickness of 2.5 m. They were modelled using geological knowledge of the deposit, grade continuity and a weighted average zinc-lead grade above 1% Pb+Zn.

The mineralized zones are sub-horizontal (Figure 14-2 to Figure 14-5) and can be separated into three types:

- **Prismatic zones** – generally high-grade zones that are vertically continuous for up to 60 m, but with limited lateral extent (generally 15 m to 50 m, up to 140 m);
- **Tabular zones** – laterally continuous zones that follow the general carbonate stratigraphy and platform trend and generally having lower grade than the prismatic zones;
- **N-204 zones** – located in the northeastern portion of the Pine Point project, these zones consist of finer crystalline mineralization that was deposited within the "B-Spongy" horizon. It differs from the more typical Sulphur Point-hosted tabular mineralization and from the sulphide-rich prismatic-type.

The topographic surface was created by BBA in Geovia GEMS™ and is based on the drillholes collar coordinates and elevation. A similar approach was chosen for the overburden-rock interface. The mineralized zones were clipped to the overburden/bedrock interface when necessary.

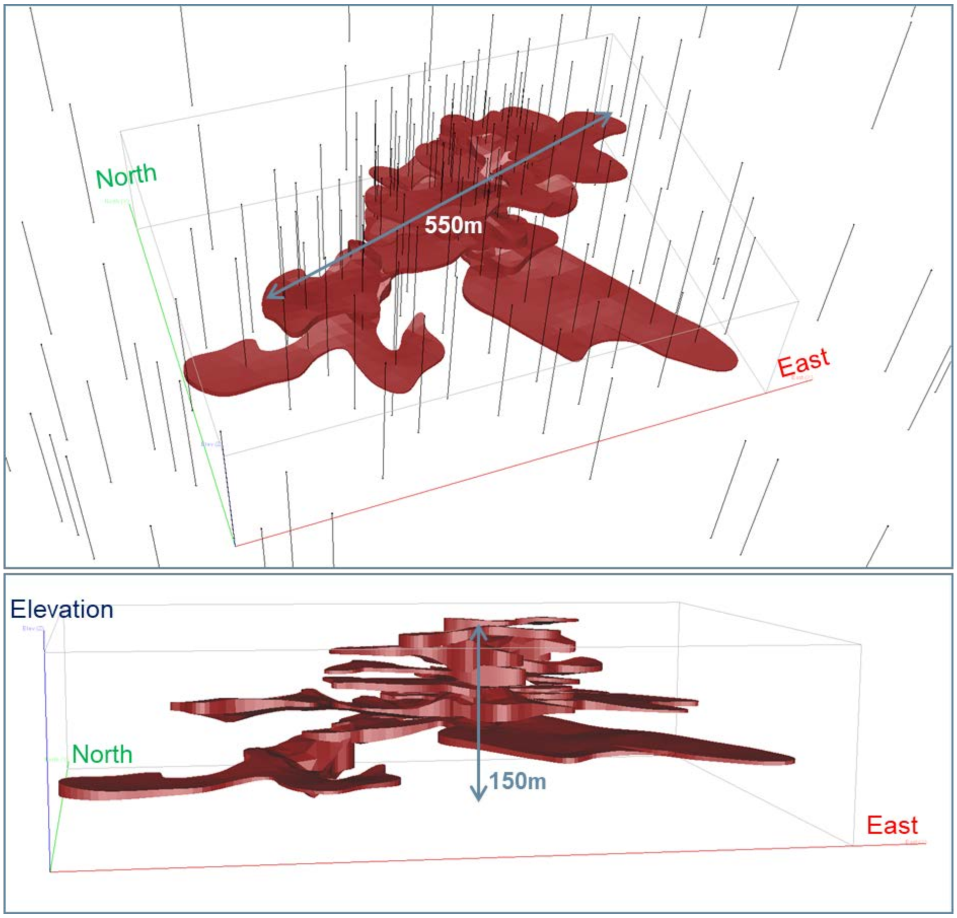


Figure 14-3: Example of prismatic zones (C2W_1004) in plan view (top) and in 3D view looking north (bottom)[1]

[1] The reader should note that the zone presented in this image was not necessarily entirely interpolated because some area did not have sufficient drilling at this time. No resources were classified as such without at least two drillholes being used and at least one drillhole within 50 m from each block.

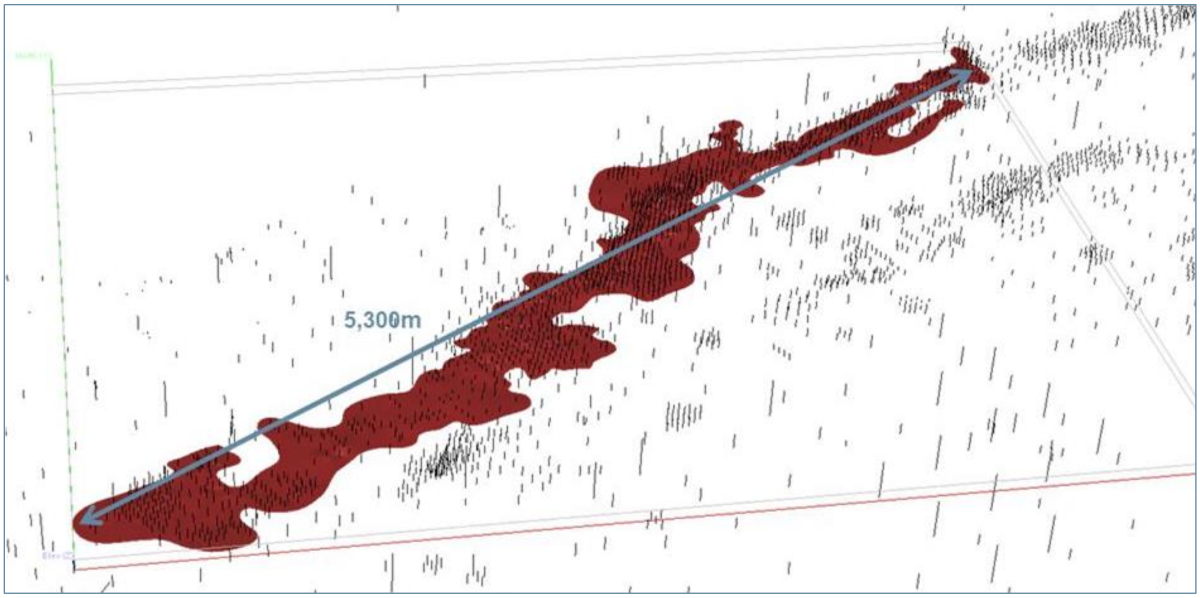


Figure 14-4: Example of a tabular zone (C1_2002)[2]

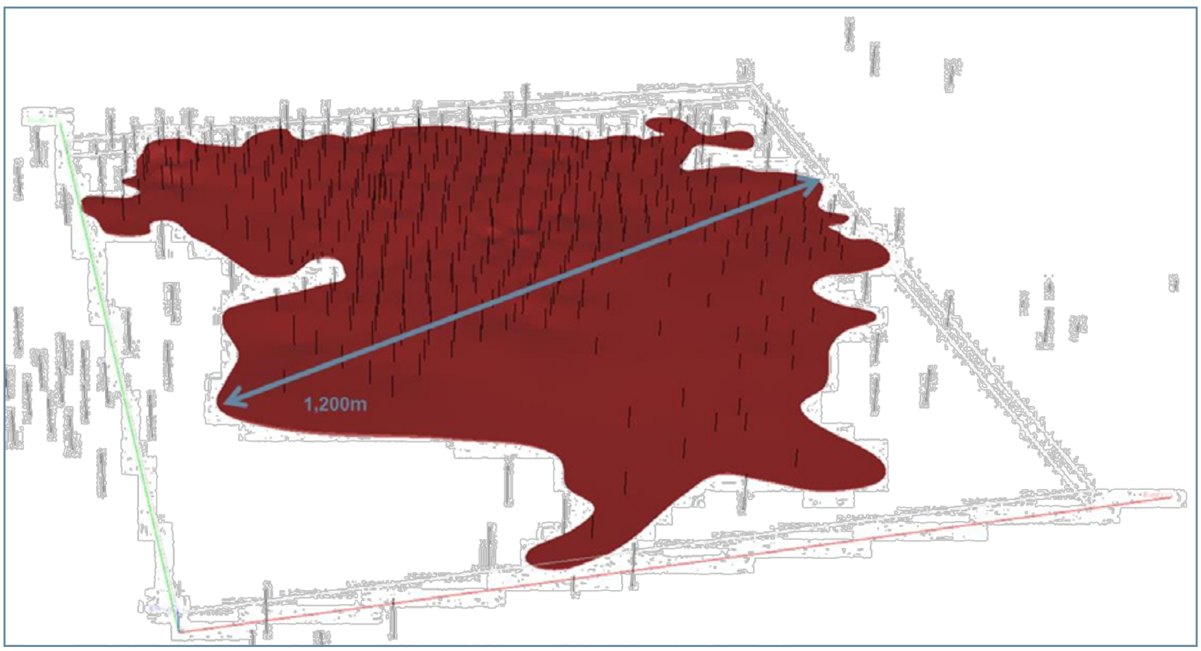


Figure 14-5: Example of N-204 zones (N-204_2001)[2]

[2] The reader should note that the zone presented in this image was not necessarily entirely interpolated because some area did not have sufficient drilling at this time. No resources were classified as such without at least two drillholes being used and at least one drillhole within 50 m from each block.

14.4 Voids Model

It was decided to sterilize all of the blocks that lay within historical pit surface contours. Without a precise topographic survey, such as a LiDAR survey, and a bathymetric survey, this approach was judged to be the most appropriate one although it penalizes some of the deposits. Any future MRE update should consider the LiDAR and bathymetric surveys currently being carried out by Pine Point Mining Limited at the time of completing the current MRE). The same approach was applied to historical underground workings.

14.5 Compositing

All raw assay data that intersected mineralized zones were assigned individual rock codes. These coded intercepts were used to produce basic statistics on sample lengths and grades. A total of 31,120 assays are included in the mineralized zones.

Compositing of drillhole samples was conducted in order to homogenize the database for the statistical analysis and remove any bias associated to the sample length that may exist in the original database. The composite length was determined using original sample length statistics and the thickness of the mineralized zones.

Inside the mineralized zones, 95% of the samples are between 0.5 m and 3.10 m in length. The average sample length is 1.45 m. As a result, 26,133 composites were generated with a length of 2 m, but ranging from 1 m to 3 m when necessary after redistributing the tails. Figure 14-6 shows the sample length distribution within the mineralized zones.

Grades of 0.00 % Zn and 0.00% Pb were assigned to all missing intervals during the compositing process.

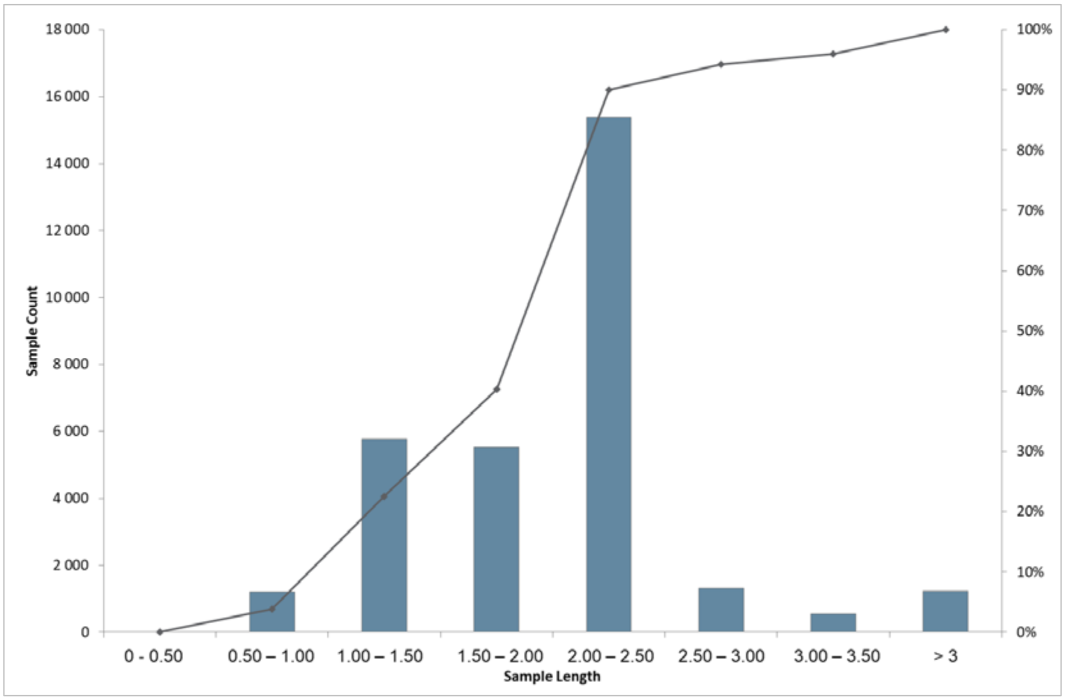


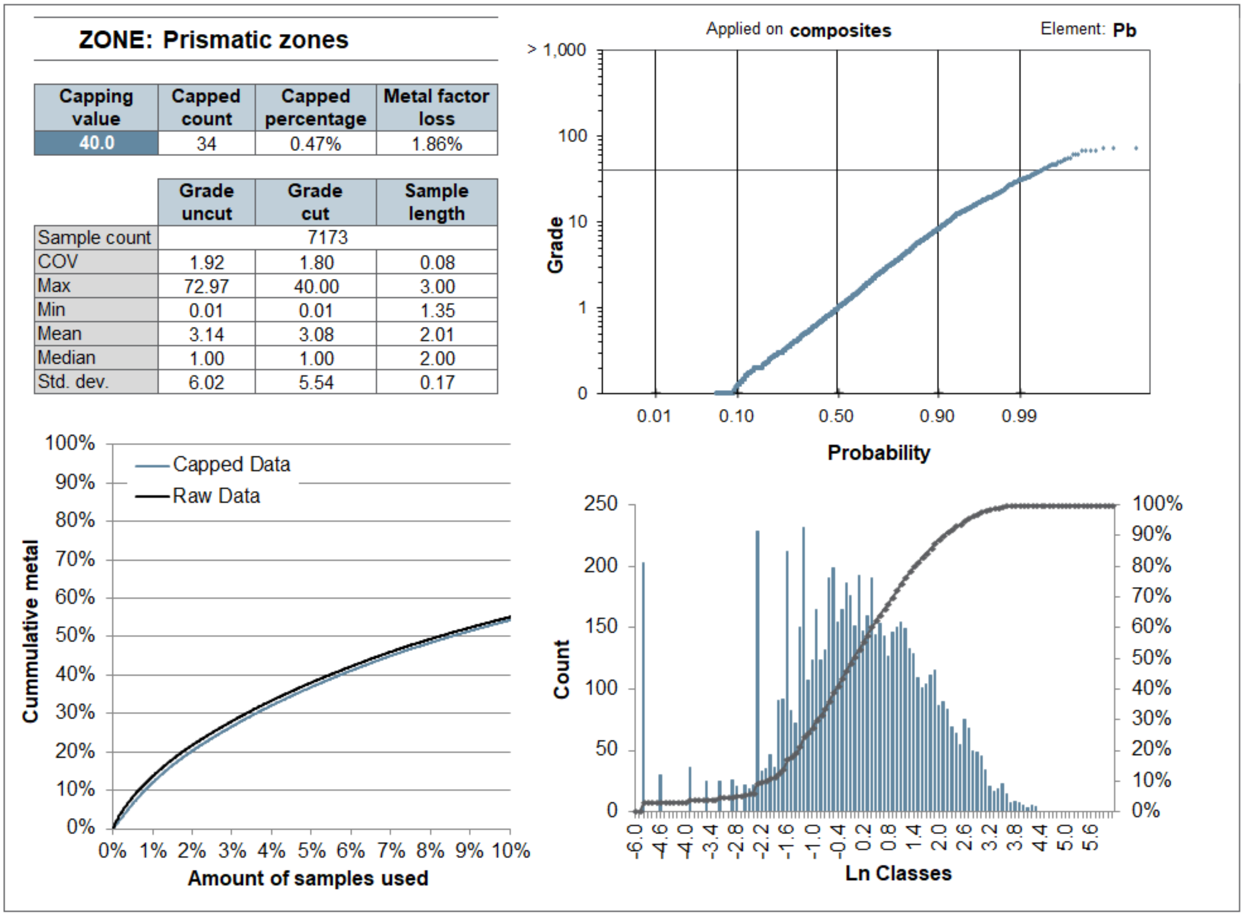
Figure 14-6: Sample length distribution within the mineralized zones

14.5.1 Capping

It is common practice to statistically examine the higher grades within a population and to trim them to a lower grade value based on the results of a statistical study. The capping is performed on high-grade values considered to be outliers. High-grade capping was done on the composited assay data and established on a per zone type basis (tabular, prismatic, or N-204).

The capping values were defined by checking for abnormal breaks or change of slope on the grade distribution probability plot while making sure that the coefficient of variation of the capped data was ideally lower than 2.00 and no more than 10% of the total contained metal was enclosed within the first 1% of the highest-grade samples. The use of various statistical methods allows for a selection of the capping threshold in a more objective and justified manner. Capping grades vary from 10% to 35% Zn and 5% to 40% Pb.

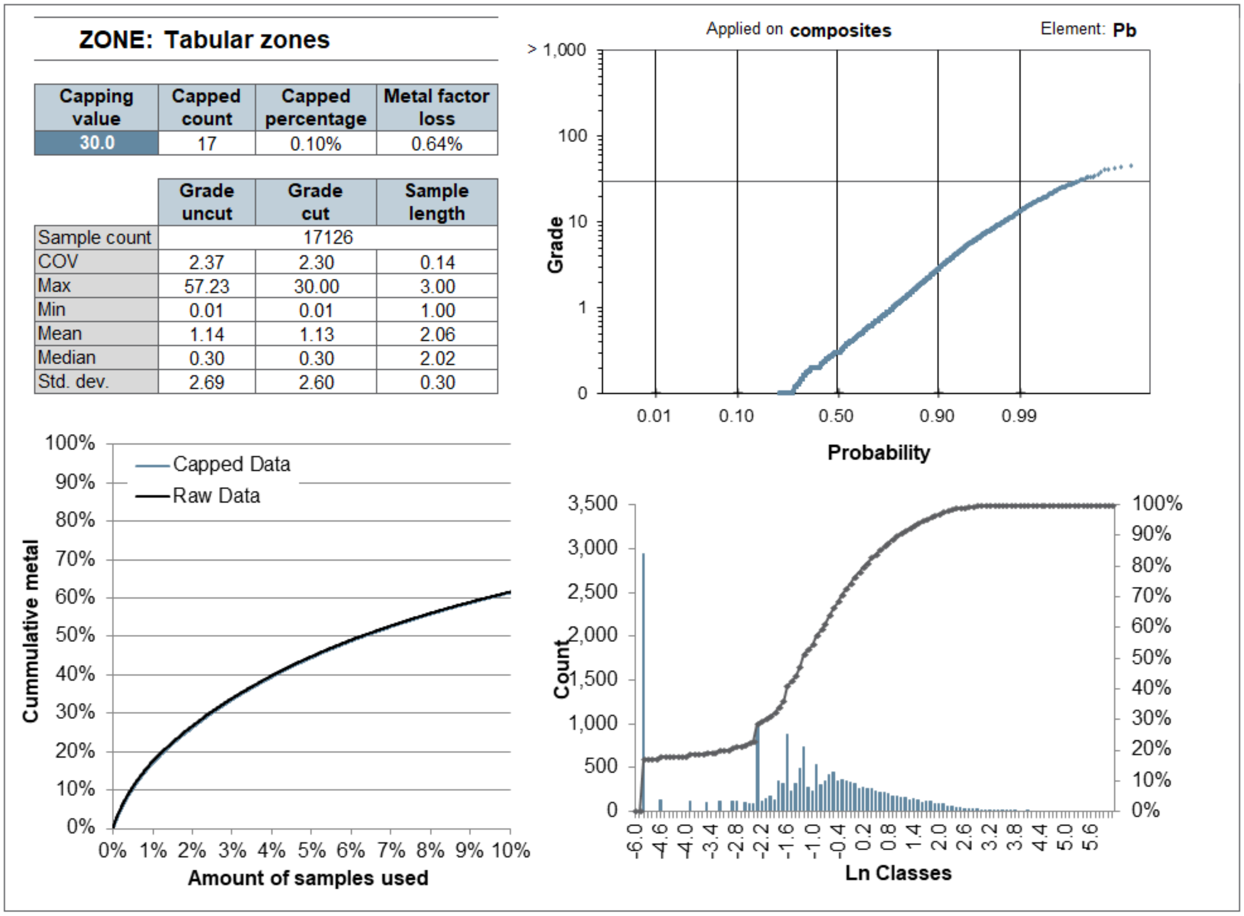
Basic statistics for Pb and Zn assays and capped assays are summarized in Table 14-1. Figure 14-7 to Figure 14-12 show graphs supporting the capping threshold decisions.



Capping value	Capped count	Capped percentage	Metal factor loss
40.0	34	0.47%	1.86%

	Grade uncut	Grade cut	Sample length
Sample count		7173	
COV	1.92	1.80	0.08
Max	72.97	40.00	3.00
Min	0.01	0.01	1.35
Mean	3.14	3.08	2.01
Median	1.00	1.00	2.00
Std. dev.	6.02	5.54	0.17

Figure 14-7: Graphs supporting capping on composites for the prismatic zones for Pb



ZONE: Tabular zones

Capping value	Capped count	Capped percentage	Metal factor loss
30.0	17	0.10%	0.64%

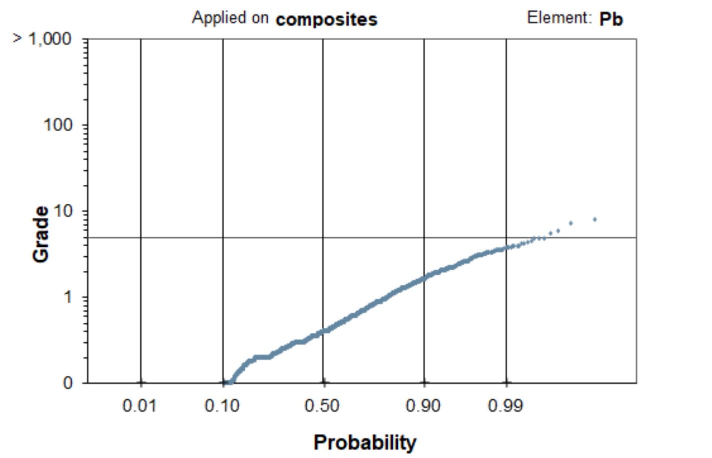
	Grade uncut	Grade cut	Sample length
Sample count	17126		
COV	2.37	2.30	0.14
Max	57.23	30.00	3.00
Min	0.01	0.01	1.00
Mean	1.14	1.13	2.06
Median	0.30	0.30	2.02
Std. dev.	2.69	2.60	0.30

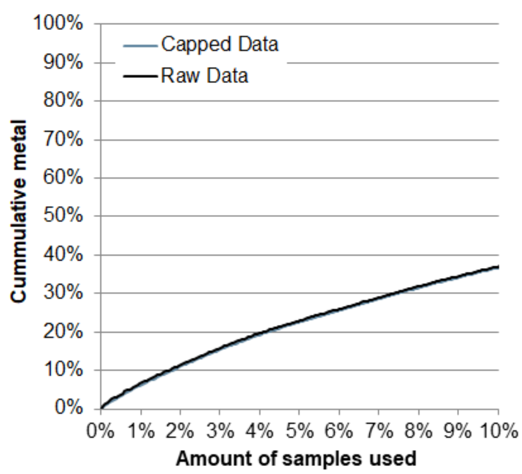
Figure 14-8: Graphs supporting capping on composites for the tabular zones for Pb

ZONE: N204

Capping value	Capped count	Capped percentage	Metal factor loss
5.0	4	0.22%	0.52%

	Grade uncut	Grade cut	Sample length
Sample count	1834		
COV	1.16	1.13	0.14
Max	7.89	5.00	2.95
Min	0.01	0.01	1.05
Mean	0.68	0.68	2.08
Median	0.40	0.40	2.03
Std. dev.	0.79	0.76	0.29





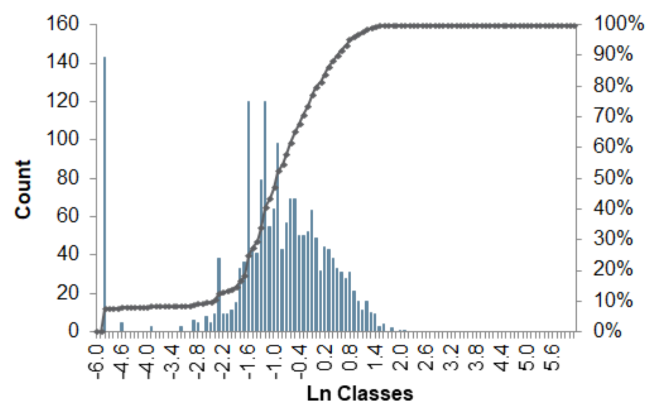
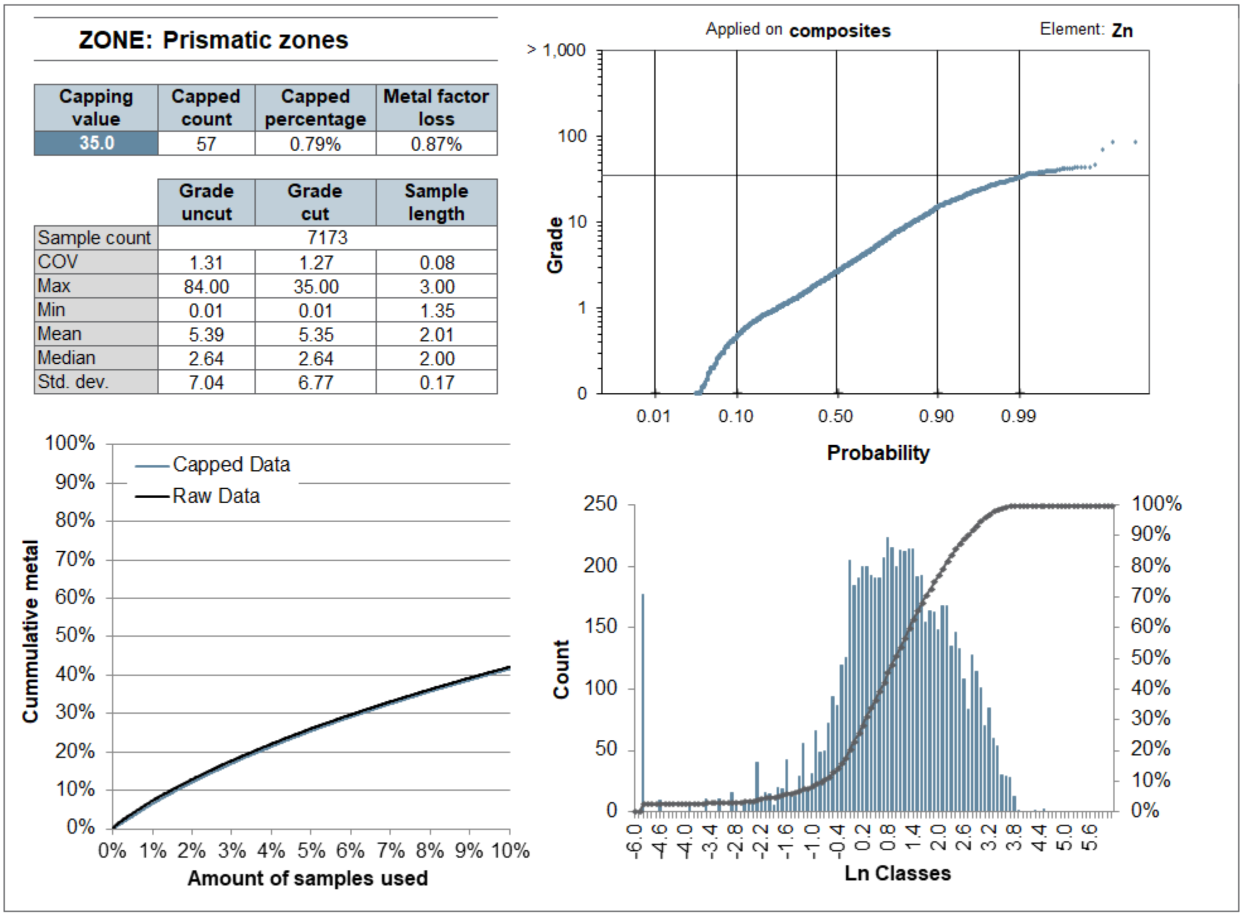


Figure 14-9: Graphs supporting capping on composites for the N-204 zones for Pb

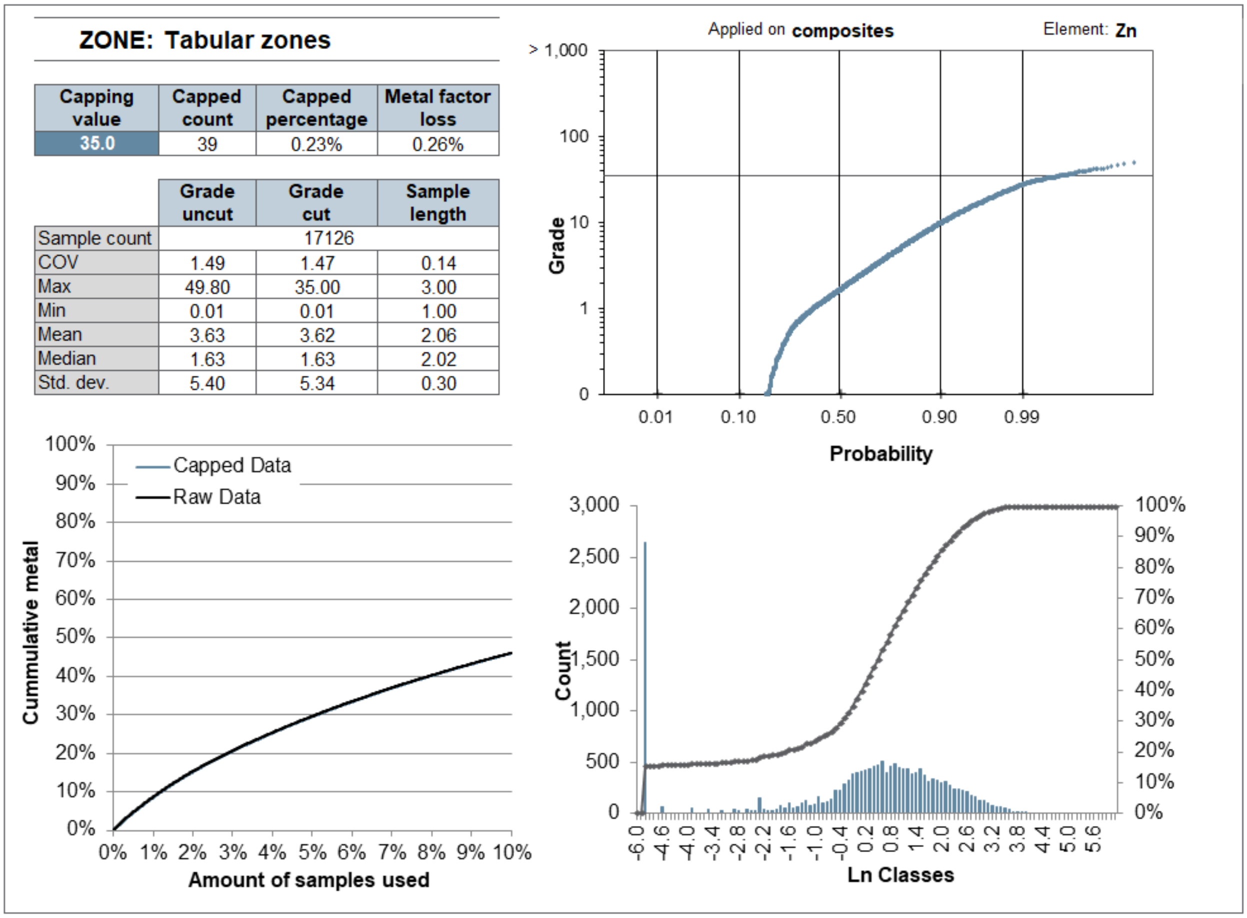


ZONE: Prismatic zones

Capping value	Capped count	Capped percentage	Metal factor loss
35.0	57	0.79%	0.87%

	Grade uncut	Grade cut	Sample length
Sample count		7173	
COV	1.31	1.27	0.08
Max	84.00	35.00	3.00
Min	0.01	0.01	1.35
Mean	5.39	5.35	2.01
Median	2.64	2.64	2.00
Std. dev.	7.04	6.77	0.17

Figure 14-10: Graphs supporting capping on composites for the prismatic zones for Zn

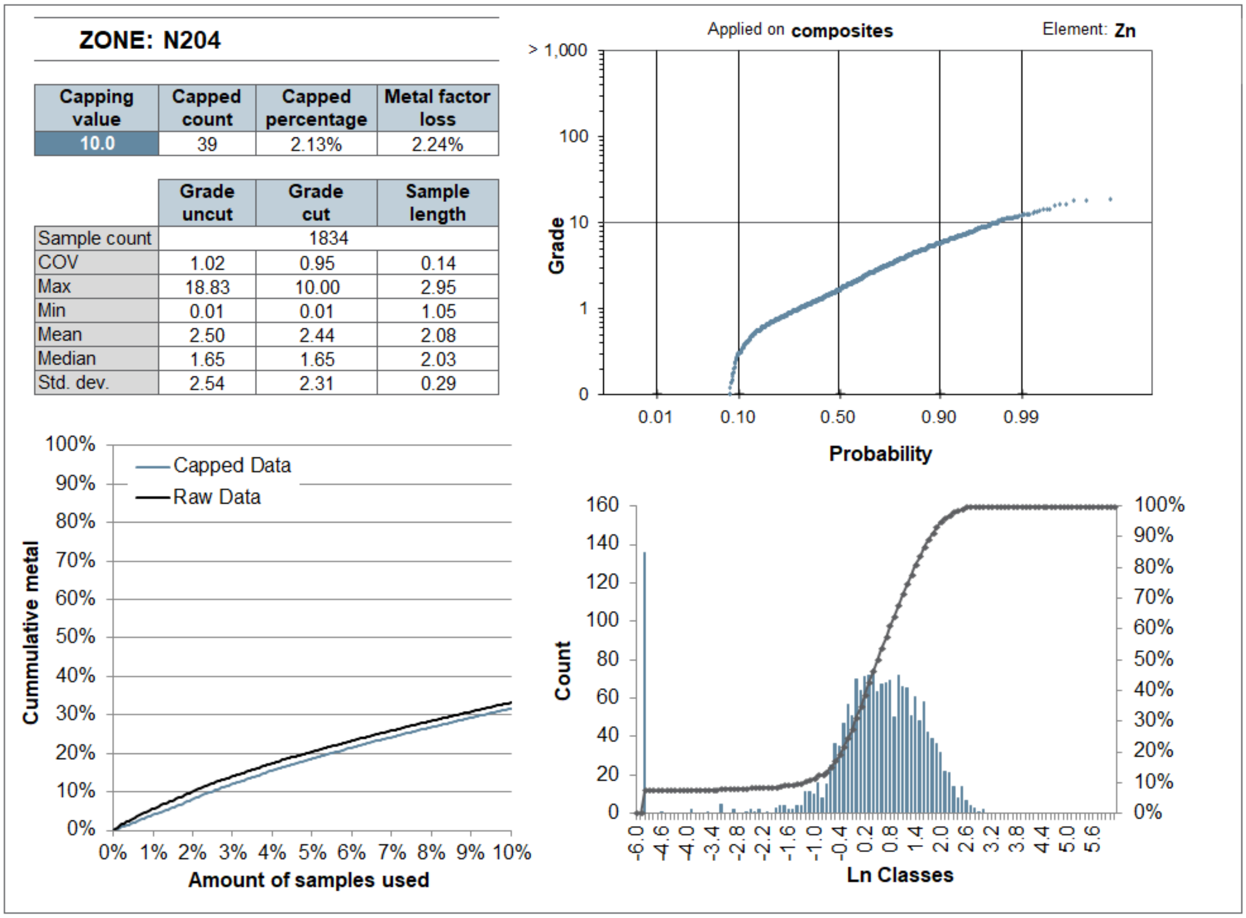


ZONE: Tabular zones

Capping value	Capped count	Capped percentage	Metal factor loss
35.0	39	0.23%	0.26%

	Grade uncut	Grade cut	Sample length
Sample count		17126	
COV	1.49	1.47	0.14
Max	49.80	35.00	3.00
Min	0.01	0.01	1.00
Mean	3.63	3.62	2.06
Median	1.63	1.63	2.02
Std. dev.	5.40	5.34	0.30

Figure 14-11: Graphs supporting capping on composites for the tabular zones for Zn



Capping value	Capped count	Capped percentage	Metal factor loss
10.0	39	2.13%	2.24%

	Grade uncut	Grade cut	Sample length
Sample count	1834		
COV	1.02	0.95	0.14
Max	18.83	10.00	2.95
Min	0.01	0.01	1.05
Mean	2.50	2.44	2.08
Median	1.65	1.65	2.03
Std. dev.	2.54	2.31	0.29

Figure 14-12: Graphs supporting capping on composites for the N-204 zones for Zn

Table 14-1: Basic statistics on composites and high-grade capping value for each type of zone

		Composites													
	Zones	Composites count	Uncut mean (%)	Std. dev.	COV	Max (%)	Uncut median (%)	Capping value	Number capped	Capped (%)	Metal loss (%)	Capped mean (%)	Capped std. dev.	Capped COV	Capped median (%)
Pb	Prismatic	7,173	3.14	6.02	1.92	72.97	1.00	40	34	0.47	1.86	3.08	5.54	1.80	1.00
	Tabular	17,126	1.14	2.69	2.37	57.23	0.30	30	17	0.10	0.64	1.13	2.60	2.30	0.30
	N-204	1,834	0.68	0.79	1.16	7.89	0.40	5	4	0.22	0.52	0.68	0.76	1.13	0.40
Zn	Prismatic	7,173	5.39	7.04	1.31	84.00	2.64	35	57	0.79	0.87	5.35	6.77	1.27	2.64
	Tabular	17,126	3.63	5.40	1.49	49.80	1.63	35	39	0.23	0.26	3.62	5.34	1.47	1.63
	N-204	1,834	2.50	2.54	1.02	18.83	1.65	10	39	2.13	2.24	2.44	2.31	0.95	1.65

14.6 Density

Density values were calculated based on the formula established and used by Cominco Ltd. during their operational period between 1964 and 1987:

$$P = \frac{1 - 0.05}{0.3509 - ((0.0025 \times \text{Pb}) + (0.0015 \times \text{Zn}) + (0.0033 \times \text{Fe}))}$$

Density values were calculated from the density of dolomite (host rock), adjusted by the amount of sphalerite, galena, and marcasite/pyrite as determined by metal assays. A porosity of 5% was assumed, and is taken into consideration in the formula. Waste material was assigned the density of porous dolomite (2.71 g/cm^3).

Validations were performed to gain confidence that the Cominco Ltd. formula presented above can be used for the purpose of the current MRE.

14.7 Contact Plot

A significant portion of the drillhole database is historical from the Cominco Ltd. period. As part of the historical data validation process, and in order to make sure there were no biases between recent and historical data, contact plots were generated comparing both populations.

Contact plots compare the nature of grade between two domains: they graphically display average grades of all pairs of data from both populations at increasing distances. Commonly used to determine if a hard or a soft interpolation boundary is justified, it can also be used to compare different populations within a mineralized zone. If there is a significant difference in grade across a domain boundary or different datasets (i.e. RC versus DDH, historical holes versus recent holes, etc.), the resource geologist must figure out a way to take that into consideration in the model, and in some cases discard one of the populations. Conversely, if a more gradual change in grade occurs across the boundary, the two datasets can be used as if they were from a single dataset.

Despite a disproportionate amount of samples in both populations (756 recent and 22,071 historical), the distributions shown in Figure 14-13 and Figure 14-14 demonstrate that both populations are similar in nature and that no bias is believed to exist. Such graphs should improve with additional data being added on the more recent dataset side. The slight surge in grade at the 0-5 m threshold can be explained by the twin holes that were recently conducted in an attempt to validate some of the high-grade historical holes. As for the more important surge at the 30-35 m threshold on the historical side of the graphs, they are considered artifacts with no significance due, in part, to the significant distance involved between compared data and the low amount of data being compared at this threshold.

Based on these graphs, the general overview of the Project, is that the significant historical production from Cominco Ltd., and resampling from historical core during this mandate, it is the QP's opinion that no bias exists between historical and recent holes and therefore both datasets can be used for the mineral resource estimate.

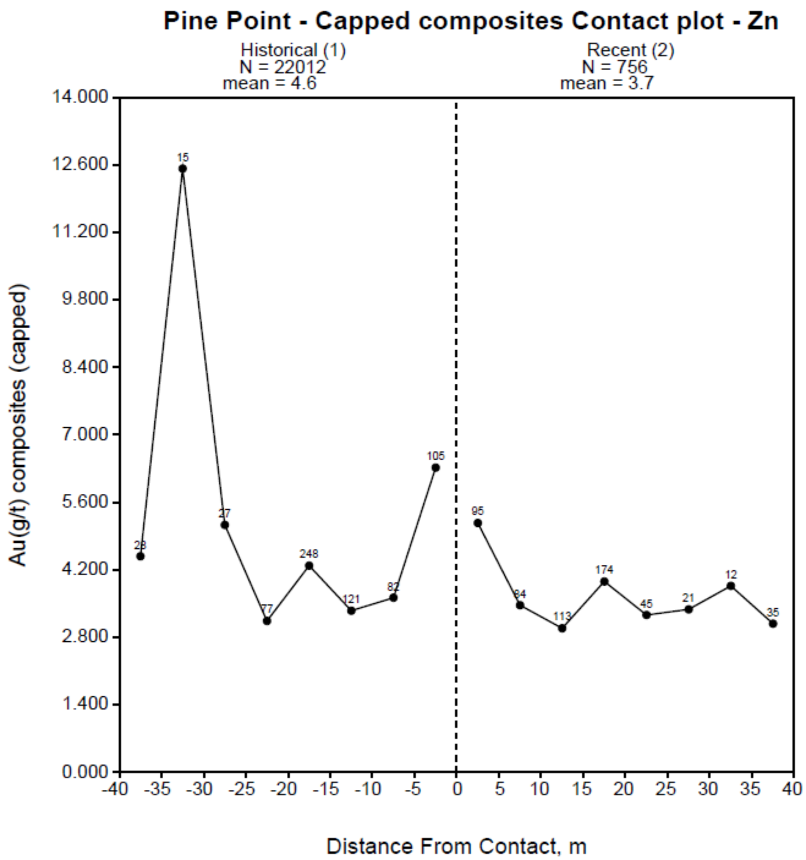


Figure 14-13: Contact analysis on the Zn capped composites between the historical and the recent drillholes

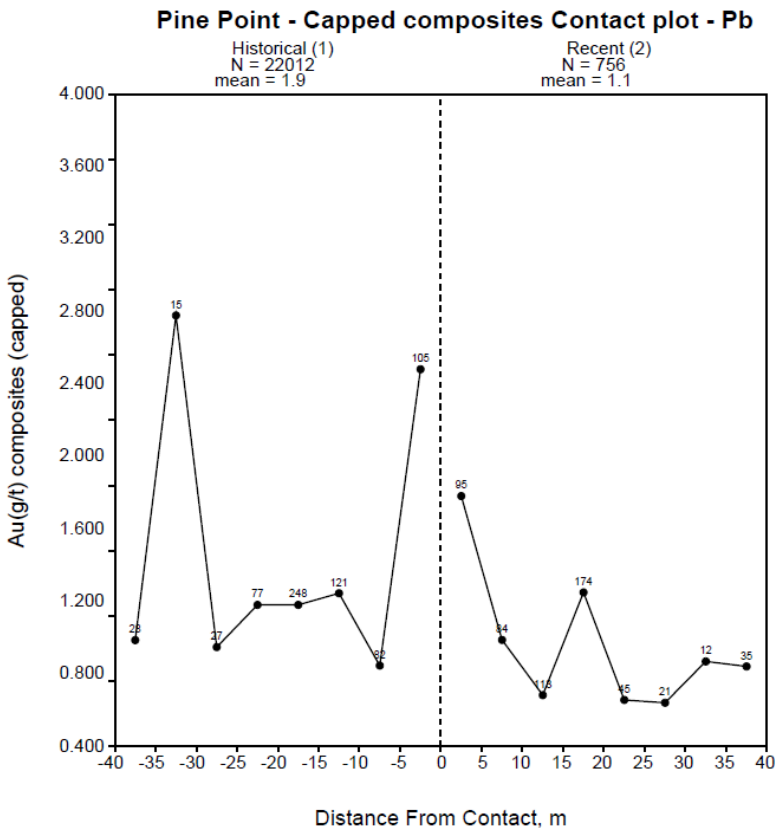


Figure 14-14: Contact analysis on the Pb capped composites between the historical and the recent drillholes

14.8 Variogram Analysis and Search Ellipsoids

A semi-variogram is a common tool used to measure the spatial variability within a zone. Typically, samples taken far apart will vary more than samples taken close to each other. A variogram gives a measure of how much two samples taken from the same mineralized zone will vary in grade depending on the distance between those samples, and therefore allowing to build search ellipsoids to be used during interpolation.

Three dimensional directional variography was carried out on the composites using the Snowden Supervisor v8.9 software. Variograms were modelled in the three orthogonal directions to define a 3D ellipsoid for each group of zone (tabular, prismatic and N-204) using the most representative zone of each group. The three directions of ellipsoid axes were set by using the variogram fans and visually confirmed with geological knowledge of the deposit.

Then, a mathematical model was interpreted in order to best-fit the shape of the calculated variogram for each direction. Three components were defined for the mathematical model: the nugget effect, the sill, and the range.

Table 14-2 presents the chosen variogram model parameters for each zone and Figure 14-15 illustrates an example of the variography results.

Table 14-2: Variogram model parameters for each mineralized zone

				First structure				Second structure			
	Zones	Blockcode	Nugget	Sill	Range X (m)	Range Y (m)	Range Z (m)	Sill	Range X (m)	Range Y (m)	Range Z (m)
Pb	N-204	2001	0.2	0.67	96	35	8	0.13	114	55	10
	Prismatic	1004	0.23	0.69	45	52	25	0.08	74	61	40
	Tabular	2002	0.44	0.38	34	22	2	0.18	103	80	5
Zn	N-204	2001	0.1	0.76	119	35	8	0.14	141	55	10
	Prismatic	1004	0.07	0.83	36	56	20	0.09	68	61	40
	Tabular	2002	0.46	0.37	49	32	2	0.17	103	63	5

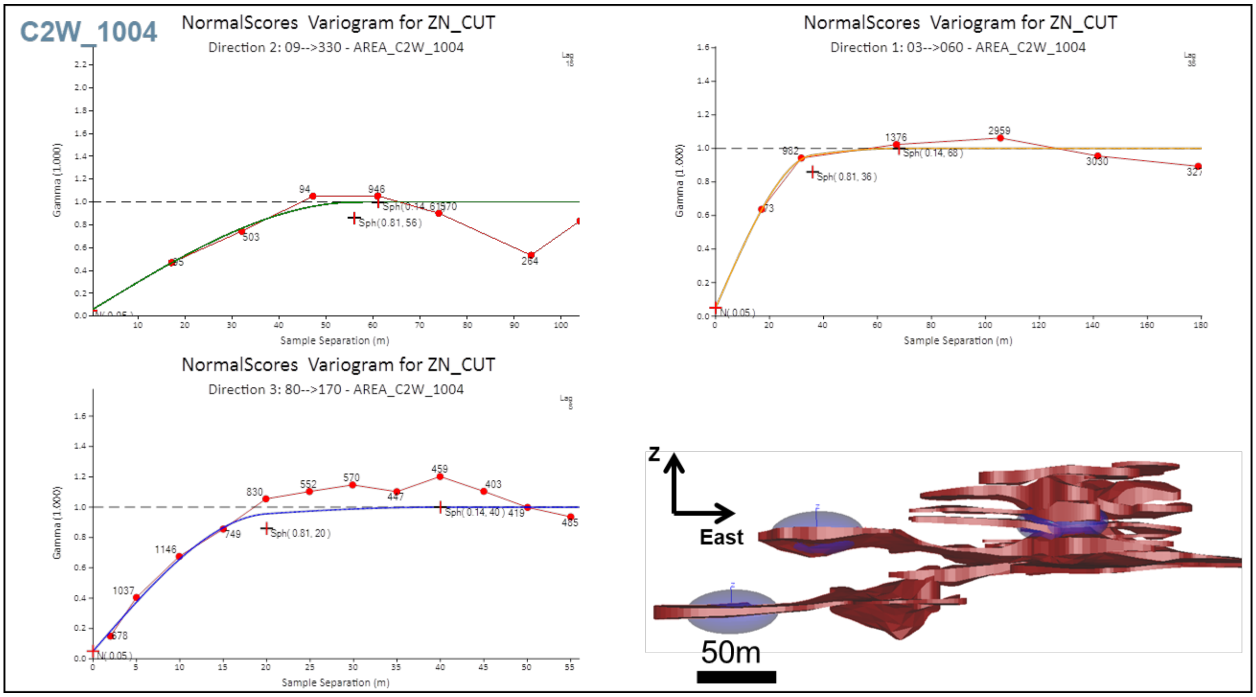


Figure 14-15: Example of the variography study for the C2W_1004 (prismatic zone)

14.9 Block Model

Due to the significant area covered by the Pine Point property, ten block models were created to facilitate the technical execution of the MRE. It should be noted that two block models (N-99 and K-77) were discarded from the MRE statement. The N-99 deposit was discarded due to the fact that the deposit was too close to the Buffalo River, and K-77 because of its lower grade not yielding a reasonable prospect of being economically extracted. Deposits G-03 and H-02, that are part of the NTW block model, were also discarded for being too close to the Buffalo River.

The block models were constructed in Geovia GEMS™ for the current mineral resource estimate using the block model parameters provided in Table 14-3. Individual block cells have dimensions of 10 m long (X-axis) by 10 m wide (Y-axis) by 5 m vertical (Z-axis).

Table 14-3: Pine Point block models parameters

		Origin coordinates	Number of blocks	Block extent (m)
C1	X (column)	639,572.25	822	8,220
	Y (row)	6,748,515.17	222	2,220
	Z (level)	235.00	20	100
C2	X (column)	627,238.394	609	6,090
	Y (row)	6,742,506.817	213	2,130
	Z (level)	250.00	26	130
R67	X (column)	627,984.268	59	590
	Y (row)	6,740,335.428	60	600
	Z (level)	260.00	30	150
C2W	X (column)	600,142.428	1,349	13,490
	Y (row)	6,731,882.328	290	2,900
	Z (level)	220.00	47	235
MID	X (column)	632,775.461	794	7,940
	Y (row)	6,744,714.512	296	2,960
	Z (level)	260.00	25	125
N-204	X (column)	658,240.753	300	3,000
	Y (row)	6,758,547.245	248	2,480
	Z (level)	210.00	20	100
NT	X (column)	624,719.191	1,375	13,750
	Y (row)	6,747,065.577	225	2,250
	Z (level)	220.00	30	150

		Origin coordinates	Number of blocks	Block extent (m)
NTW	X (column)	617,451.555	370	3,700
	Y (row)	6,744,857.237	90	900
	Z (level)	215.00	33	165

The block models were rotated 25° counter-clockwise (X-axis oriented along N65°) to honour the orientation of most of the mineralized zones and were coded using the percent model method typical of Geovia GEMS™, reflecting the proportion of each solid inside every block. All blocks falling within a solid were assigned the corresponding solid block code.

14.10 Search Ellipsoid Strategy

The ranges of the ellipsoids used for the interpolation were established using the variography study and correspond to the range of the first structure for the first pass, to the second structure for the second pass. For prismatic zones, a third pass was built using the ranges from the second pass of tabular zones in order to adequately populate lateral branches that are similar in orientation and composition to the tabular zones.

It is noteworthy to mention at this point that the classification was mostly based on drillhole spacing and, therefore, some interpolated blocks were not converted into the Inferred classification. Please refer to the Resource Classification section further below for more details.

Table 14-4 presents the orientation and ranges of the search ellipsoids for each pass.

Table 14-4: Search ellipsoid ranges by interpolation passes for the mineralized zones

						Pass 1			Pass 2			Pass 3		
			GEMS orientation			Search ellipsoid ranges			Search ellipsoid ranges			Search ellipsoid ranges		
	Zones	Blockcode	Azimut	Dip	Azimut	X (m)	Y (m)	Z (m)	X (m)	Y (m)	Z (m)	X (m)	Y (m)	Z (m)
Pb	N-204	2001	60	3	330	96	35	8	114	55	10	-	-	-
	Prismatic	1004	330	-1	60	45	52	25	74	61	40	103	80	10
	Tabular	2002	70	-1	160	34	22	2	103	80	10	-	-	-
Zn	N-204	2001	60	3	330	119	35	8	141	55	10	-	-	-
	Prismatic	1004	330	-1	60	36	56	20	68	61	40	103	63	10
	Tabular	2002	70	-1	160	49	32	2	103	63	10	-	-	-

14.11 Interpolation Parameters

A kriging neighbourhood analysis ("KNA") was conducted on the most representative zone of each group with the Snowden Supervisor software. KNA provides a quantitative method of testing different estimation parameters (i.e. block size, discretization and min/max of composites used for the interpolation) by evaluating their impact on the quality of the results. The interpretation of these helps select the optimal value for each parameter.

Following this study, the parameters provided in Table 14-5 were chosen for the interpolation of the Pine Point block model.

Table 14-5: Interpolation parameters

	Zn			Pb		
Interpolation parameters	**Pass 1**	**Pass 2**	**Pass 3**	**Pass 1**	**Pass 2**	**Pass 3**
Minimum number of composites used	4	4	4	4	4	4
Maximum number of composites per drillhole used	16	16	16	12	12	12
Maximum number of composites used	3	3	3	3	3	3
Minimum number of drillhole used	2	2	2	2	2	2

14.12 Interpolation Method

The interpolation was run on a set of points extracted from the capped composited data. The block model grades were estimated using ordinary kriging ("OK") methods. Hard boundaries between the mineralized zones were used in order to prevent grades from adjacent zones being used during interpolation. As a block was estimated, it was tagged with the corresponding pass number.

For comparison purposes, additional grade models were generated using 1) inverse distance squared ("ID^2"); 2) nearest neighbour ("NN"); and 3) OK on uncapped composited data.

14.13 Block Model Validation

The Pine Point block models were validated using several methods including a visual review of the grades in relation to the underlying drillhole and statistical methods.

14.13.1 Visual Validation

Block model grades were visually compared against drillhole composite grades and raw assays in cross-section, plan, longitudinal, and 3D views. This visual validation process also included confirming that the proper coding was done within the various domains. The visual comparison shows a good correlation between the values without excessive smoothing. Visual comparisons were also conducted between ID^2, OK and NN interpolation scenarios. The OK scenario used for the resource estimate produced a grade distribution honouring drillhole data and the style of mineralization observed at Pine Point.

14.13.2 Statistical Validation

Grade averages for the OK and the ID^2 models were tabulated in Table 14-6. This comparison did not identify significant issues. As expected, block grade averages are generally lower than the composite grades.

Table 14-6: Comparison of the block and composite mean grades at a zero cut-off grade for Inferred blocks (blocks > 50% inside a mineralized zone)

			Zn			Pb		
Sector	**Number of composites**	**Number of blocks**	**Composite grade (%)**	**OK grade model (%)**	**ID^2 grade model (%)**	**Composite grade (%)**	**OK grade model (%)**	**ID^2 grade model (%)**
C1	5,709	13,973	2.60	1.82	1.84	0.82	0.60	0.60
C2	1,938	9,372	3.52	3.17	3.22	1.16	0.88	0.90
C2W	3,782	10,012	5.27	4.55	4.70	2.69	2.44	2.52
MID	2,615	4,030	4.78	3.22	3.20	3.28	0.79	0.77
N-204	1,834	12,798	2.49	2.54	2.58	0.68	0.70	0.71
NT	8,511	13,744	4.45	3.01	3.10	1.35	1.10	1.12
NTW	1,744	3,249	5.03	3.89	3.98	2.66	1.75	1.78
Total	**26,133**	**67,178**	**4.03**	**2.98**	**3.04**	**1.65**	**1.10**	**1.12**

Swath plots were also generated as part of the block model validation. A swath plot is a graphical display of the grade distribution derived from a series of bands (or swaths), generated in several directions throughout the deposit. Using the swath plots, grade variations from the OK model are compared to the distribution of grade interpolated with the NN and ID^2 methods and to the composite grades (Figure 14-16 and Figure 14-17).

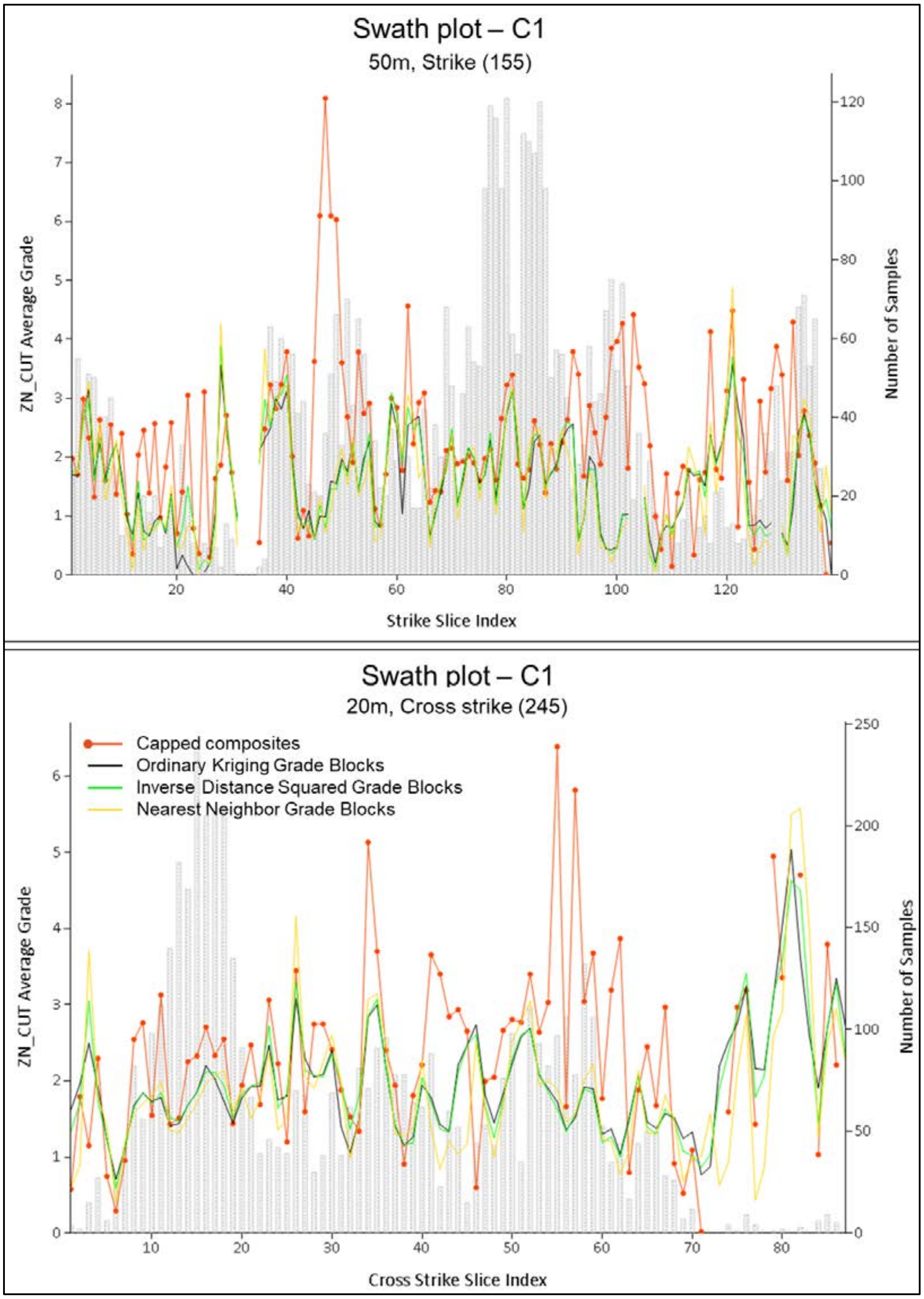


Figure 14-16: Pine Point model validation swath plots along strike (N155, top) and across strike (N245, bottom) for Zn

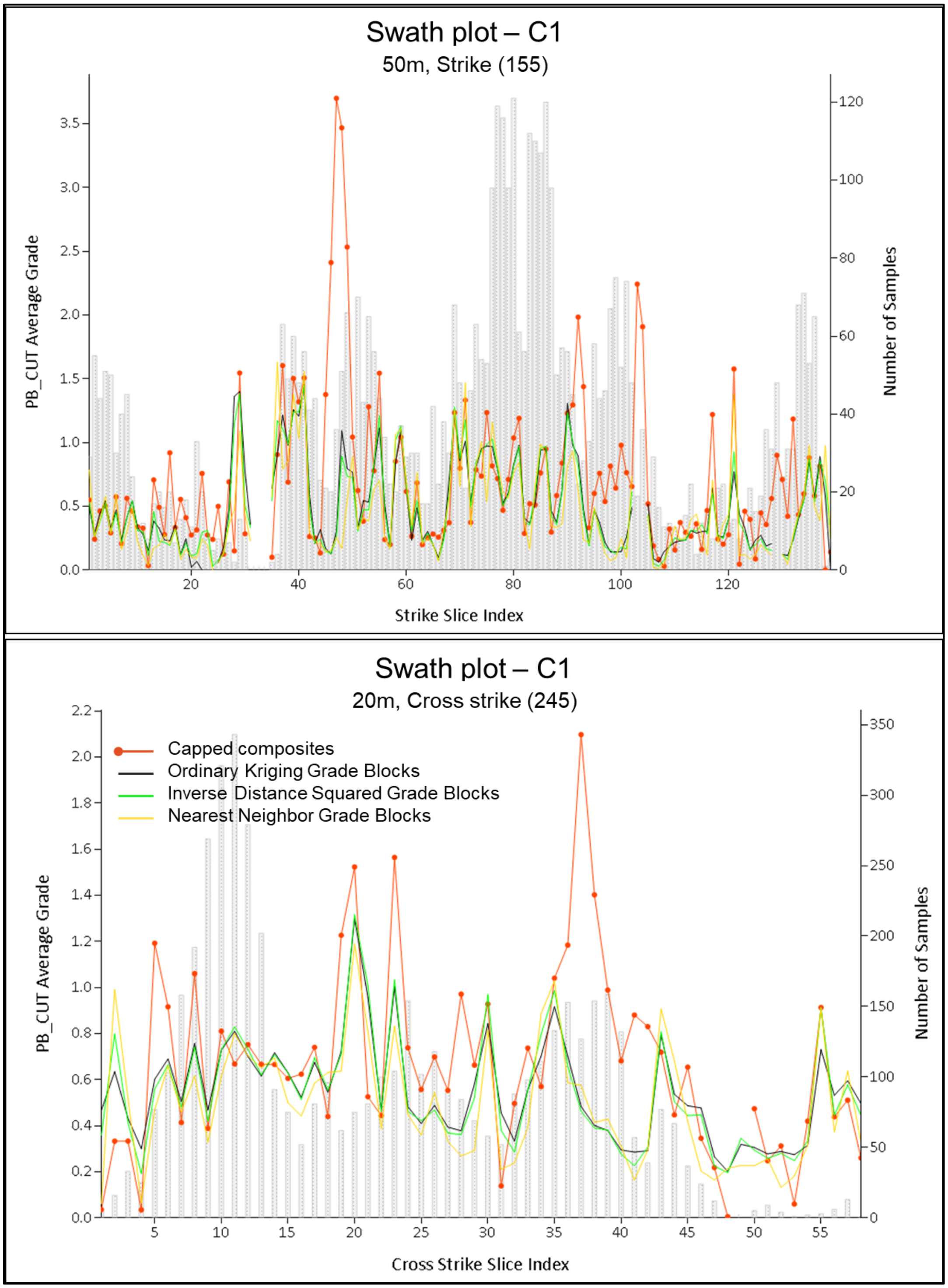


Figure 14-17: Pine Point model validation swath plots along strike (N155, top) and across strike (N245, bottom) for Pb

Based on visual and statistical reviews, it is the QP's opinion that the Pine Point block model provides a reasonable estimate of in situ Pb-Zn resources.

14.14 Resource Classification

The mineral resources for the Pine Point project were classified in accordance with CIM Standards.

14.14.1 Mineral Resource Definition

The "CIM Definition Standards for Mineral Resources and Reserves" published by the Canadian Institute of Mining, Metallurgy and Petroleum for the resource classification clarifies the following:

"Inferred Mineral Resource:

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

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

Indicated Mineral Resource:

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

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

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

Measured Mineral Resource:

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

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

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

14.14.2 Mineral Resource Classification for the Pine Point Project

The estimated block grades were classified into Inferred Mineral Resource category using drill spacing and geological continuity of mineralization. No indicated or measured resources were defined for the Project at this stage.

Inferred Mineral Resources were defined for blocks within the mineralized zones that have been informed by a minimum of two drillholes within 50 m of a drillhole (100 m of drill spacing).

When needed, a series of clipping boundaries were created manually in longitudinal views to either upgrade or downgrade classification in order to avoid artifacts due to automatically generated classification. All remaining estimated but unclassified blocks were flagged as "Exploration Potential".

14.15 Cut-off Grade and Pit Optimization

According to CIM's Definition Standards, in order for a deposit to be considered a Mineral Resource it must be proven that there are "reasonable prospects for eventual economic extraction". This requirement implies that the quantity and grade estimates meet certain economic thresholds and that the Mineral Resources are reported at an appropriate cut-off grade that takes into account extraction scenarios and processing recoveries. In order to determine the quantity of mineralization that shows a "reasonable prospect for eventual economic extraction" using open pits mining methods, BBA carried out a pit optimization analysis using Hexagon MineSight's Economic Planner. This analysis evaluates the profitability of each mineralized block in the model based on its value. The pit optimization parameters that are presented in Table 14-7 are based on discussions with Pine Point Mining Limited and benchmarking against similar projects.

It is important to note that the results from the pit optimization exercise are used solely for testing the “reasonable prospects for eventual economic extraction” by open pit mining methods and do not represent an economic study.

The cut-off grade used for the Mineral Resource Estimate ranges from 1.70% to 2.00% equivalent zinc (Table 14-8). The reason for the cut-off grade variation is due to the fact that different haulage costs are applied depending on the distance of the deposit to the assumed mill site (Figure 14-18). It should be noted that no mill is currently present on the Property, and that mill distance were estimated based on the most likely location where a mill could potentially be built if the project moves forward. Additionally, different mine dewatering costs were used for several of the deposits and lower mill recoveries were used for the N-204 deposit.

The pit optimization analysis was carried out using overall pit slopes of 50 degrees and mining dilution and material losses were not considered. The pit shells that were selected for the Mineral Resource Estimate were those that were run at a Revenue Factor (“RF”) equal to 1.1. Very small open pits that resulted from this analysis as well as small areas that were adjacent to existing historical open pits were removed from the Mineral Resource Estimate since they were deemed not practical to be mined at this time. This reduction amounts to 2.8% of the total tonnage of Mineral Resources.

The cut-off grade and pit optimization analysis resulted in a Mineral Resource Estimate that incorporates 52 new open pits and expansions of five historical open pits (Figure 14-25).

Figure 14-24 shows a 3D view from the East Mill Area.

Table 14-7: Cut-off grade and pit optimization parameters

Parameter	Unit	Input
Mine Site Costs		
Mining Cost – Mineralized Material	$/t mined	4.00
Mining Cost – Waste	$/t mined	4.00
Mine Dewatering Cost(1)	$/t mined	0.88
Pre-concentration Cost	$/t mineralized material	3.50
Processing Cost(2)	$/t milled	22.50
General & Administration Cost(2)	$/t milled	33.60
Recoveries		
Overall Lead(3)	%	87.8
Overall Zinc(3)	%	83.1
Pre-concentration Mass Pull	%	37.3
Zinc Concentrate Grade	%	55.0
Lead Concentrate Grade	%	55.0

Parameter	Unit	Input
Payables		
Zinc	%	85.0
Lead	%	96.0
Zinc Concentrate Costs		
Transport to Rail	$/dmt	27.00
Transport to Smelter	$/wmt	178.00
Smelter Cost	$/dmt	295.00
Lead Concentrate Costs		
Transport to Rail	$/dmt	27.00
Transport to Smelter	$/wmt	221.00
Smelter Cost	$/dmt	262.00
Metal Prices		
Zinc	USD/lb	1.10
Lead	USD/lb	0.90
Exchange Rate (CAD:USD)		1.31
Royalties (3%)	% NSR	3.0

(1) Applied to both mineralized material and waste tonnages.

(2) Costs per tonne milled are based on a 37.27% Pre-concentration Mass Pull.

(3) Inclusive of sorting test program results.

Table 14-8: Cut-off grade per area

Area	ZnEq (%)
Central Zone	1.70
East Mill Zone	1.70
North Zone	1.70
West Zone	1.90
N-204 Zone	2.00

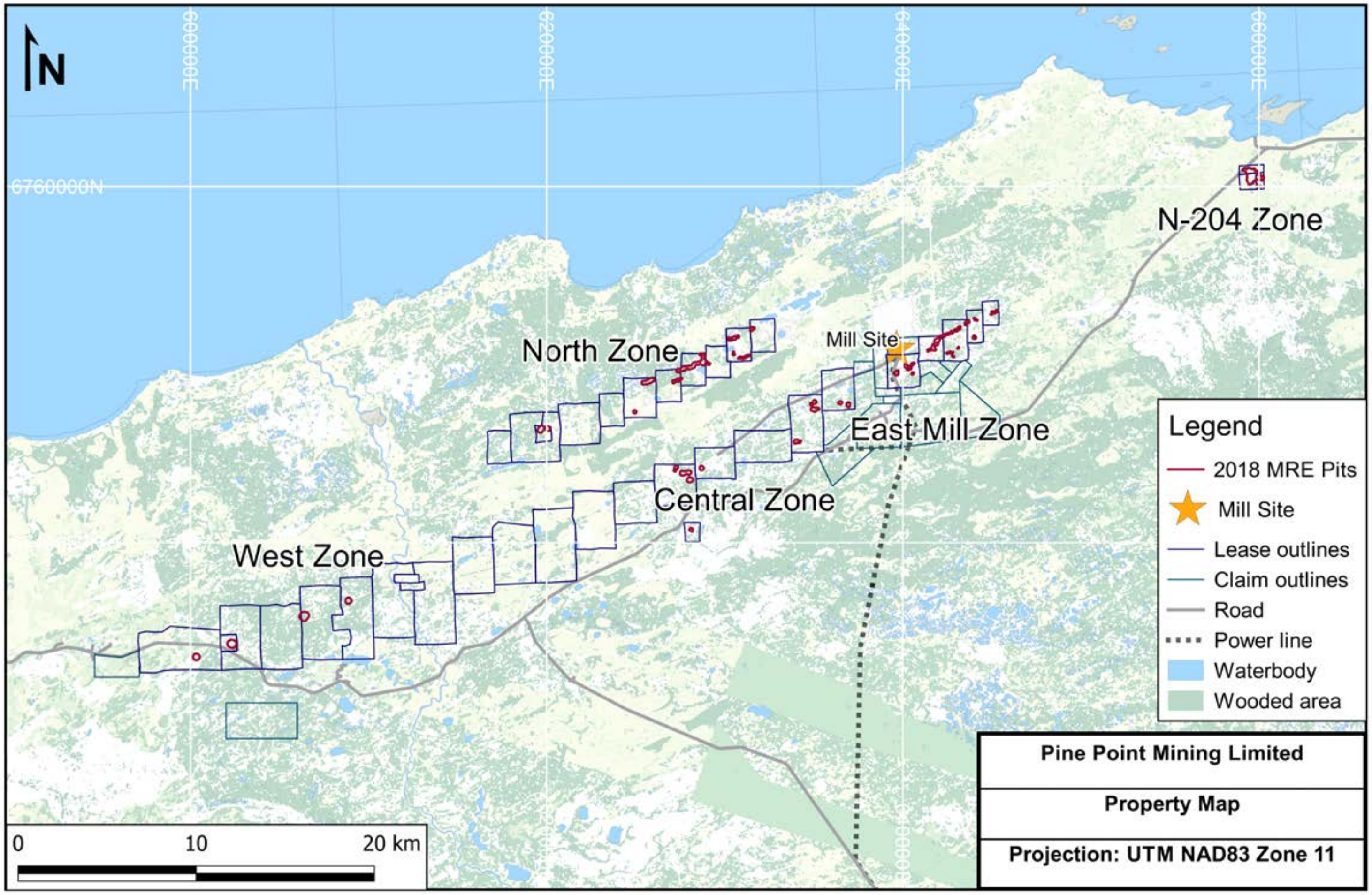


Figure 14-18: General view of the Property showing the location of the different zones and the historical mill, also the assumed location of a future mill. It should be noted that no mill is currently present on the Property, and that the most likely location where a mill could potentially be built if the project moves forward was established based on the distribution of the deposits and the historical location of the mill.

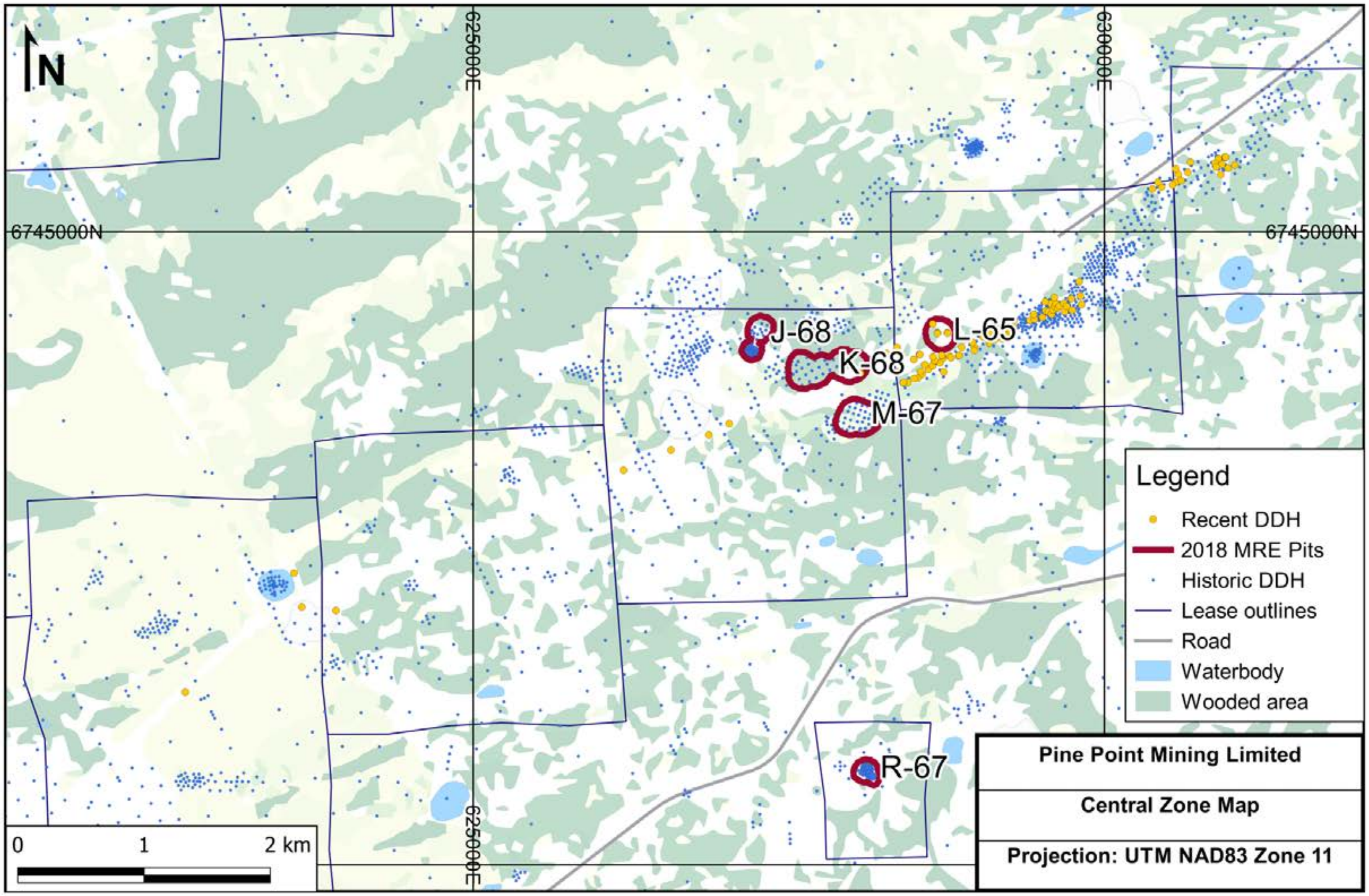


Figure 14-19: Central Zone Map showing the location of drillholes and pits from the current MRE.

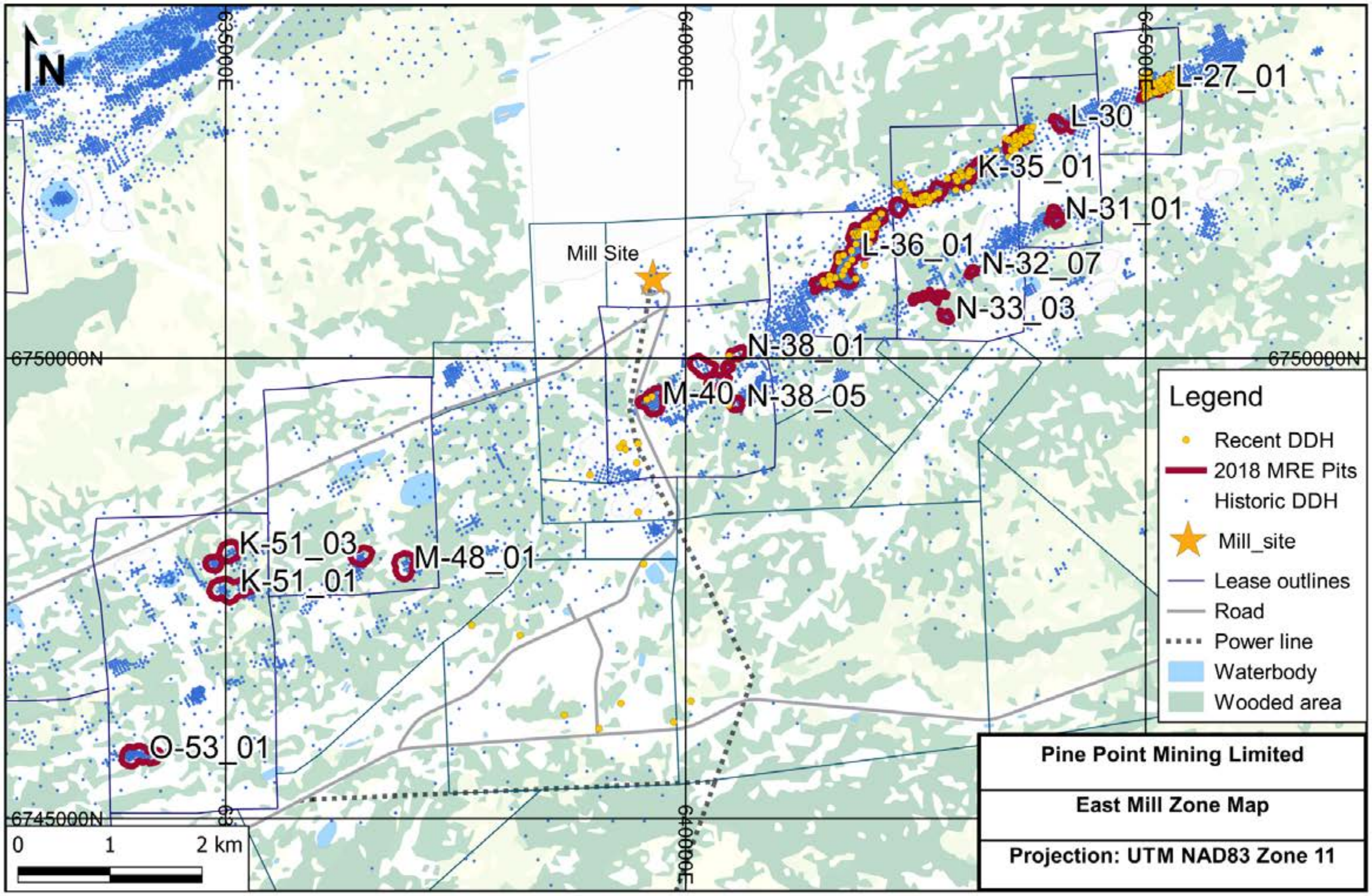


Figure 14-20: East Mill Zone Map showing the location of drillholes and pits from the current MRE and the historical mill, also the assumed location of a future mill.
It should be noted that no mill is currently present on the Property, and that the most likely location where a mill could potentially be built if the project moves forward was established based on the distribution of the deposits and the historical location of the mill.

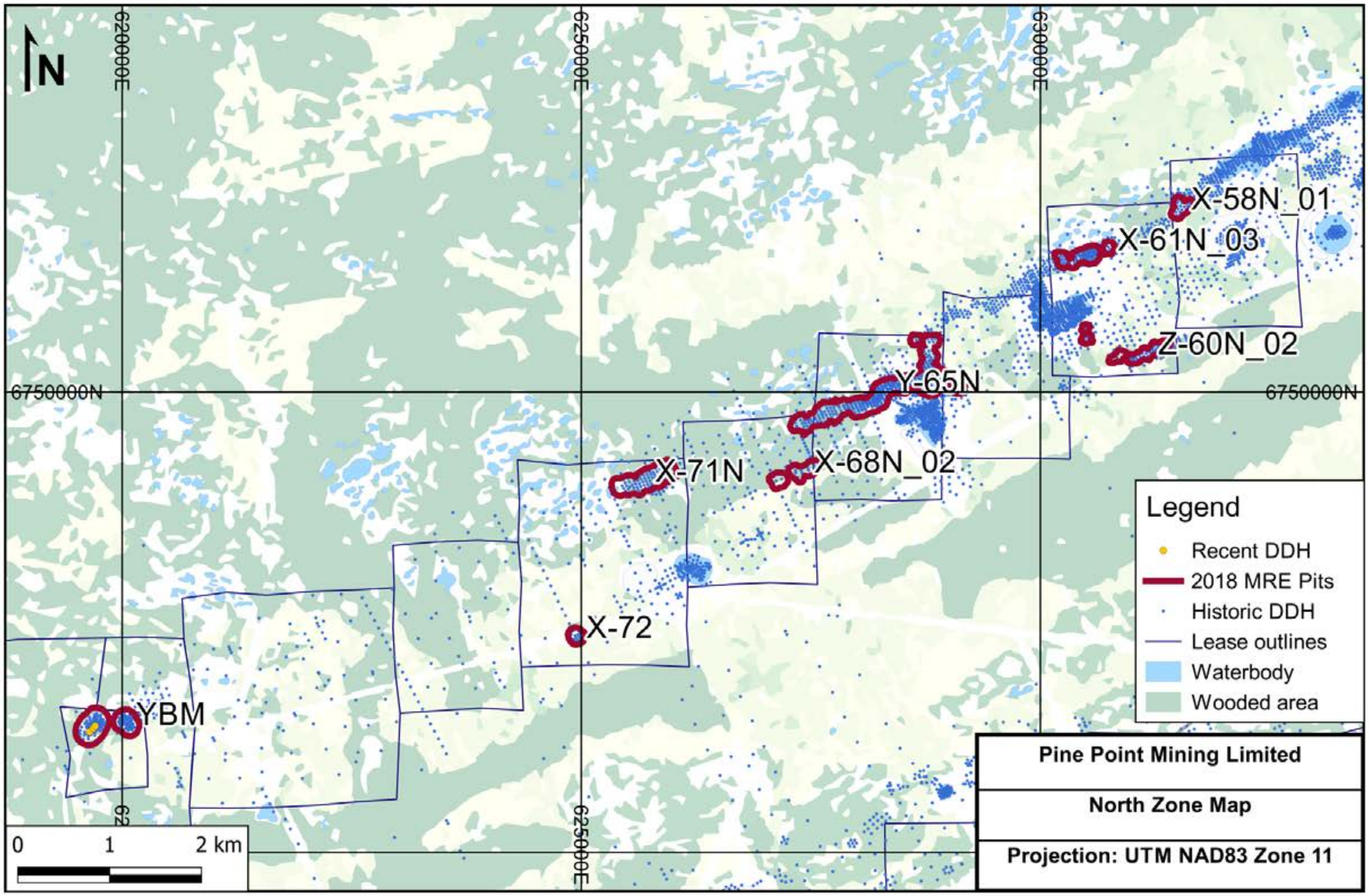


Figure 14-21: North Zone Map showing the location of drillholes and pits from the current MRE

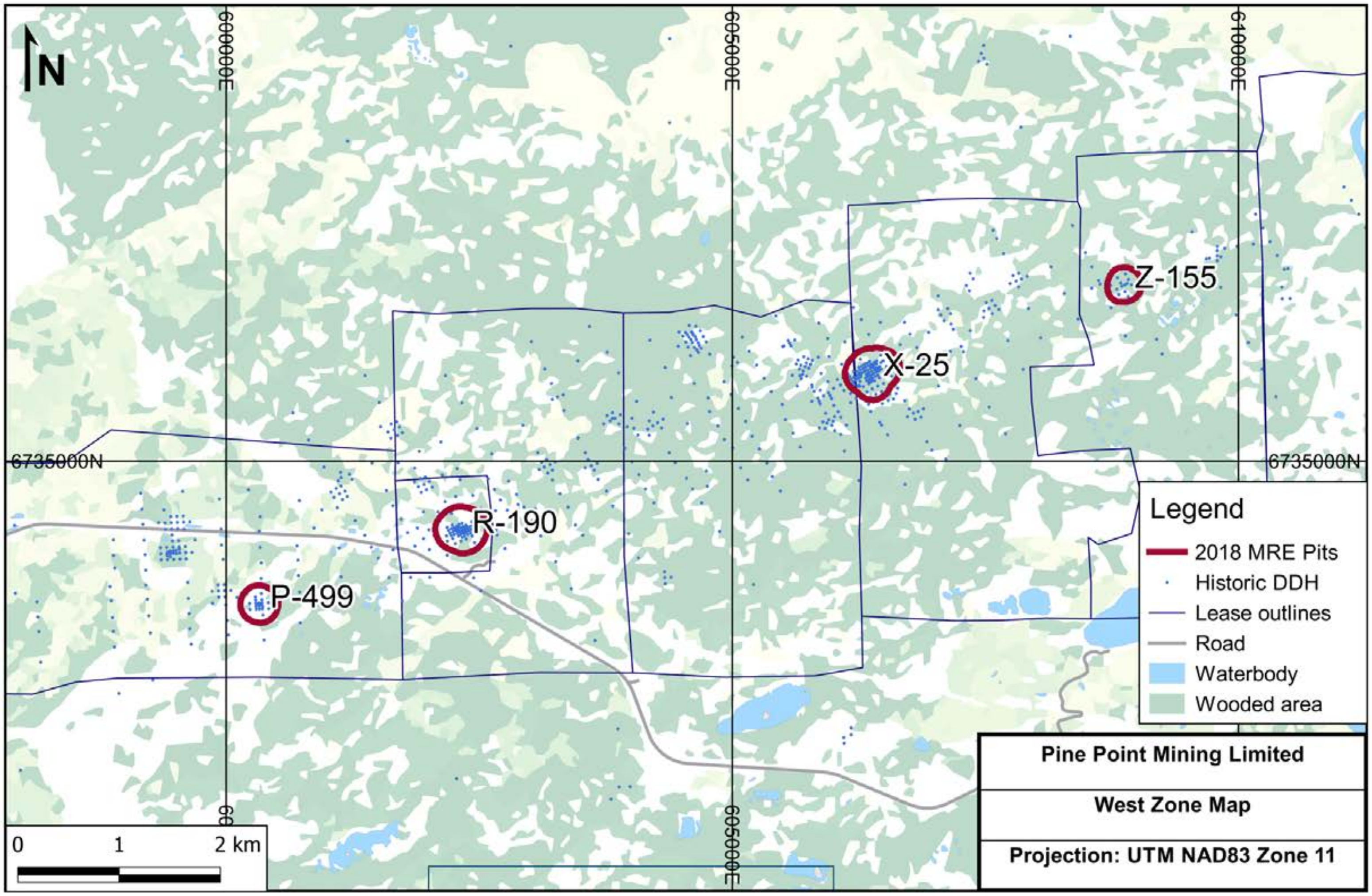


Figure 14-22: West Zone Map showing the location of drillholes and pits from the current MRE

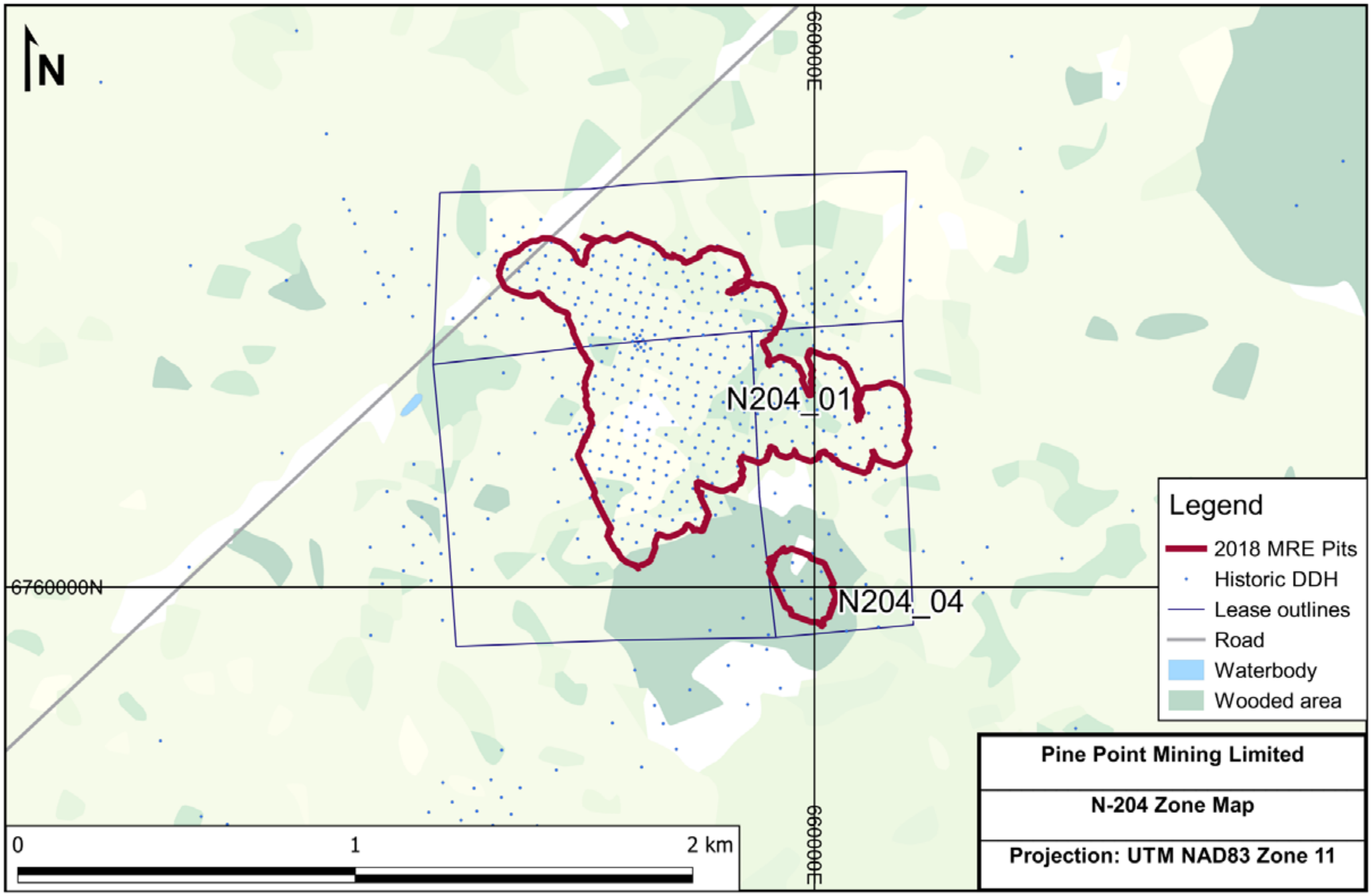


Figure 14-23: N-204 Zone Map showing the location of drillholes and pits from the current MRE

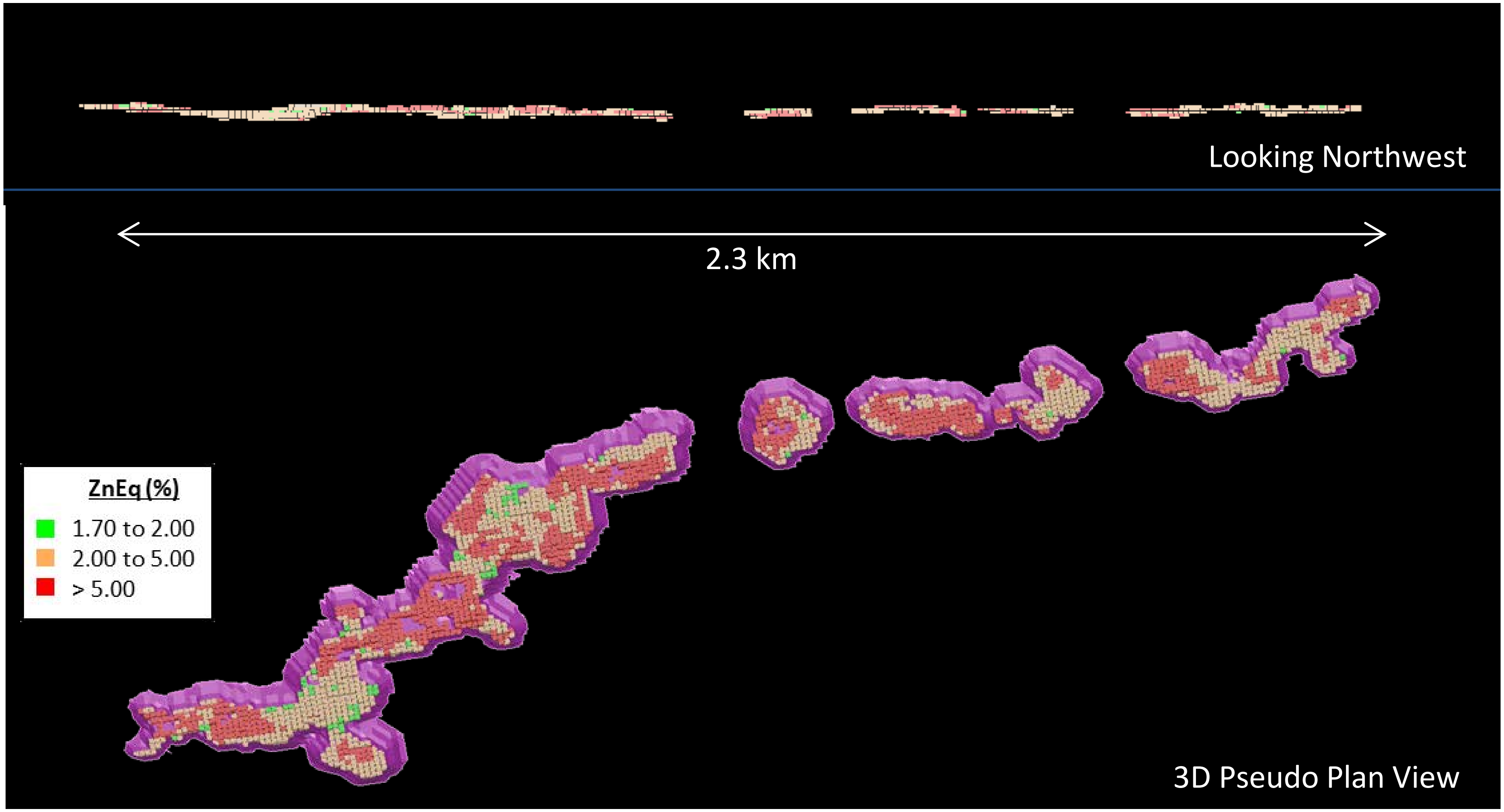


Figure 14-24: 3D view from the East Mill Area. Blocks shown are 1) within pit optimized shells; 2) above the cut-off grade; and 3) classified as Inferred

14.16 Pine Point Mineral Resource Estimate

The Inferred Mineral Resource Estimate presented herein is constrained within pit shells developed from the pit optimization analysis discussed above. All deposits for which no pit was generated (because they were not economic using the parameters presented in the previous section) were removed from the resource estimate.

Table 14-9: Pit-constrained Inferred Mineral Resource Estimate

Area	Tonnage (Mt)	ZnEq (%)	Zn (%)	Pb (%)	Strip Ratio
Central Zone	4.80	7.69	5.84	1.72	11.70
East Mill Zone	5.50	5.16	3.76	1.30	5.70
North Zone	13.10	6.27	4.26	1.87	5.30
West Zone	6.40	10.09	6.30	3.53	14.50
N-204 Zone	8.60	4.74	3.61	1.02	5.40
Total	**38.40**	**6.58**	**4.58**	**1.85**	**7.70**

Notes to Table 14-9:

1. The independent qualified person for the 2018 MRE, as defined by NI 43-101 guidelines, is Pierre-Luc Richard, P. Geo., of BBA Inc. The effective date of the estimate is November 14, 2018.
2. These mineral resources are not mineral reserves as they do not have demonstrated economic viability. The quantity and grade of reported Inferred resources in this MRE are uncertain in nature and there has been insufficient exploration to define these Inferred resources as Indicated or Measured, however It is reasonably expected that the majority of Inferred Mineral Resources could be upgraded to Indicated Mineral Resources with continued exploration.
3. Resources are presented as undiluted and in situ for an open-pit scenario and are considered to have reasonable prospects for economic extraction. The constraining pit shells were developed using pit slopes of 50 degrees.
4. The MRE was prepared using GEOVIA GEMS 6.8.2 and is based on 18,542 surface drillholes, of which 6,880 intercepted mineralization, and a total of 31,120 assays. The drillhole database includes Osisko Metals infill drilling of 23,751 metres in 318 drillholes and also incorporates Cominco Ltd.'s historical drillholes, the use of which was validated by a drillhole collar survey and a partial core resampling program. The cut-off date for drillhole assays was September 12, 2018.
5. The estimate encompasses 243 zinc-lead-bearing zones each defined by individual wireframes with a minimum true thickness of 2.5 m. A value of zero grade was applied in cases of core not assayed.
6. High-grade capping was done on the composited assay data and established on a per zone basis for zinc and lead. Capping grades vary from 10% to 35% Zn and 5% to 40% Pb.
7. Density values were calculated based on the formula established and used by Cominco Ltd. during their operational period between 1964 and 1987. Density values were calculated from the density of dolomite, adjusted by the amount of sphalerite, galena, and marcasite/pyrite as determined by metal assays. A porosity of 5% was assumed. Waste material was assigned the density of porous dolomite.
8. Grade model resource estimation was calculated from drillhole data using an Ordinary Kriging interpolation method in a block model using blocks measuring 10 m x 10 m x 5 m (vertical) in size.
9. Zinc equivalency percentages are calculated using metal prices, forecasted metal recoveries, concentrate grades, transport costs, smelter payable metals and charges.

10. The estimate is reported using a Zn Equivalent ("ZnEq") cut-off varying from 1.70% to 2.00%. Variations take into consideration trucking distances from the open pits to the mill and metallurgical parameters for each area. The cut-off grade was calculated using the following parameters (amongst others): zinc price = USD1.10/lb; lead price = USD0.90/lb; CAD:USD exchange rate = 1.31. The cut-off grade will be re-evaluated in light of future prevailing market conditions and costs.
11. The MRE presented herein is categorized as an Inferred resource. The Inferred mineral resource category is only defined within the areas where drill spacing is less than 100 m and shows reasonable geological and grade continuity.
12. The pit optimization to develop the resource constraining pit shells was done using Hexagon's MineSight Version 15.10.
13. Calculations used metric units (metre, tonne). Metal contents are presented in percent or pounds. Metric tonnages were rounded and any discrepancies in total amounts are due to rounding errors.
14. CIM definitions and guidelines for Mineral Resource Estimates have been followed.
15. The author is not aware of any known environmental, permitting, legal, title-related, taxation, socio-political or marketing issues, or any other relevant issues not reported in this Technical Report, that could materially affect the Mineral Resource Estimate.

Table 14-10 and Figure 14-25 show the sensitivity of the block model estimate to grade cut-off for the in situ mineral resource estimate. Table 14-11 compares the results of the official statement to an ID^2 and an uncut scenario.

The reader is cautioned that the numbers presented in the following tables should not be misconstrued with a mineral resource statement.

Table 14-10: Pine Point project Inferred Mineral Resource cut-off grade sensitivity table

Area	Cut-off grade (%)	Tonnage (Mt)	Zn (%)	Pb (%)
ALL	10.00	5.15	10.66	5.34
	9.00	6.39	9.90	4.88
	8.00	8.18	9.07	4.38
	7.00	10.54	8.25	3.89
	6.00	13.87	7.43	3.38
	5.00	18.65	6.58	2.89
	4.00	25.16	5.77	2.43
	3.00	32.14	5.11	2.09
	2.00	37.63	4.64	1.88
	1.80	38.46	4.57	1.85
	1.60	39.19	4.51	1.82
	1.40	39.86	4.46	1.80
	1.20	40.44	4.41	1.78
	1.00	40.92	4.36	1.76

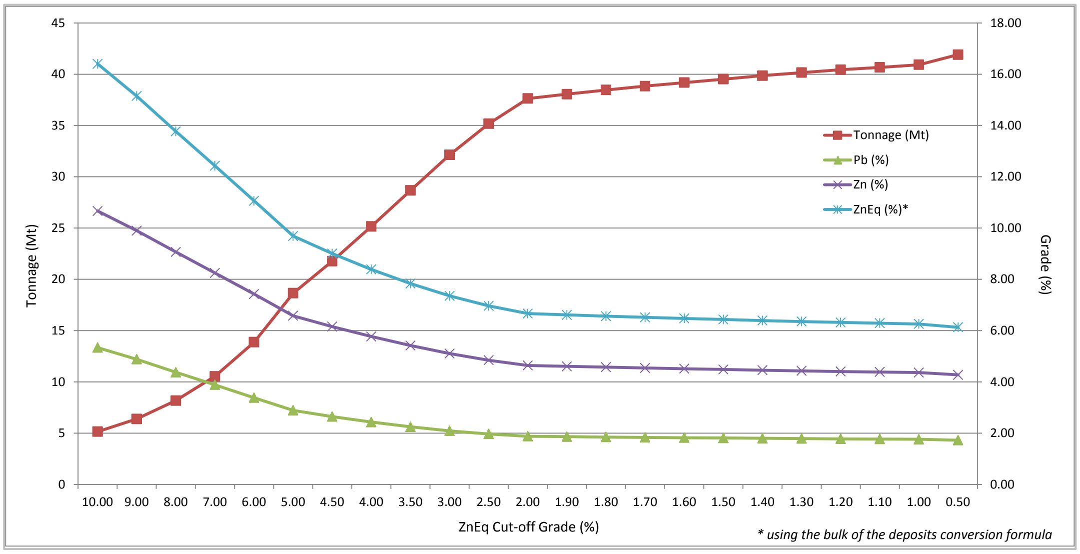


Figure 14-25: Pine Point project Inferred Mineral Resource cut-off grade sensitivity graph

Table 14-11: Pine Point project Inferred Mineral Resource interpolation method sensitivity table

Area	Inferred (OK)				Inferred (UNC)				Inferred (ID^2)			
	Tonnage (Mt)	ZnEq (%)	Pb (%)	Zn (%)	Tonnage (Mt)	ZnEq (%)	Pb (%)	Zn (%)	Tonnage (Mt)	ZnEq (%)	Pb (%)	Zn (%)
Central Zone	4.80	7.69	1.72	5.84	4.80	7.76	1.73	5.89	4.70	8.28	1.85	6.29
East Mill Zone	5.50	5.16	1.30	3.76	5.50	5.17	1.31	3.76	5.30	5.51	1.39	4.02
North Zone	13.10	6.27	1.87	4.26	13.00	6.33	1.89	4.29	12.50	6.66	1.99	4.51
West Zone	6.40	10.09	3.53	6.30	6.40	10.24	3.55	6.42	6.30	10.57	3.70	6.60
N-204 Zone	8.60	4.74	1.02	3.61	8.60	4.84	1.02	3.71	8.50	4.90	1.05	3.74
	38.40	**6.58**	**1.85**	**4.58**	**38.20**	**6.58**	**1.87**	**4.64**	**37.20**	**6.96**	**1.96**	**4.84**

15. MINERAL RESERVE ESTIMATES

This chapter is not required for a Technical Report on Mineral Resources.

16. MINING METHODS

This chapter is not required for a Technical Report on Mineral Resources.

17. RECOVERY METHODS

This chapter is not required for a Technical Report on Mineral Resources.

18. PROJECT INFRASTRUCTURE

This chapter is not required for a Technical Report on Mineral Resources.

19. MARKET STUDIES AND CONTRACTS

This chapter is not required for a Technical Report on Mineral Resources.

20. ENVIRONMENTAL STUDIES, PERMITTING, AND SOCIAL OR COMMUNITY IMPACT

This chapter is not required for a Technical Report on Mineral Resources.

21. CAPITAL AND OPERATING COSTS

This chapter is not required for a Technical Report on Mineral Resources.

22. ECONOMIC ANALYSIS

This chapter is not required for a Technical Report on Mineral Resources.

23. ADJACENT PROPERTIES

Currently, there are no adjacent mineral claims or leases to the Project as the surrounding area is withdrawn from staking. Historically, several mining and exploration companies have held and explored ground outside the current area controlled by Pine Point Mining Limited in the area between Hay River and Fort Resolution. Within this large area, 100 km by 40 km, regional exploration consisted of geophysical surveys, typically induced polarization followed up by a few test drillholes. After the rush in the 1960s, there were two major companies/groups in the area: Cominco Ltd. and the Westmin Resources led the Great Slave Reef joint venture.

Cominco Ltd. explored an area of roughly 60 km by 18 km, covering 1,182 km^2 and stretching from Dawson Landing in the east to 3 km west of the Buffalo River in the west. Cominco Ltd. explored the area intensely with induced polarization geophysics and follow-up drilling. Unfortunately, the induced polarization data has been lost but most of Cominco Ltd.'s drillhole locations, geology and assay data were still available. The Cominco Ltd. drillhole database has been essential to the current exploration and development work underway at Pine Point and its reliability is discussed in Chapter 12 - Data Verification.

Westmin Resources Limited ("Westmin") and partners explored an area to the west of and contiguous to the Cominco Ltd. holdings. It was explored in the 1970s and covered an area extending 43 km to the west of the Buffalo River all the way to Hay River. Exploration was guided by induced polarization geophysics and geological concepts. Westmin turned over much of their geological and drilling data to the government, which is now summarized in an open file report NWTGEO_2002_001. The drillhole database has been very useful in evaluating the deposits found by Westmin and currently within the Pine Point leases, west of the Buffalo River.

24. OTHER RELEVANT DATA AND INFORMATION

BBA knows of no additional relevant data that might materially impact the interpretations and conclusions presented in this Technical Report.

25. INTERPRETATIONS AND CONCLUSIONS

25.1 Overview

The objective of BBA's mandate was to produce a Mineral Resource Estimate for the Pine Point Lead-Zinc project and a supporting NI 43-101 Technical Report. This Report and the 2018 MRE herein meet this objective.

The mineral resource estimation parameters and geological interpretation for the Pine Point deposit were established by BBA. Historical operating data, past metallurgical testwork and recent metallurgical testing was also reviewed.

25.2 Mineral Tenure, Surface Rights, Agreements and Royalties

The information provided by Osisko Metals support the conclusion that the mining tenure held is valid.

25.3 Geology and Mineralization

The understanding of the regional geology, lithological and structural controls of the mineralization at Pine Point are sufficient to support estimation of Mineral resources.

25.4 Environmental

The Pine Point project is not subject to any known environmental liabilities. As the area has a long history of exploration and mining, BBA does not anticipate any barriers to access the Project for work planned going forward.

25.5 Metallurgy

Overall metallurgical recoveries of 83.1% for zinc and 87.8% for lead were estimated using data from the historic Cominco Ltd. operations and past metallurgical testwork. These recovery factors have been applied to the Mineral Resource Estimate and are considered acceptable and appropriate. Recent XRT sorting investigations indicate that the Pine Point material is well suited for sensor-based sorting. Prior to including this technology within the flowsheet for Pine Point, a techno-economic evaluation should be initiated comparing mineral sorting with dense media separation.

25.6 2018 Pine Point Resource Estimate

The 2018 Pine Point Mineral Resource Estimate (the "2018 MRE") was prepared by Pierre-Luc Richard, P. Geo., using all available information including historical and recent diamond drillholes.

The mineral resources in the 2018 MRE are not mineral reserves as they do not have demonstrated economic viability. The estimate is categorized as Inferred Resources based on data density, search ellipse criteria, drillhole density and specific interpolation parameters. The effective date of the estimate is November 14, 2018 based on the compilation status and cut-off grade parameters.

BBA considers the 2018 MRE to be reliable and based on quality data, reasonable hypotheses and parameters that follow CIM Definition Standards. After completing the MRE and a detailed review of all pertinent information, BBA concluded the following:

- The in-pit MRE is divided into five geographic zones and includes 52 new pits and the expansion of five historical pits;
- Using a variable cut-off grade of 1.7% and 2.0% ZnEq, the Inferred In-pit Resources amount to 38.4 Mt grading 4.58% zinc and 1.85% lead (6.58% ZnEq) containing approximately 3.9 billion pounds of zinc and 1.6 billion pounds of lead;
- No Indicated Resources have been defined in the 2018 MRE;
- At the deposit scale the grade and tonnage show very little variability at lower cut-off grades;
- The In-pit MRE is robust and relatively insensitive to metal prices.

25.7 Exploration Potential

Following an overall review of all pertinent information, including the MRE, BBA concluded the following:

- The exploration potential remains high at the property scale, justifying compilation and target generation programs;
- The Pine Point project hosts a significant amount of mineralized intercepts that merit follow-up work;
- The potential is high for adding additional resources to Pine Point project by drilling lateral extensions of numerous of the currently identified zones;
- It is likely that drilling additional holes therefore improving the current drill spacing would translate into upgrading Inferred resources to the Indicated category;
- A sampling program of the historical core currently stored on the property is likely to improve the grade of the MRE presented in this Report.

25.8 Risk and Opportunities

BBA is not aware of any known environmental, permitting, legal, title, taxation, socio-economic, marketing, political or relevant issues could be expected to affect the reliability or confidence in the exploration information and Mineral Resource discussed herein or the right or ability to perform future work on the Pine Point project. As with all mineral projects, there is an inherent risk associated with mineral exploration. The mineral resources may be affected by a future conceptual study assessment of mining, processing, environmental, permitting, taxation, socio-economic and other factors. Additional technical factors which may impact the Mineral Resource estimate include:

- Metal prices, smelter terms and valuation assumptions;
- Changes to technical inputs used to estimate zinc and lead content (e.g. bulk density estimation and grade model methodology);
- Changes to geotechnical, hydrogeology and mining assumptions including the application of alternative mining methods;
- Changes to process plant recovery estimates if the metallurgical recovery in certain domains is less or greater than currently assumed including the application of alternative processing methods.

26. RECOMMENDATIONS

26.1 Overview

Based on the results of the 2018 MRE, BBA recommends additional exploration/delineation drilling and further geological interpretation to gain a better understanding of the deposit before updating the current Mineral Resource Estimate.

It is also recommended that metallurgical and hydrogeological studies continue and they are described below.

26.2 Recommended Activities

The following activities are recommended:

26.2.1 Geotechnical and Hydrogeological Studies

To refine the revenue estimates and to facilitate mine operations, dewatering and water management has been identified as being a key factor. Detailed knowledge of the hydrogeological conditions of the area as well as for each open pit sector will be required. The Project benefits from one of the most extensive hydrogeological databases in the industry, including real time dewatering measurements. This knowledge will be increased in completing these prioritized tasks as follows:

- Continuing the background data review and gap analyses taking into account various life of mine ("LOM") schedules for the Project;
- Updating of the conceptual large-scale hydrogeological model for Pine Point property;
- Building and running a numerical model to simulate groundwater flow and dewatering sequences for several mine schedules and identify the most attractive schedule from economical and technical points of view;
- Implementing an appropriate complementary hydrogeological fieldwork program taking into account existing information and data gaps to be filled, as per modelling results and gap analyses.

Subsequent efforts will also include geotechnical work regarding the Project infrastructure and open pit geotechnical design. Given the brownfield nature of the Project, a large project database is available and will be fully used. Based on the current understanding, no particular issues are anticipated. Work will typically include:

- Background data review and gap analyses for characterizing the condition of foundations for the plant area and overburden in the area of open pits, waste rock piles and roads, as well as bedrock conditions in order to confirm the open pit geotechnical design parameters;

- Implementing a geotechnical field program to complement existing information, consisting of conventional overburden characterization and sampling (test pits and drilling), laboratory analyses and engineering analyses and reporting. Open pit design will require complementing oriented core drilling in a few locations as per the data gap analyses. Results will be used to define the appropriate slopes for overburden excavations, to verify stability for all impoundments and to provide or confirm parameters for the open pit designs.

26.2.2 Ground DGPS Survey

Continued ground Differential Global Positioning Systems ("DGPS") surveying of historical drillholes is recommended for correction factors in relation to holes that were not identified in the field, but that have original Cominco Ltd. survey coordinates. It is recommended that a number of visible historical drill grid lines be followed and historical holes identified and surveyed where such gridlines connect the three mineralized trends (North Trend, Main Trend and South Trend). Each area containing a resource estimate should have at least 10% of the original historical collars identified and surveyed. Furthermore, should future exploration focus on an area of historical drilling that has not undergone a DGPS survey of historical collars, surveying of those historical holes is recommended. As well, ground survey base station control points should be established for any new areas being investigated.

26.2.3 LiDAR Survey

A LiDAR survey covering the entire area of the Pine Point Camp (Figure 26-1) is recommended for terrain mapping and precise elevation control for purposes of collar elevation confidence, precision and accuracy.

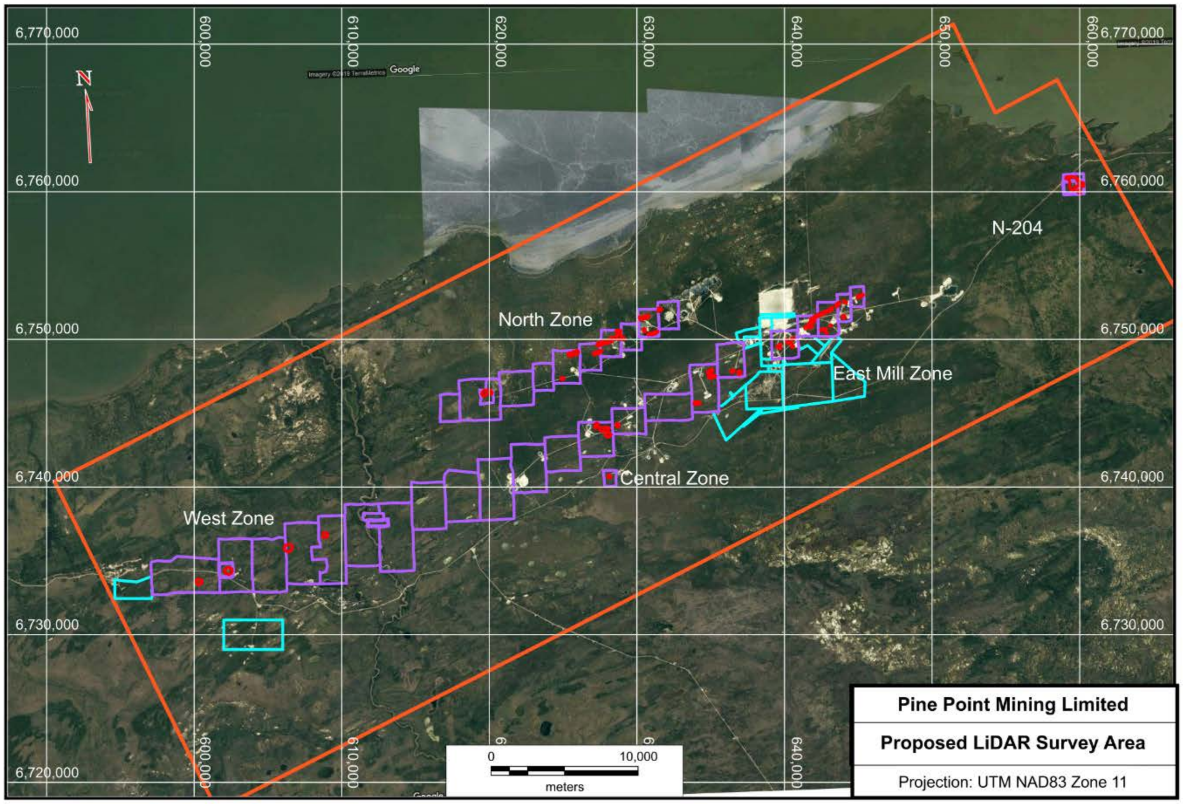


Figure 26-1: Proposed LiDAR survey area

26.2.4 Pit Bathymetry

Bathymetry surveys of all water filled open pits should be conducted and referenced to the DGPS, the survey base station control points and the LiDAR survey.

26.2.5 Photogrammetry Survey and Structural Analyses

Photogrammetry surveys of all pit wall exposures should be conducted where appropriate for structural analysis and interpretation with the objective of defining any structural controls on mineralization. As well, pit wall rock stability should be assessed.

26.2.6 Merging of Surveys

With the established DGPS control points, all of the above surveys should be tied into each other following QA/QC protocols to remove the sources of error between ground base station control points and LiDAR.

26.2.7 Historical Exploration Data Compilation

Compile all historical exploration data along with both current and historical drilling data using the new DGPS and LiDAR controls to support future exploration efforts and further mineral resource estimation.

26.2.8 Historical Core Re-logging and Sampling

With the identification of a significant amount of intact core available from the original Cominco Ltd. drilling, re-logging and re-assaying a portion of the available holes for each resource area and areas of exploration interest is recommended.

26.2.9 Additional In-fill Definition Drilling

To achieve a drillhole spacing of approximately 30 m, or better, and to upgrade the entire modelled Inferred Mineral Resources across the PPMC to the Indicated category, approximately 920 drillholes totalling approximately 54,000 m are required (Figure 26-2).

Two phases of in-fill drilling are recommended, these estimates would complement the 605 drillholes totalling 41,379 m of in-fill drilling that are completed as of December 31, 2018, but not included in the MRE reported herein, as these results arrived after the September resource cut-off date. This drilling in and of itself will meet the 30 m spacing criteria for an estimate of Inferred Resources to be conducted that can then be used for future economic studies.

When considering a phased approach, the Phase 1 drill program would include 600 drillholes totalling 32,000 m and would provide the 30 m spacing criteria needed to conduct Indicated Mineral Resource estimates for the North, Central, and East Mill Zones where the MRE reported herein outlined 23.4 million tonnes of Inferred Mineral Resources.

Phase 2 drilling would include 320 drillholes totalling 22,000 m and would provide the 30 m spacing criteria needed to conduct Indicated Mineral Resource estimates for the N-204 and West Zones where the MRE reported herein outlined 15 million tonnes of Inferred Mineral Resources.

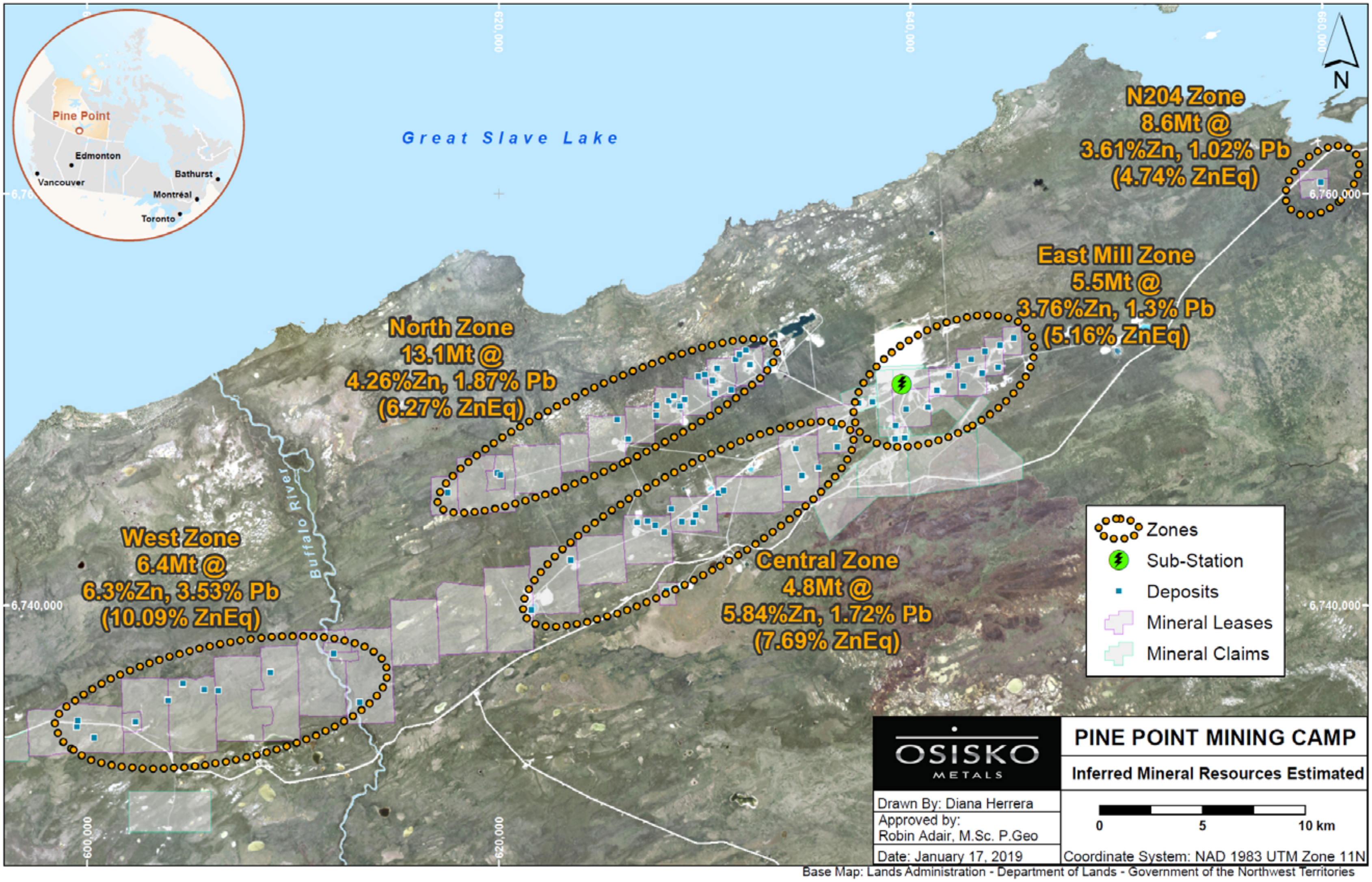


Figure 26-2: Inferred Mineral Resources Estimate

26.2.10 Exploration

The following activities related to exploration are recommended:

- It is recommended that a helicopter airborne gravity survey, covering the central area of the PPMC, be flown (Figure 26-3). Approximately 4,700-line kilometers are recommended to be flown at a 50 m line spacing that will cover the central 220 km^2. Based on the success in the Central area of the PPMC where Cominco Ltd. mined for 25 years, a larger regional survey may also be warranted.
- A trial Induced Polarization geophysical survey using modern equipment and analytical tools is recommended over both in situ tabular and prismatic mineralization types to determine the viability of this type of geophysical survey and its application to screen gravity targets.
- Investigation of structural controls on mineralization using pit mapping, photogrammetry, LiDAR topographic elevation data, and historical airborne electromagnetic and magnetic survey datasets.
- A brownfield exploration drilling program totalling 32,800 m is recommended to follow up on priority targets developed from the compilation work described above.

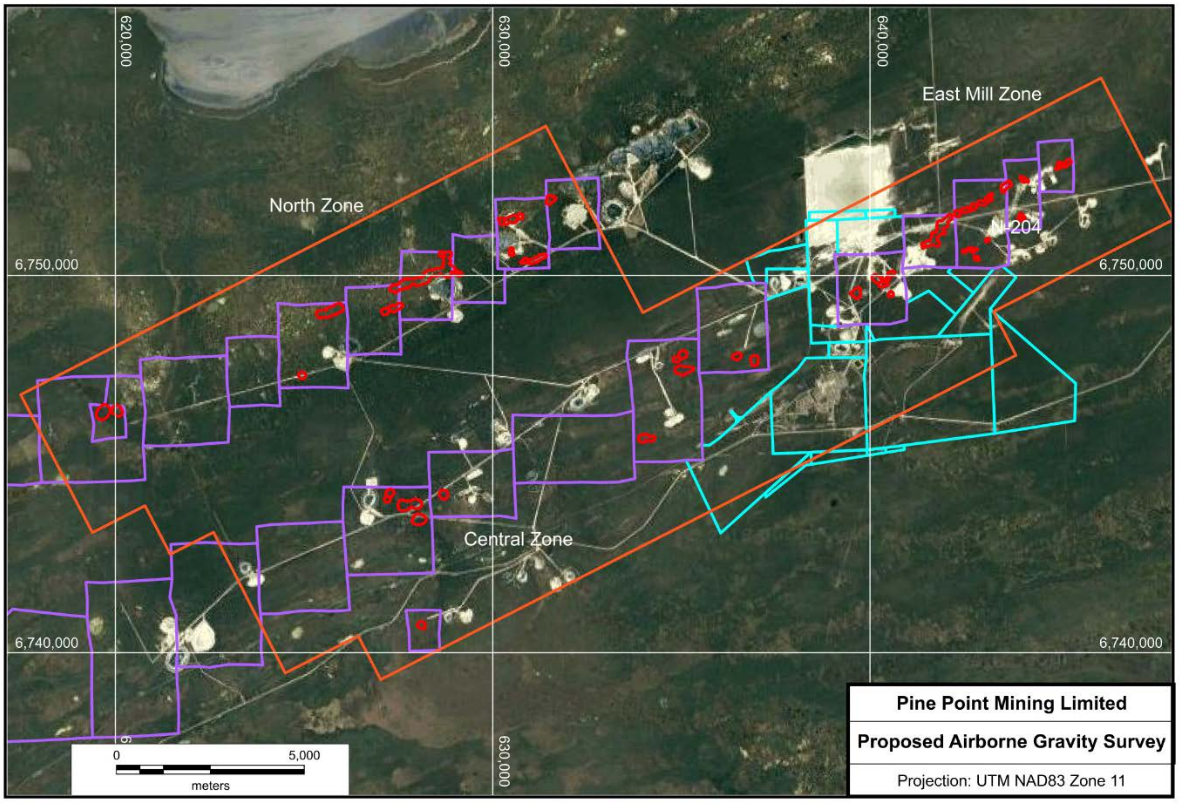


Figure 26-3: Proposed airborne gravity survey area

26.2.11 Metallurgical Testwork

The following testwork and studies are recommended to validate the metallurgical characteristics and variability of the major project zones (Central, East Mill, North Trend N-204 and West Zones) of the PPMC to determine the most suitable process flowsheet:

- Pre-concentration
  - Complete ongoing XRT mineral sorting and dense media testing programs;
  - Process technology selection trade off studies.
- Crushing and grinding
  - Comminution testwork to assess deposit variability and to gather data for mill sizing and power calculation purposes.

- Flotation
  - Investigate the impact of liberation grind size on recovery;
  - Reagent evaluation study;
  - Validate impact of slimes and bitumen;
  - Flowsheet definition.
- Concentrate characterization
  - Validate the quality of the Pb and Zn concentrates for potential impurities which could lead to smelter penalties.
- Dewatering testwork (concentrate and tailings)
  - Gather sedimentation and filtration data for equipment sizing purposes.

26.3 Work Plan Budget

The recommendations in Section 26.2 are budgeted at an estimate based on current site costs with details as follows:

Table 26-1: Work program budget

Description	Unit	Cost ($)
DGPS and survey		120,000
LiDAR survey	1,783 km^2	200,000
Pit bathymetry		100,000
Hydrogeology and Geotechnical		200,000
Photogrammetry and structural analyses		60,000
Merging of surveys		25,000
Historical data compilation		75,000
Phase 1 & 2 Infill drilling for Indicated Resources	54,000 m	16,470,000
Airborne gravity survey	220 km^2	1,000,000
Brownfield exploration drilling	32,800 m	10,000,000
Metallurgical Testwork		350,000
Total		**28,600,000**

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