



NI 43-101 Technical Report

Preliminary Economic Assessment for the Pine Point Lead-Zinc Project

Hay River, Northwest Territories, Canada

Prepared for:

Osisko Metals Inc. and Pine Point Mining Ltd.



Prepared by the following Qualified Persons:

▪ Colin Hardie, P. Eng.	BBA Engineering Inc.
▪ Hugo Latulippe, P. Eng.	BBA Engineering Inc.
▪ Michael Verreault, P. Eng.	Hydro-Ressources Inc.
▪ Pierre-Luc Richard, P. Geo.	PLR Resources Inc.
▪ Paul Gauthier, P. Eng.	WSP Canada Inc.
▪ Simon Latulippe, P. Eng.	WSP Canada Inc.
▪ Zakaria Moctar, P. Eng.	WSP Canada Inc.
▪ Trent Purvis, P. Eng.	WSP Canada Inc.

Effective Date: July 11, 2022

Signature Date: August 26, 2022





IMPORTANT NOTICE

This report was prepared as a National Instrument 43-101 Standards of Disclosure for Mineral Projects Technical Report for Osisko Metals Incorporated ("Osisko Metals") and Pine Point Mining Limited ("PPML") by BBA Engineering Inc. ("BBA"), PLR Resources Inc. ("PLR"), Hydro-Ressources Inc. ("HRI") and WSP Canada Inc. ("WSP"), known as 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 and PPML subject to the terms and conditions of its contract with the report authors and relevant securities legislation. The contract permits Osisko Metals to file this report as a Technical Report with Canadian Securities Regulatory Authorities pursuant to National Instrument 43-101. Except for the purposes legislated under provincial securities law, any other use of this report by any third party is at that party's sole risk. The responsibility for this disclosure remains with Osisko Metals. The user of this document should ensure that this is the most recent Technical Report for the property as it is not valid if a new Technical Report has been issued.



DATE AND SIGNATURE PAGE

This technical report is effective as of the 11th day of July, 2022.

Original signed and sealed on file

Colin Hardie, P. Eng.
BBA Engineering Inc.

August 26, 2022

Date

Original signed and sealed on file

Hugo Latulippe, P. Eng.
BBA Engineering Inc.

August 26, 2022

Date

Original signed and sealed on file

Michael Verreault, P.Eng.
Hydro-Ressources Inc.

August 26, 2022

Date

Original signed and sealed on file

Pierre-Luc Richard, P.Geo.
PLR Resources Inc.

August 26, 2022

Date

Original signed and sealed on file

Paul Gauthier, P. Eng.
WSP Canada Inc.

August 26, 2022

Date



Original signed and sealed on file

Simon Latulippe, P.Eng.
WSP Canada Inc.

August 26, 2022

Date

Original signed and sealed on file

Zakaria Ould Mockett, P.Eng.
WSP Canada Inc.

August 26, 2022

Date

Original signed and sealed on file

Trent Purvis, P.Eng.
WSP Canada Inc.

August 26, 2022

Date



2020 Robert-Bourassa Blvd., Suite 300
Montréal, QC H3A 2A5
T +1 514.866.2111
F +1 514.866.2116
BBA.CA

CERTIFICATE OF QUALIFIED PERSON

Colin Hardie, P. Eng.

This certificate applies to the NI 43-101 Technical Report titled "Pine Point Lead-Zinc Project Preliminary Economic Assessment" (the "Technical Report"), prepared for Osisko Metals Inc. (Osisko), dated August 26, 2022, with an effective date of July 11, 2022.

I, Colin Hardie, P.Eng., as a co-author of the Technical Report, do hereby certify that:

1. I am the National Director, Mining and Metals Business Line with the firm BBA Inc., located at 2020 Robert-Bourassa Blvd., Suite 300, Montréal, Québec, H3A 2A5, Canada.
2. I graduated from the University of Toronto, Ontario Canada, in 1996 with a B.A.Sc. in Geological and Mineral Engineering. In 1999, I graduated from McGill University of Montréal, Québec Canada, with an M.Eng. in Metallurgical Engineering and in 2008 obtained a Master of Business Administration (MBA) degree from the University of Montréal (HEC), Québec Canada.
3. I am a member in good standing of the Northwest Territories Association of Professional Engineers and Geoscientists (NAPEG Member No. L4147) and of the Canadian Institute of Mining, Metallurgy, and Petroleum (Member No. 140556). I have practiced my profession continuously since my graduation.
4. I have been employed in mining operations, consulting engineering and applied metallurgical research for over 20 years. My relevant project experience includes metallurgical testwork analysis, flowsheet development, cost estimation and financial modeling. Since joining BBA in 2008, I have worked as a senior process engineer, project financial analyst and/or lead study integrator for numerous North American iron ore, precious metal, industrial mineral, and base metal projects.
5. I have read the definition of "qualified person" set out in the NI 43-101 – Standards of Disclosure for Mineral Projects ("NI 43-101") and certify that, by reason of my education, affiliation with a professional association, and past relevant work experience, I fulfill the requirements to be a qualified person for the purposes of NI 43-101.
6. I am independent of the issuer applying all the tests in Section 1.5 of NI 43-101.
7. I am author and responsible for the following Chapters and Sections: 2, 3, 4, 5, 13, 15, 17, 18 (Sections 18.1, 18.2, 18.2.1, 18.2.2, 18.2.3 (Process Plant only) and 18.2.4), 19, 20 (Section 20.7), 21 (Sections 21.1.1, 21.1.2, 21.1.3, 21.1.4, 21.1.5.1, 21.1.5.2, 21.1.5.3, 21.1.5.5, 21.1.5.7, 21.1.5.8, 21.1.5.9, 21.1.6, 21.1.6.3, 21.1.6.7, 21.2.1, 21.2.3, 21.2.4, 21.2.6, and 21.2.7), 22 and 24. I am also a co-author and responsible for the integration of Chapters 1, 25, 26 and 27.
8. I visited the Pine Point Property that is the subject of the Technical Report on October 9, 2019.
9. I have had prior involvement with the property that is the subject of the Technical Report having co-authored the technical report entitled " Preliminary Economic Assessment of the Pine Point Lead-Zinc Project, Hay River, Northwest Territories, Canada) (effective date June 11, 2020 and signature date of July 30, 2020).
10. I have read NI 43-101 and the sections of the Technical Report for which I am responsible have been prepared in compliance with NI 43-101.
11. As at the effective date of the Technical Report, to the best of my knowledge, information and belief, the sections of the Technical Report for which I am responsible contain all scientific and technical information that is required to be disclosed to make the portions of the Technical Report for which I am responsible not misleading.

Signed and sealed this 26th day of August 2022.

Original signed and sealed on file

Colin Hardie, P. Eng.



2020 Robert-Bourassa Blvd., Suite 300
Montréal, QC H3A 2A5
T +1 514.866.2111
F +1 514.866.2116
BBA.CA

CERTIFICATE OF QUALIFIED PERSON

Hugo Latulippe, P. Eng., B.Sc.A

This certificate applies to the NI 43-101 Technical Report titled "Pine Point Lead-Zinc Project Preliminary Economic Assessment" (the "Technical Report"), prepared for Osisko Metals Inc. (Osisko), dated August 26, 2022, with an effective date of July 11, 2022.

I, Hugo Latulippe, P.Eng., do hereby certify that:

1. I am a Mining engineer with BBA Inc. located at 990, route de l'Église, bureau 590, Québec, QC, G1V 3V5, CANADA.
2. I am a graduate of Mining and Mineralogy Engineering at Laval University, Québec, Québec, Canada, 2001.
3. I am a member of the Ordre des ingénieurs du Québec (OIQ 126558), Northwest Territories and Nunavut Association of Professional Engineers and Geoscientists (NAPEG No. L4521), Professional Engineers Ontario (PEO No. 100520994) and Engineers and Geoscientists British Columbia (No 209460).
4. I have worked in the mining industry since 2001. I began as a mining engineer in underground mine in Abitibi and then worked in open pit operation in Baie-James and New-Caledonia. I acquired solid experience in mining operation prior to work in three project's developments. I have been involved in mining studies since 2012.
5. I have read the definition of "qualified person" set out in the NI 43-101 – Standards of Disclosure for Mineral Projects ("NI 43-101") and certify that, by reason of my education, affiliation with a professional association, and past relevant work experience, I fulfill the requirements to be a qualified person for the purposes of NI 43-101.
6. I am independent of the issuer applying all the tests in Section 1.5 of NI 43-101.
7. I am author and responsible for the preparation of Section 14.15. I am also co-author and responsible for the relevant portions of Chapters 1, 2, 3, 25, 26 and 27 of the Technical Report.
8. I have not visited the Pine Point Property that is the subject of the Technical Report, as it was not required for the purpose of this mandate.
9. I have had prior involvement with the property that is the subject of the Technical Report by working on the mining section of the 2020 PEA.
10. I have read NI 43-101 and the sections of the Technical Report for which I am responsible have been prepared in compliance with NI 43-101.
11. As at the effective date of the Technical Report, to the best of my knowledge, information and belief, the sections of the Technical Report for which I am responsible contain all scientific and technical information that is required to be disclosed to make the portions of the Technical Report for which I am responsible not misleading.

Signed and sealed this 26th day of August 2022.

Original signed and sealed on file

Hugo Latulippe, P. Eng., B.Sc.A

CERTIFICATE OF QUALIFIED PERSON

Michael Verreault, P. Eng., M.Sc.A

This certificate applies to the NI 43-101 Technical Report titled "Pine Point Lead-Zinc Project Preliminary Economic Assessment" (the "Technical Report"), prepared for Osisko Metals Inc. (Osisko), dated August 26, 2022, with an effective date of July 11, 2022.

I, Michael Verreault, P.Eng., M.Sc.A., as a co-author of the Technical Report, do hereby certify that:

1. I am the President and Hydrogeologist with the firm Hydro-Ressources Inc., located at 1855 Campanules Street, Jonquiere, Québec, G7S 0B5, Canada.
2. I graduated from Université du Québec at Chicoutimi in 2000 with a Bachelor in Geological Engineering. In 2001, I graduated from the Université du Québec at Chicoutimi with a master's degree in Applied Sciences (Hydrogeology).
3. I am a member in good standing of the Northwest Territories and Nunavut Association of Professional Engineers and Geoscientists (NAPEG), License number L4932 and of the Ordre des Ingénieurs du Québec (OIQ), License number 125243. I have practiced my profession continuously since my graduation.
4. I have been in the mining industry since 2008. I have an extensive experience in mine dewatering and I have worked in multiple mines around world. I have been involved in open pit and underground dewatering. I have also developed and patented a tracer technique to optimize mine dewatering.
5. I have read the definition of "qualified person" set out in the NI 43-101 – Standards of Disclosure for Mineral Projects ("NI 43-101") and certify that, by reason of my education, affiliation with a professional association, and past relevant work experience, I fulfill the requirements to be a qualified person for the purposes of NI 43-101.
6. I am independent of the issuer applying all the tests in Section 1.5 of NI 43-101.
7. I am co-author for the relevant portions of Sections 16.3, 16.4.3, 16.4.4, and 6.4.5 of the Technical Report. I am also co-author and responsible for the relevant portions of Chapters 1, 2, 3, 25, 26 and 27 of the Technical Report.
8. I have visited the Pine Point Property that is the subject of the Technical Report.
9. I have had no prior involvement with the property that is the subject of the Technical Report.
10. I have read NI 43-101 and the sections of the Technical Report for which I am responsible have been prepared in compliance with NI 43-101.
11. As at the effective date of the Technical Report, to the best of my knowledge, information and belief, the sections of the Technical Report for which I am responsible contain all scientific and technical information that is required to be disclosed to make the portions of the Technical Report for which I am responsible not misleading.

Signed and sealed this 26th day of August 2022.

Original signed and sealed on file

Michael Verreault, P. Eng., M.Sc.A.

CERTIFICATE OF QUALIFIED PERSON

Pierre Luc Richard, P. Geo.

This certificate applies to the NI 43-101 Technical Report titled "Pine Point Lead-Zinc Project Preliminary Economic Assessment" (the "Technical Report"), prepared for Osisko Metals Inc. (Osisko), dated August 26, 2022, with an effective date of July 11, 2022.

I, Pierre Luc Richard, P.Geo., as a co-author of the Technical Report, do hereby certify that:

1. I am the President of the firm PLR Resources Inc., located at 2000 McGill College Avenue, Suite 600, Montréal, Québec, H3A 3H3, Canada.
2. I graduated from the University of Quebec in Montreal, Quebec, Canada, in 2004 with a B.A.Sc. in Resource Geology. I also obtained an M.Sc. from University of Quebec in Chicoutimi, Quebec, Canada in Earth Sciences in 2012.
3. I am a member in good standing of the Ordre des Géologues du Québec (OGQ Member No. 1119), the Association of Professional Geoscientists of Ontario (PGO Member No. 1714), and the Northwest Territories Association of Professional Engineers and Geoscientists (NAPEG Member No. L2465).
4. I have been employed in mining operations, research groups, and consulting firms for over 20 years. My exploration expertise has been acquired with Richmond Mines Inc., the Ministry of Natural Resources of Quebec (Geology Branch), and numerous companies through my career as a consultant. My mining expertise was acquired at the Beaufor mine and many other producers through my career. I managed numerous technical reports, mineral resource estimates, and audits as a consultant for InnovExplo from 2007 to 2018, as a consultant for BBA from 2018 and 2022, and as a consultant for PLR Resources since 2022. I have practiced my profession continuously since my graduation.
5. I have read the definition of "qualified person" set out in the NI 43-101 – Standards of Disclosure for Mineral Projects ("NI 43-101") and certify that, by reason of my education, affiliation with a professional association, and past relevant work experience, I fulfill the requirements to be a qualified person for the purposes of NI 43-101.
6. I am independent of the issuer applying all the tests in Section 1.5 of NI 43-101.
7. I am author and responsible for the preparation of Chapters 6, 7, 8, 9, 10, 11, 12, 14 (except Section 14.15), and 23. I am also co-author for the relevant portions of Chapters 1, 2, 3, 25, 26 and 27 of the Technical Report.
8. I have visited the Pine Point Property that is the subject of the Technical Report, Project from August 9 to 12, 2018, from October 18 to 20, 2019, and more recently during the course of this mandate from October 28 to 31, 2021.
9. I have had prior involvement with the property that is the subject of the Technical Report having co-authored previous technical reports on the project between 2018 and 2020. I have also provided some technical support to the project on various mandates since 2018.
10. I have read NI 43-101 and the sections of the Technical Report for which I am responsible have been prepared in compliance with NI 43-101.
11. As at the effective date of the Technical Report, to the best of my knowledge, information and belief, the sections of the Technical Report for which I am responsible contain all scientific and technical information that is required to be disclosed to make the portions of the Technical Report for which I am responsible not misleading.

Signed and sealed this 26th day of August, 2022.

Original signed and sealed on file

Pierre Luc Richard, P. Geo.



1305, boul. Lebourgneuf, bureau 510
Quebec City, Quebec, Canada
G2K 2E4
WSP.COM

CERTIFICATE OF QUALIFIED PERSON

Paul Gauthier, P. Eng.

This certificate applies to the NI 43-101 Technical Report titled "Pine Point Lead-Zinc Project Preliminary Economic Assessment" (the "Technical Report"), prepared for Osisko Metals Inc. (Osisko), dated August 26, 2022, with an effective date of July 11, 2022.

I, Paul Gauthier, P.Eng., as a co-author of the Technical Report, do hereby certify that:

1. I am a Senior Mining Consultant with the firm WSP Canada Inc., located at 1305, boul. Lebourgneuf, bureau 510, Quebec, Canada G2K 2E4.
2. I graduated from Laval University, Quebec City, Quebec in 1977 with a B.Sc. in Mining Engineering.
3. I am a Professional Engineer in good standing with Ordre des Ingénieurs du Québec (OIQ #31178), Professional Engineers Ontario (PEO #100080984) and the Northwest Territories and Nunavut Association of Professional Engineers (L5137). I have practiced my profession continuously since my graduation.
4. I have worked in mining engineering operation departments in different roles for over 40 years. I have managed multiple feasibility studies of simple and complex natures from scoping through feasibility studies, detailed engineering, and mine development across Canada.
5. I have read the definition of "qualified person" set out in the NI 43-101 – Standards of Disclosure for Mineral Projects ("NI 43-101") and certify that, by reason of my education, affiliation with a professional association, and past relevant work experience, I fulfill the requirements to be a qualified person for the purposes of NI 43-101.
6. I am independent of the issuer applying all the tests in Section 1.5 of NI 43-101.
7. I am author and responsible for the preparation of Sections 16.5 of the Technical Report. I am also co-author and responsible for the relevant portions of Chapters 1, 2, 3, 25, 26 and 27 of the Technical Report.
8. I have not visited the Pine Point Property that is the subject of the Technical Report as it was not required for the purpose of this mandate.
9. I have had no prior involvement with the property that is the subject of the Technical Report.
10. I have read NI 43-101 and the sections of the Technical Report for which I am responsible have been prepared in compliance with NI 43-101.
11. As at the effective date of the Technical Report, to the best of my knowledge, information and belief, the sections of the Technical Report for which I am responsible contain all scientific and technical information that is required to be disclosed to make the portions of the Technical Report for which I am responsible not misleading.

Signed and sealed this 26th day of August 2022.

Original signed and sealed on file

Paul Gauthier, P. Eng.



CERTIFICATE OF QUALIFIED PERSON

Simon Latulippe, P. Eng.

This certificate applies to the NI 43101 Technical Report titled "Pine Point Lead-Zinc Project Preliminary Economic Assessment" (the "Technical Report"), prepared for Osisko Metals Inc. (Osisko), dated August 26, 2022, with an effective date of July 11, 2022.

I, Simon Latulippe, Eng., B.Sc.A, as a co-author of the Technical Report, do hereby certify that:

1. I am an engineer and team leader Geotechnical & Water Management with WSP Canada Inc., located at 1135, boulevard Lebourgneuf, Québec, Québec, Canada G2K 0M5.
2. I graduated from Laval University, Québec, Québec, Canada, 1998, in geological engineering.
3. I am a member in good standing of the Ordre des ingénieurs du Québec (OIQ 121 692). I have practiced my profession continuously since my graduation.
4. I have been employed in mining operations and consulting engineering for over 24 years. My relevant project experience includes geological engineer for a gold mine in northern Abitibi, where I gained experience in mining and environment. As a consultant I have also acquired solid experience in mining projects, specifically mining studies, tailings and water management design and mine site restoration, which I have been involved in for the last sixteen years.
5. I have read the definition of "qualified person" set out in the NI 43-101 – Standards of Disclosure for Mineral Projects ("NI 43-101") and certify that, by reason of my education, affiliation with a professional association, and past relevant work experience, I fulfill the requirements to be a qualified person for the purposes of NI 43-101.
6. I am independent of the issuer applying all the tests in Section 1.5 of NI 43-101.
7. I am author and responsible for the preparation of Chapter 18, Sections 18.2.6, 18.2.7 (Partial), 18.2.8, Chapter 20, (except 20.7), and Chapter 21, Sections 21.1.5.6 (partial), 21.1.6.5 (partial), 21.1.6.6, and 21.2.5. I am also co-author for the relevant portions of Chapters 1, 2, 3, 25, 26 and 27 of the Technical Report.
8. I have not visited the Pine Point Property that is the subject of the Technical Report, as it was not required for the purpose of this mandate.
9. I have had no prior involvement with the property that is the subject of the Technical Report.
10. I have read NI 43-101 and the sections of the Technical Report for which I am responsible have been prepared in compliance with NI 43-101.
11. As at the effective date of the Technical Report, to the best of my knowledge, information and belief, the sections of the Technical Report for which I am responsible contain all scientific and technical information that is required to be disclosed to make the portions of the Technical Report for which I am responsible not misleading.

Signed and sealed this 26th day of August, 2022.

Original signed and sealed on file

Simon Latulippe, Eng.



CERTIFICATE OF QUALIFIED PERSON

Zakaria Ould Moctar, P. Eng.

This certificate applies to the NI 43-101 Technical Report titled "Pine Point Lead-Zinc Project Preliminary Economic Assessment" (the "Technical Report"), prepared for Osisko Metals Inc. (Osisko), dated August 26, 2022, with an effective date of July 11, 2022.

I, Zakaria Ould Moctar, P.Eng., as a co-author of the Technical Report, do hereby certify that:

1. I am the Principal Mining Engineer and Project Manager with the firm WSP Canada Inc., located at 1600 René-Levesque West, Montréal, Québec, H3H 1P9, Canada.
2. I graduated from the Rabat School of Mines (ENSMR, Morocco) in 2004 with a Bachelor of Science in Mining Engineering. In 2006, I graduated from the French Institute of Petroleum in Paris (IFP, France) with a master's degree in environmental and Petroleum Economics. In 2020, I obtained a Master of Business Administration (MBA) degree from Laval University, Québec Canada.
3. I am a member in good standing of the Northwest Territories and Nunavut Association of Professional Engineers and Geoscientists (NAPEG), License number L5007 and of the Ordre des Ingénieurs du Québec (OIQ), License number 5000433. I have practiced my profession continuously since my graduation.
4. I have been in the mining Industry since 2004. I began as a mining engineer in an iron ore mine in Mauritania, then I worked in gold/silver mines in Baie-James and Abitibi, Québec. I acquired solid open pit operation experience in 5 operating mines in Canada and Burkina Faso. I have been involved in mining studies since 2011.
5. I have read the definition of "qualified person" set out in the NI 43-101 – Standards of Disclosure for Mineral Projects ("NI 43-101") and certify that, by reason of my education, affiliation with a professional association, and past relevant work experience, I fulfill the requirements to be a qualified person for the purposes of NI 43-101.
6. I am independent of the issuer applying all the tests in Section 1.5 of NI 43-101.
7. I am author and responsible for the preparation of 16.1, 16.2, 16.4.1, 16.4.2, 16.4.3, 16.4.6, 16.4.7, 16.4.8, 16.4.9, and 16.6 and Sections 21.1.5 (partial), 21.1.5.2, 21.1.6.1, 21.1.6.2, 21.1.6.1, 21.1.6.2 and 21.2.2. I am also co-author for the relevant portions of Chapters 1, 2, 3, 25, 26 and 27 of the Technical Report.
8. I have not visited (or not visited) the Pine Point Property that is the subject of the Technical Report, as it was not required for the purpose of this mandate.
9. I have not had prior involvement with the property that is the subject of the Technical Report.
10. I have read NI 43-101 and the sections of the Technical Report for which I am responsible have been prepared in compliance with NI 43-101.
11. As at the effective date of the Technical Report, to the best of my knowledge, information and belief, the sections of the Technical Report for which I am responsible contain all scientific and technical information that is required to be disclosed to make the portions of the Technical Report for which I am responsible not misleading.

Signed and sealed this 26th day of August, 2022.

Original signed and sealed on file

Zakaria Ould Moctar, P. Eng.



3509 6th Avenue North
Lethbridge Alberta Canada
T1H 5C1
WSP.COM

CERTIFICATE OF QUALIFIED PERSON

Trent Purvis, P. Eng.

This certificate applies to the NI 43-101 Technical Report titled "Pine Point Lead-Zinc Project Preliminary Economic Assessment" (the "Technical Report"), prepared for Osisko Metals Inc. (Osisko), dated August 26, 2022, with an effective date of July 11, 2022.

I, Trent Purvis, P.Eng., as a co-author of the Technical Report, do hereby certify that:

1. I am a Senior Engineer and Manager with the firm WSP Canada Inc., located at 3509 6th Avenue North Lethbridge Alberta, Canada T1H 5C1.
2. I graduated from the University of Alberta, Edmonton Alberta in 1996 with a B.Sc. in Civil Engineering.
3. I am a Professional Engineer in good standing of The Northwest Territories and Nunavut Association of Professional Engineers and Geoscientists and the Association of Professional Engineers and Geoscientists of Alberta. I have practiced my profession continuously since my graduation.
4. I have been employed in consulting engineering for over 25 years. My relevant project experience includes providing enabling infrastructure engineering on large scale industrial and resource development projects across Canada.
5. I have read the definition of "qualified person" set out in the NI 43-101 – Standards of Disclosure for Mineral Projects ("NI 43-101") and certify that, by reason of my education, affiliation with a professional association, and past relevant work experience, I fulfill the requirements to be a qualified person for the purposes of NI 43-101.
6. I am independent of the issuer applying all the tests in Section 1.5 of NI 43-101.
7. I am author and responsible for the preparation of co-author for the relevant portions of Sections 18.2.3 (except Process Plant), 18.2.5, 18.2.7 (partial), 18.2.9 and 18.3; and Sections 21.1.5.4, 21.1.5.6 (partial), 21.1.6.4, and 21.1.6.5 (partial) of the Technical Report. I am also co-author and responsible for the relevant portions of Chapters 1, 2, 3, 25, 26 and 27 of the Technical Report.
8. I have not visited the Pine Point Property that is the subject of the Technical Report as it was not required for the purpose of this mandate.
9. I have had no prior involvement with the property that is the subject of the Technical Report.
10. I have read NI 43-101 and the sections of the Technical Report for which I am responsible have been prepared in compliance with NI 43-101.
11. As at the effective date of the Technical Report, to the best of my knowledge, information and belief, the sections of the Technical Report for which I am responsible contain all scientific and technical information that is required to be disclosed to make the portions of the Technical Report for which I am responsible not misleading.

Signed and sealed this 26th day of August, 2022.

Original signed and sealed on file

Trent Purvis, P. Eng.



TABLE OF CONTENTS

1. Summary	1-1
1.1 Introduction	1-1
1.2 Terms of Reference	1-1
1.3 Contributors	1-2
1.4 Key Project Outcomes	1-3
1.5 Property Description, Location and Ownership	1-4
1.6 Project Royalties.....	1-7
1.7 Geology, Mineralization and Exploration	1-7
1.8 Status of Drilling and Exploration	1-9
1.9 Data Verification.....	1-9
1.10 Mineral Processing and Metallurgical Testing	1-10
1.11 Mineral Resource Estimation.....	1-11
1.12 Mining Methods	1-15
1.12.1 Open-pit Mining	1-15
1.12.2 Underground Mining.....	1-17
1.12.3 Mining Production Schedule	1-19
1.13 Mineral Reserves	1-21
1.14 Recovery Methods	1-21
1.15 Project Infrastructure	1-23
1.15.1 Process Plant Site	1-24
1.15.2 Waste Rock and Overburden Management	1-27
1.15.3 Pine Point Junction	1-29
1.16 Environmental, Permitting and Closure	1-29
1.17 Market Studies and Contracts	1-30
1.17.1 Metal Pricing.....	1-30
1.17.2 Concentrate Logistics	1-30
1.17.3 NSR Calculations.....	1-30
1.17.4 Contracts.....	1-30
1.18 Capital Cost Estimate.....	1-31
1.19 Operating Cost Estimate.....	1-33
1.20 Project Economics	1-36
1.20.1 Cautionary Note	1-36
1.20.2 Methodology and Input Parameters.....	1-37
1.20.3 Results.....	1-37



1.21	Project Schedule and Organization	1-41
1.22	Interpretations and Conclusions	1-42
1.22.1	Project Risks and Opportunities	1-42
1.23	Recommendations	1-43
2.	Introduction.....	2-1
2.1	Scope of Study	2-1
2.2	Report Responsibility and Qualified Persons.....	2-1
2.3	Effective Dates and Declaration	2-4
2.4	Sources of Information	2-5
2.4.1	General.....	2-5
2.4.2	Specialist Input – BBA.....	2-6
2.4.3	Specialist Input – PLR	2-6
2.4.4	Specialist Input – WSP	2-7
2.5	Site Visit.....	2-8
2.6	Currency, Units of Measure and Calculations.....	2-9
2.7	Previous Technical Reports	2-9
2.8	Acknowledgments	2-10
3.	Reliance on Other Experts	3-1
3.1	Introduction	3-1
3.2	Mineral Tenure and Surface Rights	3-1
3.3	Duties, Royalties and Taxation	3-1
3.4	Markets.....	3-2
4.	Property Description and Location	4-1
4.1	Mineral Tenure	4-1
4.2	Surface Leases	4-1
4.3	Project Areas and Deposits	4-2
4.4	Royalties and Encumbrances	4-5
4.5	Environmental Liabilities	4-5
4.6	Environmental Studies, Permitting, and Social or Community Impact.....	4-6
4.6.1	Introduction	4-6
4.6.2	Environmental Studies	4-6
4.6.3	Permitting.....	4-7
4.6.4	Social and Community Impact.....	4-7
5.	Accessibility, Climate, Local Resources and Infrastructure, Physiography	5-1
5.1	Accessibility	5-1



5.1.1	Road Access	5-1
5.1.2	Climate	5-2
5.2	Local Resources and Infrastructure	5-3
5.2.1	Airports, Rail Terminals and Ports	5-3
5.2.2	Historical Mine Infrastructure.....	5-3
5.2.3	Local Workforce	5-4
5.2.4	Additional Support Services.....	5-4
5.3	Exploration Camp	5-5
5.4	Physiography	5-5
6.	History	6-1
6.1.	Prior to 1920	6-1
6.2.	1920 to 1960	6-2
6.3.	1960 to 2000	6-3
6.4.	2000 to 2017	6-7
6.5.	2018	6-23
6.6.	2019	6-24
6.7.	2020	6-25
6.8.	2020 to Present	6-26
7.	Geological Setting	7-1
7.1	General Statement on Nomenclature	7-1
7.2	Regional Geology.....	7-1
7.3	Pine Point Mining Camp Geology	7-5
7.4	Project Geology.....	7-8
7.5	Stratigraphy	7-9
7.6	Hydrothermal Alteration.....	7-12
7.7	Fine to Medium Crystalline Dolomite	7-19
7.8	Coarse Crystalline Dolomite	7-19
7.9	White Vein Dolomite (White Matrix Breccia, Saddle Dolomite)	7-20
7.10	Hydrothermal Dolomitization as a Control.....	7-21
7.11	Hydrothermal Alteration Within the Main Trend	7-21
7.12	Hydrothermal Alteration Within the North Trend	7-22
7.13	Hydrothermal Paleokarst.....	7-23
7.14	Alignment of Mineralization and Structural Controls on Mineralization	7-24



8.	Deposit Types.....	8-1
8.1.	Mineralization	8-1
8.2.	Deposit Types.....	8-4
8.2.1.	Prismatic Deposits	8-5
8.2.2.	Tabular Deposits.....	8-5
8.2.3.	N204 Deposit	8-6
8.3.	Mineralized Trends	8-6
8.4.	Gangue Mineralogy	8-6
8.5.	Nature of the Pine Point Sulphide Deposits	8-7
8.6.	Stratigraphic Control	8-8
9.	Exploration	9-1
9.1.	Gallium and Germanium (Ga/Ge)	9-1
9.1.1.	Re-verifying Pulps	9-1
9.1.2.	Changing Standards to Certified Ga/Ge	9-1
9.2.	LiDAR – Exploration Applications	9-1
9.2.1.	Terrain Corrections.....	9-2
9.2.2.	Collapse Structure Analysis.....	9-2
9.2.3.	Structural Analysis	9-2
9.3.	Unmanned Aerial Vehicle (UAV) Photogrammetry	9-2
9.4.	Project Scale Structural Analyses.....	9-3
9.5.	Gravity Gradiometry.....	9-4
9.6.	Aeromagnetic	9-6
9.7.	Induced Polarization (IP)	9-6
9.8.	Orphan Holes.....	9-7
9.9.	Galena Vectoring	9-9
9.10.	Exploration Approach	9-11
10.	Drilling	10-1
10.1.	Historical Drill Core Relogging	10-1
10.2.	Incorporation of Historical Drillholes.....	10-1
10.2.1.	Historical Drillhole Locations and Validation	10-2
10.2.2.	LiDAR Surveys.....	10-2
10.2.3.	Historical Drillhole Location	10-2
10.2.4.	Transformation of Un-surveyed Holes.....	10-5
10.2.5.	2019 Historical Drill Collar Survey	10-8
10.3.	Pit Bathymetry	10-9



10.4. Drilling Methodology	10-9
10.4.1. Drillhole Selection	10-9
10.4.2. Drillhole Location / Set-up	10-10
10.4.3. Drillhole Orientation at Start-Up	10-10
10.4.4. Drillhole Orientation during Operation	10-10
10.4.5. Drillhole Numbering	10-11
10.4.6. Drillhole Coring	10-11
10.4.7. Drilling and Core Handling	10-13
10.4.8. Receiving Core at the Pine Point Mining Camp	10-13
10.5. Core Logging	10-13
10.5.1. Core Fitting	10-14
10.5.2. Geotechnical Logging Procedure	10-14
10.5.3. Geological Logging Procedure	10-16
10.6. Assay Sample Selection	10-17
10.6.1. Core Sampling (Core Saw Splitting)	10-17
10.6.2. Sample Shipment Preparation	10-18
10.7. Core Photography	10-19
10.8. Core Storage	10-20
10.9. 2017 Drilling Program	10-20
10.10. Winter 2018 and 2019 Drilling Program	10-21
10.11. Post 2020 MRE Drilling Program	10-21
11. Sample Preparation, Analyses, and Security	11-1
11.1 Historic Data	11-1
11.2 Pine Point Mining Limited Data	11-2
11.2.1 Assay Samples	11-2
11.2.2 Density Samples	11-2
11.2.3 Lab Methods of Preparation, Processing and Analysis	11-3
11.2.4 Sample Shipping and Security	11-4
11.3 Quality Assurance and Quality Control (QA/QC)	11-5
11.3.1 Duplicates	11-6
11.3.2 Blanks	11-12
11.3.3 Certified Reference Materials (Standards)	11-18
11.4 Assessment of Results	11-39
11.4.1 Conclusion	11-40
12. Data Verification	12-1
12.1 Site Visit	12-1



12.2	Drilling and Sampling Procedure	12-3
12.3	Recent Drillhole Database.....	12-4
12.3.1	Assays.....	12-4
12.3.2	Drillhole Location	12-5
12.3.3	Downhole Survey	12-5
12.3.4	Independent Resampling Program	12-5
12.3.5	QA/QC.....	12-6
12.4	Historical Drillhole Database.....	12-6
12.4.1	Drillhole Location	12-6
12.4.2	Logging and Re-sampling of Historical Cominco Ltd. Core	12-7
12.4.3	Unsamped Historical Drillholes	12-11
12.5	Conclusion	12-12
13.	Mineral Processing and Metallurgical Testing.....	13-1
13.1.	Introduction	13-1
13.2.	Geology	13-1
13.3.	Historic Operating Data	13-1
13.3.1.	Historical Operations	13-1
13.3.2.	Production History	13-5
13.3.3.	Historical Metallurgy	13-7
13.4.	Past Metallurgical Testwork.....	13-9
13.4.1.	Reference Reports.....	13-9
13.4.2.	Tamerlane Ventures (2004 to 2016)	13-10
13.4.3.	Darnley Bay Resources (2017)	13-17
13.5.	Metallurgical Testwork - Osisko Metals (2018 and 2019)	13-18
13.5.1.	Sample Source	13-18
13.5.2.	Mineral Sorting.....	13-18
13.5.3.	Flotation	13-21
13.6.	Recovery and Grade Projections	13-26
13.7.	Recommendations for Future Work.....	13-27
14.	Mineral Resource Estimate.....	14-1
14.1	Methodology.....	14-1
14.2	Resource Database.....	14-3
14.3	Geological Model.....	14-4
14.4	Voids Model.....	14-9
14.5	Compositing	14-9
14.5.1	Capping	14-10



14.6	Density	14-18
14.7	Contact Plot	14-18
14.8	Variogram Analysis and Search Ellipsoids	14-20
14.9	Block Model	14-22
14.10	Search Ellipsoid Strategy	14-24
14.11	Interpolation Parameters	14-25
14.12	Interpolation Method	14-25
14.13	Block Model Validation	14-25
14.13.1	Visual Validation	14-26
14.13.2	Statistical Validation	14-26
14.14	Mineral Resource Classification	14-29
14.14.1	Mineral Resource Definition	14-29
14.14.2	Mineral Resource Classification for the Pine Point Project	14-30
14.15	Pit Constrained Cut-off Grade and Pit Optimization	14-31
14.16	Underground Cut-off Grade	14-42
14.17	Pine Point Mineral Resource Estimate	14-45
15.	Mineral Reserve Estimates	15-1
16.	Mining Methods	16-1
16.1	Summary	16-1
16.2	Geotechnical Considerations	16-3
16.2.1	Open-pit Geotechnical Considerations	16-3
16.2.2	Underground Geotechnical Considerations	16-3
16.3	Hydrogeological Settings	16-11
16.4	Open-pit Summary	16-12
16.4.1	Pit Optimization	16-12
16.4.2	Pit Design	16-33
16.4.3	Pit Dewatering	16-33
16.4.4	Site Wide Numerical Model; A Dewatering Overview	16-35
16.4.5	Pit Drilling and Blasting	16-38
16.4.6	Material Stockpile and Disposal	16-39
16.4.7	Pit Mining Equipment Fleet	16-39
16.4.8	Open-pit Workforce	16-42
16.5	Underground Mining	16-44
16.5.1	Underground Mining Summary	16-44
16.5.2	Underground Mining Methods	16-45
16.5.3	Underground Mine Design	16-47



16.5.4	Development Rates.....	16-52
16.5.5	Mine Development Schedule	16-53
16.5.6	Production Forecast	16-55
16.5.7	Underground Mining Equipment.....	16-56
16.5.8	Underground Mine Personnel.....	16-56
16.5.9	Mine Support Services	16-57
16.6	Mine Production Schedule	16-65
17.	Recovery Methods.....	17-1
17.1.	Process Plant Design Criteria	17-4
17.2.	Crushing and Mineral Sorting Circuit	17-5
17.3.	Grinding Circuit	17-6
17.4.	Flotation	17-7
17.5.	Concentrate Dewatering	17-9
17.6.	Tailings Dewatering and Disposal	17-9
17.7.	Reagent Systems.....	17-10
17.8.	Energy, Water and Consumable Requirements	17-11
17.8.1.	Energy Requirements.....	17-11
17.8.2.	Water Requirements.....	17-12
17.8.3.	Consumable Requirements	17-12
18.	Project Infrastructure.....	18-1
18.1	Overview.....	18-1
18.1.1	Project Area	18-2
18.2	Process Plant Site	18-4
18.2.1	Overview.....	18-4
18.2.2	Project Area	18-4
18.2.3	Process Plant Site Facilities	18-5
18.2.4	Power	18-13
18.2.5	IT and Communications.....	18-18
18.2.6	Waste Management.....	18-18
18.2.7	Water Management	18-20
18.2.8	Tailings and Reclaim Water Management	18-24
18.2.9	Fresh Water.....	18-26
18.3	Pine Point Junction Infrastructure	18-26
18.3.1	Overview.....	18-26
18.3.2	Facilities.....	18-27



19. Market Studies and Contracts.....	19-1
19.1 Zinc and Lead Price Projections.....	19-1
19.2 Zinc Concentrate.....	19-1
19.2.1 Zinc Market Overview.....	19-2
19.2.2 Zinc Concentrate Marketability	19-3
19.2.3 Zinc Concentrate Sales Terms	19-4
19.2.4 Zinc Concentrate Transport	19-4
19.3 Lead Concentrate	19-4
19.3.1 Lead Market Overview.....	19-5
19.3.2 Lead Concentrate Marketability	19-5
19.3.3 Lead Concentrate Sales Terms	19-6
19.3.4 Lead Concentrate Transport	19-6
19.4 Royalties.....	19-7
19.5 Contracts	19-7
19.6 Comments on Chapter 19.....	19-7
20. Environmental Studies, Permitting, and Social or Community Impact.....	20-1
20.1 Environmental Studies	20-1
20.2 Social or Community Considerations	20-3
20.3 Anticipated Concerns.....	20-4
20.3.1 Water Management	20-4
20.3.2 Water Quality	20-5
20.3.3 Caribou	20-6
20.3.4 Indigenous Land and Resource Use	20-6
20.3.5 Social and Economic Conditions.....	20-7
20.3.6 Legacy Effects and Expectations for Closure and Reclamation.....	20-7
20.4 Mine Waste and Water Management Strategy	20-8
20.4.1 Geochemistry.....	20-8
20.4.2 Waste Rock (WR) and Overburden (OB)	20-8
20.4.3 Tailings.....	20-8
20.4.4 Mineralized Material and Mineral Sorter Rejects	20-9
20.4.5 Contact Water Management – WRSF, TDA, and Surface Water	20-9
20.4.6 Mine Dewatering	20-11
20.5 Regulatory Context	20-11
20.5.1 Environmental Assessment Process.....	20-12
20.5.2 Regulatory Process	20-13
20.5.3 Regulatory Timeline	20-15



20.6	Mine Closure Requirements.....	20-16
20.6.1	Closure Concept	20-17
20.6.2	Reclamation Objectives	20-18
20.6.3	Closure Activities.....	20-18
20.7	Greenhouse Gas (“GHG”) Emissions Assessment	20-19
20.7.1	Methods.....	20-20
20.7.2	Results.....	20-20
20.7.3	Conclusions and Recommendations	20-22
21.	Capital and Operating Costs.....	21-1
21.1	Capital Costs	21-1
21.1.1	Summary	21-1
21.1.2	Scope and Structure of Capital Cost Estimate	21-2
21.1.3	Work Breakdown Structure (WBS) and Estimate Responsibilities	21-3
21.1.4	Exclusions	21-4
21.1.5	Pre-production Capital Costs	21-4
21.1.6	Sustaining Capital Costs.....	21-14
21.2	Operating Costs	21-20
21.2.1	Summary	21-20
21.2.2	Mining.....	21-23
21.2.3	Process Plant	21-30
21.2.4	Power Generation and Supply	21-33
21.2.5	Tailings, Waste and Water Management.....	21-34
21.2.6	General and Administration	21-35
21.2.7	Site Personnel Summary – All Areas.....	21-36
22.	Economic Analysis.....	22-1
22.1	Overview	22-1
22.2	Cautionary Statement.....	22-1
22.3	Assumptions and Basis.....	22-3
22.4	Metal Production	22-5
22.5	Capital and Sustaining Costs	22-5
22.6	Royalties	22-6
22.7	Taxation.....	22-7
22.8	Financial Analysis Summary	22-8
22.9	Production Costs	22-11
22.10	Value Drivers.....	22-12
22.11	Sensitivity Analysis.....	22-14



22.12 Comments on Chapter 22.....	22-18
23. Adjacent Properties	23-1
24. Other Relevant Data and Information	24-1
24.1 Project Organization.....	24-1
24.1.1 Management.....	24-1
24.1.2 Construction Management.....	24-3
24.2 Project Execution Schedule.....	24-4
25. Interpretation and Conclusions.....	25-1
25.1 Overview	25-1
25.2 Mineral Tenure, Surface Rights, Agreements and Royalties.....	25-1
25.3 Geology and Mineralization.....	25-1
25.4 Data Verification and Mineral Resources.....	25-1
25.5 Exploration Potential.....	25-2
25.6 Mining Methods	25-3
25.6.1 Mine Dewatering	25-3
25.6.2 Open-pit Mining	25-4
25.6.3 Underground Mining.....	25-5
25.7 Metallurgy	25-6
25.8 Recovery Methods	25-7
25.9 Environmental Studies	25-8
25.10 Infrastructure.....	25-8
25.11 Market Studies and Contracts	25-8
25.12 Capital Costs	25-9
25.13 Operating Costs.....	25-10
25.14 Indicative Economic Results.....	25-11
25.15 Project Risks and Opportunities.....	25-12
26. Recommendations	26-1
26.1 Geology and Mineral Resources	26-1
26.1.1 Geological Modelling and Target Evaluation	26-1
26.1.2 Additional In-fill Definition Drilling	26-1
26.1.3 Exploration	26-2
26.2 Geotechnical and Hydrogeological Studies	26-2
26.2.1 Mine Dewatering	26-4
26.2.2 Open-pit Deposits.....	26-4
26.2.3 Underground Deposits.....	26-5



26.2.4	Waste Rock, Mineral Sorter Reject and Overburden Storage Facility.....	26-5
26.2.5	Infrastructure	26-5
26.3	Mining	26-6
26.3.1	Open-pit Mining	26-6
26.3.2	Underground Mining.....	26-6
26.4	Metallurgical Testwork.....	26-7
26.5	Processing Plant	26-7
26.6	Infrastructure.....	26-8
26.6.1	On-site Roads	26-8
26.6.2	Sewage System.....	26-8
26.6.3	Transloading Site	26-8
26.6.4	Communications	26-8
26.6.5	Potable Water.....	26-8
26.6.6	Power	26-9
26.7	Tailings, Mine Waste and Water Management	26-9
26.8	Environment and Permitting	26-10
26.9	Market Studies and Contracts	26-10
26.10	Proposed Work Program and Budget	26-11
27.	References.....	27-1



LIST OF TABLES

Table 1-1: Report contributors	1-2
Table 1-2: Pine Point Indicated and Inferred 2022 Mineral Resource Estimate	1-13
Table 1-3: Mining production schedule	1-20
Table 1-4: Site power demand	1-25
Table 1-5: Project capital costs (pre-production and sustaining).....	1-31
Table 1-6: General rate and unit cost assumptions	1-34
Table 1-7: Project operating costs	1-34
Table 1-8: Employee summary – all areas.....	1-35
Table 1-9: Financial analysis summary	1-38
Table 1-10: Key Project activities (preliminary)	1-42
Table 2-1: Qualified persons and areas of report responsibility	2-2
Table 4-1: Mineral leases and mineral claims.....	4-1
Table 4-2: Details of mineral leases (current as of August 8, 2022)	4-8
Table 4-3: Details of mineral claims (current as of August 8, 2022).....	4-10
Table 4-4: Details of surface leases (Current as of August 8, 2022).	4-14
Table 6-1: Pine Point mines production from 1964 to 1983	6-5
Table 6-2: Tamerlane 2008 historical Mineral Resource Estimate for deposits R190, X25, G03, P499, O556, Z155	6-9
Table 6-3: Tamerlane 2008 historical Reserve Estimate for deposits R190, X25, G033, P499, O556, Z155	6-10
Table 6-4: Tamerlane summary of life of mine cut-off grade inputs for the Tamerlane 2008.....	6-11
Table 6-5: Tamerlane 2011 historical Mineral Resource Estimate for deposit N204	6-12
Table 6-6: Tamerlane 2014 pit shell parameters.....	6-14
Table 6-7: Tamerlane 2014 historical Mineral Reserve Estimate	6-15
Table 6-8: Darnley Bay 2017 historical Mineral Resource Estimate for deposits	6-16
Table 6-9: Darnley Bay 2017 historical Mineral Resource Estimate for deposits	6-18
Table 6-10: Darnley Bay 2017 historical Mineral Resource Estimate for deposits N204	6-18
Table 6-11: Darnley Bay 2017 historical Mineral Resource Estimate for the “cluster pits”	6-19
Table 6-12: Darnley Bay 2017 pit shell parameters for the N204 historical Mineral Resource Estimate	6-20
Table 6-13: Darnley Bay 2017 pit shell parameters for the “cluster pits” historical Mineral Resource Estimate	6-21
Table 6-14: 2017 Darnley Bay Limited drilling summary	6-22
Table 6-15: 2018 Pit-constrained Inferred Mineral Resource Estimate	6-23



Table 6-16: 2019 Inferred Mineral Resource Estimate	6-25
Table 6-17: Pine Point Mineral Resource Estimate.....	6-26
Table 6-18: Drillhole database status.....	6-26
Table 6-19: Historical relogging program	6-27
Table 7-1: List of formations after Skall (1975), Lantos (1983), and Rhodes et al. (1984)	7-11
Table 10-1: Quality codes	10-3
Table 10-2: DGPS surveying of historical holes.....	10-8
Table 10-3: Bathymetry survey parameters	10-9
Table 10-4: Drillholes from PPML	10-22
Table 10-5: Historical relogging program	10-22
Table 11-1: Detection limits for elements of interest	11-4
Table 11-2: Standard reference materials used at Pine Point	11-19
Table 11-3: Results for the Zn CRMs used for the 2017, 2018 and Winter 2019 drilling programs	11-20
Table 11-4: Results for the Pb CRMs used for the 2017, 2018 and Winter 2019 drilling programs	11-20
Table 11-5: Results for the Zn CRMs used for the late 2019 drilling and historical relogging programs	11-21
Table 11-6: Results for the Pb CRMs used for the late 2019 drilling and historical relogging programs	11-21
Table 12-1: Independent re-sampling program results	12-5
Table 12-2: Preliminary re-sampling program results	12-10
Table 12-3: Results from the 2019 twining program	12-11
Table 13-1: Pine Point production history (Silke, 2009)	13-6
Table 13-2: DMS test results for O556 / Z155	13-12
Table 13-3: Locked cycle flotation test results for O556 / Z155.....	13-13
Table 13-4: Proposed general reagent scheme	13-14
Table 13-5: N204 Composite head assays	13-14
Table 13-6: DMS Test results for N204.....	13-15
Table 13-7: Locked cycle flotation test results for N204	13-15
Table 13-8: Metallurgical projections (R190).....	13-17
Table 13-9: Metallurgical projections (N204)	13-17
Table 13-10: Mineral sorting testwork summary (-60 mm/+10 mm)	13-19
Table 13-11: Initial rougher flotation testwork results.....	13-22
Table 13-12: Initial cleaner flotation testwork results.....	13-23
Table 13-13: Optimized rougher flotation testwork results	13-24
Table 13-14: Optimized cleaner flotation testwork results.....	13-24
Table 13-15: Zinc concentrate elemental analysis (ALS)	13-25



Table 13-16: Recovery projections (Central, East Mill, North and West Zones)	13-26
Table 13-17: Recovery projections (N204 Zone).....	13-26
Table 14-1: Basic statistics on composites and high-grade capping value for each type of zone	14-17
Table 14-2: Variogram model parameters for each mineralized zone	14-21
Table 14-3: Pine Point block models parameters.....	14-22
Table 14-4: Search ellipsoid ranges by interpolation passes for the mineralized zones	14-24
Table 14-5: Interpolation parameters	14-25
Table 14-6: Comparison of the block and composite mean grades.....	14-26
Table 14-7: Cut-off grade and pit optimization parameters	14-32
Table 14-8: MRE Open-pit cut-off grade per area.....	14-33
Table 14-9: Pine Point Inferred Mineral Resource Estimate	14-45
Table 14-10: Pit constrained Inferred cut-off grade sensitivity.....	14-47
Table 14-11: Pit constrained Indicated cut-off grade sensitivity	14-48
Table 14-12: Underground Inferred cut-off grade sensitivity	14-48
Table 16-1: Comparison of rock mass rating values (AMC. 2012)	16-4
Table 16-2: Excavation type vs. Ground support system	16-9
Table 16-3: Sills ground support	16-10
Table 16-4: Pit optimization parameters.....	16-13
Table 16-5: Pit optimization results summary for N1 Area	16-19
Table 16-6: Discounted cash flow @ 8% per revenue factor for N1 Area	16-20
Table 16-7: Pit optimization results for each area @ revenue factor = 1.....	16-23
Table 16-8: Pit optimization results for all areas per revenue factor	16-24
Table 16-9: Inventory by pit.....	16-31
Table 16-10: Inventory by area.....	16-33
Table 16-11: Hydrogeological model calibration results.....	16-35
Table 16-12: Open-pit major equipment fleet	16-38
Table 16-13: Open-pit major equipment fleet	16-40
Table 16-14: Open-pit Workforce	16-42
Table 16-15: Drilling design criteria.....	16-46
Table 16-16: Overburden (OB) volume and tonnage for portal excavation – West Zone	16-48
Table 16-17: Development advance rates.....	16-53
Table 16-18: Development Schedule – West Zone	16-53
Table 16-19: Development schedule – Central Zone	16-54
Table 16-20: Production forecast including development in mineralized zone – West Zone.....	16-55
Table 16-21: Mineralized tonnes – Central Zone	16-55



Table 16-22: Production forecast for upper method only – Central Zone	16-56
Table 16-23: Underground mine equipment list	16-56
Table 16-24: Underground mine personnel list	16-57
Table 16-25: Mobile equipment list and air demands for the West Zone	16-60
Table 16-26: Selected fan models	16-63
Table 16-27: Mobile equipment list for the Central Zone	16-64
Table 16-28: Mine production schedule for underground area C1	16-67
Table 16-29: Mine production schedule for underground area W1.....	16-68
Table 16-30: Detailed production schedule for Open-pit and Underground	16-69
Table 16-31: Mine production table	16-71
Table 17-1: Process plant design basis	17-4
Table 17-2: Reagent systems	17-11
Table 17-3: Process plant power demand by area	17-11
Table 17-4: Reagents – Application and consumption.....	17-12
Table 18-1: Mine infrastructure - Design criteria summary	18-11
Table 18-2: Site power demand	18-15
Table 18-3: Volumes of waste rock or overburden capacities in each pile or pit group	18-20
Table 18-4: Summary of pumps required for open-pit dewatering	18-22
Table 18-5: Summary of pumps required for underground dewatering	18-23
Table 19-1: Indicated zinc concentrate assays (ALS).....	19-2
Table 19-2: Zinc concentrate terms used in derivation of the NSR	19-4
Table 19-3: Indicated lead concentrate assays (SGS).....	19-5
Table 19-4: Lead concentrate terms used in derivation of the NSR	19-6
Table 20-1: Authorizations likely to be required for the Project	20-14
Table 20-2: Fuel consumption during the LOM.....	20-21
Table 21-1: Project LOM capital cost summary.....	21-1
Table 21-2: CAPEX estimate responsibilities by WBS	21-4
Table 21-3: Project pre-production capital cost summary	21-5
Table 21-4: General administration (Owner's costs) pre-production capital cost summary	21-6
Table 21-5: Open-pit pre-production capital costs (Area 300).....	21-7
Table 21-6: Electrical and communication pre-production capital costs (Area 400)	21-8
Table 21-7: Site infrastructure pre-production capital costs (Area 500).....	21-9
Table 21-8: Process plant pre-production capital costs (Area 600)	21-10
Table 21-9: Tailings, waste rock and water management infrastructure pre-production capital costs.....	21-12
Table 21-10 Indirect pre-production capital costs (Area 900)	21-13



Table 21-11: Project sustaining capital cost summary.....	21-14
Table 21-12: Sustaining capital cost breakdown	21-16
Table 21-13: Underground sustaining capital costs (Area 200).....	21-17
Table 21-14: Open-pit mining sustaining capital costs (Area 300).....	21-18
Table 21-15: Electricity and communication sustaining capital costs (Area 400)	21-18
Table 21-16: Site infrastructure sustaining capital costs (Area 500)	21-18
Table 21-17: Tailings, mine waste and water management sustaining capital costs (Area 800)	21-19
Table 21-18: Site rehabilitation and closure	21-20
Table 21-19: Salvage value.....	21-20
Table 21-20: Pine Point Project operating cost summary.....	21-21
Table 21-21: OPEX estimate responsibilities.....	21-22
Table 21-22: General rate and unit cost assumptions.....	21-23
Table 21-23: Mining operating cost summary	21-23
Table 21-24: Open-pit mining cost summary	21-24
Table 21-25: Underground mining operating costs - West Zone	21-26
Table 21-26: Underground mining operating costs - Central Zone.....	21-27
Table 21-27: Open-pit mine operating costs per year	21-28
Table 21-28: Underground mine operating costs per year - West Zone	21-29
Table 21-29: Underground mine operating costs per year - Central Zone.....	21-29
Table 21-30: Process plant operating costs	21-30
Table 21-31: Average reagent costs	21-31
Table 21-32: Process plant maintenance costs by area	21-32
Table 21-33: Media consumption rate	21-32
Table 21-34: Power Generation and Supply operating cost summary	21-34
Table 21-35: Tailings, Waste and Water Management operating cost summary	21-34
Table 21-36: Average general and administrative costs	21-36
Table 21-37: Summary of personnel – All areas.....	21-37
Table 22-1: Financial model parameters	22-4
Table 22-2: Financial analysis summary (pre-tax and after-tax)	22-8
Table 22-3: Pine Point Project financial model summary	22-9
Table 22-4: Production cost summary (USD).....	22-12
Table 22-5: NPV sensitivity results (after-tax) for zinc price and exchange rate variations.....	22-14
Table 22-6: IRR sensitivity results (after-tax) for zinc price and exchange rate variations.....	22-14
Table 22-7: NPV sensitivity results (after-tax) for lead price and exchange rate variations	22-15
Table 22-8: IRR sensitivity results (after-tax) for lead price and exchange rate variations.....	22-15



Table 22-9: NPV sensitivity results (after-tax) for zinc and lead feed grade variations.....	22-15
Table 22-10: IRR sensitivity results (after-tax) for zinc and lead feed grade variations.....	22-16
Table 22-11: NPV sensitivity results (after-tax) for operating and capital cost variations.....	22-16
Table 22-12: IRR sensitivity results (after-tax) for operating and capital cost variations.....	22-16
Table 22-13: NPV sensitivity results (after-tax) for discount rate.....	22-17
Table 24-1: Key milestones (preliminary)	24-4
Table 25-1: Project LOM capital cost summary.....	25-9
Table 25-2: Project operating costs	25-10
Table 25-3: Project risks (preliminary risk assessment)	25-14
Table 25-4: Project opportunities	25-25
Table 26-1: Proposed Work Program Budget	26-11

LIST OF FIGURES

Figure 1-1: Project location map.....	1-5
Figure 1-2: Pine Point Project mining leases and mineral claims.....	1-6
Figure 1-3: Pine Point Project geology (PPML, 2022).....	1-8
Figure 1-4: Resource Project areas	1-15
Figure 1-5: Process flowsheet.....	1-22
Figure 1-6: Pine Point Project area	1-23
Figure 1-7: Capital cost summary (pre-production)	1-32
Figure 1-8: Overall capital cost profile	1-33
Figure 1-9: Operating cost summary (by area).....	1-35
Figure 1-10: Annual lead and zinc production (Mlb)	1-39
Figure 1-11: Net present value (8% discount rate, after-tax) sensitivity analysis.....	1-40
Figure 1-12: Internal rate of return (after-tax) sensitivity analysis.....	1-41
Figure 4-1: Pine Point Mining Limited mineral claims, mineral leases and surface claims	4-2
Figure 4-2: Pine Point Project zones and areas	4-3
Figure 4-3: Deposit names referenced in the Mineral Resource Estimate.	4-4
Figure 5-1: Location of the Pine Point Project.....	5-2
Figure 7-1: Regional geological setting.....	7-2
Figure 7-2: Pine Point Project geology.....	7-4
Figure 7-3: Pine Point Mining Camp revised geology map and structural interpretation	7-5
Figure 7-4: Geological section across the Pine Point barrier reef complex.....	7-6
Figure 7-5: Mineralized trends.....	7-7
Figure 7-6: Bedding – Parallel replacement by HTD - Sulphur Point Formation	7-13
Figure 7-7: Sub-overburden map of interpreted distribution of HTD alteration, largely focused in the Sulphur Point Formation	7-15
Figure 7-8: On-reef lithology and stratigraphy with schematic HTD superimposed	7-16
Figure 7-9: On-reef lithology and stratigraphy with schematic HTD and mineralization superimposed	7-17
Figure 7-10: Alteration map of the L37 pit.....	7-18
Figure 7-11: Schematic cross-section through the L37 pit.....	7-18
Figure 7-12: Composite section through North Trend showing Prismatic deposits	7-23
Figure 7-13: Paragenetic sequence at Pine Point	7-24
Figure 8-1: Fine colloform and replacive sphalerite marginal to the coarse-crystalline	8-3
Figure 8-2: Examples of mineralization in hand specimen	8-3
Figure 8-3: Coarse-crystalline sphalerite-rich mineralization at the core of the L37 deposit	8-4

Figure 9-1: Interpreted structural geology	9-4
Figure 9-2: Airborne gravity gradiometry survey coverage area	9-5
Figure 9-3: Airborne gravity gradiometry lows – collapse structures, LiDAR base map	9-6
Figure 9-4: R67 chargeability model base and anomalous iso-surfaces (perspective view, looking southwest)	9-7
Figure 9-5: Orphan holes within the current MRE	9-8
Figure 9-6: Regional orphan drillholes	9-9
Figure 9-7: Example of metal zonation	9-10
Figure 9-8: Vectoring using lead ratios	9-10
Figure 9-9: Exploration approach used by PPML	9-11
Figure 10-1: DGPS Historical drillhole collar survey	10-5
Figure 10-2: General view of the Pine Point Project showing the location of the different zones	10-23
Figure 10-3: Central Zone map showing the location of historical collars (black) and recent holes from 2017-2019 (orange)	10-24
Figure 10-4: East Mill Zone map showing the location of historical collars (black) and recent holes from 2017-2019 (orange)	10-25
Figure 10-5: North Zone map showing the location of historical collars (black) and recent holes from 2017-2019 (orange)	10-26
Figure 10-6: West Zone map showing the location of historical collars (black) and recent holes from 2017-2019 (none in this case)	10-27
Figure 10-7: N204 Zone map showing the location of historical collars (black) and recent holes from 2017-2019 (none in this case)	10-28
Figure 10-8: Northeast Zone map showing the location of historical collars (black) and recent holes from 2017-2019 (none in this case)	10-29
Figure 10-9: Cross-section view in the East Mill Area looking NW	10-30
Figure 10-10: Cross-section view in the N204 Area looking NW	10-30
Figure 10-11: Cross-section view in the Central Area looking NW	10-31
Figure 10-12: Cross-section view in the North Area looking NE (Prismatic Zone)	10-31
Figure 10-13: Cross-section view in the North Area looking NE (Tabular Zones)	10-32
Figure 10-14: Cross-section view in the West Area looking NW (Prismatic Zone)	10-32
Figure 11-1: Linear graph of Zn duplicates for the 2017 drilling campaign	11-7
Figure 11-2: Linear graph of Pb duplicates for the 2017 drilling campaign	11-8
Figure 11-3: Linear graph of Zn duplicates for the 2018 drilling campaign	11-8
Figure 11-4: Linear graph of Pb duplicates for the 2018 drilling campaign	11-9
Figure 11-5: Linear graph of Zn duplicates for the Winter 2019 drilling campaign	11-9
Figure 11-6: Linear graph of Pb duplicates for the Winter 2019 drilling campaign	11-10



Figure 11-7: Linear graph of Zn duplicates for the late 2019 drilling campaign.....	11-10
Figure 11-8: Linear graph of Pb duplicates for the late 2019 drilling campaign.....	11-11
Figure 11-9: Linear graph of Zn duplicates for the late 2019 historical relogging program	11-11
Figure 11-10: Linear graph of Pb duplicates for the late 2019 historical relogging program	11-12
Figure 11-11: Results of Zn blanks samples used during the 2017 drilling program	11-13
Figure 11-12: Results of Pb blanks samples used during the 2017 drilling program.....	11-13
Figure 11-13: Results of Zn blanks samples used during the 2018 drilling program	11-14
Figure 11-14: Results of Pb blanks samples used during the 2018 drilling program.....	11-14
Figure 11-15: Results of Zn blanks samples used during the Winter 2019 drilling program	11-15
Figure 11-16: Results of Pb blanks samples used during the Winter 2019 drilling program.....	11-15
Figure 11-17: Results of Zn blank samples used during the late 2019 drilling program	11-16
Figure 11-18: Results of Pb blank samples used during the late 2019 drilling program	11-16
Figure 11-19: Results of Zn blank samples used during the late 2019 historical relogging program	11-17
Figure 11-20: Results of Pb blank samples used during the late 2019 historical relogging program	11-17
Figure 11-21: 2017 results of standard ME-1301	11-22
Figure 11-22: 2017 results of standard ME-1601	11-23
Figure 11-23: 2017 results of standard OREAS-132b	11-23
Figure 11-24: 2017 results of standard OREAS-133a	11-24
Figure 11-25: 2017 results of standard OREAS-352.....	11-24
Figure 11-26: 2018 results of standard ME-1301	11-25
Figure 11-27: 2018 results of standard ME-1601	11-25
Figure 11-28: 2018 results of standard OREAS-132b	11-26
Figure 11-29: 2018 results of standard OREAS-133a	11-26
Figure 11-30: 2017 and 2018 Zn results for the ME-1301 standard	11-27
Figure 11-31: 2017 and 2018 Zn results for the ME-1601 standard	11-28
Figure 11-32: Winter 2019 results of standard ME-1301	11-28
Figure 11-33: Winter 2019 results of standard ME-1409	11-29
Figure 11-34: Winter 2019 results of standard ME-1601	11-29
Figure 11-35: Winter 2019 results of standard OREAS-133a	11-30
Figure 11-36: Winter 2019 results of standard OREAS-132b	11-30
Figure 11-37: Winter 2019 results of standard OREAS-131b	11-31
Figure 11-38: Standard ME-1409 late 2019 drilling results.....	11-31
Figure 11-39: Standard ME-1409 2019 historical relogging results	11-32
Figure 11-40: Standard ME-1601 late 2019 drilling results.....	11-32

Figure 11-41: Standard ME-1601 2019 historical relogging results	11-33
Figure 11-42: Standard OREAS 131b late 2019 drilling results.....	11-33
Figure 11-43: Standard OREAS 131b 2019 historical relogging results.....	11-34
Figure 11-44: Standard OREAS 132b late 2019 drilling results.....	11-34
Figure 11-45: Standard OREAS 132b 2019 historical relogging results.....	11-35
Figure 11-46: Standard OREAS 133a late 2019 drilling results.....	11-35
Figure 11-47: Standard OREAS 133a 2019 historical relogging results.....	11-36
Figure 11-48: Standard OREAS 34h late 2019 drilling results	11-36
Figure 11-49: Standard OREAS 34h 2019 historical relogging results.....	11-37
Figure 11-50: Standard OREAS 34i late 2019 drilling results	11-37
Figure 11-51: Standard OREAS 34i 2019 historical relogging results	11-38
Figure 11-52: Standard OREAS 34k late 2019 drilling results	11-38
Figure 11-53: Standard OREAS 34k 2019 historical relogging results	11-39
Figure 12-1: Historical pits visited during the site visits	12-2
Figure 12-2: Core review in the core logging facility	12-3
Figure 12-3: Drilling underway during one of the site visits and drill collar review	12-4
Figure 12-4: Sample preparation room and samples ready for shipment to the laboratory	12-4
Figure 12-5: Orthomosaic air photo map of the Pine Point core yard	12-8
Figure 12-6: A pallet of NQ core at the Pine Point core yard	12-9
Figure 13-1: Historical process plant block diagram.....	13-3
Figure 13-2: Historical lead recovery and concentrate grade (1974 to 1987)	13-7
Figure 13-3: Historical zinc recovery and concentrate grade (1974 to 1987)	13-8
Figure 13-4: Historical head grade versus recovery relationship (1974 to 1987)	13-9
Figure 13-5: Feed grade versus mass pull to concentrate.....	13-20
Figure 13-6: Zn feed grade versus recovery	13-20
Figure 13-7: Pb feed grade versus recovery.....	13-21
Figure 14-1: Overall plan view for the Pine Point Project	14-2
Figure 14-2: Overall plan view of the 3D model of the mineralized zones (A) and of the drillholes	14-4
Figure 14-3: Example of Prismatic Zones in plan view (top) and in 3D view looking north (bottom)	14-6
Figure 14-4: Example of a Tabular Zone	14-7
Figure 14-5: Example of N204 Zones ²	14-8
Figure 14-6: Sample length distribution within the mineralized zones	14-10
Figure 14-7: Graphs supporting capping on composites for the Prismatic Zones for Pb	14-11
Figure 14-8: Graphs supporting capping on composites for the Tabular Zones for Pb.....	14-12

Figure 14-9: Graphs supporting capping on composites for the N204 Zones for Pb	14-13
Figure 14-10: Graphs supporting capping on composites for the Prismatic Zones for Zn	14-14
Figure 14-11: Graphs supporting capping on composites for the Tabular Zones for Zn	14-15
Figure 14-12: Graphs supporting capping on composites for the N204 Zones for Zn	14-16
Figure 14-13: Contact analysis on the Zn capped composites	14-19
Figure 14-14: Contact analysis on the Pb capped composites	14-20
Figure 14-15: Example of the variography study for Prismatic Zones	14-22
Figure 14-16: Pine Point model validation swath plots along strike (N155, top) and across strike	14-27
Figure 14-17: Pine Point model validation swath plots	14-28
Figure 14-18: General view of the Project – Location of the different pit-constrained zones, historical mill	14-34
Figure 14-19: Central Zone map showing the location of drillholes and pits from the current MRE	14-35
Figure 14-20: East Mill Zone map showing the location of drillholes and pits from the current MRE	14-36
Figure 14-21: North Zone map showing the location of drillholes and modelled pits from the current MRE	14-37
Figure 14-22: West Zone Map showing the location of drillholes and pits from the current MRE	14-38
Figure 14-23: N204 Zone map showing the location of drillholes and pits from the current MRE	14-39
Figure 14-24: Northeast Zone map showing the location of drillholes and pits from the current MRE	14-40
Figure 14-25: 3D view from the East Mill Area	14-41
Figure 14-26: Typical cross-section with pit constrained and underground mineralization	14-42
Figure 14-27: Central Zone map showing the location of drillholes and underground mineral resources from the current MRE	14-43
Figure 14-28: West Zone map showing the location of drillholes and underground mineral resources from the current MRE	14-44
Figure 16-1: Map of the Pine Point Project deposits	16-2
Figure 16-2: Stability graph results – West Zone longhole stopes with the new proposed stope geometries	16-5
Figure 16-3: Plan view of the room and pillar configuration	16-6
Figure 16-4: Underground mine development from Cominco Era	16-7
Figure 16-5: Ground support – RMR spacing rating 10	16-8
Figure 16-6: Ground support – RMR spacing rating 20	16-9
Figure 16-7: Typical development drift ground support layouts	16-10

Figure 16-8: Pine Point geological formations	16-11
Figure 16-9: Discounted cash flow graph @ 8% for N1 Area	16-22
Figure 16-10: East Mill pit shells.....	16-25
Figure 16-11: C1 Pit shells.....	16-25
Figure 16-12: C2 Pit shells.....	16-26
Figure 16-13: C3 Pit shell	16-26
Figure 16-14: N1 Pit shells.....	16-27
Figure 16-15: N2 Pit shells.....	16-27
Figure 16-16: N204 Pit shells.....	16-28
Figure 16-17: NE1 Pit shell	16-28
Figure 16-18: NE2 Pit shells.....	16-29
Figure 16-19: Cross-section looking North-East – East Mill pit L35.....	16-29
Figure 16-20: Summary of the inflow on the whole property: Inflow Rates vs Time	16-37
Figure 16-21: Underground workings locations, excerpt from Figure 16-1 (not to scale).....	16-44
Figure 16-22: Isometric view of typical transverse longhole stoping - R190 Deposit (not to scale)	16-46
Figure 16-23: Longitudinal view - Portal cut – West Zone	16-49
Figure 16-24: Section view - Portal cut – West Zone	16-49
Figure 16-25: Isometric view – Connection ramp - West Zone (not to scale).....	16-50
Figure 16-26: Plan view – C1 Area (not to scale)	16-51
Figure 16-27: Development phase ventilation (R190) isometric view looking west (concept, not to scale)	16-61
Figure 16-28: Production phase (R190) isometric view looking west (concept, not to scale) ...	16-62
Figure 16-29: Production phase ventilation (G03) isometric view looking east (concept, not to scale)	16-63
Figure 17-1: Simplified process plant flow diagram	17-2
Figure 17-2: Process plant 3D general layout	17-3
Figure 17-3: Primary crushing arrangement.....	17-5
Figure 17-4: Secondary and tertiary crushing arrangement	17-6
Figure 17-5: Ball mill and cyclone cluster arrangement.....	17-7
Figure 17-6: Flotation circuit arrangement.....	17-8
Figure 17-7: Concentrate dewatering arrangement	17-9
Figure 17-8: Tailings dewatering arrangement.....	17-10
Figure 18-1: Pine Point Project area	18-3
Figure 18-2: Process Plant Site infrastructure overview	18-5
Figure 18-3: Camp complex - Typical room aisle.....	18-7



Figure 18-4: Camp complex - Kitchen.....	18-7
Figure 18-5: Camp complex - Administrative and dry facility building	18-8
Figure 18-6: 3D view of the process plant.....	18-13
Figure 18-7: Historic and mined-out open-pits identified for tailings storage areas (TDAs)	18-25
Figure 18-8: Pine Point Junction transloading site location	18-27
Figure 18-9: Pine Point Junction concentrate transloading general arrangement	18-28
Figure 20-1: Tailings and reclaim water management overview	20-10
Figure 21-1: Annual and cumulative project capital costs	21-2
Figure 21-2: Distribution of pre-production capital costs	21-5
Figure 21-3 Project sustaining capital cost summary.....	21-15
Figure 22-1: Annual lead and zinc production (Mlb)	22-5
Figure 22-2: Overall Pine Point Project capital cost profile	22-6
Figure 22-3: Life of mine cash flow projection (cumulative, pre-tax and after-tax).....	22-11
Figure 22-4: Main cost value drivers (CAD, discounted at 8%)	22-13
Figure 22-5: Sensitivity of the net present value (after-tax) to financial variables	22-17
Figure 22-6: Sensitivity of the internal rate of return (after-tax) to financial variables	22-18
Figure 24-1: Construction total labour requirements	24-6



TABLE OF ABBREVIATIONS	
Abbreviation	Description
3D	Three dimensional
a	Annum (year)
AA	Atomic absorption
AACE	American Association of Cost Engineers
AAS	Atomic Absorption Spectrometry
ADR	Adsorption-desorption-recovery
AES	Atomic emission spectrometry
Ag	Silver
AGG	Airborne Gravity Gradiometry
AISC	All-in sustaining cost
Al	Aluminum
ALS	ALS Minerals
ARD	Acid rock drainage
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 (quick lime)
Cd	Cadmium
Ce	Cerium
CGL	Challenger Geomatics Ltd.
CIM	Canadian Institute of Mining, Metallurgy and Petroleum
CIRNAC	Crown-Indigenous Relations and Northern Affairs Canada
CMC	Carboxymethylcellulose
CMT	Construction Management Team
CNG	Compressed Natural Gas
CO ₂	Carbon dioxide
CO ₂ eq	CO ₂ equivalent
CoA	Certificate of authorization



TABLE OF ABBREVIATIONS	
Abbreviation	Description
conc.	Concentrate
CP	Cluster Pit
CRF	Cemented rock fill
CRMs	Certified reference materials
CRP	Closure and reclamation plan
CSRS	Canadian Spatial Reference System
Cu	Copper
CV	Coefficient of variation
DAR	Developer's Assessment Report
DDH	Diamond drillhole
DEM	Digital Elevation Model
DFO	Department of Fisheries and Oceans Canada
DGPS	Differential Global Positioning Systems
DKFN	Deninu Kue First Nation
DL	Detection limit
DM	Dismicrite marker
DMS	Dense Media Separation
DSM	Direct Shipment Material
DUP	Duplicate
EA	Environmental assessment
EBA	EBA Engineering Consultants Ltd.
ECCC	Environment and Climate Change Canada
EIA	Environmental Impact Assessment
EIS	Environmental Impact Statement
EM	East Mill
EOH	End of hole
ESA	Environmental Site Assessment
ESR	Permanent mine opening
et al.	and others
Fe	Iron
FIFO	Fly-in / fly-out
FS	Feasibility study
FW	Foot wall
Ga/Ge	Gallium and Germanium
GEMS	Geovia GEMS software
GHG	Greenhouse gas



TABLE OF ABBREVIATIONS	
Abbreviation	Description
GNWT	Government of Northwest Territories
HG	High-grade
HLS	Heavy liquid separation
HQ	HQ- Caliber drillhole
HRI	Hydro-Ressources Inc.
HTD	Hydrothermal dolomite
HW	Hanging wall
IBA	Impact and Benefit Agreements
ICP-AES	Atomic emission spectrometry
ICP-MS	Inductively Coupled Plasma Mass Spectrometry
ICRP	Interim Closure and Reclamation Plan
ID ²	Inverse distance square
IP	Induced Polarization
IR	Information Requests
IRR	Internal rate of return
K	Hydraulic Conductivity
K ₈₀	80% passing – Particle size
Karst	Karst Investments LLC
KFN	K'at'l'odeeche First Nation
KNA	Kriging neighbourhood analysis
LCT	Locked-cycle test
LG	Low-grade
LHD	Load haul dump
LNG	Liquid natural gas
LOM	Life of mine
LSI	LiDAR Services International
LUP	Land use permit
M	Million
m.a.s.l.	Metres above sea level
Ma	Mega annum (Million years)
MAG	Magnetic
MCC	Motor control centre
MDMER	Metal and Diamond Mining Effluent Regulations
Mg	Magnesium
MG	Medium-grade
MIBC	Methyl isobutyl carbinol



TABLE OF ABBREVIATIONS	
Abbreviation	Description
Mn	Manganese
Mpa	Mega pascals
MRE	Mineral Resource Estimate
MS	Massive sulphide
MSO	Mineable Shape Optimizer
MVEIRB	Mackenzie Valley Environmental Review Board
MVLWB	Mackenzie Valley Land and Water Board
MVRMA	<i>Mackenzie Valley Resource Management Act</i>
MVT	Mississippi Valley-Type
Na	Sodium
Na ₂ O	Sodium Oxide
NaCN	Sodium cyanide
NAG	Non-acid generating
NaOH	Sodium hydroxide
NI	National Instrument
Ni	Nickel
NN	Nearest neighbour
No.	Number
Northwest Territories	NWT or NT
NPAG or Non-PAG	Non-potentially Acid Generating
NQ	NQ- Caliber drillhole
NRCan	Natural Resources Canada
NSA	Not sufficient assay
NSR	Net smelter return
NTPC	Northwest Territories Power Corporation
NTS	National topographic system
NWTMN	Northwest Territory Metis Nation
O ₂	Oxygen
OB	Overburden material
OK	Ordinary kriging
ORE	Ore Research & Exportation Pty Ltd.
OREAS	Ore Research & Exportation Pty Ltd. Assay Standards
OSA	On-stream analyzer
Osisko Metals	Osisko Metals Incorporated
Osisko Royalties	Osisko Gold Royalties Ltd.
OTS	Osisko Technical Services



TABLE OF ABBREVIATIONS	
Abbreviation	Description
P ₈₀	80% passing - Product size
Pb	Lead
PEA	Preliminary economic assessment
pH	Potential of hydrogen
PhD	Doctor of philosophy
PMA	Particle Mineral Analysis
PPMC	Pine Point Mining Camp
PPML	Pine Point Mining Limited
PRS	Pressure Reduction System
PV	Present value
PYSTR	Pyrite stringer
Q1	First quarter
Q2	Second quarter
Q3	Third quarter
Q4	Fourth quarter
QA	Quality Assurance
QA/QC	Quality Assurance / Quality Control
QC	Quality Control
QP	Qualified person
R ²	Coefficient of determination
Rescan	Rescan Environmental Services Ltd.
RF	Revenue Factor
RQD	Rock Quality Designation
RWi	Rod work index
S	Sulphur
S.U.	Standard Unit
SAG	Sub-Arctic Geomatics Ltd.
SD	Standard deviation
SEDAR	System for electronic document analysis and retrieval
SG	Specific gravity
SiO ₂	Silicon dioxide / silica
SO ₂	Sulphur dioxide
SO ₄	Sulphate
SRMs	Standard Reference Materials
Std	Standard S.U.
Tamerlane	Tamerlane Ventures Inc.



TABLE OF ABBREVIATIONS	
Abbreviation	Description
tCO ₂ eq	Tonnes of CO ₂ equivalent
tCO ₂ eq/t	tCO ₂ eq per tonne
TDA	Tailings Disposal Areas
Ti	Titanium
TiO ₂	Titanium dioxide
TLRU	Traditional land and resource use
TSM	Total sustainable mining
TSS	Total suspended solids
TSX	Toronto Stock Exchange
TSXV	TSX Venture Exchange
U	Uranium
UAV	Unmanned aerial vehicle
USD or US\$	United States dollar (examples of use: USD2.5M / US\$2.5M)
UTM	Universal Transverse Mercator
vs.	Versus
WBS	Work breakdown structure
Westmin	Westmin Resources Limited
WR	Waste rock
WRSF	Waste rock storage facilities
WSP	WSP Canada Inc.
WTP	Water treatment plant
XRF	X-ray fluorescence
XRT	X-ray transmission
y	Year (365 days)
Zn	Zinc
ZnSO ₄	Zinc sulphate



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





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





TABLE OF ABBREVIATIONS – UNITS OF MEASURE	
Metric	
Unit	Description
\$/t	Dollars per metric tonne
°C	Degrees Celsius
µm	micron
d	day (24 hours)
deg. or °	angular degree
dmt	dry metric ton
G	Giga
g	gram
g/t	grams per (metric) tonne
gWh	Gigawatt hours
h	hour (60 minutes)
K	Thousand (000)
kg	kilogram
kg/t	kilograms per tonne
km	kilometres
km ²	square kilometre
kt	kilotonne
kW	kilowatt
L	litre
m	metre
M	Million
m ²	square metre
m ³	cubic metre
m ³ /h	cubic metres per hour
mg	milligram
ML	millilitre
mm	millimetre
mm ²	square millimetres
Mt	Million tonne
MW	Megawatt
Ø	diameter
ppm	parts per million
SG	specific gravity
t	tonne (1,000 kg) (metric ton)



TABLE OF ABBREVIATIONS – UNITS OF MEASURE	
Metric	
Unit	Description
tpa	tonnes per annum
tpd	tonnes per day
tpy	tonnes per year
usgpm	US gallons per minute
W	Watt
wmt	wet metric ton

1. SUMMARY

1.1 Introduction

In January 2022 Osisko Metals through its 100% owned subsidiary Pine Point Mining Limited ("PPML") commissioned BBA Engineering Inc. ("BBA"), PLR Resources ("PLR"), Hydro-Ressources Inc. ("HRI") and WSP Canada Inc. ("WSP") to perform an update of the Preliminary Economic Assessment ("PEA") based on a revised Mineral Resource Estimate ("2022 MRE") for the Pine Point Lead-Zinc Project (the "Project" or the "Pine Point Project") in accordance with the guidelines of the Canadian Securities Administrators National Instrument 43101 ("NI 43-101") and Form 43101 F1. The Project is located near the town of Hay River in the 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.

The Pine Point Mining Camp ("PPMC") was discovered in 1898 and exploited from 1964 to 1988 by Cominco Ltd. 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.

The preliminary economic analysis contained within this study is partly based on Inferred Mineral Resources that are considered too speculative geologically to have the economic considerations applied to them that would enable them to be categorized as Mineral Reserves. There is no certainty that the PEA based on these mineral resources will be realized.

1.2 Terms of Reference

This Report supports the disclosure in Osisko Metal's news release dated July 13, 2022, titled "OSISKO METALS RELEASES POSITIVE UPDATE FOR PINE POINT PEA AFTER-TAX NPV OF C\$602M AND AFTER-TAX IRR OF 25%". The overall effective date of the report is July 11, 2022.

All measurement units used in this Report are metric. Currency is expressed in Canadian dollars ("CAD" or "\$") unless otherwise noted.

The Mineral Resources and Mineral Reserves are reported in accordance with the Canadian Institute of Mining, Metallurgy and Petroleum ("CIM") Definition Standards for Mineral Resources and Mineral Reserves (May 2014; the 2014 CIM Definition Standards) and the CIM Estimation of Mineral Resources and Mineral Reserves Best Practice Guidelines (November 2003; 2003 CIM Best Practice Guidelines).

1.3 Contributors

The major PEA study contributors, Qualified Persons (QPs) and their respective areas of responsibility are presented in Table 1-1.

Table 1-1: Report contributors

Qualified Person / Consulting Firm	General overview of responsibilities
BBA Engineering Inc.	
- Hugo Latulippe, P.Eng. - Colin Hardie, P.Eng.	- Supervision of Pit Constrained Cut-off Grade and Pit Optimization; - Review of operating cost and capital estimate for the Project - Establish an economic framework for the PEA mineral resources; - Metallurgical testwork analysis; - Develop a conceptual flowsheet, specification and selection of process equipment; - Develop electrical supply and distribution concept; - Capital cost estimation for the electrical infrastructure and process plant facilities; - Operating cost estimate for the Project G&A, power generation and mineral processing; - Prepare a financial model and conduct an economic evaluation; - Interpret the results and make conclusions that lead to recommendations to improve value and reduce risks; - Provide recommendations for additional studies and data collection required to advance the Project to a feasibility study.
PLR Resources Inc.	
Pierre-Luc Richard, P.Geo.	2022 Updated Mineral Resource Estimate - QP Site visit - Description of historical work - Description of geology - Description of deposit type - Description of exploration and drilling programs



Qualified Person / Consulting Firm	General overview of responsibilities
	■ Drillhole Database validation ■ Updated Mineral Resource Estimate
Hydro-Ressources Inc.	
■ Michael Verreault, P.Eng., M.Sc.A.	■ Groundwater investigations related topics: data gathering, field testing and groundwater flow simulations ■ Mine Site Dewatering
WSP Canada Inc.	
■ Zakaria Moctar, P.Eng. ■ Paul Gauthier, P.Eng. ■ Trent Purvis, P.Eng. ■ Simon Latulippe, P.Eng.	■ Open-pit and underground mine planning and equipment selection; ■ Design of the waste rock, tailings, and water management infrastructure; ■ Operating cost estimate for open-pit mining, underground mining and waste and water management; ■ Capital cost estimate for mining, site infrastructure and waste and water management; ■ Estimation of closure costs.

1.4 Key Project Outcomes

The following list details the key outcomes as determined from the updated Mineral Resource Estimate and PEA Study:

- Pit constrained Indicated Mineral Resources of 15.8 M tonnes grading 1.53% Pb and 4.17% Zn;
- Underground and pit constrained Inferred Mineral Resources of 46.9M tonnes grading 1.76% Pb and 4.39% Zn;
- Production of 3,951 Mlb of zinc and 1,687 Mlb of lead over a 12-year mine life from 46.9 Mt of mineral inventory;
- LOM lead and zinc concentrate grades of 65% and 60%, respectively. LOM lead and zinc recoveries of 93% and 87%, respectively;
- Gross revenue ("NSR") of \$5,625 million (payable metal) before royalty;
- Initial capital costs of \$653.3 million, including a \$87.8 million contingency. Sustaining capital costs of \$616.8 million. Reclamation and closure costs of \$68 million.
- LOM operating costs of \$1.8 billion and royalties of \$183.7 million;



- LOM Mine Site operating cost of 0.34 USD/lb ZnEq. C1 Cash cost of 0.61 USD/lb ZnEq. All-in sustaining cost ("AISC") of 0.71 USD/lb ZnEq;
- NPV (8%) of \$1,098.7 million and an IRR of 35.1% before taxes and mining duties;
- NPV (8%) of \$602.3 million and an IRR of 24.8% after taxes and mining duties;
- Payback period of 2.8 years pre-tax and 3.8 years post-tax;
- Average of approximately 285 workers during construction and approximately 485 employees, staff and labour (local and FIFO), will be required during operations;
- Project Construction completed in Q1 2028 (Year 1) and process plant commercial production by Q4 2028 (Year 1).

1.5 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 ("NT") of Canada.

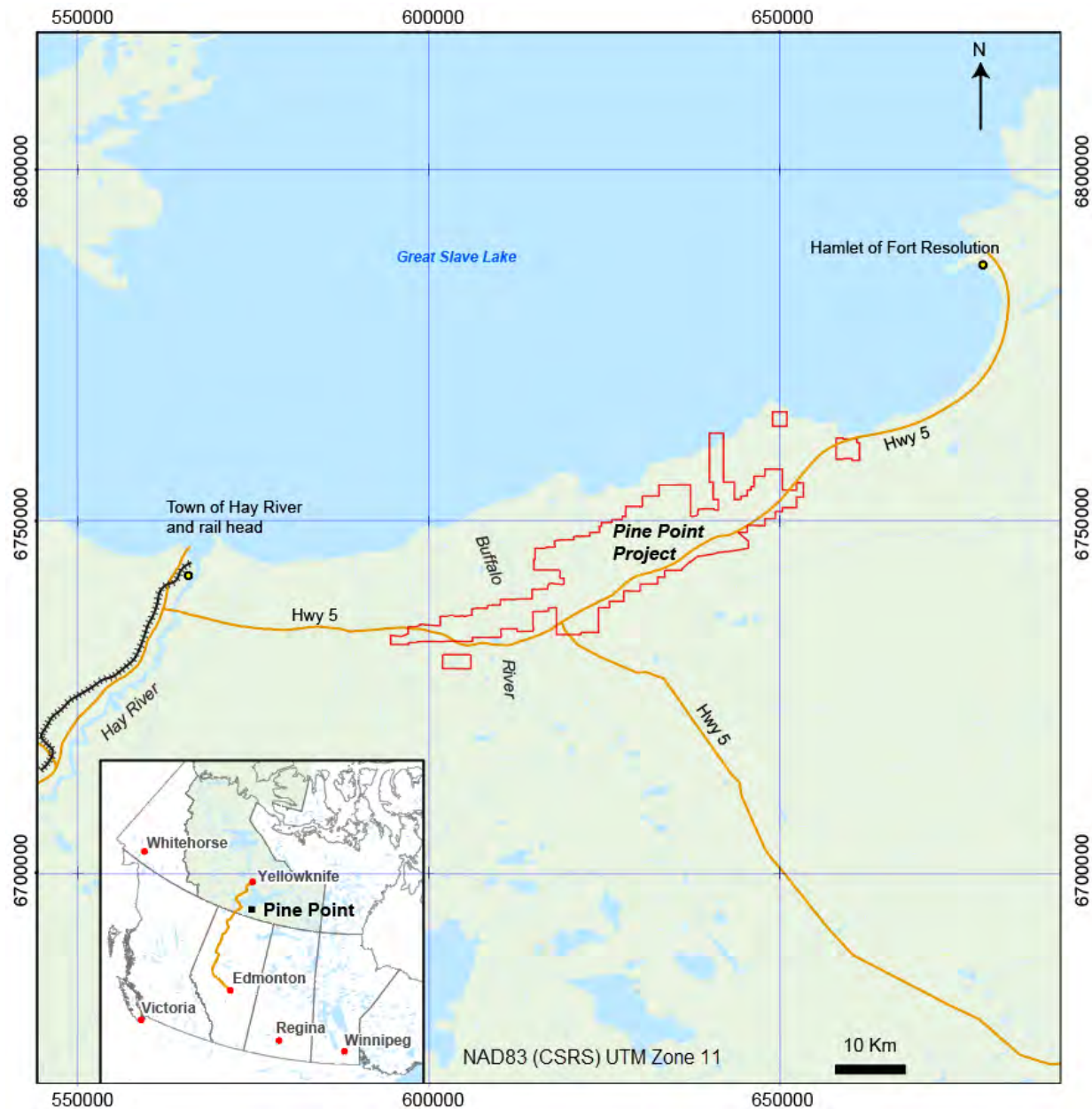


Figure 1-1: Project location map

The Project is composed of mineral leases and claims that cover an approximate 67 km strike length in the PPMC and range between 42 km and 110 km from Hay River, NT. 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 (m.a.s.l.). 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 1-2).

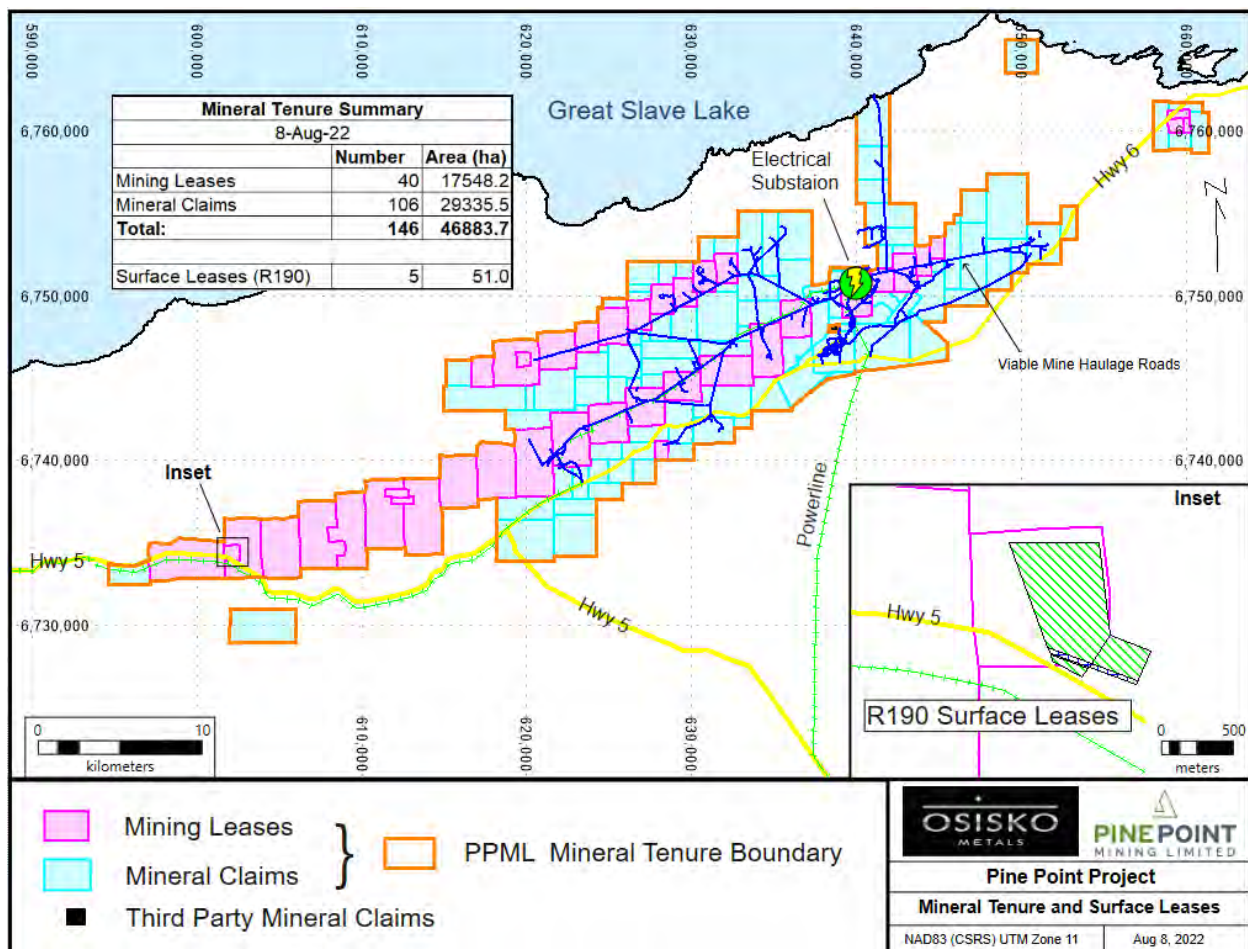


Figure 1-2: Pine Point Project mining leases and mineral claims

The Pine Point mineral deposits are covered by 40 mineral leases and 106 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.



1.6 Project Royalties

On February 11, 2022 Osisko Metals announced that it had entered into a binding letter agreement with Osisko Gold Royalties Ltd ("OGR"), pursuant to which OGR was granted an additional 1.0% net smelter returns royalty on the Pine Point Project in exchange for a cash consideration of \$6.5 million.

As of the effective date of this report, OGR holds a combined 3.0% NSR royalty on the Pine Point Project. This royalty is not subject to buy-back rights in favour of Osisko Metals. There are no other royalties associated to the project.

1.7 Geology, Mineralization and Exploration

The Project is located 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 Carbonate Hosted Zinc-Lead deposits which have similarities to, and differences from, the Mississippi Valley-Type ("MVT") base metal model sensu stricto. Recent observations by PPML staff suggest that while the stratigraphic controls remain, there is evidence that low temperature hydrothermal fluids caused the development dolomite alteration and the development of increased porosity as a result. Further, there is mounting evidence of structural influences on mineralizing fluids and that hydrocarbons may have played a role in the development and precipitation of sulphide mineralization.

Zinc-lead mineralization at Pine Point is hosted in a dolomitized carbonate barrier reef complex that transects the entire property. 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 km². Most historically mined deposits contained between 0.20 Mt and 2 Mt of mineralized material, although the largest, the X15 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 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.

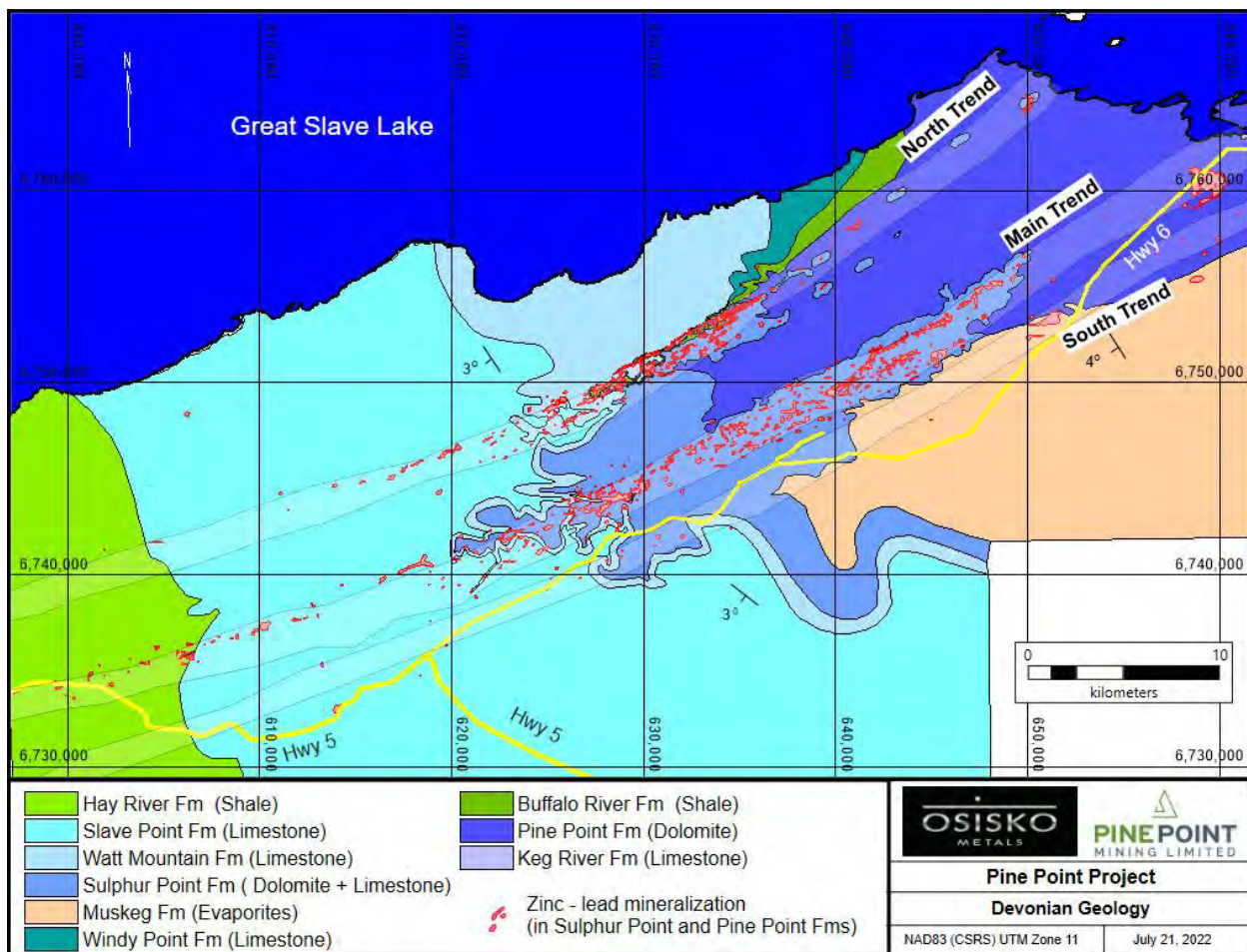


Figure 1-3: Pine Point Project geology (PPML, 2022)

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

1.8 Status of Drilling and Exploration

Diamond drilling has continued since the 2020 PEA with the objective of reducing the intercept spacing to approximately 30 metres within Inferred Mineral Resource areas reported in the 2020 PEA. This spacing is needed to convert these resources to the Indicated Mineral Resource category. To minimize the impact on the environment, drilling is mostly conducted in winter. The program is approximately 45% complete and the Company expects to complete infill drilling by H2, 2023.

1.9 Data Verification

Pierre-Luc Richard, QP, of PLR Resources Inc., visited the Pine Point Project from August 9 to 12, 2018, from October 18 to 20, 2019, and more recently during the course of this mandate from October 28 to 31, 2021. The site visits included a visual inspection of historical core and core drilling in progress, field tours (Figure 12 1), resampling, and discussions of the geological interpretations with geologists and engineers of Osisko Metals. The site visits included a visual inspection of historical core and core drilling in progress, field tours, resampling, and discussions of the current geological interpretations with geologists and engineers of Osisko Metals.

The QP 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 the QP's site visits, which provided an opportunity for Osisko Metals 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.

The QP was granted access to the original assay certificates for all holes drilled from Osisko Metals included in the current Mineral Resource Estimate. 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.

The QP 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 current MRE. In the QP's opinion, the Pine Point database is appropriate to be used for the estimation of Mineral Resources.

1.10 Mineral Processing and Metallurgical Testing

Historically, Pine Point Mining Ltd. (Cominco Ltd.) mined and concentrated over 69.4 Mst (64.3 Mt) of mineralized material from many deposits over a period of 24 years with only slight variations to the process being required to achieve economic recoveries.

Since the closure of operations at Pine Point in 1988, metallurgical test programs were conducted on samples from across the Pine Point District including R190, O556, Z155 and N204 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. The test program 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. Based on an average of the samples tested, mineral sorting was able to increase the grade of both Pb and Zn in the concentrate by approximately two times, achieve recoveries of 98% for Pb and 93% for Zinc while rejecting approximately 40% to 50% of the mass to a waste product.

In 2019, flotation testwork based on a conventional Pb/Zn flowsheet (roughing and two stages of cleaning) using mineral sorting testwork concentrate from the East Mill, Central and North Zones was performed. The testwork program demonstrated that zinc recoveries of 93% to 95% are possible at concentrate grades of 63% to 64% Zn while lead recoveries of 87% to 91% are achievable at concentrate grades ranging from 68% to 72% Pb. Subsequent to the 2020 Preliminary Economic Assessment ("PEA"), a small flotation testwork program was completed in December 2020. The primary findings from the test program indicate that both zinc and lead recoveries from the N204 zone material will likely be lower than originally estimated due its fine grain mineralization.

Elemental analysis of the zinc concentrate during the laboratory testing program indicates that it contains low levels of deleterious elements except for Magnesium Oxide, which is at the typical smelter penalty threshold. Based on the preliminary testwork completed to date, it is not anticipated that there will be any smelter or refinery penalties for the Pine Point zinc or lead concentrates.

Additional material sorting, grinding, flotation and dewatering testwork on all zones at Pine Point is recommended to confirm metallurgical performance and variability as well as for gathering design data for flowsheet development and future economic studies.

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

1.11 Mineral Resource Estimation

The 2022 Pine Point Deposit Mineral Resource Estimate ("2022 MRE") was prepared by Pierre-Luc Richard, P. Geo., using all available information and is effective as of July 11, 2022. The drillhole database used for the 2022 MRE includes 19,509 surface drillholes of which 7,852 intercepted mineralization. The drillhole database includes recent drilling of 78,195 m in 1,182 drillholes between 2017 and 2020 and also incorporates Cominco Ltd.'s historical drillholes, the use of which was partially validated by a drillhole collar survey, twinning programs, and a partial core resampling program.

The estimated block grades were classified into either the Inferred or Indicated Mineral Resource categories using drill spacing, geological continuity of mineralization, grade continuity, presence of recent drilling, and overall level of confidence. No Measured Mineral Resources were defined in this phase of the Project.

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).

Indicated Mineral Resources were defined where the following criteria were met:

- Drill spacing of 30 m or less;
- Demonstrated geological continuity;
- Grade continuity at the reported cut-off grade;
- Recent drillholes confirming the model (geologically and grade-wise);
- Elevated level of confidence on the collar location.

It should also be mentioned that Indicated Mineral Resources were only declared where recent drillholes were sufficient to do so. Any sector solely defined by historical drillholes were kept in the Inferred category even if they met most of the criteria above.

For example, resampling historical holes was not considered sufficient and recommendations were made to add recent holes in zones where none existed before upgrading the classification from Inferred to Indicated. Therefore, multiple sectors have 30 m drill spacing or tighter, geological and grade continuity, and are covered by the recent resurveying program, but are currently declared as Inferred pending additional drilling from the issuer.

When needed, a series of clipping boundaries were created manually in longitudinal views to either upgrade or downgrade classification in order to avoid issues due to automatically generated classification. All remaining estimated but unclassified blocks were not reported.



Based on data density, search ellipse criteria, drillhole density and interpolation parameters, the total Indicated Mineral Resource for the Pine Point deposit is estimated at 15.8 Mt with an average grade of 4.17% Zn and 1.53% Pb (5.55% Zn Eq) and the total Inferred Mineral Resource is estimated at 47.2 Mt with an average grade of 4.43% Zn and 1.68% Pb (5.94% Zn Eq) based on using a Zn equivalent cut-off grade varying from 1.25% to 1.50% for open-pit mineral resources and 4.15% to 4.50% for underground mineral resources depending on location, mining methods, and their metallurgical parameters (Table 1-2 and Figure1-2).

The Mineral Resource Estimate presented herein is either constrained within pit shells developed from the pit optimization analysis discussed in this Report or presented as underground mineral resources using appropriate cut-off grades and sufficient grade continuity.

Table 1-2: Pine Point Indicated and Inferred 2022 Mineral Resource Estimate (effective date July 11, 2022)

Method	Zone	Cut-off Grade (ZnEq %)	Indicated				Inferred			
			Tonnage	ZnEq	Pb	Zn	Tonnage	ZnEq	Pb	Zn
			(kt)	(%)	(%)	(%)	(kt)	(%)	(%)	(%)
Pit Constrained Mineral Resources	Central	1.25	2,424	6.36	1.47	5.04	4,373	6.58	1.65	5.09
	East Mill	1.25	7,232	4.74	1.23	3.63	4,624	4.46	0.89	3.65
	North	1.25 - 1.35	6,097	6.18	1.91	4.46	13,707	4.92	1.43	3.64
	N204	1.50					11,707	4.08	0.90	3.28
Underground Mineral Resources	Central	4.50					2,735	6.91	1.57	5.49
	West	4.15					10,060	9.62	3.31	6.64
Total Pit Constrained			15,753	5.55	1.53	4.17	34,411	4.78	1.21	3.70
Total Underground			-	-	-	-	12,796	9.04	2.94	6.39
Total Combined			15,753	5.55	1.53	4.17	47,207	5.94	1.68	4.43

Notes to Table 1-2:

1. The independent qualified person for the 2022 MRE, as defined by National Instrument ("NI") 43-101 guidelines, is Pierre-Luc Richard, P.Geo. The effective date of the 2022 MRE is July 11, 2022.
2. These mineral resources are not mineral reserves as they do not have demonstrated economic viability. The quantity and grade of reported Inferred Mineral Resources in this MRE are uncertain in nature and there has been insufficient exploration to define these Inferred Mineral 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. Mineral resources are presented as undiluted and in situ for an open-pit and underground scenario and are considered to have reasonable prospects for economic extraction. The constraining pit shells were developed using overall pit slopes of 45 degrees in bedrock and 26.6 degrees in overburden. Mineral resources show sufficient continuity and isolated blocks were discarded; therefore, the herein MRE meets the CIM Guidelines published in November 2019.
4. The 2022 MRE was prepared using GEOVIA GEMS 6.8.3 and is based on 19,509 surface drillholes and 166,376 samples, of which 7,852 drillholes and a total of 47,998 assays were included the modelled mineralization. The drillhole database includes recent drilling of 78,195 m in 1,182 drillholes since 2017 and also incorporates Cominco Ltd.'s historical drillholes, the use of which was partially validated by a drillhole collar survey, twinning programs, and a partial core resampling program. The cut-off date for the drillhole database was December 31, 2019. Approximately 35,000m in 550 drillholes were added to the project since the drillhole database cut-off date.
5. The 2022 MRE encompasses 254 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 1988. 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 mineral resource estimation was calculated from drillhole data using an Ordinary Kriging interpolation method in a percent block model using blocks measuring 10 m x 10 m x 5 m in size.
9. Zinc equivalency percentages are calculated using long term metal prices indicated below (item 10), forecasted metal recoveries, concentrate grades, transport costs, smelter payable metals and charges.
10. The estimate is reported using a ZnEq cut-off varying from 1.25% to 1.50% for open-pit mineral resources and 4.15% to 4.50% for underground mineral resources. Variations take into consideration trucking distances from the pit constrained mineralization to the mill and metallurgical parameters for each area. The cut-off grade was calculated using the following parameters (amongst others): zinc price = USD1.30/lb; lead price = USD1.00/lb; CAD:USD exchange rate = 1.27. The cut-off grade will be re-evaluated in light of future prevailing market conditions and costs.
11. The 2022 MRE presented herein is categorized as Inferred and Indicated Mineral Resources. The Inferred Mineral Resource category is constrained to areas where drill spacing is less than 100 metres and the Indicated Mineral Resource category is constrained to areas where drill spacing is less than 30 metres. In both cases, reasonable geological and grade continuity were also a criteria during the classification process.
12. The pit optimization to develop the mineral resource constraining pit shells was done using Hexagon's MinePlan 3D Version 15.8.
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 QP is not aware of any known environmental, permitting, legal, title-related, taxation, sociopolitical or marketing issues, or any other relevant issues that could materially affect this MRE.

The Project areas that describe the Mineral Resources Estimate are illustrated in Figure 1-4.

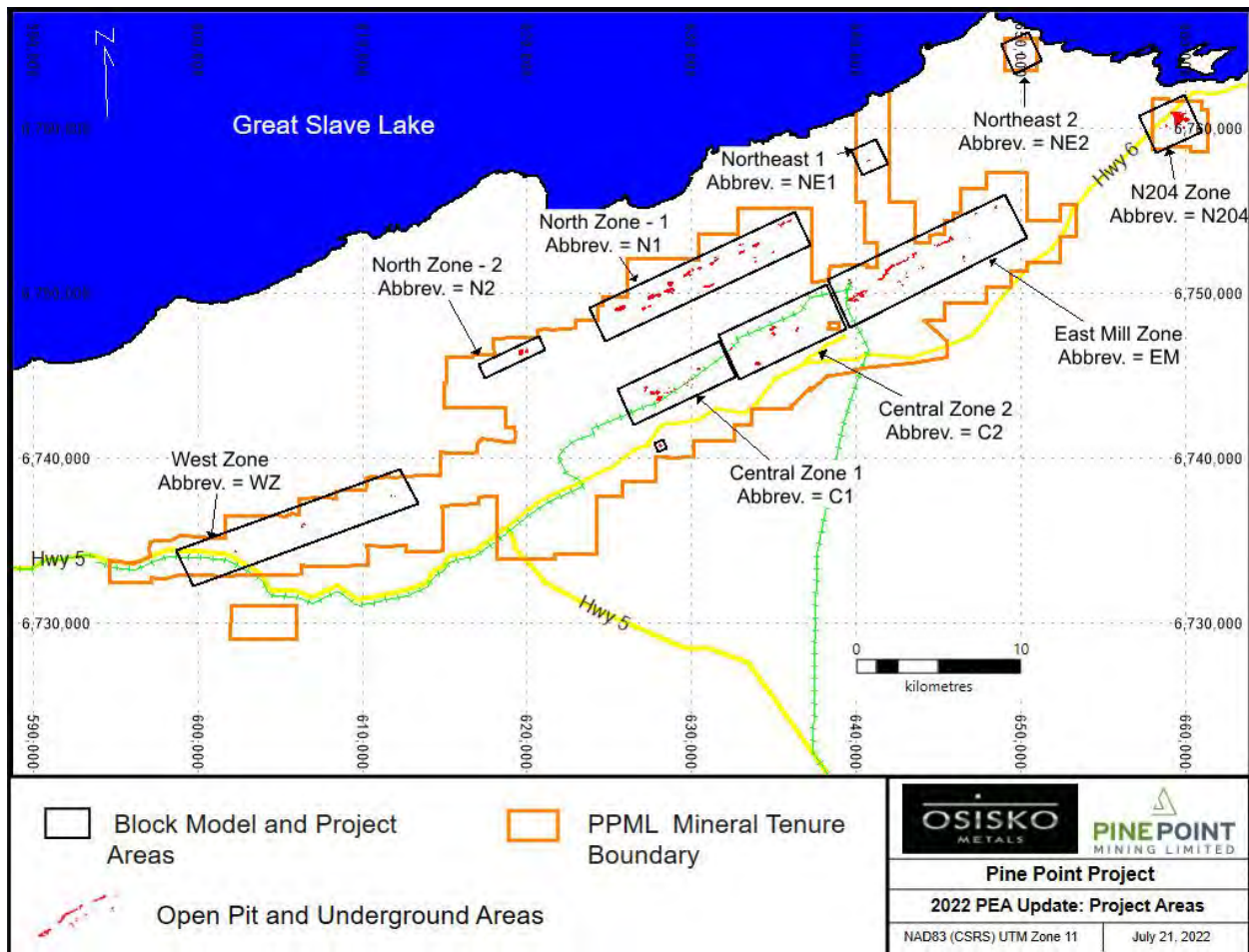


Figure 1-4: Resource Project areas

1.12 Mining Methods

1.12.1 Open-pit Mining

Most of the deposits are mined as open-pits with truck and shovel. The mine life duration is 12 years. It is assumed that overburden material will not require drilling and blasting. The hauling of rock material and overburden will be performed by an average of fifteen to twenty 100 t class-trucks. The loading of waste rock material will be handled by four 12 m³ bucket capacity hydraulic shovels. In addition, three 7 m³ bucket capacity hydraulic excavators will be used in the overburden and mineralized material for more selectivity.

1.12.1.1 Geotechnical and Hydrogeological Considerations

No geotechnical study was available for open-pit mining. The parameters used in this PEA for open-pits are conservative at a 45° overall slope angle considering that the pits are shallow, but this takes into consideration the ramp and the berm widths.

A hydrogeological study was performed by Hydro-Ressources Inc. The existing data was compiled to design and implement a hydrogeological testing campaign. Multiple sources of information were reviewed, including reports during the Cominco production era and a TetraTech report submitted for the PEA in 2020.

The field testing campaign was initiated in early 2021, using new exploration drillholes. Slotted PVC pipe was inserted into several holes to allow for invasive testing. A total of 26 holes were successfully tested on the overall Project. Tests performed include Slug Tests, Injection Tests, Profile Tracer Tests and Chemical Profiles. These tests allow for the measurement of the aquifer properties and to isolate preferential water bearing discontinuities.

A site wide numerical model (SWNM) was built to simulate flow. The model was developed using Feflow 7 and calibration was achieved using 14 historical pits on the Project. Testing results do not suggest any preferential flow associated to specific lithostratigraphic units, but rather flow within discontinuities. The limestone stratigraphy was combined for this model. Inflows were forecasted in all open pits and underground mines, following a mining schedule developed by PPML.

1.12.1.2 Mine Design

Forty-seven deposits from four zones (East Mill, Central, North, and N204) are mined as open-pits considering the deposit's size, shape, orientation, and proximity to the surface as well as economic parameters. Drilling, blasting, loading, and hauling are used to mine the open-pit mineralized material to meet the mine production schedule.

1.12.1.3 Open-Pit Dewatering

Dewatering is planned to be completed using dewatering wells that will be located for optimal dewatering. Their position will be guided by a preliminary sequence of tests to isolate water bearing zones. Then, most wells will be productive and their effect on the piezometric surface will be maximized. The dewatering wells are planned to be drilled and developed in the year before the pit is put into production. Well diameters are planned to be 500 mm in the overburden to above the bedrock; well casings will be installed. The drilling will continue beyond the casing into the bedrock with a hole diameter of 400 mm in the bedrock. The wells will be completed and then

submersible pumps ranging in size from 75 hp to 400 hp, to suit the pit depths and flow, will be installed in the wells and connected to a network of pipes.

Dewatering water will be discharged into nearby mined-out pits or into settling ponds, re-injection will still be investigated but most of the water will be managed on surface using piping and ditches. Pumps and piping equipment will be reused once the pits are depleted, and additional pumps procured as needed for the succeeding pits.

1.12.1.4 Mine Services

Prior to mining the selected deposits, stripping is required to remove topsoil and overburden.

Mineral sorter rejects will need a dedicated fleet, composed of one small excavator and two 45 t articulated trucks on a continuous basis. This fleet could also be provided by a contractor.

1.12.2 Underground Mining

The underground deposits are mechanized ramp access deposits that will use longhole stope and room and pillar mining methods. The underground mine production rate starts in the West Zone at an average of 2,500 t/day in Year 3 and ramps up to an average of 4,000 t/day from Years 4 to 6. After Year 6, the mineralized material from underground mining comes mainly from the Central Zone with an average productivity of around 1,500 t/day until Year 9. The overall underground mine life is expected to be approximately six years of production. All underground mining activities are carried out by contractor teams.

The deposits that are planned to be mined from underground are located in two zones: the West Zone (W1 Area) and the Central Zone (C1 Area). Longhole stope was chosen for Prismatic bodies while room and pillar method is conducted for Tabular deposits. The West Zone contains mainly Prismatic deposits that are more vertically continuous and therefore the main mining method is the longhole stope method. Room and pillar is used in the edges of the mineralized zones for the W1 Area and in C1 Area where only Tabular deposits are identified. In the proposed mine plan, and for both zones combined, the longhole method provides 71% of the total tonnage, development will provide approximately 8%, and the room-and-pillar method will account for 21%.

In order to reach the productivity objectives, the average LOM ramp advance is 1,800 m/year for single-face excavations, and 3,600 m/year when multi-face excavations are available. For the longhole method in the West Zone, sublevels are developed at 25 m vertical intervals. Each level or sublevel is accessed using crosscuts (5 m x 5 m) from the main ramp to allow effective loading of haulage trucks. For the room and pillar method (West Zone and Central Zone), development is

mainly done in the mineralized material. In general, stope accesses will have a dimension of 5 m x 5 m, the same dimensions as the main ramp to allow free movement of equipment throughout the mine. The mineralized material and waste rock are hauled by LHDs from the production area to either a re-muck or loading bay excavated close to the main ramp. The mineralized material will then be loaded in 45 t trucks and hauled directly to surface. For the West Zone, the haul trucks that will transport mineralized material to the concentrator are loaded at the portal and will take Highway 5 to cross the Buffalo River and then continue on Highway 6 towards the nearest haul road to limit the distance used on the highway. For the C1 Area, 45 t articulated trucks will transport the mineralized material.

1.12.2.1 Geotechnical and Hydrogeological Considerations

A geotechnical review for the West Zone (W1 Area) was conducted in 2011-2012 by AMC Consultants, and provided rock mass classification, stability assessments, and ground support guidelines (AMC, 2012). Underground design recommendations are based entirely on this report and most of the geotechnical information is taken from this review.

The rock mass classification Q-RMR values indicate generally a fair to good rock mass quality overall.

There is no geotechnical information for the Central Zone underground workings.

1.12.2.2 Mine Design

Primary underground access for the West Zone deposits will be provided by portals (except for P499 and O556) and excavation of the overburden is necessary beforehand, while for the C1 Area, portals are excavated from the bottom of existing mined-out and dewatered historical pits. A connection ramp is planned to be excavated between R190, P499, and O556 to allow the haulage of mined material from deposits P499 and O556 to surface. After the completion of decline/ramp development, ventilation raises (fresh air) will be excavated in priority to create a second emergency exit before the start of production work.

1.12.2.3 Underground Dewatering

Dewatering of the underground workings is planned to be performed in a similar fashion to open-pit mining. However, the submersible pumps in the W1 Area are larger at 300 and 400 hp due to higher discharge head, and some deposits (R190, P499, O556 and X25) require grouting/till injection beforehand in order to reduce hydraulic conductivity by 1.5 orders of magnitude. Dewatering wells will be developed within each grout curtain to dewater the remaining infiltration

that will flow towards underground workings. In the C1 Area, the underground workings are mainly dewatered from surrounding open-pits (mostly M67) that will continue to be pumped during the underground mining operations.

The dewatering wells will be connected to a network of pipes that will discharge into settling ponds. The pumps and piping equipment will also be reused once the underground workings are depleted.

1.12.2.4 Mine Services

The ventilation system is planned to perform during the development phase, with fans installed in parallel at the portal and above the ventilation raise during the production phase. Primary stopes are backfilled with cemented rock fill ("CRF") and secondary stopes are filled with 1/3 CRF and 2/3 rock fill in order to mine adjacent stopes without leaving a pillar behind.

1.12.3 Mining Production Schedule

The mining production schedule is based on a throughput of 11,250 tpd at the Process Plant during 12 years of operation. Indicated and Inferred Mineral Resources account for 21.14% and 78.86% of total mill feed, respectively. Most of the pits are mined during a period ranging from 1 to 3 months, except for Y65, X71, L35 and N204A which are mined respectively during a period of 6, 8, 11 and 18 months.

Underground workings feed the plant at the same time as open pits between Year 1 and Year 12.

The mining production schedule is presented in Table 1-3.



Table 1-3: Mining production schedule

	Unit	LOM	Y1	Y2	Y3	Y4	Y5	Y6	Y7	Y8	Y9	Y10	Y11	Y12
Open-pit Production														
Open-pits	#		6	6	5	12	12	5	4	9	7	1	1	3
Mineralized Material	Mt	37.9	3.2	4.0	3.5	4.9	2.6	2.5	3.3	4.7	1.9	1.0	5.2	1.0
Zn Grade	%	3.98	4.87	4.56	4.73	3.90	4.04	4.37	3.24	3.61	3.52	3.20	3.48	3.65
Pb Grade	%	1.44	2.13	1.94	1.59	1.30	1.39	1.25	1.63	1.38	1.15	0.90	0.97	1.05
ZnEq Grade	%	5.41	7.01	6.51	6.32	5.20	5.42	5.62	4.87	4.99	4.67	4.10	4.44	4.70
Total Waste Rock	Mt	212.8	31.4	31.6	23.8	15.2	17.5	17.6	16.8	15.3	14.5	15.5	11.2	2.4
Overburden	Mt	75.1	8.6	10.8	5.4	9.5	10.9	11.5	5.7	6.9	4.9	0.8	0.0	0.1
Waste Rock	Mt	137.7	22.8	20.8	18.5	5.7	6.6	6.1	11.2	8.4	9.7	14.7	11.2	2.3
Underground Production														
Mineralized Material	Mt	9.0	0.1	0.2	0.7	1.2	1.6	1.5	1.6	1.0	0.7	0.4		
Zn Grade	%	6.14	5.19	4.41	8.72	7.00	8.10	5.68	5.16	3.80	5.49	5.42		
Pb Grade	%	3.11	2.12	3.76	4.08	2.67	3.41	3.87	3.04	3.26	1.35	1.49		
Total Production to Mill														
Mineralized Material to Mill	Mt	46.9	3.3	4.2	4.2	6.2	4.1	4.1	4.9	5.7	2.6	1.3	5.2	1.0
Zn Grade	%	4.39	4.88	4.55	5.36	4.52	5.56	4.87	3.88	3.64	4.06	3.82	3.48	3.65
Pb Grade	%	1.76	2.13	2.02	1.99	1.57	2.15	2.25	2.10	1.72	1.20	1.06	0.97	1.05

1.13 Mineral Reserves

This section is not relevant to this technical report.

1.14 Recovery Methods

The selection of a conventional lead and zinc flotation-based processing circuit for the Pine Point material and associated flowsheet is based on metallurgical testwork results and historical operational data described in Chapter 13 and BBA's experience with operations treating similar mineralization and of the same scale as proposed for the Pine Point process plant (Figure 1-5).

Metallurgical testwork has shown that the mineralized material is amenable to preconcentration by mineral sorting ("XRT") in the crushing circuit with 99% lead and 93 % zinc recovery at 42 % weight recovery and to froth flotation at a primary grind size of P_{80} 100 μm achieving a 65% lead concentrate with 93 % recovery and a 60% zinc concentrate with 87% recovery. Lead and zinc will be recovered in dedicated standard rougher and two-stage cleaner flotation circuits. No regrind circuit is included, based on preliminary testwork results.

The processing plant will consist of the following unit operations:

- Primary jaw crushing;
- Secondary and tertiary crushing and mineral sorting;
- Fine mineralized material storage and handling;
- Ball mill grinding in a closed circuit with hydrocyclone classification;
- Lead flotation cells including rougher and two cleaning stages;
- Zinc flotation cells including rougher and two cleaning stages;
- Concentrate dewatering (zinc and lead concentrates);
- Flotation tailings thickening and pumping;
- Fresh and reclaim water supply;
- Reagent preparation and distribution.

Two types of final concentrates will be produced: a zinc concentrate and a lead concentrate. Tailings will be disposed of in already mined pits. Process plant energy requirements are expected to average about 79 GWh/a (excluding heating) over the planned life-of-mine. The average installed power for these facilities will be about 10.8 MW (excluding heating). Fresh water requirements are estimated to be 42 m^3/h . The major reagents are anticipated to be collectors

(Aerofloat 3418A and Cytec 3894), frother (MIBC), CMC, lime, copper sulphate activator, zinc sulphate depressant, and flocculant.

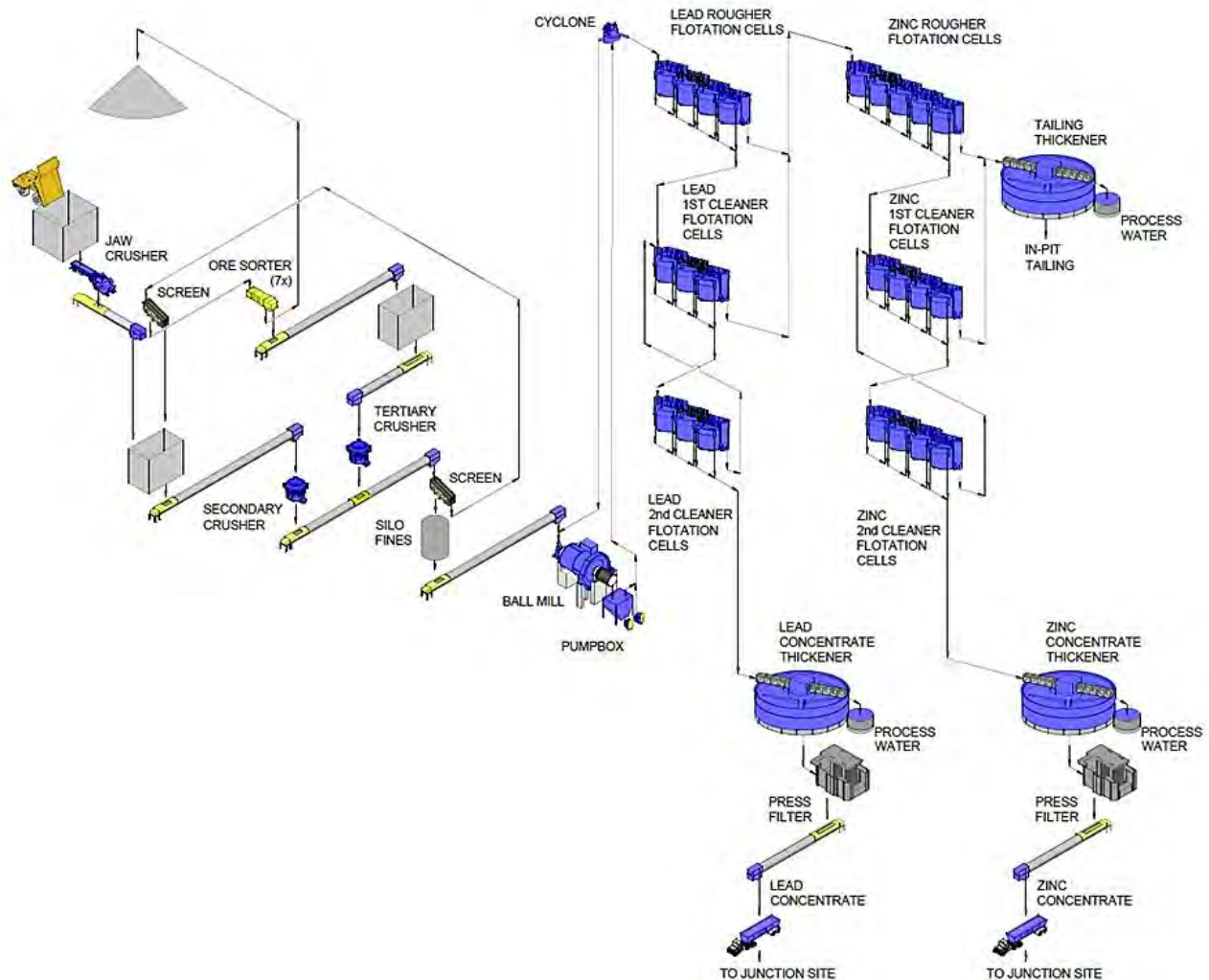


Figure 1-5: Process flowsheet

1.15 Project Infrastructure

The Pine Point Project benefits from the presence of existing infrastructure from the Cominco era and the proximity of provincial highways (Nos. 5 and 6) and railways (Mackenzie Northern Railway). The Project proposed infrastructure are located on two sites: 1) at the Pine Point Process Plant Site (mine site), beginning 42 km east from Hay River and extending over approximately 67 km of strike length parallel to the highways, with its central area being located near the historic and deconstructed Pine Point town; 2) at Pine Point Junction, 10 km south of Hay River. Figure 1-6 shows a regional overview of the Pine Point Project Area.

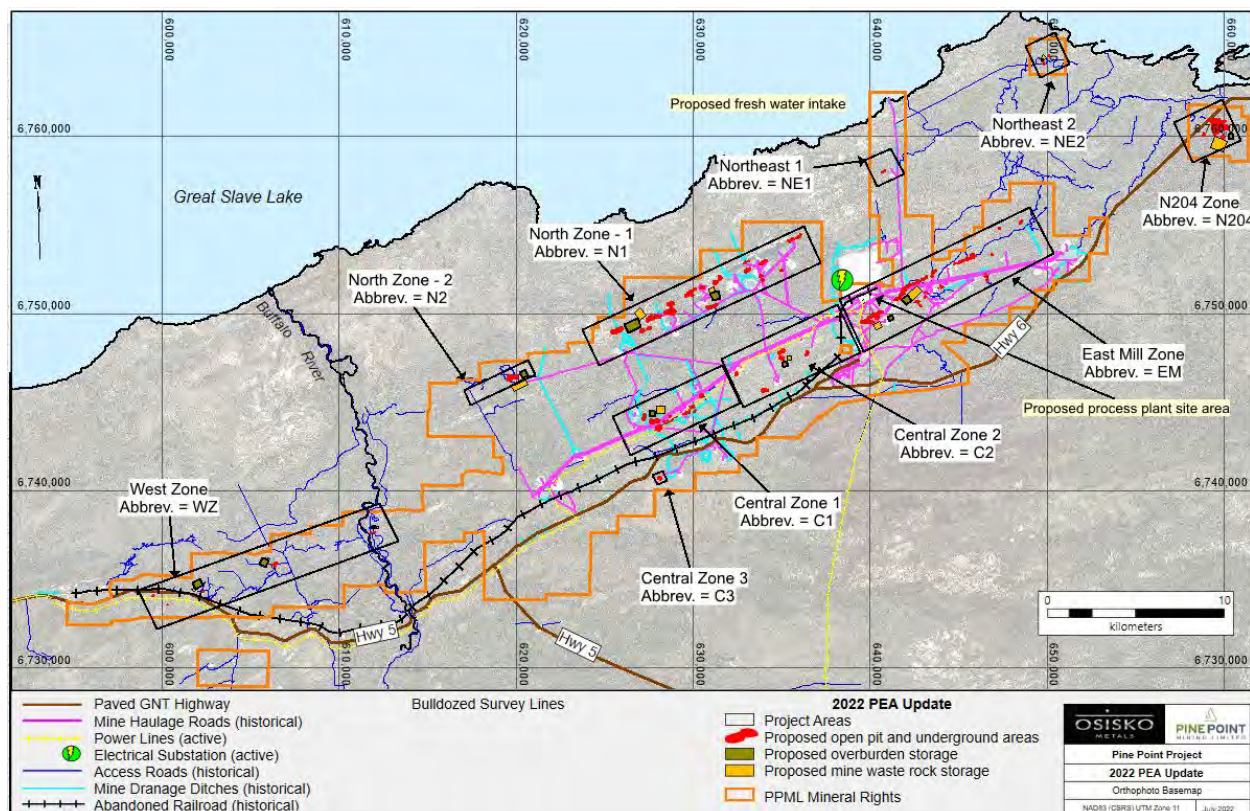


Figure 1-6: Pine Point Project area

1.15.1 Process Plant Site

1.15.1.1 Infrastructure

The Project envisions the upgrading or construction of the following Infrastructure at the Process Plant Site location:

- Gate House, parking, weigh scale and on-site roads;
- Administration, mine dry and first aid facilities;
- Camp accommodations, kitchen and recreational complex (230 persons);
- Process Plant;
- Power generation (LNG) and distribution facilities;
- Bulk fuel storage and distribution;
- Fresh water pumping station;
- Truck shop and Warehouse;
- Mobile equipment;
- Waste Rock ("WR") and overburden ("OB") storage facilities including conventional piles, as well as historical and proposed mined-out pits for waste rock management;
- Mine water management infrastructure including collection ponds, ditches, rehabilitated ponds, mobile water treatment units, pumps and pipelines;
- Mine dewatering wells, and sumps;
- Historical pits (8) used as Tailings Disposal Areas ("TDA") and their associated pumps and pipelines;
- Final effluent water treatment plant;
- West underground mine offices, lunch-room and mine dry.

1.15.1.2 Power

Demand

The total power demand for the site varies over the years of operations mainly because of the mining and resulting dewatering plan. Table 1-4 below identifies the average total power demand over the LOM.



Table 1-4: Site power demand

Year		Y1	Y2	Y3	Y4	Y5	Y6	Y7	Y8	Y9	Y10	Y11	Y12
Area	Description	Average power demand (MW)											
200	Underground Mine	-	0.06	0.53	1.01	1.35	0.83	0.88	0.08	-	-	-	-
300	Open-pit Mine	0.00	2.19	1.64	1.93	2.39	1.60	1.75	0.71	2.05	1.62	1.44	1.87
500	Site Infrastructure	0.85	0.85	0.85	0.85	0.85	0.85	0.85	0.85	0.85	0.85	0.85	0.85
600	Process plant	10.95	10.95	10.95	10.95	10.95	10.95	10.95	10.95	10.95	10.95	10.95	10.95
Total power demand (MW)		11.80	14.04	13.96	14.74	15.54	14.23	14.43	12.59	13.85	13.42	13.24	13.67

Supply and Distribution

Most of the power required for operations is at the process plant. The power could come from the local public utility, Northwest Territories Power Corporation ("NTPC"). However, NTPC does not have the production and transmission capacity required to supply all the Process facility and dewatering energy needs. Therefore, a local power generation strategy is required and has been developed to maintain mining operations annually and provide some redundancy. The power required for the site will therefore be a combination of NTPC supplied power at an annual average of 8.4MW combined with 2.6MW Photovoltaic solar farm. The remaining power would be provided by a local Liquid Natural Gas ("LNG") power plant.

The available power from NTPC will be mainly used for fixed installations such as the process plant, camp, etc., while on-site power generation equipment will be mobilized for the mining operations (dewatering and ventilation) which will be located according to pits and underground mines that are in production.

The Project will seek to benefit from an existing LNG delivery network (virtual pipeline) for industrial clients sourced out of Fort Nelson BC. This network can supply the southern area of the Northwest Territories through an existing public road network. A supply contract would be established with a local LNG provider to source, transport and transform the natural gas onsite for local generator use.

Energy cost provided by NTPC would be 0.145\$/kwh and due to its relatively low-cost, power supply from NTPC would be favoured over on-site power generation.

As for the electricity transmission and distribution, since NTPC currently owns and operates an electrical substation (138 - 12.5 kV) located near old Pine Point infrastructures, it was agreed with NTPC that the electricity required for the site would be available at 12.5 kV from two existing main breakers. From the NTPC substation, two full capacity interconnexions have been foreseen between the existing substation and the process plant facilities in order to insure reliability and flexibility for maintenance. Local distribution at the underground mines and open-pits will be built at 12.5 kV.

Comparison with 2020

The new power generation strategy enables the integration of renewables to the power mix, which cuts NG consumption by a substantial 79% compared to an all fossil-fuel based solution as proposed for the 2020 PEA

In addition to the diversification of the power mix, an optimization of power transmission and distribution have been developed. In this strategy, a definite quantity of transformers and distribution cables would be used interchangeably between the pumps and the mobile generators, which significantly cuts the costs related to power transmission and voltage step-down.

Since dewatering represented a big portion of power load in 2020, a strategy to use grouting/till injection for the underground mines would be adopted in 2022, which would cut the dewatering load for the western underground mines by almost 90%. This load reduction would be reflected on the power needs and would reduce the number of pumps and, consequently reduce the number of gensets required for underground dewatering.

1.15.2 Waste Rock and Overburden Management

WR and OB will be stored in historical and new pits as much as possible to reduce the site footprint. A total of 28 storage facilities (14 WR dumps and 14 OB piles) are required to store the remaining 52.6 Mm³ of waste rock and 36.4 Mm³ of overburden. The total WR volume produced over the LOM is estimated to be 66.0 Mm³ and the estimated total OB volume is 44.2 Mm³. Reclamation piles will also be built to store organic material. OB and reclamation piles will be used for reclamation purposes.

1.15.2.1 Tailings Management

Tailings from the processing plant will be pumped into eight historical and mined-out open-pits. Over the life of the Project, the pipeline networks will be expanded to reach the pits that have been identified to be used to store tailings. Thus, no surface tailings storage facility is required for the Project.

Reclaim water is brought back to the plant in a pipeline installed next to the tailings line.

The total of 23.9 Mm³ of tailings produced during the LOM will be disposed in eight historical or new open-pits, namely pits I46, J44, N42, A55, Z64, L37, N39 and M40. Reclaim water pumps and pipelines will allow the supernatant water in the tailings pits to be transported to the reconstructed sedimentation ponds. Thickened tailings will be discharged in the open pits up to the bedrock rim. Backfill of all open pits used for tailings disposal will be completed to ground surface with the addition of mineral sorter rejects and/or WR.

1.15.2.2 Surface Water Management

Collection ponds and ditches will be constructed for each sector to collect contact water from the WR storage facilities (WRSF) and OB piles. The water from these ponds will be directed to the nearest historical pit. If required, a mobile water treatment unit may be added to ensure water quality meets environmental regulations.

Two historical ponds will be reconstructed and used to store reclaim and mill water. These ponds are located 3 km south of the mill site and will be connected to a water treatment plant capable of receiving a flow rate of 135 m³/h, which will ensure the water quality is in accordance with regulations prior to discharging.

1.15.2.3 Mining Sites Dewatering

Prior to mining each pit, groundwater will be lowered to a depth of 10 m below the lowest current excavation level by pumping from submersible pumps in wells into ditches and discharging the water towards historical mined-out pits or surface settling ponds and eventually discharged to the Great Slave Lake in accordance with discharge limits. This is to be confirmed in more detail in subsequent stages of engineering. The dewatering requirements for each pit in terms of numbers of boreholes and pumped flows have been estimated based on numerical modelling by Hydro-Ressources Inc.

In accordance with the mine production schedule, once an open-pit is no longer in operation, the submersible pumps are planned to be relocated to another pit. Rainwater collected from adjacent operating pits would be pumped to a common booster pump located on the surface that transfers this water to a nearby receiving pond, either directly or combined with runoff from waste rock and overburden piles.

Similar to open-pits, dewatering requirements for the five underground workings have been established by HRI. The same pumping strategy with submersible pumps in wells, with ditches transferring the groundwater to previously exploited open-pits or ponds is being used, as well as the relocation to the next deposit when an underground mine is no longer in operation.

The underground workings located west of the Buffalo River will however require groundwater control by means of grout/till injection prior to dewatering, as described in Chapter 16. Some underground workings in the Central Area are expected to be fully or partially dewatered as nearby open-pits are dewatered.

1.15.3 Pine Point Junction

The following infrastructure items are to be built at Pine Point Junction:

- Concentrate transloading installation;
- Warehouse, offices, and assay laboratory;
- Railway spur lines.

1.16 Environmental, Permitting and Closure

The regulating authorities have established requirements for mine closure for projects in the Northwest Territories, which are presented in the Guidelines for the Closure and Reclamation of Advanced Mineral Exploration and Mine Sites in the Northwest Territories (MVLWB/GNWT/CIRNAC, 2022) (hereinafter, the Guidelines). For the Pine Point Project, a conceptual closure and reclamation plan is required as part of the Environmental Assessment process and is updated as necessary during the permitting phase. An Interim Closure and Reclamation Plan ("ICRP") is typically required within 12 months of the issuance of the Water Licence.

Mine site reclamation is carried out with the intent to limit as much as possible the impacts of the Project on the environment. General closure concepts presented consider the Design for Closure practice, upon which designing a mine site while considering its closure allows for anticipation and mitigation of potential environmental and social impacts. This concept also helps reduce closure costs as it reduces replicated work.

A formal process of engagement with local communities is required to develop the objectives for the closure and reclamation plan. Where possible, affected areas will be reclaimed progressively during mine operation, which will provide information regarding site-specific results with respect to the efficacy of the methods used. The progressive reclamation work completed during operations will be presented in interim ICRPs and in a final CRP, as defined in the Guidelines. An application to reduce the financial security can be made once closure objectives are achieved. Progressive reclamation reduces the time required for the final reclamation works, as the partial completion of the reclamation works will be demonstrated in every interim ICRP.

The reclamation cost for the Pine Point Project is estimated to be \$68M. This cost includes the direct and indirect costs of site reclamation and closure as well as post-reclamation monitoring.

The closure concepts proposed in this PEA are based on assumptions and hypothesis for various parameters. These assumptions and hypothesis will be confirmed in the next phases of the engineering studies and will be subjected to regulatory and community reviews.

1.17 Market Studies and Contracts

1.17.1 Metal Pricing

The long-term prices for zinc and lead were estimated on the basis of discussions with experts, consensus analyst estimates and recently published economic studies that were deemed to be credible. The forecasts used are meant to reflect the average metal price expectation over the life of the Project. Project economics were estimated based on a long-term exchange rate (CAD:USD) of 1.27 and metal prices of US\$1.37/lb for zinc, and US\$0.96/lb for lead (Charles, 2022).

1.17.2 Concentrate Logistics

Trucking and rail car loading costs have been estimated at \$13.50/wmt for both lead and zinc concentrates. The freight and logistics costs for transporting the concentrates from Pine Point Junction to North American and Asian Smelters have been estimated at \$139/wmt for zinc and at \$163/wmt for zinc based on a preliminary marketing strategy.

1.17.3 NSR Calculations

The commercial terms, which include payable metal quantities, deductions, treatment charges, refining charges and penalties, where applicable, are based on benchmarks and typical industry terms as projected by Stonehouse Consulting (Vice, 2022), which are expected to apply to products with similar characteristics to the Pine Point lead and zinc concentrates. The following payability factors have been applied:

- Zinc concentrate: pay for 85.0% of the zinc content, subject to a minimum deduction of eight units;
- Lead concentrate: pay for 95.0% of lead content, subject to a minimum deduction of three units.

The terms used in derivation of the NSR for the zinc and lead concentrate are summarized in Chapter 19. Due to the relatively low impurity levels in both the lead and zinc concentrates, no penalties are expected.

1.17.4 Contracts

There are currently no contracts in place regarding the sales of concentrates from the Pine Point Lead-Zinc Project.



1.18 Capital Cost Estimate

The total capital cost ("CAPEX") of the Pine Point Lead-Zinc Project, including pre-production and sustaining costs, is estimated at \$1,318.5M. The pre-production costs were calculated at \$653.3M, including a \$87.8M contingency based on a 16-month pre-production construction period. The sustaining costs, excluding site rehabilitation costs, were calculated at \$616.8M. Closure costs including all costs related to the closure, reclamation and ongoing monitoring are estimated to be \$68M.

The overall capital cost estimate developed in this Study meets the AACE International ("AACE") Class 4 requirements and has an accuracy range of between -30% and +30%. The capital cost estimate was compiled using a combination of budgetary quotations, database costs, and database factors. Items such as sales taxes, land acquisition, permitting, licensing, price escalation, feasibility studies and financing costs are not included in the cost estimate. The Pine Point Project capital cost summary is outlined in Table 1-5 and Figure 1-7.

Table 1-5: Project capital costs (pre-production and sustaining)

WBS	Cost area	Pre-production capital cost (\$M)	Sustaining capital cost (\$M)	Total cost (\$M)
000	General Administration (Owner's costs)	22.8	0.0	22.8
200	Underground Mine	0.0	183.3	183.3
300	Open-pit Mine	15.7	80.6	96.3
400	Electricity and Communications	45.7	19.3	64.9
500	Site Infrastructure	59.7	11.8	71.5
600	Process Plant	297.3	0.0	297.3
800	Tailings, Mine Waste and Water Management	47.7	123.6	171.3
900	Indirect Costs	76.6	0.0	76.6
999	Contingency	87.8	23.7	111.5
	Capitalized Operating Costs ¹	0.0	174.5	174.5
	Total	653.3	616.8	1,270.1
	Site Reclamation and Closure	0.0	68.0	68.0
	Salvage Value	0.0	-19.6	-19.6
	Total - Forecast to spend	653.3	665.1	1,318.5

1. The operating costs attributable to overburden and waste stripping that is in excess of the average strip ratio over the life of mine have been transferred to capital expenditures.

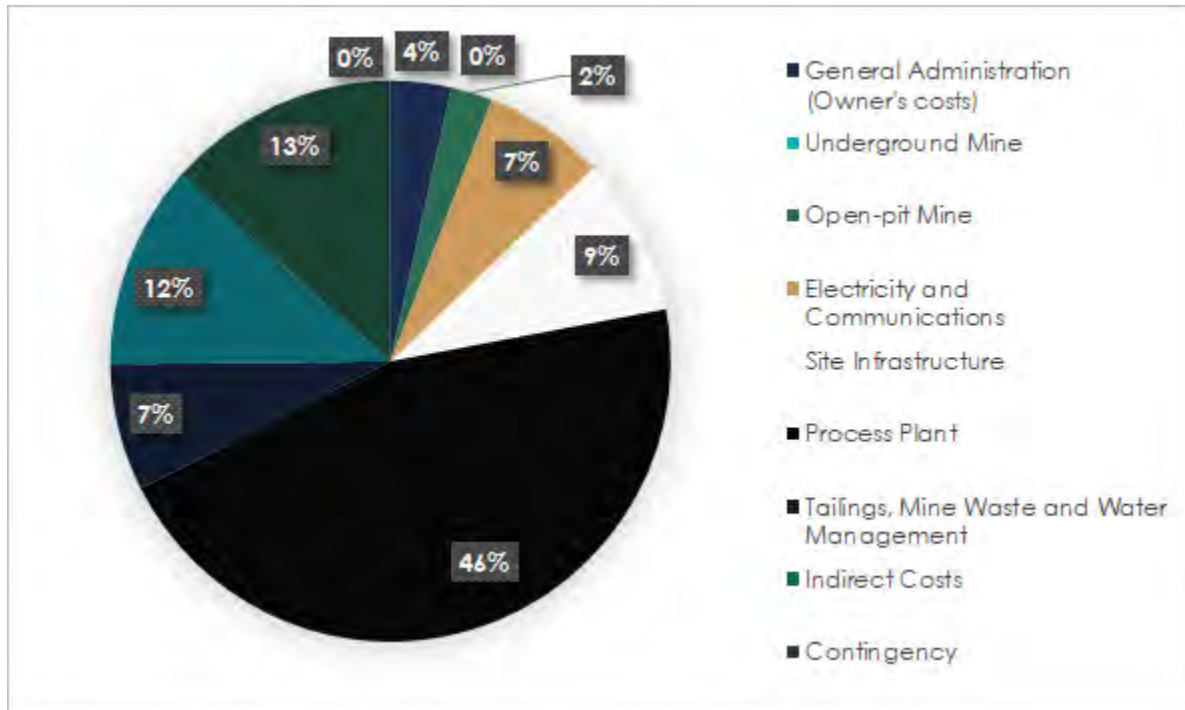


Figure 1-7: Capital cost summary (pre-production)

All capital costs for the Project have been distributed against the development schedule to support the economic cash flow model. Figure 1-8 presents the planned annual and cumulative LOM capital cost profile.

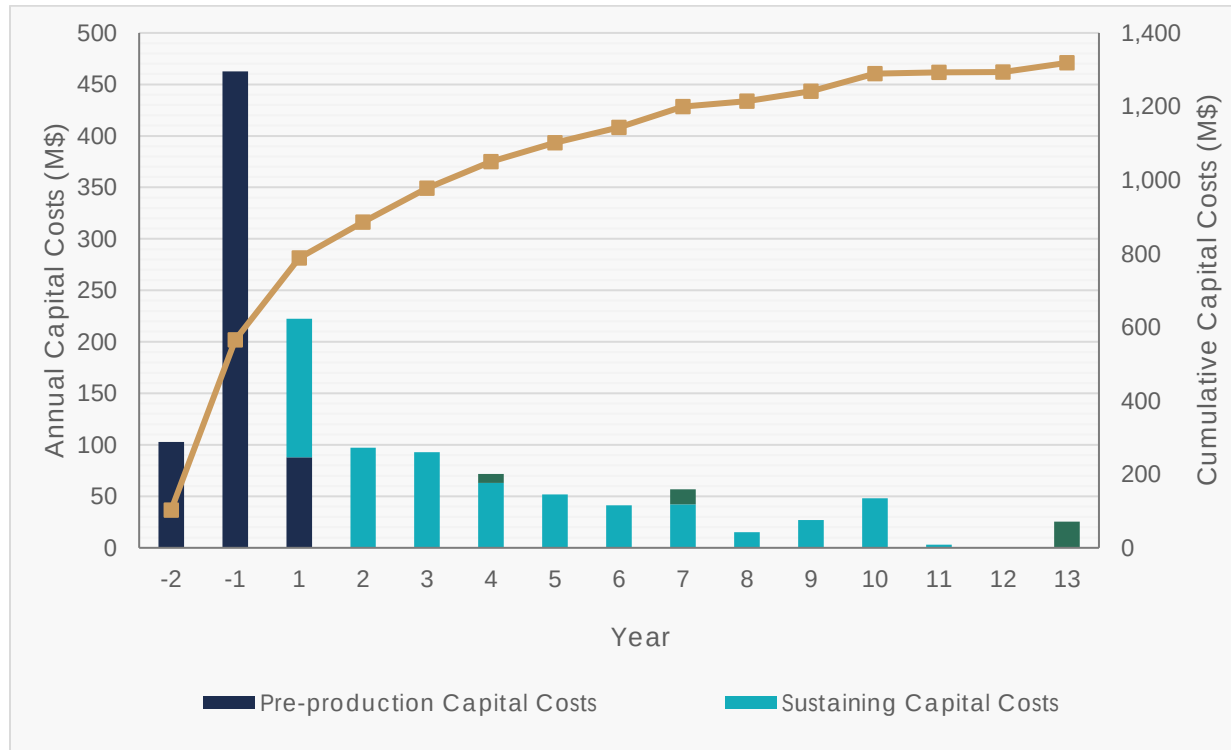


Figure 1-8: Overall capital cost profile

1.19 Operating Cost Estimate

The operating cost estimate ("OPEX") is based on a combination of experience, reference project, budgetary quotes and factors as appropriate with a preliminary study. The target accuracy of the operating cost is +/-30%. No cost escalation or contingency has been included within the operating cost estimate.

The operating cost estimate in this Study includes the costs to mine, process the mineralized material to produce lead and zinc concentrates, supplementary power generation as well as general and administration expenses ("G&A"). Underground mining will be performed on a contract basis while supplementary power is supplied through a fixed LNG power plant and rented generators which will be used to meet variable loads. The main OPEX assumptions are outlined in Table 1-6.



Table 1-6: General rate and unit cost assumptions

Parameter	Unit	Value
Average Daily Processing Tonnage	tpd	11,250
Years of Operations	year	12.0
Open-pit - Mineralized Material	Mt	37.9
West Zone - Mineralized Material	Mt	7.6
Central Zone - Mineralized Material	Mt	1.2
Total LOM - Mineralized Material	Mt	46.9
LOM Lead Grade	%	1.76
LOM Zinc Grade	%	4.39
LOM Zinc Equivalent Grade	%	5.87
NTPC Grid Power	\$/KWh	0.145
Compressed Natural Gas (delivered and transformed)	\$/GJ	18.15
Diesel Fuel	\$/L	1.41
LOM Average Manpower (including Contractors)	Employees	485

The average operating cost over the 12-year mine life of the Pine Point Lead-Zinc Project is estimated to be \$191 M/y or \$48.91/t milled. Total LOM and unit operating cost estimates are summarized in Table 1-7 and shown as a pie chart by percentage in Figure 1-9.

Table 1-7: Project operating costs

Cost area	LOM (\$M)	Annual average cost (\$M)	Average LOM (\$/t processed)	OPEX (%)
Open-pit Mining ⁽¹⁾	843.1	70.3	17.98	37
Underground Mining (West and Central Zones) ⁽²⁾	376.4	31.4	8.03	16
Processing	575.3	47.9	12.27	25
Power	216.3	18.0	4.61	9
Tailings, Waste and Water Management	76.4	6.4	1.63	3
General and Administration	380.6	31.7	8.11	17
Capitalized Operating Costs ⁽³⁾	-174.5	-14.5	-3.72	-8
Total (mine site)	2,293.6	191.1	48.91	100

1. Open-pit mining = \$3.36/t mined
2. West Zone underground mining (Years 1 to 10) = \$40.32/t mined, Central Zone underground mining (Years 7 to 10) = \$53.92/t mined
3. The operating costs attributable to overburden and waste stripping that is in excess of the average strip ratio over the life of mine have been transferred to capital expenditures.

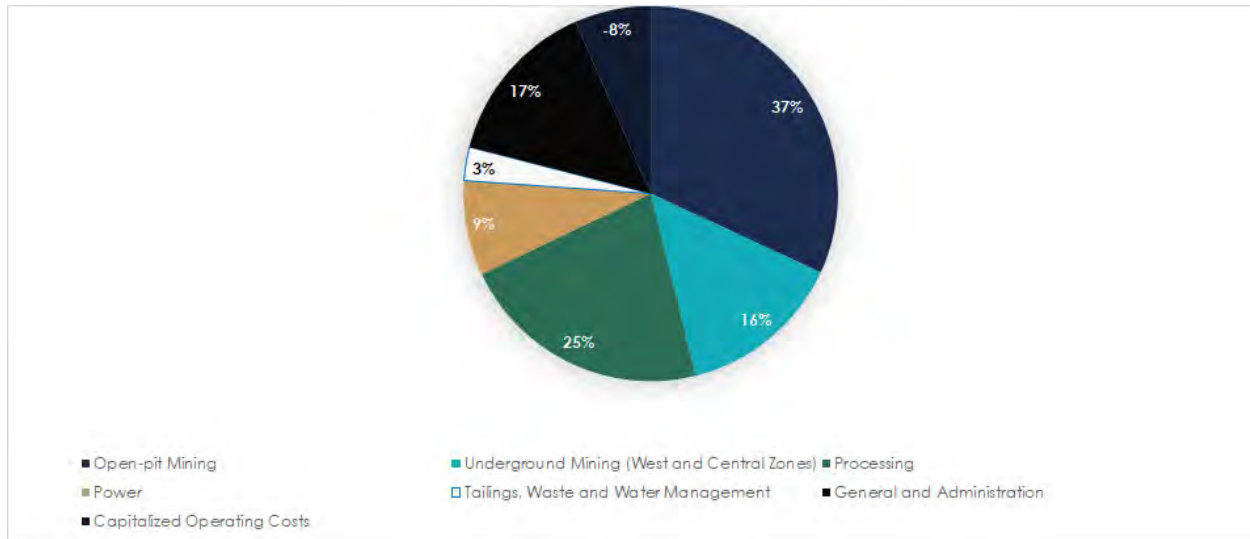


Figure 1-9: Operating cost summary (by area)

Over the life of mine, it is anticipated that 485 employees (staff, labour and contractors) will be required for operations. Employees will be sourced regionally or on a fly-in/fly-out ("FIFO") basis from Edmonton and Yellowknife. Table 1-8 provides a summary of the employees by facility area.

Table 1-8: Employee summary – all areas

Facility area	FIFO	Local	Total
General & Administration	21	44	65
Open-pit Mining	211	74	285
Underground Mining	39	2	41
Power Generation and Supply	8	2	10
Process Plant	43	23	66
Water and Waste Management	10	8	18
Overall Project	332	153	485
Contractors	35	1	36
Pine Point Employees	297	152	449

1.20 Project Economics

1.20.1 Cautionary Note

Certain information and statements contained in this section and in the Report are “forward looking” in nature. Forward-looking statements include, but are not limited to, statements with respect to the economic and study parameters of the Project; Mineral Resource Estimates; the cost and timing of any development of the Project; the proposed mine plan and mining methods; dilution and extraction recoveries; processing method and rates and production rates; projected metallurgical recovery rates; infrastructure requirements; capital, operating and sustaining cost estimates; the projected life of mine and other expected attributes of the Project; the net present value (“NPV”) and internal rate of return (“IRR”) and payback period of capital; capital; future metal prices; the timing of the environmental assessment process; changes to the Project configuration that may be requested as a result of stakeholder or government input to the environmental assessment process; government regulations and permitting timelines; estimates of reclamation obligations; requirements for additional capital; environmental risks; and general business and economic conditions.

All forward-looking statements in this Report are necessarily based on opinions and estimates made as of the date such statements are made and are subject to important risk factors and uncertainties, many of which cannot be controlled or predicted. Material assumptions regarding forward-looking statements are discussed in this Report, where applicable. In addition to, and subject to, such specific assumptions discussed in more detail elsewhere in this Report, the forward-looking statements in this Report are subject to the following assumptions:

- There being no significant disruptions affecting the development and operation of the Project;
- The availability of certain consumables and services and the prices for power and other key supplies being approximately consistent with assumptions in the Report;
- Labour and materials costs being approximately consistent with assumptions in the Report;
- The timelines for prior consultation and seasonal baseline data collection being generally consistent with PEA assumptions;
- Permitting and arrangements with stakeholders being consistent with current expectations as outlined in the Report;
- All environmental approvals, required permits, licenses and authorizations will be obtained from the relevant governments and other relevant stakeholders;

- Certain tax rates, including the allocation of certain tax attributes, being applicable to the Project;
- The availability of financing for Osisko Metal's planned development activities;
- The timelines for exploration and development activities on the Project;
- Assumptions made in Mineral Resource Estimate and the financial analysis based on that estimate, including, but are not limited to, geological interpretation, grades, commodity price assumptions, extraction and mining recovery rates, hydrological and hydrogeological assumptions, capital and operating cost estimates, and general marketing, political, business and economic conditions.

The production schedules and financial analysis annualized cash flow table are presented with conceptual years shown. Years shown in these tables are for illustrative purposes only. If additional mining, technical, and engineering studies are conducted, these may alter the Project assumptions as discussed in this Report and may result in changes to the calendar timelines presented.

The PEA is partly based on Inferred Mineral Resources that are considered too speculative geologically to have the economic considerations applied to them that would enable them to be categorized as Mineral Reserves, and there is no certainty that the PEA based on these mineral resources will be realized.

1.20.2 Methodology and Input Parameters

The economic/financial assessment of the Pine Point Lead-Zinc Project was carried out using a discounted cash flow approach on a pre-tax and after-tax basis, based on Q2-2022 metal price projections in United States currency and cost estimates (CAPEX and OPEX) in Canadian currency. Inflation or cost escalation factors were not taken into account. An exchange rate (CAD:USD) of 1.27 has been assumed over the life of the Project. The base case zinc and lead prices are US\$1.37/lb and US\$0.96/lb, respectively. A construction period of 16 months was considered and conceptually planned to start in Q4 2026 (Year -2) for the overall completion in Q1 2028. Year 1 (2028) corresponds to the first year of production. The input parameters and results of the financial analysis are presented in Table 1-9.

1.20.3 Results

Over the 12-year operational period of the Project, the process plant will produce of a total of 1,687 Mlb of Lead (Average annual: 141 Mlb) in concentrate and 3,951 Mlb of Zinc (Average annual: 329 Mlb) in concentrate. On an equivalent Zinc (ZnEq) basis, a total of 5,366 Mlb will be



produced in concentrate. Figure 1-10 provides a forecast summary of the lead and zinc production by year.

The pre-tax base case financial model resulted in an internal rate of return of 35.1% and an NPV of \$1,099M with a discount rate of 8%. The simple pre-tax payback period is 4.8 years.

On an after-tax basis, the base case financial model resulted in an internal rate of return of 24.8% and an NPV of \$602M with a discount rate of 8%. The simple after-tax payback period is 5.8 years.

The LOM mine site operating cost per lb is 0.34 USD/lb. C1 Cash costs (Mine Site + Transport and Refining + Royalties) are estimated at 0.61 USD/lb while the LOM all-in sustaining cost ("AISC") is 0.71 USD/lb derived from the total cash costs plus sustaining capital, and closure costs. All in costs including on-site, off-site and required project capital is estimated at 0.80 USD/lb.

Table 1-9: Financial analysis summary

Description	Unit	Value
Long Term Lead Price	USD/lb	0.96
Long Term Zinc Price	USD/lb	1.37
Exchange Rate	CAD:USD	1.27
Discount Rate	%	8%
Mine Life	years	12
Total Material Processed (LOM)	Mt	46.90
Lead Grade (LOM)	%	1.76
Zinc Grade (LOM)	%	4.39
ZnEq Grade (LOM)	%	6.15
Overall Lead Recovery (LOM)	%	92.9
Overall Zinc Recovery (LOM)	%	87.0
Total Lead Production (in concentrate)	Mlb	1,687
Total Zinc Production (in concentrate)	Mlb	3,951
Total Zinc Equivalent Production (in concentrate)	Mlb	5,366
Average Annual Lead Production (in concentrate)	Mlb per year	141
Average Annual Zinc Production (in concentrate)	Mlb per year	329
Open-pit Mining Operating Cost	\$/t mined	3.36
Underground Mining Operating Cost (West Zone)	\$/t mined	40.32
Underground Mining Operating Cost (Central Zone)	\$/t mined	53.92
Processing Operating Cost	\$/t processed	12.27
Power Production Operating Cost	\$/t processed	4.61
Tailings, Waste & Water Management Operating Cost	\$/t processed	1.63



Description	Unit	Value
General and Administration Operating Cost	\$/t processed	8.11
Royalties	% NSR	3.0
Pre-production Capital Cost	\$M	653.3
Sustaining Capital Cost	\$M	616.8
Reclamation and Closure Cost	\$M	68.0
Salvage Value	\$M	19.6
Pre-Tax Summary		
Net Present Value (0% disc)	\$M	2,328.6
Net Present Value (8% disc)	\$M	1,098.7
Internal Rate of Return	%	35.1
Simple Payback Period	years	4.8
Payback Period (after start of operations)	years	2.8
After-Tax Summary		
Net Present Value (0% discount rate)	\$M	1,436.0
Net Present Value (8% discount rate)	\$M	602.3
Internal Rate of Return	%	24.8
Simple Payback Period	years	5.8
Payback Period (after start of operations)	years	3.8

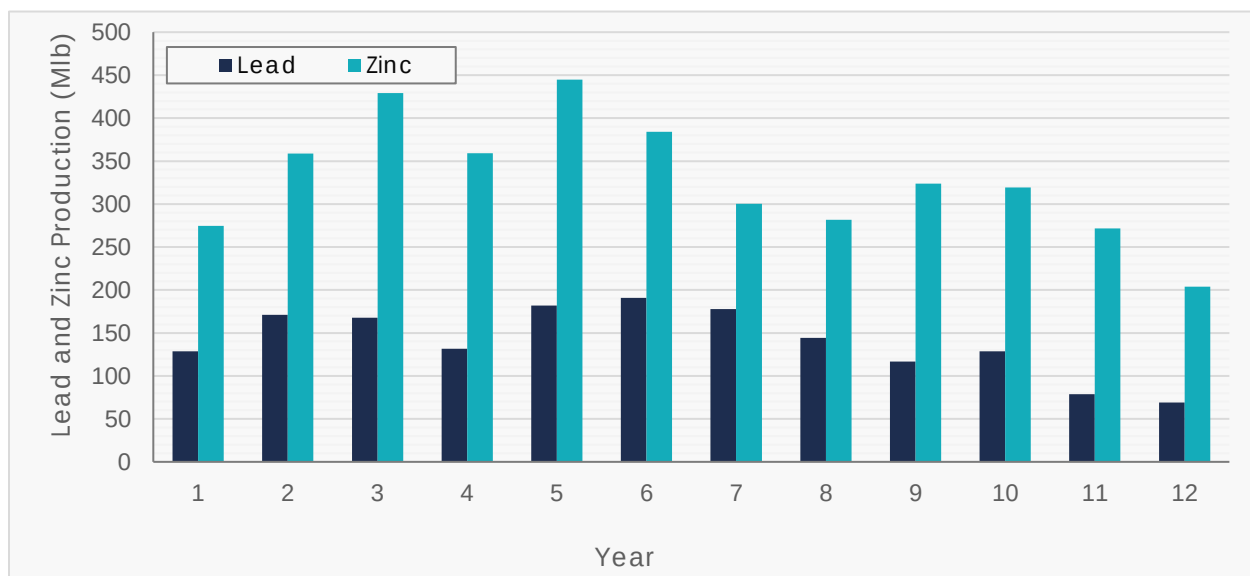


Figure 1-10: Annual lead and zinc production (Mlb)

A financial sensitivity analysis was conducted on the Pine Point Lead-Zinc Project's after tax NPV and IRR using the following variables: initial capital cost, operating costs, CAD:USD exchange rate, lead and zinc feed grades and the price of lead and zinc.

The graphical representations of the financial sensitivity analysis on NPV and IRR are depicted in Figure 1-11 and Figure 1-12. The sensitivity analysis reveals that the CAD:USD exchange rate and Zinc price has the most significant influence on both NPV and IRR compared to the other parameters, based on the range of values evaluated. It should be noted that the economic viability of the Project will not be significantly impacted by variations in the capital cost, within the margins of error associated with the PEA capital cost estimate.

Overall, the NPV and IRR of the Pine Point Lead-Zinc Project are generally positive over the range of values used for the sensitivity analysis except when the CAD:USD exchange rate is at par.

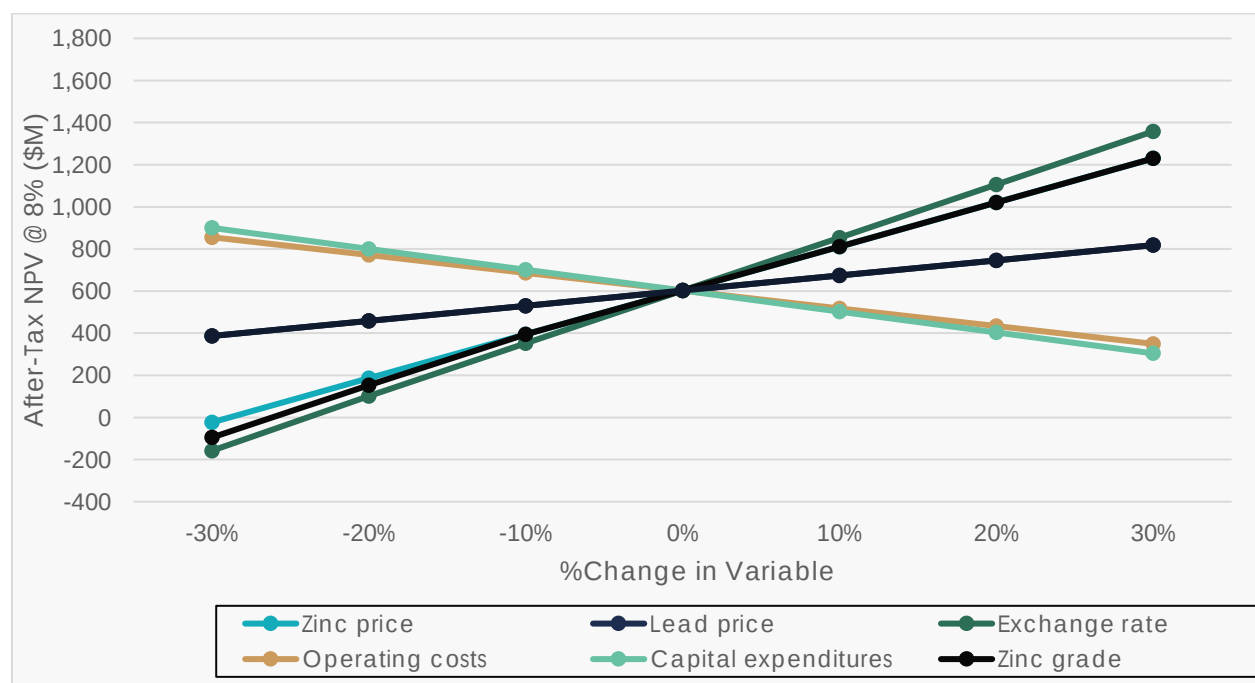


Figure 1-11: Net present value (8% discount rate, after-tax) sensitivity analysis

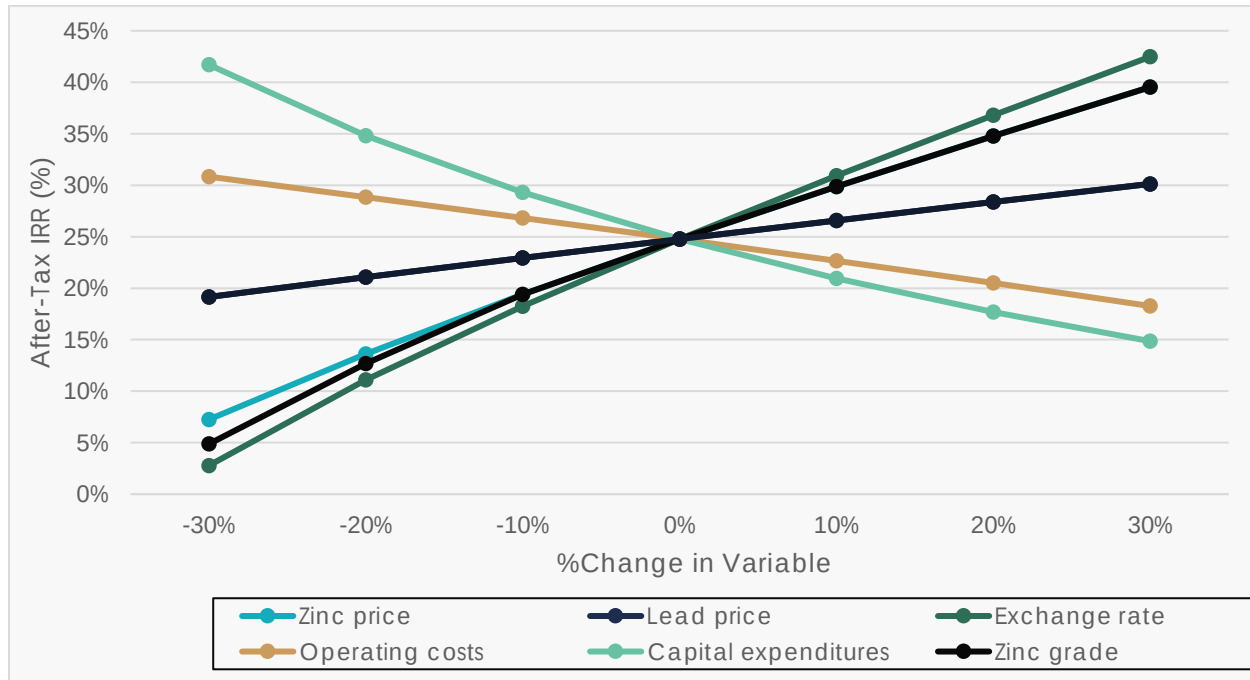


Figure 1-12: Internal rate of return (after-tax) sensitivity analysis

1.21 Project Schedule and Organization

The execution of the Pine Point Project will be directly managed by the Osisko Metals Construction Management Team ("CMT"). The engineering and construction work will be contracted out to qualified firms and contractors under the direct supervision of CMT. Procurement and Project control functions such as scheduling, cost control, Project logistics and site supervision will be executed directly by CMT personnel.

Contractors will carry out all Project development activities. The preliminary on-site workforce requirement for construction, including infrastructure and the process plant is expected to average 285 personnel with a peak at approximately 350.

The major Project activity milestones are presented in Table 1-10. For the purposes of this conceptual schedule and alignment with the mining and CAPEX/OPEX expenditure schedules, 2028 is considered year one (yr 1) of the twelve-year LOM. The Project critical path runs through the completion of the Developer's Assessment Report ("DAR"), the Environmental Assessment Certificate ("EAC") application process and the reception of the EAC, the detail engineering, the procurement and construction of the process plant facility.



Table 1-10: Key Project activities (preliminary)

Activity	Start Date	Completion Date
Complete PEA Update Study		Q3 2022
Developers Assessment Report	Q3 2022	Q2 2024
Complete Feasibility Study	Q2 2023	Q3 2024
Process plant detailed engineering	Q1 2025	Q2 2026
Project EAC* application and reception of Certificate	Q2 2024	Q4 2026
Process plant construction	Q4 2026	Q1 2028
Pre-production mine development	Q2 2027	Q3 2027
End of process plant construction/plant commissioning		Q1 2028
Process plant fully operational (commercial production)		Q3 2028

1.22 Interpretations and Conclusions

This Study was prepared by a group of independent consultants to demonstrate the technical and economic viability of a Zn/Pb project based on a mix of open-pit and underground mining and a process plant treating 11,250 tpd of mineralized material. This Report provides a summary of the results and findings from each major area of investigation. Standard industry practices, equipment and processes were used in this Study. To date the QPs are not aware of any unusual or significant risks or uncertainties that could materially affect the reliability or confidence in the Pine Point Lead-Zinc Project based on the information available.

The results of the Study indicate that the proposed Project has technical and financial merit using the base case assumptions. The results are considered sufficiently reliable to guide Osisko Metals' management in a decision to advance to the next phase of the Project development: that being the initiation of a feasibility study.

1.22.1 Project Risks and Opportunities

An analysis of the results of the investigations has identified a series of risks and opportunities associated with each of the technical aspects considered for the development of the Pine Point Project.

The most significant potential risks associated with the Project are:

- Mine dewatering volumes could be greater or lower than estimated;
- Commodity price and exchange rate fluctuations;

- Planning and operational challenges of operating a mine site consisting of multiple open pits and underground mines.

Many of the previous noted risks are common to most mining projects, many of which can be mitigated, at least to some degree, with adequate engineering, planning and pro-active management.

There are numerous opportunities that could improve the economics, timing and/or permitting potential of the Project.

The key opportunities that have been identified at this time are as follows:

- There is significant exploration potential for discoveries around the Project area. The possibility exists to increase resources and extend mine life;
- Further definition drilling may convert some of the existing Inferred Mineral Resources to the Indicated or Measured Resource categories;
- Additional metallurgical testing could potentially improve recoveries and concentrate grades;
- Implementation of progressive reclamation;
- Capacity upgrades to the NTPC network to supply more power to the Project would reduce capital and operating costs significantly.

1.23 Recommendations

In the opinion of the various QPs, the concept proposed for the Pine Point Project within the PEA is technically feasible and has positive economics. The Project warrants consideration for advancement to feasibility level engineering by Osisko Metals. This more advanced study will further detail:

- A mineral reserve estimate;
- More detailed dewatering studies;
- Process engineering;
- Project schedule;
- Capital and operating cost estimates;

The study will improve the confidence in the Project concept and design. It will also provide improved accuracy of the Project's economics and execution plan.

It is estimated that the exploration program, field studies, support work and the feasibility study would cost approximately \$62 M.

2. INTRODUCTION

In January 2022, Osisko Metals Incorporated ("Osisko Metals" or the "Company") through its wholly-owned subsidiary Pine Point Mining Limited ("PPML") requested that BBA Engineering Inc. ("BBA") lead a group of consulting firms including PLR Resources Inc. ("PLR"), WSP Canada Inc. ("WSP") and Hydro-Ressources Inc. ("HRI") to perform an update of the Mineral Resource Estimate and compile a NI 43-101 technical report summarizing the results of a updated Preliminary Economic Assessment ("PEA") study for the Pine Point Lead-Zinc Project ("Project") located in the Northwest Territories, on the south shore of Great Slave Lake, approximately 42 km east of the town of Hay River.

2.1 Scope of Study

The following Technical Report (the "Report") presents the results of an updated Preliminary Economic Assessment based on the updated 2022 Mineral Resource Estimate ("2022 MRE") for the Pine Point Lead-Zinc 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 – Preliminary Economic Assessment", was prepared by Qualified Persons ("QPs") following the guidelines of the National Instrument 43-101 ("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.

■ Hugo Latulippe, P. Eng.	BBA Engineering Inc.
■ Colin Hardie, P. Eng.	BBA Engineering Inc.
■ Pierre-Luc Richard, P. Geo.	PLR Resources Inc.
■ Simon Latulippe, P. Eng.	WSP Canada Inc.



- Paul Gauthier, P. Eng. WSP Canada Inc.
- Zakaria Moctar, P. Eng. WSP Canada Inc.
- Trent Purvis, P. Eng. WSP Canada Inc.
- Michael Verreault, P. Eng. Hydro-Ressources 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	Chapter and Section Responsibilities
1.	Executive Summary	C. Hardie	BBA	Integrator of Chapter 1. All QPs contributed based on their respective scope of work and the Chapters/Sections under their responsibility.
2.	Introduction	C. Hardie	BBA	All Chapter 2
3.	Reliance on other Experts	C. Hardie	BBA	All Chapter 3
4.	Project Property Description and Location	C. Hardie	BBA	All Chapter 4
5.	Accessibility, Climate, Local Resource, Infrastructure and Physiography	C. Hardie	BBA	All Chapter 5
6.	History	P.-L. Richard	PLR	All Chapter 6
7.	Geological Setting and Mineralization	P.-L. Richard	PLR	All Chapter 7
8.	Deposit Types	P.-L. Richard	PLR	All Chapter 8
9.	Exploration	P.-L. Richard	PLR	All Chapter 9
10.	Drilling	P.-L. Richard	PLR	All Chapter 10
11.	Sample Preparation, Analyses and Security	P.-L. Richard	PLR	All Chapter 11
12.	Data Verification	P.-L. Richard	PLR	All Chapter 12
13.	Mineral Processing and Metallurgical Testing	C. Hardie	BBA	All Chapter 13



Chapter	Description	Qualified Person	Company	Chapter and Section Responsibilities
14.	Mineral Resource Estimate	P.-L. Richard	PLR	All Chapter 14 except Section 14.15
		H. Latulippe	BBA	Section 14.15
15.	Mineral Reserve Estimate	C. Hardie	BBA	All Chapter 15. Not applicable for a PEA.
16.	Mining Methods	Z. Moctar	WSP	Sections 16.1, 16.2, 16.4.1, 16.4.2, 16.4.3, 16.4.6, 16.4.7, 16.4.8, 16.4.9, and 16.6
		P. Gauthier	WSP	Sections 16.5
		M. Verreault	HRI	Sections 16.3, 16.4.3, 16.4.4, and 6.4.5
17.	Recovery Methods	C. Hardie	BBA	All Chapter 17.
18.	Project Infrastructure	C. Hardie	BBA	Sections 18.1, 18.2, 18.2.1, 18.2.2, 18.2.3 (Process Plant only) and 18.2.4
		T. Purvis	WSP	Sections 18.2.3 (except Process Plant), 18.2.5, 18.2.7 (partial), 18.2.9 and 18.3
		S. Latulippe	WSP	Sections 18.2.6 and 18.2.7 (partial) and 18.2.8
19.	Market Studies and Contracts	C. Hardie	BBA	All Chapter 19. Stonehouse Consulting (Vice, 2022) provided transport and smelting terms.
20.	Environmental Studies, Permitting, and Social or Community Impact	S. Latulippe	WSP	All Chapter 20 except section 20.7
		C. Hardie	BBA	Section 20.7
21.	Capital and Operating Costs	C. Hardie	BBA	Sections 21.1.1, 21.1.2, 21.1.3, 21.1.4, 21.1.5.1, 21.1.5.2, 21.1.5.3, 21.1.5.5, 21.1.5.7, 21.1.5.8, 21.1.5.9, 21.1.6, 21.1.6.3, 21.1.6.7, 21.2.1, 21.2.3, 21.2.4, 21.2.6, and 21.2.7
		Z. Moctar	WSP	Sections 21.1.5 (partial), 21.1.5.2, 21.1.6.1, 21.1.6.2, 21.1.6.1, 21.1.6.2 and 21.2.2
		S. Latulippe	WSP	Sections 21.1.5.6 (partial), 21.1.6.5 (partial), 21.1.6.6, and 21.2.5
		T. Purvis	WSP	Sections 21.1.5.4, 21.1.5.6 (partial), 21.1.6.4, and 21.1.6.5 (partial)
22.	Economic Analysis	C. Hardie	BBA	All Chapter 22. OM provided metal prices, exchange rates and royalty costs (Charles, 2022). WSP provided closure costs. OM provided project taxes and after-tax cash flows.
23.	Adjacent Properties	P.-L. Richard	PLR	All Chapter 23



Chapter	Description	Qualified Person	Company	Chapter and Section Responsibilities
24.	Other Relevant Data and Information	C. Hardie	BBA	All Chapter 24. Schedule and execution plan developed by BBA based on inputs from all contributors and OM.
25.	Interpretation and Conclusions	C. Hardie	BBA	Integrator of Chapter 25. All QPs contributed based on their respective scope of work and the Chapters/Sections under their responsibility.
26.	Recommendations	C. Hardie	BBA	Integrator of Chapter 26. All QPs contributed based on their respective scope of work and the Chapters/Sections under their responsibility.
27.	References	C. Hardie	BBA	Integrator of Chapter 27. All QPs contributed based on their respective scope of work and the Chapters/Sections under their responsibility.

2.3 Effective Dates and Declaration

This Report is in support of the Osisko Metals press release, dated July 13, 2022 titled “OSISKO METALS RELEASES POSITIVE UPDATE FOR PINE POINT PEA AFTER-TAX NPV OF C\$602M AND AFTER-TAX IRR OF 25%” as well as an amended press release issued on August 29, 2022 with non-material corrections. The overall effective date of the report is July 11, 2022. The Report has several close-out dates for information:

- Drill Database close-out date: December 31, 2019;
- Metallurgical testwork close-out date: December 9, 2020;
- Effective date of the Mineral Resource Estimate: March 10, 2022;
- Date of the financial analysis: July 11, 2022;
- Mineral Lease and Claim Status: August 8, 2022.

This Report was prepared as a National Instrument 43-101 Technical Report for Osisko Metals by Qualified Persons from BBA Engineering Inc., PLR Resources Inc., WSP Canada Inc., and Hydro-Ressources 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 use of this Report by any third party is at that party's sole risk.

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.

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

2.4.1 General

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. Sections from reports authored by others may have been directly quoted or summarized in the report and are so indicated, where appropriate.

This updated MRE and PEA 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 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).

2.4.2 Specialist Input – BBA

The following individuals/groups provided specialist input to Colin Hardie, QP:

- Osisko Metals and its external advisors provided an estimate for the Owner's costs and indirect costs used in the development of the Project's pre-production capital cost estimate found in Chapter 21;
- Houssam Elwafi (BBA) and Jean-Philippe Castonguay (BBA) provided an estimate and input for the site electrical power generation and infrastructure costs in Chapters 18 and 21;
- Osisko Metals and its external advisors provided preliminary metallurgical design criteria and mass balances used in the development of the Project Process Plant found in Chapter 17;
- Helin Girgin (BBA) provided an estimate for the Project's process operational costs (OPEX, Chapter 21);
- James Alarcon (BBA) and Salah El Ghali (BBA) provided the industrial standards, norms and factors for the various material, manpower and construction costs used in the development of the process plant and overall indirect capital costs (Chapter 21);
- Vincent Clement (BBA) provided the estimate for the GHG assessment as summarized in Chapter 20;
- Claude Catudal (BBA) provided input to the Project's execution strategy and schedule as summarized in Chapter 24.

These specialists are not considered as QPs for the purposes of this NI 43-101 Report.

2.4.3 Specialist Input – PLR

The following individuals/groups provided specialist input to Pierre-Luc Richard, QP:

- Osisko Metals provided the drillhole database, property boundary, topographic surfaces, and historical workings;
- Christina Thouvenot (formerly BBA) participated in the 3D and block modelling aspects of the Project;

- Clovis Auger (BBA) participated in the validation of the database and part of the statistical analysis;

These specialists are not considered as QPs for the purposes of this NI 43-101 Report.

2.4.4 Specialist Input – WSP

The following individuals/groups provided specialist input to Zakaria Moctar, QP:

- Osisko Metals and its external advisors provided preliminary mineralized material throughput from open-pit and underground as well as the preliminary mining production schedule;
- Osisko Metals and its external advisors provided preliminary economic parameters;
- Osisko Metals provided preliminary MSO for underground workings;
- Hydro Ressources provided dewatering rate estimates required for the completion of Chapters 16, 18, 20 and 21;
- Peter White provided grouting methodology and cost estimate for the completion of Chapters 16 and 21;
- Paul Gauthier (WSP) and Vijay Sharma (WSP) provided the underground mining methodology (Chapter 16) and an estimate for the Project's underground capital and operational costs (Chapter 21). They also contributed to Chapter 1, Chapter 25 and Chapter 26;
- Paul Gauthier (WSP) reviewed the input for ventilation requirements and network in Chapter 16.

The following individuals provided specialist input to Mr. Simon Latulippe, QP:

- Osisko Metals and its external advisors provided preliminary waste rock, overburden mineral and tailings volume in the development of the Project;
- Kristine Mason (WSP) provided inputs to the environmental studies, permitting, and social or community impacts, as summarized in Chapter 20;
- John Faithful (WSP) provided inputs to the environmental studies, permitting, and social or community impacts, as summarized in Chapter 20;
- Audrey Bédard (WSP) provided calculations, design and cost estimate for waste rock and overburden stockpiles, tailings management and closure, Chapters 18 and 20;
- Patrick Couture (WSP) and Slim Kouki (WSP) provided hydrology calculations and cost estimate for the Water Treatment Plant (Chapter 18).

The following individuals provided specialist input to Paul Gauthier, QP:

- Trent Purvis, P. Eng (WSP) provided the dewatering pumping/piping sizing calculations and cost estimates for Chapters 18 and 21.
- Vijay Sharma (WSP) provided the underground mining methodology (Chapter 16) and an estimate for the Project's underground capital and operational costs (Chapter 21). They also contributed to Chapter 1, Chapter 25 and Chapter 26;

The following individuals provided specialist input to Trent Purvis, QP:

- Boris Urieu (WSP) provided the review of dewatering flow rates and mine plan staging;
- Michael Porretta (WSP) provided dewatering pump and pipeline sizing.

These specialists are not considered as QPs for the purposes of this NI 43-101 Report.

2.5 Site Visit

The following bulleted list describes which Qualified Persons visited the Pine Point Project site(s), the date of the visit, and the general objective of the visit:

- Pierre-Luc Richard (PLR) visited the Pine Point Lead-Zinc Project site from August 9 to August 11, 2018, from October 18 to 20, 2019, as well as from October 29 to 31, 2021. The purpose of the visits was to review the Pine Point Project with the PPML Geology team. The visits included overviews of the general geological conditions and Project, a tour of the core storage facility, and visual inspections of 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 (BBA) visited the Project on October 9, 2019. The purpose of the visit was to review the Pine Point Project with PPML staff and Osisko Metal's management team. The visit included an overview of the Project's geology and a tour of the exploration camp, historical open-pits and general site infrastructure.
- Michael Verreault (Hydro-Ressources) visited the Project on the week of June 14, 2022. During the visit Mr. Verreault completed a tour of most historical pits and went to the proposed open pit and underground mine areas. Multiple observations were made, including structural geology, water occurrences, historical wells and piezometers, etc.

As of the effective date of this Report, the following QPs have not visited the Project site:

- Hugo Latulippe (BBA);
- Simon Latulippe (WSP);

- Trent Purvis (WSP);
- Paul Gauthier (WSP);
- Zakaria Moctar (WSP)

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 1.27 CAD for 1.00 USD 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;
- All metal prices are expressed in US dollars ("USD" or "US\$");
- All cost estimates have a base date of the second quarter ("Q2") of 2022.

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 Previous Technical Reports

This technical report supersedes all previous technical reports issued for the Pine Point Project. Osisko Metals has previously filed the following technical reports:

- Cassoff, J., Hardie, C., Richard, P-L., Latulippe, H., Latulippe, S. and Poirier, E. 2020: Technical Report titled "Preliminary Economic Assessment, Pine Point Lead-Zinc Project, Hay River, Northwest Territories, Canada", (2020 PEA) prepared by BBA Inc. for Osisko Metals, effective date June 11, 2020;
- Cassoff, J., Hardie, C. and Richard, P-L. 2019: Technical Report titled "Mineral Resource Estimate Update, Pine Point Lead-Zinc Project, Hay River, Northwest Territories, Canada", (2019 MRE) prepared by BBA Inc. for Osisko Metals, effective date November 10, 2019;
- Cassoff, J., Hardie, C. and Richard, P-L. 2018: Technical Report titled "Mineral Resource Estimate for the Pine Point Lead-Zinc Project, Hay River, Northwest Territories, Canada", (2018 MRE) prepared by BBA Inc. for Osisko Metals, effective date December 20, 2018.



2.8 Acknowledgments

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 & COO – Osisko Metals and Pine Point Mining Limited;
- Xavier Pouchain, Project Manager – Pine Point Mining Limited;
- Robin Adair, Vice President Exploration – Osisko Metals;
- Andrew Williams, Environment Manager– Osisko Metals;
- Killian Charles, Vice President Corporate Development – Osisko Metals;
- Mary Norman, Project Assistant – BBA;
- Shane K. A. Ghouralal, Regional Director Mining & Metals Studies – BBA.

Their commitment, contributions and teamwork 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, surface rights, property agreements, royalties, taxes and Zinc/Lead commodity markets.

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. Colin Hardie, QP, consulted the Northwest 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 support of Chapter 4 (Property Description and Location), Chapter 14 (Mineral Resource Estimate) and Chapter 22 (Economic Analysis).

3.3 Duties, Royalties and Taxation

Colin Hardie, QP, has fully relied upon, and disclaims responsibility for, information supplied by Osisko Metals staff and experts retained by Osisko Metals for information related to mining duties, taxation and royalties as applied to the financial model. This information is used in support of the financial analysis in Chapter 22 (Economic Analysis).



3.4 Markets

Colin Hardie, QP, has fully relied upon, and disclaims responsibility for marketing information on zinc and lead concentrates, transport and logistics and smelter terms information derived from Osisko Metals staff and experts retained by Osisko Metals through the following documents:

- "Pine Point Zinc and Lead Pricing" by Killian Charles, VP Corporate Development of Osisko Metals dated August 15, 2022, (Charles, 2022);
- "Pine Point Marketing Report" by Jim Vice of Stonehouse Consulting Inc dated August 18, 2020, (Vice, 2022).

This information is used in Chapter 19 (Market Studies and Contracts) and in support of the financial analysis in Chapter 22 (Economic Analysis).

Colin Hardie, QP, considers it reasonable to rely upon Mr. Jim Vice (Stonehouse Consulting Services) who was contracted directly by Osisko Metals for such information, as the company provides independent analysis and advice on base metal concentrate terms, logistics and markets to the mining industry.



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 106 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 46,883.7 hectares. This is current as of Aug 08, 2022. Detailed lists of the PPML mineral leases and mineral claims are shown in Table 4-2 and Table 4-3, respectively. Currently, there is one adjacent mineral claim of 24 ha held by a local individual that is not impactful to the 2022 MRE Update. There are no other adjacent mine claims or mining leases to the Project as the surrounding areas are within interim land withdrawals and these areas are not available for staking.

Table 4-1: Mineral leases and mineral claims

Mineral Tenure Summary 8-Aug-22		
	Number	Area (ha)
Mining Leases	40	17,548.2
Mineral Claims	106	29,335.45
Total	146	46,883.65
Surface Leases (R190)	5	51.0

4.2 Surface Leases

Pine Point has five surface leases in the R190 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 the devolution of government services from the federal government to the Government of Northwest Territories, two separate lease payments are due annually.

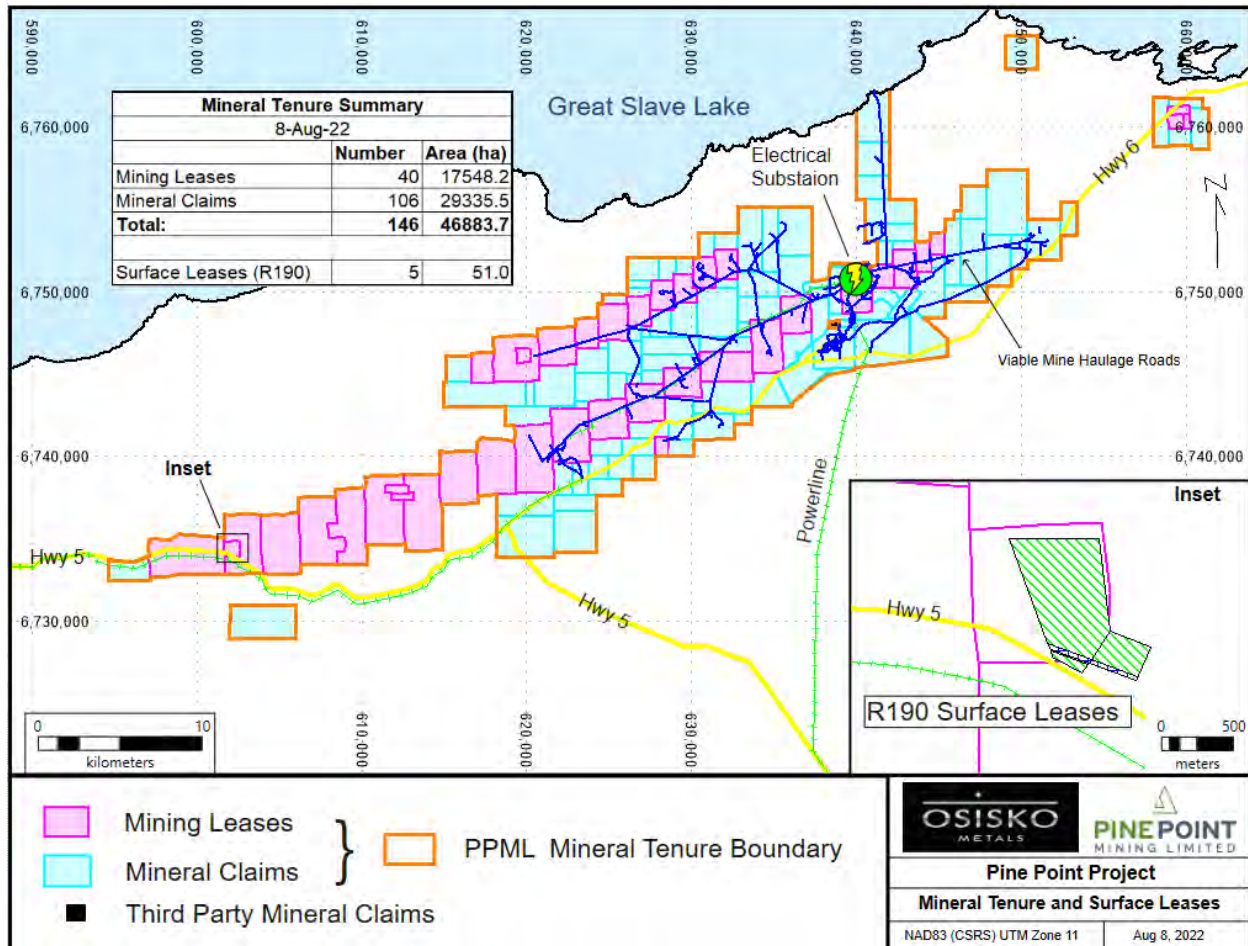


Figure 4-1: Pine Point Mining Limited mineral claims, mineral leases and surface claims

4.3 Project Areas and Deposits

The Pine Point Project covers a land area of approximately 4 km² and includes, 10 project areas and 89 deposits. Figure 4-2 and Table 4-2 provide an overview of the zones, areas, and deposits that are included in this 2022 Mineral Resource Estimate ("MRE") as well as a breakdown of the open-pit and underground deposits specifically included as part of the Preliminary Economic Assessment ("PEA") mineral inventory.

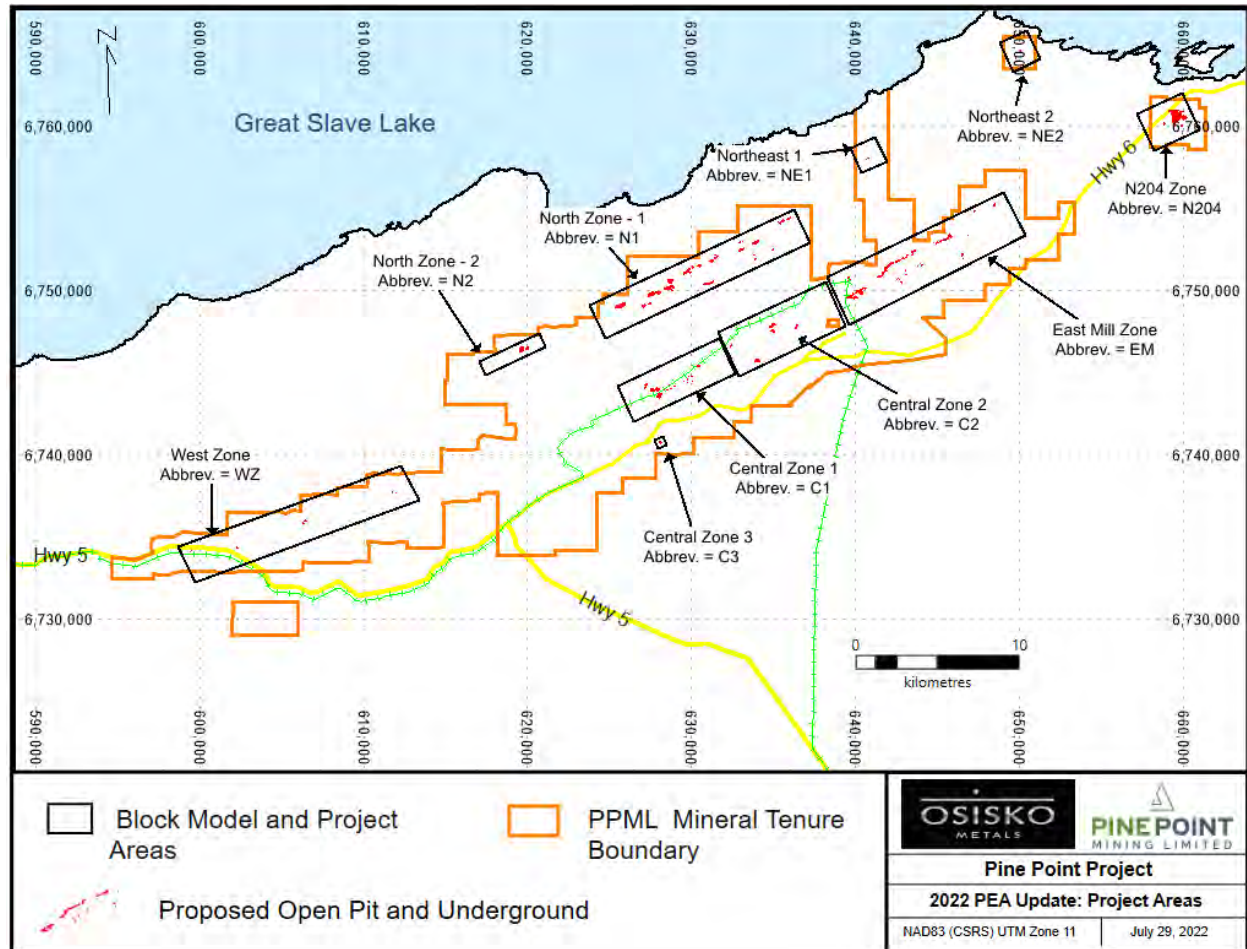


Figure 4-2: Pine Point Project zones and areas

Mineralized Trend	North Trend				Main Trend					South Trend	
Project Areas	N2	N1	NE1	NE2	WZ	C1	C2	EM	N204	C3	
Individual Deposit Names	W85	X49	T37	X17	GO3	J68	K52	L24	N204A	R67	
	YBM	X55		X18	O556	K60	K57	L25	N204B		
		X59			P499	K68	M48	L26	N204C		
		X60			R190	L65	O53	L27			
		X61			X25	M63		L35			
		X68				M67		M40			
		X71						N31			
		Y53						N32			
		Y55						N33			
		Y56						N34			
		Y57						N39			
		Y58						O26			
		Y62						O27			
		Y65									
		Y68									
		Z60									

Deposit Type

Prismatic-style deposits

Tabular-style deposits

Proposed Development Type

Open Pit

Underground

Figure 4-3: Deposit names referenced in the Mineral Resource Estimate.

4.4 Royalties and Encumbrances

On October 15 2019, Osisko Metals sold a 1.5% NSR to Osisko Gold Royalties for CAD6,500,000. On December 10 2019, Osisko Metals sold a further 0.5% to Osisko Gold Royalties for consideration of CAD6,500,000. On February 11 2022, Osisko Metals sold a 1% NSR to Osisko Gold Royalties for consideration of CAD6,500,000. Following the completion of all three transactions, the current encumbrance on the Project consists of an additional 1.5% NSR payable to Osisko Gold Royalties and Right of First Offer for any future sales of royalties or equivalent interest. There is total of 3% royalties to Osisko Gold Royalties.

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).

4.5 Environmental Liabilities

The Pine Point Project is located on a previously disturbed mine site. Cominco Ltd. operated the Pine Point mine, producing 64 Mt of material from 50 open-pits between 1964 and 1988. The Cominco Mine included the historical open-pits, waste rock stockpiles, tailings disposal areas, township water ponds, the Process Plant Site, haulage and service roads, the former townsite of Pine Point and an airstrip. The Pine Point Project will occupy a portion of these previously disturbed areas, including some of the historical open pits. However, the Project will not disturb the existing tailings impoundment area (managed by Teck Metals Ltd.), the historic townsite and the old, airstrip. There are no plans to disturb the historic waste rock stockpiles. The environmental liability will be limited to the project development footprint.

Other environmental liabilities on the site include the historic tailings facility managed by Teck Metals Ltd. Teck Metals currently holds a Water Licence and Land use permit that includes a security that will remain until Teck Metals Ltd. implements and completes a final closure reclamation plan to the satisfaction of the Mackenzie Valley Land and Water Board ("MVLWB"), Government of Northwest Territories ("GNWT"), other government regulators and the Indigenous groups and communities.

From Sections 4.1 and 4.2, 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 these lease 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).

4.6.2 Environmental Studies

The Pine Point Project benefits from 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 updated data during the regulatory process, and the data gathering process is underway. Past environmental and/or baseline studies conducted across the Pine Point Project include:

- Waste rock characterization;
- Bathymetry of existing mine pits;
- Air quality including dust fall 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;
- Hydrology studies;
- Traditional Knowledge and Socioeconomic studies; and
- Demographic data gathering.

4.6.3 Permitting

PPML currently holds a Type-A Land Use Permit ("LUP") issued November 2, 2021 and effective until November 1, 2026 and a Type A Water Licence issued on December 8, 2021 and effective until December 7, 2028. The Licence and the Permit include all but two mineral tenures (claims and leases) held by PPML in the project area when the permits were issued. The two claims not included were staked earlier in 2022. An amendment to the licence and permit to add these two claims is currently being processed by the MVLWB. Previously issued Land Use Permits have expired or will expire by September 19, 2022 and a previously issued Water Licence will expire on September 8, 2022. All activities under these expiring permits are included in the new Type-A LUP and the Type A Water Licence issued in 2021. These 84 mineral claims include claims PP 01-95 (Table 4-3).

4.6.4 Social and Community Impact

There are three Indigenous groups that have been engaged regularly regarding the Project: the Deninu Kue First Nation ("DKFN"), the K'at'l'odeeche First Nation ("KFN"), and the Northwest Territory Metis Nation ("NWTMN"). Information regarding exploration and development activities has been shared and will continue to be shared with these indigenous groups, and others (i.e., West Point First Nation, Smith's Landing First Nation, Salt River First Nation, Dene Tha First Nation, and Deh Gah Gotie First Nation), as well as with the local communities and municipal governments (i.e., Hay River, Fort Smith, and Fort Resolution), the Government of Northwest Territories and the Federal Government.

On September 30, 2019, Pine Point Mining Limited signed two separate Collaboration Agreements with the Deninu Kue and Northwest Territory Metis Nation. These agreements were amended in 2021, allowing PPML to add two new claims. These parties entered into the agreements in order to promote a cooperative and mutually respectful relationship governing the proposed exploration and development activities in the area of Pine Point. The agreements reflect the intention to work with each Indigenous community regarding education and training, employment, business and contracting opportunities, information sharing, site visits, and also broad outlines of topics for future agreements.

The Company also has an exploration agreement with the K'at'l'odeeche First Nation regarding consultation and exploration activities to be performed within the western portion of the Project.

PPML is actively undertaking engagement activities with local communities in the Pine Point Project area for both the exploration program as well as the proposed mine.



Table 4-2: Details of mineral leases (current as of August 8, 2022)

Lease No.	Lease Status	Issue Date	Term Expire	Area Ha	Owner	Land Claim
NT-4869	ACTIVE	7/16/2007	7/15/2028	251	Pine Point Mining Limited (100%)	AKAITCHO/NWTMN
NT-4866	ACTIVE	5/9/2007	5/8/2028	723	Pine Point Mining Limited (100%)	DEHCHO/NWTMN
NT-4860	ACTIVE	5/9/2007	5/8/2028	86.3	Pine Point Mining Limited (100%)	DEHCHO/NWTMN
NT-4864	ACTIVE	5/9/2007	5/8/2028	936	Pine Point Mining Limited (100%)	DEHCHO/NWTMN
NT-4868	ACTIVE	7/16/2007	7/15/2028	405	Pine Point Mining Limited (100%)	AKAITCHO/NWTMN
NT-4865	ACTIVE	5/9/2007	5/8/2028	837	Pine Point Mining Limited (100%)	DEHCHO/NWTMN
NT-4870	ACTIVE	7/16/2007	7/15/2028	522	Pine Point Mining Limited (100%)	AKAITCHO/NWTMN
NT-4862	ACTIVE	5/9/2007	5/8/2028	887	Pine Point Mining Limited (100%)	DEHCHO/NWTMN
NT-5242	ACTIVE	8/25/2011	8/24/2032	227	Pine Point Mining Limited (100%)	AKAITCHO/NWTMN
NT-5241	ACTIVE	8/25/2011	8/24/2032	249	Pine Point Mining Limited (100%)	AKAITCHO/NWTMN
NT-5240	ACTIVE	8/25/2011	8/24/2032	254	Pine Point Mining Limited (100%)	AKAITCHO/NWTMN
NT-5243	ACTIVE	8/25/2011	8/24/2032	251	Pine Point Mining Limited (100%)	AKAITCHO/NWTMN
NT-5239	ACTIVE	8/25/2011	8/24/2032	249	Pine Point Mining Limited (100%)	AKAITCHO/NWTMN
NT-5260	ACTIVE	8/25/2011	8/24/2032	722	Pine Point Mining Limited (100%)	NWTMN
NT-5249	ACTIVE	8/25/2011	8/24/2032	191	Pine Point Mining Limited (100%)	AKAITCHO/NWTMN
NT-5258	ACTIVE	8/25/2011	8/24/2032	938	Pine Point Mining Limited (100%)	NWTMN
NT-5255	ACTIVE	8/25/2011	8/24/2032	536	Pine Point Mining Limited (100%)	AKAITCHO/NWTMN
NT-5261	ACTIVE	8/25/2011	8/24/2032	89.2	Pine Point Mining Limited (100%)	AKAITCHO/NWTMN
NT-5252	ACTIVE	8/25/2011	8/24/2032	593	Pine Point Mining Limited (100%)	AKAITCHO/NWTMN
NT-5259	ACTIVE	8/25/2011	8/24/2032	786	Pine Point Mining Limited (100%)	NWTMN
NT-5251	ACTIVE	8/25/2011	8/24/2032	419	Pine Point Mining Limited (100%)	AKAITCHO/NWTMN
NT-5250	ACTIVE	8/25/2011	8/24/2032	336	Pine Point Mining Limited (100%)	AKAITCHO/NWTMN
NT-5248	ACTIVE	8/25/2011	8/24/2032	319	Pine Point Mining Limited (100%)	AKAITCHO/NWTMN



Lease No.	Lease Status	Issue Date	Term Expire	Area Ha	Owner	Land Claim
NT-5262	ACTIVE	8/25/2011	8/24/2032	165	Pine Point Mining Limited (100%)	AKAITCHO/NWTMN
NT-5247	ACTIVE	8/25/2011	8/24/2032	122	Pine Point Mining Limited (100%)	AKAITCHO/NWTMN
NT-5257	ACTIVE	8/25/2011	8/24/2032	742	Pine Point Mining Limited (100%)	AKAITCHO/NWTMN
NT-5245	ACTIVE	8/25/2011	8/24/2032	64.8	Pine Point Mining Limited (100%)	AKAITCHO/NWTMN
NT-5246	ACTIVE	8/25/2011	8/24/2032	82	Pine Point Mining Limited (100%)	AKAITCHO/NWTMN
NT-5244	ACTIVE	8/25/2011	8/24/2032	39.4	Pine Point Mining Limited (100%)	AKAITCHO/NWTMN
NT-4858	ACTIVE	5/9/2007	5/8/2028	63.1	Pine Point Mining Limited (100%)	DEHCHO/NWTMN
NT-4859	ACTIVE	5/9/2007	5/8/2028	57.9	Pine Point Mining Limited (100%)	DEHCHO/NWTMN
NT-4861	ACTIVE	5/9/2007	5/8/2028	1006	Pine Point Mining Limited (100%)	DEHCHO/NWTMN
NT-4863	ACTIVE	5/9/2007	5/8/2028	819	Pine Point Mining Limited (100%)	DEHCHO/NWTMN
NT-4867	ACTIVE	5/9/2007	5/8/2028	1067	Pine Point Mining Limited (100%)	DEHCHO/NWTMN
NT-4871	ACTIVE	7/16/2007	7/15/2028	692	Pine Point Mining Limited (100%)	NWTMN
NT-4872	ACTIVE	7/16/2007	7/15/2028	241	Pine Point Mining Limited (100%)	NWTMN
NT-4873	ACTIVE	7/16/2007	7/15/2028	76.5	Pine Point Mining Limited (100%)	NWTMN
NT-5253	ACTIVE	8/25/2011	8/24/2032	569	Pine Point Mining Limited (100%)	AKAITCHO/NWTMN
NT-5254	ACTIVE	8/25/2011	8/24/2032	389	Pine Point Mining Limited (100%)	AKAITCHO/NWTMN
NT-5256	ACTIVE	8/25/2011	8/24/2032	546	Pine Point Mining Limited (100%)	AKAITCHO/NWTMN
TOTAL				17,548.2		
Number				40		





Table 4-3: Details of mineral claims (current as of August 8, 2022)

Claim No.	Claim Status	Issue Date	Anniversary Date	Area (ha)	Owners	Claim Name	Land Claim
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
M10191	ACTIVE	2018-06-22	2028-06-22	92.49	Pine Point Mining Limited (100%)	PPM 1	AKAITCHO/NWTMN
M10192	ACTIVE	2018-06-22	2028-06-22	32.54	Pine Point Mining Limited (100%)	PPM 2	AKAITCHO/NWTMN
M10296	ACTIVE	2017-02-16	2027-02-16	309	Pine Point Mining Limited (100%)	D8	DEHCHO/NWTMN
M10297	ACTIVE	2017-02-16	2025-02-16	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
M10302	ACTIVE	2017-02-22	2027-02-22	136	Pine Point Mining Limited (100%)	D13	AKAITCHO/NWTMN
M10303	ACTIVE	2017-02-22	2027-02-22	155	Pine Point Mining Limited (100%)	D12	AKAITCHO/NWTMN
M10426	ACTIVE	2018-06-22	2028-06-22	15.26	Pine Point Mining Limited (100%)	PPM 3	AKAITCHO/NWTMN
M10427	ACTIVE	2018-06-22	2028-06-22	4.59	Pine Point Mining Limited (100%)	PPM 4	AKAITCHO/NWTMN
M10514	ACTIVE	2019-09-25	2029-09-25	625	Pine Point Mining Limited (100%)	PP 81	DEHCHO/NWTMN
M10515	ACTIVE	2019-09-25	2029-09-25	250	Pine Point Mining Limited (100%)	PP 82	AKAITCHO/NWTMN
M10516	ACTIVE	2019-09-25	2029-09-25	185	Pine Point Mining Limited (100%)	PP 83	AKAITCHO/NWTMN
M10517	ACTIVE	2019-09-25	2029-09-25	450	Pine Point Mining Limited (100%)	PP 84	AKAITCHO/NWTMN
M10518	ACTIVE	2019-09-25	2029-09-25	300	Pine Point Mining Limited (100%)	PP 85	AKAITCHO/NWTMN
M10519	ACTIVE	2019-09-25	2029-09-25	400	Pine Point Mining Limited (100%)	PP 86	AKAITCHO/NWTMN
M10520	ACTIVE	2019-09-25	2029-09-25	700	Pine Point Mining Limited (100%)	PP 87	AKAITCHO/NWTMN
M10550	ACTIVE	2019-09-25	2029-09-25	215	Pine Point Mining Limited (100%)	PP 90	AKAITCHO/NWTMN
M10551	ACTIVE	2019-09-25	2029-09-25	165	Pine Point Mining Limited (100%)	PP 91	AKAITCHO/NWTMN
M10552	ACTIVE	2019-09-25	2029-09-25	190	Pine Point Mining Limited (100%)	PP 92	AKAITCHO/NWTMN
M10553	ACTIVE	2019-09-25	2029-09-25	200	Pine Point Mining Limited (100%)	PP 93	AKAITCHO/NWTMN



Claim No.	Claim Status	Issue Date	Anniversary Date	Area (ha)	Owners	Claim Name	Land Claim
M10554	ACTIVE	2019-09-25	2029-09-25	225	Pine Point Mining Limited (100%)	PP 94	AKAITCHO/NWTMN
M10555	ACTIVE	2019-09-25	2029-09-25	235	Pine Point Mining Limited (100%)	PP 95	AKAITCHO/NWTMN
M10649	ACTIVE	2022-03-25	2024-03-25	280.39	Pine Point Mining Limited (100%)	PP96	NWTMN
M10650	ACTIVE	2022-03-25	2024-03-25	62.6	Pine Point Mining Limited (100%)	PP 97	NWTMN
M10653	ACTIVE	2018-06-22	2028-06-22	1.93	Pine Point Mining Limited (100%)	PPM 5	AKAITCHO/NWTMN
M10654	ACTIVE	2018-06-22	2028-06-22	35.2	Pine Point Mining Limited (100%)	PPM 6	AKAITCHO/NWTMN
M10658	ACTIVE	2018-06-22	2028-06-22	21.53	Pine Point Mining Limited (100%)	PPM 7	AKAITCHO/NWTMN
M10801	ACTIVE	2019-09-25	2029-09-25	300	Pine Point Mining Limited (100%)	PP 1	AKAITCHO/NWTMN
M10802	ACTIVE	2019-09-25	2029-09-25	400	Pine Point Mining Limited (100%)	PP 2	AKAITCHO/NWTMN
M10803	ACTIVE	2019-09-25	2029-09-25	1000	Pine Point Mining Limited (100%)	PP 3	AKAITCHO/NWTMN
M10804	ACTIVE	2019-09-25	2029-09-25	300	Pine Point Mining Limited (100%)	PP 4	AKAITCHO/NWTMN
M10805	ACTIVE	2019-09-25	2029-09-25	600	Pine Point Mining Limited (100%)	PP 5	AKAITCHO/NWTMN
M10806	ACTIVE	2019-09-25	2029-09-25	400	Pine Point Mining Limited (100%)	PP 6	AKAITCHO/NWTMN
M10807	ACTIVE	2019-09-25	2029-09-25	300	Pine Point Mining Limited (100%)	PP 7	AKAITCHO/NWTMN
M10808	ACTIVE	2019-09-25	2029-09-25	150	Pine Point Mining Limited (100%)	PP 8	AKAITCHO/NWTMN
M10809	ACTIVE	2019-09-25	2029-09-25	400	Pine Point Mining Limited (100%)	PP 9	AKAITCHO/NWTMN
M10810	ACTIVE	2019-09-25	2029-09-25	600	Pine Point Mining Limited (100%)	PP 10	AKAITCHO/NWTMN
M10811	ACTIVE	2019-09-25	2029-09-25	750	Pine Point Mining Limited (100%)	PP 11	AKAITCHO/NWTMN
M10812	ACTIVE	2019-09-25	2029-09-25	70	Pine Point Mining Limited (100%)	PP 12	AKAITCHO/NWTMN
M10813	ACTIVE	2019-09-25	2029-09-25	110	Pine Point Mining Limited (100%)	PP 13	AKAITCHO/NWTMN
M10814	ACTIVE	2019-09-25	2029-09-25	140	Pine Point Mining Limited (100%)	PP 14	AKAITCHO/NWTMN
M10815	ACTIVE	2019-09-25	2029-09-25	350	Pine Point Mining Limited (100%)	PP 15	AKAITCHO/NWTMN
M10816	ACTIVE	2019-09-25	2029-09-25	410	Pine Point Mining Limited (100%)	PP 16	AKAITCHO/NWTMN
M10817	ACTIVE	2019-09-25	2029-09-25	175	Pine Point Mining Limited (100%)	PP 17	AKAITCHO/NWTMN
M10818	ACTIVE	2019-09-25	2029-09-25	320	Pine Point Mining Limited (100%)	PP 18	AKAITCHO/NWTMN
M10819	ACTIVE	2019-09-25	2029-09-25	105	Pine Point Mining Limited (100%)	PP 19	AKAITCHO/NWTMN
M10820	ACTIVE	2019-09-25	2029-09-25	60	Pine Point Mining Limited (100%)	PP 20	AKAITCHO/NWTMN
M10821	ACTIVE	2019-09-25	2029-09-25	310	Pine Point Mining Limited (100%)	PP 21	AKAITCHO/NWTMN
M10822	ACTIVE	2019-09-25	2029-09-25	225	Pine Point Mining Limited (100%)	PP 22	AKAITCHO/NWTMN
M10823	ACTIVE	2019-09-25	2029-09-25	75	Pine Point Mining Limited (100%)	PP 23	AKAITCHO/NWTMN



Claim No.	Claim Status	Issue Date	Anniversary Date	Area (ha)	Owners	Claim Name	Land Claim
M10824	ACTIVE	2019-09-25	2029-09-25	40	Pine Point Mining Limited (100%)	PP 24	AKAITCHO/NWTMN
M10825	ACTIVE	2019-09-25	2029-09-25	285	Pine Point Mining Limited (100%)	PP 25	AKAITCHO/NWTMN
M10826	ACTIVE	2019-09-25	2029-09-25	600	Pine Point Mining Limited (100%)	PP 26	AKAITCHO/NWTMN
M10827	ACTIVE	2019-09-25	2029-09-25	375	Pine Point Mining Limited (100%)	PP 27	AKAITCHO/NWTMN
M10828	ACTIVE	2019-09-25	2029-09-25	370	Pine Point Mining Limited (100%)	PP 28	AKAITCHO/NWTMN
M10829	ACTIVE	2019-09-25	2029-09-25	75	Pine Point Mining Limited (100%)	PP 29	AKAITCHO/NWTMN
M10830	ACTIVE	2019-09-25	2029-09-25	450	Pine Point Mining Limited (100%)	PP 30	AKAITCHO/NWTMN
M10831	ACTIVE	2019-09-25	2029-09-25	150	Pine Point Mining Limited (100%)	PP 31	AKAITCHO/NWTMN
M10832	ACTIVE	2019-09-25	2029-09-25	140	Pine Point Mining Limited (100%)	PP 32	AKAITCHO/NWTMN
M10833	ACTIVE	2019-09-25	2029-09-25	80	Pine Point Mining Limited (100%)	PP 33	AKAITCHO/NWTMN
M10834	ACTIVE	2019-09-25	2029-09-25	75	Pine Point Mining Limited (100%)	PP 34	AKAITCHO/NWTMN
M10835	ACTIVE	2019-09-25	2029-09-25	320	Pine Point Mining Limited (100%)	PP 35	DEHCHO/NWTMN
M10837	ACTIVE	2019-09-25	2029-09-25	590	Pine Point Mining Limited (100%)	PP 37	AKAITCHO/NWTMN
M10838	ACTIVE	2019-09-25	2029-09-25	75	Pine Point Mining Limited (100%)	PP 38	AKAITCHO/NWTMN
M10839	ACTIVE	2019-09-25	2029-09-25	25	Pine Point Mining Limited (100%)	PP 39	AKAITCHO/NWTMN
M10840	ACTIVE	2019-09-25	2029-09-25	300	Pine Point Mining Limited (100%)	PP 40	AKAITCHO/NWTMN
M10841	ACTIVE	2019-09-25	2029-09-25	30	Pine Point Mining Limited (100%)	PP 41	AKAITCHO/NWTMN
M10842	ACTIVE	2019-09-25	2029-09-25	330	Pine Point Mining Limited (100%)	PP 42	DEHCHO/NWTMN
M10843	ACTIVE	2019-09-25	2029-09-25	35	Pine Point Mining Limited (100%)	PP 43	AKAITCHO/NWTMN
M10844	ACTIVE	2019-09-25	2029-09-25	740	Pine Point Mining Limited (100%)	PP 44	AKAITCHO/NWTMN
M10845	ACTIVE	2019-09-25	2029-09-25	240	Pine Point Mining Limited (100%)	PP 45	AKAITCHO/NWTMN
M10846	ACTIVE	2019-09-25	2029-09-25	275	Pine Point Mining Limited (100%)	PP 46	AKAITCHO/NWTMN
M10847	ACTIVE	2019-09-25	2029-09-25	125	Pine Point Mining Limited (100%)	PP 47	AKAITCHO/NWTMN
M10848	ACTIVE	2019-09-25	2029-09-25	100	Pine Point Mining Limited (100%)	PP 48	AKAITCHO/NWTMN
M10849	ACTIVE	2019-09-25	2029-09-25	250	Pine Point Mining Limited (100%)	PP 49	AKAITCHO/NWTMN
M10850	ACTIVE	2019-09-25	2029-09-25	25	Pine Point Mining Limited (100%)	PP 50	AKAITCHO/NWTMN
M10851	ACTIVE	2019-09-25	2029-09-25	450	Pine Point Mining Limited (100%)	PP 51	AKAITCHO/NWTMN
M10852	ACTIVE	2019-09-25	2029-09-25	200	Pine Point Mining Limited (100%)	PP 52	AKAITCHO/NWTMN
M10853	ACTIVE	2019-09-25	2029-09-25	500	Pine Point Mining Limited (100%)	PP 53	AKAITCHO/NWTMN
M10854	ACTIVE	2019-09-25	2029-09-25	200	Pine Point Mining Limited (100%)	PP 54	AKAITCHO/NWTMN



Claim No.	Claim Status	Issue Date	Anniversary Date	Area (ha)	Owners	Claim Name	Land Claim
M10855	ACTIVE	2019-09-25	2029-09-25	70	Pine Point Mining Limited (100%)	PP 55	AKAITCHO/NWTMN
M10856	ACTIVE	2019-09-25	2029-09-25	75	Pine Point Mining Limited (100%)	PP 56	AKAITCHO/NWTMN
M10857	ACTIVE	2019-09-25	2029-09-25	500	Pine Point Mining Limited (100%)	PP 57	AKAITCHO/NWTMN
M10858	ACTIVE	2019-09-25	2029-09-25	40	Pine Point Mining Limited (100%)	PP 58	AKAITCHO/NWTMN
M10859	ACTIVE	2019-09-25	2029-09-25	175	Pine Point Mining Limited (100%)	PP 59	AKAITCHO/NWTMN
M10860	ACTIVE	2019-09-25	2029-09-25	240	Pine Point Mining Limited (100%)	PP 60	AKAITCHO/NWTMN
M10861	ACTIVE	2019-09-25	2029-09-25	250	Pine Point Mining Limited (100%)	PP 61	AKAITCHO/NWTMN
M10862	ACTIVE	2019-09-25	2029-09-25	375	Pine Point Mining Limited (100%)	PP 62	AKAITCHO/NWTMN
M10863	ACTIVE	2019-09-25	2029-09-25	275	Pine Point Mining Limited (100%)	PP 63	AKAITCHO/NWTMN
M10865	ACTIVE	2019-09-25	2029-09-25	150	Pine Point Mining Limited (100%)	PP 65	AKAITCHO/NWTMN
M10866	ACTIVE	2019-09-25	2029-09-25	650	Pine Point Mining Limited (100%)	PP 66	AKAITCHO/NWTMN
M10868	ACTIVE	2019-09-25	2029-09-25	150	Pine Point Mining Limited (100%)	PP 68	AKAITCHO/NWTMN
M10869	ACTIVE	2019-09-25	2029-09-25	50	Pine Point Mining Limited (100%)	PP 69	DEHCHO/NWTMN
M10870	ACTIVE	2019-09-25	2029-09-25	400	Pine Point Mining Limited (100%)	PP 70	AKAITCHO/NWTMN
M10877	ACTIVE	2019-09-25	2029-09-25	200	Pine Point Mining Limited (100%)	PP 77	AKAITCHO/NWTMN
M10878	ACTIVE	2019-09-25	2029-09-25	225	Pine Point Mining Limited (100%)	PP 78	AKAITCHO/NWTMN
M10879	ACTIVE	2019-09-25	2029-09-25	500	Pine Point Mining Limited (100%)	PP 79	AKAITCHO/NWTMN
M10880	ACTIVE	2019-09-25	2029-09-25	875	Pine Point Mining Limited (100%)	PP 80	AKAITCHO/NWTMN
Total				29,335.45			
Number				106			



Table 4-4: Details of surface leases (Current as of August 8, 2022).

Registry Number	Lease Name	Lease Number	Anniversary Date	Annual Lease Rental	Jurisdiction
KM 42, NWT Highway No. 5	Settling Pond	85B/11-15-2 Part 1	2029-June-30	\$2,423.00	Northwest Territories Lands
KM 42, NWT Highway No. 5	Settling Pond	85B/11-15-2 Part 2	2029-June-30		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
Total \$/y				\$2,7423.00	
Total Area (ha)				21	
Number				5	

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

5.1 Accessibility

5.1.1 Road Access

The Pine Point Lead-Zinc 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 lies north of, and roughly parallels, territorial highways NT-5 and NT-6, beginning 42 km east of Hay River, NT, and extending to 110 km from Hay River. Access within the Property is via a network of paved roads and 100 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.

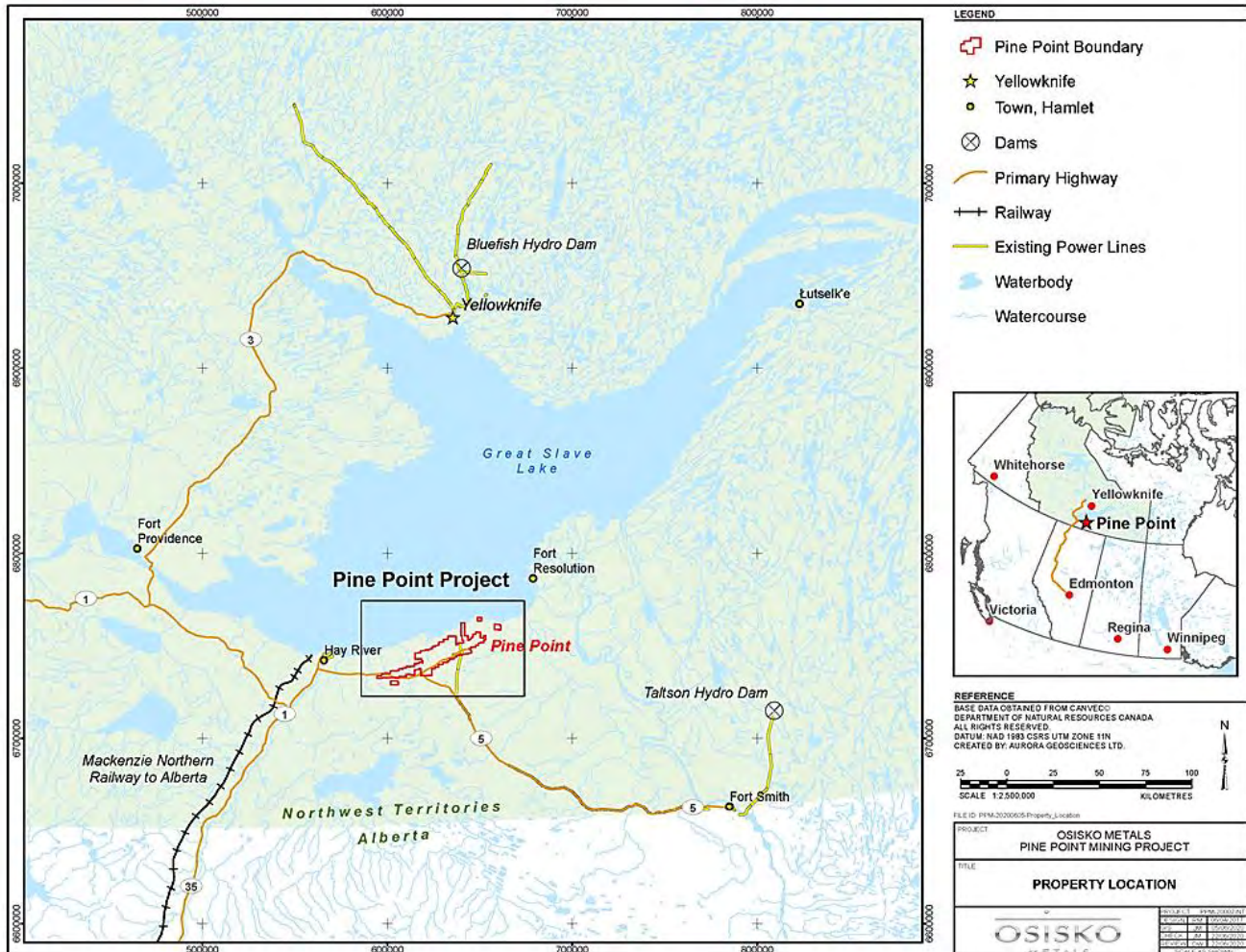


Figure 5-1: Location of the Pine Point Project

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 to 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 Project area 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 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 Project 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 Historical Mine Infrastructure

In addition to paved highway access, an electrical substation, as well as 100 km of viable historic mine haulage roads, are in place on the property. Bush roads and abundant historical survey lines provide good access through the Project, particularly for winter drill programs.

5.2.3 Local Workforce

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 Northwest Territories.

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.

5.2.4 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 Hay River 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 Project approximately 4 km northeast of the historic Pine Point town site. The dam complex currently has an 18.5 MW capacity with a forecasted capacity increase of 4 MW (power upgrade scheduled to be completed in 2022). Of that, 9 MW will be unused and available during the winter months (October to March) and 12 MW unused and available during the summer months (April to September).

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

5.3 Exploration Camp

In fall 2018, an exploration camp was commissioned by Pine Point Mining Limited within permit guidelines to facilitate geological work and site support. A semi-permanent all-season tent camp was built by Discovery Mining Services and expanded in late 2021 to add increased capacity through the addition of trailers rented from GNS Industrial Trailer Services. The Camp houses geologists, management, drilling staff, and support staff. The camp consists of sleeper tents and trailers, a kitchen trailer, generator building, dry/shower tent, lavatory building, and a staff office. On-site amenities also include satellite internet and a cell phone relay tower to allow 24-hour instant communication to Hay River and the Montreal head office. As of the fall 2019 program, all staff was moved to the site with no operations or rented accommodations in Hay River. A new core logging facility has been constructed to allow year-round logging and relogging programs on-site. The majority of drilling and future planned drilling are conducted in the winter months between November and April. Logistics and drill support are much easier with this arrangement and allow monitoring and reporting by management to be continuous (Figure 5-1).

5.4 Physiography

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

The Buffalo River flows northward through the Project area and into Great Slave Lake. Other drainages by streams within the Project area are minimal. With such poor drainage, the area is dominated by muskeg swamp and numerous small ponds. Several east-west trending sand and gravel ridges mark the 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 ft) 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, evidence 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 for 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 a few miles inland from the Ile du Mort on the Great Slave Lakeshore. Nagle staked eight claims over the showings and collected some galena samples. Nagle’s claims centered on what has become known as the historical P32 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). A 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 to form 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 a reconnaissance survey 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 the 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 T37 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 an interest in a total of 270 claims, but the T37 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, the 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. 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 was conducted in the summer of 1930 and totaled 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 that 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 kilometres from the previous surface discoveries, all covered by overburden.

In order to obtain bulk mineralized samples, shafts were sunk into the historical N42 and M40 deposits in 1954. The historical N42 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 M40 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 Project. 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 X15 pit, which eventually produced 17,474,260 tonnes of material at 2% lead and 6.2% zinc. Pyramid Mining also discovered W17 deposit near X15. Other discoveries made in 1966 include: A55 deposit on the Buffalo River Exploration property, R61 and S65 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 Project area and most of the area 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 Project 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, X15, 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 M40 and Y65. M40 was in production from 1975 to 1977 and produced 350,870 tonnes of material grading 2.2% lead and 5.5% zinc. Y65 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			Contained Metal		
		%Pb	%Zn	%Zn+Pb	Pb (tonnes)	Zn (tonnes)	Zn+Pb (tonnes)
NORTH TREND - listed from northeast to southwest							
X17	44,910	1.5	6.3	7.8	674	2,829	3,503
T37	358,960	2.1	6.3	8.4	7,538	22,614	30,152
X51	1,203,980	2.2	6.7	8.9	26,488	80,667	107,155
Y53	967,710	1.5	5.6	7.1	14,516	54,192	68,708
X52	1,104,080	1.6	6.3	7.9	17,665	69,557	87,222
X53	1,231,940	2.7	9.2	11.9	33,262	113,338	146,600
Z53	380,520	1.4	5.0	6.4	5,327	19,026	24,353
A55	1,550,830	3.0	7.6	10.6	46,525	117,863	164,388
Y54	263,840	1.3	4.0	5.3	3,430	10,554	13,984
X54/X55	216,130	2.1	6.7	8.8	4,539	14,481	19,020
X56/X57	1,319,580	1.6	6.3	7.9	21,113	83,134	104,247



Deposit	Production (tonnes)	Grade			Contained Metal		
		%Pb	%Zn	%Zn+Pb	Pb (tonnes)	Zn (tonnes)	Zn+Pb (tonnes)
Z57	827,870	1.1	4.2	5.3	9,107	34,771	43,878
Y65	149,770	7.0	12.9	19.9	10,484	19,320	29,804
Y60	512,490	2.1	7.3	9.4	10,762	37,412	48,174
Y61	549,040	3.5	9.3	12.8	19,216	51,061	70,277
Z64	913,470	1.4	5.1	6.5	12,789	46,587	59,376
A70	2,289,360	4.5	10.4	14.9	103,021	238,093	341,114
MAIN TREND - listed from northeast to southwest							
P24	496,640	3.5	7.6	11.1	17,382	37,745	55,127
L30	262,170	1.1	2.8	3.9	2,884	7,341	10,225
O28	1,483,870	2.0	3.7	5.7	29,677	54,903	84,580
P29	476,120	1.6	3.3	4.9	7,618	15,712	23,330
N31	505,200	1.6	4.1	5.7	8,083	20,713	28,796
P31	604,760	2.2	3.6	5.8	13,305	21,771	35,076
P32	694,980	3.2	3.5	6.7	22,239	24,324	46,563
O32	375,970	2.8	6.4	9.2	10,527	24,062	34,589
N32	1,862,070	3.4	8.4	11.8	63,310	156,414	219,724
L37	3,417,550	1.0	3.4	4.4	34,176	116,197	150,373
N38	1,182,110	4.9	7.4	12.3	57,923	87,476	145,399
N42	2,959,680	5.3	9.5	14.8	156,863	281,170	438,033
O42	2,742,720	8.8	11.6	20.4	241,359	318,156	559,515
P41	196,140	2.1	8.3	10.4	4,119	16,280	20,399
M40	350,870	2.2	5.5	7.7	7,719	19,298	27,017
J44	1,282,230	5.9	9.8	15.7	75,652	125,659	201,311
I46	389,870	5.1	4.2	9.3	19,883	16,375	36,258
M52	455,260	3.5	7.6	11.1	15,934	34,600	50,534
K53	468,900	3.7	9.3	13.0	17,349	43,608	60,957
K57	1,564,540	6.5	5.2	11.7	101,695	81,356	183,051
K62	1,001,590	3.6	4.8	8.4	36,057	48,076	84,133
I65	194,510	3.8	11.1	14.9	7,391	21,591	28,982
M64	178,460	4.9	8.0	12.9	8,745	14,277	23,022
R61	1,034,540	1.6	5.2	6.8	16,553	53,796	70,349
J69	854,770	1.2	5.2	6.4	10,257	44,448	54,705
K77	511,120	6.4	6.4	12.8	32,712	32,712	65,424
N81	2,699,950	7.0	14.1	21.1	188,997	380,693	569,690



Deposit	Production (tonnes)	Grade			Contained Metal		
		%Pb	%Zn	%Zn+Pb	Pb (tonnes)	Zn (tonnes)	Zn+Pb (tonnes)
SOUTH TREND - listed from northeast to southwest							
X15	17,474,260	2.0	6.2	8.2	349,485	1,083,404	1,432,889
W17	3,515,400	2.0	6.1	8.1	70,308	214,439	284,747
T58	563,310	4.5	12.6	17.1	25,349	70,997	96,346
R61	1,034,540	1.6	5.2	6.8	16,553	53,796	70,349
S65	575,550	1.2	5.7	6.9	6,907	32,806	39,713
TOTAL	65,294,130	3.10%	7.00%	10.10%	2,023,467	4,569,694	6,593,161

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 until 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 2017

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 Project area. 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 downhole 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 Project 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: O555, O556, P499, R190, T799, V46, W19, X25 and Z155. The X25 deposit is the largest with R190 and Z155 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 W85 deposit in the North Trend, the G03 deposit in the western part of the former Cominco Ltd. Pine Point Project, and the R190 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 evaluated a series of deposits that are part of the current MRE Collins et al. (2008). 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 the historical estimate in Collins et al. (2008) 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 R190, X25, G03, P499, 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 (Mineral Resource Estimate) and Table 6-3 (Mineral 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 ("ID²") was used for estimation. This historical reserve estimate was given an appropriate dilution.

Table 6-2: Tamerlane 2008 historical Mineral Resource Estimate for
deposits R190, X25, G03, P499, O556, Z155
(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 R190, X25, G033, P499, O556, Z155
(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 Prices¹		
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 N204 deposit.

In 2011, 1,821 m in nine drillholes were drilled at the R190 deposit for geotechnical studies. Between 2005 and 2011, Tamerlane Ventures drilled a total of 11,726 m in 106 drillholes at the Pine Point Project (Gann, 2016).

The same year, in March, a mineral resource estimate was completed on the N204 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 mineral 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. ID² 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 N204
(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 R190 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 N204 deposit. The nine proposed open-pits were J68, HZ, W85, X65, M67, K68, M62, M63 and O53. The W85 deposit was one of the two North Trend Cluster Pits on which Tamerlane generated Open-pit Mining Reserves, the other being the X65 deposit. The 2013 proposal was based on previous work and economic studies completed in 2012 including the R190 and N204 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: J68, M67, K68, HZ, M62/M63 and O53 prismatic deposits.

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 mineral 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 J68, K68, X65, W85, M67, M63/M62, O53, 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. ID² was used for estimation. Open-pit mining costs, dense media separation costs, milling costs, and all ancillary costs were taken from Horlacher (2011).

The N204 deposit historical mineral resource estimate was carried over from Horlacher (2011) with the 2014 study applying new pit shell parameters for the reserves statement.

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



Table 6-6: Tamerlane 2014 pit shell parameters
for the J68, K68, X65, W85, M67, M63/M62, O53, and Hinge Zone historical Mineral 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 ("PPML") 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 mineral 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 J68, K68, X65, W85, M67, M63/M62, O53, and Hinge Zone deposits were carried over from (Siega and Gann, 2014), but the classification was downgraded.

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

Table 6-8: Darnley Bay 2017 historical Mineral Resource Estimate for deposits J68, K68, X65, W85, M67, M63/M62, O53, 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 W85 (226.7 m in two drillholes), at L65 and at the nearby K66 deposit (5,756.1 m in 54 drillholes and 193 m in two drillholes respectively), and at K60 (1,565 m in 17 drillholes). Regional exploration completed eight stratigraphic drillholes (846 m) along a haul road between deposits J69 and K77 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 mineral 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 R190, X25, G03, P499, 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 N204 deposit historical mineral resource estimates were carried over from Horlacher (2011) with JDS applying new pit shell parameters (Table 6-12). A summary is presented in Table 6-10.

The Hinge Zone, J68, K68, X65, W85, M67, M63/M62 and O53 deposits, also referred to as "the cluster pits" historical mineral resource estimates were carried over from Siega and Gann (2017) with JDS applying new pit shell parameters (Table 6-13). A summary of the historical mineral resource estimate is presented in Table 6-11 below.



Table 6-9: Darnley Bay 2017 historical Mineral Resource Estimate for deposits
R190, X25, G03, P499, O556, Z155
(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 N204
(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 pit shell parameters for the N204 historical Mineral 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 pit **shell parameters for the “cluster pits” historical Mineral 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 L35, L36 and L37 deposits were conducted in August-September 2017. This consisted of 29 drillholes that twinned historical 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 N42 deposit in the fall of 2017. Quantec Geoscience Ltd. collected 1,819 stations of gravity measurements over the Townsite grid, 455 stations over N42 grid and 430 stations over the mill site grid. Smaller grids with 50-100 stations were surveyed at L65, W85 and R190 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 N42 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 M40 and N38 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
K60	17	1,565.00		
K66	2	193.00		
L65	54	5,756.10		
L35	14	676.38		
L36	14	870.66	2 met holes: 126.14 m	Twinning/Met
L37	1	50.00		1,597.04
N38	1	50.00		
M40	2	121.00		
N42	7	541.95		Exploration/Gravity
W85	3	237.37	incl. 1 abandoned at 10.67 m	
Total	131	11,695.96		

6.5. 2018

In February 2018, Osisko Metals Incorporated acquired Pine Point Mining Limited from Darnley Bay Resources and became sole owner of the Pine Point Project. Between February 2018 and September 2018, Pine Point Mining Limited, a wholly owned subsidiary of Osisko Metals Incorporated, completed definition drilling totalling 23,751 m in 318 drillholes, which were included in the technical report titled Mineral Resource Estimate for the Pine Point Lead-Zinc Project (the "2018 MRE"), filed on SEDAR. The 2018 MRE was based on the following components:

- Location control of all historical Cominco drillholes (~18,000 holes) used existing UTM NAD83 coordinates in the database inherited from Tamerlane Resources and subsequently, Darnley Bay Resources. A number of holes were verified in the field and checked by hand-held, non-differential GPS (x and y ± 10 m, z ± 20 m);
- Original Cominco mine survey coordinates are available in the inherited database for general and relative position confidence;
- A total of 318 new holes drilled in 2018 for 23,751 m, all surveyed by DGPS;
- Only estimated mineral resources on mineral tenure held by PPML;
- Confidence established on areas of mineral resources reported by Cominco historically;
- Confidence established in continuity of grade and mineralization sufficient for an Inferred Mineral Resource category.

Table 6-15: 2018 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

Additionally, in 2018, PPML established the following as a formal universal grid reference system for Differential Global Positioning System ("DGPS"), including establishing control points for precise location control of all drillholes, geophysical surveys, bathymetric surveys, LiDAR surveys and infrastructure.

- Datum: NAD83 (CSRS) epoch 2010;
- Projection: UTM (Zone 11);

- Reference Meridian: 117 West Longitude;
- Vertical datum: CGVD 2013 (orthometric, i.e. above sea level).

PPML also began an extensive field program to locate and identify historical drill collars for survey by DGPS. At total 4,698 DDH sites visited and surveyed with varying level of certainty in the vicinity of the 2018 mineral resource areas (2018 MRE). Of these, 2,138 were found of high certainty (collar labels intact and readable). Based on this data, the remainder of the Cominco historical drillholes were transformed to a best fit x and y coordinate. In addition, a LiDAR survey was flown over the Project area and all holes were converted to the LiDAR elevation. The DGPS survey, transformation, and LiDAR correction did not include historical holes drilled by Westmin that are located west of the Buffalo River.

6.6. 2019

In 2019, PPML acquired addition mineral titles and expanded the Project area. PPML completed definition drilling totalling 47,263 m in 714 drillholes. These holes as well as the 2018 DGPS transformation of un-surveyed Cominco holes and LiDAR elevation correction were included in the technical report titled Mineral Resource Estimate for the Pine Point Lead-Zinc Project (the “2019 MRE”), filed on SEDAR. The 2019 MRE was based on the following components:

- Universal grid reference system established for DGPS control of all drillholes and infrastructure, including control points;
 - Datum: NAD83 (CSRS) epoch 2010;
 - Projection: UTM (Zone 11);
 - Reference Meridian: 117 West Longitude;
 - Vertical datum: CGVD 2013 (orthometric, i.e. above sea level).
- A total of 714 new holes for 47,263 m were completed and incorporated in the database supporting the 2019 MRE;
- Drill spacing of 30 m in some areas;
- Geological continuity further established;
- Grade continuity at the given cut-off grade further established;
- Global correction of Cominco downhole intervals to correct Cominco collar elevation datum reference to surface elevation reference;
- New drillholes added confirming the model (geologically and grade-wise);
- New claims staked covering additional historical mineral resources documented by Cominco not previously in the 2018 MRE;

- Newly acquired historical areas added to the model and estimated;
- Preliminary DGPS survey relocation program completed for historical Cominco drillholes;
- Included DDH sites visited and DGPS surveyed with varying level of certainty in the vicinity of the 2018 mineral resource areas;
- Transformation of the remaining un-surveyed holes was calculated based on 4,698 located historical holes;
- Elevation was corrected for un-surveyed holes to the 2018 LiDAR elevations.

Table 6-16: 2019 Inferred Mineral Resource Estimate

Method	Area	Cut-off Grade (ZnEq %)	Tonnage (kt)	ZnEq (%)	Pb (%)	Zn (%)
Pit-constrained Resources	Central	1.85	4,200	7.81	2.01	5.80
	East Mill	1.85	8,600	5.10	1.23	3.86
	North	1.85	16,400	6.18	1.80	4.38
	West	2.00	9,200	8.82	2.96	5.86
	N-204	2.10	9,500	4.53	0.97	3.58
Underground Resources	Central	5.00	2,700	7.57	1.60	5.97
	West	5.00	1,800	9.11	3.67	5.43
Total Pit Constrained		1.85 - 2.10	47,900	6.31	1.78	4.54
Total Underground		5.00	4,500	8.19	2.43	5.76
Total Combined			52,400	6.47	1.83	4.64

An airborne gravity gradiometry survey covering the central portion of the Project area, as outlined in the 2019 MRE, was also conducted by PPML in 2019, results and interpretation were completed in early 2020.

6.7. 2020

In 2020, PPML completed a Preliminary Economic Assessment and a technical report titled Mineral Resource Estimate for the Pine Point Lead-Zinc Project (the “2020 PEA”). Table 6-17 shows the results of the MRE included in the 2020 PEA study.



Table 6-17: Pine Point Mineral Resource Estimate

Method	Zone	Cut-off Grade (ZnEq %)	Indicated				Inferred			
			Tonnage	ZnEq	Pb	Zn	Tonnage	ZnEq	Pb	Zn
			(kt)	(%)	(%)	(%)	(kt)	(%)	(%)	(%)
Pit Constrained Mineral Resources	Central	1.85	1,700	7.31	1.71	5.61	3,200	7.89	2.02	5.86
	East Mill	1.85	6,000	5.38	1.39	4.00	3,800	5.05	1.02	4.03
	North	1.90	5,300	6.98	2.12	4.86	10,800	5.70	1.64	4.06
	N204	2.05	-	-	-	-	9,400	4.58	0.99	3.59
Underground Mineral Resources	Central	5.00	-	-	-	-	2,300	7.38	1.58	5.80
	West	5.00	-	-	-	-	8,200	11.04	3.78	7.25
Total Pit Constrained		1.85 - 2.05	12,900	6.29	1.73	4.56	27,200	5.48	1.37	4.11
Total Underground		5.00	-	-	-	-	10,500	10.23	3.30	6.93
Total Combined			12,900	6.29	1.73	4.56	37,600	6.80	1.91	4.89

6.8. 2020 to Present

From the closing of the drill hole database for the 2020 PEA, PPML has completed additional drilling. The majority of this drilling was conducted in the winter season and was split 90% infill drilling and 10% exploration drilling. Relogging of historical Tamerlane (2004-2012) and Cominco (1945-1985) era holes is ongoing. This effort adds to the confidence level of the historical data.

Table 6-18: Drillhole database status

	2017	2018	2019	2020	2021	2022 Winter	Total
Number of diamond drill holes	132	830	239	111	216	252	1,780
Number of reverse circulation holes					40		40
Number of diamond drill hole metres	11,759	55,263	12,154	6,614	14,299	14,105	114,194
Number of reverse circulation metres					3,830		3,830
Number of diamond drill hole assays					2,376	3,321	5,697
Number of reverse circulation hole assays					1,865		1,865
				In 2022 PEA Update			



Table 6-19: Historical relogging program

Historical Relogging Program	
	Total Relog
Number of drill holes	970
Number of metres	60,661
Number of assays	5,553



7. GEOLOGICAL SETTING

7.1 General Statement on Nomenclature

To mitigate confusion in terminology across historical documents, and academic literature and to update nomenclature to current understanding, the following practice is now used in the context of “Presqu’ile” Formation vs “Presqu’ile” alteration:

- The Presqu’ile Formation is no longer used. In all cases, the Sulphur Point Formation is used in proper context;
- Characteristic dissolution, precipitation of coarse, dolospar ± sulphides ± calcite and development of open pore space is now called Hydrothermal Dolomite (“HTD”) alteration replacing “presqu’ile dolomite”, or relatives of “presqu’ilization”;
- The only reference to “Presqu’ile” will be the Presqu’ile Barrier Reef Complex that occurs from Pine Point to the Rocky Mountains and encompasses reefal facies of the Keg River, Pine Point, Sulphur Point, Watt Mountain and Slave Point Formations, respectively (Figure 7-1).

7.2 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 Carbonate Hosted Zinc-Lead deposits which have similarities to, and differences from the Mississippi Valley-Type (“MVT”) base metal model sensu stricto. Recent observations by PPML staff suggest that while the stratigraphic controls remain, there is evidence that low temperature hydrothermal fluids caused the development dolomite alteration and the development of increased porosity as a result. Further, there is mounting evidence of structural influences on mineralizing fluids and that hydrocarbons may have played a role in the development and precipitation of sulphide mineralization.

Zinc-lead mineralization in the Pine Point Mining Camp (“PPMC”) is hosted in a dolomitized carbonate barrier reef complex that transects the entire Project area. 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 reef lithologies of middle-Devonian age.

The carbonate sequence dips gently from 1 to 3 degrees to the southwest and extends for up to 650 km into northern Alberta. These Middle Devonian carbonate lithologies are host to the Pine Point lead and zinc mineralization. The individual Pine Point deposits are widely distributed across the 70 km southwest trending belt covering up to 1,600 km².

Most deposits contain between 0.20 Mt and 2 Mt of mineralized material, although the largest, the X15 Deposit, had nearly 18 Mt (Sangster, 1990). Fifty of the 100 drill-defined deposits were developed in the PPMC 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.).

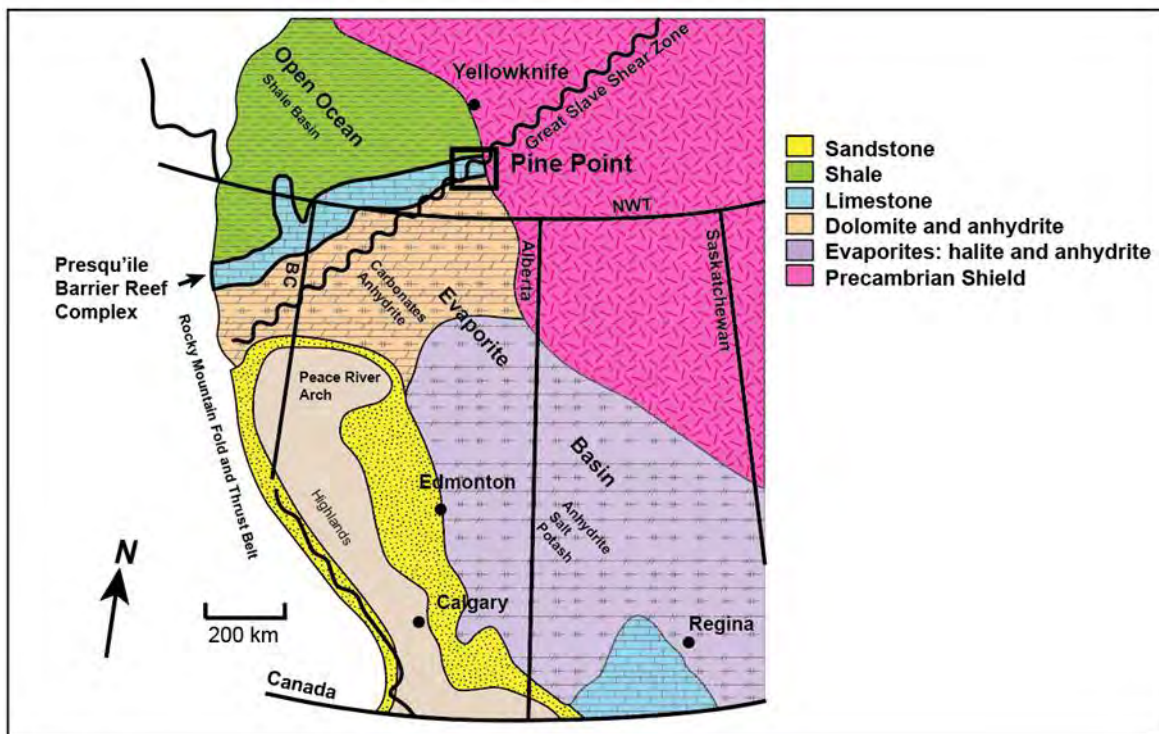


Figure 7-1: Regional geological setting
 (in Adair, 2020, after Gann, 2016)

The Pine Point deposits are hosted by low temperature (100 °C) HTD carbonates and subordinate limestone that form part of an extensive carbonate reef and platform sequence. Wall-rock alteration consists of large-scale hydrothermal dolomitization, which results in dissolution, recrystallization and local hydrothermal brecciation of the carbonate host lithologies. These properties combine with paleo-dissolution to create ground preparation for sulphide deposition at Pine Point.

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 Columbia (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 (Figure 7-1 and Figure 7-2). 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 7-3). This structural system can be traced with the aid of aeromagnetism and extends from within the Canadian Shield through the Pine Point area into northwestern Alberta and northwestern British Columbia. 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 X15 and N204 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. Structural interpretations of the area have been advanced by PPML using lineament interpretation as well as structural analyses from the available historical open-pits (Figure 7-3).

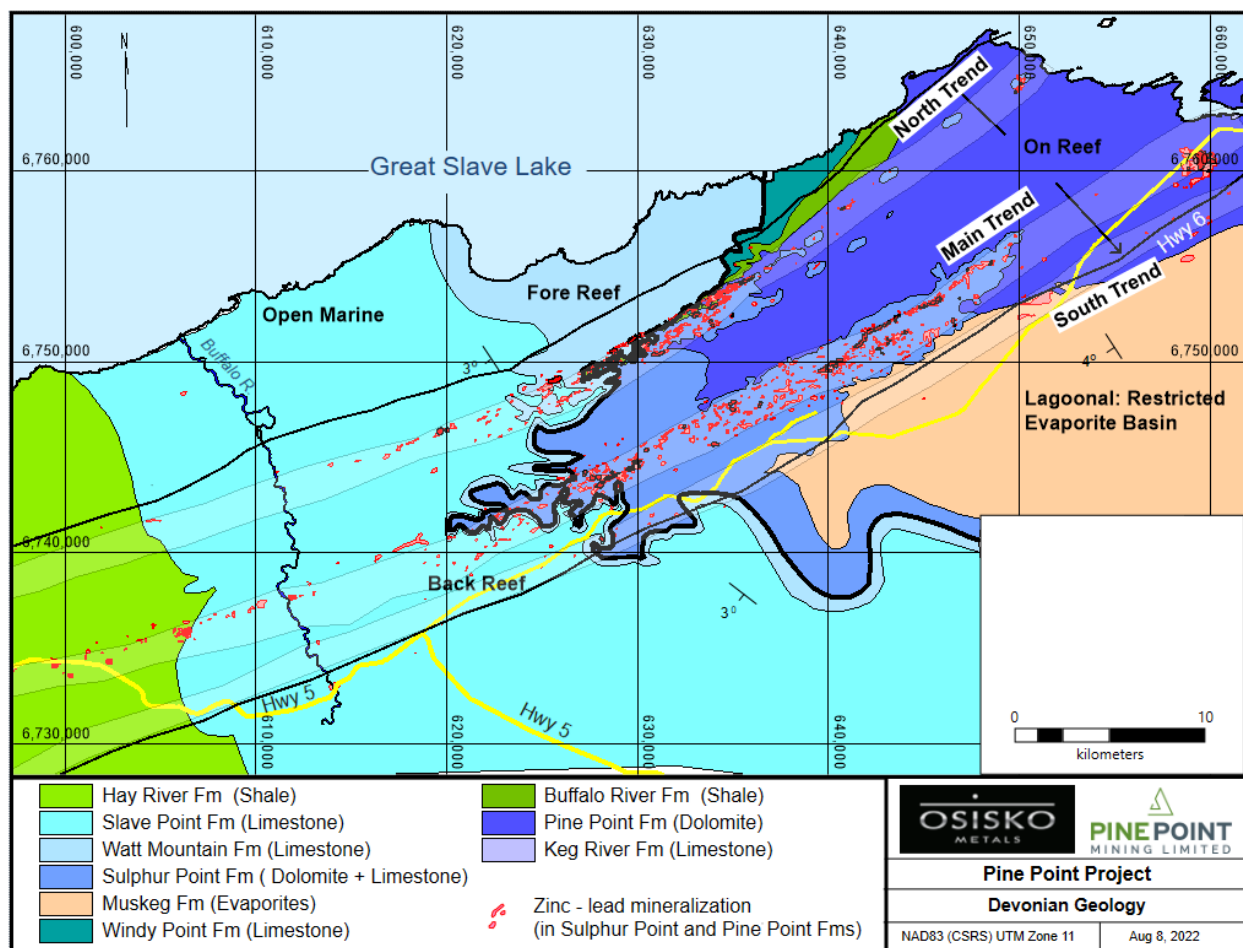


Figure 7-2: Pine Point Project geology
 (PPML, 2022)

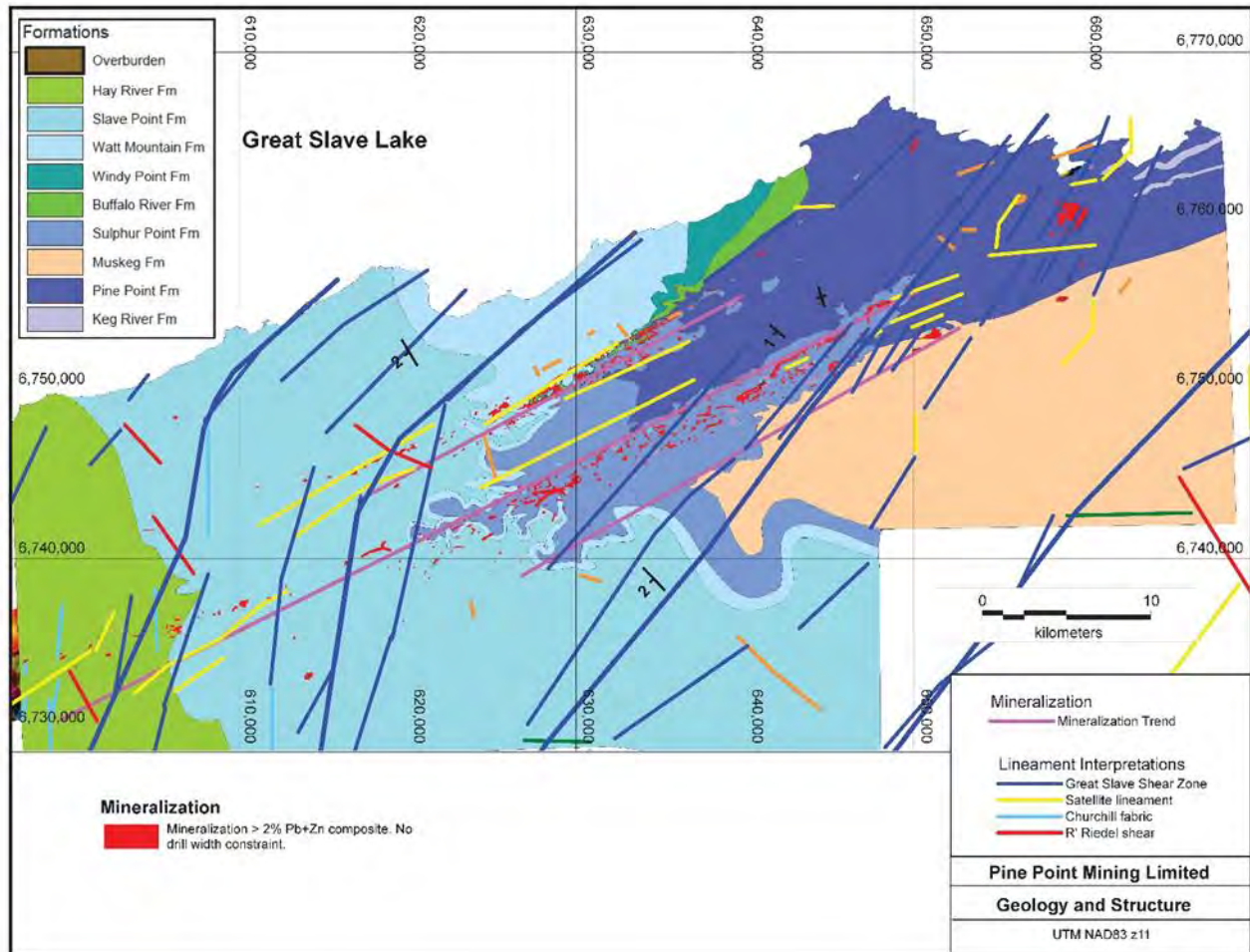


Figure 7-3: Pine Point Mining Camp revised geology map and structural interpretation

7.3 Pine Point Mining Camp Geology

The PPMC extends from east of 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 mineralization in the Pine Point and Sulphur Point formations. The Middle Devonian section is subsequently overlain by non-mineralized Middle and Upper Devonian sequences to the northwest and west whereas the Upper Devonian Sequence is represented by thick basinal shales and restricted carbonates dipping gently to the west District (Figure 7-2). This upper sequence represents the extensive open marine environment, contrasting sharply with the sub-tidal, sabkha and the shallow-marine environment represented by the Muskeg Formation comprising 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 grainstones to the east towards Fort Resolution (Figure 7-2). The gently inclined stratigraphy containing the west-dipping mineralized sequences sub-crops for over 45 km before shallowly plunging to the west under cover of the Upper Devonian basinal lithologies (Figure 7-2). 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).

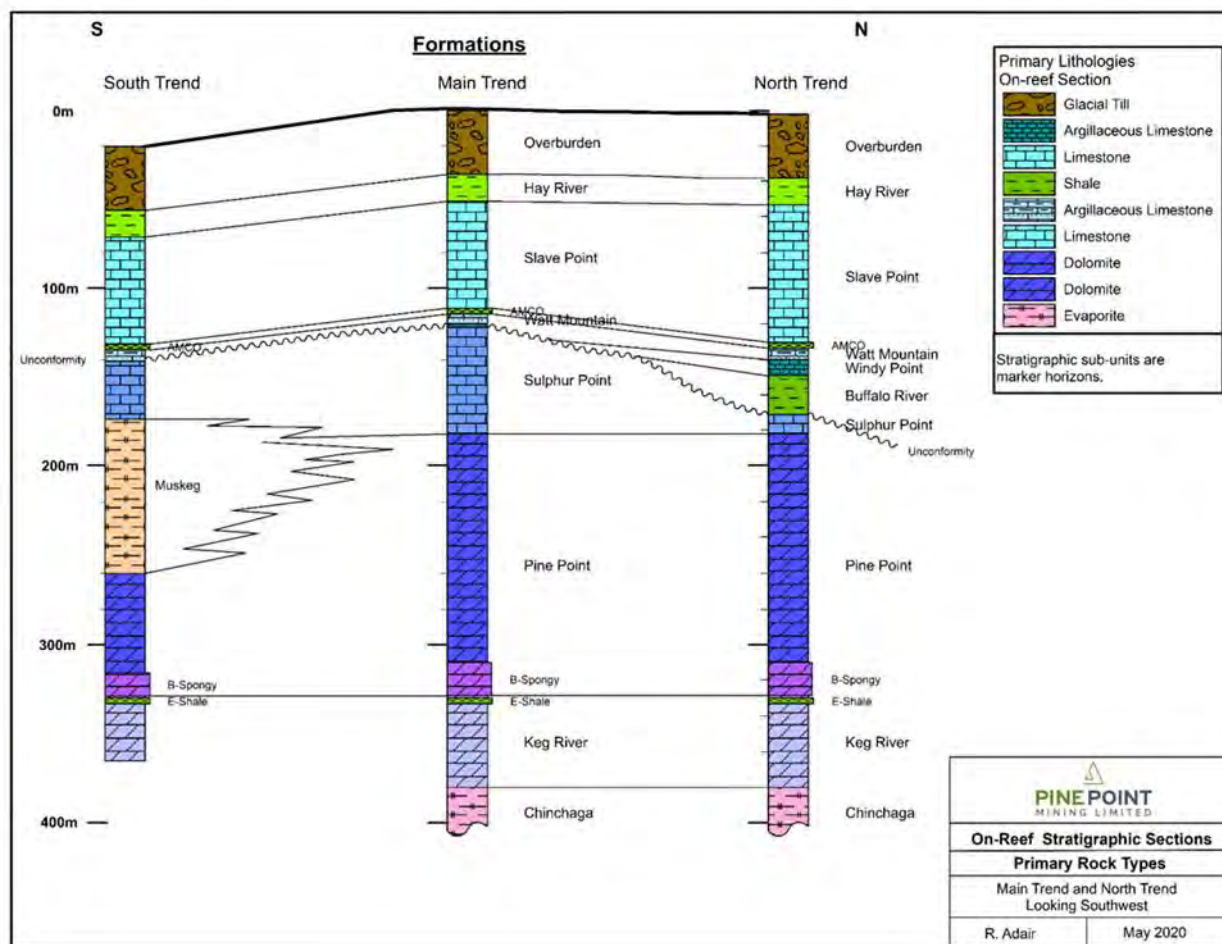


Figure 7-4: Geological section across the Pine Point barrier reef complex showing several Pine Point deposits and their stratigraphic setting within the Pine Point carbonate shelf-to-basin succession (Adair, 2020)

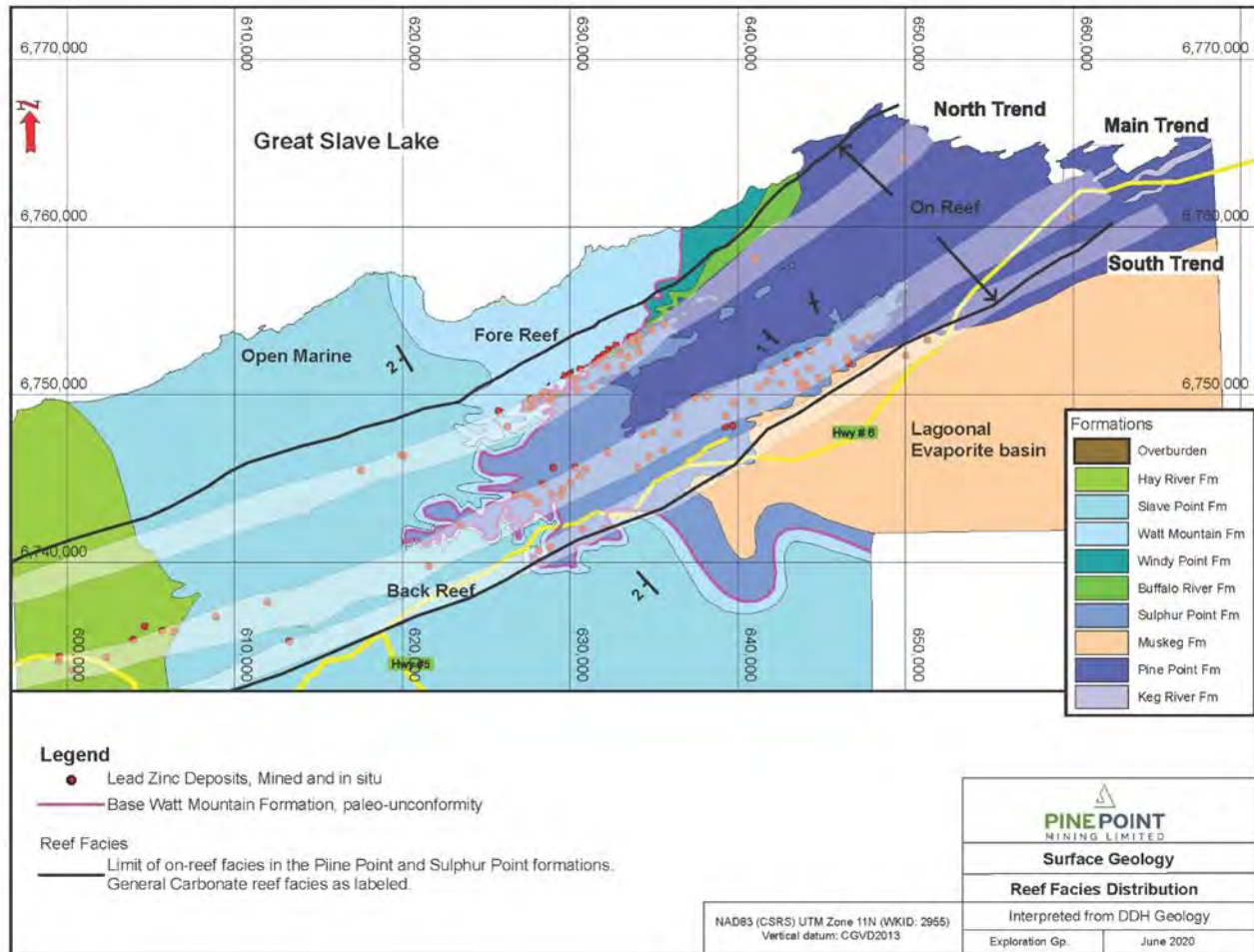


Figure 7-5: Mineralized trends

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 zinc-lead deposits, are un-deformed and form a 'layer-cake' west-dipping sedimentary sequence.

7.4 Project 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 600,000 m of core.

The stratigraphy is well defined but lacks modern carbonate stratigraphy terminology and sequence stratigraphic principals. The bulk of the Project is underlain by the prospective carbonate sequence. Underlying the western sector of the Project 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 zinc-lead 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.

7.5 Stratigraphy

The carbonate stratigraphy of the Pine Point area is elucidated and is largely based on the work of Skall (1975) (Figure 7-4). The basic subdivisions are:

Keg River Formation | Basal, regionally extensive shallow-water bioclastic and biomicritic limestone comprising a carbonate platform sequence.

Pine Point Formation | An extensive accumulation of shallow-water grainstones, calcarenites and fine biocalcarenes 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 stromatoporoid 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 stromatoporoid reef structure with stromatoporoid bioclastic debris accumulations interspersed with abundant micrite piles. The Sulphur Point Formation is a principal host to the Pine Point 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 stromatolite 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 limestone 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 limestone (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 stromatoporoidal debris. The O-facies is commonly mineralized in some of the North Zone Deposits, notably W85.



Table 7-1: List of formations after Skall (1975), Lantos (1983), and Rhodes et al. (1984)

Formation Name	Marker Horizon	Protolith	Notes
Hay River (+ 15 m)		Shale	Deeper marine platform
Slave Point (50-75 m)		Limestone	Marine platform
	AMCO (1-3 m)	Blue-grey argillaceous limestone.	Marine platform
Watt Mountain (7-15 m)		Green, shaley limestone	Lagoonal, restricted marine.
Windy Point (10-50 m variable, on-laps reef from the north)		Limestone	Marine, reefal, time stratigraphic equivalent to upper sulphur Point.
Buffalo River 20-25 m, variable, on-laps reef from the north)		Shale	Shallow, open marine platform.
Sulphur Point (0-80 m) Eroded to the north.		Limestone	Reefal and marine platform
Muskeg (85-100 m, variable, on-laps reef from the south)		Interfingered dolomite and evaporite with the later becoming dominant to the south of the reef.	Off-reef, restricted back-reef evaporite basin. Time stratigraphic equiv. of the Pine Point Fm and basal Sulphur Point Fm.
Pine Point 40-140 m		Dolomite and dolomitized limestone.	Reefal and marine platform
	B-spongy (5-20 m)	Dolomite, vuggy.	Marine bioherm, reefal?
Keg River (61-73 m)		Dolomite, locally argillaceous.	Marine Platform
	E Shale (5-10 m)	Shale or shaly dolomite	Marine platform. 3-6 m below the top of Keg River Fm.
Chinchaga (76 m)		Anhydrite, crystalline dolomite, quartz sandstone, dolomitic shale, halite.	Restricted, back-reef basin

7.6 Hydrothermal 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 (Figure 7-6, Figure 7-7 and Figure 7-8). 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.

Alteration within the PPMC stratigraphy is broadly characterized by secondary dissolution of carbonate rocks by elevated temperature fluids under hydrostatic pressure that exploit porous and permeable rocks, fractures, paleokarst and breccia zones producing pore space precipitation of secondary dolospar ± sulphides ± calcite. Dissolution may be stratabound, within beds of higher porosity and permeability (primary and paleokarst), particularly within the Sulphur Point Formation, and cross-cutting along fractures, breccia zones and paleokarst. Continued fluid flux re-exploits earlier-formed alteration zones due to the open space development and most likely results in continued development of collapse architecture. The resultant plumbing system influenced areas of precipitation of sphalerite, galena and lesser marcasite in both Tabular and Prismatic deposits. Fluids were briny, basin-derived and were likely at elevated temperatures upwards of ~100 °C (Turner, 2006). The mineralized system extends over a strike length of 65 kilometres on the Pine point Project and is documented further west and north of the Project area.

The precipitation of coarse, sparry dolomite ("saddle dolomite", "dolospar") termed hydrothermal dolomite ("HTD"), is contemporaneous with the above-mentioned fluids and preferentially precipitates in dissolved beds, fractures and breccia zones where it develops within voids as well as replacement of primary carbonate. Bedding, breccias, vugs, karst and caverns are exploited and may be re-brecciated/dissolved, followed by continued precipitation (Figure 7-6). Evidence suggests that HTD was precipitated from an elevated temperature, slightly saline, calcium-rich fluid that was funneled upward along faults and fractures that developed during the Late Cretaceous to early Tertiary Laramide Orogeny that provided the driving force of basinal fluids.

Zinc, lead, and iron sulphides are mainly precipitated through sulphur from dissolved anhydrite/gypsum (CaSO_4 / $\text{CaSO}_4 + 2\text{H}_2\text{O}$) and/or reaction of H_2S gas and/or bitumen dissolved within basinal fluids, or present within the host rock. Calcite is the last precipitated mineral. It generally forms a permeability barrier and defines the late-stage hydrothermal vectoring, often associated with porous collapse features.

The HTD process, sulphide mineralization and precipitation of calcite may partially or completely close the permeability of the system.

HTD is grouped into two general types:

1. **Lateral, bedding-parallel replacement** in susceptible beds results in broad lateral dissolution, concomitant open space development (porosity and permeability), and precipitation of HTD. The intensity of the dissolution and replacement is bedding specific and focused on preferential lithologies (Figure 7-6, Figure 7-9 and Figure 7-11). Tabular deposits and related HTD channel are associated with this style of alteration. The Sulphur Point Formation is the unit most exploited laterally and exemplified by pervasive HTD to varying degrees as below. Lateral replacement is largely restricted to the Sulphur Point Formation with discordant replacement (restricted lateral replacement) in the Pine Point, Buffalo River, Watt Mountain and lower Slave Point Formations. At the base of the Sulphur Point Formation, a porous unit related to the dissolution of a micrite facies and HTD development is present at the base of the Sulphur Point in the East Mill and Central Zones. It is called the dismicrite marker ("DM"), Figure 7-6).



Figure 7-6: Bedding – Parallel replacement by HTD - Sulphur Point Formation

2. **Discordant replacement** cross-cuts stratigraphy and is associated with both Tabular and Prismatic deposits. It tends to be much more laterally restrictive and intensities drop off over short distances laterally, likely to the lower primary porosity of the host rock, usually dolomite (Figure 7-8).

Relationships between bedding-parallel and discordant replacement is not proven, but they appear to be linked in Tabular and Prismatic deposits developed in the Sulphur Point Formation where connections between stratabound channel ways (Tabular trends) and vertical breakouts

(Prismatic deposits) is postulated. Furthermore, the BSpongy marker horizon is interpreted as a lateral channel way for mineralizing fluids.

Alteration intensity is broadly characterized as follows with a notable increase of intensity in channels and in vertical conduits that host Tabular and Prismatic deposits, respectively:

1. Weak: <15% of total rock:
 - a. Local small, Tabular voids developed along bedding planes and within beds with primary porosity.
 - b. Fracture-fill and replacement of cross-cutting features (sedimentary and structural).
2. Moderate: >15 / <60% of total rock:
 - a. Tabular voids developed along bedding planes and within beds with primary porosity, more continuous laterally with local vertical connection.
 - b. Fracture-fill and replacement of cross-cutting features (sedimentary and structural).
3. Strong: 60% - 100% of total rock. Replacing most of the original rock:
 - a. Along bedding planes. The DM (dismicrite) horizon in the Main Trend at the base of the Sulphur Point Formation is an example of a fully replaced bed underlying most of the Tabular system in the Main Trend with local large, metre-scale voids.
 - b. In vertical pipes associated and in proximity to Prismatic deposits. The pipes will exploit and replace bedding, but lateral extent is limited to the proximity of the deposit.

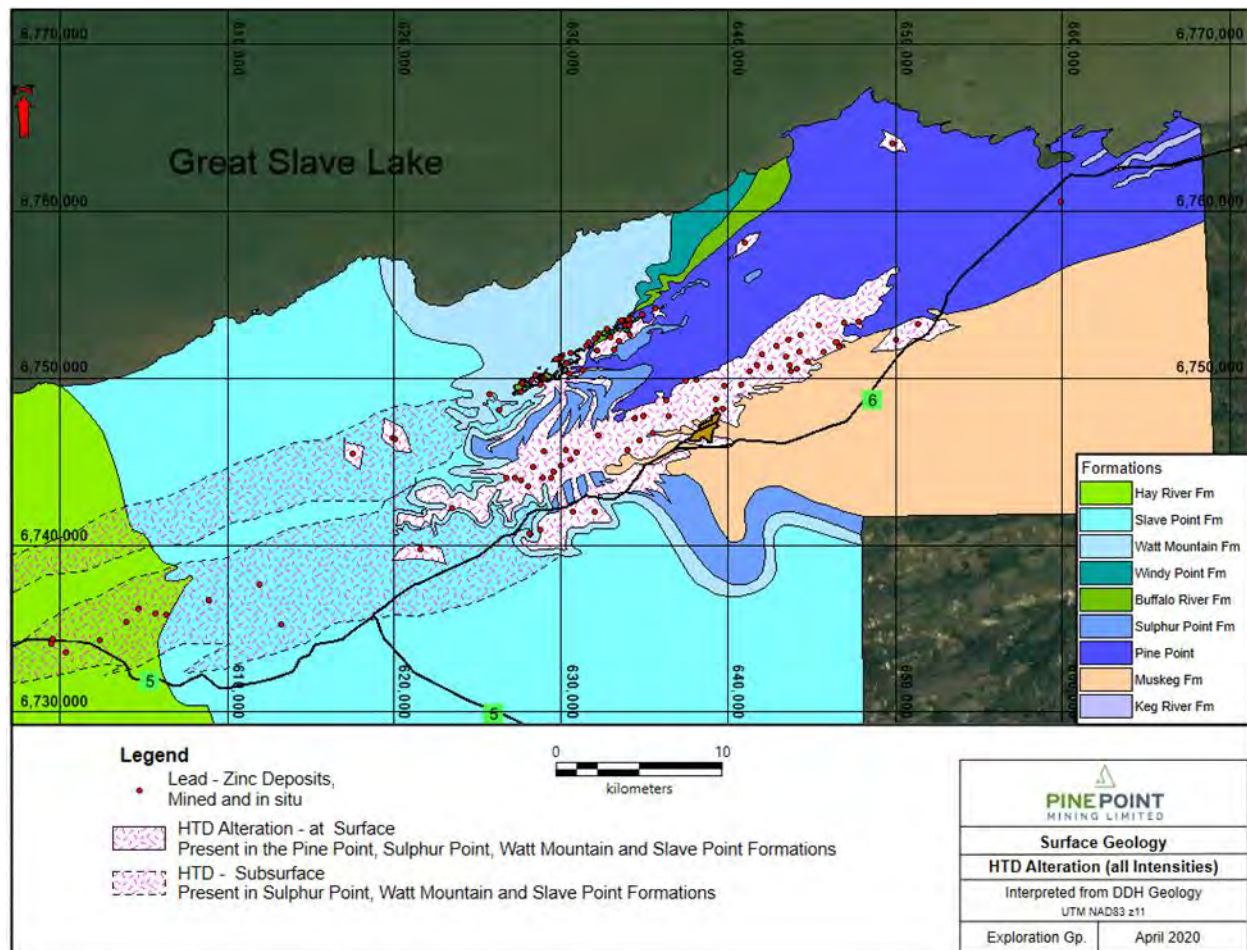


Figure 7-7: Sub-overburden map of interpreted distribution of HTD alteration, largely focused in the Sulphur Point Formation

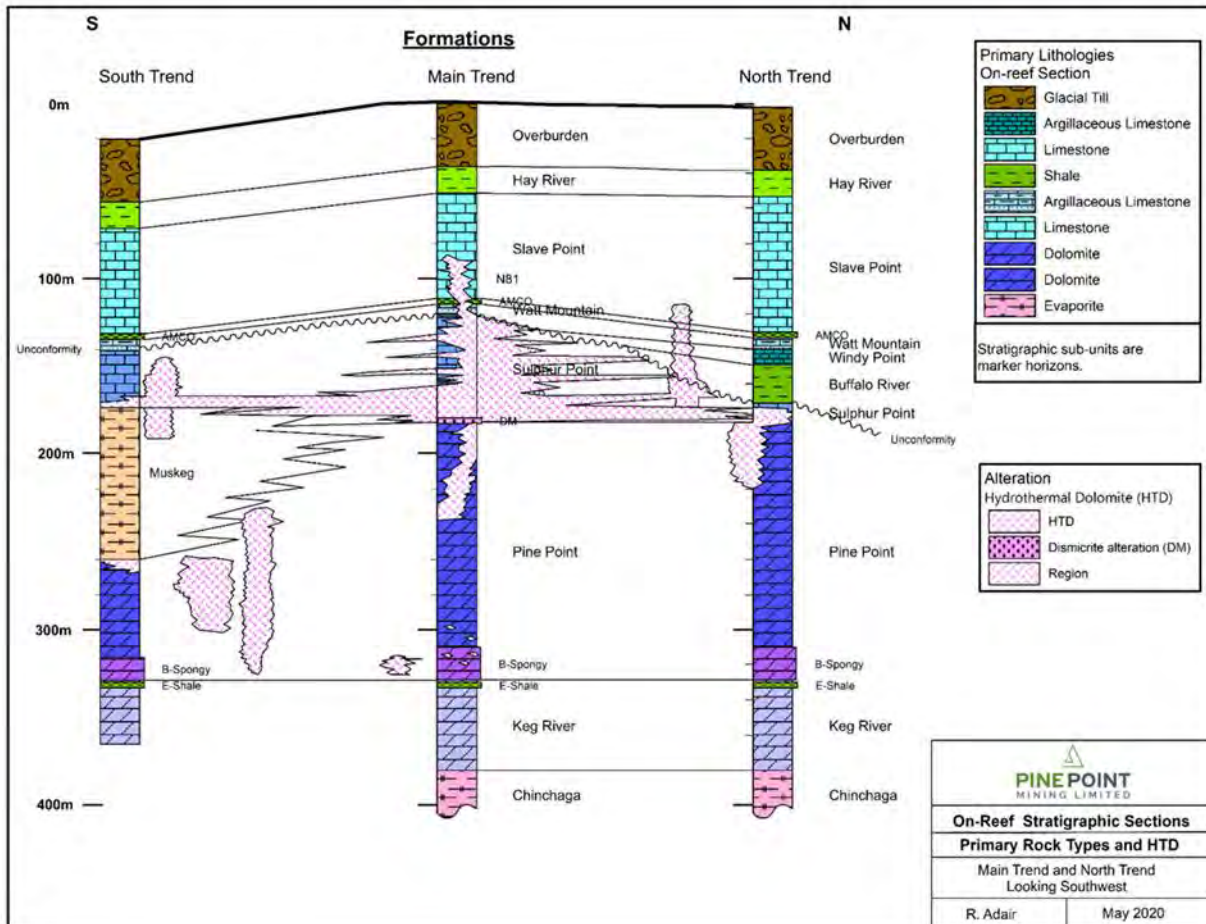


Figure 7-8: On-reef lithology and stratigraphy with schematic HTD superimposed

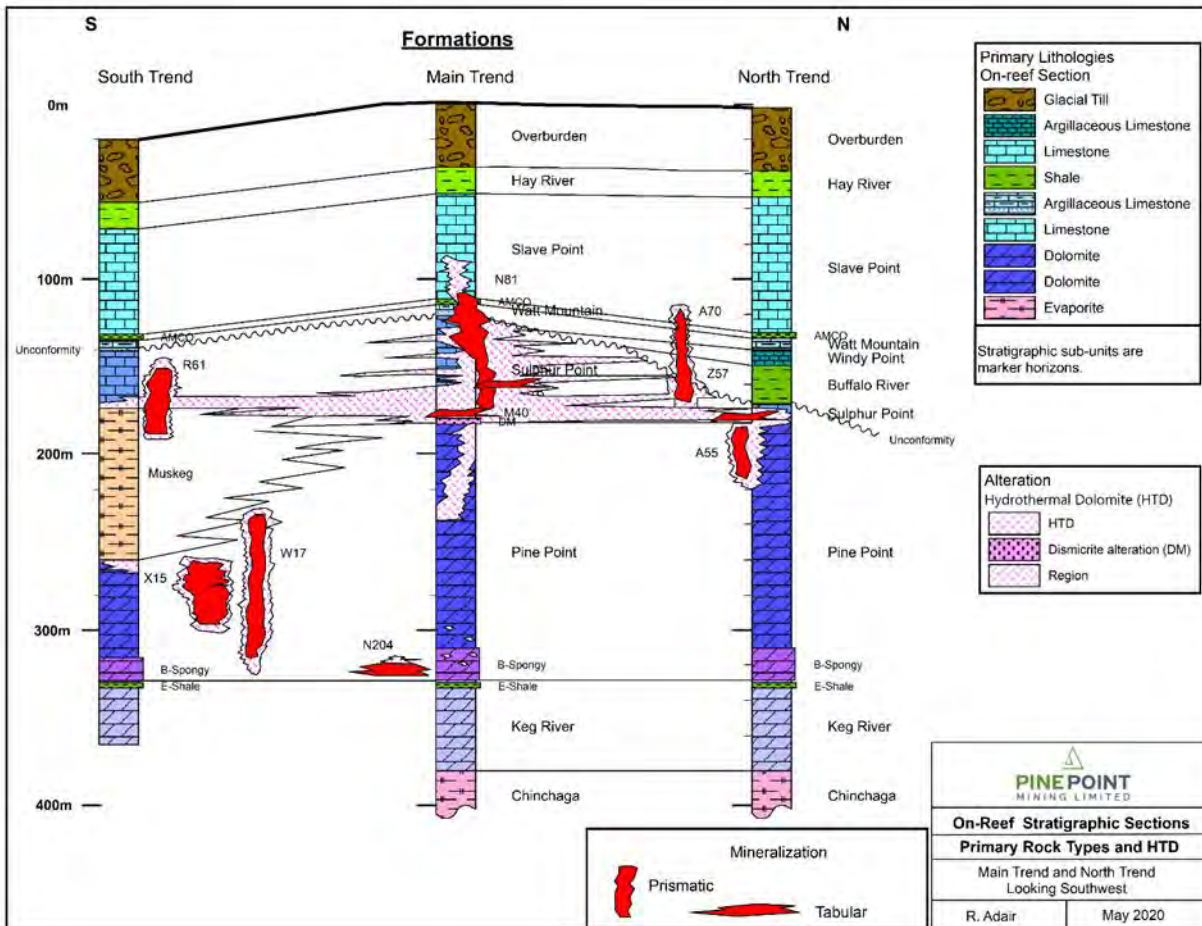


Figure 7-9: On-reef lithology and stratigraphy with schematic HTD and mineralization superimposed

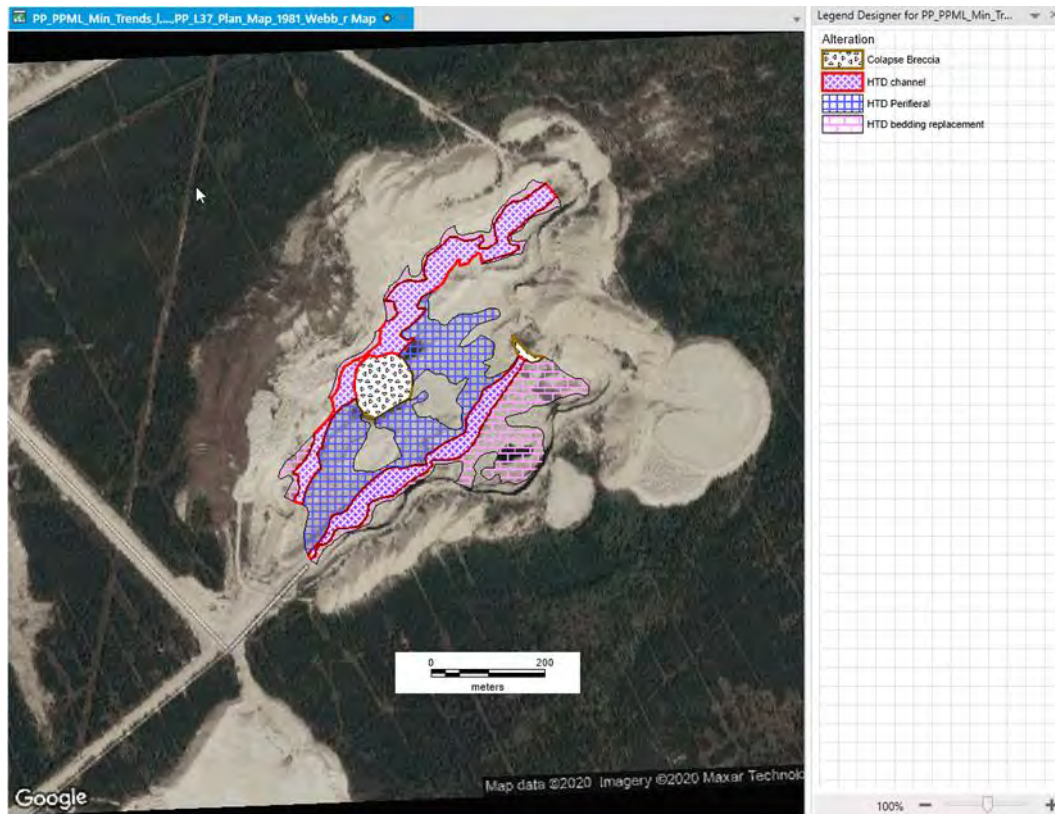


Figure 7-10: Alteration map of the L37 pit

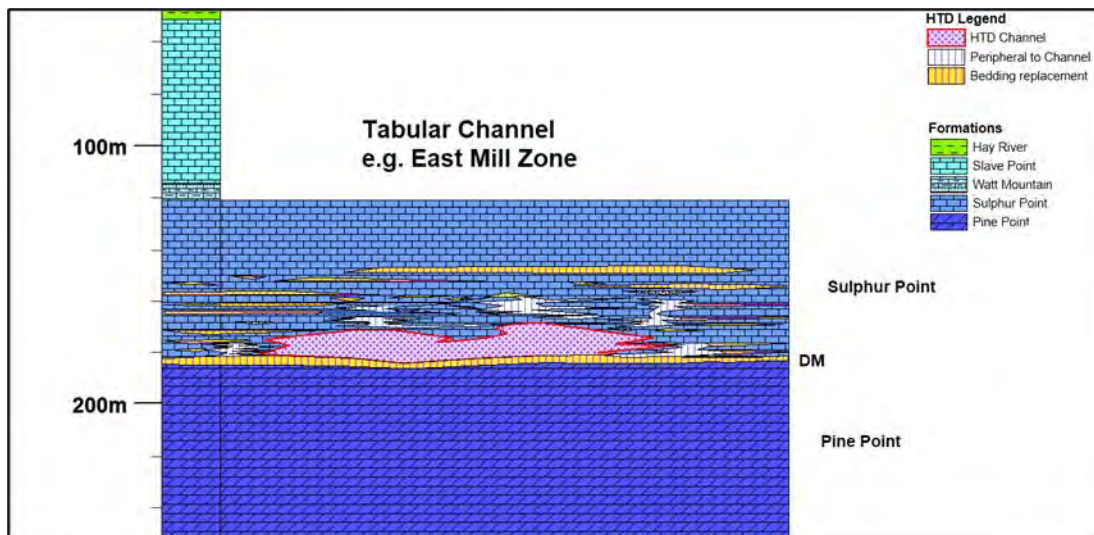


Figure 7-11: Schematic cross-section through the L37 pit
 DM = dismicrite horizon

HTD is well-developed in Sulphur Point Formation associated with the dominantly limestone lithology and primary porosity distribution strongly influenced by reef structures and likely secondary brittle structures. It is best developed in the Main Trend and to a lesser extent in the North Trend where it is dominantly stratabound within the reef build-up. In other formations it is tightly restricted to areas of mineralization. Within the Sulphur Point Formation, the intensity is highest in the Main Trend reef structure in the Tabular channels and in the vicinity of Prismatic deposits. It dissipates between the Main and North Trends and is only patchy in the South Trend.

7.7 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 ($\text{CaSO}_4 + 2\text{H}_2\text{O}$), 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.

7.8 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 unconformity 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.

7.9 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, 1994a).

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, CaCl₂, KCl, and MgCl₂ corresponding with salinities ranging from 10 to 28 wt% equivalent NaCl, three to eight times the salinity of seawater.

7.10 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 limestone and transitions between carbonates and shales. Chemical and mineralogical composition influenced how rocks were altered by hydrothermal fluids:

- Limestone was 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.

7.11 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. 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).

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).

7.12 Hydrothermal Alteration Within the North Trend

The flow of hydrothermal brines was controlled by shale or shaly formations that acted as aquitards (Figure 7-12). 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.

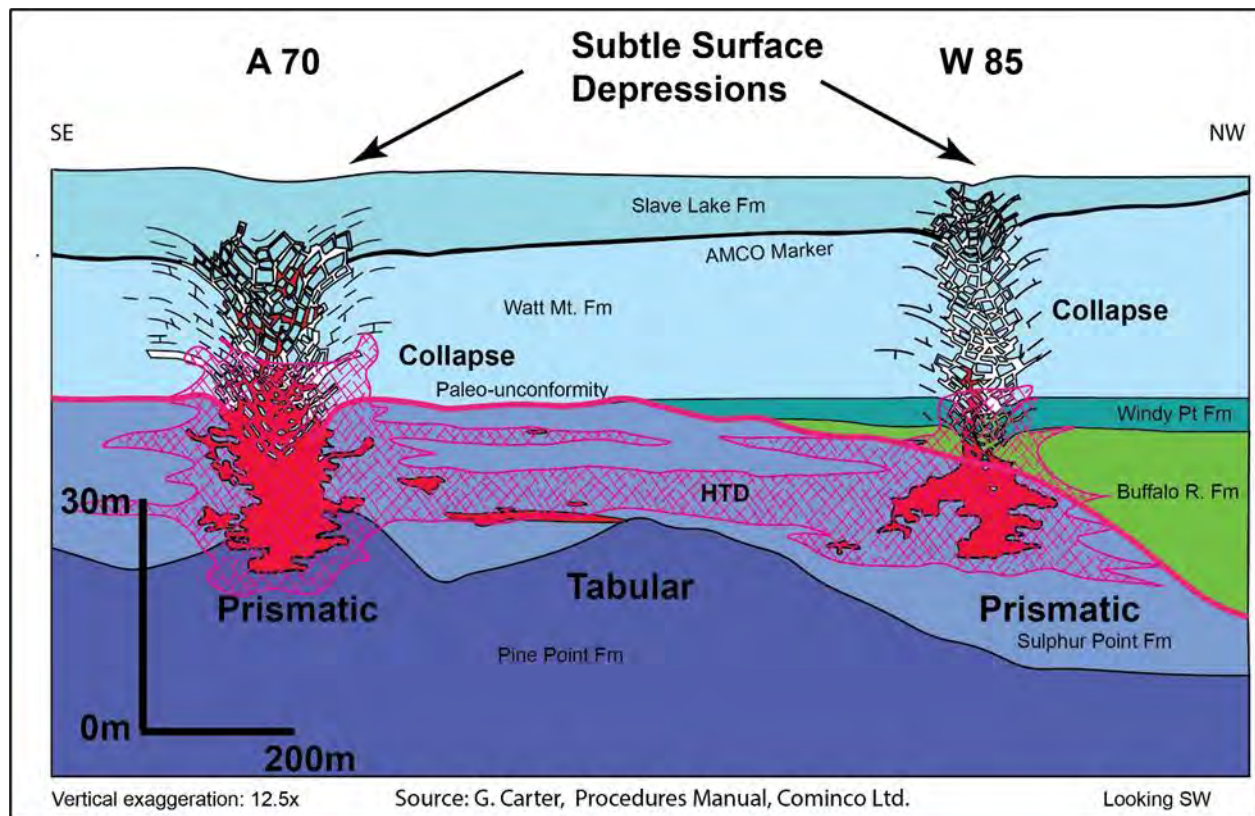


Figure 7-12: Composite section through North Trend showing Prismatic deposits

7.13 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. Consequently, 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 7-13.



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.

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 with aspects of Mississippi Valley-Type (“MVT”). MVT deposits and 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 such 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 km² and define a district comparable in size with the Tri-State MVT district in the USA (Leach et al., 2010).

The characteristic features of 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).

New characteristics:

- Likely structural controls on fluids and mineralization;
- Initial dissolution of carbonates, development of dolomitization, development of porosity/permeability and mineralization is the result of low temperature hydrothermal fluids under pressure;
- Sulphide precipitation has a relationship to the presence of hydrocarbons.

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. **N204 type mineralization.** The N204 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

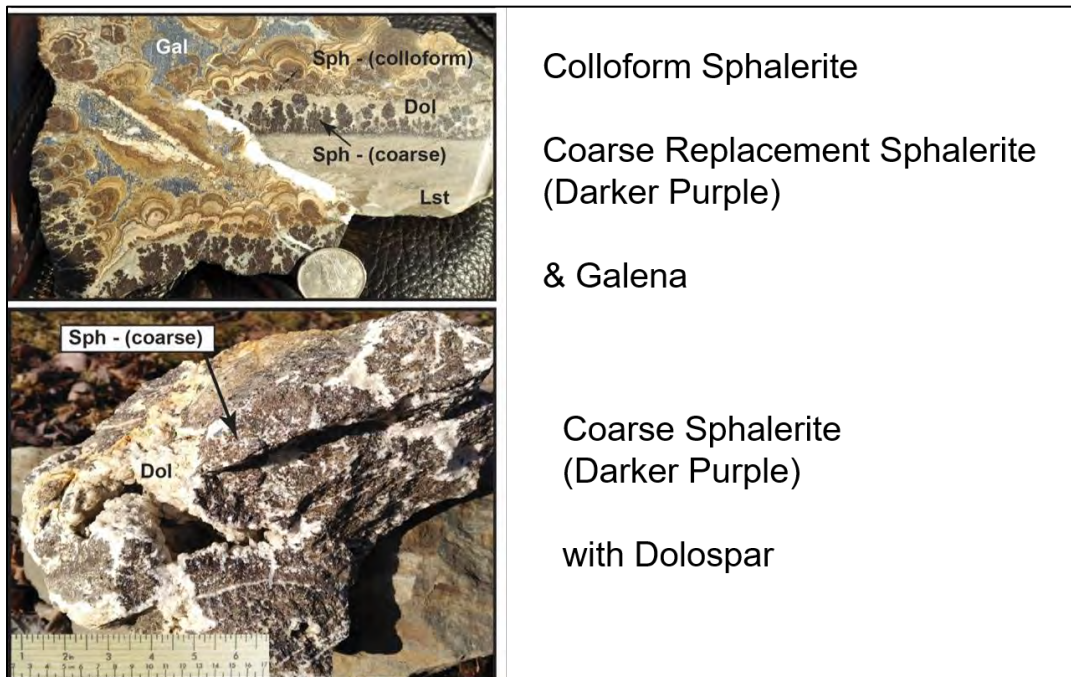


Figure 8-2: Examples of mineralization in hand specimen



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. **N204 Deposit** – N204 mineralization consists of fine crystalline mineralization deposited within the porosity of strati-bound 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 N204.

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.

8.2.3. N204 Deposit

A related type of mineralization is found at N204 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 N204 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 focused 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 N204 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 heterogeneous 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. 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 strata-bound 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.

9. EXPLORATION

Recent exploration, from the period of September 13, 2018, to the time of this report, has focused on increasing the confidence of historical mineral resources areas. Exploration drilling has targeted the edges of mineralized zones for potential expansion. Exploration programs on the Project have consisted primarily of diamond drilling but have also incorporated various studies including unmanned aerial vehicle ("UAV") photogrammetry, LiDAR-based topographic surveys and an aeromagnetic survey in conjunction with an airborne gravity gradiometry survey. In addition, analytical geochemistry has been initiated to quantify Gallium and Germanium ("Ga/Ge") contents of mineralized material.

9.1. Gallium and Germanium (Ga/Ge)

9.1.1. Re-verifying Pulps

A pulp verification program has been planned and initiated to ensure the quality of Ga/Ge values of previously obtained ICP-MS laboratory assay results. A random representative population of samples was collected from stored pulps and re-assayed. New Mississippi Valley-type ("MVT") standards with certified Ga/Ge values have been inserted at regular intervals along with blanks. This information will be used to ensure the quality of geochemical results and to increase the confidence of previously reported Ga/Ge values measured through previous assays.

9.1.2. Changing Standards to Certified Ga/Ge

In order to accurately report Ga/Ge values to NI 43-101 standards, new standards have been introduced to replace previously approved standards. The company believes these standards will better represent the lithologies at Pine Point and permit certification of all reported metals relevant to resource calculations and net smelter return ("NSR").

9.2. LiDAR – Exploration Applications

In fall 2018, and subsequently in the fall of 2019, LiDAR surveys were conducted, and now cover the entire Pine Point Project, by LiDAR Services International Inc. using a MATRIX LiDAR system installed in a Cessna 206 airplane. The survey covered an area of 1,320 km² and generated a cm-scale Digital Elevation Model ("DEM") across the modern topography of the Pine Point site. This fine scale sub-vegetation DEM will be used to identify subtle trends in ground elevation that may be

indicative of underlying bedrock conditions. In addition, the topographic data includes roads, pits, waste piles and shallow ponds that play an important role in corrections applied to gravimetric studies. In tandem with the LiDAR analysis, the airborne platform also collected ultra-high-resolution orthophotography to accompany the DEM. Updated aerial photographs are being used to facilitate exploration planning and targeting.

9.2.1. Terrain Corrections

Using the DEM generated by the LiDAR survey, a set of high-precision terrain corrections were performed to correct the Airborne Gravity Gradiometry ("AGG") survey. These terrain corrections are required for the AGG survey due to the reliance on small amplitude short wave-length features in observed gravity. In areas with high contrasts in material density, such as water-filled pits and waste piles, topographical features can have strong effects on gradiometry and produce erroneous features. Terrain corrections are essential to minimize these effects. The final QA/QC for the 2019 LiDAR survey were not received in time for incorporation into the 2020 MRE.

9.2.2. Collapse Structure Analysis

An ancillary application for the high-precision LiDAR is the ability to penetrate the vegetation and create a bare earth DEM. By removing vegetation, less obvious depressions in the landscape can be solved and used to locally discover karstic ground. The relationship between karst and mineralized terrains was defined by previous operators and can be used as an exploration vector. The bare earth DEM will be used as an exploration tool for future target generation.

9.2.3. Structural Analysis

The exposed outcrop in the historical open-pits was analyzed by Terrane Geosciences Ltd. to generate a structural interpretation of the discontinuities in the pit walls. These structures appear in the LiDAR DEM and were also investigated in a smaller scale UAV photogrammetry survey (more details in the section below). Combining both datasets, the structures were interpreted and used to better understand possible controls on mineralization.

9.3. Unmanned Aerial Vehicle (UAV) Photogrammetry

In late summer 2018, a UAV photogrammetry survey was conducted in many of the historical open-pits left open by Cominco Mining Ltd. The surveys focused on gathering geological information within the walls and benches of the historical open-pits to gain understanding of geometry and structures within mineralized terrains. The platform collected cm-scale precision



data and was used to develop 3-dimensional models of the surveyed pits to facilitate structural analysis. Terrane Geoscience submitted a report describing structural features in the pits and interpreting correlations with larger features observed across the district. This information is being used to further exploration efforts.

9.4. Project Scale Structural Analyses

Terrane Geoscience was contracted to conduct a structural analyses and interpretation based on satellite imagery, large-scale government magnetic and gravity data, the 2004 AEM survey conducted by Tamerlane, the 2018 and 2019 LiDAR, and detailed analyses from UAV photogrammetry in available (unflooded) historical open-pits. The results of this investigation revealed potential structural influences on mineralization and subsurface hydrogeological systems.

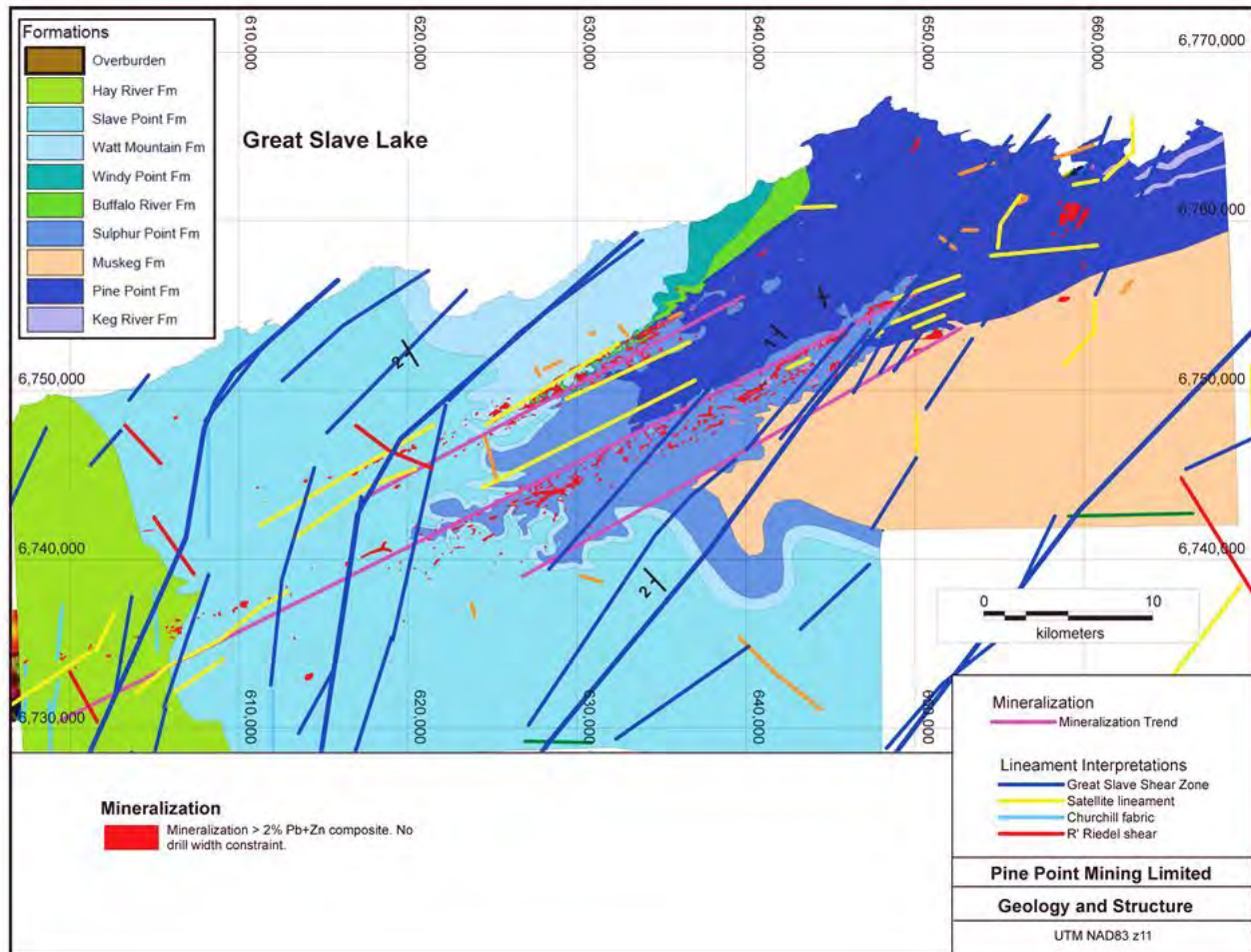


Figure 9-1: Interpreted structural geology

9.5. Gravity Gradiometry

In summer 2019, CGG Canada Services Ltd. conducted a heliborne airborne gravity gradiometry survey (Figure 9-2 and Figure 9-3) to identify areas of density contrast that may be indicative of mineralization. Line spacing was 50 m and a total of 4,682 line-km of survey was flown to cover the survey area that included known occurrences. The data was processed, and the results are now compiled with the geological and geophysical datasets to identify new targets for exploration purposes. Furthermore, the 2019 Gravity Gradiometry as well as baseline induced polarization ("IP") surveys conducted in 2020 have both returned responses indicative of collapse features, with the latter directly associated with mineralization at the W85 and R67 Prismatic deposits.



Figure 9-2: Airborne gravity gradiometry survey coverage area

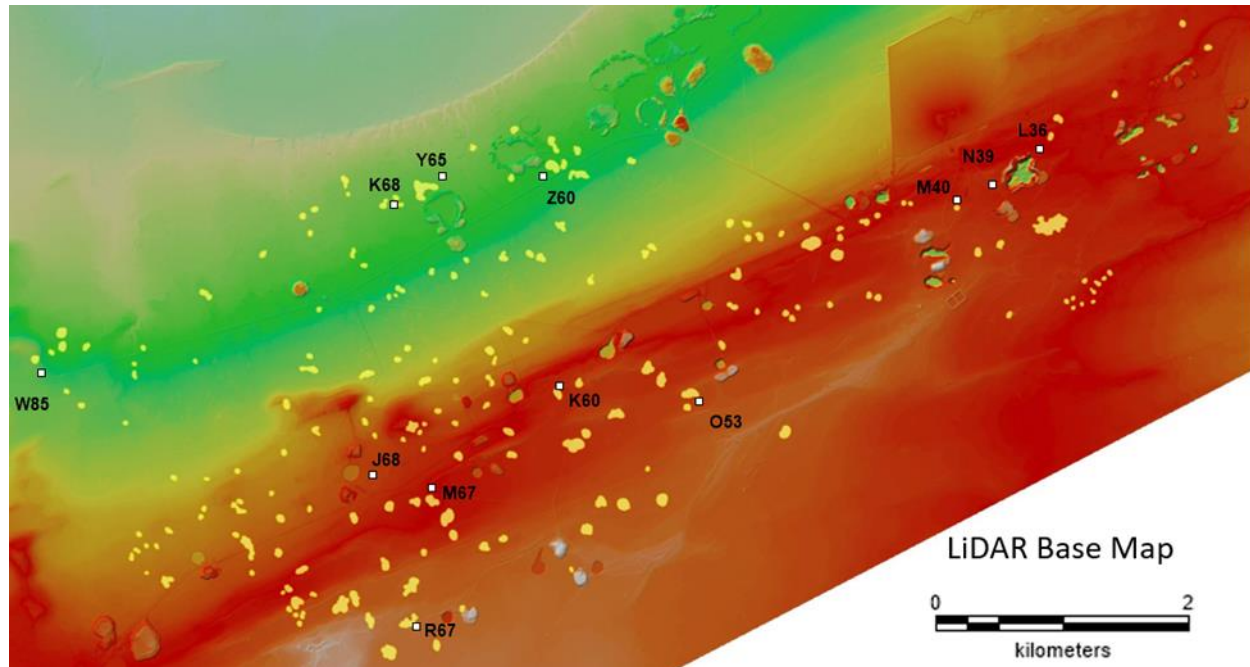


Figure 9-3: Airborne gravity gradiometry lows – collapse structures, LiDAR base map

9.6. Aeromagnetic

Concurrent with the airborne gravity gradiometry survey, CGG Canada Services Ltd. conducted an aeromagnetic survey. The magnetic gradient data was collected with a two-sensor horizontal magnetometer. This data is now processed and is being used to refine exploration targets. This survey provided hyper-detailed mapping of the magnetic characteristics within the Airborne Gravity Gradiometry and matched very well with the 2004 AeroTEM magnetic data flown at 200 m linespacing.

9.7. Induced Polarization (IP)

In early 2020, IP baseline surveys were conducted over the W85 and R67 Prismatic deposits that are within the 2020 MRE using an offset pole-dipole array (Dziuba and Geoph, 2020) (Figure 9-4). The results successfully imaged the deposits, supporting the use of IP as a key survey in assessing collapse structures, orphan holes containing mineralization that have not been followed up and airborne gravity gradiometry surveys.

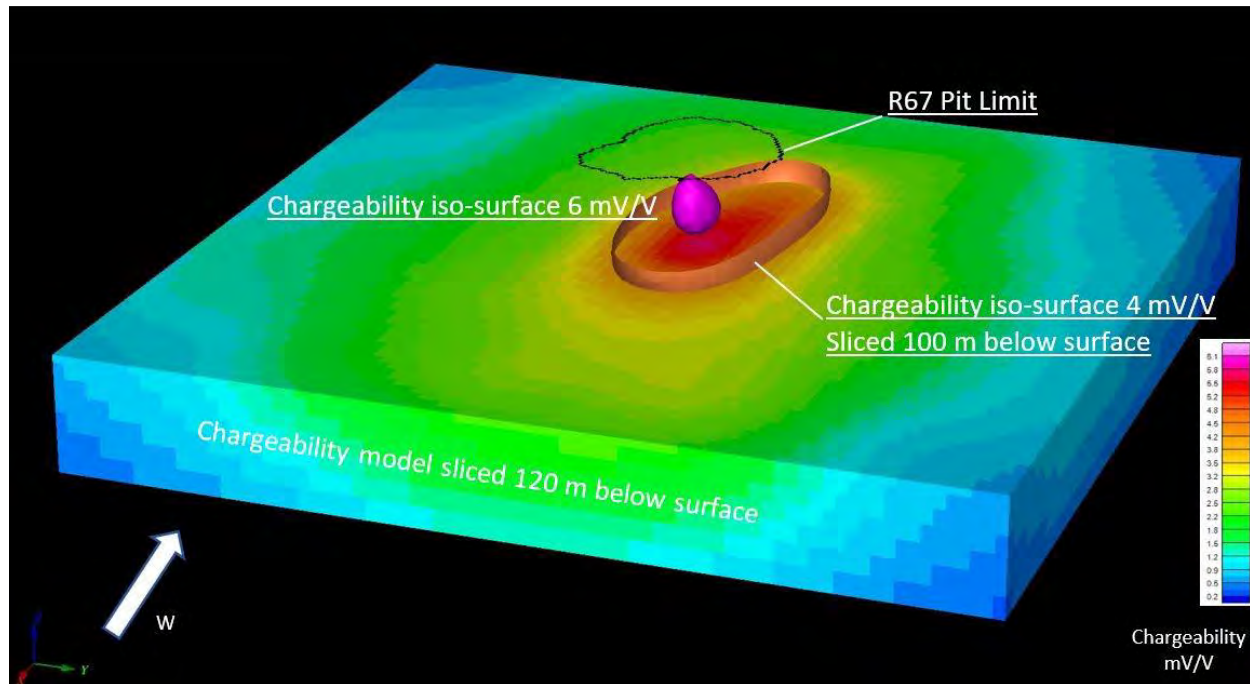


Figure 9-4: R67 chargeability model base and anomalous iso-surfaces (perspective view, looking southwest)
(Dziuba and Geoph, 2020)

9.8. Orphan Holes

Based on reported assays in historical Cominco and Westmin drillholes, there are multiple occurrences of significant mineralization in drillholes that have not been investigated or fully investigated by subsequent drilling. These are “orphan holes” and these occur within and near MRE resource boundaries, and within widely-spaced exploration areas well outside of the current MRE (Figure 9-5 and Figure 9-6).

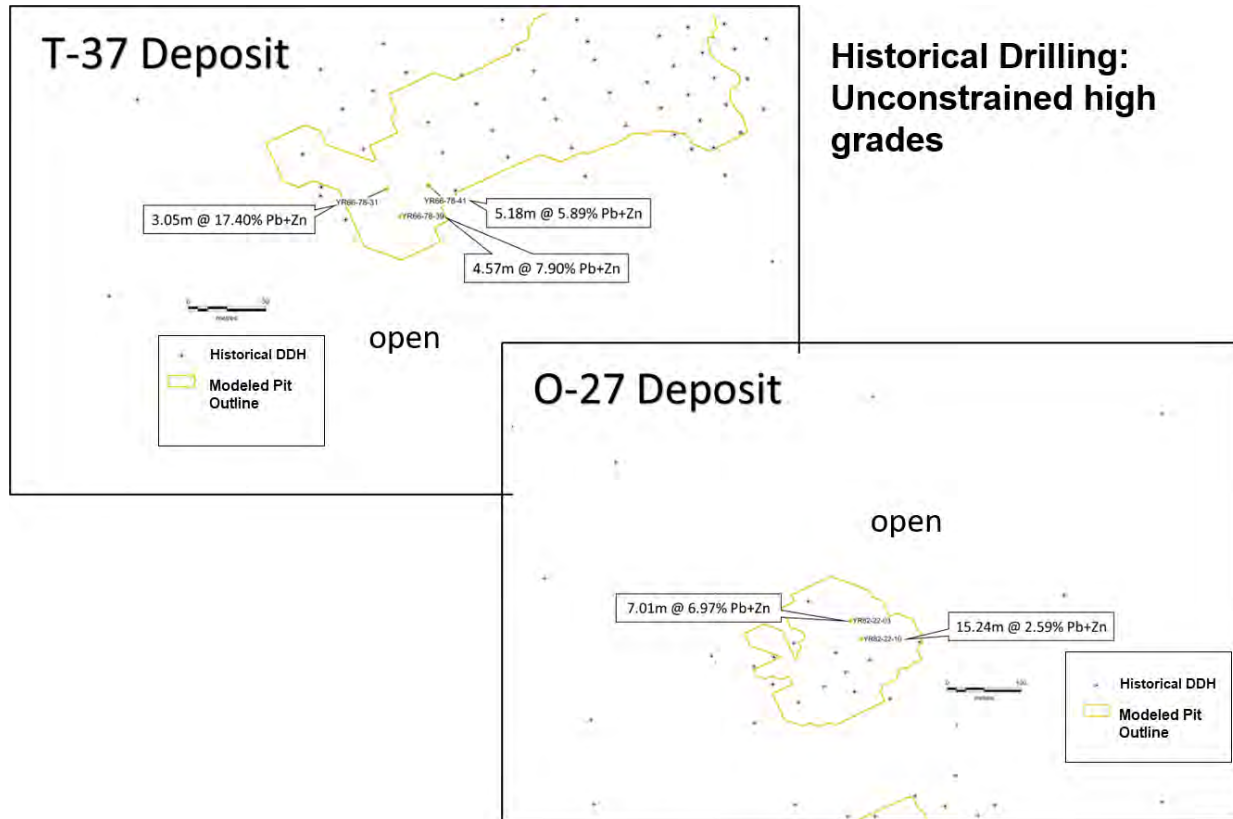


Figure 9-5: Orphan holes within the current MRE

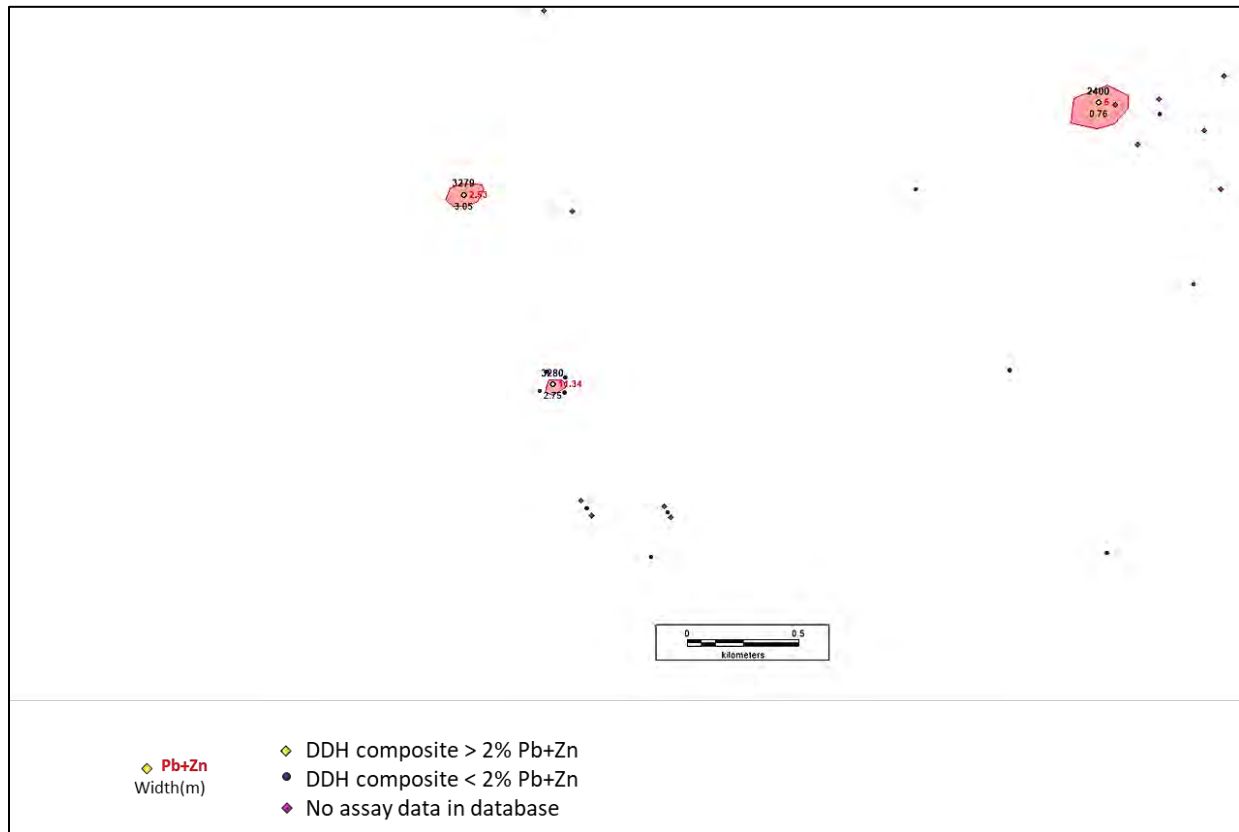


Figure 9-6: Regional orphan drillholes

9.9. Galena Vectoring

The cores of both Tabular and Prismatic deposits are characterized by the precipitation of galena, representing the heart of the mineralizing system and defining zones of maximum fluid passage. This also corresponds to a decrease of iron content. All of these features can be demonstrated in Figure 9-7 in a representative vein structure. With the newly upgraded drillhole database, Pb/Zn assay ratios are used by Pine Point Mining Limited ("PPML") to develop drill targets (Figure 9-8).

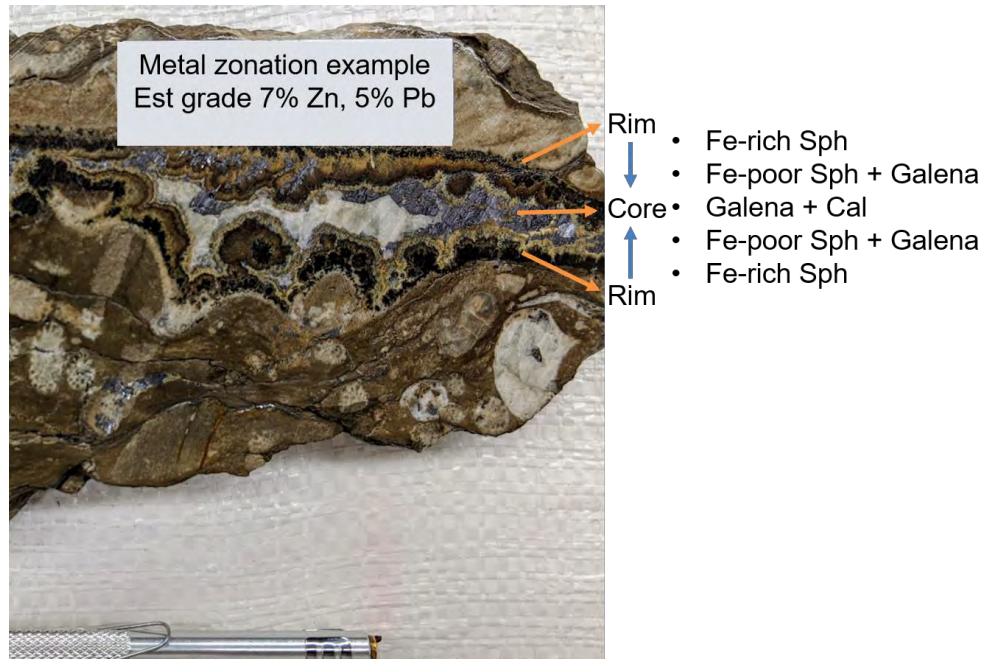


Figure 9-7: Example of metal zonation

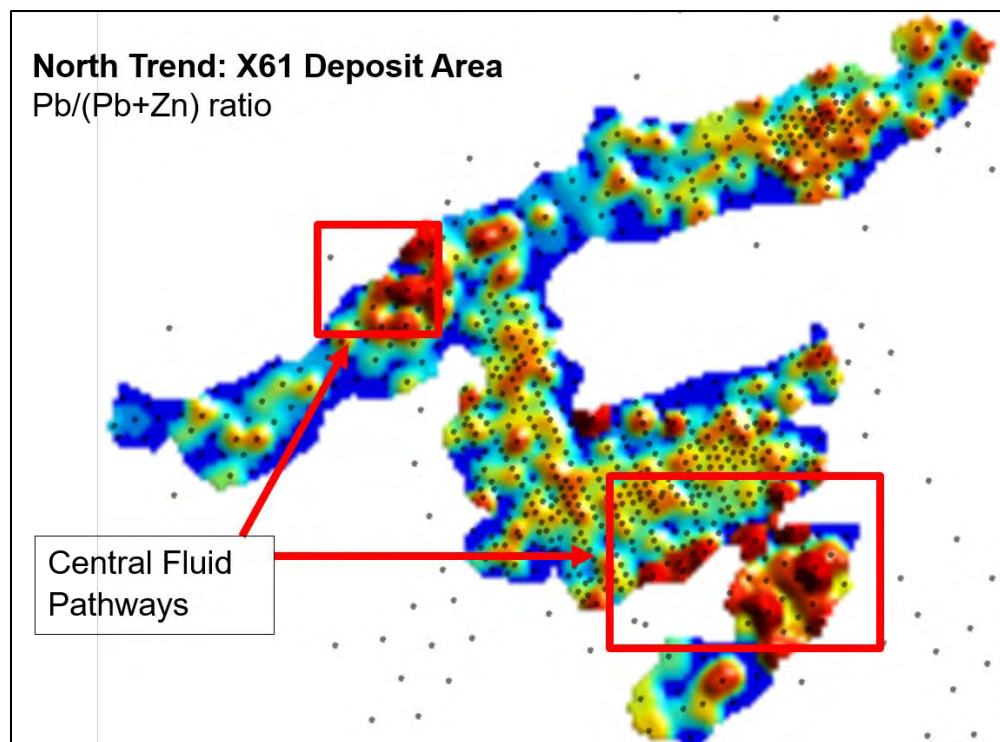


Figure 9-8: Vectoring using lead ratios

9.10. Exploration Approach

PPML uses a hybrid exploration approach that considers the classical MVT model used by Cominco and combines this with newly-developed aspects of a wider “carbonate hosted Pb+Zn” model approach. This allows consideration of the features listed in Figure 9-9 with the application of modern geophysical systems.

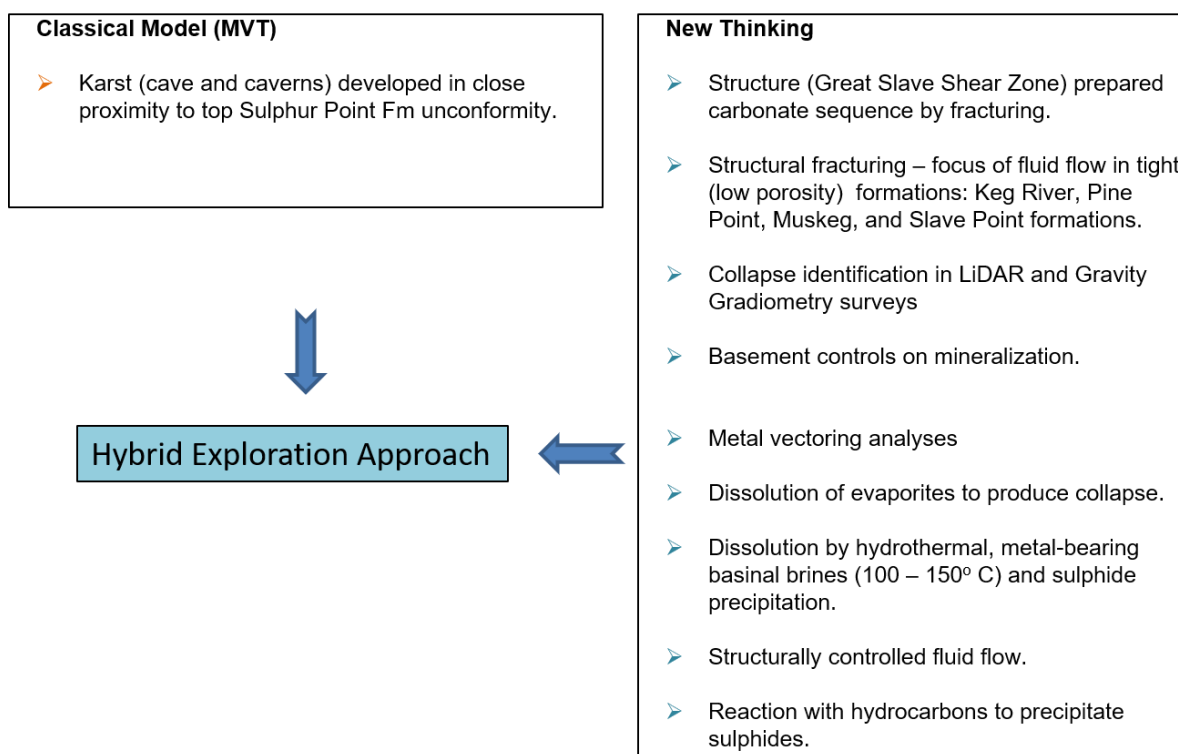


Figure 9-9: Exploration approach used by PPML

10. DRILLING

10.1. Historical Drill Core Relogging

For confirmation purposes, 970 holes were recovered from the Cominco Core graveyard and relogged. Box and downhole tags were re-marked. Where mineralization was present, either as unsampled or previously sampled, new samples were taken and submitted for assay using standards and blanks. These results were incorporated into the database.

10.2. Incorporation of Historical Drillholes

The drillhole dataset acquired by Pine Point Mining Limited ("PPML") in 2018 contained some 18,000 historical drillholes completed by Cominco Ltd. as part of their exploration and deposit delineation program. Missing from the dataset are a number of pre-production, definition drillholes on the deposits previously mined by Cominco Ltd. Also included in the dataset are drillholes completed by Western Mines (later known as Westmin Resources Ltd.) in the western portion of the Pine Point Mining Camp ("PPMC"). The Cominco Ltd. holes date from 1930 to 1988 and the Westmin holes date from 1975 to 1981.

The database consists of:

- Cominco Ltd.
 - Collar information, year of drilling, location in Cominco Ltd. coordinate system and an unknown conversion to UTM NAD 83 zone 11, elevation (Cominco Ltd. and elevation converted to sea-level), and depth of hole;
 - Survey information, azimuth and dip;
 - Assay intervals for many, but not all, drillholes;
 - Lithology and alteration descriptions for approximately 30% of the drillholes;
 - Hard copy drill logs were not preserved by Cominco Ltd., although in 2019 PPML has recovered a small subset of holes in GNWT archives in Yellowknife;
 - Cominco Ltd. destroyed most surface maps and reference to drilling on the Project.
- Westmin
 - Collar information, year of drilling, location in Westmin coordinate system and an unknown conversion to UTM NAD 83 zone 11, elevation (converted to sea-level), and depth of hole;
 - Survey information, azimuth and dip;

- Assay intervals for many, but not all, drillholes;
- Lithology and alteration descriptions of a limited number of drillholes;
- Hard copy drill logs are preserved and PPML intends to enter pertinent data into the database.

Tamerlane drilling completed between 2005 and 2011 (92 holes and 10,897.24 metres) are not included in the current database and are not used in the 2020 Mineral Resource Estimate ("MRE"). Information for these holes is available in a digital database and hard copy drill logs; however, attempts to find the collar locations in the field has, to date, met with limited success.

10.2.1. Historical Drillhole Locations and Validation

10.2.2. LiDAR Surveys

The LiDAR surveys provide high precision elevation data that is used to correct historical drill collars that have not been surveyed. Further, LiDAR adequately images and defines dry historical open-pits and waste piles for use in defining in-situ mineralization and mineral resource models and to understand where historical drillholes were buried by subsequent waste piles.

In 2018 and 2019, LiDAR Services International ("LSI") was contracted to fly a LiDAR survey over the core of the PPMC covering 720 km². The survey utilized ground control points established by Sub-Arctic Geomatics ("SAG"), of Yellowknife, to verify that horizontal and vertical accuracies were within accuracy requirements. In the preparation of the 2019 MRE, the historical and unsurveyed drillholes covered by this survey were corrected to the LiDAR surface elevation.

In 2019, LSI was contracted to complete a LiDAR survey over the remainder of the PPMC. The survey was completed in August.

10.2.3. Historical Drillhole Location

The dataset acquired by Pine Point Mining Limited in 2018 contained drillhole coordinates that were un-tested to determine location accuracy. It was found the readings by hand-held GPS in the field returned northing and easting coordinates to within 5 m to 15 m of the database coordinates on a sparse population of holes investigated in the field. In late summer 2018, SAG was contracted to locate and determine the absolute positions of approximately 4,000 of roughly 18,000 historic drillholes in the Pine Point mining area to verify the database hole locations and correct the drillhole database. Selected holes were in deposit areas to support the MRE. Quality codes were given based on the identification of hole collars in the field described in Table 10-1.



Table 10-1: Quality codes

Quality confidence level	Description	Accuracy estimate
A	Historical drillhole located with readable tag	<1 m
B	Historical drillhole located without readable name	<1 m
C	Historical drill site located within reasonable	~5 m proximity
D	No drillhole evidence, ground disturbance	Variable
E	New drillhole located, readable tag	<1 m

Original Cominco Ltd. surveying of the drillholes was carried out over a number of years at the time of various drill programs across the mine site, however PPML does not have detailed records of these surveys although the Cominco Ltd. coordinates are present in the drill database. It is expected that Cominco Ltd. surveyed the drillholes with standard 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 area without the benefit of established control points and verification with high precision differential GPS (“DGPS”). The process used to establish the transformation was largely unknown and therefore the quality of the resultant UTM coordinate locations was suspect.

In 2018, Sub-Arctic Geomatics established long-term, semi-permanent control 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. The quality code noted above was established and recorded to indicate the level of confidence in both the position and identification of the drillhole.

Summary

- In 29% of the cases, drillholes had posts with readable tags correctly identifying the drillhole;
- For 19% 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 49% of the cases, no drillhole was located at the approximate location although there was evidence of drill activity nearby.

When the drillhole was located it was possible to make a comparison between the observed UTM coordinates and the original Cominco Ltd. coordinates and best-fit UTM provided coordinates. 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.

Confidence Levels and Drillhole Accuracy

- For the 29% of drillholes (1,172) located and identified in the field, accuracies are within 2-4 cm in both horizontal and vertical position;
- Positional accuracies for 16% of drillholes (704) located in the field but not identified are estimated to be better than 1 m;
- Positional accuracies for the 46% of drillholes (1,963) 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 11% of the cases (469 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.

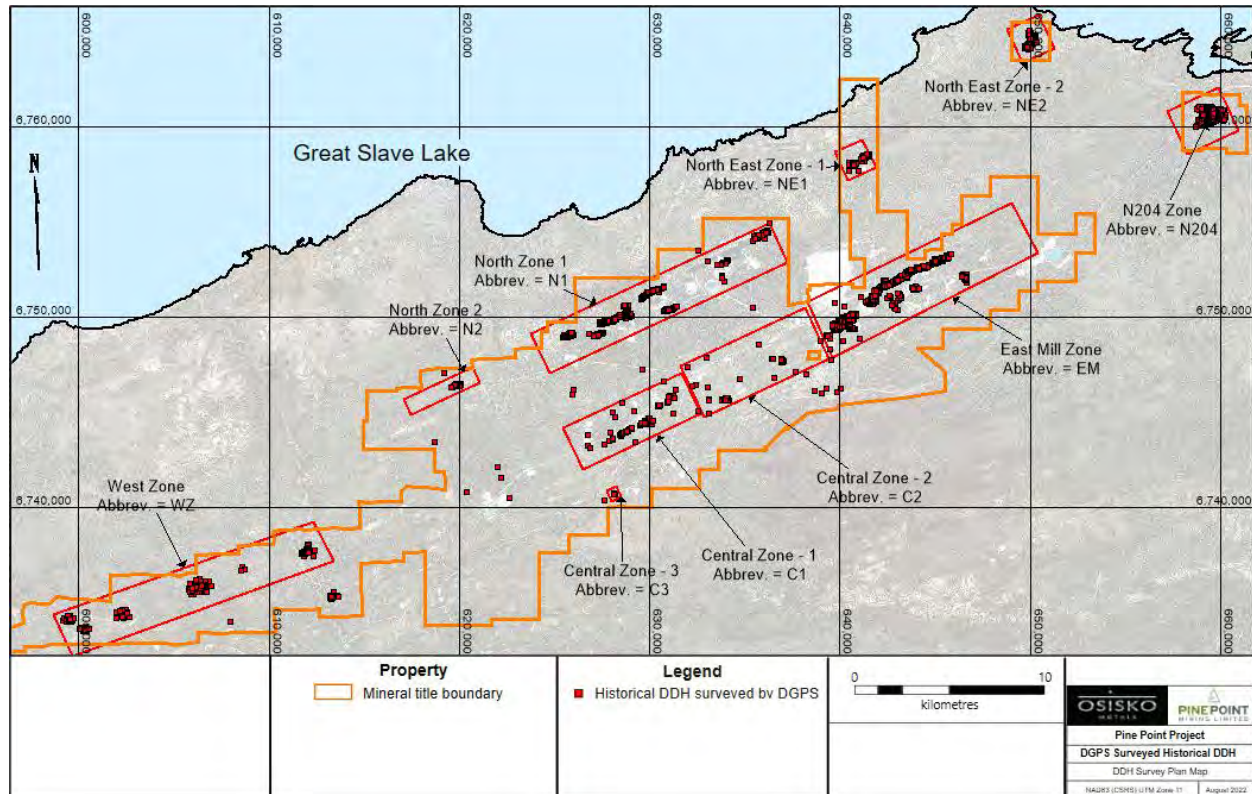


Figure 10-1: DGPS Historical drillhole collar survey

10.2.4. Transformation of Un-surveyed Holes

Background

On February 18, 2019 Challenger Geomatics Ltd. ("CGL") was contacted by Dave Finley, of DBF GeoConsulting, working on behalf of PPML to examine the results of the SAG 2018 Drillhole Survey Program at Pine Point Mine, Northwest Territories. It was noted by SAG during the survey that the UTM_Best_Fit historical coordinates provided by PPML were not always accurate, with some deposits showing up to 15 m horizontal difference between the surveyed collar locations and provided UTM coordinates. It was also noted by Dave Finley that the origin of the UTM_Best_Fit coordinates was unknown, and that the conversion from Mine Grid-feet to Cominco_metre appeared to have used an incorrect conversion factor.

CGL was asked to derive a new, better fit, mathematical transformation from Mine Grid to UTM, incorporating the 2018 surveyed collar locations.

Transformation and Calculated Coordinates

Initially, the Mine Grid_feet coordinates were converted to metres using $1 \text{ ft} = 0.3048 \text{ m}$.

Then the deposits for which drillholes were surveyed in 2018 were analyzed individually to determine “best fit” transformation parameters. This was only done using the survey points coded as Code 1 through 5, to ensure that actual collars were located and surveyed. The best fit was determined to be that which resulted in the majority of the transformed coordinates being within 1 to 2 m of the surveyed collars. It should be noted that this was not always possible to achieve.

After the individual deposits were analyzed however, it was determined that there were some consistencies between the calculated transformations, particularly as it related to the original survey date of the drillholes. It was then determined that one global transformation could be applied to the 1984 – 1987 drillholes of (nearly) all the deposits and yield coordinates that were extremely close to the actual surveyed locations. The results for the drillholes from other years varied depending on the deposit, but overall the global transformation provided coordinates that were within 1-3 m of the surveyed collar locations.

The global transformation parameters are:

- Combined Scale Factor: $\text{UTM Distance} = \text{Local Distance} * 0.9998045$
- Rotation: $\text{UTM bearing} = \text{Local Bearing MINUS } 27^{\circ}17'51.5''$
- Shift: UTM coordinate of 6750718.750N, 639644.147E is equivalent to a Local coordinate of (0N, 0E)

Coordinate Reference System

To ensure that the new UTM coordinates are consistent with ongoing drillhole survey work, the coordinate reference system/datum used is the NAD83 (CSRS) horizontal reference framework and the CGVD2013 vertical reference framework. The Map Projection is the UTM Zone 11 Epoch 2010. This system was adopted by SAG in 2018, and control points were established at that time.

Corrected Coordinates

On March 18, 2019 Challenger Geomatics Ltd. issued a spreadsheet titled “PINE POINT CALCULATED UTM”, which incorporated the original drillhole data from PPML as well as additional columns showing the 2018 SAG surveyed locations and the calculated (via transformation) UTM coordinates.

Within the spreadsheet, “Corrected Coordinates” are either the surveyed location (for drillholes surveyed and identified as Code 1 through 5) or calculated (based on the transformation parameters). Columns were also added showing the deltas requested by Dave Finley, and the confidence level in the calculated coordinates.

Confidence Level

As part of the analysis, CGL assigned each calculated UTM coordinate a rough confidence level, expressed as a distance in metres. It is anticipated that the actual collar will be located within this many metre of the calculated coordinates. This value is rough only and is based on the drillhole location, the drillhole year, and the relative fit of the Code 1 to 5 data (if any) for that location and year based on the transformation.

It was noted during the analysis and derivation of transformation parameters that deposit N204 appears to have been originally surveyed utilizing a slightly different origin and/or coordinate system. The original Mine Grid coordinates, when transformed using the global transformation parameters, yielded locations that deviated from the 2018 surveyed locations by approximately 10 m. Therefore, a horizontal confidence level of 10 m was assigned to the N204 drillholes.

A horizontal confidence level of 10 m was also assigned to any drillholes that fall far outside the range of the 2018 surveyed data.

May 3, 2019 Revision

On May 3, 2019 Challenger Geomatics Ltd. issued a revised “PINE POINT CALCULATED UTM” spreadsheet. This revision did not alter any of the transformation parameters, but it did correct the survey codes for 38 surveyed points which were incorrectly numbered during the 2018 survey. As a result, some of these drillholes saw their corrected coordinates change from surveyed to calculated, or vice versa.

West Zone Deposits

For drillholes in the West Zone deposits the current MRE relies on the original derived coordinates. Subsequently, however, drill holes in the West Zone have been identified in the field and surveyed by DGPS (Figure 10-1).

Results

Although a comprehensive analysis of the 2019 Drillhole Survey Program has not yet been completed, it appears that the application of the transformation resulted in surveyed collar locations being generally within 1 m to 4 m of the supplied coordinates. And this appears to be the case even when the calculated coordinates differ from the original UTM_Best_Fit coordinates by more than 20 m.

10.2.5. 2019 Historical Drill Collar Survey

In the summer of 2019, Sub-Arctic Geomatics was contracted to continue DGPS surveying of historical holes in deposit areas not previously surveyed. Table 10-2 summarizes the survey:

Table 10-2: DGPS surveying of historical holes

Mineral deposit area	# Drillholes to locate	Total # drillholes located	% Confidence Level by Category									
			A (Codes 1 & 3)		B (Codes 2,4,5)		C (Codes 6,7,8,9)		D (Code 10)		E (Code 11)	
			#	%	#	%	#	%	#	%	#	%
X17	166	166	28	17	79	48	29	17	30	18	0	0
T37	110	109	5	5	44	40	40	37	20	18	0	0
X15	97	97	9	9	31	32	20	21	37	38	0	0
L27	128	128	39	30	54	42	11	9	24	19	0	0
X49	713	583	36	6	44	8	367	63	136	23	0	0
G03	173	164	26	16	89	54	49	30	0	0	0	0
West	119	115	17	15	22	19	74	64	2	2	0	0
Central	215	214	88	41	58	27	62	30	6	3	0	0
Tamerlane	92	92	30	33	15	16	43	47	4	4	0	0
Total count	1,813	1,668	278		436		695		259		0	
Percentage		100%		17%		26%		42%		16%		0%

The 2019 drillhole survey data were not included in the 2020 MRE due to separate transformations needed for each deposit area that were not completed prior to the closure of the database. However, the results confirmed reasonable accuracy of the drillhole database in these areas. Full integration of the 2019 DGPS into the database is expected by the end of summer 2020.

10.3. Pit Bathymetry

Pit Bathymetry surveys provide pit geometries for the purpose of in defining in-situ mineralization and mineral resource models in the context of what has been removed and what remains in context of drillhole information.

In 2018, Sub-Arctic Geomatics was contracted to conduct bathymetry surveys on 11 water-filled historical open-pits on former Cominco Ltd. mine sites. These data were merged with the 2018 LiDAR survey and the combined data set was used in the 2019 MRE.

Table 10-3: Bathymetry survey parameters

Pit Name	Volume (m ³)	H2O surface height above MSL (m)	Approximate depth (m)	Approximate dimensions (m) EW-NS
K53	492,898	203.80	35	190 X 170
M52	208,818	200.37	20	140 X 160
K57	1,305,509	198.81	50	280 X 250
K62	867,144	200.87	50	240 X 210
M64	557,759	208.66	45	175x190
165	236,286	203.27	30	130 X 155
N42	197,210	190.59	15	490 X 125
P24	93,727	186.72	15	115 X 90
W17	2,479,730	189.40	> 75	310 X 290
X15	9,182,453	184.64	40	900 X 600
P31	31,787	190.51	10	90 X 80

In 2019, Sub-Arctic Geomatics was contracted to conduct bathymetry surveys in the remaining water-filled pits on the Project. These data are not included in the 2019 MRE.

10.4. Drilling Methodology

10.4.1. Drillhole Selection

The process of selecting drillhole locations at Pine Point is currently a combined effort between the Osisko Metals/PPML geological team and BBA. Drillholes target previously drilled zones across the Project, mainly using the extensive Cominco Ltd. database and concentrating on the many historical deposits defined by Cominco Ltd.

10.4.2. Drillhole Location / Set-up

The coordinate system in use is Nad83/UTM Zone 11. Magnetic declination in the Pine Point region is: 15°35.22' East.

On the Project, drill collar locations are pre-surveyed by the PPML surveyor using a Hemisphere S320-type surveying instrument. This is the most accurate method for locating drill collars 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 identification, the intended azimuth, and the anticipated depth of the hole. In the case of inclined drillholes, a separate set of clearly marked wooden pickets mark the foresight and back sight for the alignment of the drill rig. These pickets are placed at a sufficient distance away from the collar location so as not to be disturbed by the Drilling Contractor during equipment installation and drill set-up. Foresight and back sight pickets are accurately surveyed and installed by the PPML 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, 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 footprint of the area of operation are as small as possible to limit disturbance of the environment.

10.4.3. 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 Drilling Contractor shall not initiate activity prior to having received the approval to commence from PPML staff on-site.

10.4.4. Drillhole Orientation during Operation

- Once drilling commences, a PPML geologist or surveyor visits the drill site daily to maintain regular checks of the drilling progress, to inspect the drill site for any environmental or safety issues, and to monitor the downhole survey readings. The hole orientation is checked and monitored using a downhole surveying device as follows: First reading is taken 6 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 downhole survey instrument manufacturer. The readings include: i) operator name, ii) date and time, iii) depth of reading, iv) inclination, 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 PPML geologist or surveyor collects the downhole survey data sheets from the drillers. The downhole survey data is added to the geological logging sheet in due course by the logging geologists. The original paper downhole survey data sheets are scanned and stored digitally, and the paper copies are filed in the on-site PPML office.

10.4.5. Drillhole Numbering

Drillhole numbering is assigned by a Pine Point geologist, generally in a sequential manner, and is confirmed with the driller or drill supervisor prior to drilling. The drillhole number contains information on the Project area (based on historical deposits defined by Cominco Ltd.), year drilled, Company name, and hole number. The numbering convention is as follows:

Area-Year(YY)-Company-Hole(###)

For example, drillhole "L37-19-PP-081" is the 81st hole drilled in the L37 Area by Pine Point Mining Limited in 2019. In cases where the same area is drilled over multiple years, the first hole drilled in each calendar year is assigned hole number 001. Drillhole numbers are consecutive for each individual area.

10.4.6. Drillhole Coring

Drill cores are provided by the Drilling Contractor in either NQ (46 mm diameter) or HQ (60 mm diameter) size. 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 PPML. The driller marks the depth in metres (m) after each run, usually every 3 m, sometimes at shorter intervals when appropriate.

The drillhole is terminated by the PPML 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, the drill may be 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 grainstones beneath the reefal carbonate lithologies (host to the Pine Point mineralization)).

Once the drillhole is terminated and the final downhole survey reading is collected, the drill crew pulls 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 bedrock for the plug, and then cemented into place; this method mitigates the possibility that any groundwater will escape to the surface. The casing may be left in the hole and cut to surface level if necessary. A special marker consisting of a 2 m-long piece of metal pipe is inserted into the hole, and a high-visibility painted 2"x2" wooden stake with drillhole identification inscribed on a metal tag is hammered into the ground in very close proximity (typically less than 50 cm).

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 the equipment is removed and the site is cleaned, the PPML surveyor collects a final GPS reading of the drill collar location which is added to the drillhole database. Photographs of the site are also taken by the surveyor 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 spring or summer months to finalize clean-up and review any remediation requirements. A completed drillhole inspection sheet is filled out and signed by the surveyor, and photographs of the fully remediated site are collected for the database.

10.4.7. 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 cohesive core in three 1.5 m rows while HQ core trays hold a nominal 2.4 m of cohesive core in two 1.2 m rows. The 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. Drillhole numbering and box numbering are also 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, rock conditions or mechanical failures will often 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, cavities in the bedrock, or changing water conditions are encountered that result in core loss. Once the core tray is filled, it is secured shut using a plywood lid and wood screws. It is then carefully stacked for transport to the PPMC upon completion of the hole.

10.4.8. Receiving Core at the Pine Point Mining Camp

Securely-boxed drill core is transported daily to the core logging facility on-site at the PPMC. Care is exercised to ensure that the lids are securely attached to minimize core disturbance, breakage, and loss during transport from the drill site.

All core trays are verified in the 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.

10.5. 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. In the long term (if success is inherent), it is used to lead into mineral resource and reserve estimations, as well as 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 regard to lithologies, mineralization styles, and controls. This data has now been brought into a modern classification system and is being used as another tool for identifying drilling targets.

10.5.1. Core Fitting

The geotechnicians are responsible for realigning the core to achieve the best fit along all natural and mechanical fractures. This realignment 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, misalignment during the mechanical breaking of the 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 the core is noted and immediately reported to the driller or drill supervisor as appropriate.

10.5.2. Geotechnical Logging Procedure

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

Geotechnicians fill out pre-made spreadsheets for each drillhole that collect all relevant data. The columns are designed to calculate core recovery as well as rock quality designation ("RQD") once intervals are checked. The technicians then mark the contents contained within each box number and file the spreadsheet digitally in the relevant drillhole folder. The following sections describe procedures to ensure the correct and representative measurements are collected.

10.5.2.1. Core Recovery

Core recovery is collected from all drillholes. The core recovery is calculated by the measurement of the core in the core tray divided by the length claimed to be drilled on the meterage blocks. This number multiplied by 100 is recorded as a percent recovery. For example, 183 cm of core recovered in a 300 cm (3 m) run is reported as a core recovery of 61%.

$$\text{Recovery \%} = \left(\frac{\text{Actual length of core in core tray}}{\text{Total length of core run}} \right) \times 100$$

Core recovery is recorded for each drill run. Specific areas of loss are noted if possible and marked by the 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 loss of drill core during the coring process (e.g. grinding, etc.).

10.5.2.2. Rock Quality Designation (RQD)

The RQD 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 by incorporating only those pieces of solid core longer than twice the diameter of the core. For NQ core, the nominal diameter is 5 cm, so the length index is 10 cm. Shorter lengths of the 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:

$$\text{RQD \%} = \left(\frac{\text{Sum of all core pieces longer than } 2x \text{ core diameter}}{\text{Total length of core run}} \right) \times 100$$

It is important to distinguish between mechanical breaks and natural breaks identified in the core. Core that has been mechanically broken (e.g. by the driller's helper breaking the core to fit it into the core tray) may be used in RQD calculations.

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.

10.5.3. Geological Logging Procedure

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

Logging data is recorded in a core logging software named Geotic Log. In this program, tabs are completed by the logging geologist. The recording intervals of lithology, alteration, assays, and header tabs are completed and saved. The program uses Microsoft Access to compile data in spreadsheet format for later export to other geological software.

10.5.3.1. Projects Section - Survey Information Subsection

The survey information section collects relevant data as it pertains to the drillhole, including drillhole number, orientation, total length, coordinates and elevation, drilling start and end dates, logging geologist, core size, and storage location. The survey information section also contains the downhole survey data collected by the drillers.

10.5.3.2. Descriptions Section

The Descriptions section is the main logging part where geological information is entered and collected. All geological characteristics including lithologies, structures, alteration, sulfide mineralization, assay sample numbers and intervals, density sample numbers and intervals, etc. The main geological observations are described within these subsections.

A few specific notes are required regarding data input:

- Lithological description includes formation identification, lithology, textures, fossil content, and observational comments for each associated meterage;
- Alteration description includes alteration type and observational comments for each associated meterage. Intervals often cross lithological contacts;
- Assay sample intervals are broken out and numbered sequentially. Descriptions within this section include mineralization type and a visual estimate at time of logging. After assay results have been received, the results are imported into the software by a PPML geologist and displayed alongside the visual estimates. The meterage sequence in the "From" to "To" columns need to be sequentially complete in order to facilitate later manipulation outside of the spreadsheet;
- Density (specific gravity) sample intervals and numbers are supplemented with a rock type description and a visual estimate of sulphide mineralization. After specific gravity results

have been received, the results are imported into the software by a PPML geologist and displayed alongside the sample numbers.

10.6. Assay Sample Selection

Assay samples are broken at major rock code contacts to represent homogeneous units. The minimum assay sample interval in the hole will be not less than 50 cm except in unique circumstances (e.g. lithological units which are mineralized and less than 50 cm in length). The maximum sample interval will not exceed 150 cm. 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 QC samples (blank samples, duplicate samples, and Standard Reference Materials or "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 Assay tab of the Descriptions section in the Geotic Log software. Sample intervals and QC sample information (sample type and SRM identification) are also manually recorded in the sample tag books provided by ALS Geochemistry. All sample tag books are scanned and filed in the assay database upon completion.

Density samples are selected by the logging geologist as representative samples of the main lithologies within the drillhole. Density samples range in length from 10 cm to 20 cm. Within mineralized intervals, samples are collected every 10 m. Outside of mineralized intervals, samples are collected every 20 m or more frequently when required. Samples are selected from cohesive core.

10.6.1. Core Sampling (Core Saw Splitting)

A geotechnician trained in core cutting procedures executes the core cutting at the Pine Point field 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 and sample interval onto 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 is returned to the core box. 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 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 $\frac{1}{2}$ core sample in half again lengthwise to produce two $\frac{1}{4}$ duplicate core samples. Care is taken to ensure that both $\frac{1}{4}$ core samples are virtually identical and thus representative (even though in a lesser amount) to the original $\frac{1}{2}$ core sample. The two individual $\frac{1}{4}$ core sample intervals are placed in separate sample bags for analysis, with unique but sequential sample numbers.

A “standard” sample consisting of material of known metal content and internationally recognized and verified (SRM) is included in the sample sequence by the trained core sampler. PPML includes a standard after every 20 samples. Standard QC samples are selected and placed in sequence by a PPML geologist. Multiple SRMs are used as standards for the QA/QC program in order to test the laboratory's reporting for select key elements at various concentrations.

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 PPML on-site 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.

10.6.2. Sample Shipment Preparation

Assay sample bags are packed in large “rice” bags with a ‘full’ weight not exceeding 22 kg. 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. In the case of a security breach of the rice bags (e.g., a cut security tag or torn rice bags) prior to arrival at the laboratory, the breach would be reported by ALS Geochemistry to PPML. The compromised samples would be discounted, and replacement samples would be shipped to ALS Geochemistry by PPML.

Density samples are collected as $\frac{1}{4}$ core samples from intervals which are also serving as assay samples. The purpose of collecting only $\frac{1}{4}$ core samples is to retain a $\frac{1}{4}$ core “witness” in the core tray: $\frac{1}{2}$ core is collected for assay, $\frac{1}{4}$ core is collected for density, and $\frac{1}{4}$ core is left as a witness. Density samples in unmineralized/unsampled intervals are collected as full core samples not exceeding 20 cm in length. All density samples are stored in individual sample bags with unique

sample tags, sealed with a zip tie. The sample bags are placed in a plastic 5-gallon pail which is sealed once full. Sealed pails cannot be opened without removing a plastic strip from the pail lid, which indicates to the laboratory that the pail may have been compromised. As an additional security measure, a numbered security tag is also zipped tightly through the lid and the pail handle.

For shipping purposes, the address of the analytical laboratory is written on the rice bags and pails, along with an identification of PPML as the sender. A unique number is assigned to each rice bag and pail and is never repeated year-to-year.

The rice bags are piled onto a wooden pallet for transport and wrapped in pallet wrap to prevent the bags from moving during transport. When pails containing density, samples are also included in the shipment, the pails are placed on the centre of the pallet first, with rice bags piled around them to provide security and stability. A waterproof bag containing the Laboratory Sample Submission Form as well as a hard copy of the sample dispatch sheet is included with the sample shipment.

The palletized rice bags and pails are stored in the warehouse at the PPMC until shipped to the laboratory in Yellowknife. Currently, Manitoulin Transport collects the pallets of samples from the PPMC every Wednesday afternoon.

The lab is notified by email that the samples are en-route and is instructed to notify the PPML 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 the PPMC. Any additional instructions for processing (such as expedited service) are communicated to the laboratory manager at this time.

10.7. Core Photography

All drill core is photographed. Once the core has been geotechnically logged and the geologist has marked all contacts/samples, the 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 later be used 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, geotechnical logging, geological logging, and sample interval selection;
- All core trays from the drillhole are photographed, regardless of lithology or mineralization;

- Core trays are placed sequentially on a flat, level table. A high-quality digital camera is mounted on a gimble at a fixed height above the table, and the orientation of the camera may be adjusted via the gimble such that the core trays can be kept in focus. This method allows for all core trays to be photographed at the same scale and in a consistent manner;
- 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 with the inscribed metal sample tags affixed to the core tray, but without the paper tags also affixed. This allows for clear identification of sample intervals in the photographs, if needed, and prevents the core from being obscured by the paper tags. Paper tags are affixed after photography is complete, and are later collected by the core cutter during sample preparation;
- 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.

After the core is photographed, the core is stored on a rack beside the cutting room for the core cutter to proceed with sampling.

10.8. Core Storage

Following sampling, the ends of the core trays are labelled using an inscribed metal tag which is durable and will survive weathering far longer than permanent marker. 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 PPML's permanent storage facility on-site beside the PPMC. The core boxes are stored on the pallets in a crisscross manner to prevent tipping, with the metal tags clearly visible.

10.9. 2017 Drilling Program

In 2017, 25 holes totalling 2,276 m were drilled. 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. 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.

10.10. Winter 2018 and 2019 Drilling Program

In 2018-2019, 1,032 drillholes totalling 71,014 m were drilled. Some of the holes targeted areas of mineralization intersected by Cominco Ltd. in the past and others were directed at expanding the mineral resources of existing occurrences. Holes were drilled in the North Zone, Central Zone and East Mill Zone. This was largely in-fill drilling to increase the confidence of mineral resource estimates and delineate edges of mineralization. Of the total number of drillholes listed above, 23,751 m in 318 drillholes were included in the 2018 MRE and 714 additional holes totalling 47,263 m were added to support the 2019 MRE.

10.11. Post 2020 MRE Drilling Program

Since the 2020 PEA, PPML drilling activities focused mainly on the conversion of Inferred Mineral Resources reported in the 2020 PEA to the indicated category. Drilling has been focused on infill drilling and limited exploration. The objective of the infill drilling is to reduce the intercept spacing to approximately 30 metres within Inferred Mineral Resource areas reported in the 2020 PEA. To minimize the impact on the environment, drilling is mostly conducted in winter. The program is approximately 45% complete and the Company expects to complete infill drilling by the end of winter 2023.

Results for this partial drilling program were, in some cases, not available and deemed not material enough to justify remodeling of the entire mining camp before the entire program is completed. It is the QP's opinion that adding these holes to the model would not have had a material impact on the current MRE.

Drilling, surveying, sampling, assaying, and storage protocols for the post-2020 PEA drillholes are the same as the protocols described in this Technical Report for the 2017-2019 drillholes.

In addition to core drilling PPML is currently re-logging and re-assaying a significant amount of historical drill core catalogued and recovered from historical core storage areas (Table 10-4).



Table 10-4: Drillholes from PPML

	2017	2018	2019	2020	2021	2022 (so far)	Total
DDH count	132	830	239	111	197	252	1,761
RC count					40		40
DDH metres	11,759.06	55,263.15	12,154.00	6,613.90	14,298.50	14,105.00	114,193.61
RC metres					3,829.50		3,829.50

Table 10-5: Historical relogging program

Pit Name	Cominco Historical 2019- 2021	Tamerlane Historical 2021	Cominco Historical 2022	Tamerlane Historical 2022	Total Relogged
Number of relogged drillholes	500	68	334	68	970
Number of relogged metres	34,882.02	7,656.50	18,121.98	0.00	60,660.50
Number of relogged assays	3,544	1,626	383	0	5,553

Figure 10-2 to Figure 10-8 show the location of both recent and historical drillholes throughout the Project used for the current MRE.

Figure 10-9 to Figure 10-14 show typical cross-section views throughout the Project.

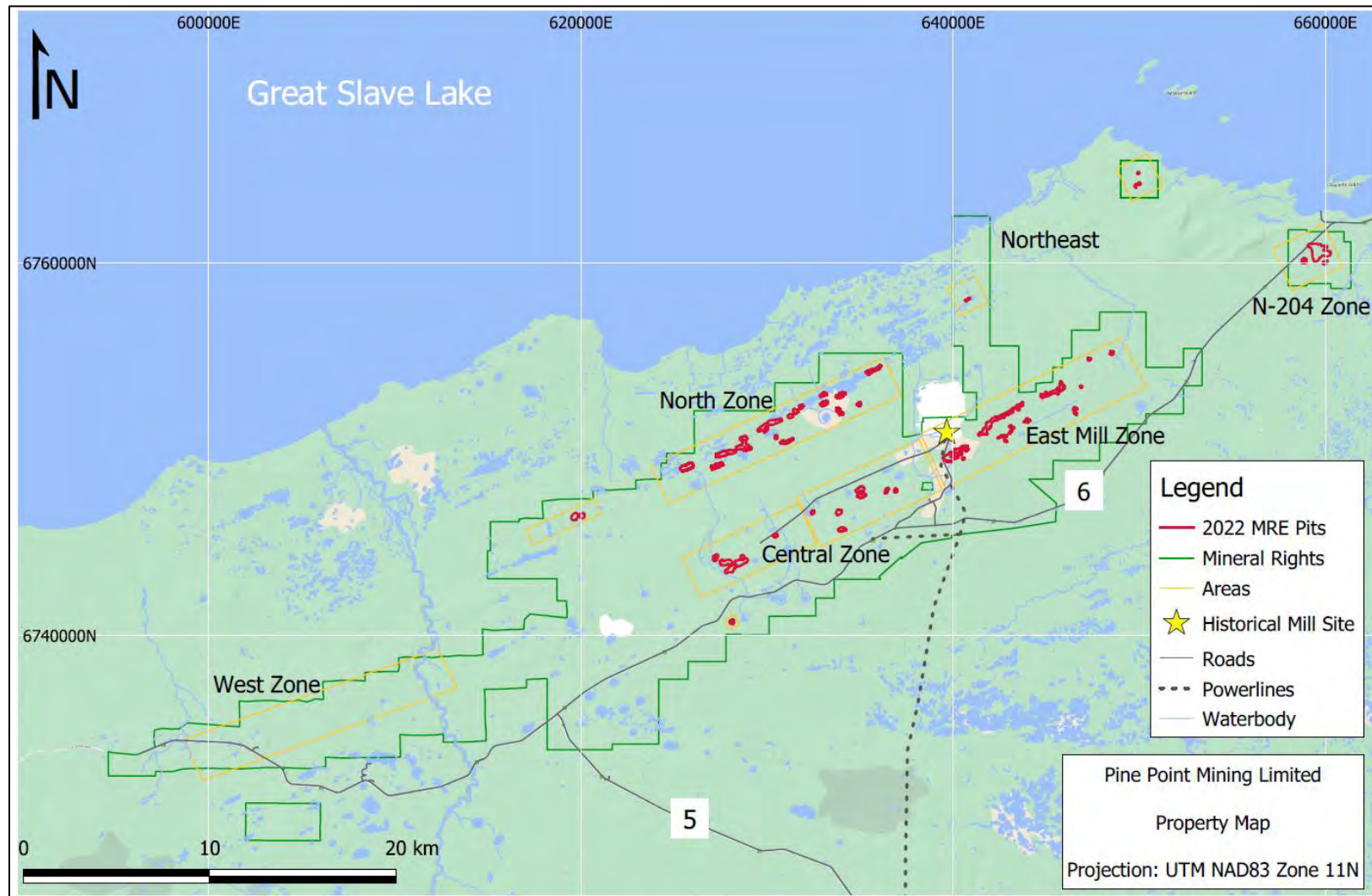


Figure 10-2: General view of the Pine Point Project showing the location of the different zones

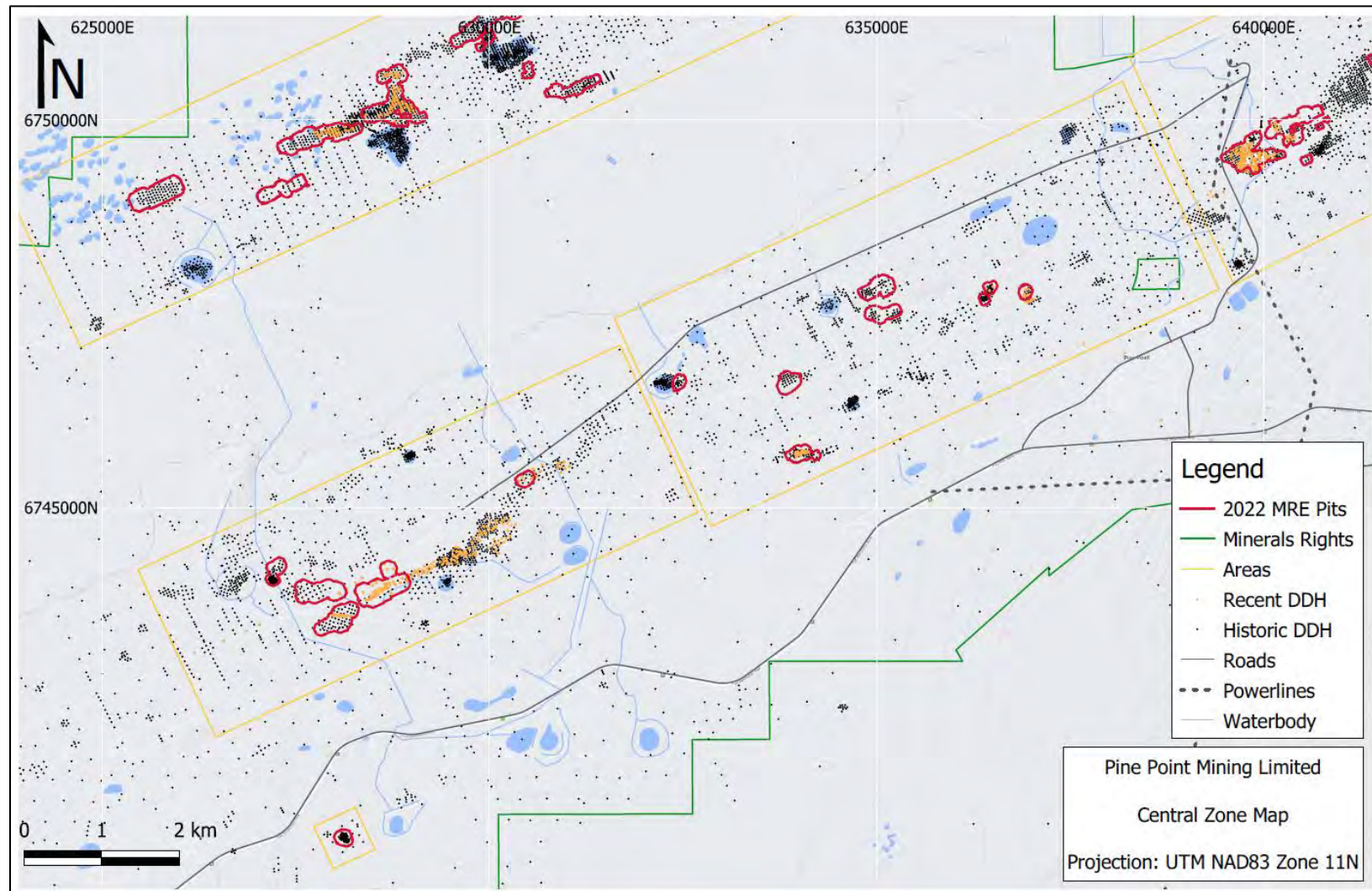


Figure 10-3: Central Zone map showing the location of historical collars (black) and recent holes from 2017-2019 (orange)

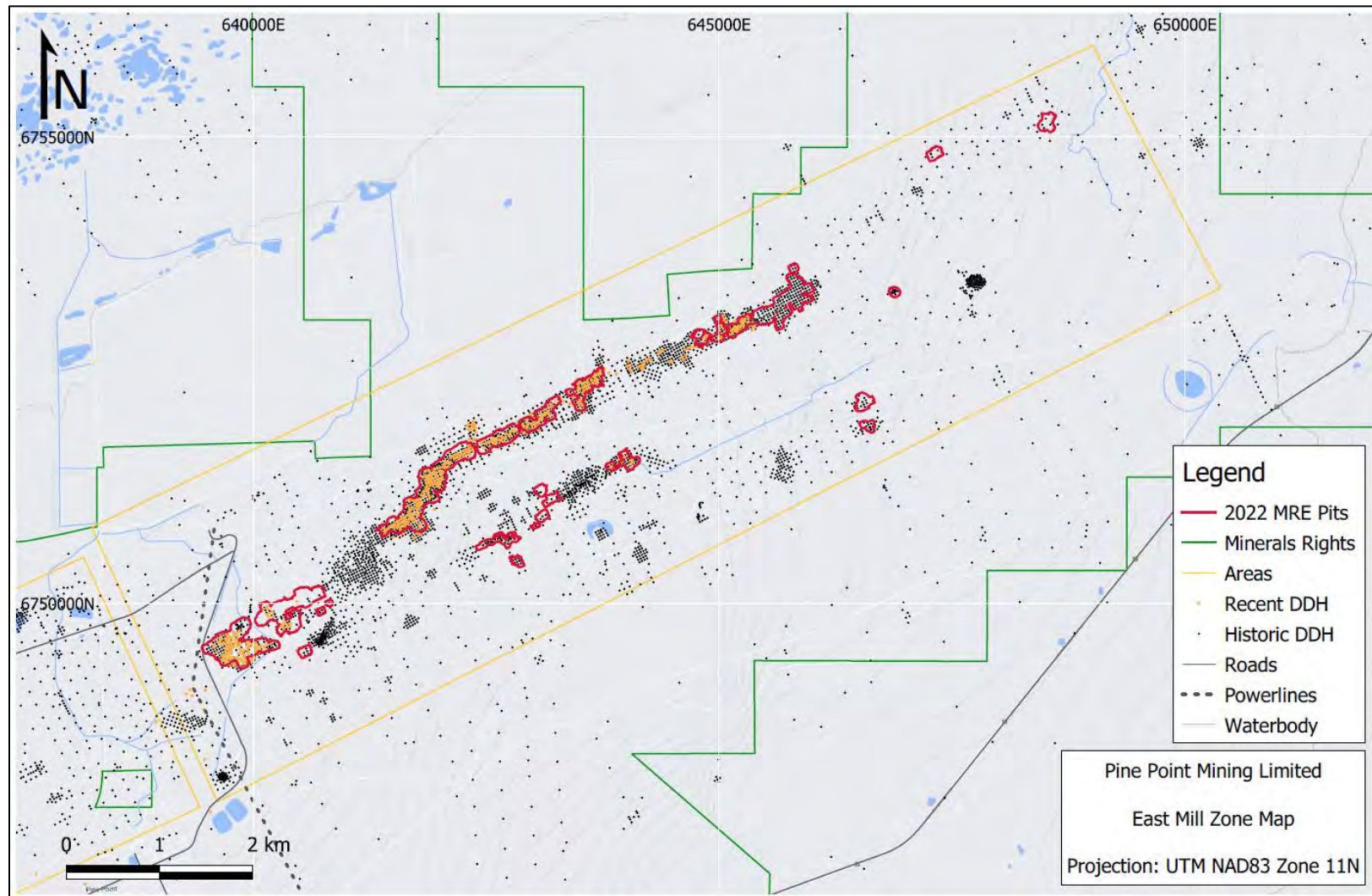


Figure 10-4: East Mill Zone map showing the location of historical collars (black) and recent holes from 2017-2019 (orange)

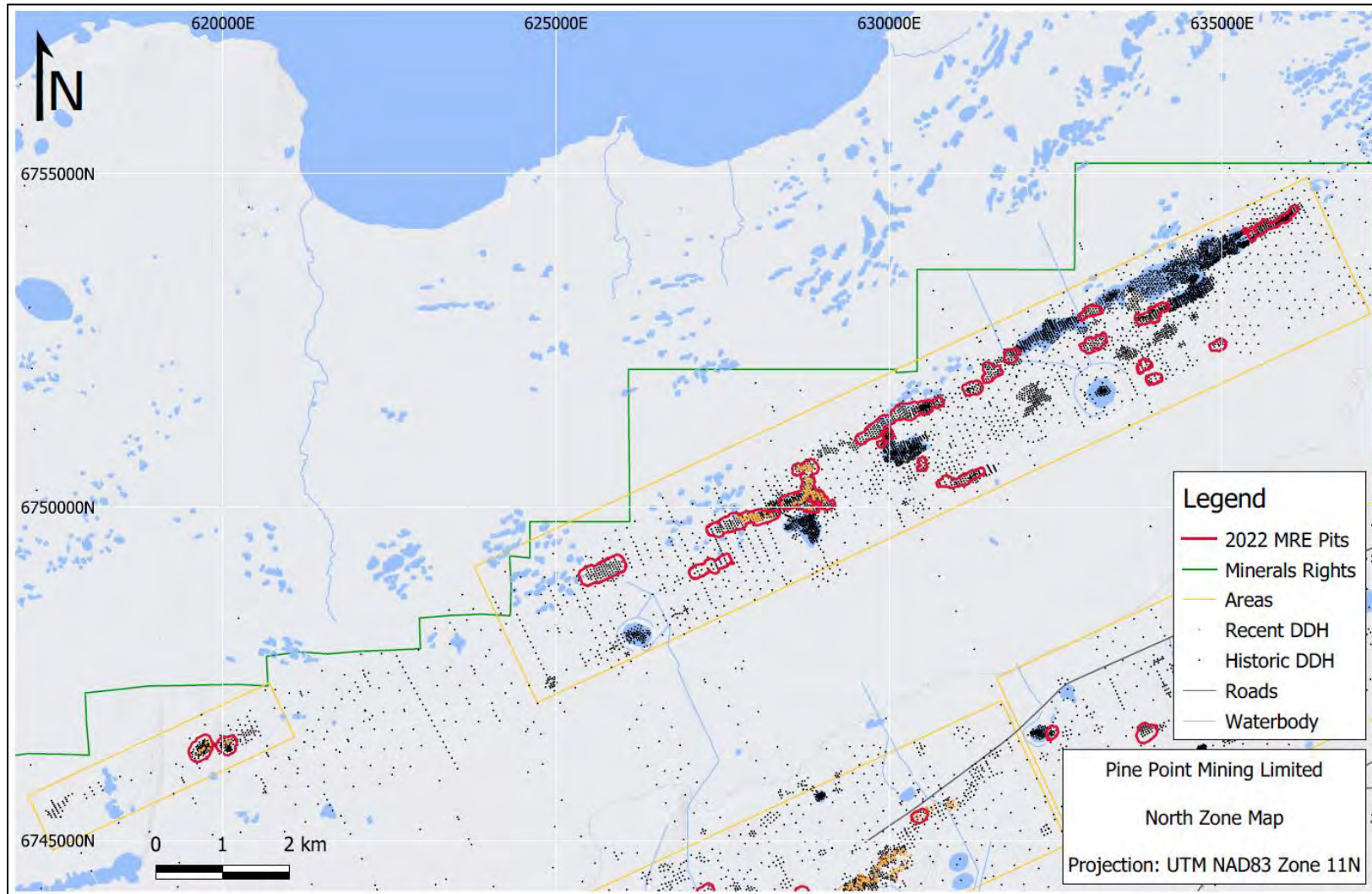


Figure 10-5: North Zone map showing the location of historical collars (black) and recent holes from 2017-2019 (orange)

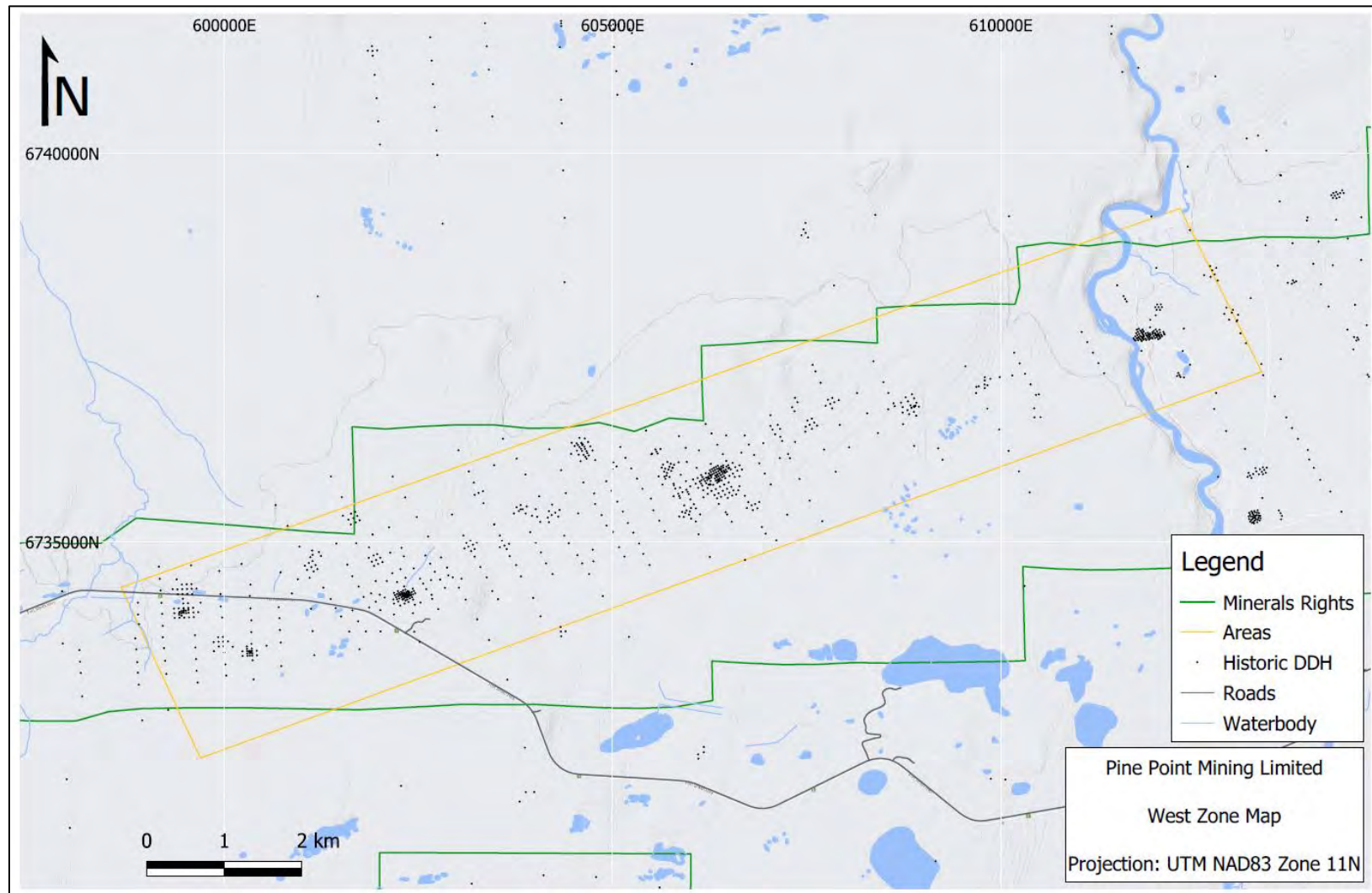


Figure 10-6: West Zone map showing the location of historical collars (black) and recent holes from 2017-2019 (none in this case)

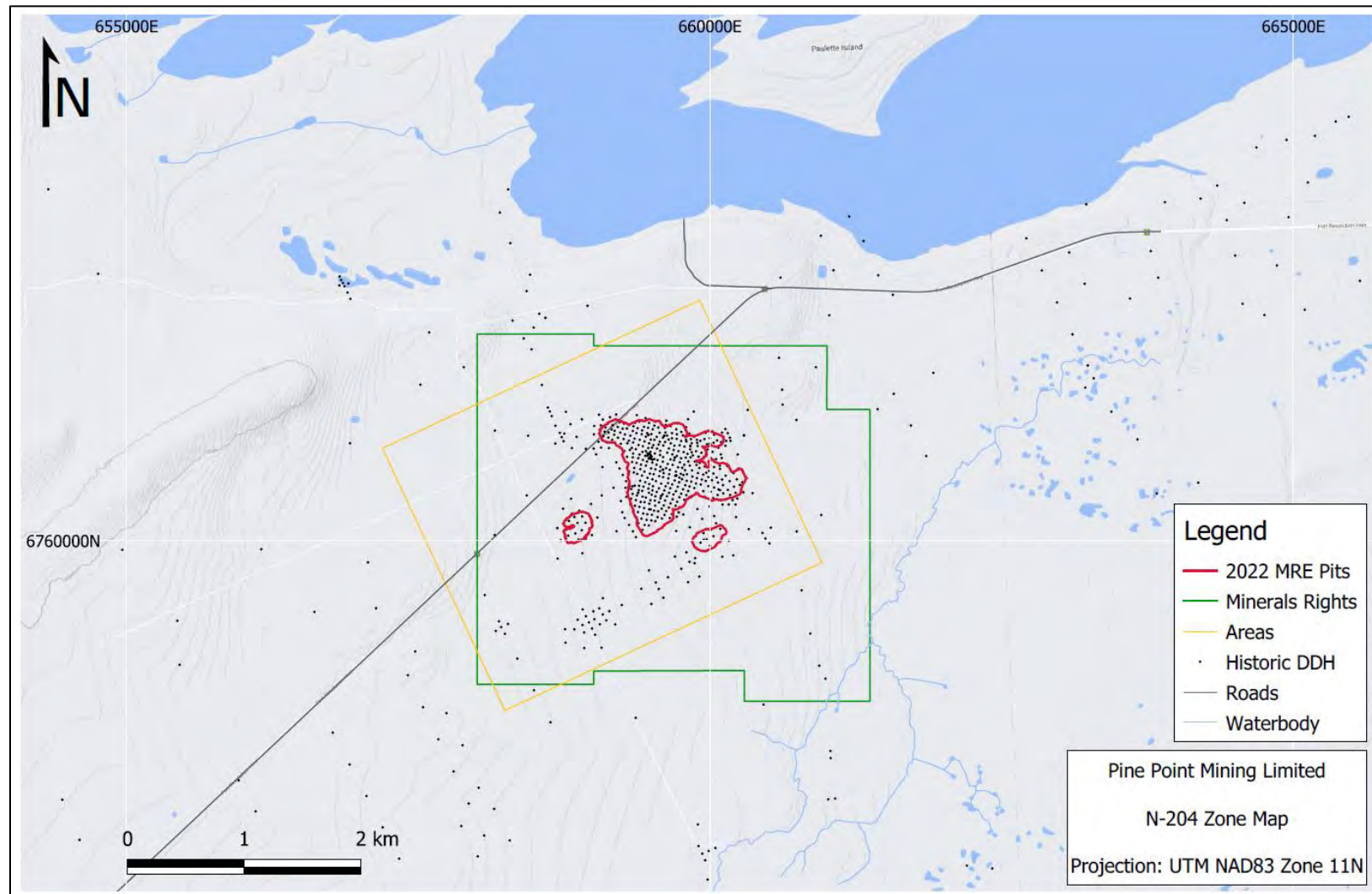
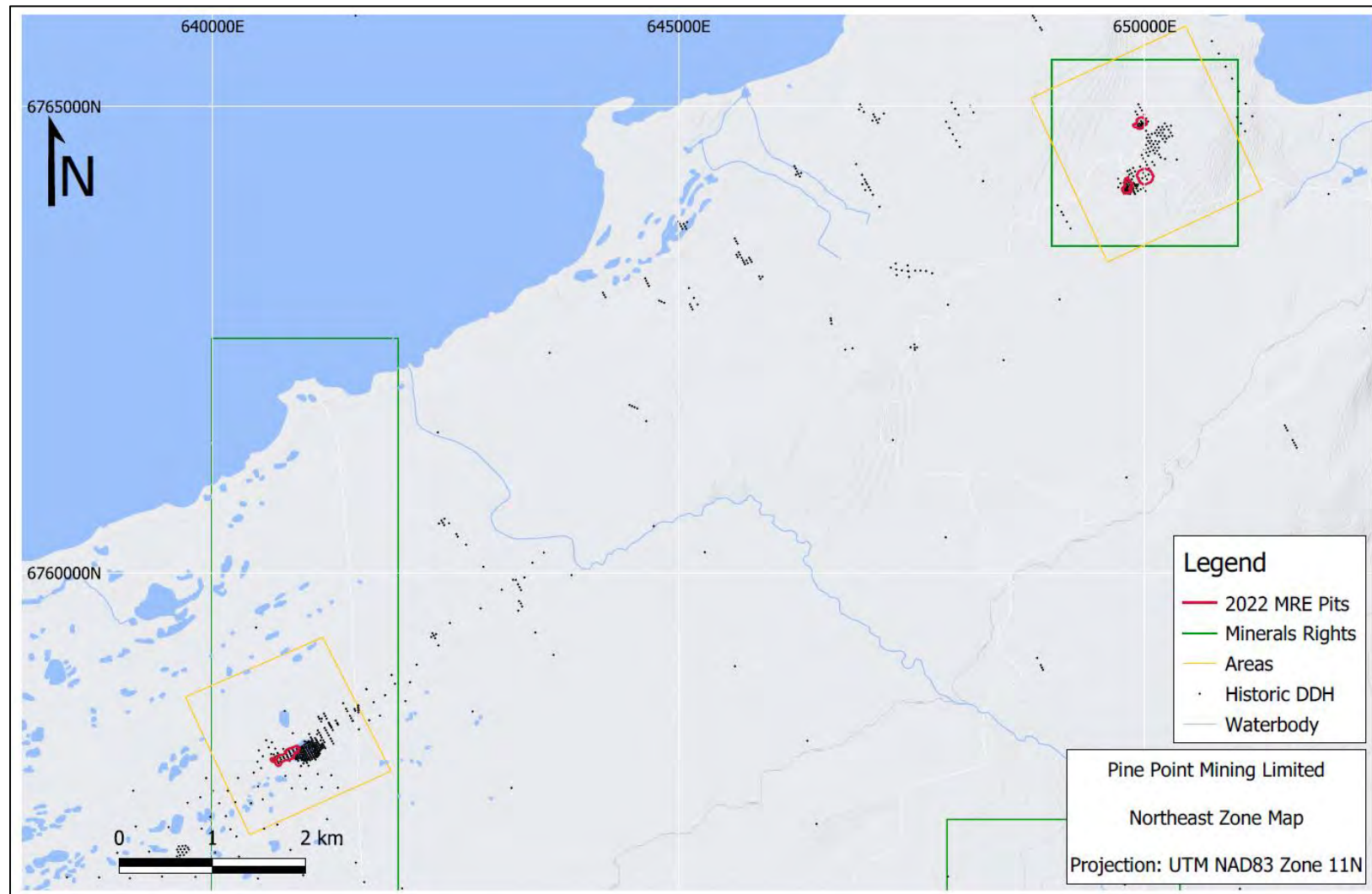


Figure 10-7: N204 Zone map showing the location of historical collars (black) and recent holes from 2017-2019 (none in this case)



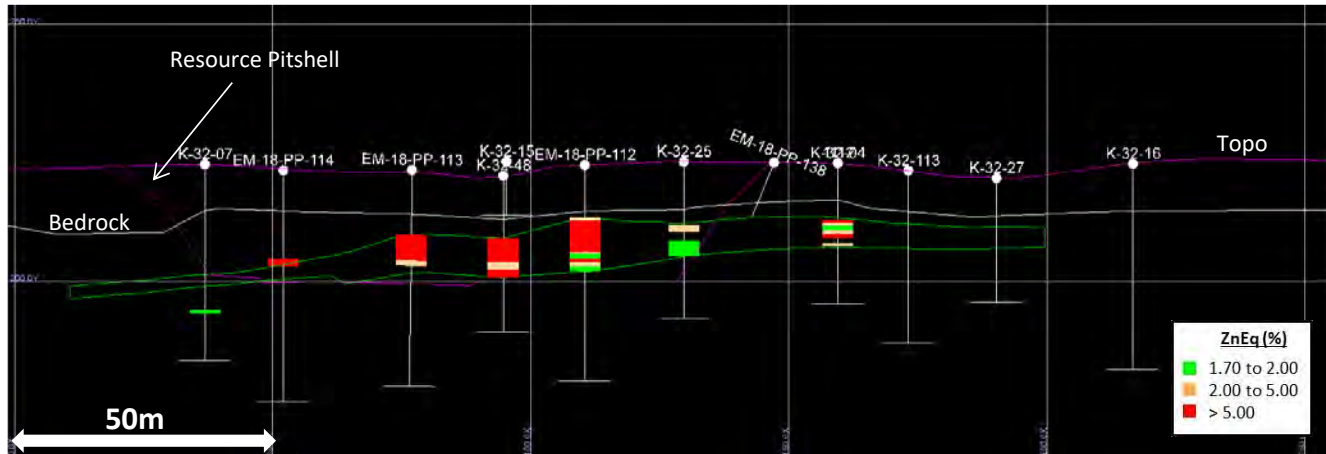


Figure 10-9: Cross-section view in the East Mill Area looking NW

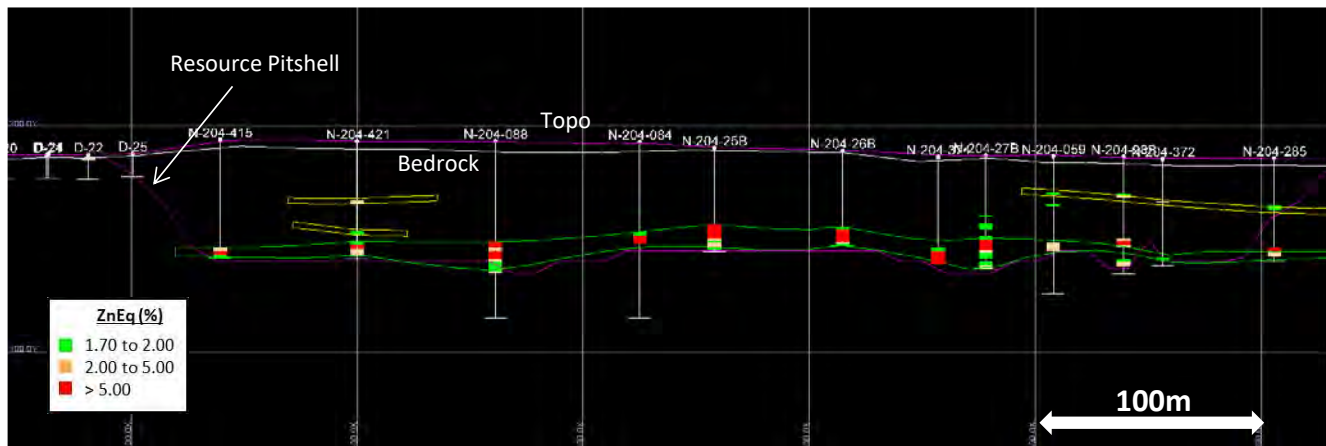


Figure 10-10: Cross-section view in the N204 Area looking NW

It should be noted that yellow zones to the left are pinch-outs from significantly larger zones showing geological continuity north of that section

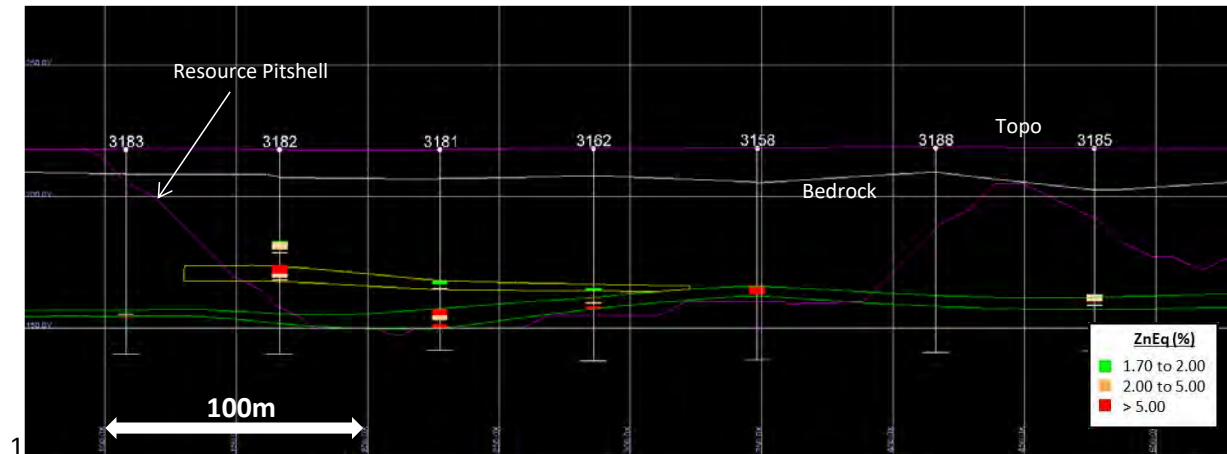


Figure 10-11: Cross-section view in the Central Area looking NW

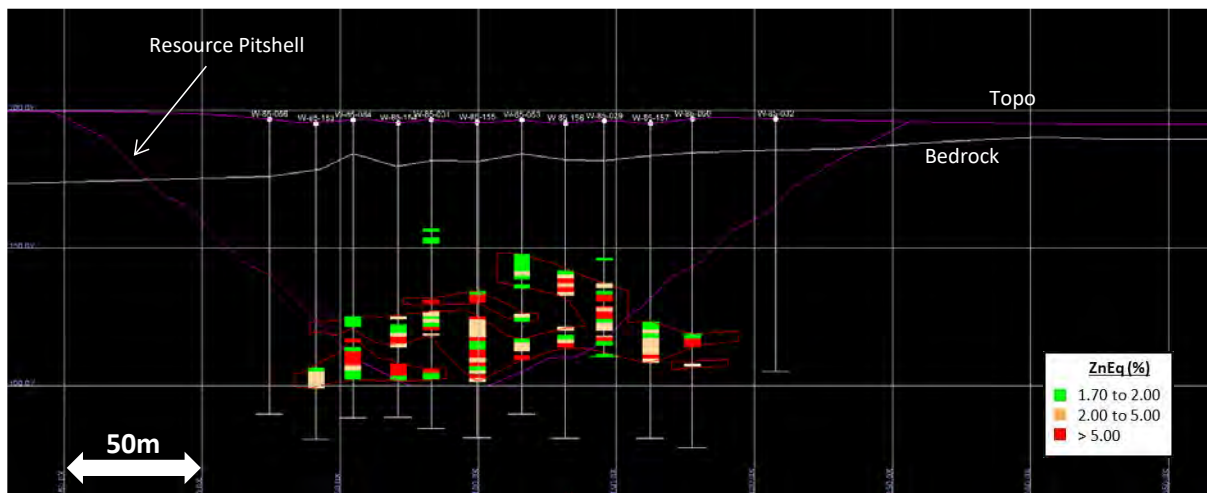


Figure 10-12: Cross-section view in the North Area looking NE (Prismatic Zone)



Figure 10-13: Cross-section view in the North Area looking NE (Tabular Zones)

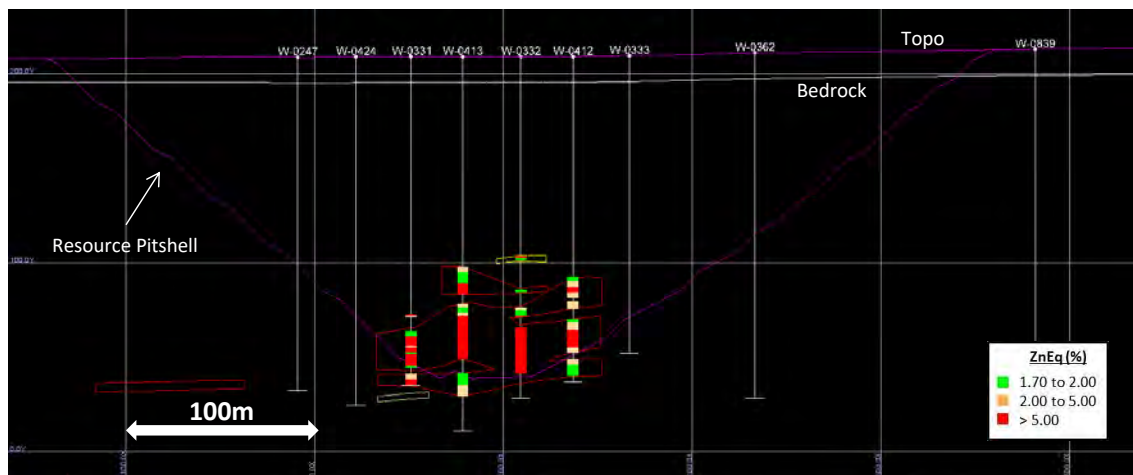


Figure 10-14: Cross-section view in the West Area looking NW (Prismatic Zone)

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 Quality Assurance / Quality Control ("QA/QC") procedures or recovery is available. Previous technical reports, including Siegal and Gann 2017, quote Mr. Ross Burns former Pine Point geologist, who indicates Cominco Ltd. used an x-ray fluorescence ("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, rock quality designation ("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 Project from February 2017 to date of this report. At the time of the data cut-off for the 2019 MRE, PPML completed infill drilling of 59,105 m in 900 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 5% of the sample intervals are less than 0.5 m and taken in areas of high-grade mineralization to reflect sharp control boundaries. Approximately 3% 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 $\frac{1}{4}$ 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 2018-2019, samples were shipped to the ALS prep laboratory in Yellowknife and on to Vancouver for analysis. Both Bureau Veritas 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, and 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-MS 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 2018-2019 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 ICP-MS detection limits 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 atomic emission spectrometry ("ICP AES")) 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 placed 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 programs.

QA/QC 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 as listed below:

- 152 blank samples, 148 CRM pulps and 138 core duplicates were sent to Bureau Veritas as part of the 2017 QA/QC program;
- 100 blank samples, 101 CRM pulps and 101 core duplicates were sent to ALS Chemex as part of the early 2018 QA/QC program;
- 643 blank samples, 884 CRM pulps and 653 core duplicates were sent to ALS Chemex as part of the late 2018 and early 2019 QA/QC program (referred below as the "2019 Winter Program");
- 294 blank samples, 190 CRM pulps and 219 core duplicates were sent to ALS Chemex as part of the late 2019 drilling QA/QC program;
- 119 blank samples, 83 CRM pulps and 87 core duplicates were sent to ALS Chemex as part of the 2019 Cominco historical re-logging 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 ($\frac{1}{2}$ core or $\frac{1}{4}$ 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 $\frac{1}{4}$ of core being used for the primary analysis and $\frac{1}{4}$ core for the subsequent duplicate analysis, leaving $\frac{1}{2}$ 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.

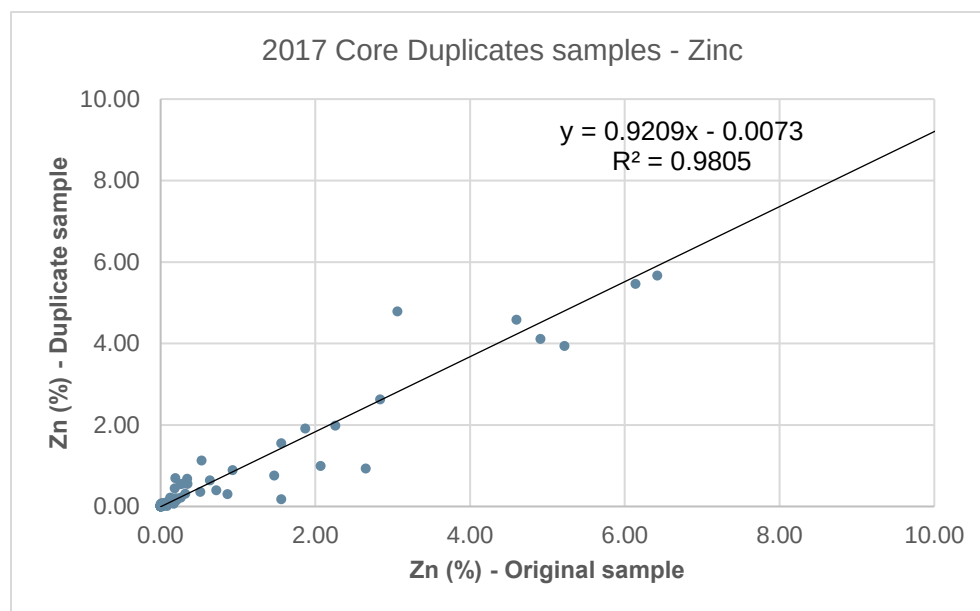


Figure 11-1: Linear graph of Zn duplicates for the 2017 drilling campaign

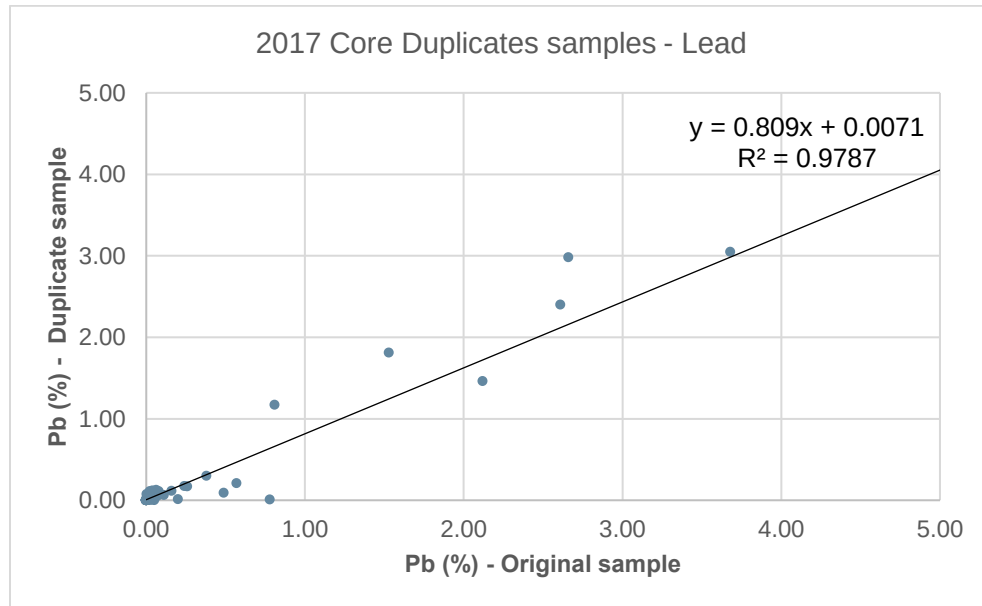


Figure 11-2: Linear graph of Pb duplicates for the 2017 drilling campaign

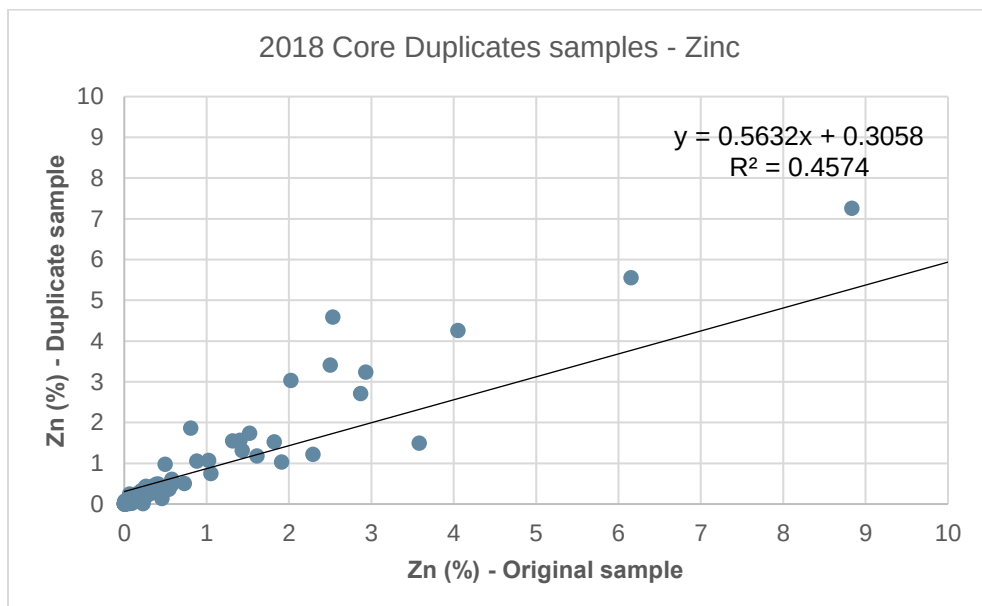


Figure 11-3: Linear graph of Zn duplicates for the 2018 drilling campaign

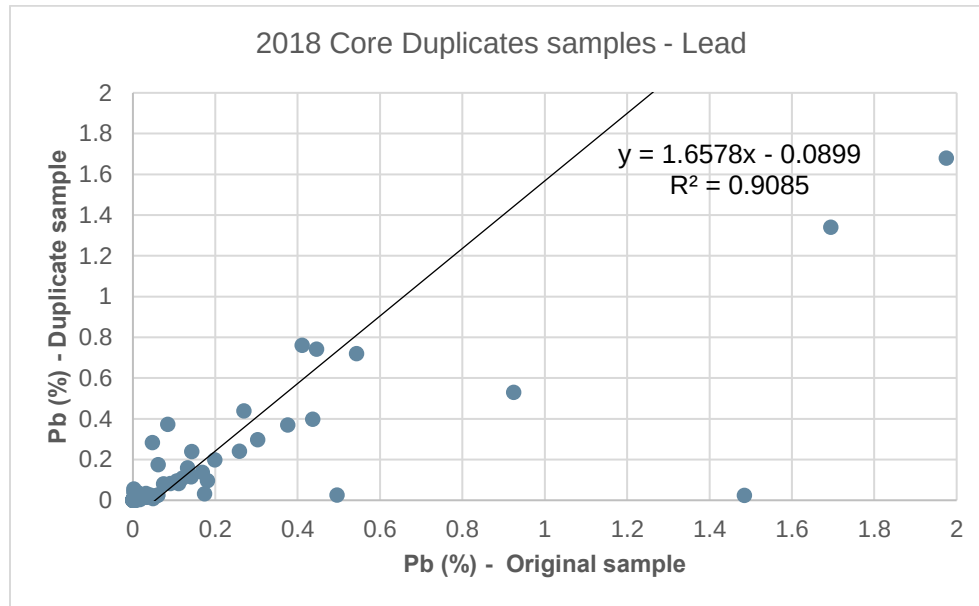


Figure 11-4: Linear graph of Pb duplicates for the 2018 drilling campaign

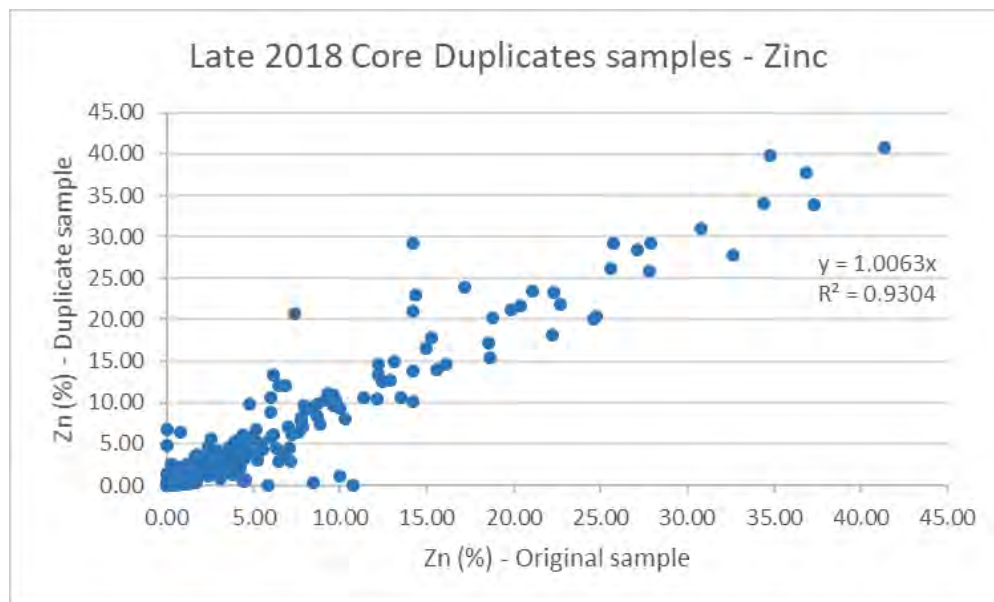


Figure 11-5: Linear graph of Zn duplicates for the Winter 2019 drilling campaign

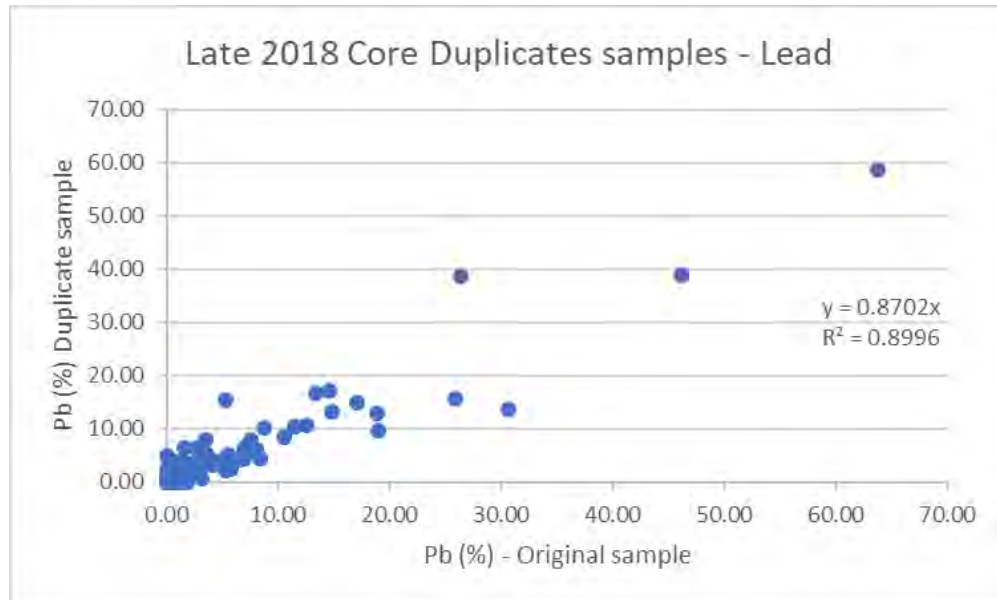


Figure 11-6: Linear graph of Pb duplicates for the Winter 2019 drilling campaign

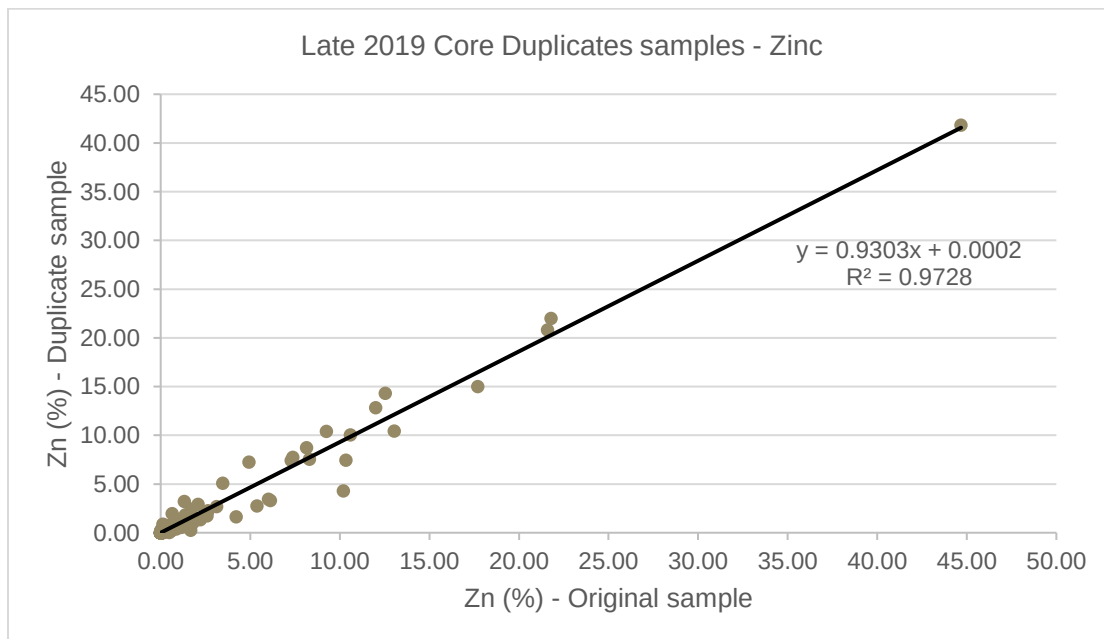


Figure 11-7: Linear graph of Zn duplicates for the late 2019 drilling campaign

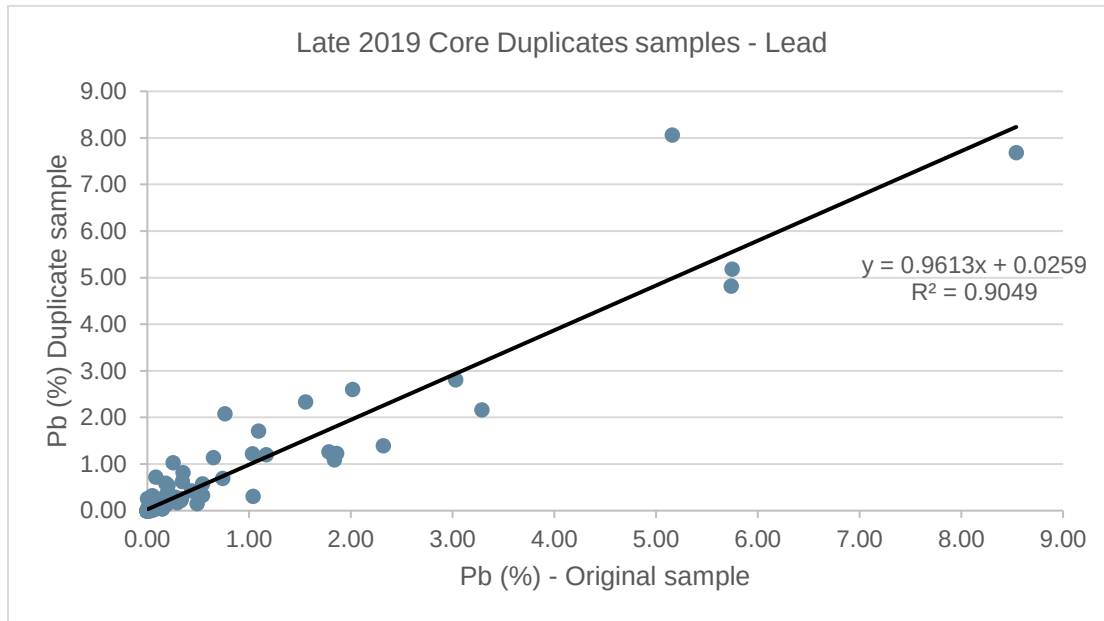


Figure 11-8: Linear graph of Pb duplicates for the late 2019 drilling campaign

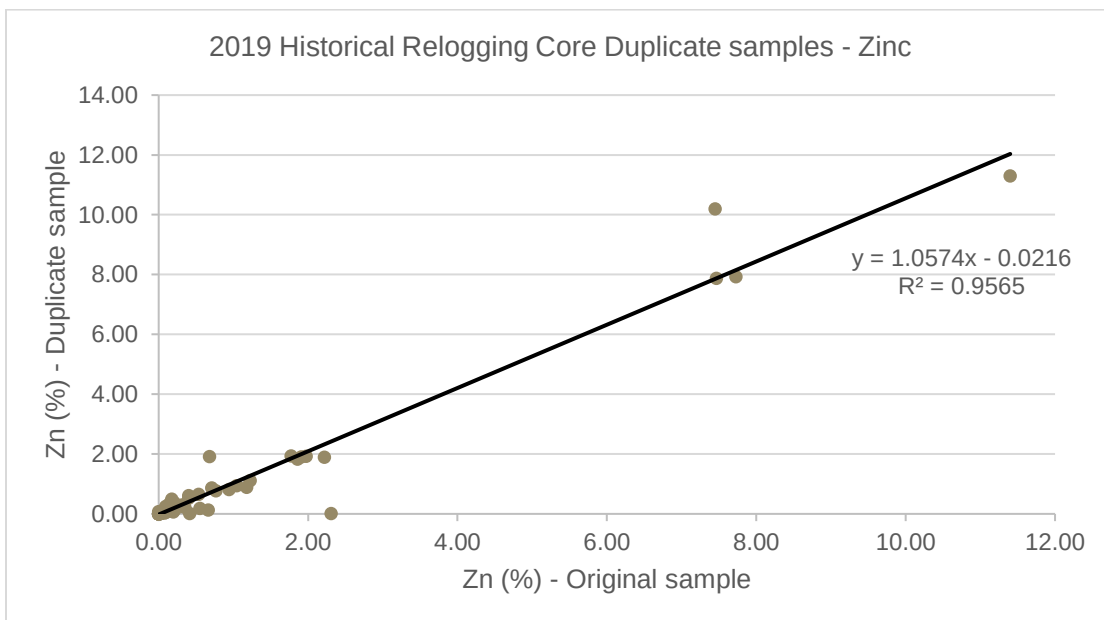


Figure 11-9: Linear graph of Zn duplicates for the late 2019 historical relogging program

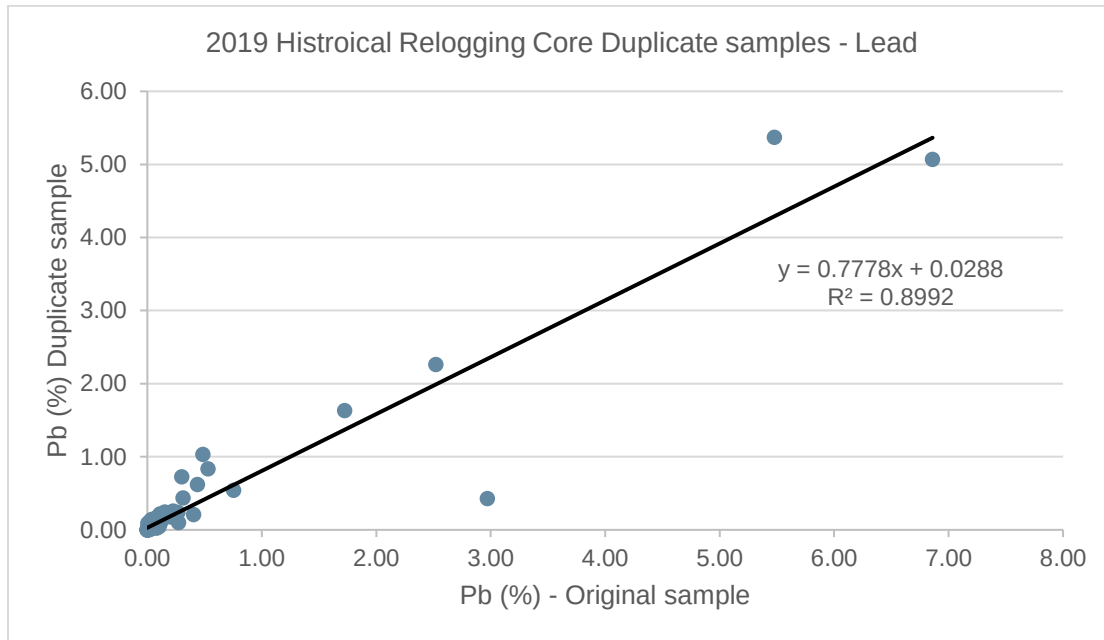


Figure 11-10: Linear graph of Pb duplicates for the late 2019 historical relogging program

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, 2018 and Winter 2019 programs.

One blank sample was inserted for every 20 samples.

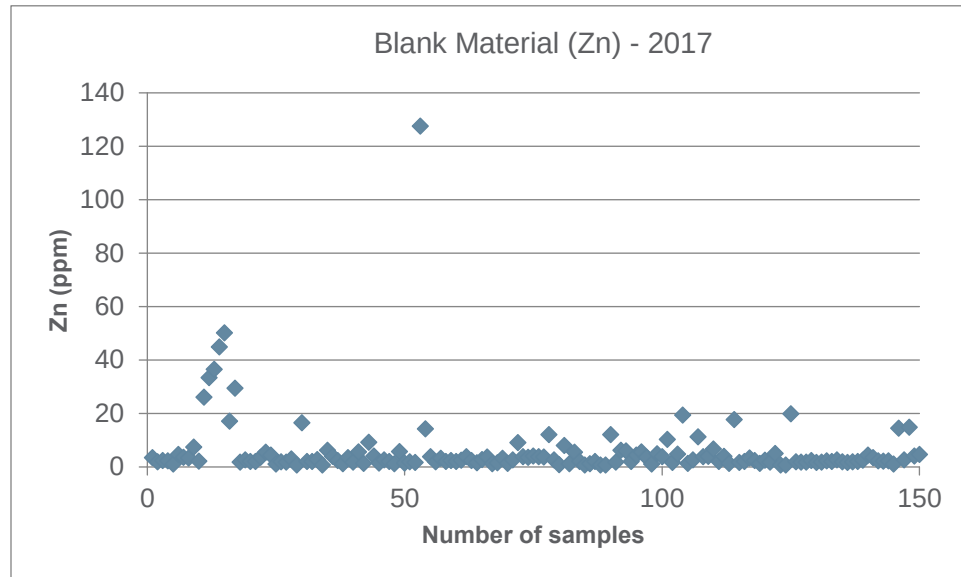


Figure 11-11: Results of Zn blanks samples used during the 2017 drilling program

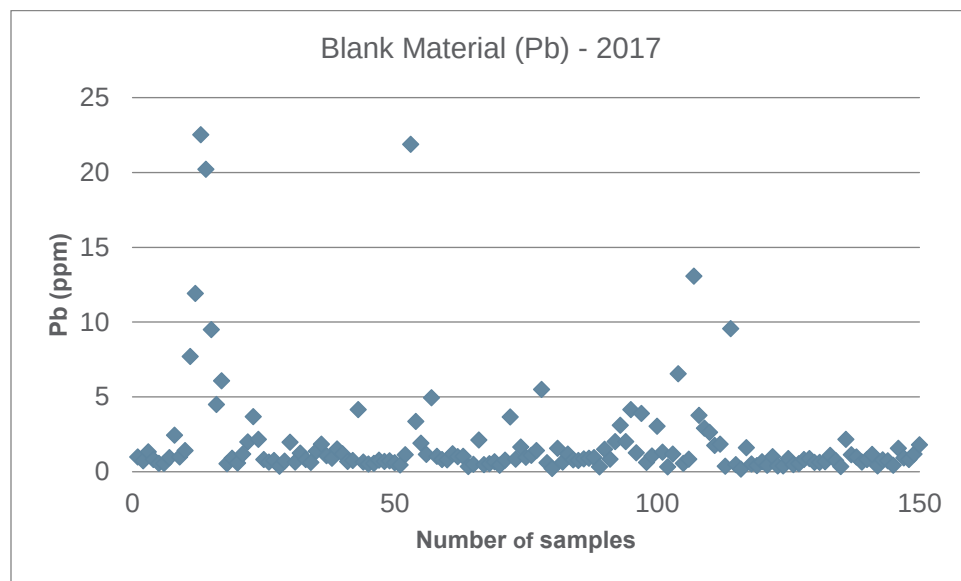


Figure 11-12: Results of Pb blanks samples used during the 2017 drilling program

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.

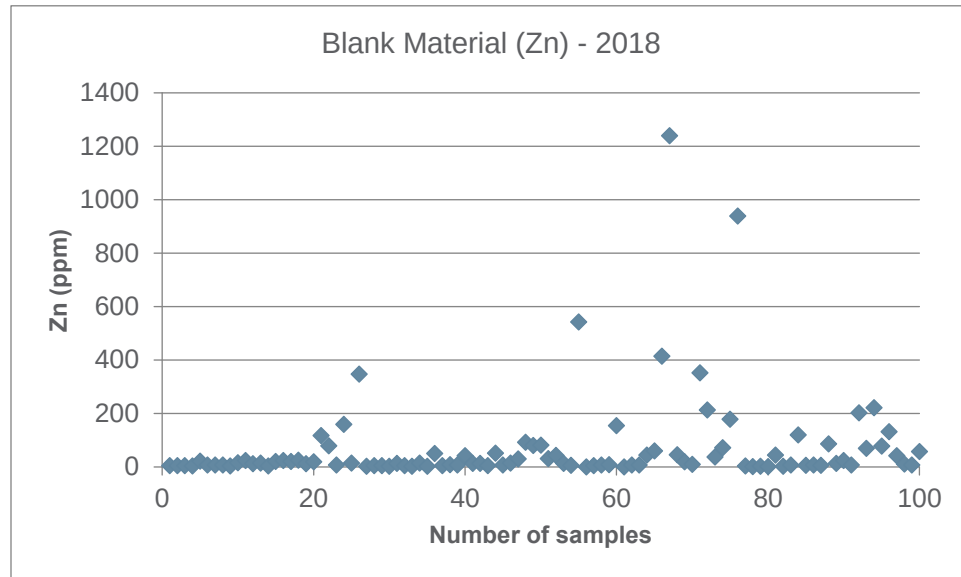


Figure 11-13: Results of Zn blanks samples used during the 2018 drilling program

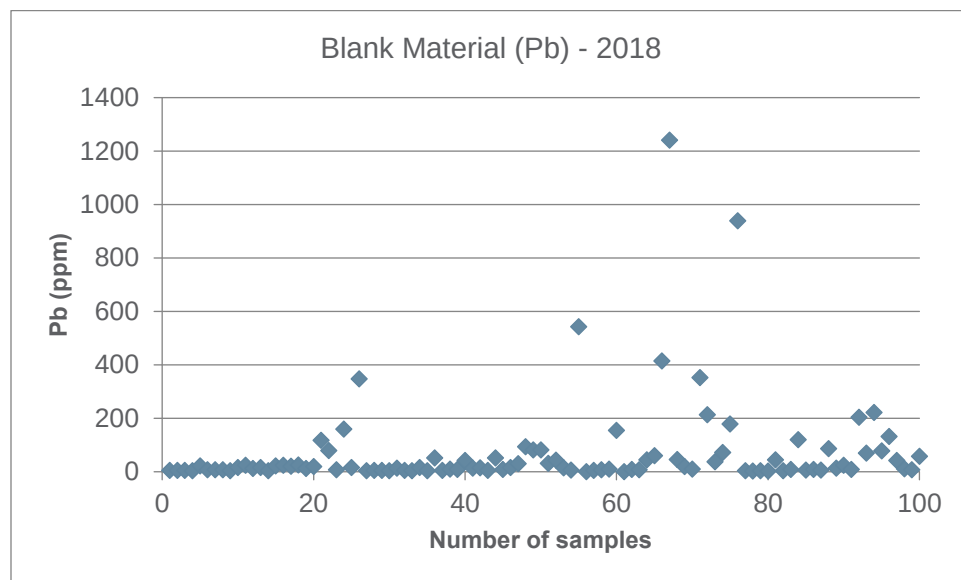


Figure 11-14: Results of Pb blanks samples used during the 2018 drilling program

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 obtained, including one in particular for lead with 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.

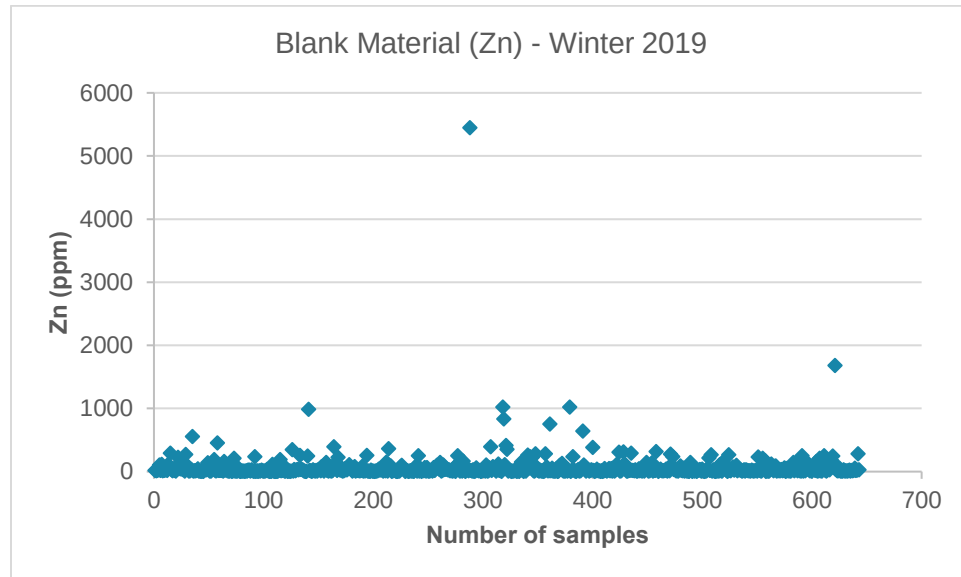


Figure 11-15: Results of Zn blanks samples used during the Winter 2019 drilling program

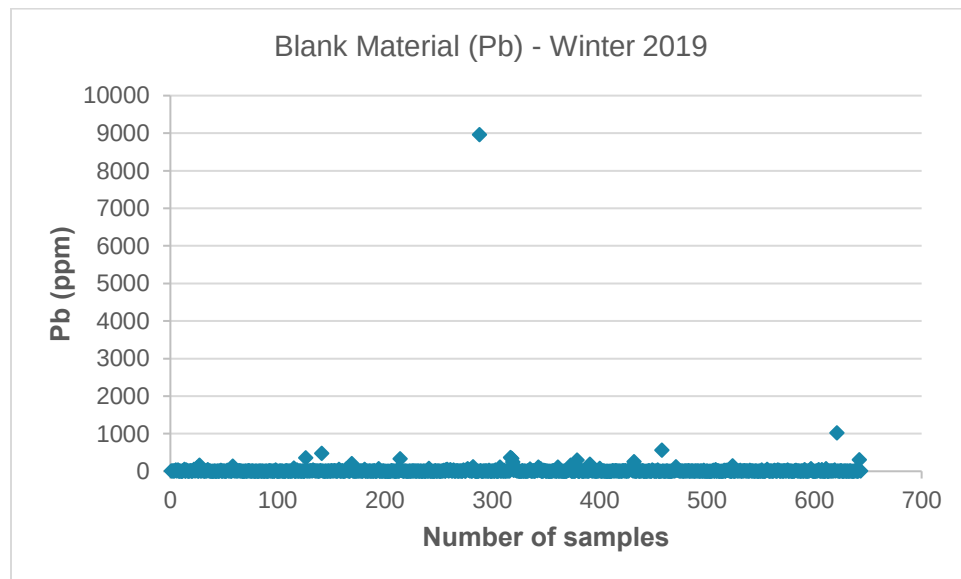


Figure 11-16: Results of Pb blanks samples used during the Winter 2019 drilling program

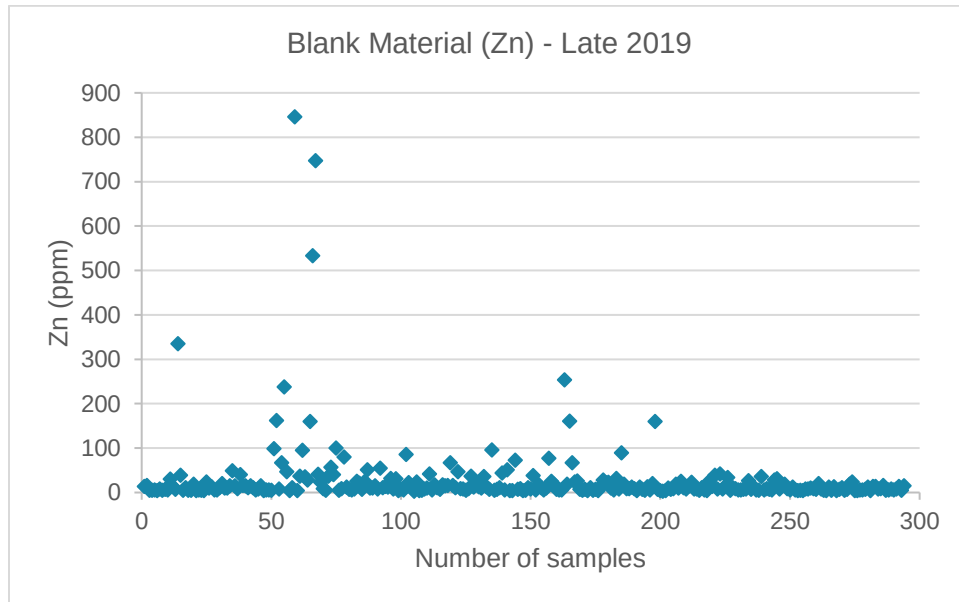


Figure 11-17: Results of Zn blank samples used during the late 2019 drilling program

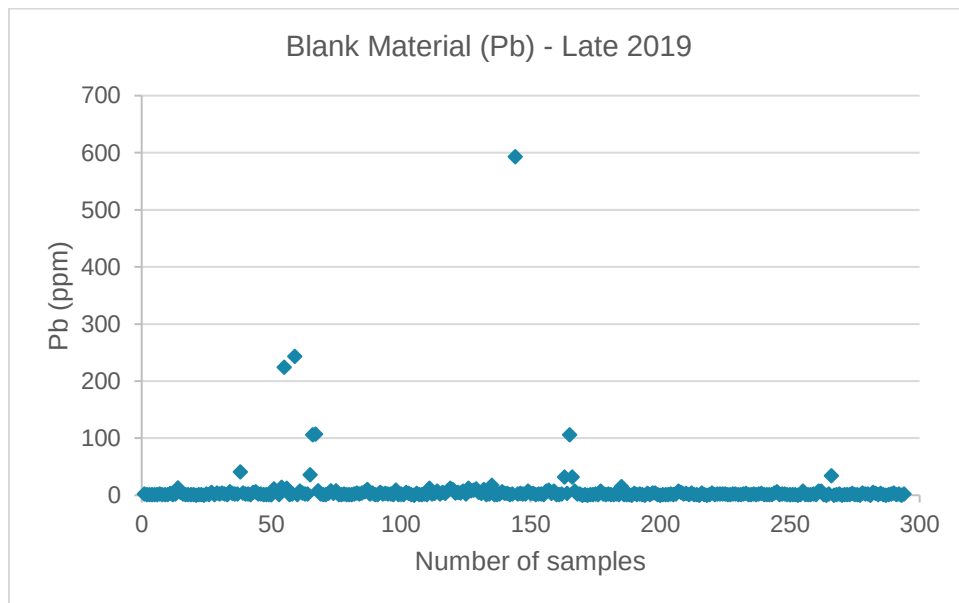


Figure 11-18: Results of Pb blank samples used during the late 2019 drilling program

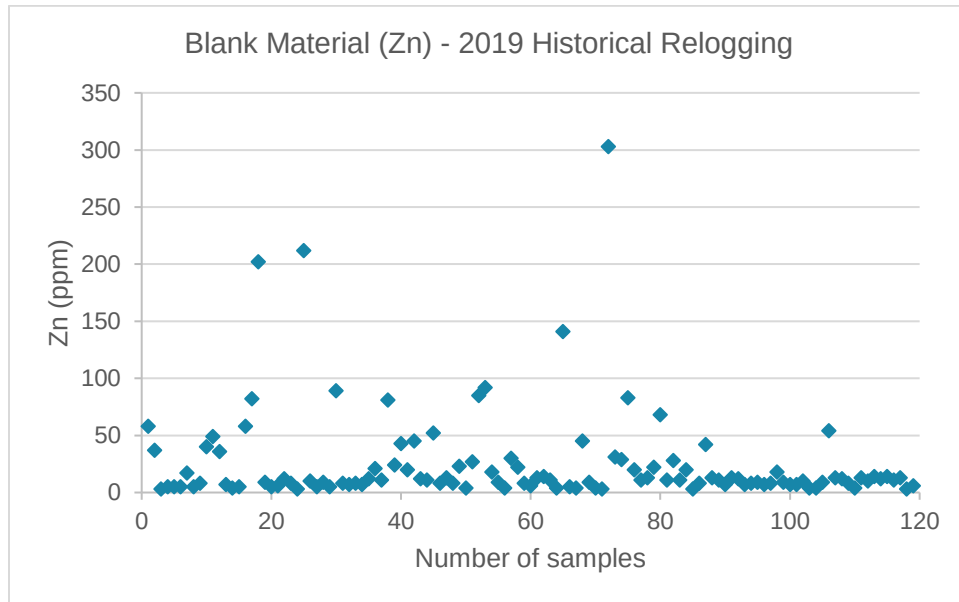


Figure 11-19: Results of Zn blank samples used during the late 2019 historical relogging program

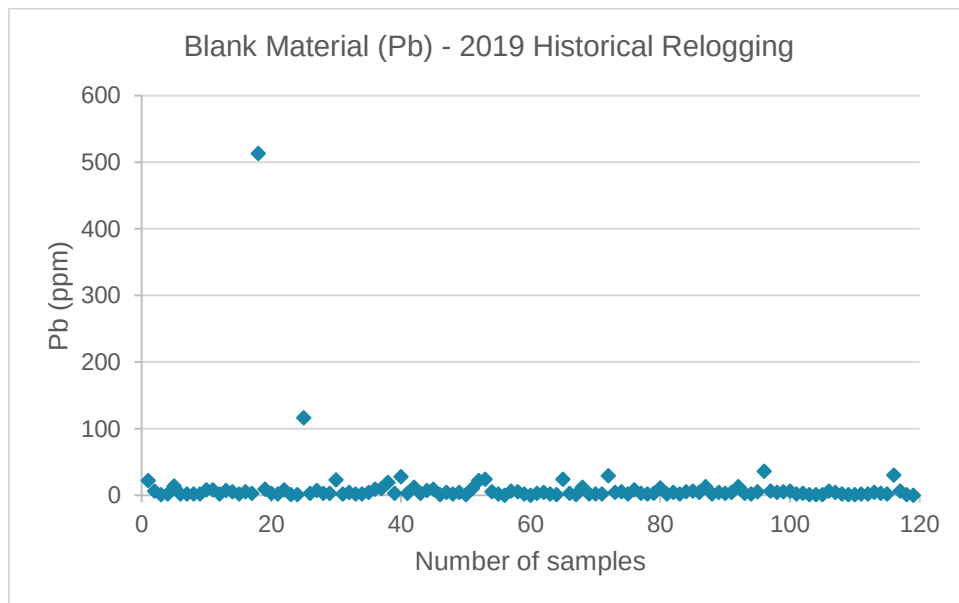


Figure 11-20: Results of Pb blank samples used during the late 2019 historical relogging program

One sample in the Winter 2019 sampling showed extreme lead and zinc contamination. This was found to be within a zone of high grade and the error occurred at the laboratory. Measures, including the addition of circuit washes to be done at locations indicated by the sampling geologist, have been taken to correct the occurrence. In order to prevent sample to sample contamination during crushing, samples with >2% visually estimated sulphides were flagged for the lab to run an additional clean sand flush through the crusher. Results obtained after the 288th blank showed minimal contamination thereafter.

The first several batches of blanks from the late 2019 drilling and 2019 historical relogging programs contained several blanks with elevated Zn and Pb. These samples were all found to be processed immediately after high-grade core samples, indicating that sample to sample contamination occurred during crushing and/or pulverizing.

The ALS preparation lab in Yellowknife was contacted about the contamination issue and several staff geologists visited the lab to better understand their procedures. The following corrective measures were taken to address the contamination:

- Samples flagged for post-prep equipment cleaning now triggered flushes of the pulverizer as well as the crusher;
- The sulphide % threshold to trigger cleaning sand flushes was reduced to >0.5% from 2%;
- It was discovered that ALS were splitting samples and processing even and odd sample numbers on separate machines. In order test both sets of equipment, additional blank samples were inserted on even sample numbers.

The results of these corrective actions can be seen starting from sample 170 in the late 2019 drilling and sample 90 in the 2019 historical relogging.

11.3.3 Certified Reference Materials (Standards)

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



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

Standard #	Pb (%)	Zn (%)
ME-1301	0.188	0.797
ME-1409	0.065	0.771
ME-1601	0.219	0.942
OREAS 34h	1.44	12.18
OREAS 34i	0.292	1.34
OREAS 34k	2.08	23.76
OREAS 131b	1.88	3.04
OREAS 132b	3.86	5.25
OREAS 133a	4.9	10.87
OREAS 352	58.14	2.21

Commercially available Certified Reference Materials ("CRMs") were used as standards during the 2017-2019 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. Towards the end of the 2019 program, CRMs ME-1409, ME-1601, OREAS 131b, OREAS 132b, and OREAS 133a were replaced with OREAS' new MVT style carbonate matrix matched CRMs: OREAS 34h, OREAS 34i, and OREAS 34k.

One CRM was inserted for every 20 samples.



Table 11-3: Results for the Zn CRMs used for the 2017, 2018 and Winter 2019 drilling programs

CRMs	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	84	0.797	0.027	0.723	0.829	3	96.40	1	98.8
ME-1409	138	0.771	0.023	0.726	0.818	0	100.00	0	100.00
ME-1601	215	0.942	0.032	0.854	0.983	3	98.60	7	96.74
OREAS 131b	8	3.04	0.084	2.880	3.220	0	100.00	1	87.50
OREAS 132b	222	5.25	0.112	4.997	5.444	3	98.65	5	97.75
OREAS 133a	202	10.87	0.265	10.212	11.274	4	98.02	3	98.51
OREAS 352	15	2.21	0.112	2.098	2.322	0	100.00	0	100.00

Table 11-4: Results for the Pb CRMs used for the 2017, 2018 and Winter 2019 drilling programs

CRMs	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	84	0.188	0.009	0.171	0.208	4	95.24	0	100.00
ME-1409	138	0.065	0.003	0.060	0.073	6	95.65	1	99.28
ME-1601	215	0.219	0.008	0.203	0.233	5	97.77	1	99.53
OREAS 131b	8	1.88	0.055	1.760	1.980	0	100.00	1	87.50
OREAS 132b	222	3.86	0.080	3.716	4.036	4	98.20	4	98.20
OREAS 133a	202	4.90	0.109	4.675	5.111	4	98.02	1	99.50
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 laboratory were under-reported (Figure 11-30 and Figure 11-31), but this issue was resolved by changing the laboratory 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 Pb value was considered too high.

Table 11-5: Results for the Zn CRMs used for the late 2019 drilling and historical relogging programs

CRMs	Quantity inserted	Certified Zn value (%)	Between Lab SD	Lower limit (-2SD)	Upper limit (+2SD)	Number failed (lower limit)	% Passing quality control	Number failed (upper limit)	% Passing quality control
ME-1409	44	0.771	0.038	0.695	0.847	0	100.00	0	100.00
ME-1601	43	0.942	0.050	0.842	1.042	0	100.00	0	100.00
OREAS 131b	23	3.04	0.119	2.802	3.278	0	100.00	0	100.00
OREAS 132b	47	5.25	0.195	4.860	5.640	0	100.00	0	100.00
OREAS 133a	42	10.87	0.354	10.162	11.578	1	97.62	0	100.00
OREAS 34h	27	12.18	0.253	11.674	12.686	2	92.59	2	92.59
OREAS 34i	24	1.34	0.041	1.258	1.422	1	95.83	0	100.00
OREAS 34k	23	23.76	0.508	22.744	24.776	1	95.65	8	65.22

CRM OREAS 34k had 8 of 23 samples over the certified Zn value +2SD making it the only CRM with <90% pass rate. This was likely due to the analytical over-range value for Zn by ICP that ALS uses to trigger a re-run using titration is 25%, very close to the certified Zn value of OREAS 34k, which is 23.76%.

Table 11-6: Results for the Pb CRMs used for the late 2019 drilling and historical relogging programs

CRMs	Quantity inserted	Certified Pb value (%)	Between Lab SD	Lower limit (-2SD)	Upper limit (+2SD)	Number failed (lower limit)	% passing quality control	Number failed (upper limit)	% passing quality control
ME-1409	44	0.065	0.002	0.061	0.069	0	100.00	4	90.91
ME-1601	43	0.219	0.012	0.195	0.243	0	100.00	0	100.00
OREAS 131b	23	1.88	0.086	1.708	2.052	0	100.00	0	100.00
OREAS 132b	47	3.86	0.066	3.728	3.992	2	95.74	1	97.87
OREAS 133a	42	4.90	0.162	4.576	5.224	0	100.00	0	100.00
OREAS 34h	27	1.44	0.031	1.378	1.502	2	92.59	1	96.30
OREAS 34i	24	0.292	0.011	0.270	0.314	0	100.00	0	100.00
OREAS 34k	23	2.08	0.050	1.980	2.180	2	91.30	1	95.65

The 2017 CRM results are shown in Figure 11-21 to Figure 11-25.

The 2018 CRM results are shown in Figure 11-26 to Figure 11-29. No sample batches were rejected. Figure 11-30 and Figure 11-31 show the comparison between the Zn results for the ME-1301 standard for 2017 and 2018.

The Winter 2019 CRM results are shown in Figure 11-32 to Figure 11-53. No sample batches were rejected.

The late 2019 drilling and 2019 historical relogging CRM results are shown in Figure 11-38 to Figure 11-53. No sample batches were rejected.

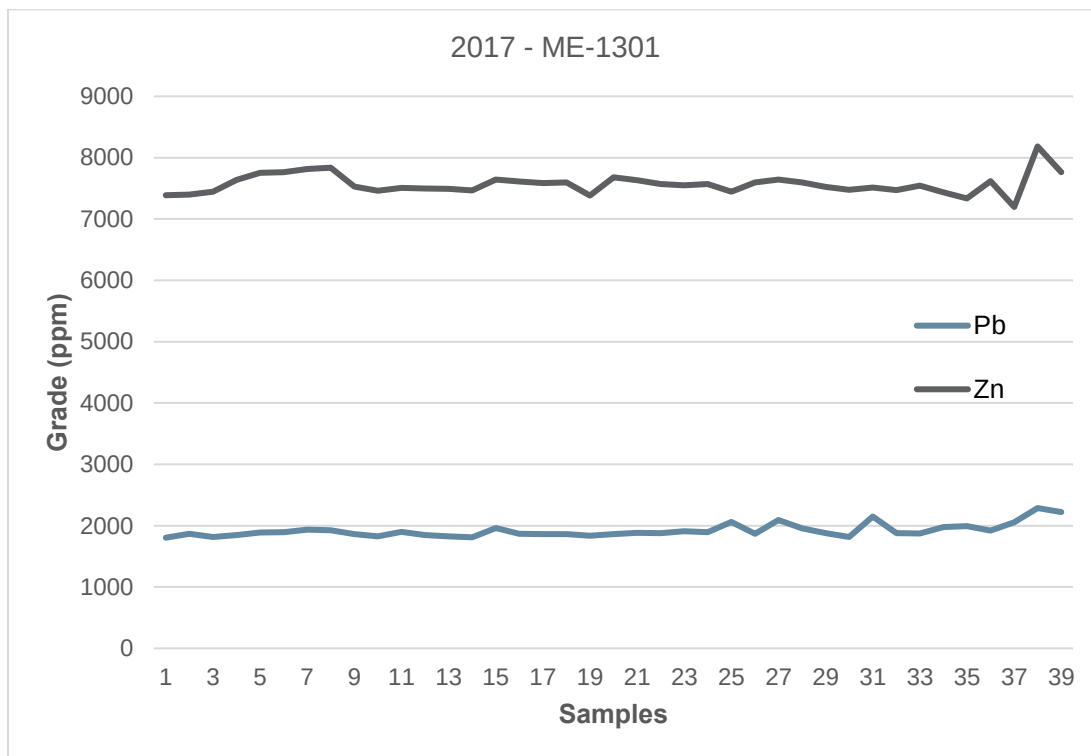


Figure 11-21: 2017 results of standard ME-1301

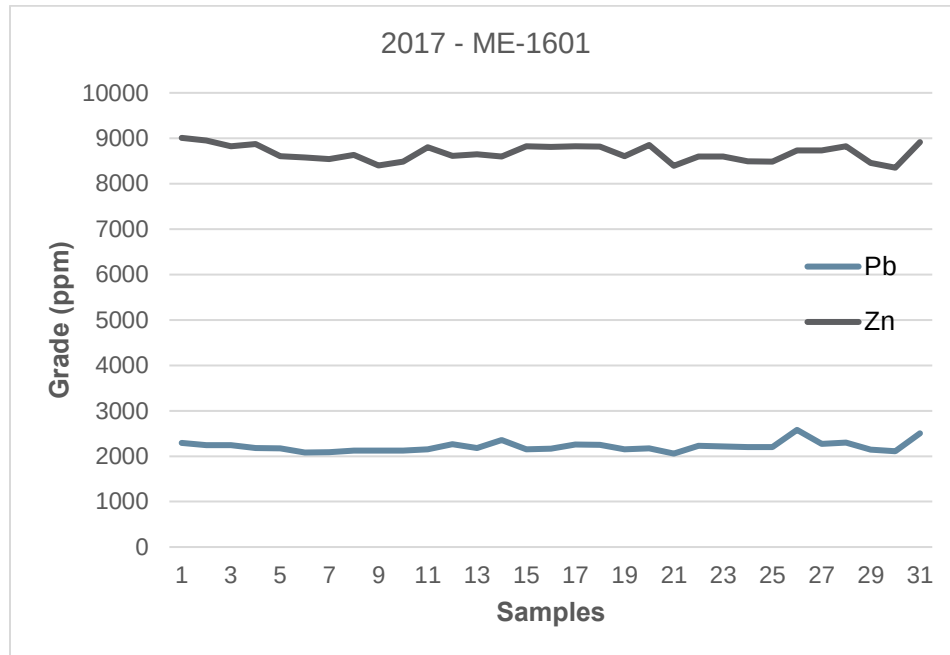


Figure 11-22: 2017 results of standard ME-1601

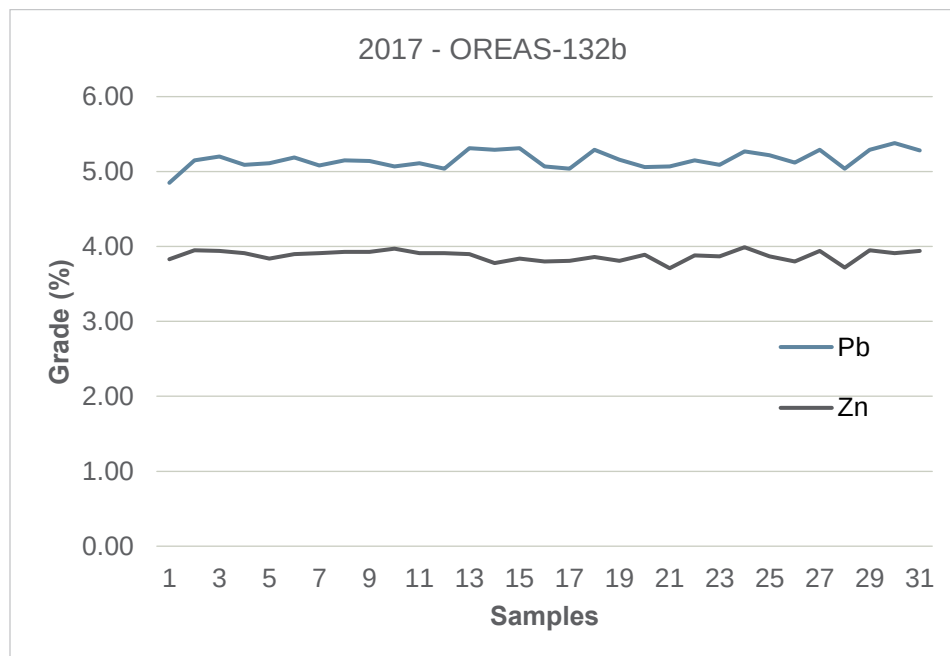


Figure 11-23: 2017 results of standard OREAS-132b

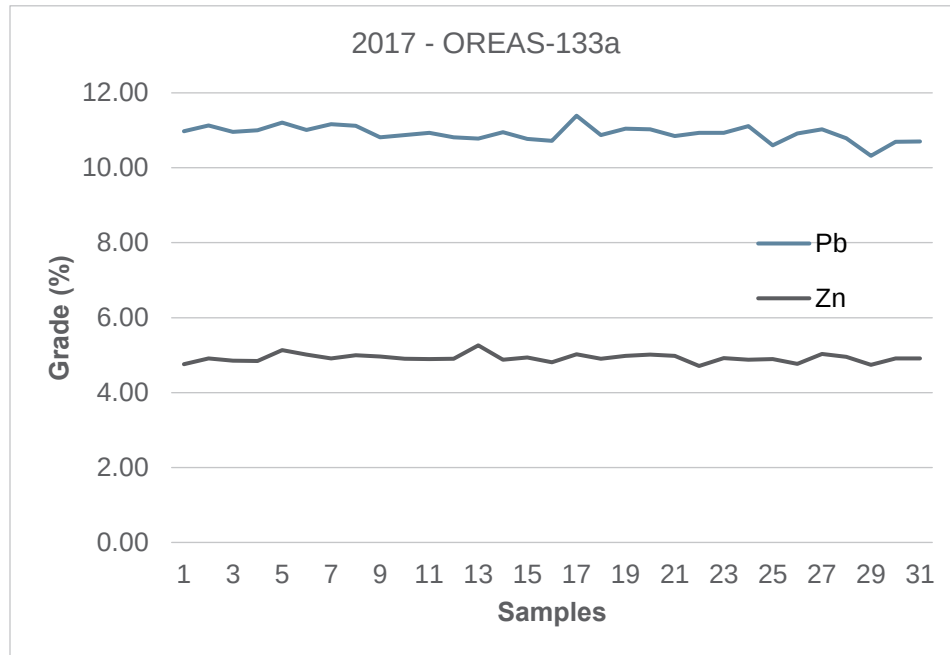


Figure 11-24: 2017 results of standard OREAS-133a

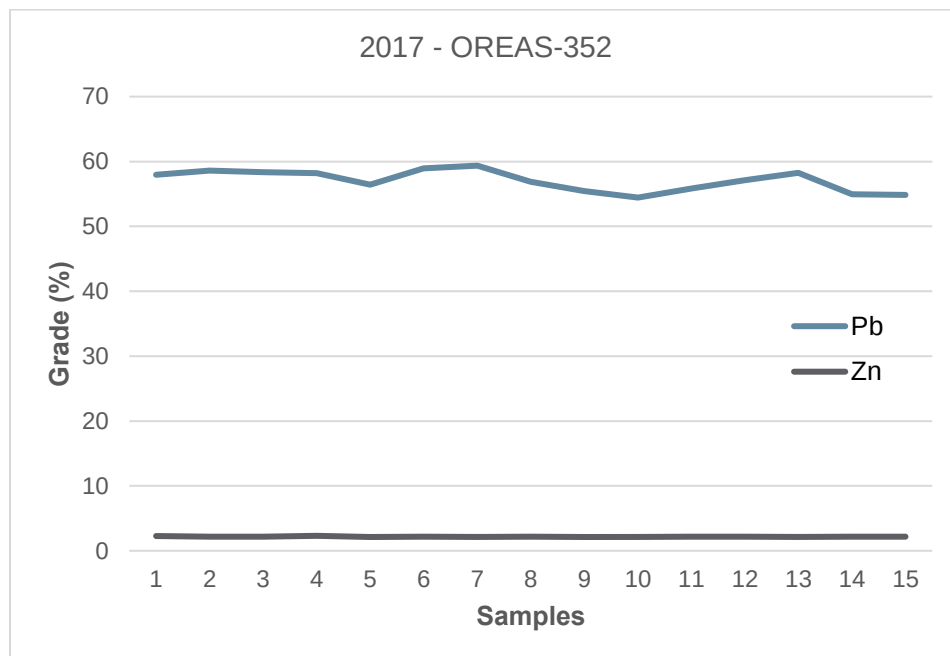


Figure 11-25: 2017 results of standard OREAS-352

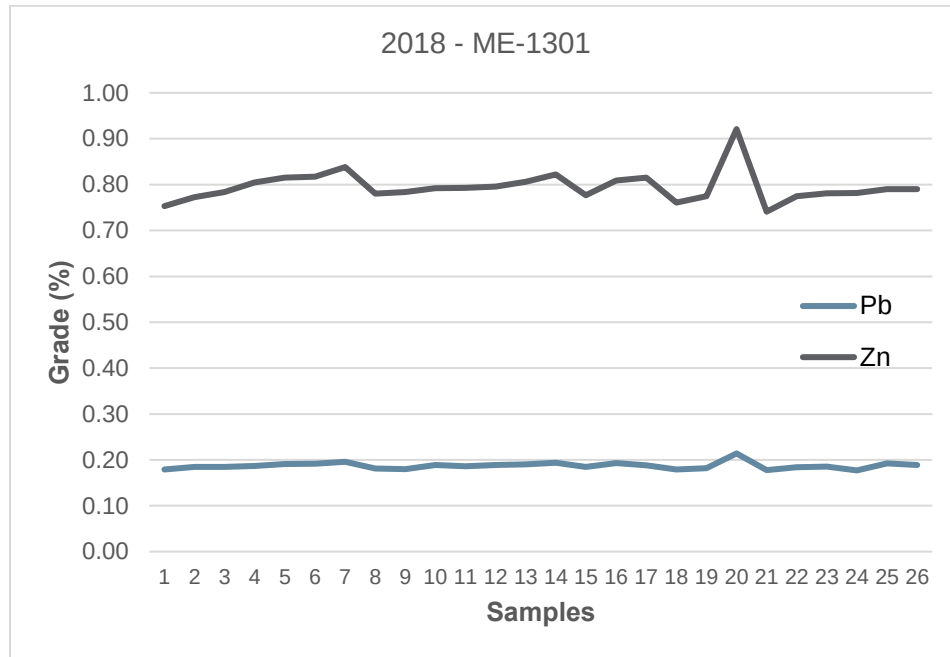


Figure 11-26: 2018 results of standard ME-1301

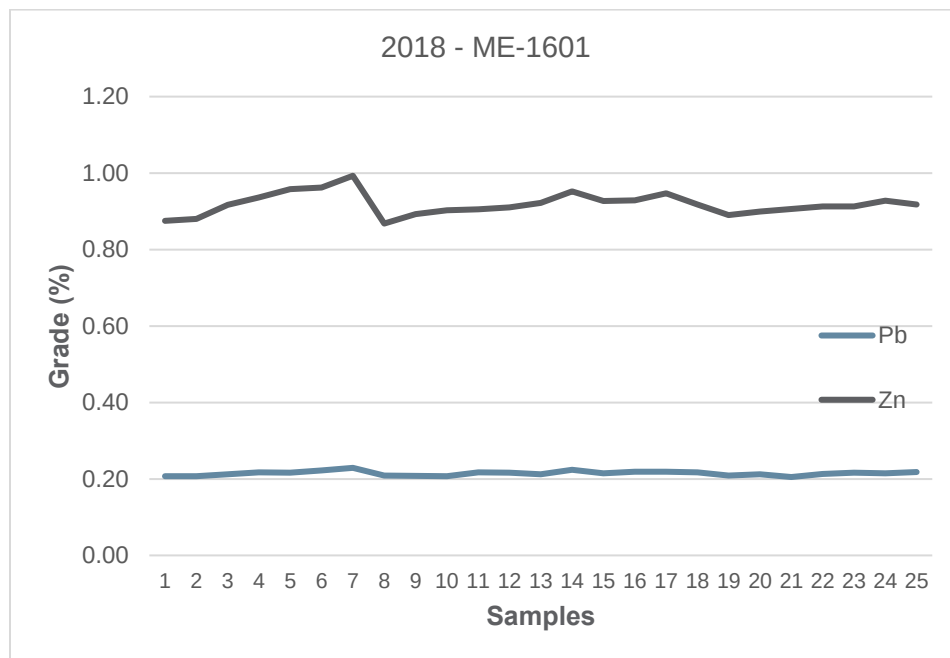


Figure 11-27: 2018 results of standard ME-1601

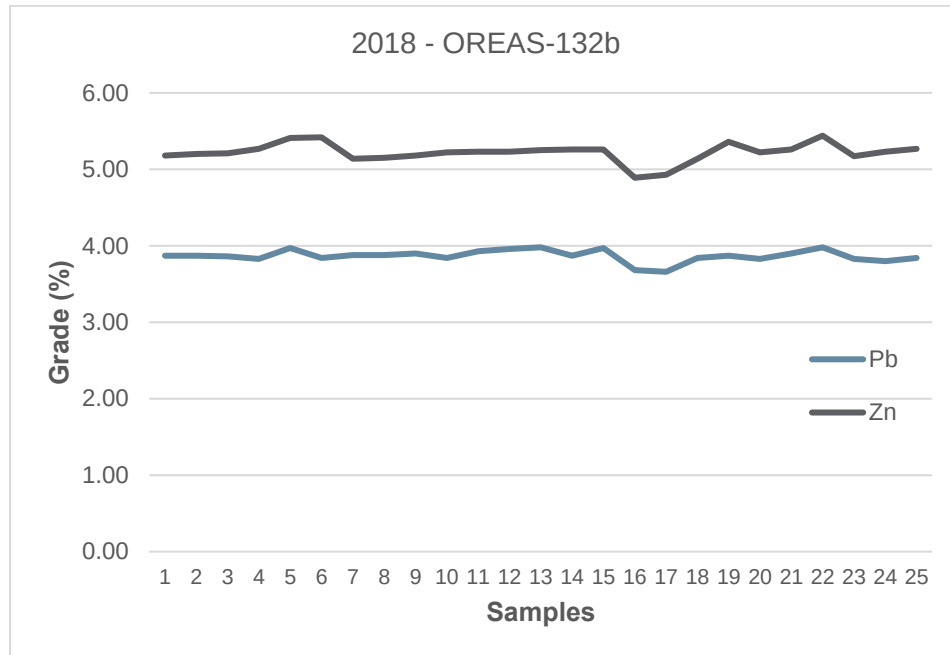


Figure 11-28: 2018 results of standard OREAS-132b

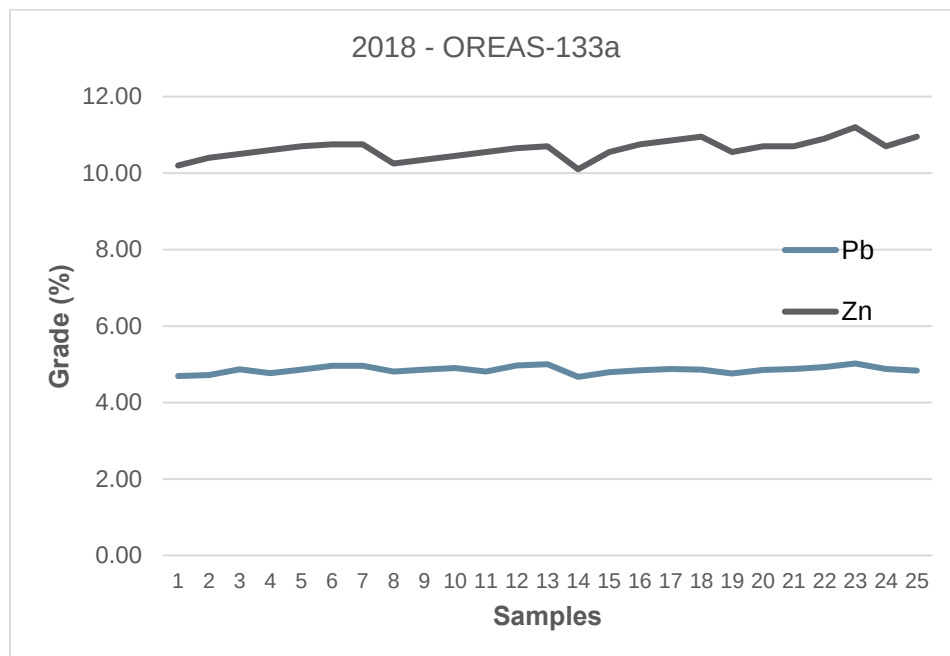


Figure 11-29: 2018 results of standard OREAS-133a

The Zn assays for 2017 and 2018 are plotted in Figure 11-30 and Figure 11-31 for ME-1301 and ME-1601, respectively. These plots show that the underreporting bias for these standards was corrected by changing labs to ALS.

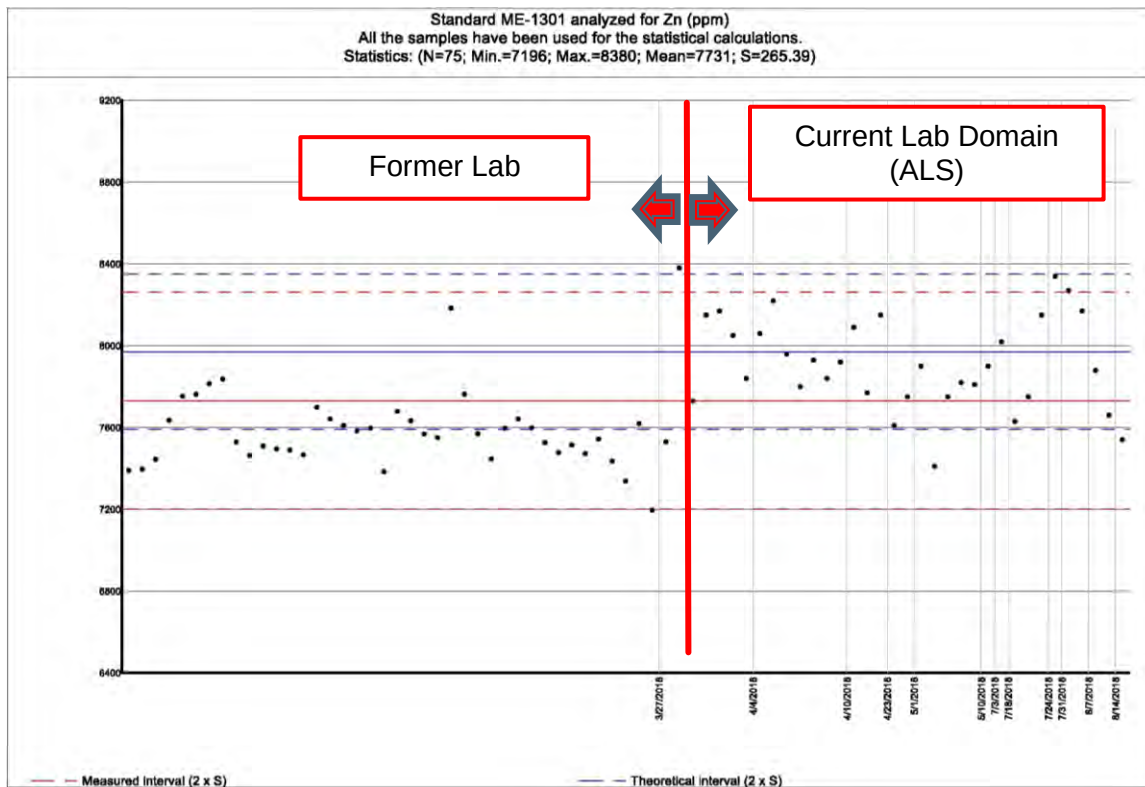


Figure 11-30: 2017 and 2018 Zn results for the ME-1301 standard

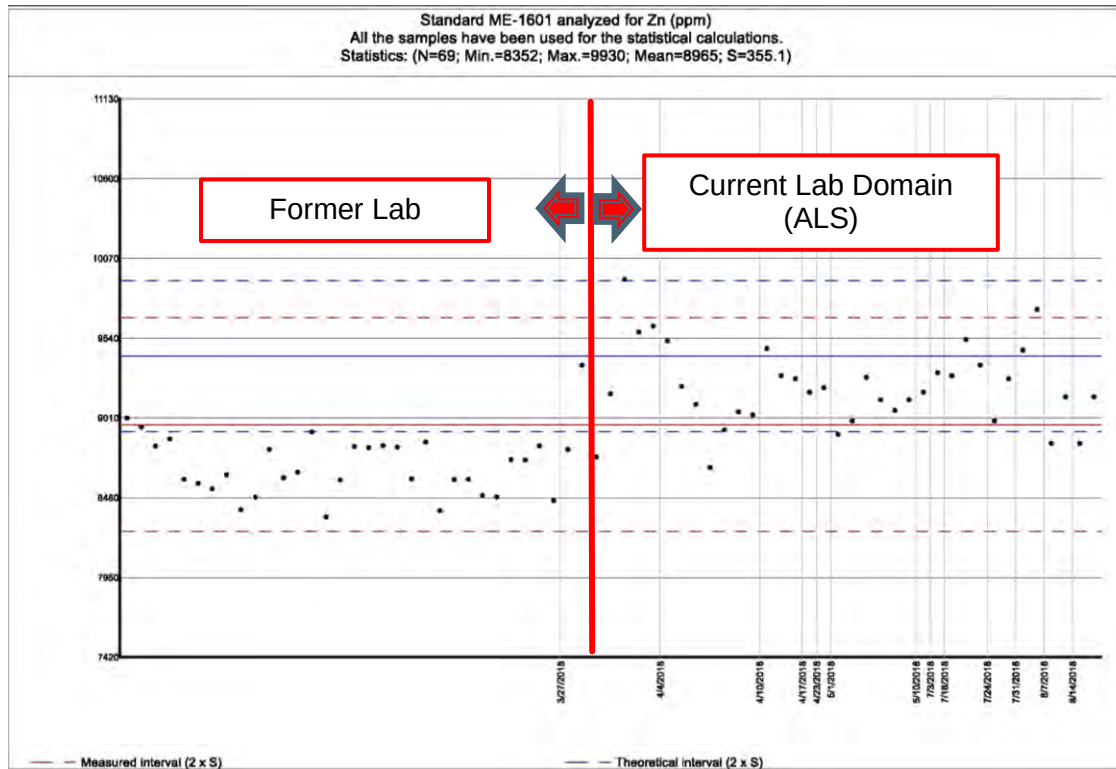


Figure 11-31: 2017 and 2018 Zn results for the ME-1601 standard

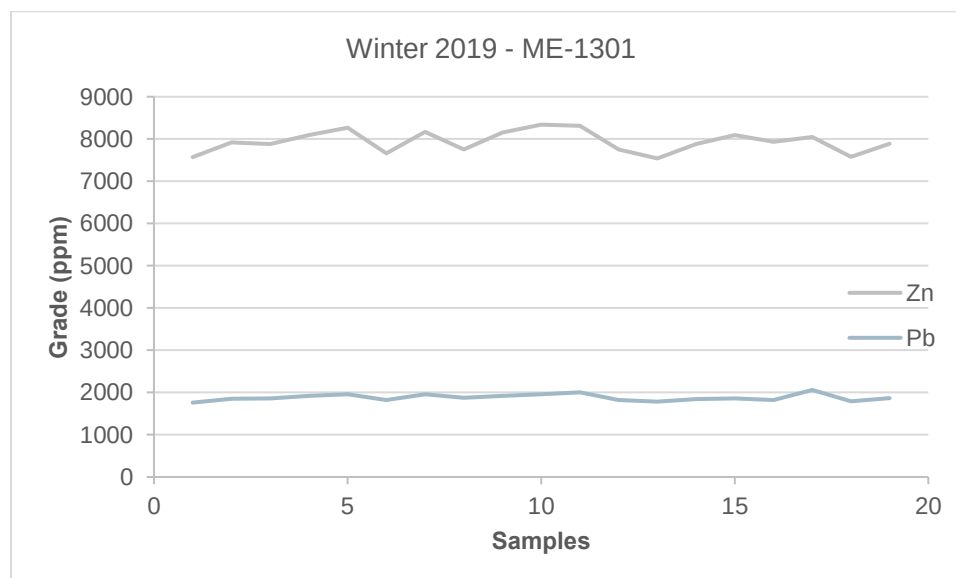


Figure 11-32: Winter 2019 results of standard ME-1301

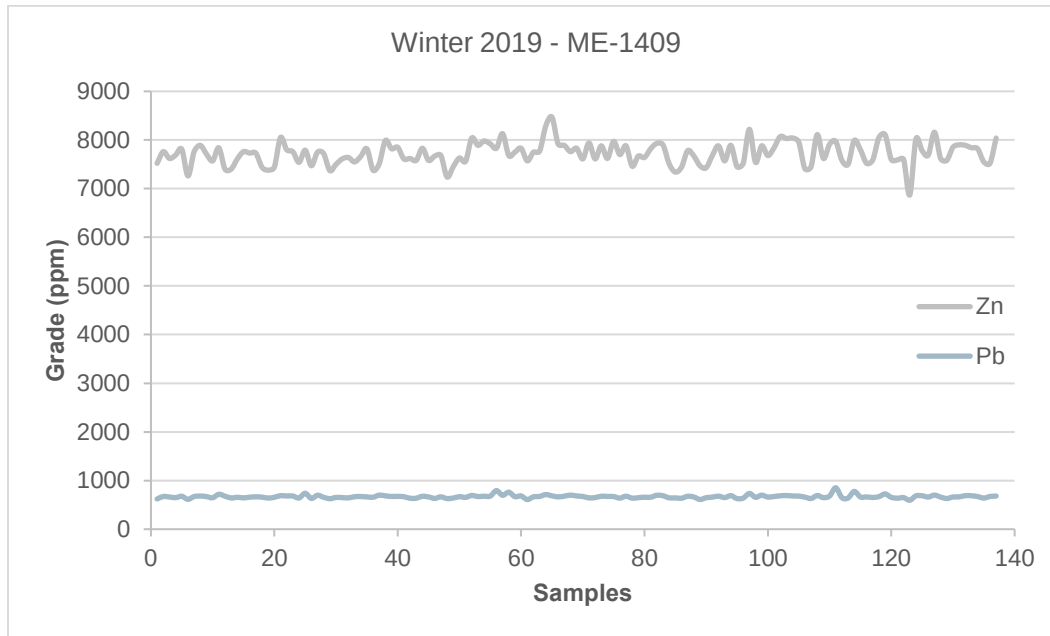


Figure 11-33: Winter 2019 results of standard ME-1409

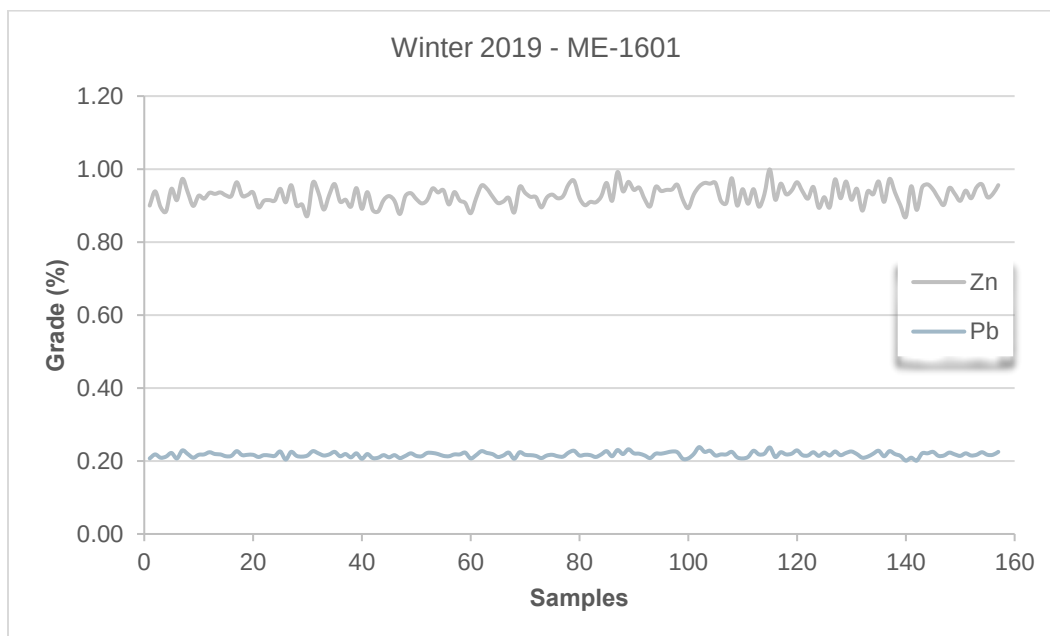


Figure 11-34: Winter 2019 results of standard ME-1601

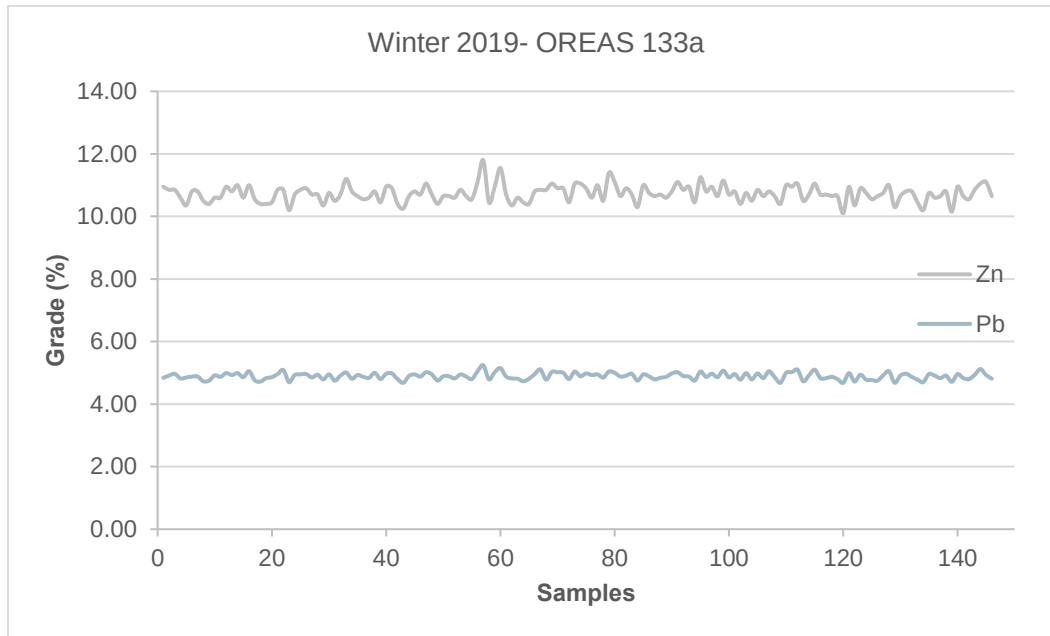


Figure 11-35: Winter 2019 results of standard OREAS-133a

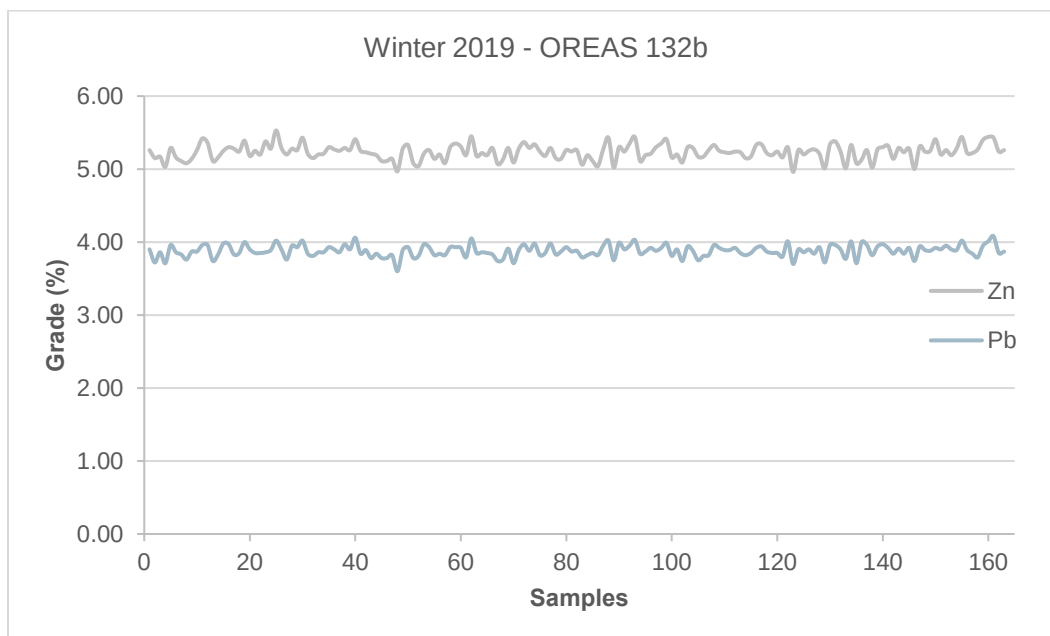


Figure 11-36: Winter 2019 results of standard OREAS-132b

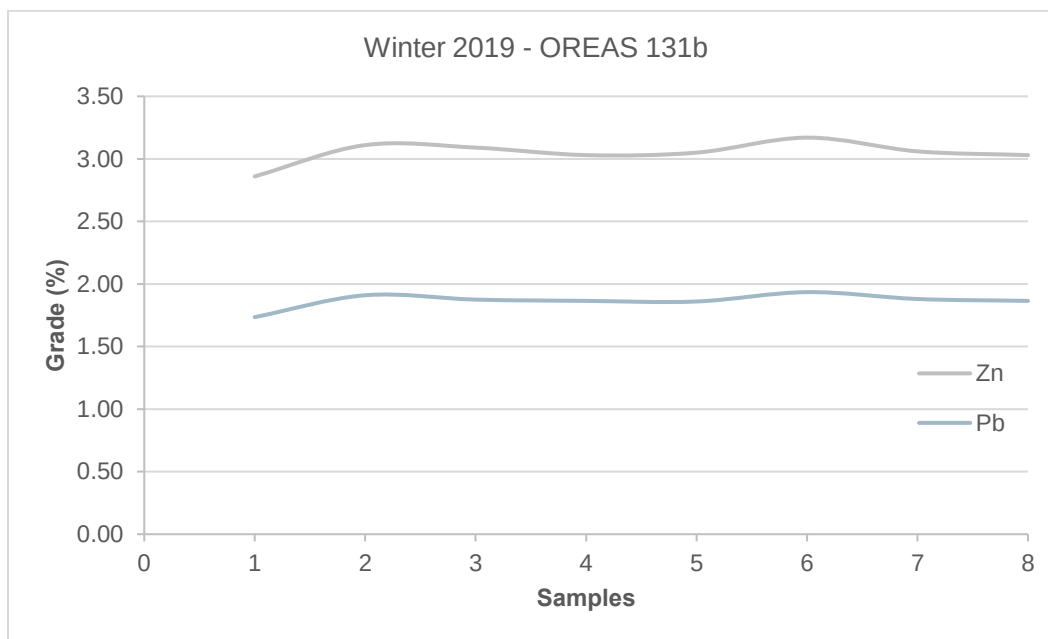


Figure 11-37: Winter 2019 results of standard OREAS-131b

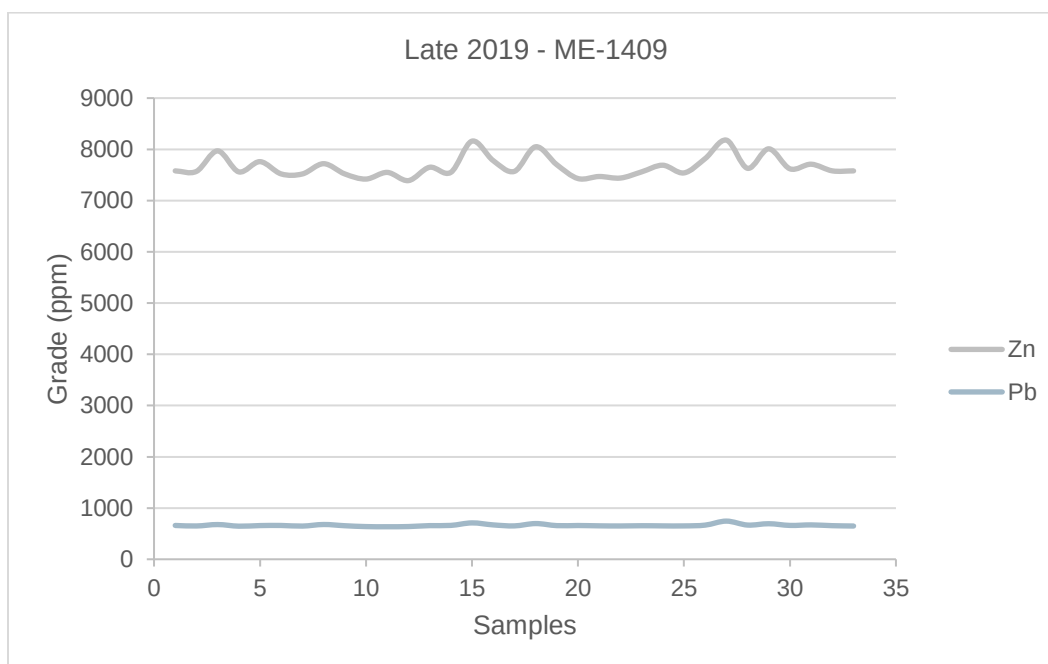


Figure 11-38: Standard ME-1409 late 2019 drilling results

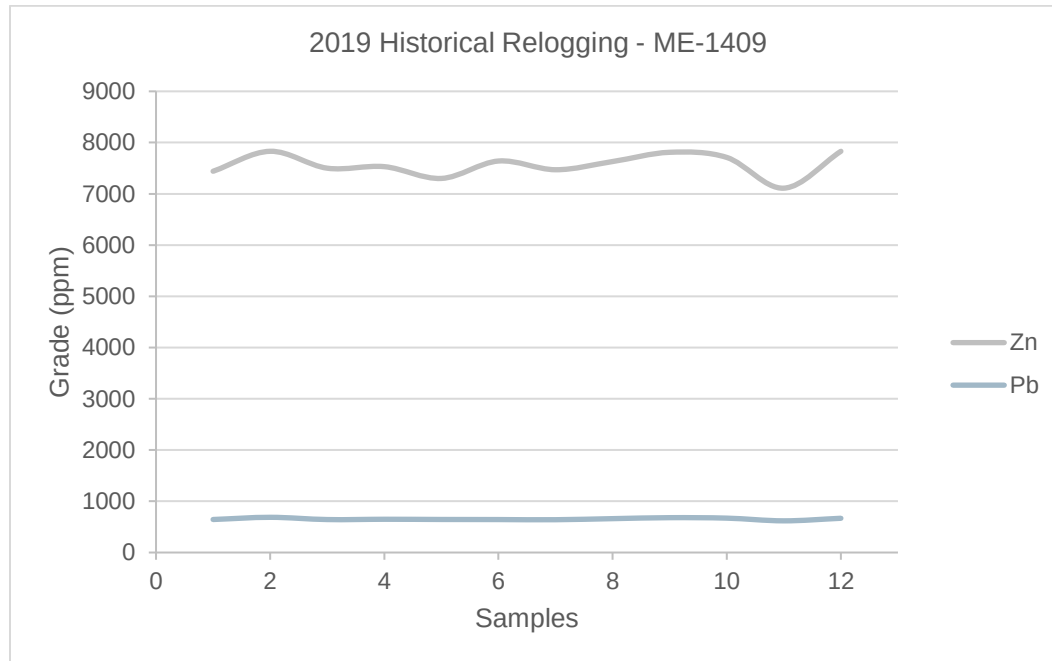


Figure 11-39: Standard ME-1409 2019 historical relogging results

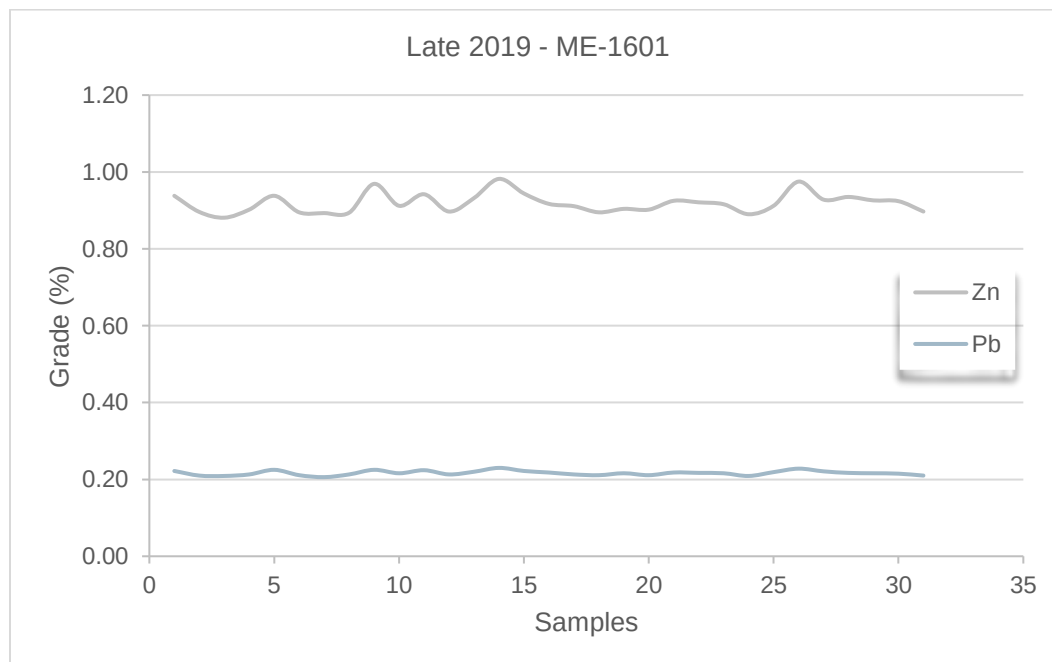


Figure 11-40: Standard ME-1601 late 2019 drilling results

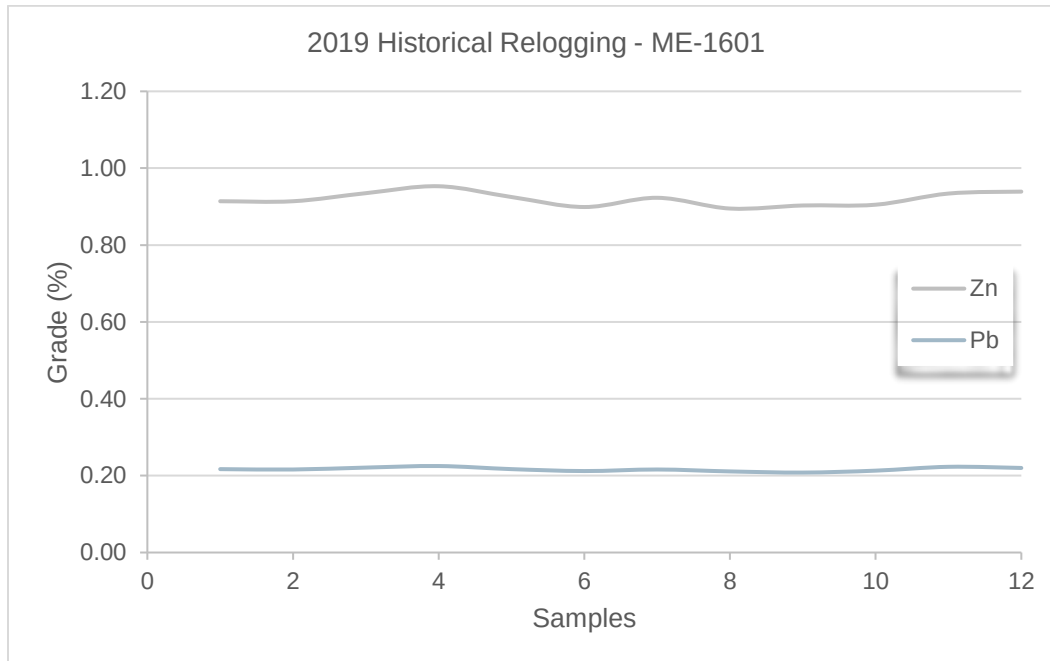


Figure 11-41: Standard ME-1601 2019 historical relogging results

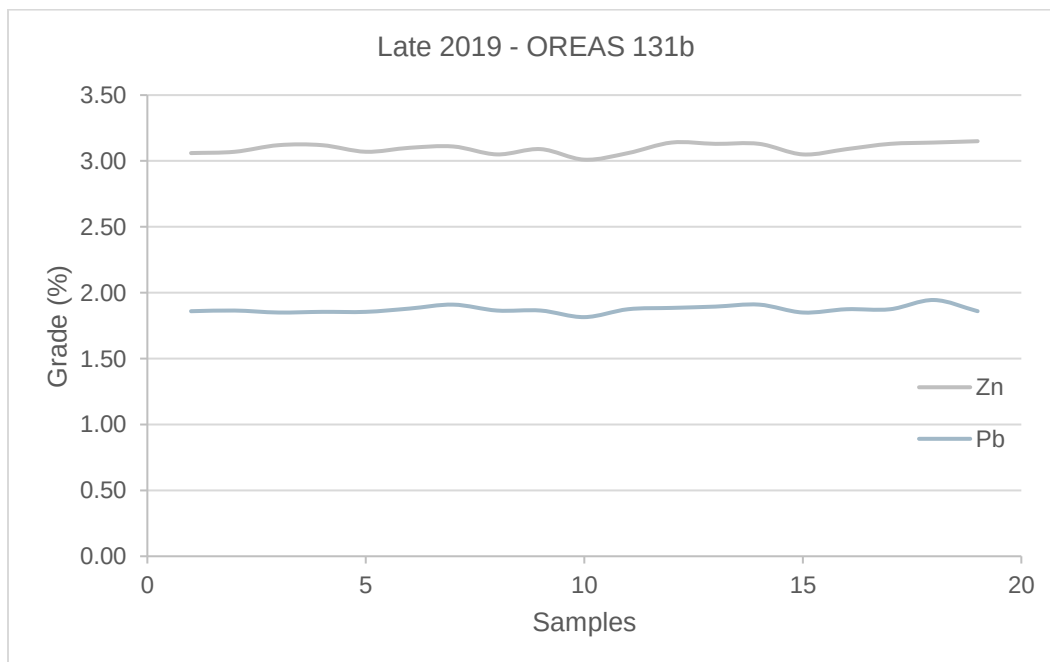


Figure 11-42: Standard OREAS 131b late 2019 drilling results

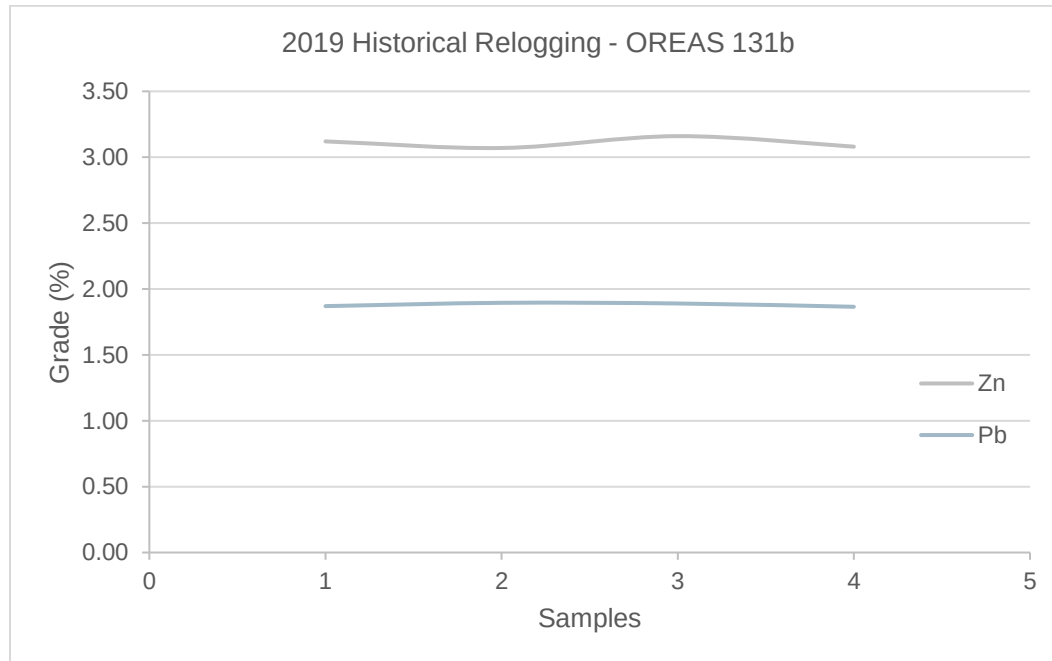


Figure 11-43: Standard OREAS 131b 2019 historical relogging results

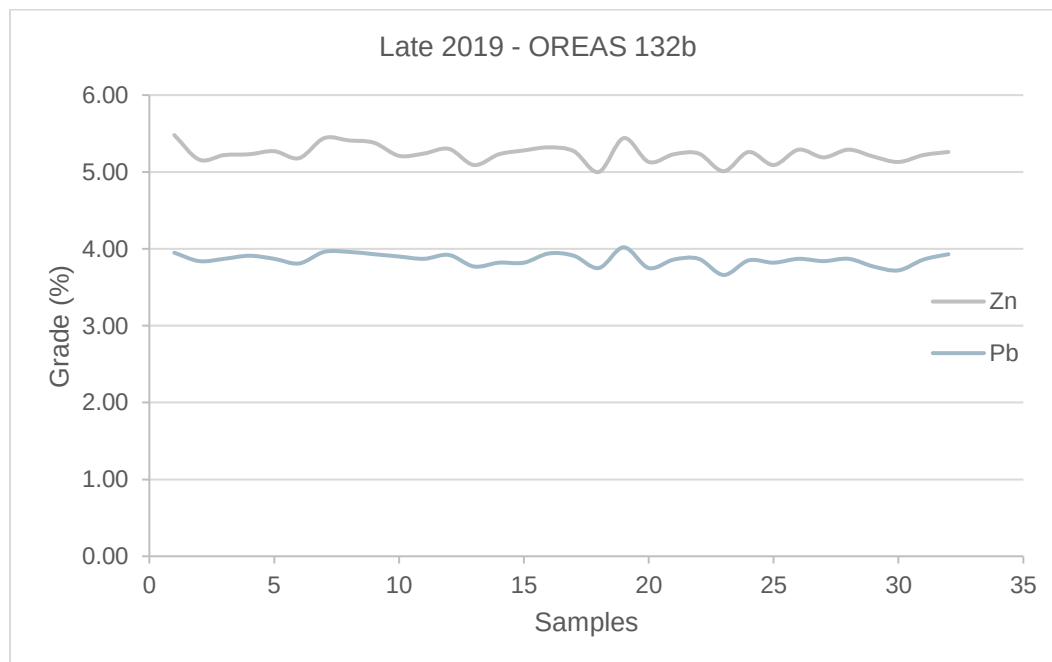


Figure 11-44: Standard OREAS 132b late 2019 drilling results

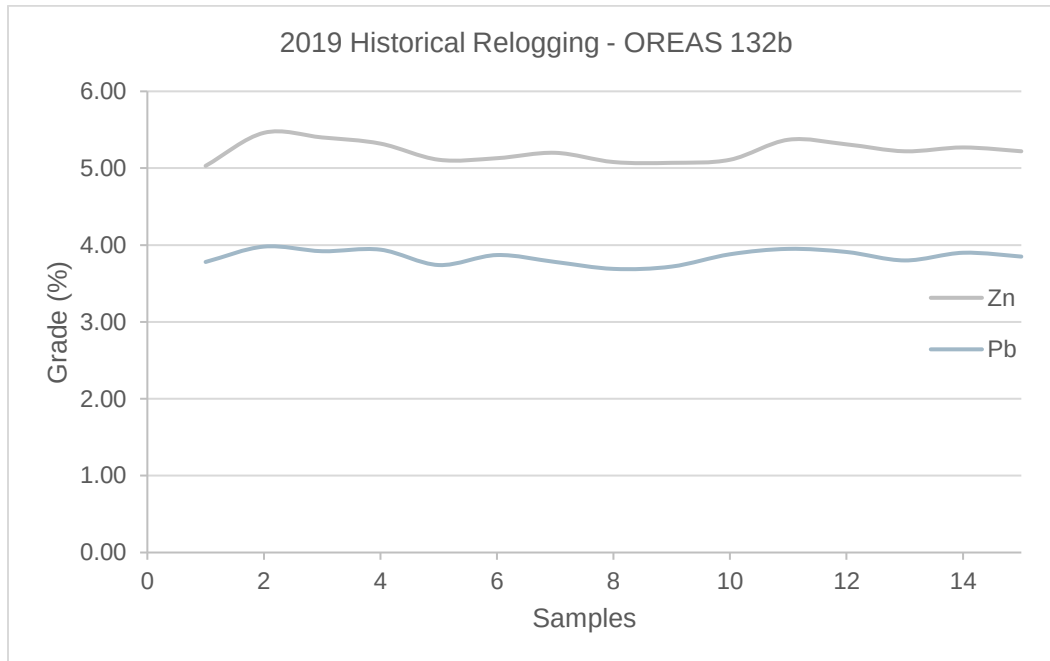


Figure 11-45: Standard OREAS 132b 2019 historical relogging results

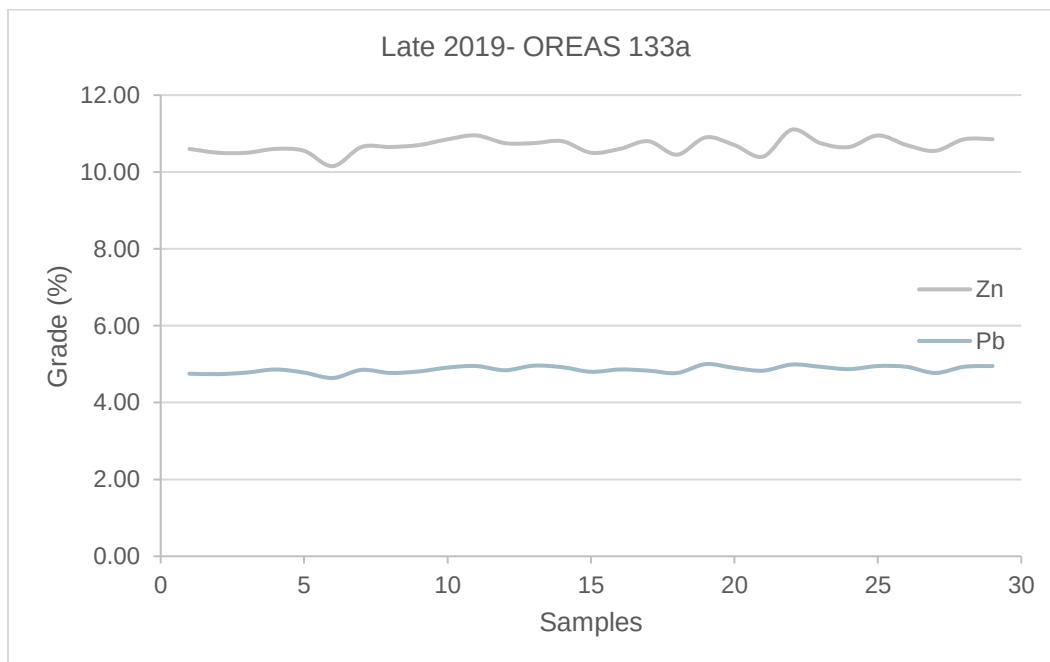


Figure 11-46: Standard OREAS 133a late 2019 drilling results

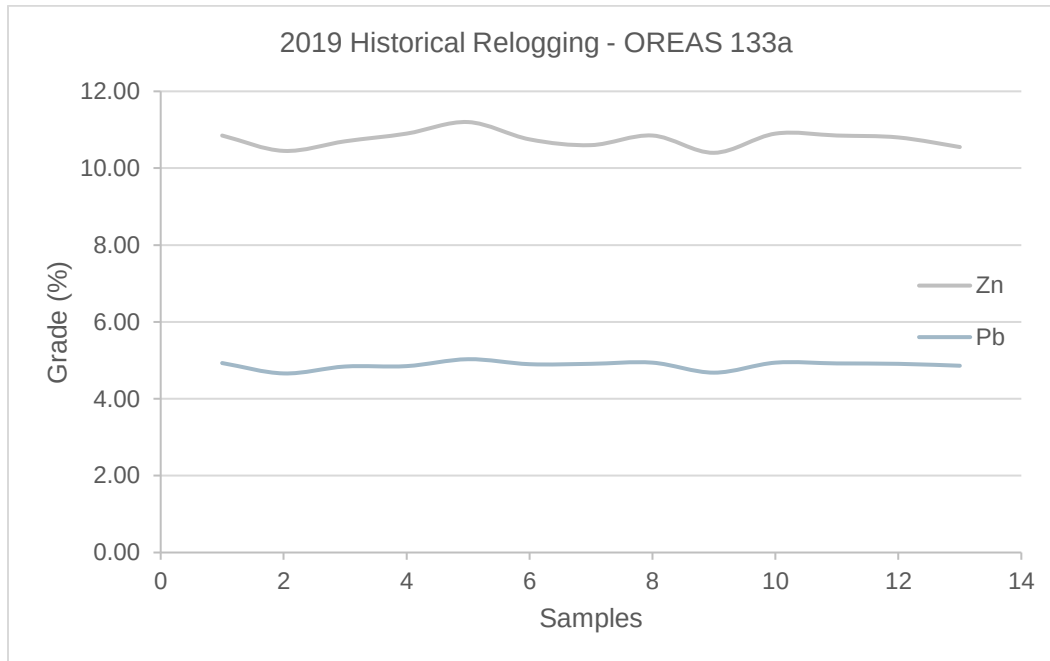


Figure 11-47: Standard OREAS 133a 2019 historical relogging results

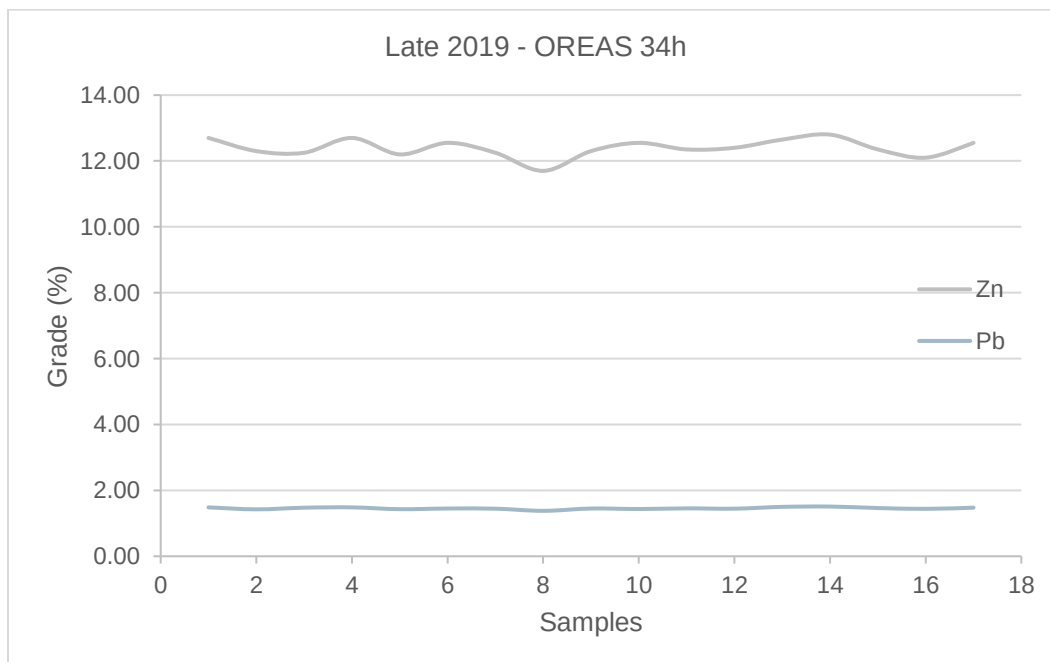


Figure 11-48: Standard OREAS 34h late 2019 drilling results

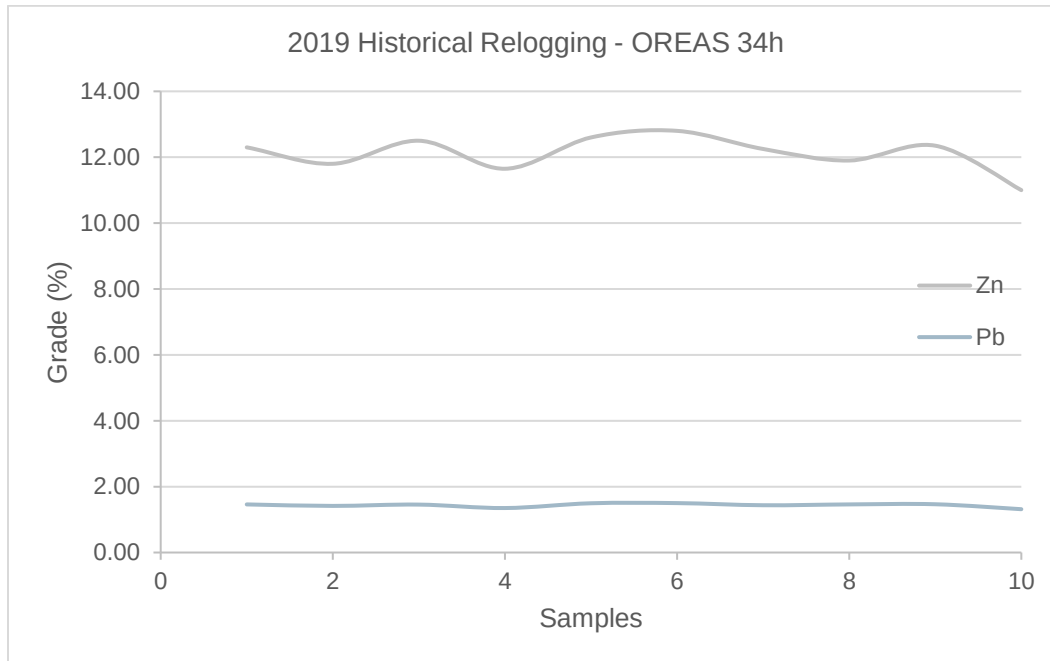


Figure 11-49: Standard OREAS 34h 2019 historical relogging results

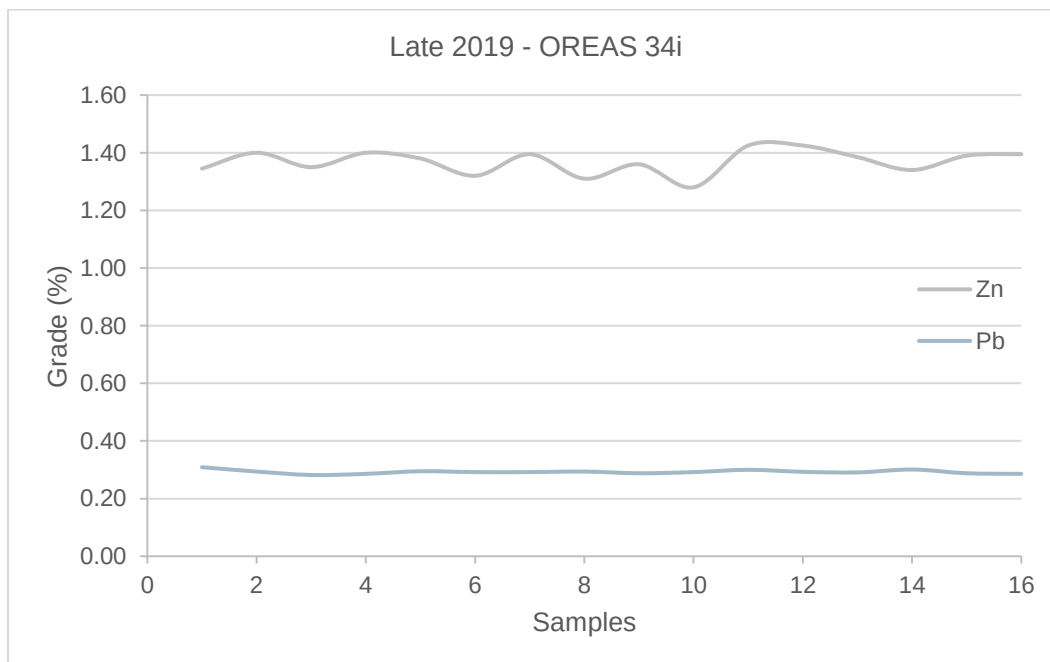


Figure 11-50: Standard OREAS 34i late 2019 drilling results

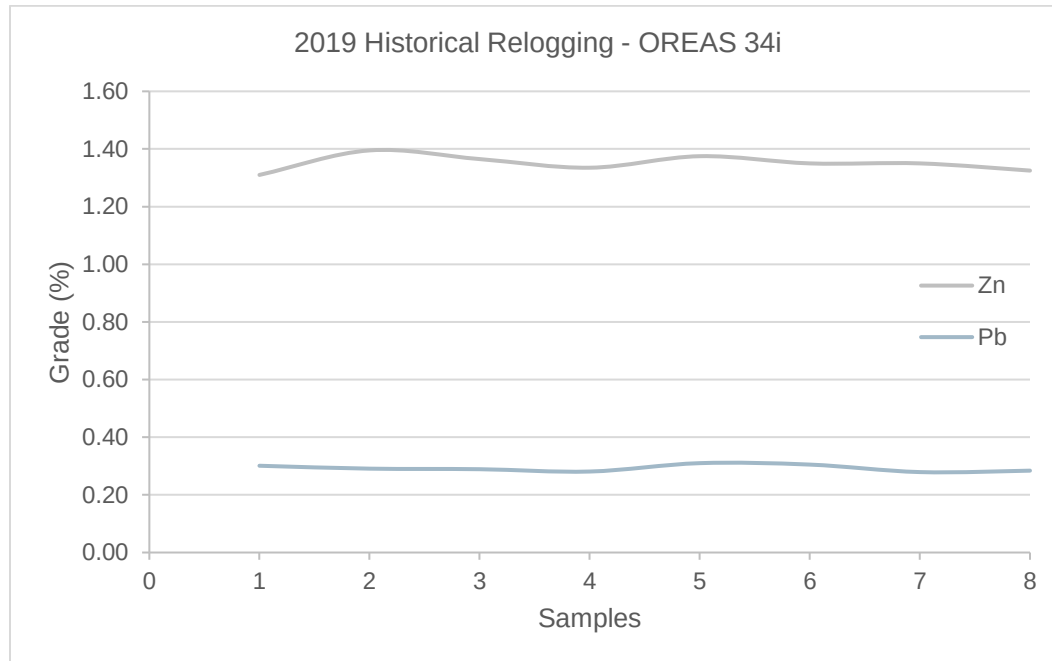


Figure 11-51: Standard OREAS 34i 2019 historical relogging results

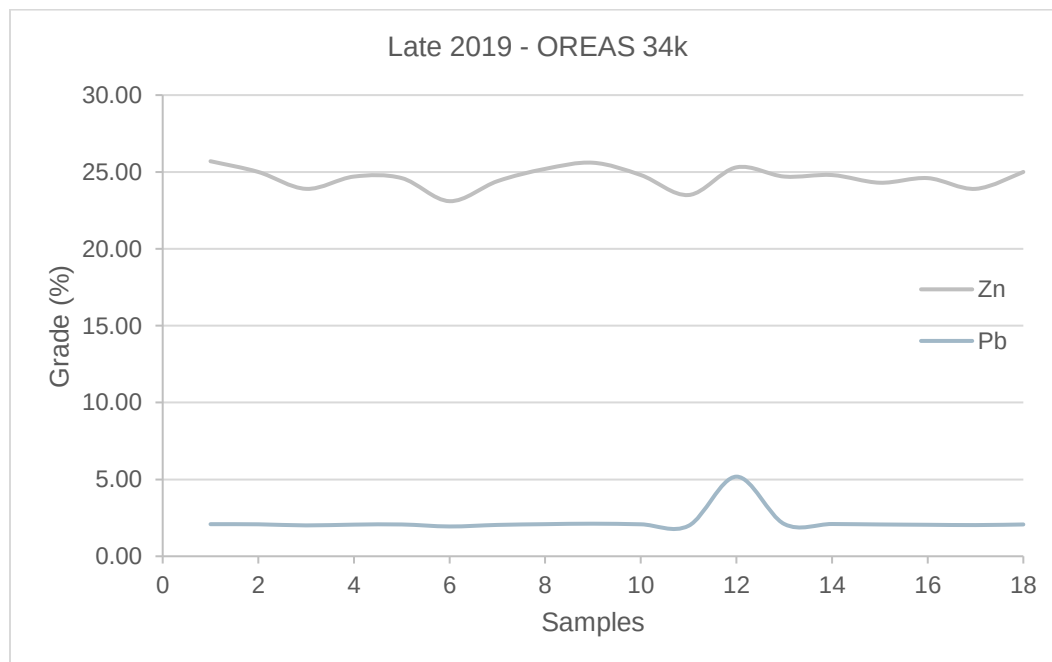


Figure 11-52: Standard OREAS 34k late 2019 drilling results

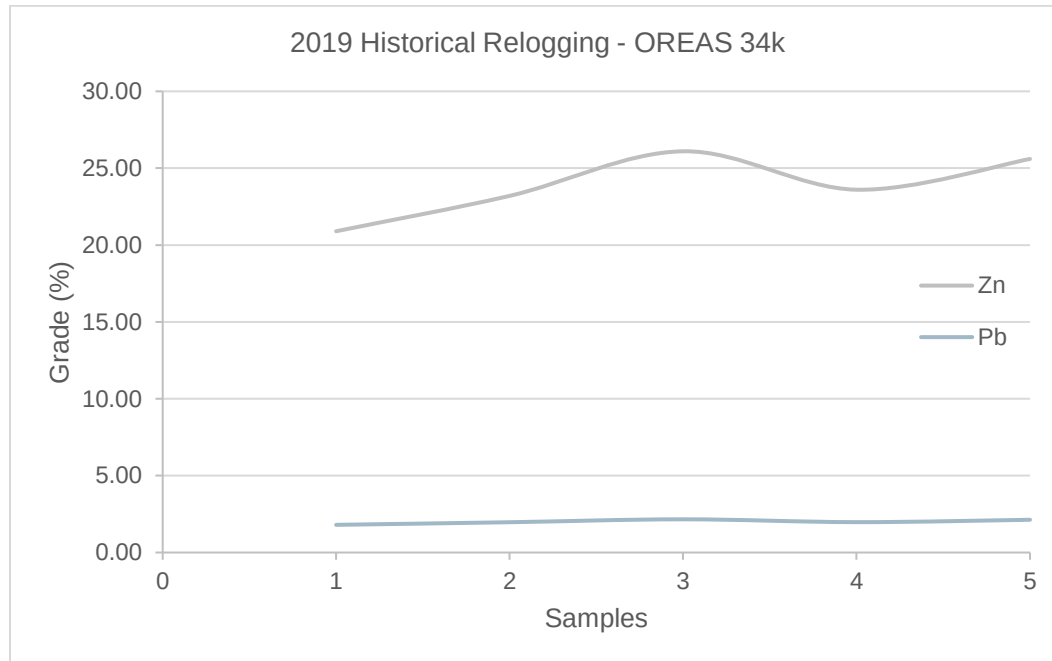


Figure 11-53: Standard OREAS 34k 2019 historical relogging results

11.4 Assessment of Results

The following procedure assesses the QA/QC program during the drilling program to identify immediate issues. If they do exist, they are immediately addressed:

- The Chief Geologist assesses the analytical results as they are received and communicates results to the Geological Team;
- The results are 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 depends on the kind of failure observed or what type of other issues are observed. Some of the follow up actions 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 includes drillhole ID, sample number, certificate number for each line of quality control data and 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, 2018, Winter 2019, and late 2019 drilling programs (the data used for the current MRE) 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 ("MRE") in this report is based on drill data from several eras of drilling at the Pine Point Mine Site including the Cominco Ltd. era holes 1930 to 1986, Westmin Resources Ltd. era holes 1975 to 1984, and the current programs of Pine Point Mining Limited ("PPML") from 2017 to December 31, 2019.

For the purpose of this MRE, the qualified persons ("QPs") performed a basic validation on the entire database. All data were provided by PPML in UTM NAD 83 Zone 11. The database close-out date for the resource estimate is December 31, 2019.

PPML has done significant work verifying historical hole locations and validating/correcting the digital drillhole database through differential GPS surveys and LiDAR surveys. These activities are documented in Chapter 10 (Drilling).

The Pine Point database contains 20,223 surface diamond drillholes. Of these 20,223 drillholes, a subset of 7,852 holes cutting across the mineralized zones were used for the current MRE.

12.1 Site Visit

Pierre-Luc Richard, QP, of PLR Resources Inc., visited the Pine Point Project from August 9 to 12, 2018, from October 18 to 20, 2019, and more recently during the course of this mandate from October 28 to 31, 2021. The site visits included a visual inspection of historical core and core drilling in progress, field tours (Figure 12-1), resampling, and discussions of the geological interpretations with geologists and engineers of Osisko Metals.



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

Selected drill collars in the field were also validated. The site visit also included a review of sampling and assays procedures, the Quality Assurance / Quality Control ("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 that said procedures were adequately applied.

The QP 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 Mr. Richard's site visits (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 one of the site visits and drill collar review



Figure 12-4: Sample preparation room and samples ready for shipment to the laboratory

12.3 Recent Drillhole Database

12.3.1 Assays

The QP was granted access to the original assay certificates for all holes drilled from Osisko Metals included in the current MRE. 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 2022 all drill collars have been surveyed using differential GPS equipment. Random field checks with handheld GPS units and comparison to plotted locations on air photos confirmed holes are in their planned locations.

12.3.3 Downhole Survey

For the 2017-2022 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 Independent Resampling Program

A total of 18 drillhole intervals were resampled by the QP during Mr. Richard's first site visit. These were selected and brought to Yellowknife by the QP where they were shipped to the Laboratory. Table 12-1 shows the results of this program. The reader should be advised that the point of such a resampling exercise is to validate the order of magnitude of the original database and confirm the presence of mineralization.

Table 12-1: Independent re-sampling program results

DDH	From	To	Sample						
				Zn (%)	Pb (%)	Fe (%)	Zn (%)	Pb (%)	Zn (%)
EM-18-PP-319	44.50	45.00	Y664029	2.64	0.57	0.30	4.19	0.66	0.31
EM-18-PP-321	30.87	31.87	Y664030	0.36	0.11	0.47	0.34	0.09	0.42
W85-18-PP-010	70.15	70.92	Y664027	4.43	3.37	0.32	3.79	3.79	0.35
W85-18-PP-010	71.70	72.70	Y664028	3.62	0.40	0.21	3.43	0.63	0.21
W85-18-PP-011	38.14	38.76	Y664041	14.20	12.95	1.07	29.20	18.85	1.74
W85-18-PP-011	59.00	59.66	Y664039	32.65	38.86	2.16	27.83	46.07	1.79
W85-18-PP-011	59.66	60.30	Y664040	7.33	4.53	1.41	20.70	6.92	2.10
W85-18-PP-012	81.96	82.93	Y664024	18.75	6.33	0.92	20.30	6.92	1.00
W85-18-PP-012	87.35	88.01	Y664025	8.10	2.28	0.97	9.29	5.29	1.01
W85-18-PP-012	96.69	97.40	Y664026	27.80	16.55	1.24	25.90	13.35	1.23
X65-19-PP-051	32.07	32.72	Y664038	1.78	0.17	0.24	2.23	0.10	0.23
X65-19-PP-065	21.85	22.85	Y664031	1.19	0.33	5.35	0.73	0.08	3.47
X65-19-PP-065	41.00	42.00	Y664032	2.74	0.86	0.34	3.61	0.98	0.39



DDH	From	To	Sample						
				Zn (%)	Pb (%)	Fe (%)	Zn (%)	Pb (%)	Zn (%)
X65-19-PP-066	19.71	20.62	Y664033	8.47	0.37	1.07	0.28	0.01	1.31
X65-19-PP-066	25.00	25.75	Y664034	3.15	1.14	4.56	2.43	2.44	3.57
X65-19-PP-070	26.70	27.70	Y664035	2.30	0.40	0.65	2.74	0.52	0.76
X65-19-PP-070	27.70	28.70	Y664036	4.39	0.83	0.50	6.10	0.60	0.56
X65-19-PP-070	28.70	29.70	Y664037	2.89	1.58	0.37	4.15	0.36	0.41

12.3.5 QA/QC

QA/QC reports were reviewed and did not yield issues. Some out-of-range lengths in some assays were detected. More details are available in Chapter 11 (Sample Preparation, Analyses, and Security) of this report.

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, the QP 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 MRE.

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

Chapter 10 provides details on the work to validate historical drillhole locations. In summary, Sub-Arctic Geomatics, out of Yellowknife, was contracted to locate and determine the absolute positions of a subset of 18,000 historic drillholes in the Pine Point mining area over in-situ mineralization in historical Cominco Ltd. unmined deposit areas. Further, LiDAR and Pit Bathymetry are ongoing, but 2018 surveys have been utilized for elevation correction and resource geometry cut-off around former open-pits.

Westmin Resources Ltd. 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 for a small sub-set were verified by the Sub-Arctic field work in 2018 and further holes were surveyed in 2019. Twin holes and confirmation drilling was also conducted in 2019.

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, the QP, 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: 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 Relogging and Sampling

Once a list of required drillholes had been submitted by the QP, 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 Ltd., 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 12-2 presents the results from the preliminary re-sampling program conducted as part of this study.

Table 12-2: 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 was initiated afterwards to recover core (see Chapter 10). Although there must be a mutual understanding that some holes are partially disturbed or completely sampled.

12.4.3 Unsampled Historical Drillholes

A significant amount of historical drillholes are completely unsampled in the database. Investigation into the matter led to indications that these might have encountered mineralization but were not assayed by the project operator (Cominco Ltd.). A twining program was initiated in 2019 on a selection of these unsampled historical drillholes to provide new core and validate this hypothesis.

12.4.3.1 Twining Program

Laboratory results for 13 twin holes are shown in Table 12-3. Significant mineralized intervals were encountered for 12 out of the 13 twin holes. Although not materially, the QP believes that these mineralized intervals are significant enough to improve the grade, and perhaps the tonnage, during a future MRE update since unsampled historical holes were assigned a grade of 0% Zn and 0% Pb for the current MRE.

Table 12-3: Results from the 2019 twining program

DDH	From	To	Length	Significant Intercepts		
				Zn (%)	Pb (%)	Zn + Pb (%)
4105-PP	22.00	25.00	3.00	2.02	0.20	2.22
	30.00	30.70	0.70	3.07	0.23	3.30
	35.70	36.20	0.50	2.87	0.05	2.92
4266-PP	6.00	7.90	1.90	0.41	0.52	0.93
	9.50	12.00	2.50	2.17	0.60	2.77



DDH	From	To	Length	Significant Intercepts		
				Zn (%)	Pb (%)	Zn + Pb (%)
K-32-110-PP	11.50	12.64	1.14	2.64	0.00	2.65
L-27-110-PP	17.90	18.30	0.40	3.16	1.03	4.19
L-35-038-PP	27.56	29.56	2.00	0.04	0.63	0.67
	33.20	39.50	6.30	3.18	0.23	3.41
	43.32	45.32	2.00	1.44	0.11	1.55
L-36-314-PP	34.56	35.40	0.84	0.72	1.86	2.58
	42.26	45.00	2.74	7.64	3.11	10.74
M-62-073-PP	53.30	54.60	1.30	0.48	0.97	1.44
M67-02-PP	65.00	66.50	1.50	1.44	0.54	1.97
	74.10	79.75	5.65	0.39	0.29	0.67
N204-267-PP	16.00	26.00	10.00	0.66	0.30	0.96
	41.60	48.00	6.40	1.56	0.37	1.93
N204-292-PP	18.00	20.00	2.00	0.89	0.17	1.07
	24.00	27.00	3.00	0.47	0.07	0.54
	38.00	42.00	4.00	1.02	0.15	1.18
	49.00	51.00	2.00	0.49	0.19	0.68
N32-092-PP	15.60	17.90	2.30	1.39	0.04	1.44
	19.90	23.00	3.10	0.62	0.37	0.99
X62-019-PP	42.10	48.30	6.20	0.51	0.21	0.73
	50.30	52.30	2.00	0.44	0.05	0.50
	56.30	58.30	2.00	0.38	0.05	0.43

12.5 Conclusion

The QP 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 current 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. 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. Subsequent to the 2020 Preliminary Economic Assessment ("PEA"), a small flotation testwork program was completed in December 2020. The primary findings from the test program indicate that both zinc and lead recoveries from the N204 zone material will likely be lower than originally estimated. The revised recovery estimates for material from the N204 zone are summarized in Section 13.6.

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, hydrothermal dolomite and calcite.

13.3. Historic Operating Data

13.3.1. Historical Operations

Historical operations 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 X15, located to the east of Pine Point's land package at the time. 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 principal 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 flowsheet 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 included 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.

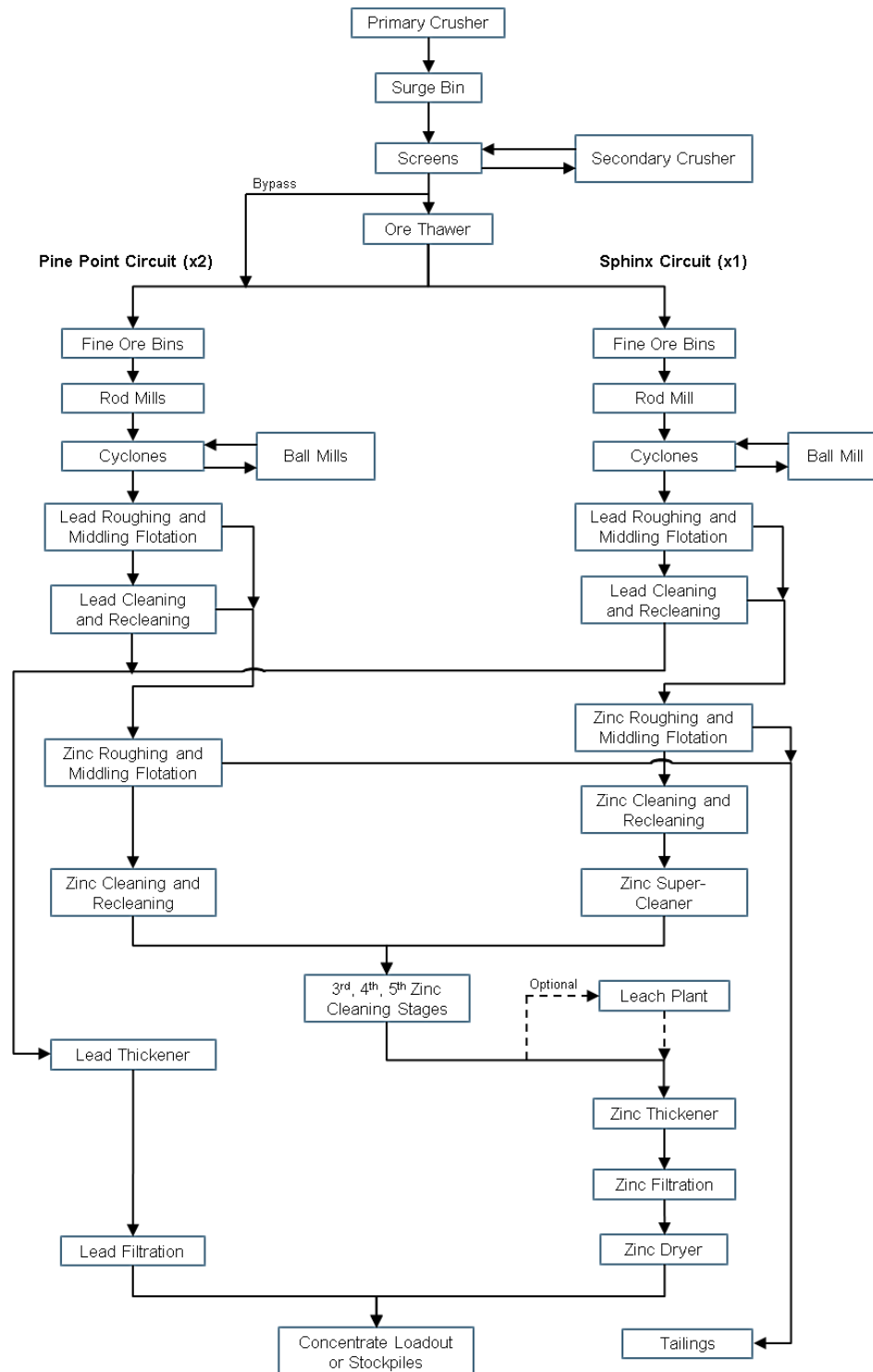


Figure 13-1: Historical process plant block diagram
 (Pickett et al., 1978)

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 $\frac{3}{4}$ " 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 X15 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 were 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, Na₂SO₃ (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 (Carmode, 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 Mining Ltd. (Cominco Ltd.) concentrator operated between 1964 and 1988, 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 Mst 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 plots 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.

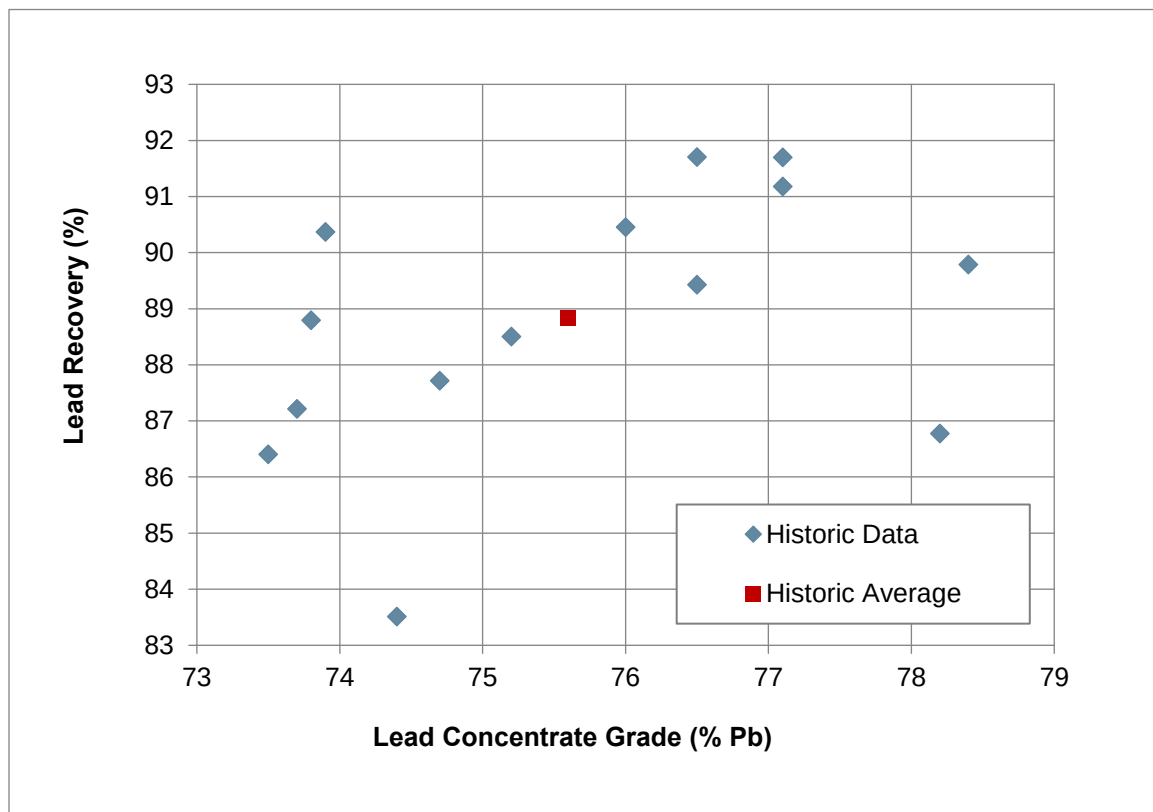


Figure 13-2: Historical lead recovery and concentrate grade (1974 to 1987)

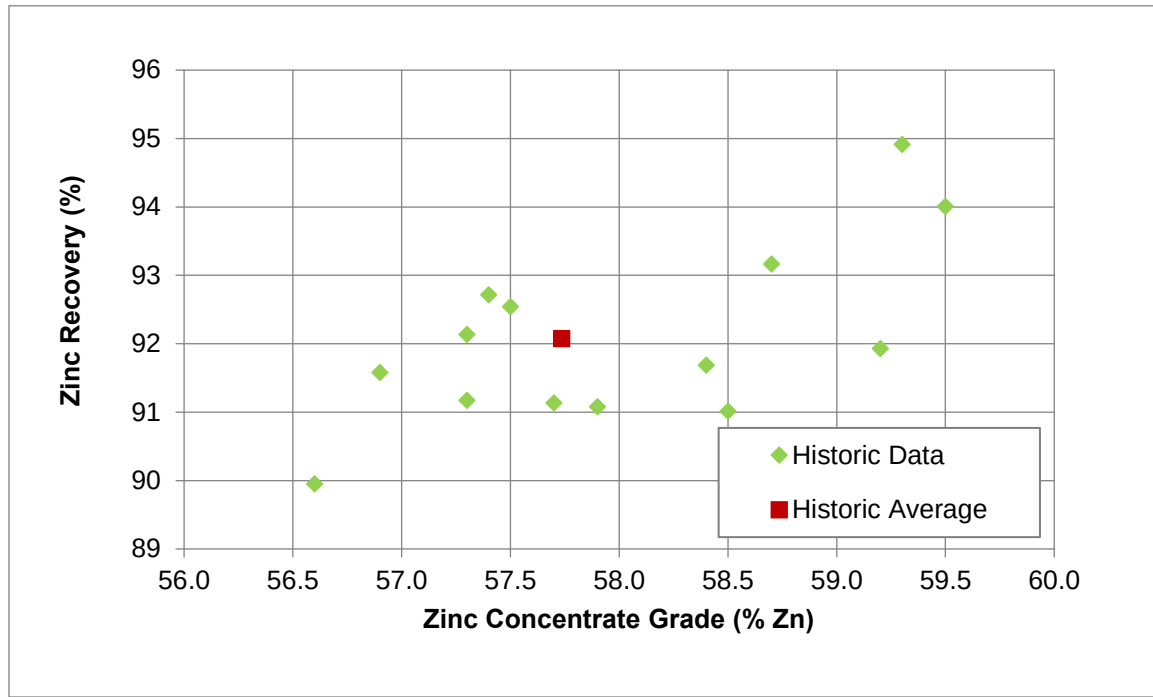


Figure 13-3: Historical zinc recovery and concentrate grade (1974 to 1987)

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.

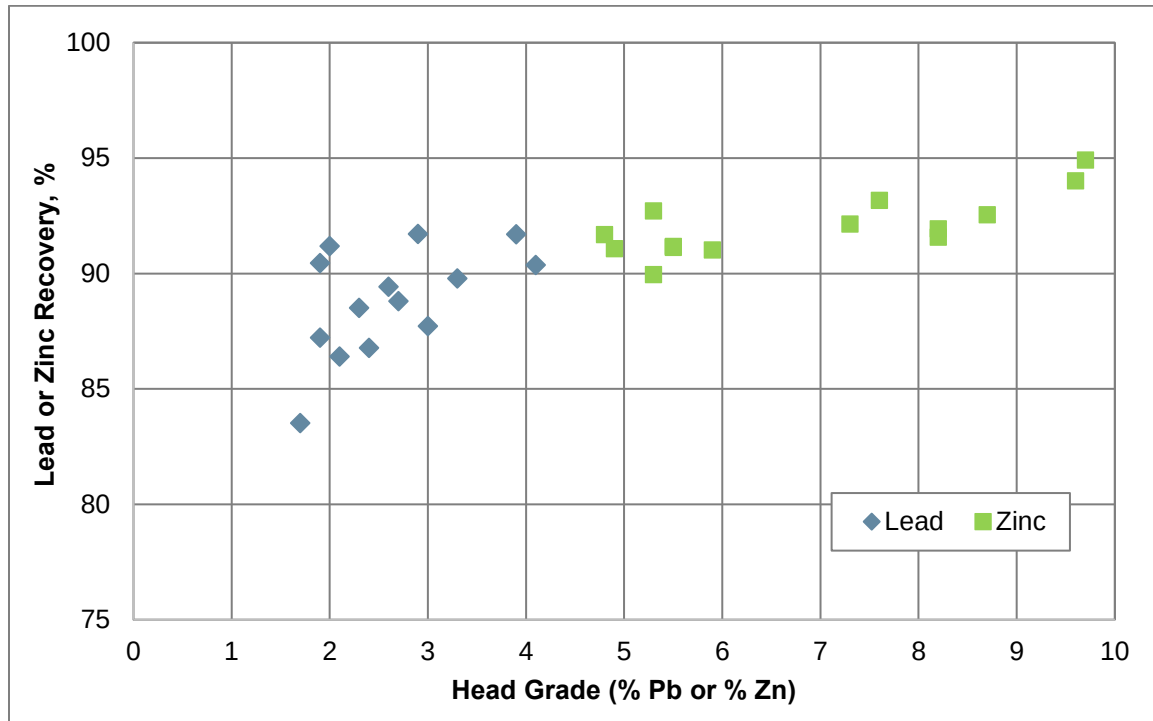


Figure 13-4: Historical head grade versus recovery relationship (1974 to 1987)

13.4. Past Metallurgical Testwork

13.4.1. Reference Reports

Numerous historical metallurgical test programs were conducted on samples from multiple deposits in the Pine Point District to support the various publicly 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 (“Tamerlane”), 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 R190 deposit and nine open-pit mines based on eight deposits in the centrally located Cluster Pit (“CP”) area as well as the N204 deposit area on the eastern extremity of the district.

The metallurgical characteristics of the N204 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 additional 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 (R190)

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 R190 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 only a minimal upgrade with a minor weight rejection to the float product. This result indicated that high-grade material may bypass the DMS

stage and report directly to the grinding 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 R190 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 R190 material to DSM quality. Although the material was upgraded significantly, these process options failed to improve the concentrate product grade.

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) mineral sorting step followed by conventional grinding and selective lead and 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 R190 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.

13.4.2.2. Phase 2 (O556 and Z155)

The second phase of process development work was performed on samples from two deposits adjacent to the R190 deposit, the O556 and the Z155 deposits. All available mineralized diamond drill core intersections from each deposit were combined into a composite sample for each deposit.

Comminution and Liberation Results

One bond ball mill work index test was performed on a composite of R190 (G&T, 2011). A work index of 7.4 kWh/t was measured, indicating a soft material relative to industry databases. 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 P₈₀. 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.

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 O556 and Z155 material.

The results of this bulk DMS test are shown in Table 13-2. The bulk DMS test confirmed that the Pine Point materials are amenable to mineral sorting 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 13-2: DMS test results for O556 / Z155

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)

Flotation

Flotation flowsheet design and reagent scheme development continued with the HLS mineral sorted 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 material from the O556 deposit contained a bituminous oil that 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 13-3.

Table 13-3: Locked cycle flotation test results for O556 / Z155

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 13-4.

Table 13-4: Proposed general reagent scheme

Process step	Flotation reagents (g/t of mill feed)							
	Lime	P92 ⁽¹⁾	SQ6 ⁽²⁾	Na ₂ S	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		
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 ZnSO₄ (66%), Na₂S₂O₅ (17%) and Oxalic Acid (17%).
2. SQ6 is composed of Acryloxymethyl 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 (N204)

The third phase of process development work (DMS and Flotation) was performed in 2011 on samples from the N204 deposit, which is located to the northeast of the Central Zone. The metallurgical characteristics of the N204 deposit, hosted in the B Spongy horizon at the base of the Pine Point Formation, differ from the deposits hosted in other formations higher up in Pine Point and Sulphur Point Formations, such as those found in the East Mill, Central, North and West Zones, in that it is finer grained mineralization. The head grade assay for the sample is shown in Table 13-5.

Table 13-5: N204 Composite head assays

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

Source: SGS (2011)

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 resulted 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 13-6.

Table 13-6: DMS Test results for N204

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)

Flotation

A limited flotation testing program was conducted on a Master Composite N204 (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 13-7.

Table 13-7: Locked cycle flotation test results for N204

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)

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 R190 deposit as an underground operation followed by open-pit mining of selected pits from the Central Zone. The metallurgical testwork program demonstrated that standard lead and zinc flotation, preceded by DMS, would yield recoveries above 90% and good concentrate grades. The proposed Tamerlane flowsheet is described in detail in the 2014 NI 43-101 summary report by Siega and Gann (Siega and Gann, 2014).

To reduce transportation and processing costs, Tamerlane proposed that the material be first processed at two crushing/DMS mineral sorting plants. One DMS plant was to be located near the Central Zone and the other near to the N204 deposit. The DMS concentrate from both plants would then be trucked to a centralized grinding/flotation plant (1,800 tpd capacity) at the R190 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, DMS, ball milling, pre-flotation to remove bitumen-activated slimes, and 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 13-8 and Table 13-9). It was assumed that the testwork performed on R190 material could be used to predict the overall metallurgy of the Central Zone material due to geological similarities. The recovery and grade predictions for N204 material were shown separately due to its differing mineralogy.

Table 13-8: Metallurgical projections (R190)

	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	-	-	85.0	88.1

Table 13-9: Metallurgical projections (N204)

	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	-	-	71.4	78.0

13.5. Metallurgical Testwork - Osisko Metals (2018 and 2019)

In mid-2018, Osisko Metals initiated a small-scale 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. In early 2019, Osisko Metals used the resulting mineral sorting concentrates and fine tailings as feed material for a flotation test program.

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.

These samples were used for the mineral sorting testwork and subsequent flotation testwork program described in the following sections.

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 were 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 13-10:

Table 13-10: 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%
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 zinc and lead feed grade on concentrate mass pull, zinc recovery and lead recovery.

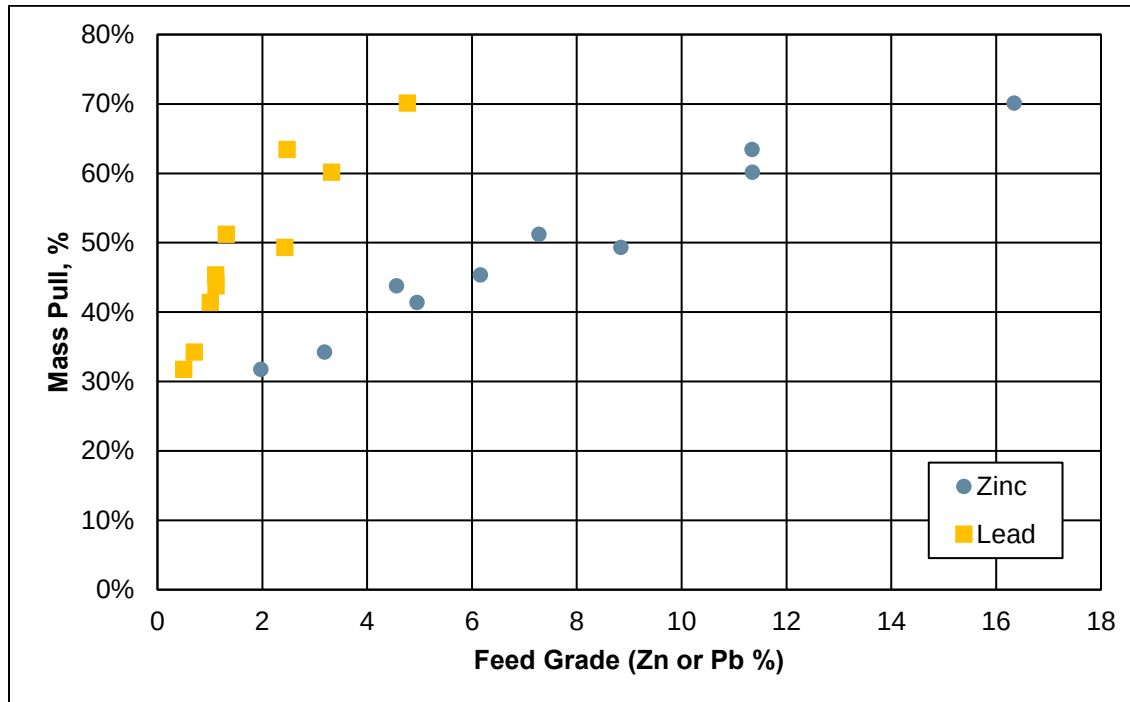


Figure 13-5: Feed grade versus mass pull to concentrate

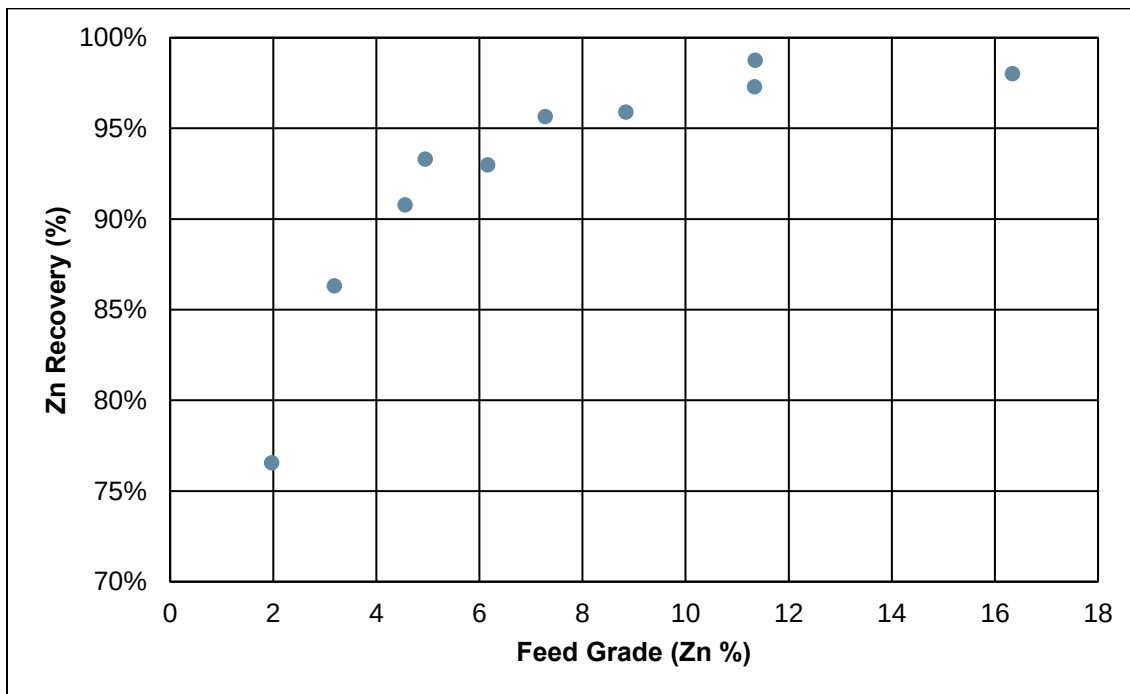


Figure 13-6: Zn feed grade versus recovery

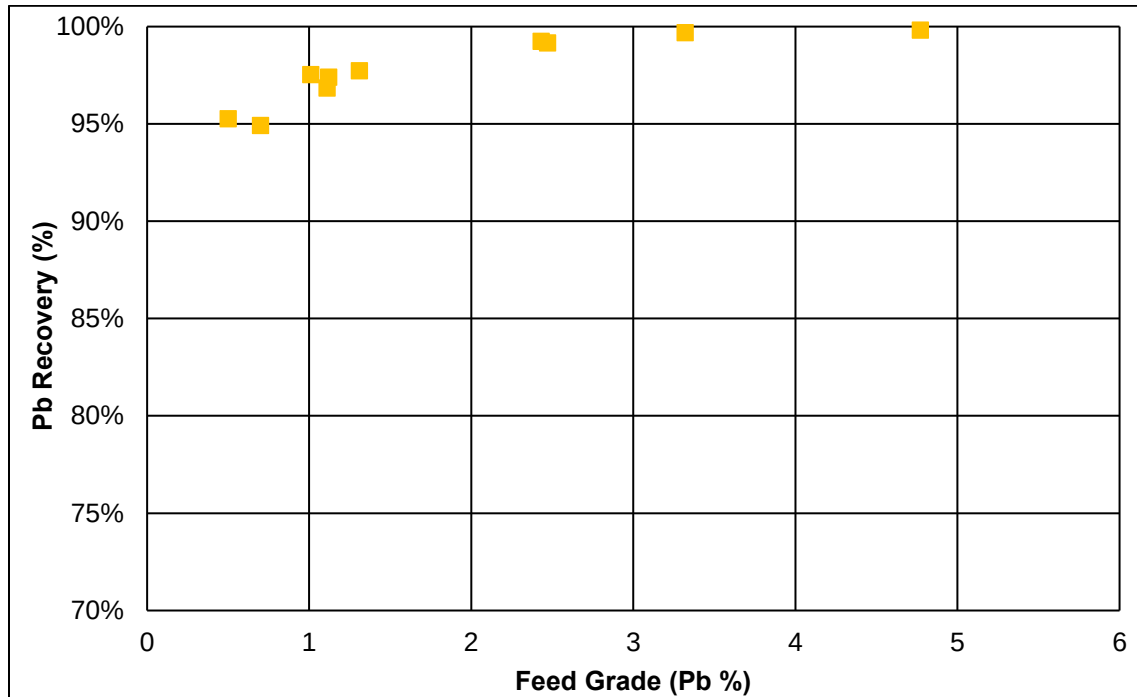


Figure 13-7: Pb feed grade versus recovery

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 and achieve recoveries of 98% for Lead 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

In early 2019, a bench scale flotation testwork program was initiated at SGS Canada (SGS, April 8, 2019) using fine tailings and concentrates produced by the 2018 material sorting evaluation program. A total of 20 flotation tests, including 15 rougher kinetic tests and five cleaner tests were completed on composites of LG, HG, HG-LG, MG, LOM, LOM-LG, HG-LOM, LOM-MIX and LOM-FINES.



Scoping Testwork

The standard rougher flotation test procedure involved grinding a 1 kg test charge at 50% solids in a laboratory rod mill to a target grind size (range: 65 µm to 100 µm) with lime, P92 (66% ZnSO₄, 17% Na₂S₂O₅ and 17% Oxalic Acid), SQ6 (60% CMC and 40% ETAA), and Na₂S added in the mill. After grinding, the density of the pulp was adjusted to 33% solids in a Denver D1 flotation cell. For lead, roughing PAX and 3419A were added, and for zinc, roughing SQ6, CuSO₄, PAX and 3894 were added. Air was introduced into the pulp and concentrates that were collected over specified time periods, in stages ranging from 3 to 6. For cleaner tests, no regrind was performed before cleaning, which was conducted in a 250 g cell using the same procedures. All products were filtered, dried, weighed and assayed. The following two tables (Table 13-11 and Table 13-12) summarize the results from the initial rougher and cleaning tests.

Table 13-11: Initial rougher flotation testwork results

Test	Test No.	Head grade		Pb Ro Concentrate			Pb Ro Conc Recovery		Zn Ro Concentrate			Zn Ro Conc Recovery	
		Pb (%)	Zn (%)	Mass (%)	Pb (%)	Zn (%)	Pb Rec (%)	Zn Rec (%)	Mass (%)	Pb (%)	Zn (%)	Pb Rec (%)	Zn Rec (%)
1	LG-F1	1.2	4.6	13.1	9.1	3.2	97.6	9.3	13.4	0.1	30.3	1.2	89.3
2	HG-F1	6.4	22.5	18.3	34.5	14.5	99.0	11.8	36.2	0.2	54.5	0.9	87.9
3	MG-F1	2.4	13.5	12.7	18.6	8.8	98.2	8.3	25.5	0.1	48.3	1.6	91.2
4	LOM-F1	2.2	9.1	11.2	19.7	6.1	98.2	7.5	19.4	0.1	43.2	1.2	91.5
5	LOM-F2	2.4	9.5	16.7	14.1	6.9	98.8	12.1	19.7	0.1	42.1	0.9	87.4
6	LOM-F3	2.2	9.4	18.9	11.8	7.1	98.8	14.3	20.4	0.1	39.1	0.9	85.2
7	LOM-F4	2.2	9.2	20.1	10.7	7.3	98.7	16.0	21.8	0.1	35.3	1.0	83.5
8	LOM MIX-F1	2.3	8.7	19.4	11.7	6.5	99.0	14.5	18.6	0.1	39.6	0.8	85.0
9	LOM FINES-F1	1.9	7.4	24.5	7.8	5.5	98.9	18.2	17.9	0.1	33.4	0.8	80.4
10	LOM-LG-F1	1.9	7.4	19.4	9.6	5.5	99.0	14.3	17.0	0.1	37.1	0.7	85.1
11	LOM-LG-F2	2.1	7.3	19.5	10.4	5.5	99.0	14.7	19.1	0.1	32.5	0.7	84.8
12	LOM-LG-F3	1.9	7.4	20.2	9.1	5.9	98.8	16.1	20.2	0.1	30.3	0.9	83.2
13	HG-LOM-F1	4.2	17.9	14.3	28.7	11.7	98.7	9.4	33.2	0.1	48.6	1.1	90.3
14	HG-LOM-F2	4.1	17.4	16.1	25.0	12.3	98.5	11.4	32.6	0.1	47.0	1.1	88.3
15	HG-LOM-F3	4.1	17.6	20.1	20.4	13.4	98.9	15.3	31.9	0.1	46.6	1.0	84.4
Minimum		1.2	4.6	11.2	7.8	3.2	97.6	7.5	13.4	0.1	30.3	0.7	80.4
Average		2.8	11.3	17.6	16.1	8.0	98.7	12.9	23.1	0.1	40.5	1.0	86.5
Maximum		6.4	22.5	24.5	34.5	14.5	99.0	18.2	36.2	0.2	54.5	1.6	91.5

Table 13-12: Initial cleaner flotation testwork results

Test	Test No.	Head grade		Pb 2nd Cleaner Concentrate			Pb 2nd Cleaner Recovery		Zn 2nd Cleaner Concentrate			Zn 2nd Cleaner Recovery	
		Pb (%)	Zn (%)	Mass (%)	Pb (%)	Zn (%)	Pb Rec (%)	Zn Rec (%)	Mass (%)	Pb (%)	Zn (%)	Pb Rec (%)	Zn Rec (%)
16	HG-LG-F1	5.2	17.5	8.0	62.0	9.1	94.4	4.1	23.1	0.1	63.6	0.6	83.9
17	MG-F2	2.4	13.8	3.5	65.0	4.1	95.4	1.1	20.2	0.1	62.0	1.0	90.8
18	LOM-F5	2.3	9.3	3.2	66.7	4.2	92.2	1.4	13.2	0.1	61.7	0.8	87.7
19	LOM-LG-F4	1.8	7.3	3.2	52.5	4.1	92.6	1.8	10.4	0.1	61.2	0.7	87.3
20	HG-LOM-F4	4.0	17.6	5.6	67.6	6.8	95.9	2.2	24.1	0.1	63.7	0.7	87.4
Minimum		1.8	7.3	3.2	52.5	4.1	92.2	1.1	10.4	0.1	61.2	0.6	83.9
Average		3.1	13.1	4.7	62.8	5.7	94.1	2.1	18.2	0.1	62.4	0.8	87.4
Maximum		5.2	17.6	8.0	67.6	9.1	95.9	4.1	24.1	0.1	63.7	1.0	90.8

All tested composites responded well to conventional flotation. The lead rougher recoveries ranged from 97.6% to 99.0%, while the zinc rougher recoveries ranged from 80.4% to 91.5% for the 15 tests. This initial testwork confirmed high recoveries for lead. An important measurement of Pb/Zn flotation separation efficiency is the loss of zinc units to the lead rougher concentrate. For the 15 tests, the zinc distribution in the lead rougher concentrate ranged from 7.5% to 18.2%.

For the five cleaner tests (2 stages), the lead recoveries ranged from 92.2% to 95.9% at a lead concentrate grade of 52.5% to 67.7%, while the zinc cleaner recoveries ranged from 83.9% to 90.8% at a zinc concentrate grade ranging from 61.2% to 63.7%. It should be noted that, typically, the final lead and zinc recoveries will be higher in locked cycle flotation testing.

Optimization Testwork

To optimize zinc recovery, three additional rougher tests following the standard testing procedure were performed using a simplified suite of reagents consisting of Lime, ZnSO₄, 3418A in the lead rougher and Lime, CuSO₄ and 3894 in the zinc rougher. The new composite – New Comp – was prepared by combining the remaining flotation test charges of LG, LG-MG, LOM, and MG samples to have a similar lead and zinc grade with LOM. The results of the three tests using different reagent dosages are shown in Table 13-13:

Table 13-13: Optimized rougher flotation testwork results

Test	Sample	Head		Pb Rougher				Zn Rougher			
		Grade (%)		Grade (%)		Recovery (%)		Grade (%)		Recovery (%)	
		Pb	Zn	Pb	Zn	Pb	Zn	Pb	Zn	Pb	Zn
1	New Comp -F1	1.9	9.5	38.9	4.7	96.7	2.3	0.2	46.7	2.0	97.1
2	New Comp -F2	2.2	9.6	33.6	6.2	97.4	4.1	0.1	49.9	1.2	95.3
3	New Comp -F3	1.9	9.4	31.0	5.6	96.7	3.5	0.2	48.6	1.7	95.8
	Minimum	1.9	9.4	31.0	4.7	96.7	2.3	0.1	46.7	1.2	95.3
	Average	2.0	9.5	34.5	5.5	97.0	3.3	0.2	48.4	1.6	96.0
	Maximum	2.2	9.6	38.9	6.2	97.4	4.1	0.2	49.9	2.0	97.1

For the three optimization tests, the lead rougher recoveries ranged from 96.7% to 97.4%, while the zinc rougher recoveries ranged from 95.3% to 97.1%. The zinc distribution in the lead rougher concentrate range improved significantly over the initial scoping tests and ranged from 2.3% to 4.1%. Overall zinc recovery improved without a significant impact on lead recovery.

Two additional cleaning tests, using a simplified suite of reagents, were performed to compare the impact of regrind on both the lead and zinc rougher concentrates. The results of the tests are shown in Table 13-14 below:

Table 13-14: Optimized cleaner flotation testwork results

Test	Sample	Head		Pb 2nd Cleaner				Zn 2nd Cleaner			
		Grade (%)		Grade (%)		Recovery (%)		Grade (%)		Recovery (%)	
		Pb	Zn	Pb	Zn	Pb	Zn	Pb	Zn	Pb	Zn
1	New Comp - F4 - Regrind (Zn: 35 µm, Pb: 20 µm)	1.9	9.3	67.9	4.2	91.4	1.2	0.2	63.6	1.3	94.5
2	New Comp - F5 - Not Regrind	1.9	9.3	72.1	3.0	87.4	0.7	0.3	63.6	2.0	93.1

Regrinding the lead rougher concentrate to 20 µm reduced the lead 2nd cleaner concentrate grade from 72.1% to 67.9% while increasing lead recovery from 87.4% to 91.4%. Regrinding the zinc rougher concentrate to 35 µm had no impact on the zinc concentrate grade (63.6% Zn) but increased the zinc recovery from 93.1% to 94.5%. Additional testing will be required to determine if the implementation of a regrind stage would result in improved recoveries and concentrate grades.

Zinc and Lead Concentrate Quality

Samples of the zinc 2nd cleaner concentrate and the lead 2nd cleaner concentrate were sent to SGS and ALS laboratories for further analysis to measure concentrations of potentially deleterious elements commonly associated with smelter or refinery penalties. Table 13-15 summarizes the main deleterious elements analyzed for the zinc concentrate.

Table 13-15: Zinc concentrate elemental analysis (ALS)

Element	Symbol	Unit	Reported Concentration
Arsenic	As	ppm	< 2*
Antimony	Sb	ppm	< 0.5*
Bismuth	Bi	ppm	< 0.1*
Cadium	Cd	ppm	864
Cobalt	Co	ppm	3
Copper+Lead	Cu+Pb	%	0.23
Fluorine	F	ppm	< 20*
Iron	Fe	%	2.6
Magnesium	MgO	%	0.36
Manganese	Mn	ppm	100
Mercury	Hg	ppm	0.31
Silica	SiO ₂	%	< 0.021*

* Indicates analytical result below detection limits.

Analysis of the zinc concentrate indicates that it is relatively clean and free of impurities except for Magnesium Oxide (0.36% MgO), which is at the current smelter penalty threshold (typical: 0.35% MgO).

Analysis of the lead concentrate indicates that it is free of impurities and any measurement elements are well below typical smelter penalty thresholds.

13.6. Recovery and Grade Projections

Preliminary processing recovery and grade projections were estimated for the Central, North, West and East Mill zones (Table 13-16) and the N204 deposit (Table 13-17). Mineral sorting recoveries of Zn and Pb are based on the recovery versus head grade relationships developed from the 2018 Mineral Sorting Test Program (Section 13.5.2), while the Zn and Pb flotation recoveries are based on recovery versus head grade relationships from the 2019 Testwork Program (Section 13.5.3) and historical production data. As mentioned in Section 13.1, the flotation recovery for N204 material has been revised based on a small testwork program completed in December 2020 and is now assumed to be 10% lower than the recoveries projected for Central, North, West and East Mill zones. Final concentrate grades of 60% Zn and 65% Pb were assumed.

Table 13-16: Recovery projections (Central, East Mill, North and West Zones)

Stage	Recovery (%) ⁽¹⁾	
	Pb	Zn
Feed	100	100
Mineral sorting (including fines bypass)	98.7 (99.1)	91.2 (93.9)
Flotation Concentrate	94.2	92.9
Overall Recovery	93.4	87.2

⁽¹⁾ Recovery estimates are based on a feed grade of 4.6% Zn and 1.9% Pb.

Table 13-17: Recovery projections (N204 Zone)

Stage	Recovery (%) ⁽¹⁾	
	Pb	Zn
Feed	100	100
Mineral sorting (including fines bypass)	97.2 (98.0)	87.6 (91.3)
Flotation Concentrate	82.5	85.0
Overall Recovery	80.9	77.6

⁽¹⁾ Recovery estimates are based on a feed grade of 3.4% Zn and 1% Pb.

Metallurgical projections, based on the mineral sorting and flotation testwork for the mineral resources as described in this report, indicate an overall lead and zinc recovery of 93% and 87%, respectively. The lead and zinc concentrate grades are projected to be 65% Pb and 60% Zn, respectively. It should be noted that these are preliminary estimates and additional confirmation testwork is planned.

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, West Zones, and N204) of the Pine Point Project, as well as to confirm the most economic process flowsheet:

- Pre-concentration:
 - Perform additional mineral sorting tests on composites that reflect the LOM plan;
 - Process technology selection trade-off study (Dense Media Separation vs Mineral Sorting).
- Crushing and grinding:
 - Complete ongoing comminution testwork program to assess deposit variability and to gather data for mill sizing and power calculation purposes.
- Flotation:
 - Investigate the impact of regrinding the rougher concentrate (zinc and lead);
 - Perform additional tests to understand the lower recoveries of N204 material;
 - Reagent evaluation and optimization;
 - Validate impact of slimes and bitumen;
 - Locked cycle tests.
- Concentrate characterization:
 - Validate the quality of the Pb and Zn concentrates for potential impurities that could lead to smelter penalties;
 - Identify methods to reduce magnesium oxide in the zinc concentrate.
- Dewatering testwork:
 - Gather sedimentation and filtration data of the concentrate and tailings for equipment sizing purposes.

14. MINERAL RESOURCE ESTIMATE

The Pine Point Mining Camp ("PPMC"), acquired by Osisko Metals, was discovered in 1898 and exploited from 1964 to 1988. During this period, Cominco Ltd. extracted around 64.3 Mt of material grading 7.0% Zn and 3.1% Pb from approximately 50 open-pits and two underground deposits.

The Mineral Resource Estimate ("MRE") for the Pine Point Lead-Zinc Project (the "Project" or the "Pine Point Project"), incorporates historical drilling data and recent drilling programs. Drillhole information up to December 31, 2019 was considered for this estimate.

This herein MRE does not include new drill holes. Economical parameters were updated to reflect new information since the previous MRE and metal prices were updated to better reflect recent changes. Updating economical parameters and metal prices had an impact on cut-off grades and pitshell shapes.

14.1 Methodology

The herein MRE covers the whole Pine Point Project with a strike length of approximately 67 km and a width of approximately 9 km, down to a vertical depth of 200 m below the surface. Figure 14-1 shows the Project along with the names of the different areas.

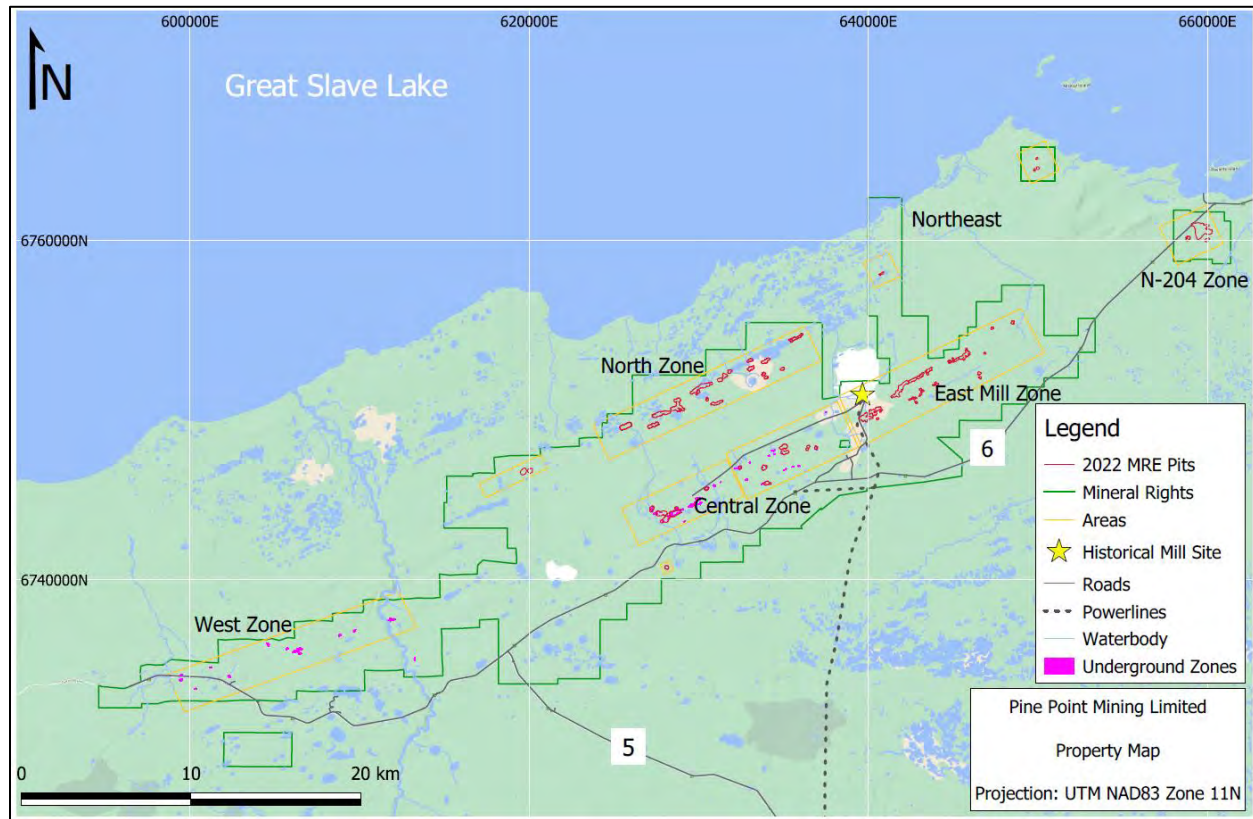


Figure 14-1: Overall plan view for the Pine Point Project

Leapfrog Geo™ v.4.5 was used for the modelling of 254 mineralized zones and for the generation of the drillhole intercepts for each solid. GEOVIA GEMS v.6.8.3 was used for the compositing, the 3D block modelling and for 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;
- Mineral 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 December 31, 2019, consisted of 19,509 surface drillholes with a cumulative length of 1,365,895 m (Figure 14-2). The average length of a drillhole is 70 m. Of these 19,509 drillholes, a subset of 7,852 holes cut across the mineralized zones with a total of 47,998 assays. The drillhole database includes recent drilling of 78,195 metres in 1,182 drillholes since 2017 and it also incorporates Cominco Ltd.'s historical drillholes, the use of which was partially validated by a drillhole collar survey, twinning programs, and a partial core resampling program. The database was validated by the QP.

Since the 2020 PEA (same database cut-off date), Pine Point Mining Limited ("PPML") drilling activities focused mainly on the conversion of Inferred Mineral Resources reported in the 2020 PEA to the Indicated category. Drilling has been focused on infill drilling and limited exploration. The objective of the infill drilling is to reduce the intercept spacing to approximately 30 metres within Inferred Mineral Resource areas reported in the 2020 PEA. To minimize the impact on the environment, drilling is mostly conducted in winter. The program is approximately 45% complete and the Company expects to complete infill drilling by the end of winter 2023.

Results for this partial drilling program were, in some cases, not available and deemed not material enough to justify remodeling of the entire mining camp before the entire program is completed. It is the QP's opinion that adding these holes to the model would not have had a material impact on the current MRE.

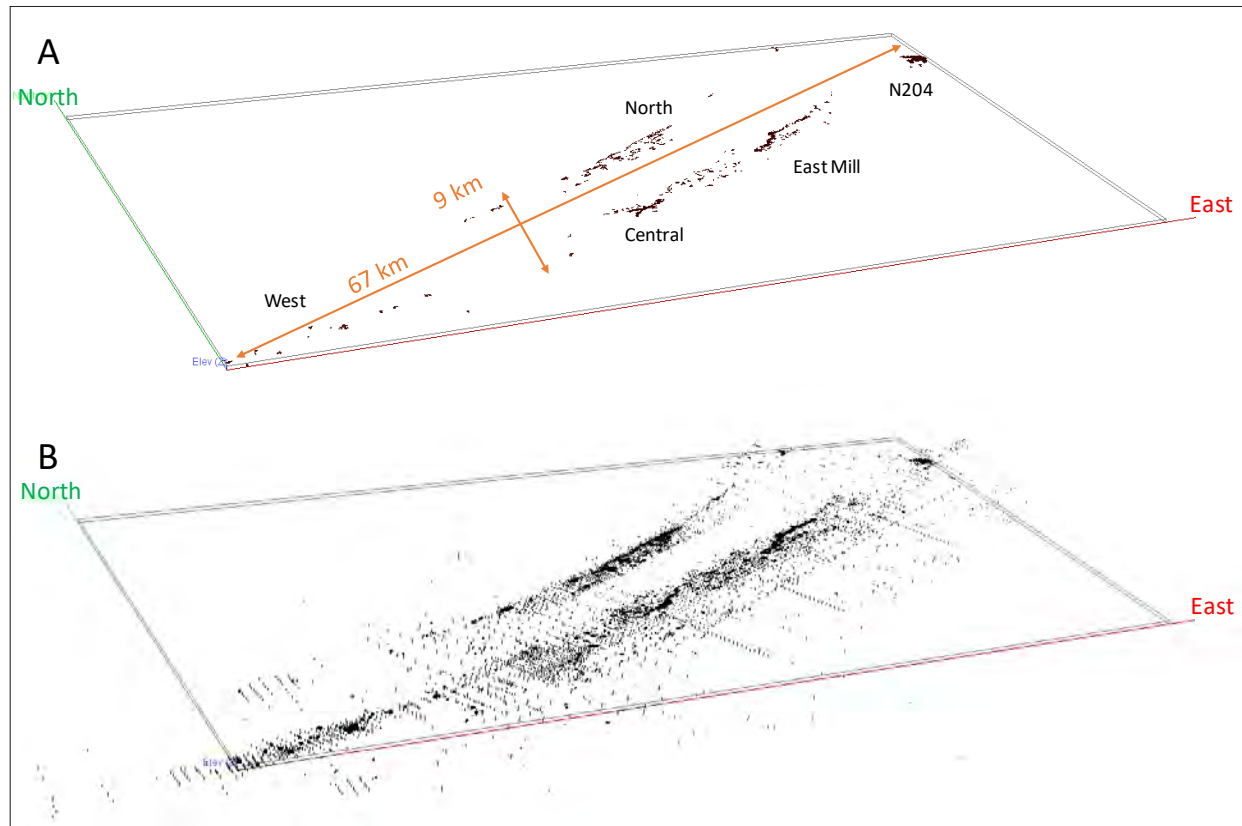


Figure 14-2: Overall plan view of the 3D model of the mineralized zones (A) and of the drillholes included in this mineral resource estimate (B)

14.3 Geological Model

Geological wireframes were constructed in Leapfrog Geo™. The model comprises 254 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 have lower grades than the Prismatic Zones;



- N204 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 in Geovia GEMST[™] and Leapfrog Geo[™] and is based on the drillholes collar coordinates and elevation as well as a LIDAR survey flown in 2019. Drillholes downhole overburden description was used for the overburden-rock interface. The mineralized zones were clipped to the overburden/bedrock interface when necessary.

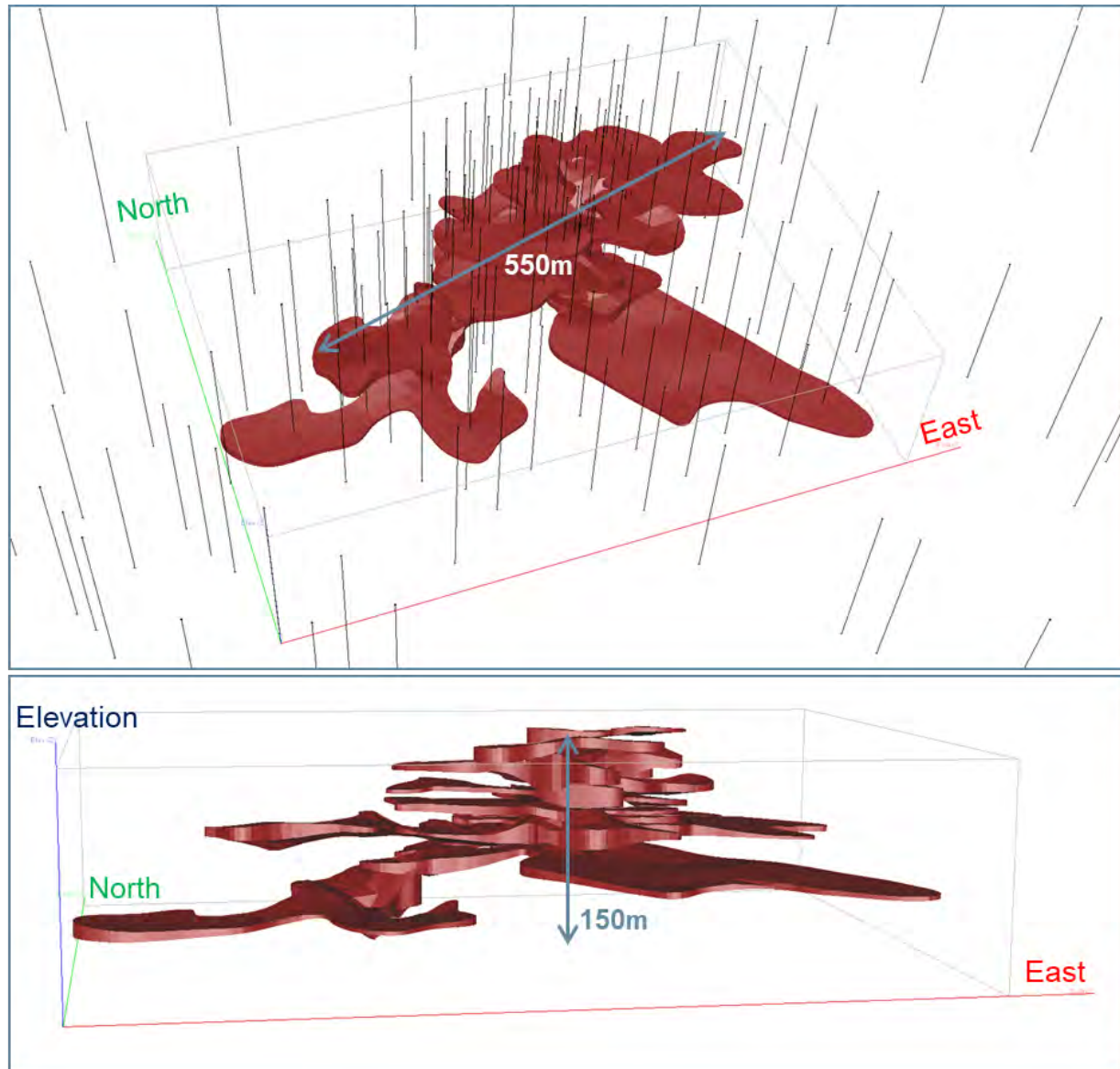


Figure 14-3: Example of Prismatic Zones in plan view (top) and in 3D view looking north (bottom)¹

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 mineral resources were classified as such without at least two drillholes being used and at least one drillhole within 50 m from each block.

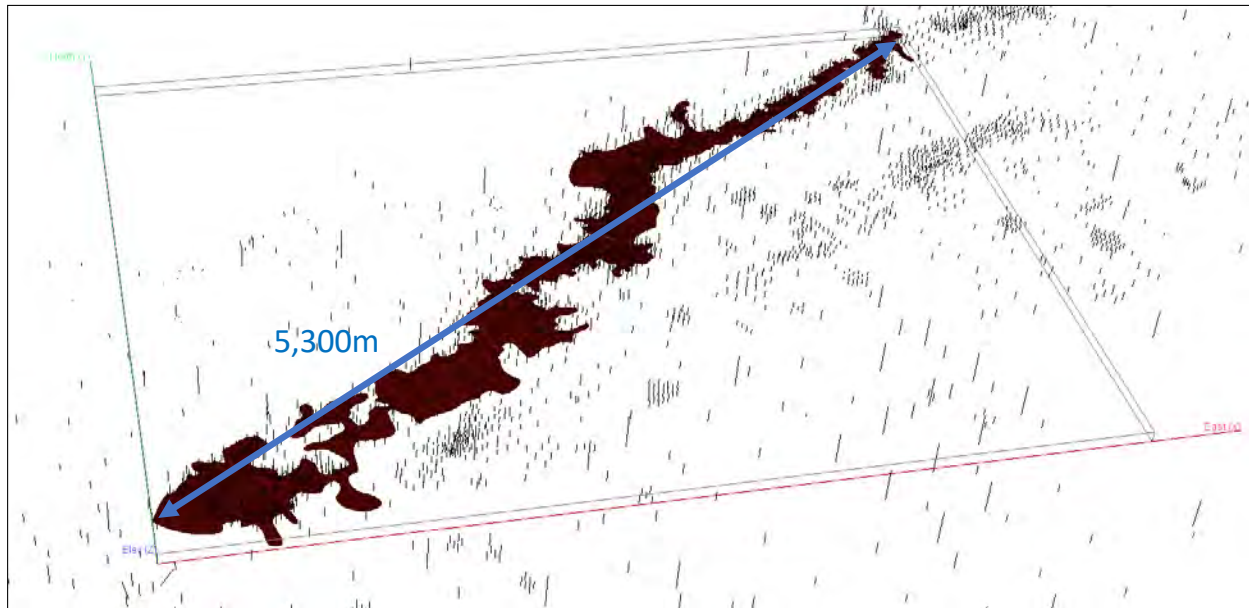


Figure 14-4: Example of a Tabular Zone²

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 mineral resources were classified as such without at least two drillholes being used and at least one drillhole within 50 m from each block.

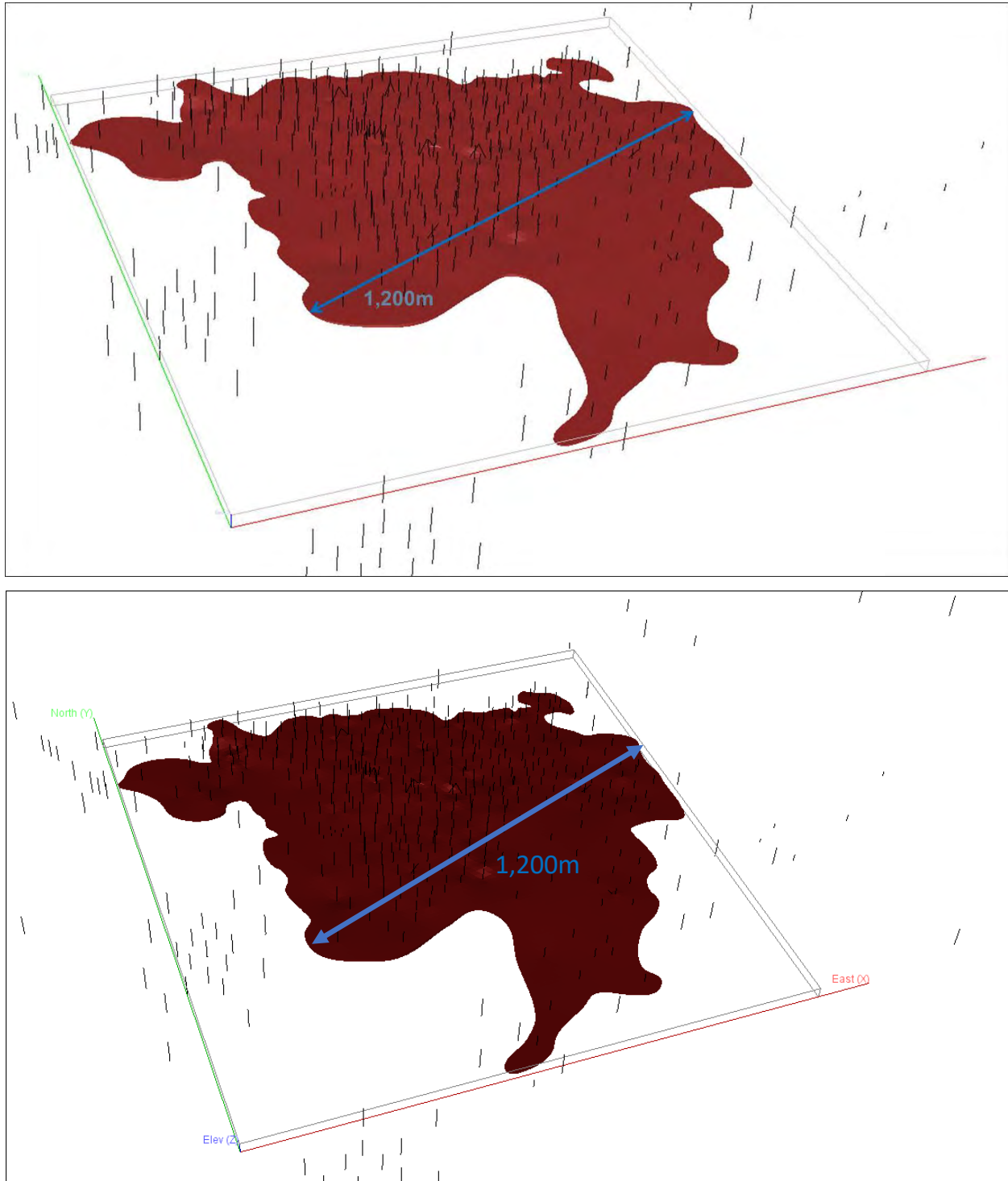


Figure 14-5: Example of N204 Zones²

14.4 Voids Model

It was decided to sterilize all the blocks that lay within historical pit surface contours with the exception of some blocks at the edge of historical pits believed to be remaining in the pit walls. Blocks affected by historical underground workings were also sterilized. In some cases, where the level of precision of the historical underground workings was not sufficiently precise, a buffer zone was created to sterilize all blocks within the area.

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 47,998 assays are included in the mineralized zones.

Compositing of drillhole samples was conducted in order to homogenize the database for statistical analysis and remove any bias associated with 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 1.41 m. As a result, 39,392 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.

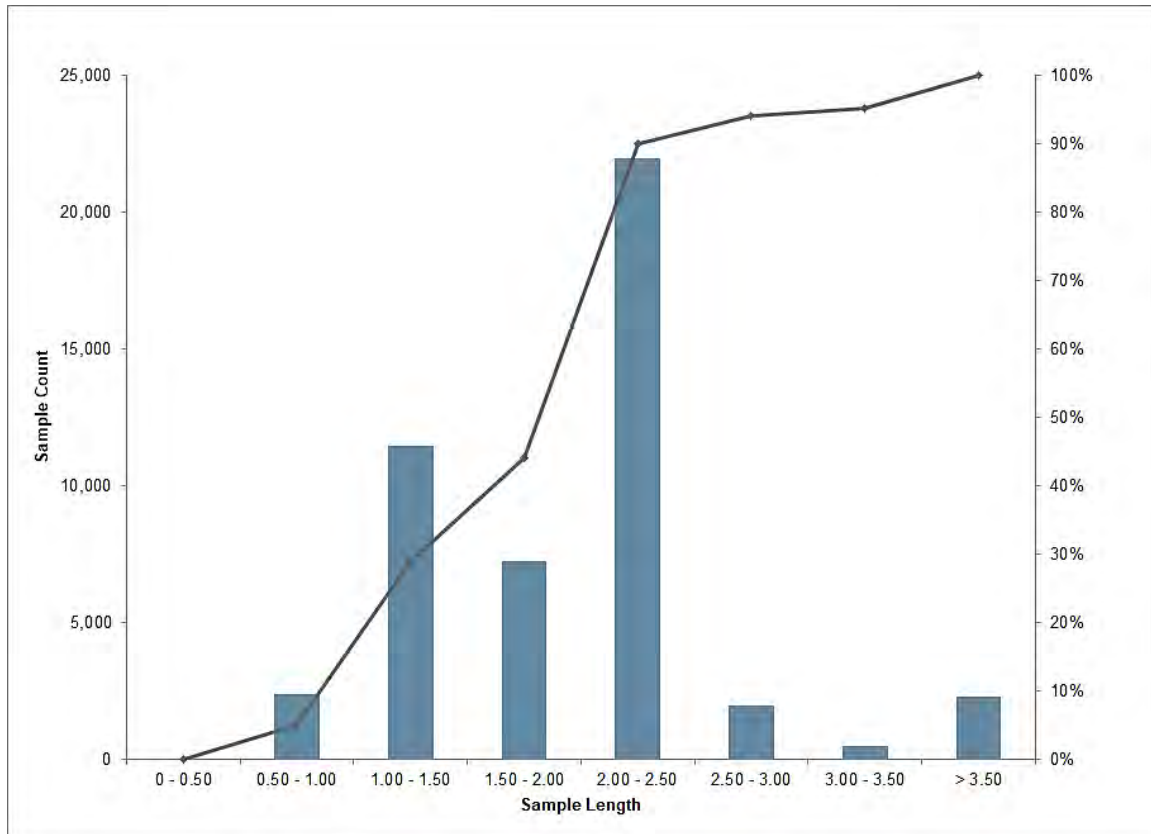


Figure 14-6: Sample length distribution within the mineralized zones

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 N204).

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.

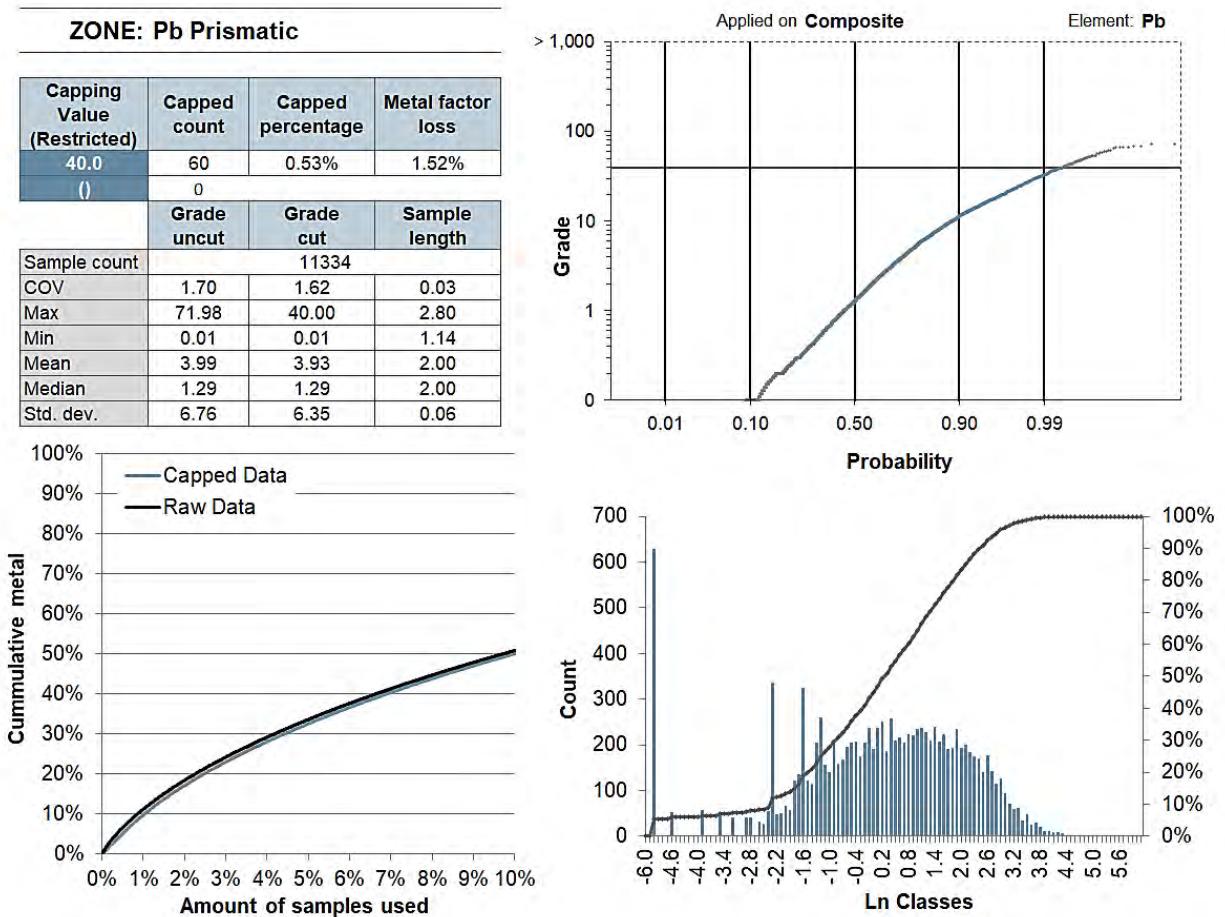


Figure 14-7: Graphs supporting capping on composites for the Prismatic Zones for Pb

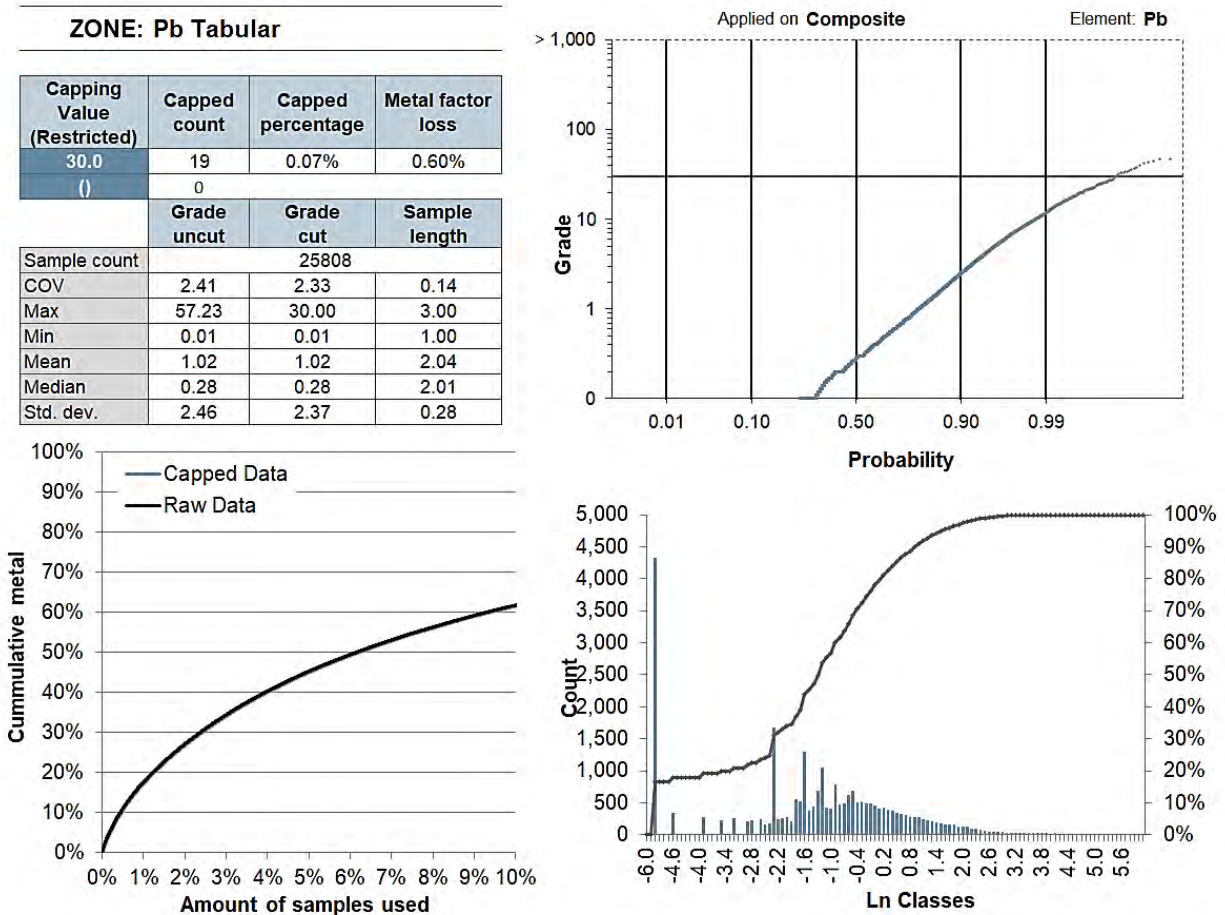


Figure 14-8: Graphs supporting capping on composites for the Tabular Zones for Pb

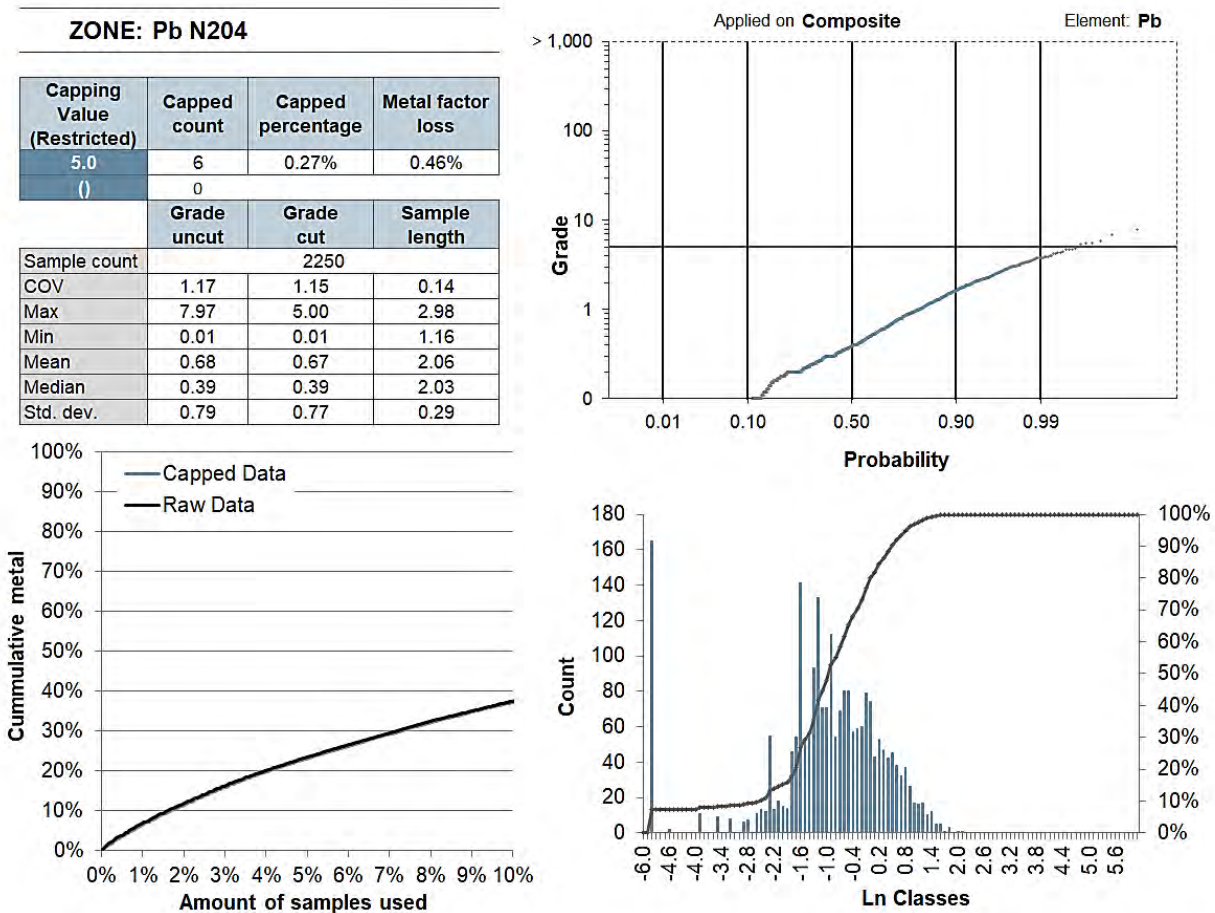


Figure 14-9: Graphs supporting capping on composites for the N204 Zones for Pb

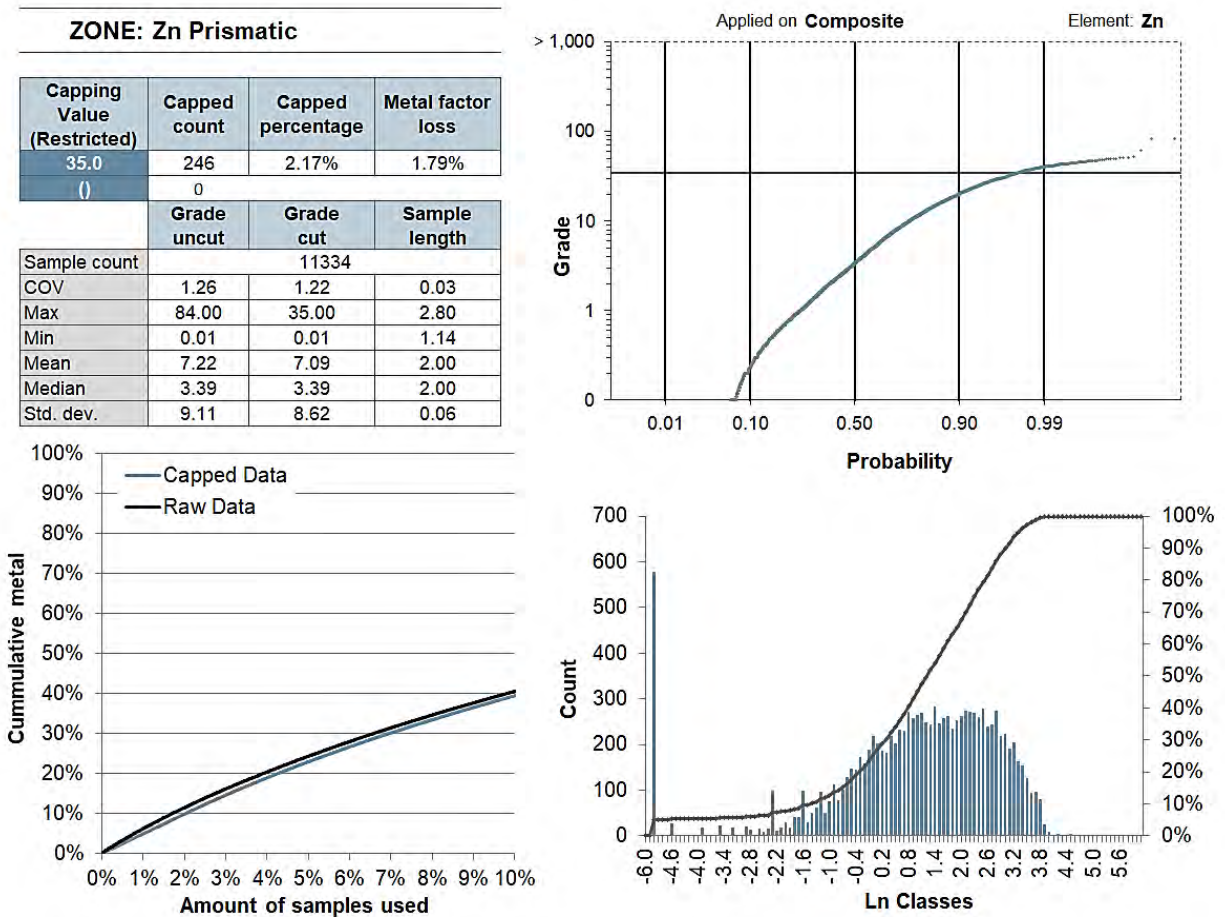


Figure 14-10: Graphs supporting capping on composites for the Prismatic Zones for Zn

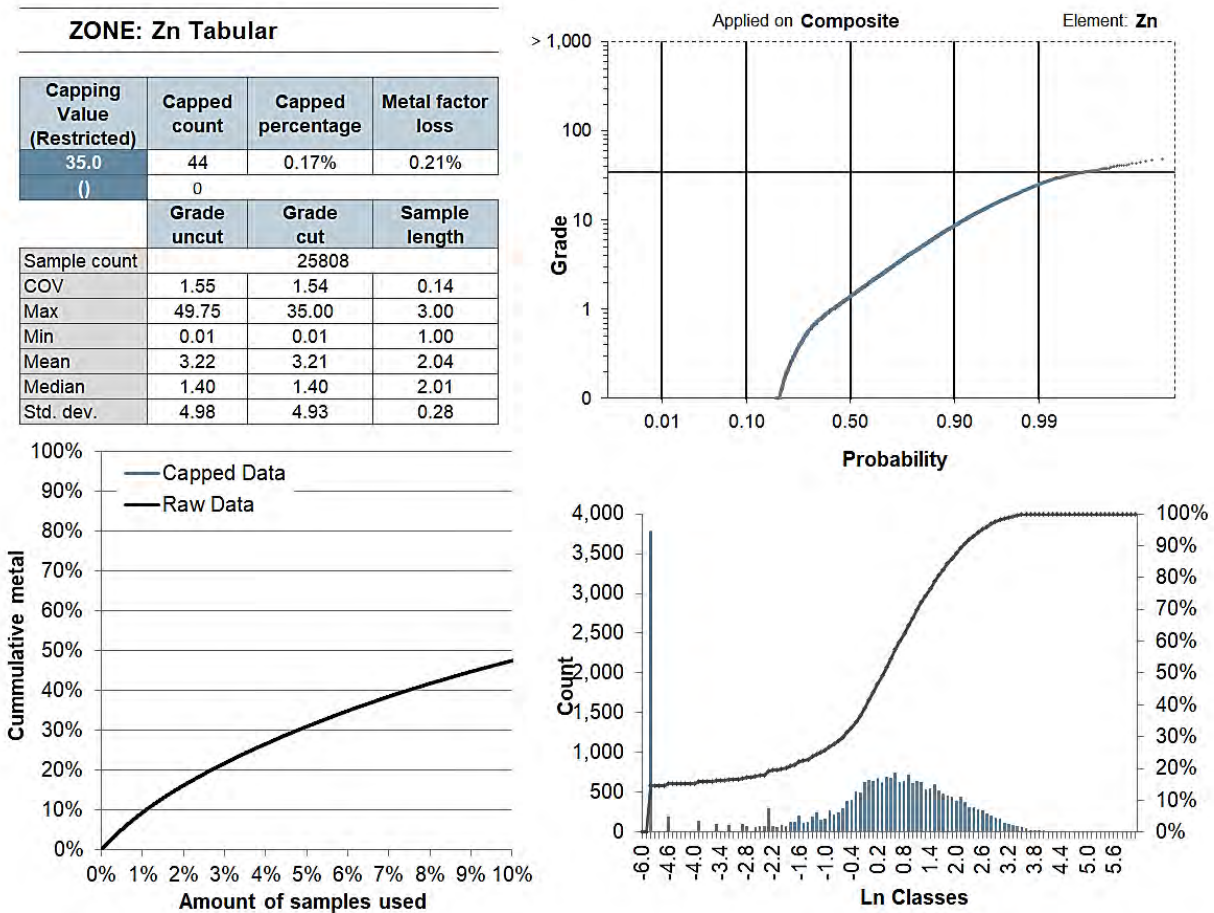


Figure 14-11: Graphs supporting capping on composites for the Tabular Zones for Zn

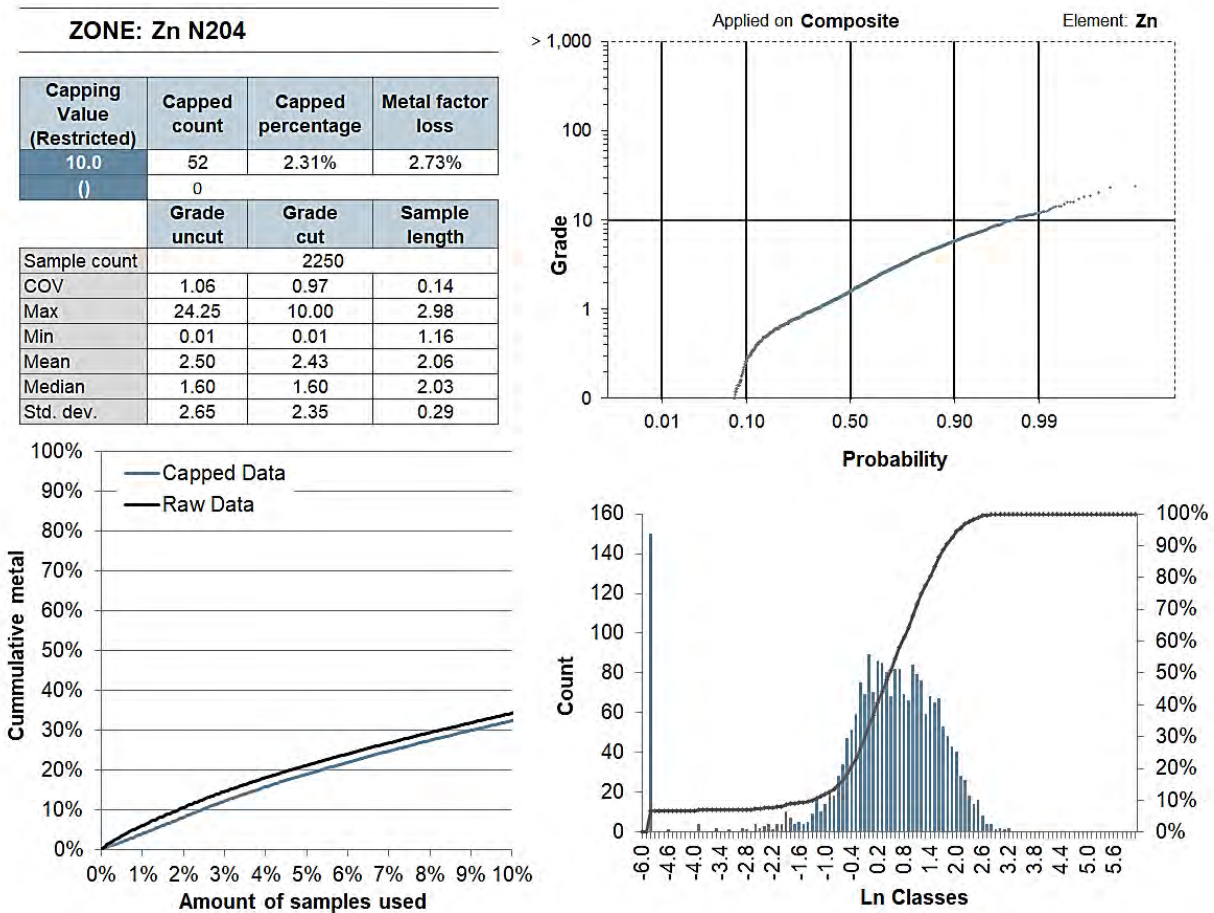


Figure 14-12: Graphs supporting capping on composites for the N204 Zones for Zn



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

	Zones	Composites count	Uncut mean (%)	COV	Max (%)	Uncut median (%)	Capping value	Number capped	Capped (%)	Metal loss (%)	Capped mean (%)	Capped COV	Capped median (%)
Pb	Prismatic	11,334	3.99	1.70	71.98	1.29	40	60	0.53%	1.52%	3.93	1.62	1.29
	Tabular	25,808	1.02	2.41	57.23	0.28	30	19	0.07%	0.60%	1.02	2.33	0.28
	N204	2,250	0.68	1.17	7.97	0.39	5	6	0.27%	0.46%	0.67	1.15	0.39
Zn	Prismatic	11,334	7.22	1.26	84.00	3.39	35	246	2.17%	1.79%	7.09	1.22	3.39
	Tabular	25,808	3.22	1.55	49.75	1.40	35	44	0.17%	0.21%	3.21	1.54	1.40
	N204	2,250	2.50	1.06	24.25	1.60	10	52	2.31%	2.73%	2.43	0.97	1.60

14.6 Density

Density values were calculated based on the formula established and used by Cominco Ltd. during their operational period between 1964 and 1988:

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

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/cm³).

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 number of samples in both populations (4,368 recent and 35,024 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.

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.

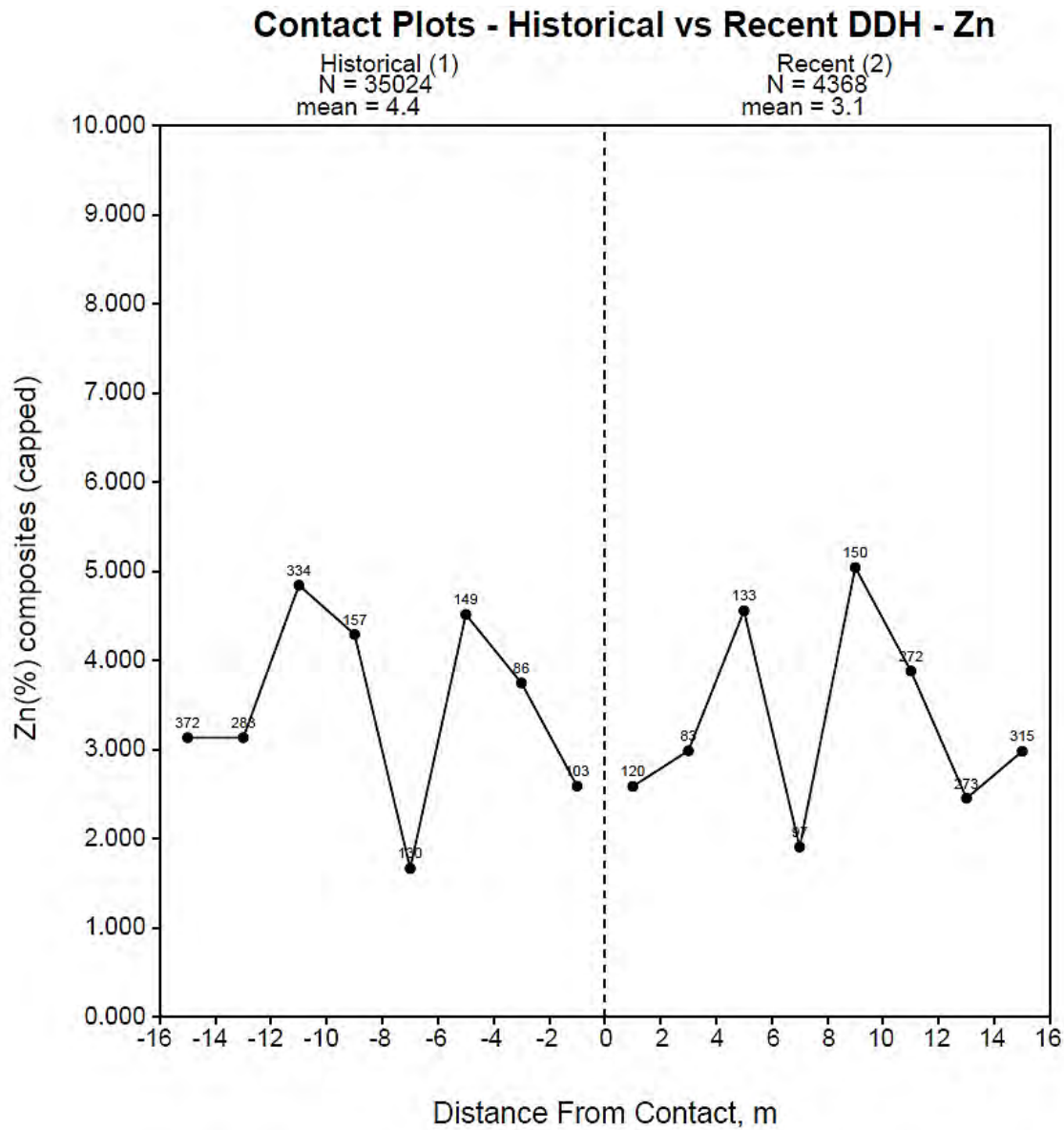


Figure 14-13: Contact analysis on the Zn capped composites between the historical and the recent drillholes

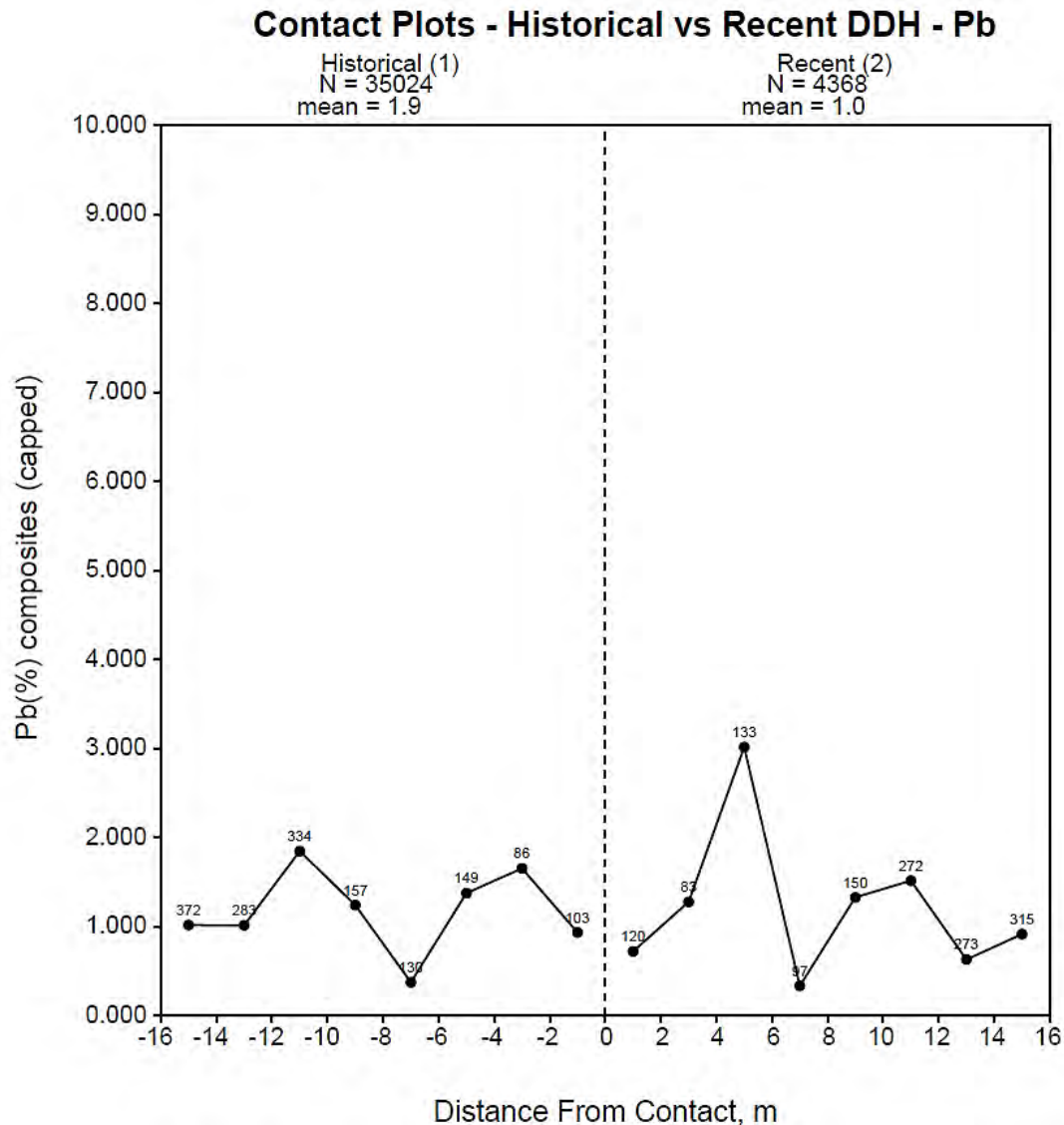


Figure 14-14: Contact analysis on the Pb capped composites between the historical and the recent drillholes

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 N204) 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. In all cases where a Normal Score Transformation was used, the results were back transformed prior to using the results to define the ellipsoids and interpolation parameters,

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	Nugget	Sill	Range X (m)	Range Y (m)	Range Z (m)	Sill	Range X (m)	Range Y (m)	Range Z (m)
Pb	N204	0.20	0.67	96	35	8	0.13	114	55	10
	Prismatic	0.23	0.69	45	52	25	0.08	74	61	40
	Tabular	0.44	0.38	34	22	2	0.18	103	80	5
Zn	N204	0.10	0.76	119	35	8	0.14	141	55	10
	Prismatic	0.07	0.83	36	56	20	0.09	68	61	40
	Tabular	0.46	0.37	49	32	2	0.17	103	63	5

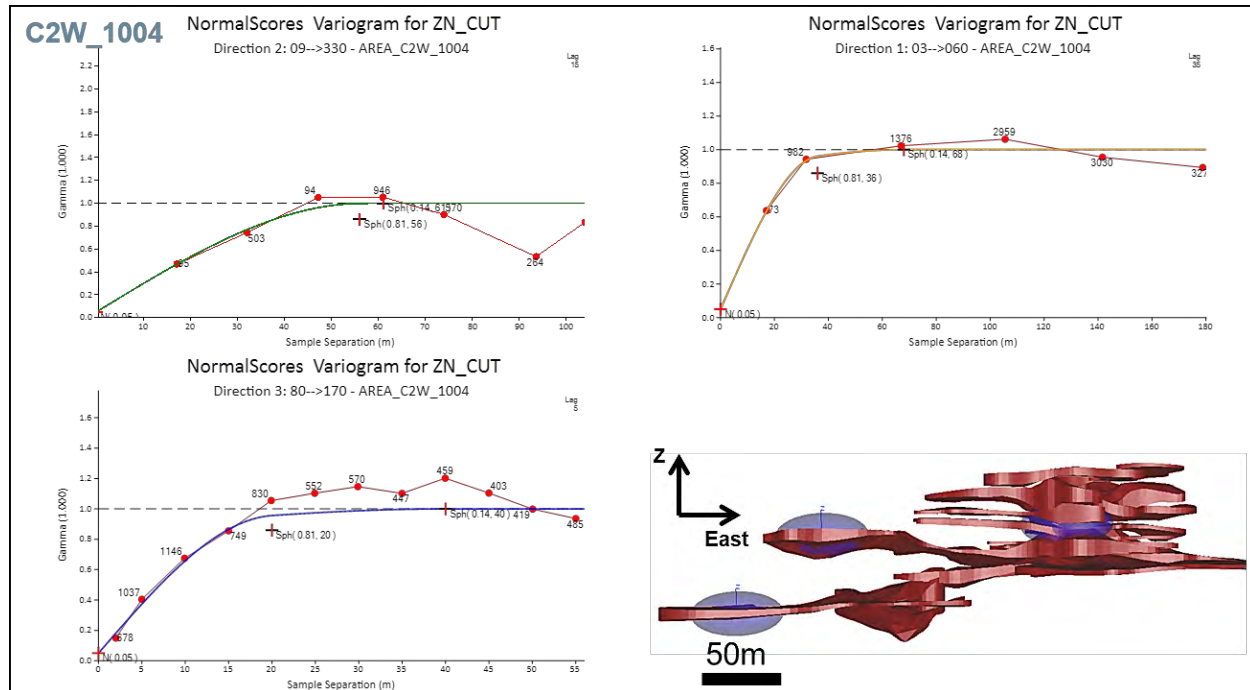


Figure 14-15: Example of the variography study for Prismatic Zones

14.9 Block Model

Due to the significant area covered by the Pine Point Project, twelve block models were created to facilitate the technical execution of the MRE.

The block models were constructed in Geovia GEMSTM 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)
EastMill_01	X (column)	639,615	1,322	13,220
	Y (row)	6,747,875	395	3,950
	Z (level)	235	30	150
Central_01	X (column)	626,465	687	6,870
	Y (row)	6,742,035	238	2,380
	Z (level)	250	34	170



		Origin coordinates	Number of blocks	Block extent (m)
Central_02	X (column)	632,865	724	7,240
	Y (row)	6,744,755	296	2,960
	Z (level)	260	30	150
Central_03	X (column)	627,985	59	590
	Y (row)	6,740,335	60	600
	Z (level)	260	30	150
Central_04	X (column)	621,290	294	2,940
	Y (row)	6,739,020	261	2,610
	Z (level)	235	35	175
North_01	X (column)	624,720	1,380	13,800
	Y (row)	6,747,065	225	2,250
	Z (level)	220	40	200
North_02	X (column)	617,450	400	4,000
	Y (row)	6,744,855	90	900
	Z (level)	215	33	165
NorthEast_01	X (column)	640,395	330	3,300
	Y (row)	6,757,190	163	1,630
	Z (level)	210	20	100
NorthEast_02	X (column)	649,600	183	1,830
	Y (row)	6,763,330	194	1,940
	Z (level)	210	30	150
West_01	X (column)	600,140	1349	13,490
	Y (row)	6,731,880	290	2,900
	Z (level)	220	47	235
West_02	X (column)	613,140	56	560
	Y (row)	6,734,910	63	630
	Z (level)	250	30	150
N204_01	X (column)	658,240	300	3,000
	Y (row)	6,758,545	248	2,480
	Z (level)	210	20	100

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 GEMST[™], 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 geological confidence, grade continuity, the presence of recent drillholes, and drillhole spacing and, therefore, some interpolated blocks were not converted into neither the Inferred nor the Indicated classification. Refer to the Mineral 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

	Zone	GEMS orientation			Pass 1			Pass 2			Pass 3		
					Search ellipsoid ranges			Search ellipsoid ranges			Search ellipsoid ranges		
		Azimuth	Dip	Azimuth	X (m)	Y (m)	Z (m)	X (m)	Y (m)	Z (m)	X (m)	Y (m)	Z (m)
Pb	N204	60	3	330	96	35	8	114	55	10	-	-	-
	Prismatic	330	-1	60	45	52	25	74	61	40	103	80	10
	Tabular	70	-1	160	34	22	2	103	80	10	-	-	-
Zn	N204	60	3	330	119	35	8	141	55	10	-	-	-
	Prismatic	330	-1	60	36	56	20	68	61	40	103	63	10
	Tabular	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

Interpolation parameters	Zn			Pb		
	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 ("ID²"); 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 ID², OK and NN interpolation scenarios. The OK scenario used for the mineral 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 model and composites 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 all classified blocks
(blocks > 50% inside a mineralized zone)

Sector	Number of composites	Number of blocks	Zn		Pb	
			Composite grade (%)	OK grade model (%)	Composite grade (%)	OK grade model (%)
Central	9,672	19,250	5.03	3.81	2.57	1.36
EastMill	8,653	18,217	3.26	2.34	1.24	0.70
N204	2,250	15,103	2.50	2.42	0.67	0.68
North	14,328	23,136	4.53	2.98	1.66	1.12
West	4,489	12,265	5.17	4.24	2.74	2.18
Total	39,392	87,971	4.33	3.11	1.86	1.16

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 composite grades (Figure 14-16 and Figure 14-17).

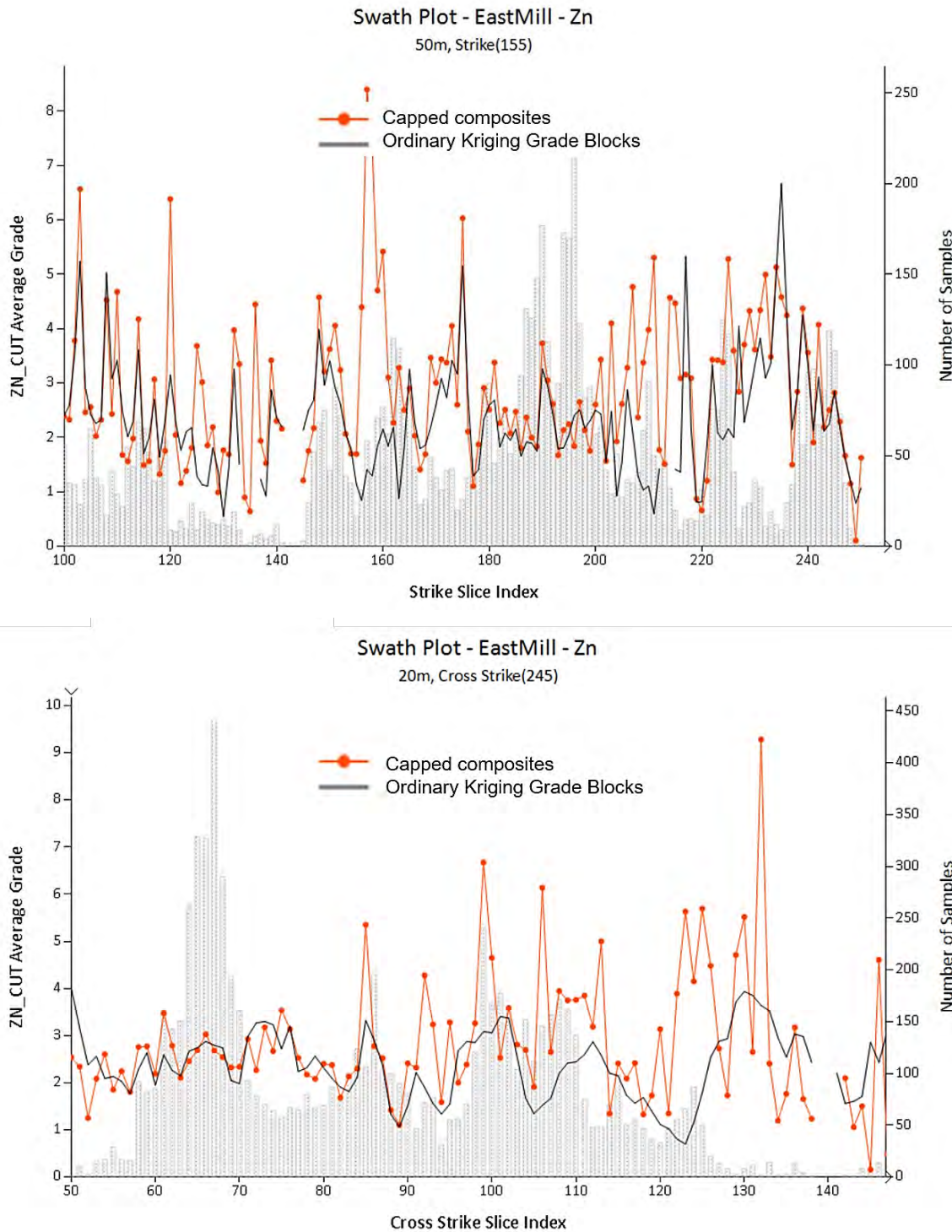


Figure 14-16: Pine Point model validation swath plots along strike (N155, top) and across strike (N245, bottom) for Zn

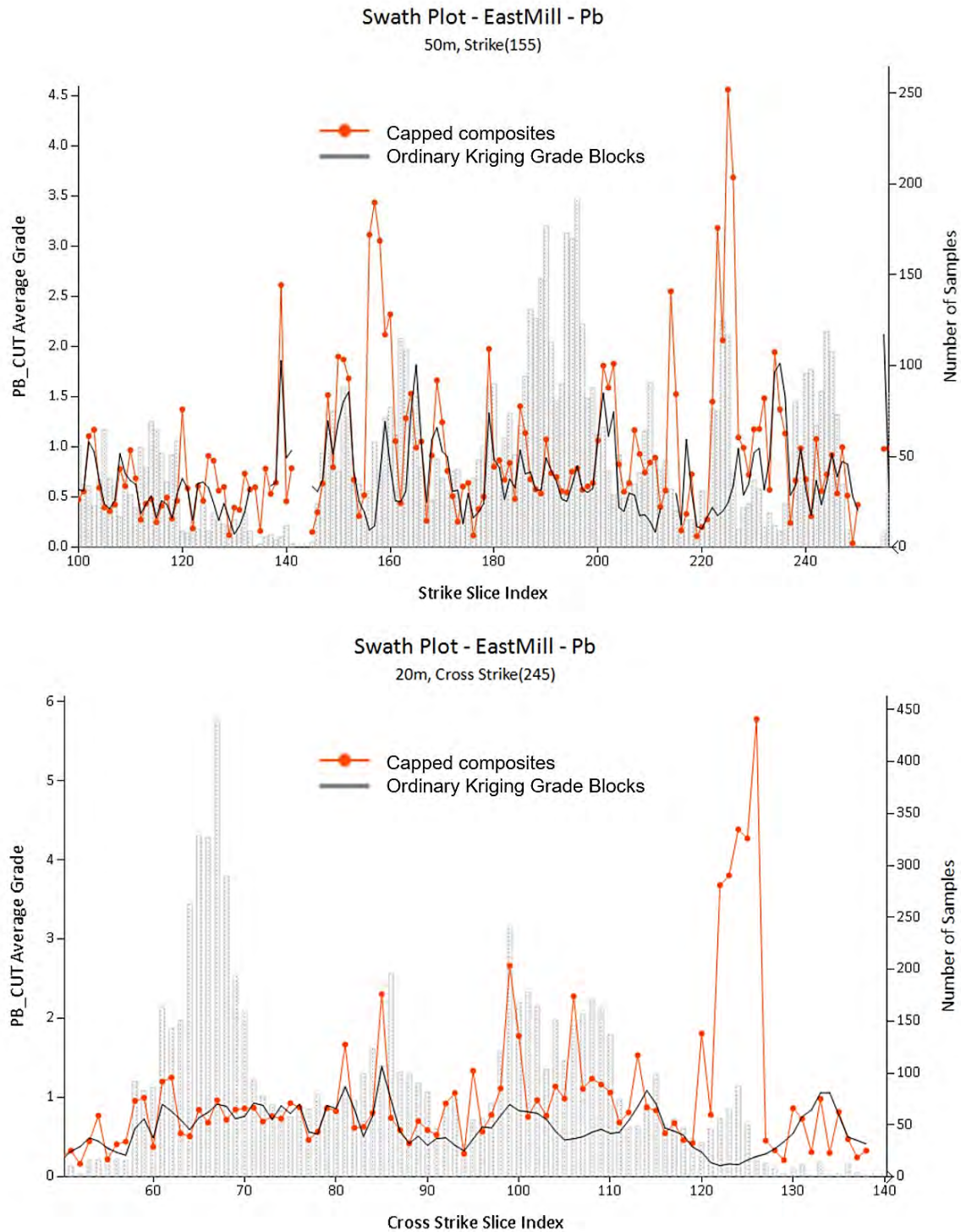


Figure 14-17: Pine Point model validation swath plots
 along strike (N155, top) and across strike (N245, bottom) for Pb

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 Mineral 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 either Inferred or Indicated Mineral Resource categories using drill spacing, geological continuity of mineralization, grade continuity, presence of recent drilling, and overall level of confidence. No Measured Mineral Resources were defined in this phase of the Project.

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).

Indicated Mineral Resources were defined where the following criteria were met:

- Drill spacing of 30 m or less;
- Demonstrated geological continuity;
- Grade continuity at the reported cut-off grade;
- Recent drillholes confirming the model (geologically and grade-wise);
- Elevated level of confidence on the collar location.

It should also be mentioned that significant work was completed since the latest MRE in order to increase confidence in the new block model:

- Improved DGPS Survey data for historical Cominco drillhole collars;
- Additional drillholes (168 DDH; 8,312 m) confirming the previous model (geologically and grade-wise);

- Partial relogging and resampling program for historical Cominco drillholes confirming the model;
- Twin holes for some of the historical drillholes for which no assays were present in the database;
- Some unsampled historical holes were either sampled or twinned. Grades are now coherent with the model and recent drillholes in the area.

It should also be mentioned that Indicated Mineral Resources were only declared where recent drillholes were sufficient to do so. Any sector solely defined by historical drillholes were kept Inferred even if they met most of the criteria above.

For example, resampling historical holes was not considered sufficient and recommendations were made to add recent holes in zones where none existed before upgrading the classification from Inferred to Indicated. Therefore, multiple sectors have 30 m drill spacing or tighter, geological and grade continuity, and are covered by the recent resurveying program, but are currently declared as Inferred pending additional drilling from the issuer.

When needed, a series of clipping boundaries were created manually in longitudinal views to either upgrade or downgrade classification in order to avoid issues due to automatically generated classification. All remaining estimated but unclassified blocks were not reported.

14.15 Pit Constrained 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, the QP carried out a pit optimization analysis using Hexagon's MinePlan 3D 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 1.25% to 1.50% equivalent zinc (Table 14-8). The reason for the variation in the cut-off grade is due to the fact that different haulage costs are applied depending on the distance between the deposit to the assumed mill site. It should be noted that no mill is currently present on the Project, 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 N204 deposit. The variation between the cut-off grade used in the 2020 PEA and this current report is mainly due to commodity price adjustment.

The pit optimization analysis was carried out using an overall pit slope of 26.6 degrees in the overburden and an overall pit slope in the rock formation of 45 degrees. Mining dilution and mining 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 1.0. 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.

The cut-off grade and pit optimization analysis resulted in a Mineral Resource Estimate that incorporates 53 open-pits (Figure 14-18 to Figure 14-24).

Figure 14-25 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 – Overburden	\$/t mined	2.20
Mining Cost – Rock	\$/t mined	3.58
Mining Cost – Mineralized material	\$/t mined	3.85
Mine Dewatering Cost ⁽¹⁾	\$/t mined	0.00;0.48;0.88 depending on which zone
Haulage Cost to Mill	\$/t-km	0.13
Pre-concentration Cost	\$/t mineralized material	5.43
Processing Cost ⁽²⁾	\$/t milled	13.40
General & Administration Cost ⁽²⁾	\$/t milled	14.12
Recoveries		
Overall Zinc ⁽³⁾	%	87.0
Overall Lead ⁽³⁾	%	92.0
Pre-concentration Mass Pull	%	59.5
Zinc Concentrate Grade	%	59.0
Lead Concentrate Grade	%	64.0



Parameter	Unit	Input
Payables		
Zinc	%	85.0
Lead	%	96.0
Zinc Concentrate Costs		
Transport to Rail	\$/dmt	33.00
Transport to Smelter	\$/wmt	198
Smelter Cost	\$/dmt	295.00
Lead Concentrate Costs		
Transport to Rail	\$/dmt	33.00
Transport to Smelter	\$/wmt	223.30
Smelter Cost	\$/dmt	262.00
Metal Prices		
Zinc	USD/lb	1.30
Lead	USD/lb	1.00
Exchange Rate (CAD:USD)		1.27
Royalties	% NSR	3.0

1. Applied to both mineralized material and waste rock tonnages.
2. Costs per tonne milled are based on a 59.5% Pre-concentration Mass Pull.
3. Inclusive of sorting test program results.

Table 14-8: MRE Open-pit cut-off grade per area

Area	ZnEq (%)
Central Zone	1.25
East Mill Zone	1.25
North Zone	1.25 - 1.35
N204 Zone	1.50

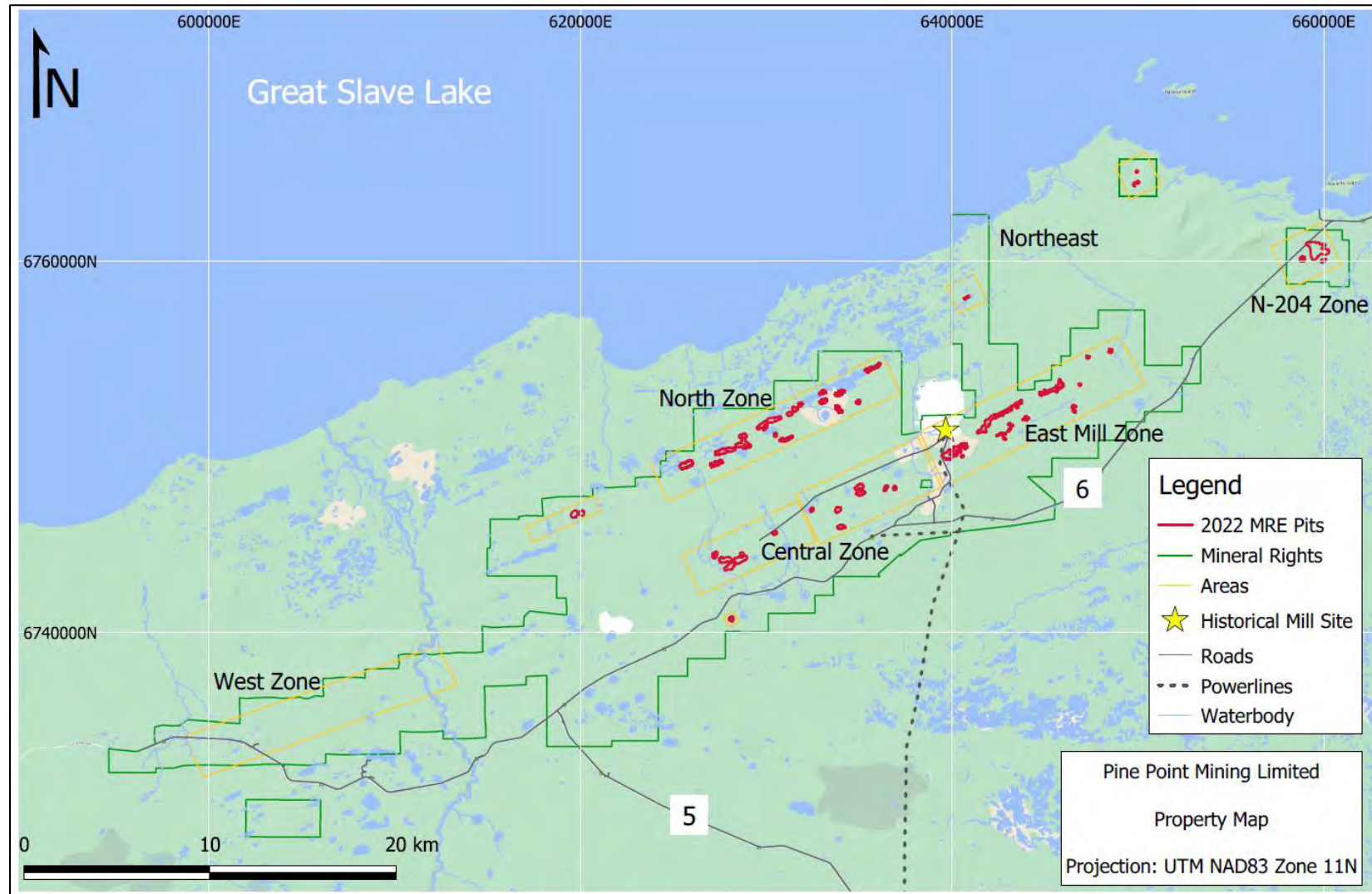


Figure 14-18: General view of the Project – Location of the different pit-constrained zones, historical mill and the assumed location of a future mill

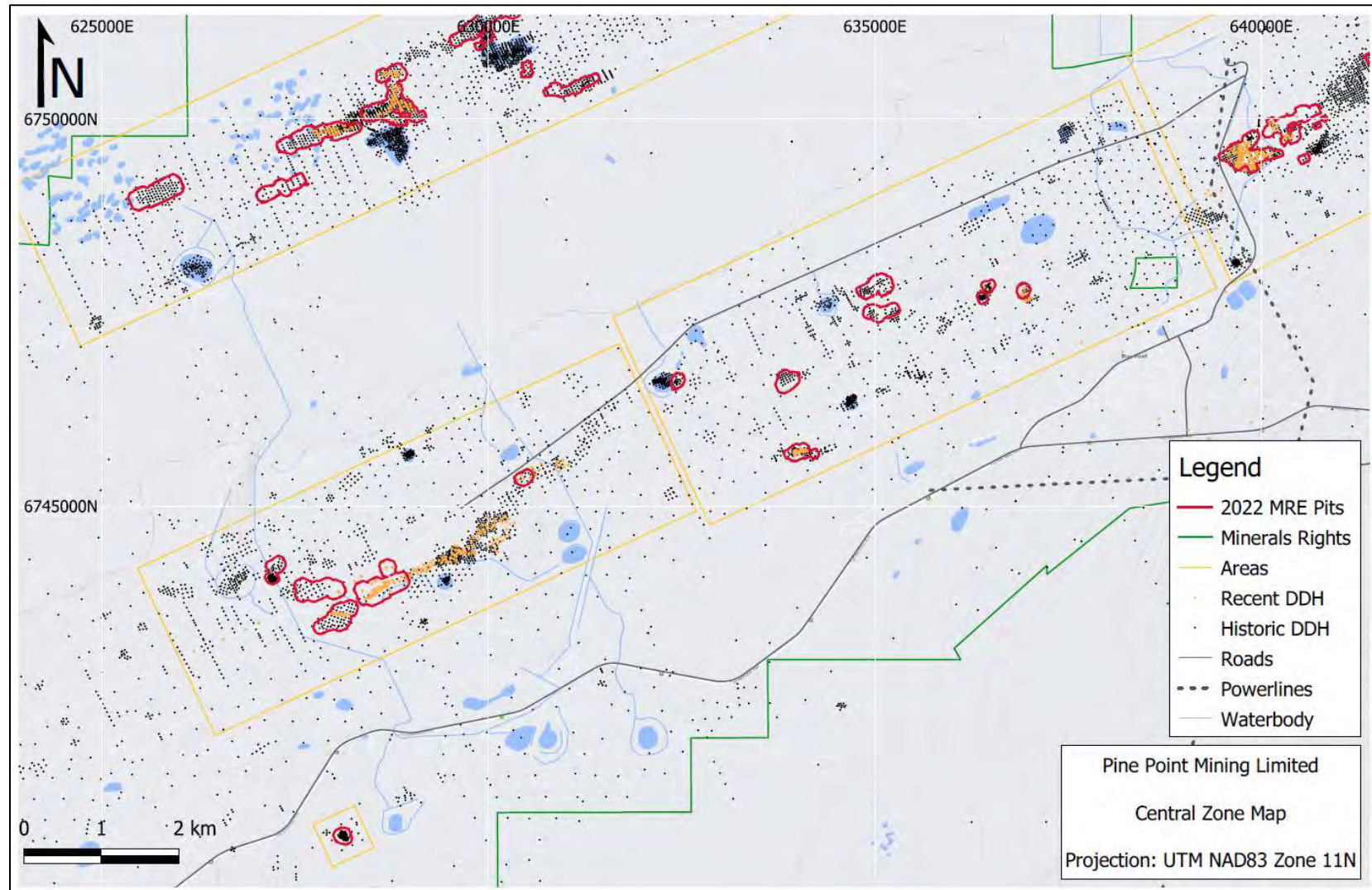


Figure 14-19: Central Zone map showing the location of drillholes and pits from the current MRE

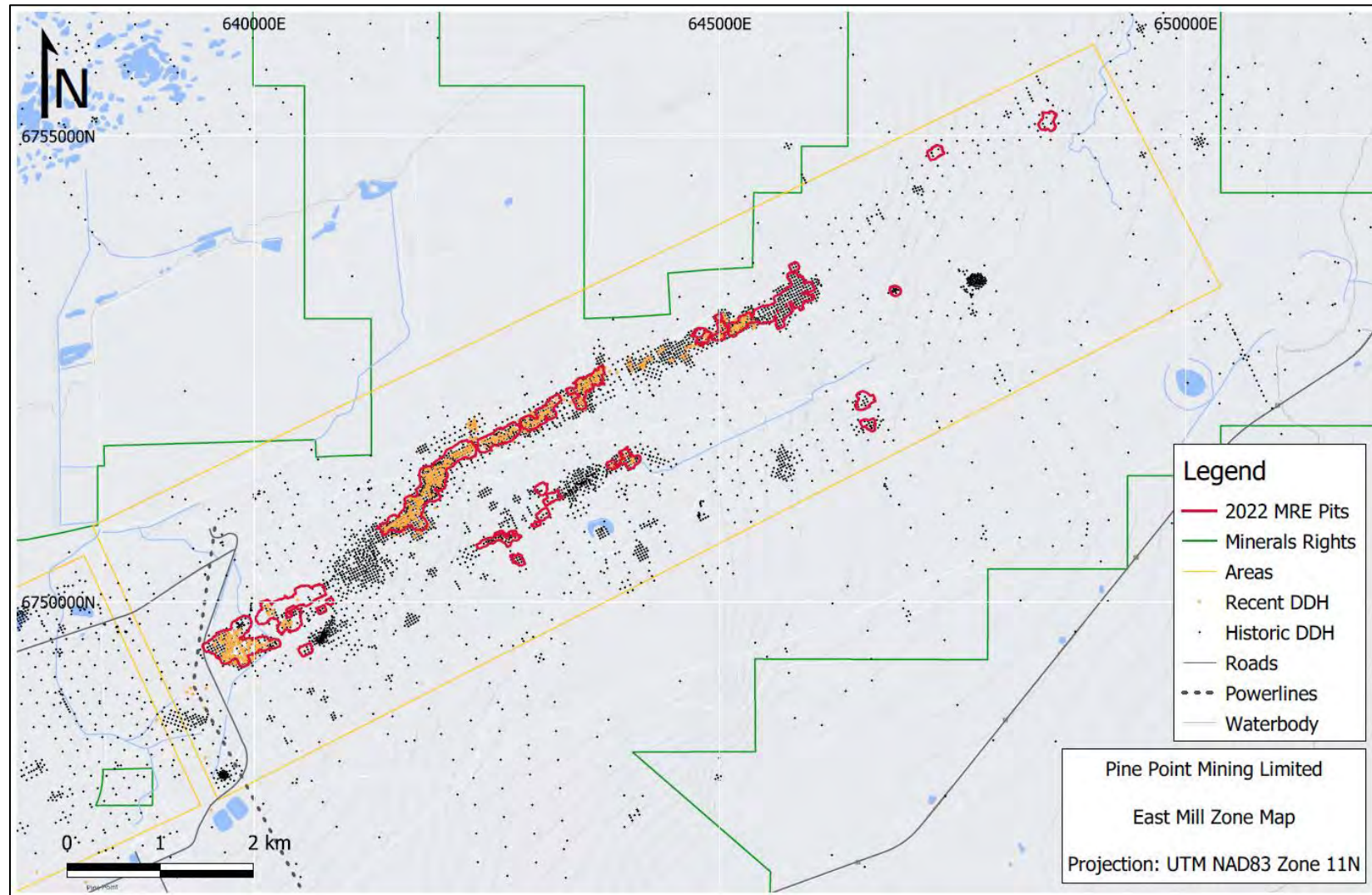


Figure 14-20: East Mill Zone map showing the location of drillholes and pits from the current MRE

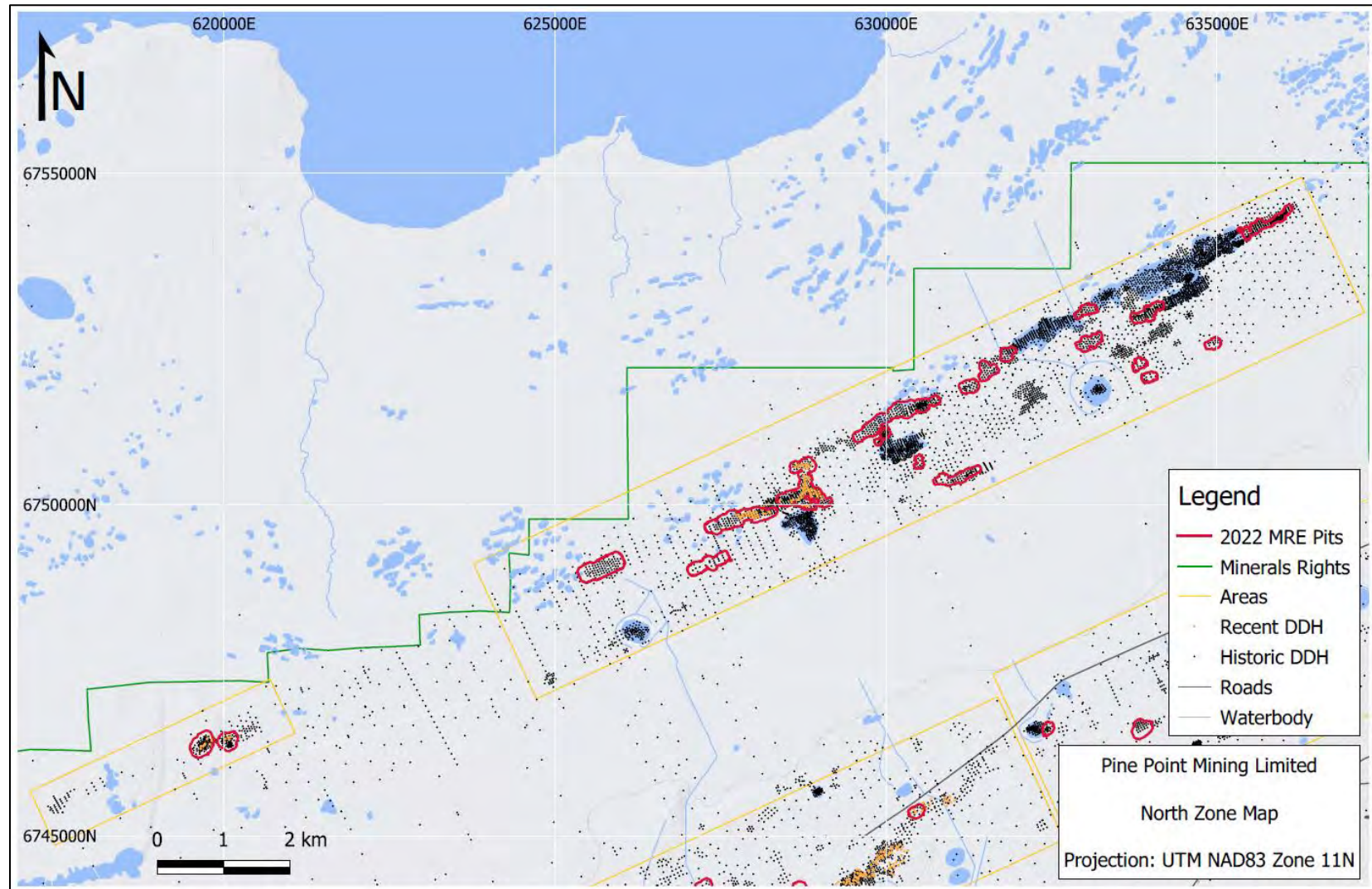


Figure 14-21: North Zone map showing the location of drillholes and modelled pits from the current MRE

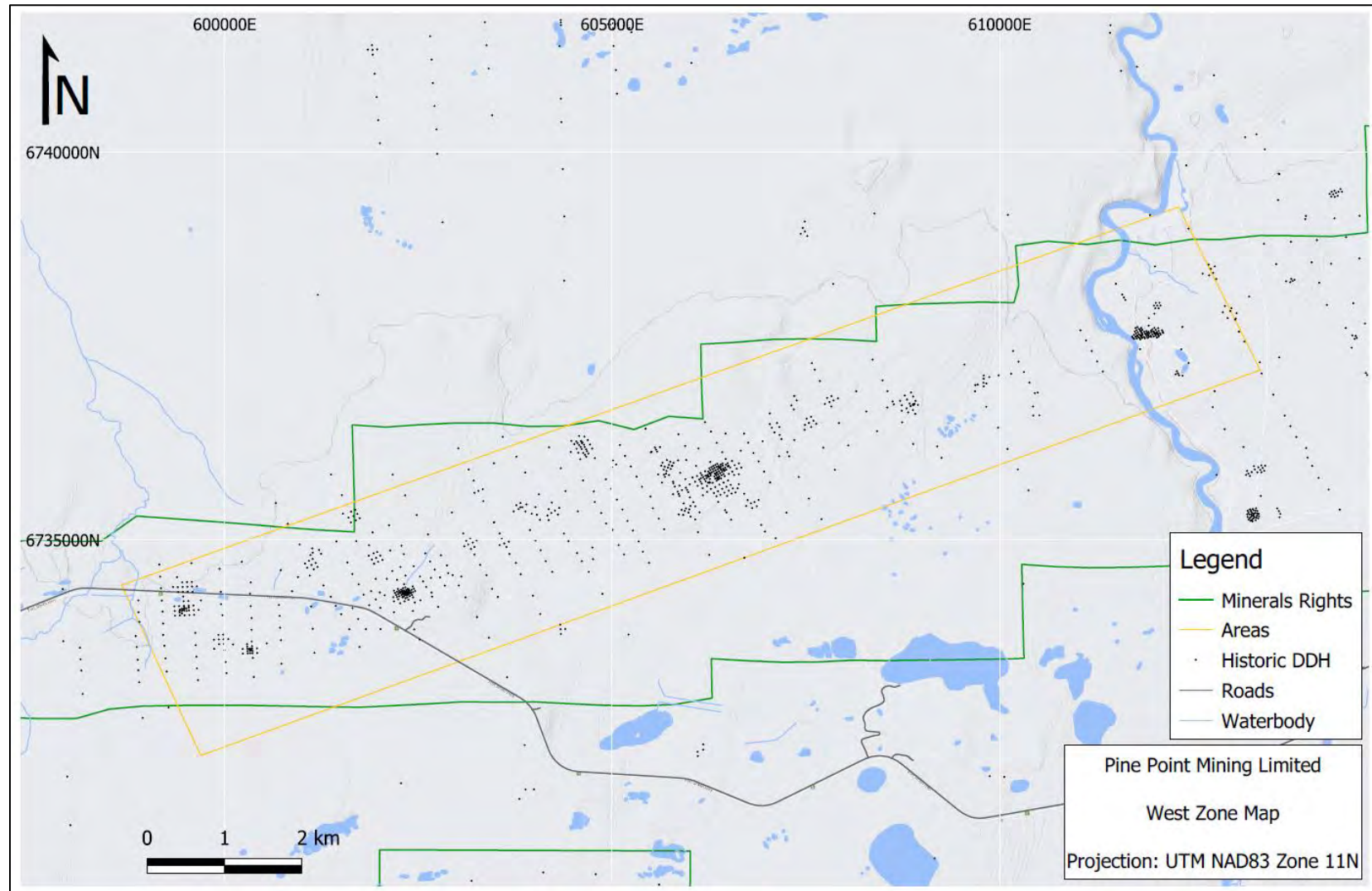


Figure 14-22: West Zone Map showing the location of drillholes and pits from the current MRE

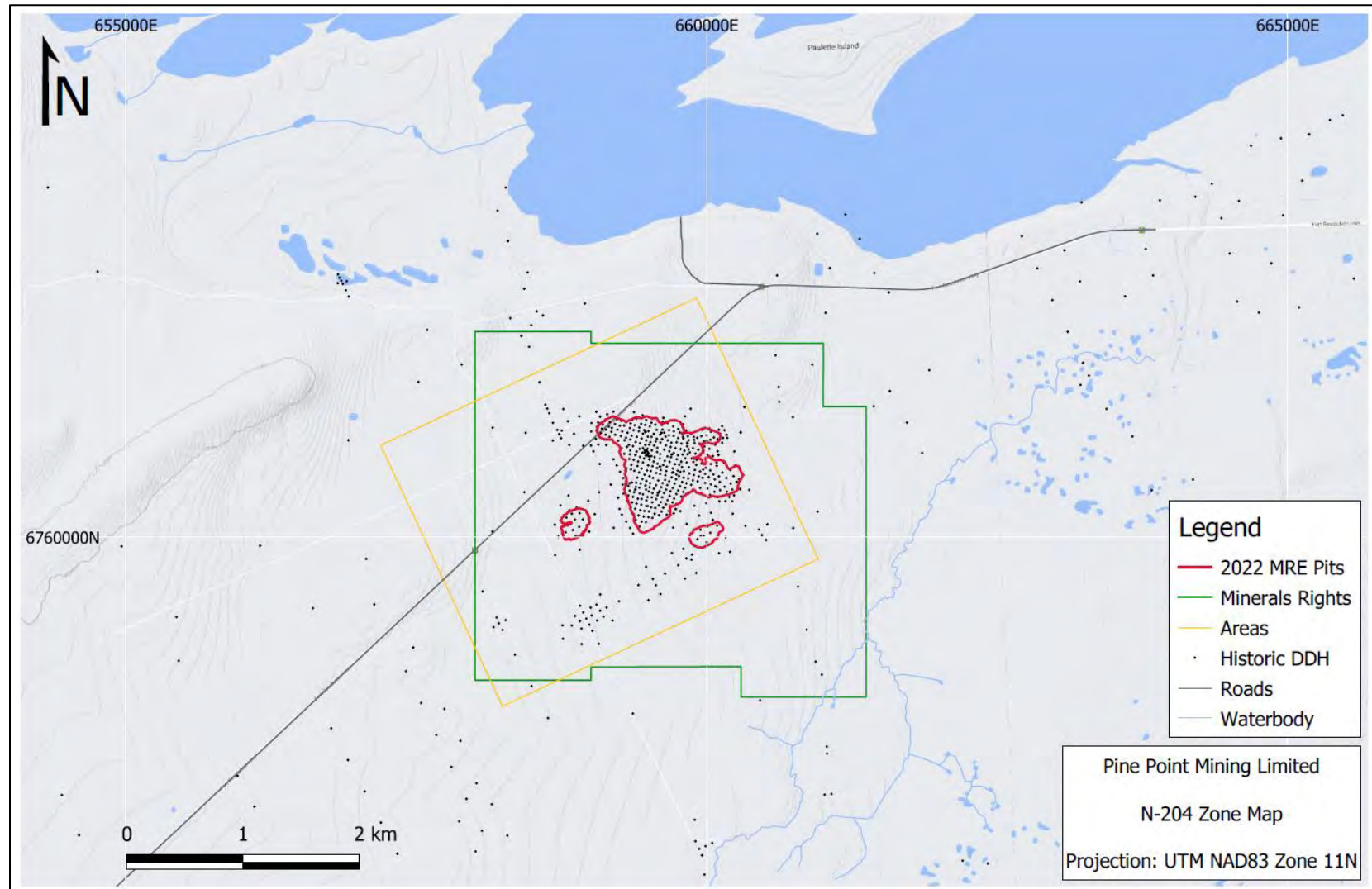


Figure 14-23: N204 Zone map showing the location of drillholes and pits from the current MRE

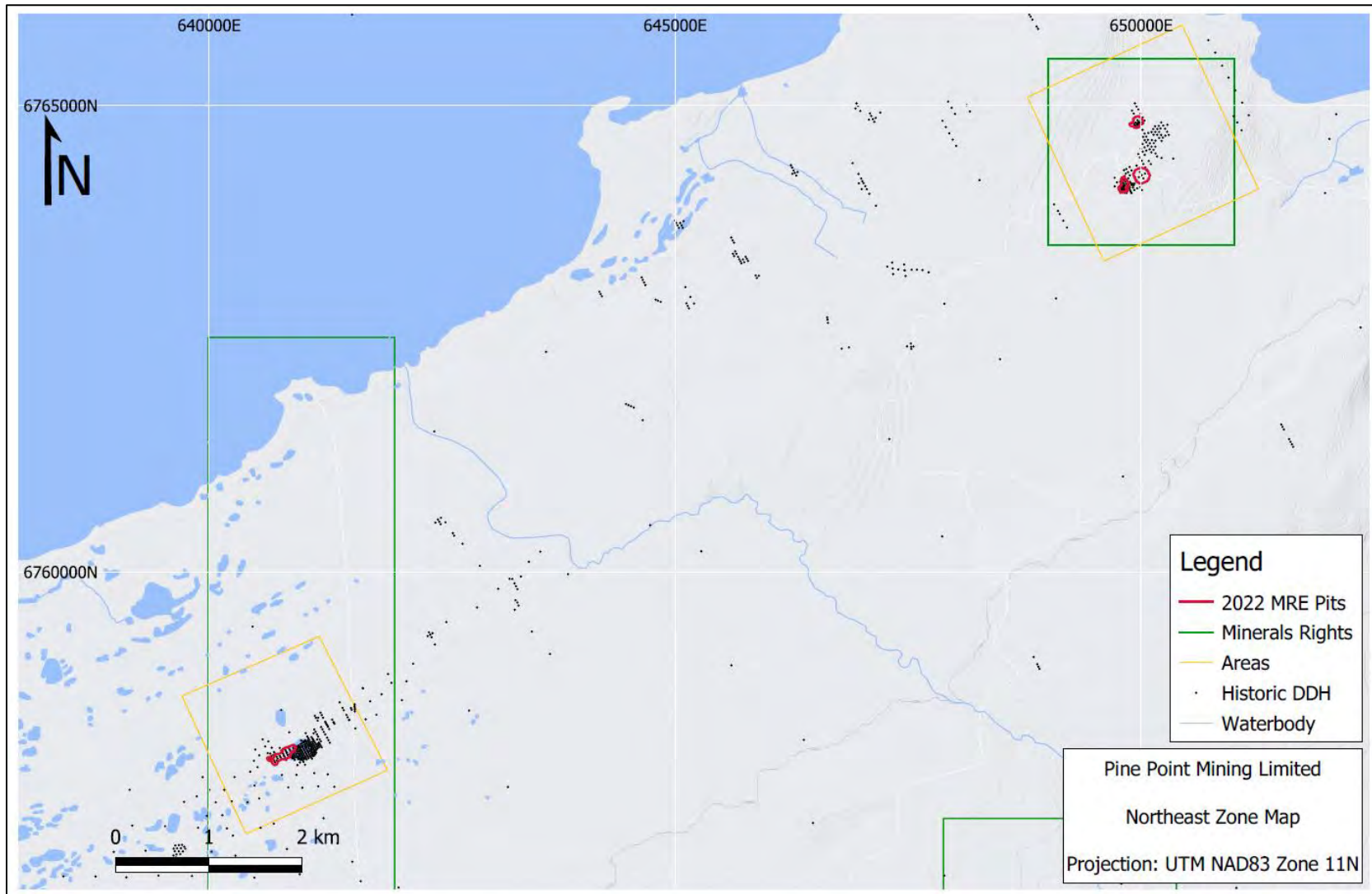


Figure 14-24: Northeast Zone map showing the location of drillholes and pits from the current MRE

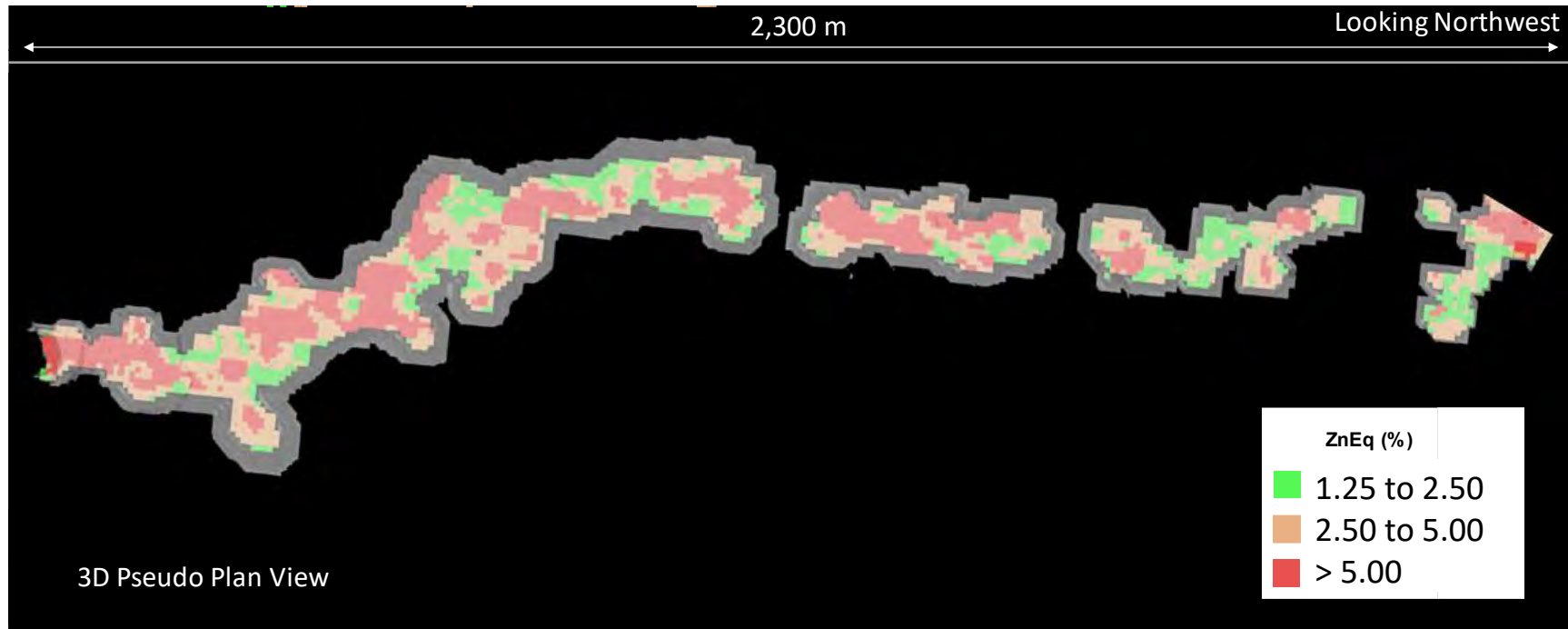


Figure 14-25: 3D view from the East Mill Area
Blocks shown are 1) within pit optimized shells; 2) above the cut-off grade; and 3) classified as either Indicated or Inferred

14.16 Underground Cut-off Grade

The underground portion of the 2022 MRE utilizes similar financial and smelting assumptions to the pit-constrained portion of the 2022 MRE. Importantly, it differs from previous operators' estimates and comprises mostly easily accessible mineralization found adjacent to the pit wall boundaries of certain deposits, as well as mineralization that is relatively continuous over longer distances between pit-constrained mineralization.

Figure 14-26 shows a cross-section with both pit constrained and underground mineralization. Part of the underground mineralization outlined in the 2022 MRE could be accessed by ramp at the bottom of adjacent pits.

Figure 14-27 and Figure 14-28 present maps showing the location of drillholes and underground mineral resources from the current MRE.

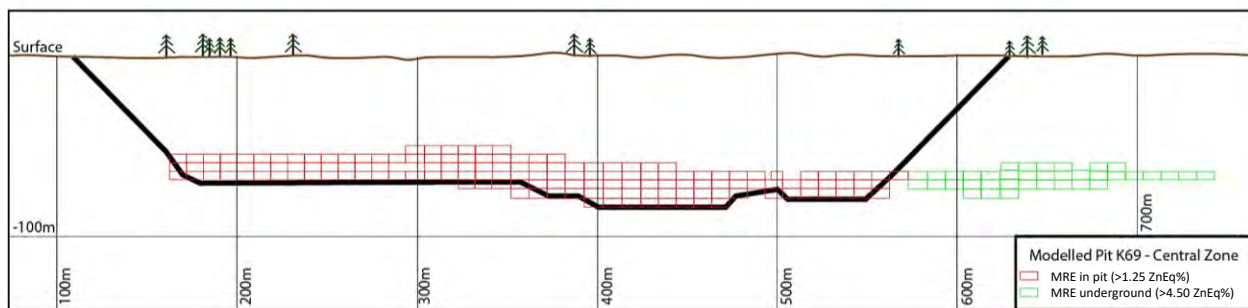


Figure 14-26: Typical cross-section with pit constrained and underground mineralization

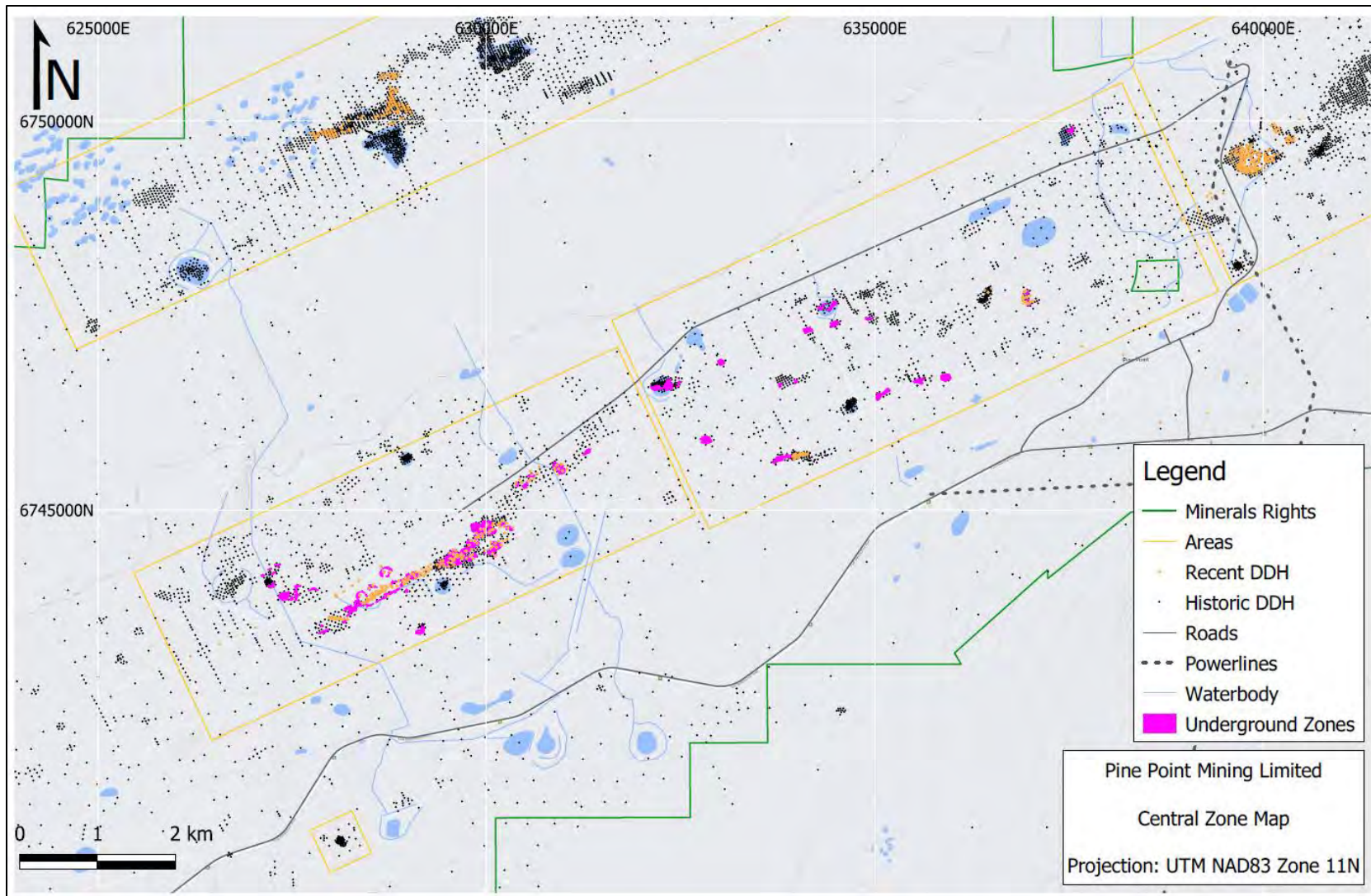


Figure 14-27: Central Zone map showing the location of drillholes and underground mineral resources from the current MRE

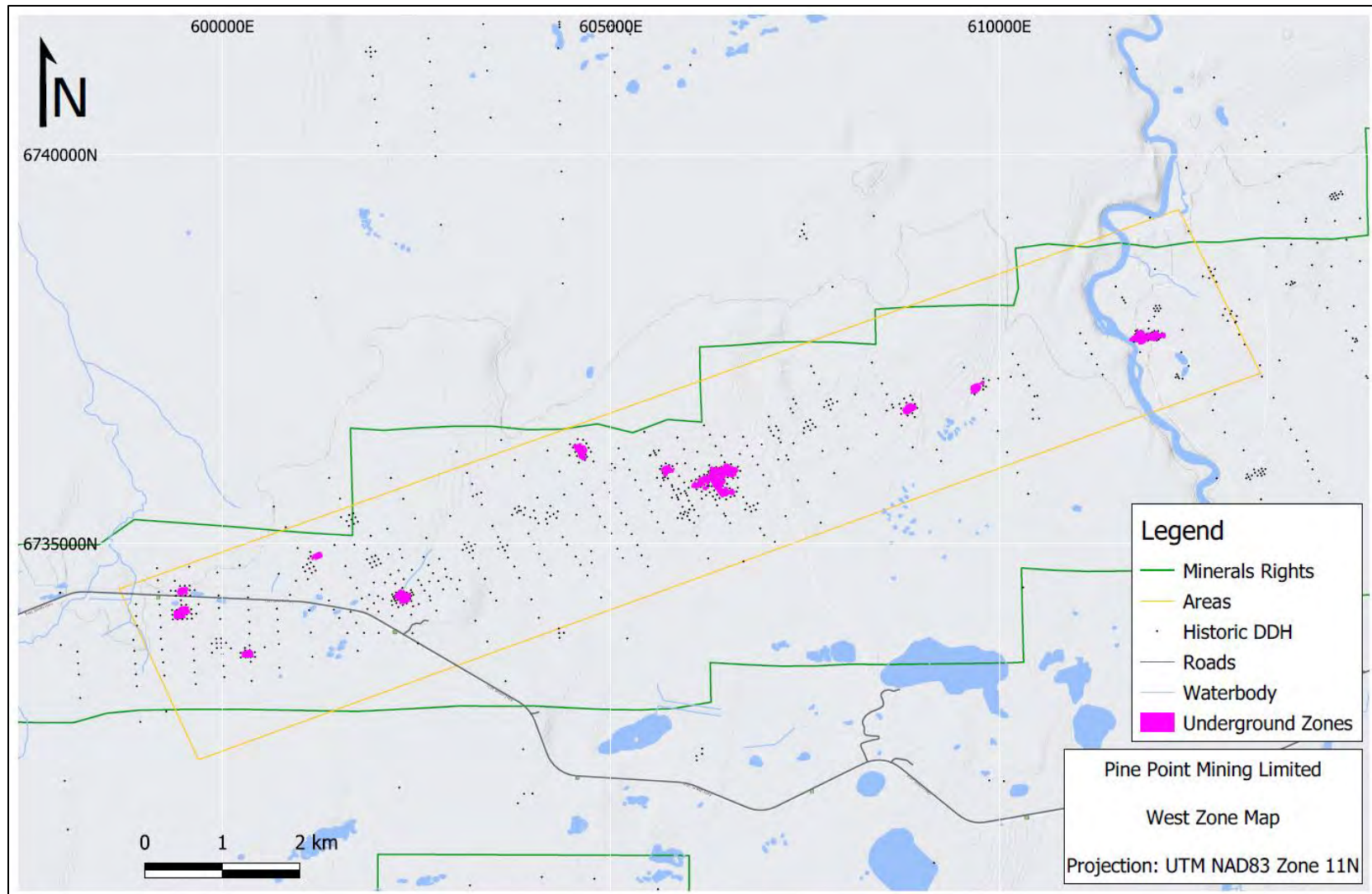


Figure 14-28: West Zone map showing the location of drillholes and underground mineral resources from the current MRE

14.17 Pine Point Mineral Resource Estimate

The Mineral Resource Estimate presented herein is either constrained within pit shells developed from the pit optimization analysis discussed above or presented as underground mineral resources using appropriate cut-off grades. All deposits for which no pit was generated (because they were not economic using the parameters presented in the previous section) were either removed from the mineral resource estimate or presented as mineral underground resources if they met the criteria for such a type of operation.

Table 14-9: Pine Point Inferred Mineral Resource Estimate

Method	Zone	Cut-off Grade (ZnEq %)	Indicated				Inferred			
			Tonnage	ZnEq	Pb	Zn	Tonnage	ZnEq	Pb	Zn
			(kt)	(%)	(%)	(%)	(kt)	(%)	(%)	(%)
Pit Constrained Mineral Resources	Central	1.25	2,424	6.36	1.47	5.04	4,373	6.58	1.65	5.09
	East Mill	1.25	7,232	4.74	1.23	3.63	4,624	4.46	0.89	3.65
	North	1.25 - 1.35	6,097	6.18	1.91	4.46	13,707	4.92	1.43	3.64
	N204	1.50					11,707	4.08	0.90	3.28
Underground Mineral Resources	Central	4.50					2,735	6.91	1.57	5.49
	West	4.15					10,060	9.62	3.31	6.64
Total Pit Constrained		1.25 - 1.50	15,753	5.55	1.53	4.17	34,411	4.78	1.21	3.70
Total Underground		4.15 - 4.50	-	-	-	-	12,796	9.04	2.94	6.39
Total Combined			15,753	5.55	1.53	4.17	47,207	5.94	1.68	4.43

Notes to Table 14-9:

1. The independent qualified person for the 2022 MRE, as defined by National Instrument ("NI") 43-101 guidelines, is Pierre-Luc Richard, P.Geo. The effective date of the 2022 MRE is July 11, 2022.
2. These mineral resources are not mineral reserves as they do not have demonstrated economic viability. The quantity and grade of reported Inferred Mineral Resources in this MRE are uncertain in nature and there has been insufficient exploration to define these Inferred Mineral 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. Mineral resources are presented as undiluted and in-situ for an open-pit and underground scenario and are considered to have reasonable prospects for economic extraction. The constraining pit shells were developed using overall pit slopes of 45 degrees in bedrock and 26.6 degrees in overburden. Mineral resources show sufficient continuity and isolated blocks were discarded; therefore, the herein MRE meets the CIM Guidelines published in November 2019.

4. The 2022 MRE was prepared using GEOVIA GEMS 6.8.3 and is based on 19,509 surface drillholes and 166,376 samples, of which 7,852 drillholes and a total of 47,998 assays were included in the modelled mineralization. The drillhole database includes recent drilling of 78,195 m in 1,182 drillholes since 2017 and also incorporates Cominco Ltd.'s historical drillholes, the use of which was partially validated by a drillhole collar survey, twinning programs, and a partial core resampling program. The cut-off date for the drillhole database was December 31, 2019. Approximately 35,000 m in 550 drillholes were added to the project since the drillhole database cut-off date.
5. The 2022 MRE encompasses 254 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 1988. 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 mineral resource estimation was calculated from drillhole data using an Ordinary Kriging interpolation method in a percent block model using blocks measuring 10 m x 10 m x 5 m in size.
9. Zinc equivalency percentages are calculated using long term metal prices indicated below (item 10), forecasted metal recoveries, concentrate grades, transport costs, smelter payable metals and charges.
10. The estimate is reported using a ZnEq cut-off varying from 1.25% to 1.50% for open-pit mineral resources and 4.15% to 4.50% for underground mineral resources. Variations take into consideration trucking distances from the pit constrained mineralization to the mill and metallurgical parameters for each area. The cut-off grade was calculated using the following parameters (amongst others): zinc price = USD1.30/lb; lead price = USD1.00/lb; CAD:USD exchange rate = 1.27. The cut-off grade will be re-evaluated in light of future prevailing market conditions and costs.
11. The 2022 MRE presented herein is categorized as Inferred and Indicated Mineral Resources. The Inferred Mineral Resource category is constrained to areas where drill spacing is less than 100 metres and the Indicated Mineral Resource category is constrained to areas where drill spacing is less than 30 metres. In both cases, reasonable geological and grade continuity were also a criteria during the classification process.
12. The pit optimization to develop the mineral resource constraining pit shells was done using Hexagon's MinePlan 3D Version 15.8.
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 QP is not aware of any known environmental, permitting, legal, title-related, taxation, sociopolitical or marketing issues, or any other relevant issues that could materially affect this MRE.



Table 14-10, Table 14-11 and Table 14-12 show the sensitivity of the block model estimate to grade cut-off for the in situ Mineral Resource Estimate. Cut-off grades lower than 2.00% ZnEq do not significantly change the pit-constrained mineral resource, indicating that the bulk of the mineralization within the wireframed model is above said threshold. Higher cut-off grades significantly increase the average grade of the deposit, as expected, with a complimentary drop in tonnage.

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

Table 14-10: Pit constrained Inferred cut-off grade sensitivity

Area	Cut-off grade (ZnEq %)	Tonnage (kt)	ZnEq (%)	Pb (%)	Zn (%)
ALL	5.00	11,401	8.02	2.14	6.10
	4.00	17,455	6.73	1.73	5.18
	3.00	24,926	5.75	1.45	4.45
	2.00	31,664	5.05	1.27	3.91
	1.75	32,181	5.00	1.26	3.87
	1.50	34,013	4.82	1.21	3.73
	1.35	34,565	4.77	1.20	3.69
	1.25	34,897	4.74	1.19	3.67
	1.00	35,693	4.66	1.17	3.60
	0.75	36,408	4.58	1.16	3.55



Table 14-11: Pit constrained Indicated cut-off grade sensitivity

Area	Cut-off grade (ZnEq %)	Tonnage (kt)	ZnEq (%)	Pb (%)	Zn (%)
ALL	5.00	6,799	8.71	2.60	6.37
	4.00	8,984	7.67	2.22	5.67
	3.00	11,613	6.73	1.90	5.01
	2.00	14,388	5.92	1.64	4.44
	1.75	14,938	5.77	1.60	4.33
	1.50	15,412	5.64	1.56	4.24
	1.35	15,627	5.58	1.54	4.19
	1.25	15,779	5.54	1.53	4.16
	1.00	16,084	5.46	1.50	4.10
	0.75	16,354	5.38	1.48	4.05

Table 14-12: Underground Inferred cut-off grade sensitivity

Area	Cut-off grade (ZnEq %)	Tonnage (kt)	ZnEq (%)	Pb (%)	Zn (%)
West and Central	10.00	3,589	15.99	5.44	11.09
	9.00	4,319	14.89	5.04	10.35
	8.00	5,270	13.73	4.64	9.55
	7.00	6,540	12.51	4.18	8.75
	6.00	8,133	11.33	3.76	7.95
	5.00	10,557	9.99	3.27	7.05
	4.50	12,032	9.34	3.04	6.61
	4.15	12,756	9.06	2.95	6.40



15. MINERAL RESERVE ESTIMATES

This chapter is not required for a Preliminary Economic Assessment Technical Report.

16. MINING METHODS

16.1 Summary

The Pine Point Lead-Zinc Project is composed of numerous lead-zinc deposits that cover a 65-km strike length.

The Project is to be mined both as open-pits and underground workings. Two types of geological deposits (Tabular and Prismatic) are included in the mineralized material. The Tabular deposits are a rather flat layer, at shallow depth (less than 200 m) and are generally thin (5-10 m). The Prismatic deposits have a more vertical extent that could have up to 60 m thickness and are still shallow. The shape of these two types of deposits creates a series of aligned pit shells or underground workings that have generally less than 1,000 m in diameter. Some of the deposits are extensions of pits from the Cominco era that were not known or not economical at the time, while others are new discoveries.

Most of the deposits are planned to be mined as open-pits and are located in the northern and main trend. The deeper deposits with higher strip ratio are planned to be mined from underground, when the grade can support such mining method operating costs. The total mineralized material is 37.9 Mt grading 3.98% Zn and 1.44% Pb for open-pits, and 9.0 Mt grading 6.14% Zn and 3.11% Pb for underground. The total combined mineralized material is 46.9 Mt at 4.39% Zn and 1.76% Pb.

- Forty-seven deposits from four zones (East Mill, Central, North, and N204) are mined as open-pits, considering the deposit's size, shape, orientation, and proximity to the surface as well as economic parameters. Drilling, blasting, loading, and hauling are used to mine the open -pit mineralized material to meet the mine production schedule.
- Two zones involve underground mining methods: the West Zone (W1 Area) and the Central Zone (C1 Area). The West Zone (W1 Area) includes five underground workings mined by longhole methods with some stopes mined by the room and pillar method (less than 10%). Three deposits from the Central Zone (C1 Area) are mined by underground methods given their high strip ratio and good grades. These deposits have portal access through the wall of mined-out pits M64 (historical) and M67(proposed).

Figure 16-1 shows the 47 open-pits and six underground working locations.

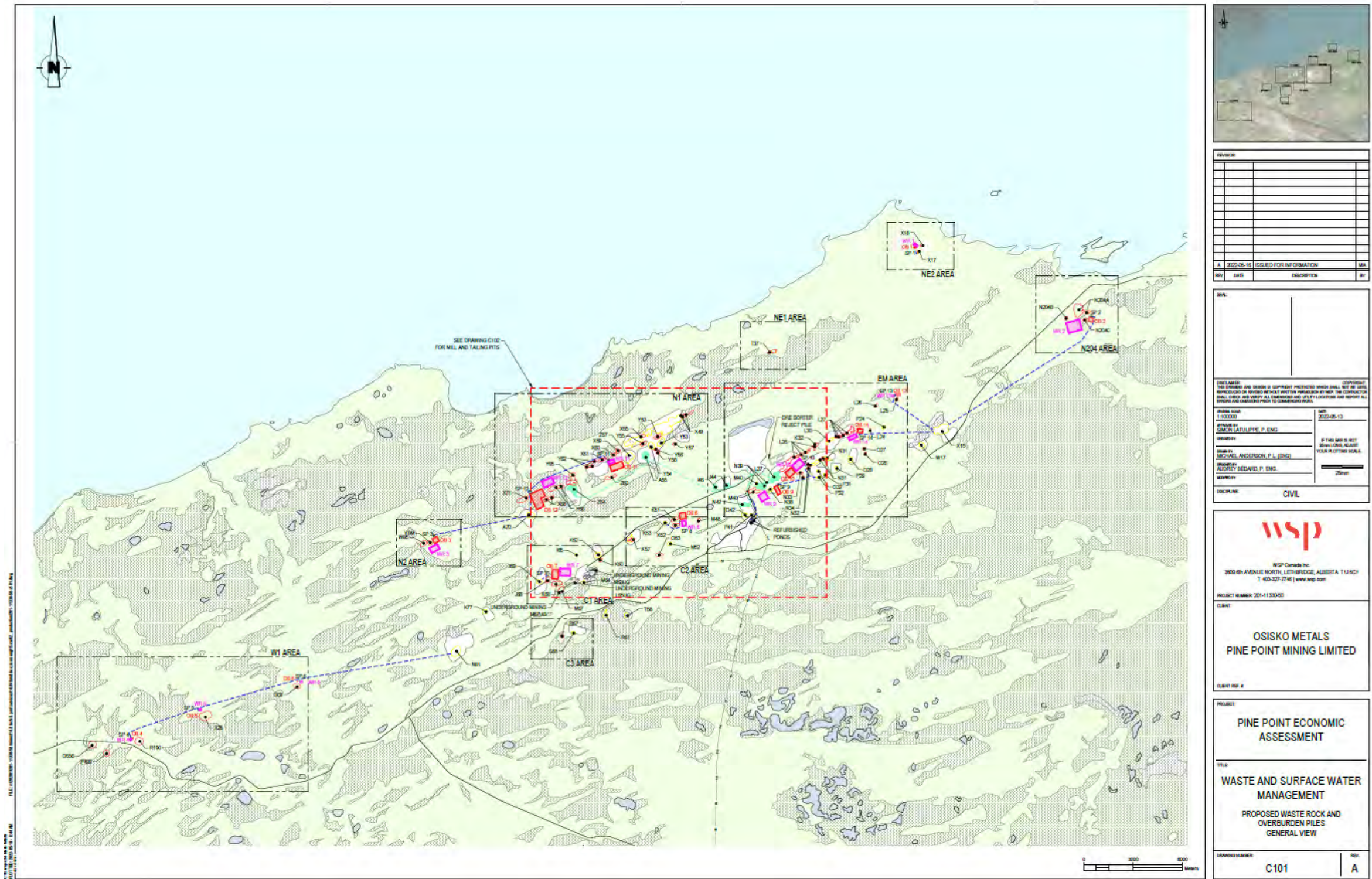


Figure 16-1: Map of the Pine Point Project deposits



16.2 Geotechnical Considerations

No geotechnical investigation was conducted for this PEA. Additional investigations are recommended for the next phase of the Project development. It should be noted that during the Cominco era, 50 open-pits and some underground workings were operating without noticeable issues.

16.2.1 Open-pit Geotechnical Considerations

No geotechnical study was available for open-pit mining. The parameters used in this PEA for open-pits are conservative at a 45° overall slope angle considering that the pits are shallow, but this takes into consideration the ramp and the berm widths. Further geotechnical investigation before the next phase of the Project development is required for the bigger pits.

16.2.2 Underground Geotechnical Considerations

A geotechnical review for the West Zone (W1) was conducted in 2011-2012 by AMC Consultants and provided rock mass classification, stability assessments, and ground support guidelines (AMC, 2012). Underground design recommendations are based entirely on this report and most of the geotechnical information is taken from this review.

16.2.2.1 Underground Rock Mass Classification

Based on AMC Consultants' geotechnical review (AMC, 2012), upper and lower bound values of Q' and RMR are summarized in Table 16-1.

Table 16-1: Comparison of rock mass rating values (AMC. 2012)

Rock Group	Q' Average		RMR Upper Bound (0.3m-1m spacing, rating=20)		RMR Lower Bound (0.05m-0.3m spacing, rating=10)	
	Q' logged	$RMR_{eq} = 9 \ln Q + 44$	$Q_{eq} = \exp[(RMR - 44)/9]$	RMR logged	$Q_{eq} = \exp[(RMR - 44)/9]$	RMR logged
Hay River Shale	4.5 (Jn=6) <i>fair</i>	57 <i>fair</i>	1.1 <i>poor</i>	45 <i>fair</i>	0.3 <i>very poor</i>	34 <i>poor</i>
Slave Point Limestone	11.5 (Jn=9) <i>good</i>	66 <i>good</i>	5.9 <i>fair</i>	60 <i>fair-good</i>	1.9 <i>poor</i>	44 <i>fair</i>
Watt Mountain Shale	4.3 (Jn=12) <i>fair</i>	57 <i>fair</i>	2.9 <i>poor</i>	54 <i>fair</i>	0.9 <i>very poor</i>	44 <i>fair</i>
Pine Point Host/Ore Body	5.9 (Jn=9) <i>fair</i>	60 <i>fair-good</i>	3.4 <i>poor</i>	55 <i>fair</i>	1.1 <i>poor</i>	45 <i>fair</i>
E-Facies	5 (Jn=6) <i>air</i>	58 <i>fair</i>	2.4 <i>poor</i>	52 <i>fair</i>	0.8 <i>very poor</i>	42 <i>poor-fair</i>

The rock mass classification values above give a general indication of the relative rock mass quality and competency that would be encountered during drive development and mining. For example, the Slave Point Limestone Q-RMR values indicate a competent rock, with fair to good rock mass quality overall.

16.2.2.2 Stopes Dimensions

Mineralization in the West Zone has an average height of 50 m, a strike length of 240 m, and a width of 120 m. To optimize the mineralized material extraction and reduce the number of top sills, stope sizes have been slightly modified from what AMC Consultants recommended in their 2012 report. For design purposes, instead of 30 mH x 10 mW x 25 mL stopes, lower and wider stopes were chosen with a dimension of 25 mH x 15 mW x 25 mL. The Mathews-Potvin stability method was applied to confirm the stability of this new stope dimension. The stability analysis results are shown on Figure 16-2.

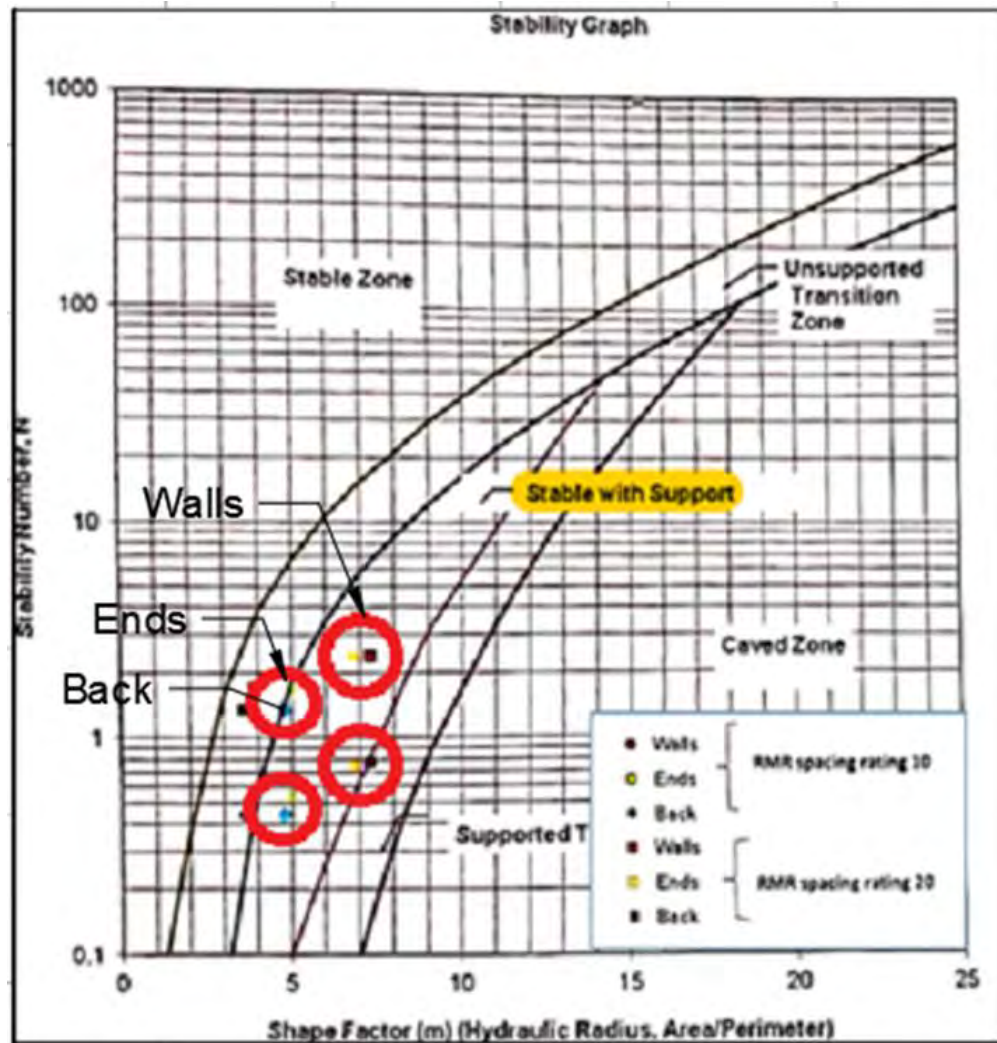


Figure 16-2: Stability graph results – West Zone longhole stopes with the new proposed stope geometries

From the stability graph, it can be noted that even for the worst-case scenario, i.e., the RMR spacing rating 10, stope walls remain in the Stable zone with Support. This area refers to the zone where ground support is required to ensure stope stability during stopes life. From the stability graph, this stope dimension is retained for the underground mining design in the West Zone.

16.2.2.3 Crown Pillar Thicknesses

The crown pillar thickness will require further assessment that will be carried out in the next phase of the Project.

16.2.2.4 Room and Pillar Stopes

To complement the longhole stoping method for the West Zone, the room and pillar method is used. It is also the main mining method for the C1 Area. The proposed stope dimension for the room and pillar mining method is based on geotechnical parameters and the rock mass condition (see Figure 16-3).

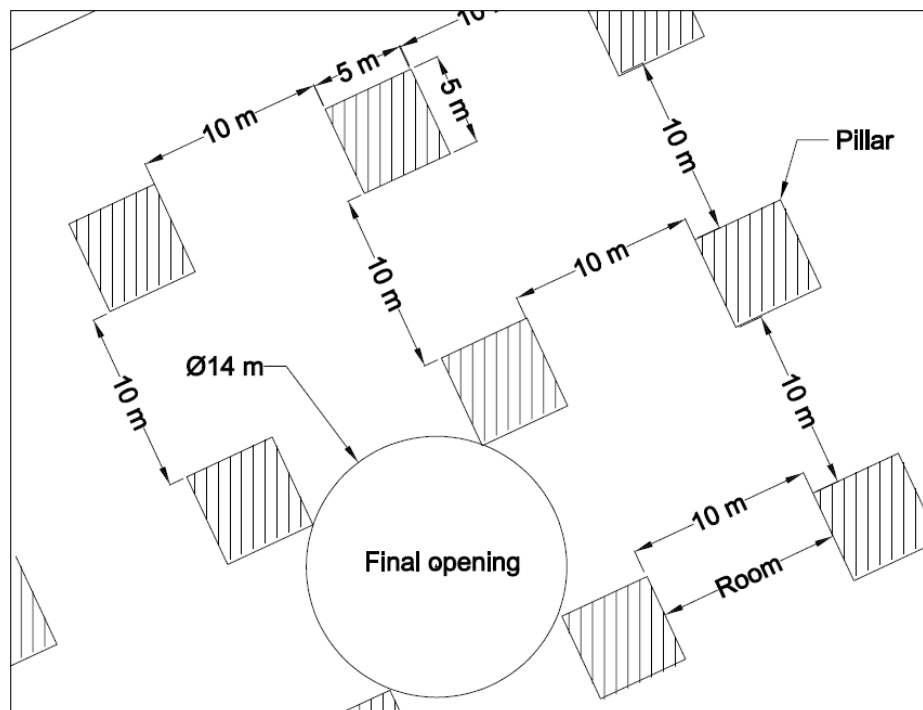


Figure 16-3: Plan view of the room and pillar configuration

The height of the rooms and pillars is 5 m and therefore the same as that of the development headings. This allows for the use of the same equipment as was used in the West Zone. Mining is done in stages. The first step is a development cut (5 m x 5 m), in the mineralized zone. During this primary development phase, the support is installed as proposed by AMC (2012). While advancing the secondary cut (5 m x 5 m), secondary ground support is installed in the back to retain the wider 10m span of the rooms. The proposed support consists of rebars of 3 m (10') installed on a 1.5 m x 1.5 m pattern. This support must be installed as the development progresses. The last cut is mined by wall slash, retreating towards the accesses, which does not expose the workers to the final opening of 14 m in diameter.

Historically, mined deposits from the Cominco era also used the room and pillar method (see Figure 16-4). This methodology will be investigated in more detail at the next phase of the Project development.

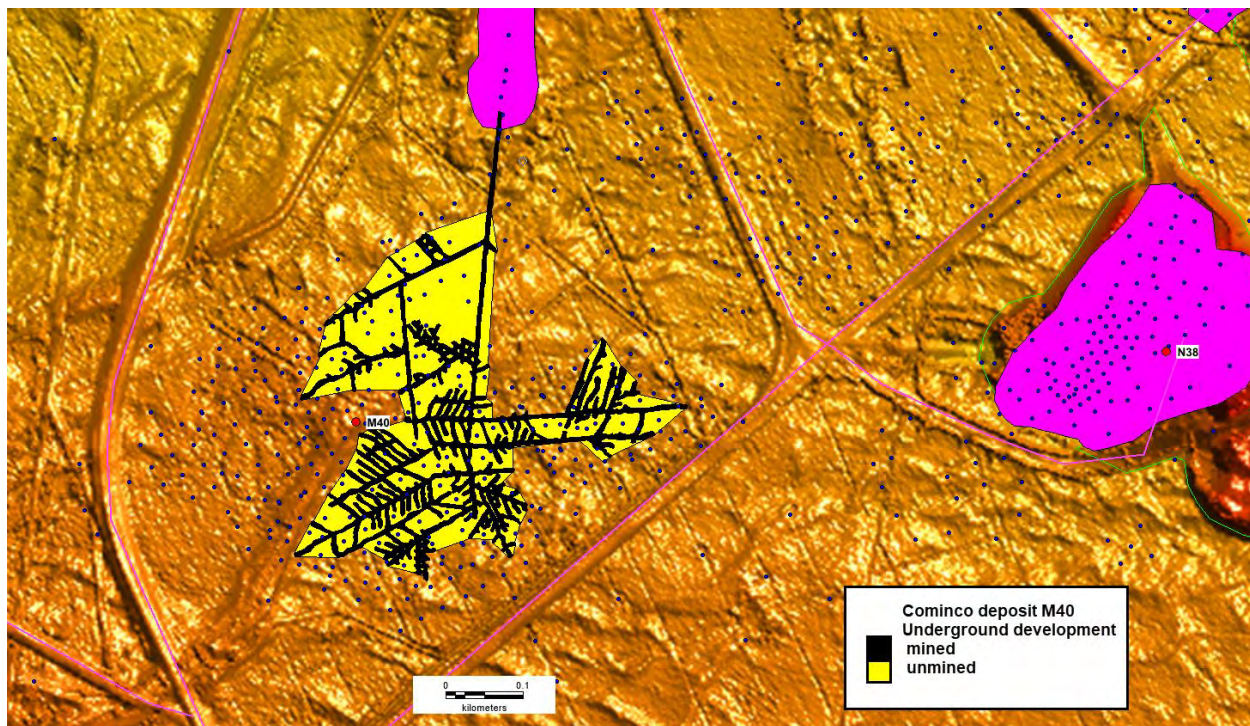


Figure 16-4: Underground mine development from Cominco Era

16.2.2.5 Ground Support

The geotechnical review (AMC, 2012), provided estimates of ground support requirements for different underground excavations in the various rock units. These estimates have been evaluated and validated by WSP. The ground support systems discussed below will be installed for both zones, regardless of the mining method used. Additional clarifications are made for the room and pillar method.

16.2.2.6 Development Headings

The NGI-Q ground support chart method was used to provide an approximation of ground support requirements for long-term development in the major lithologic units. The assumptions made are that the development headings are 5 m x 5 m, and an excavation support ratio (ESR) (Hutchinson and Diederichs, 1996) of 1.6 for permanent mine openings is applied. The results are shown in

Figure 16-5 and Figure 16-6 below. A range of ground support types has been estimated from the expected variability in ground conditions. For mine design and cost estimation purposes, an average ground support system for development drifts was decided (see Table 16-2 for excavation type vs. ground support system).

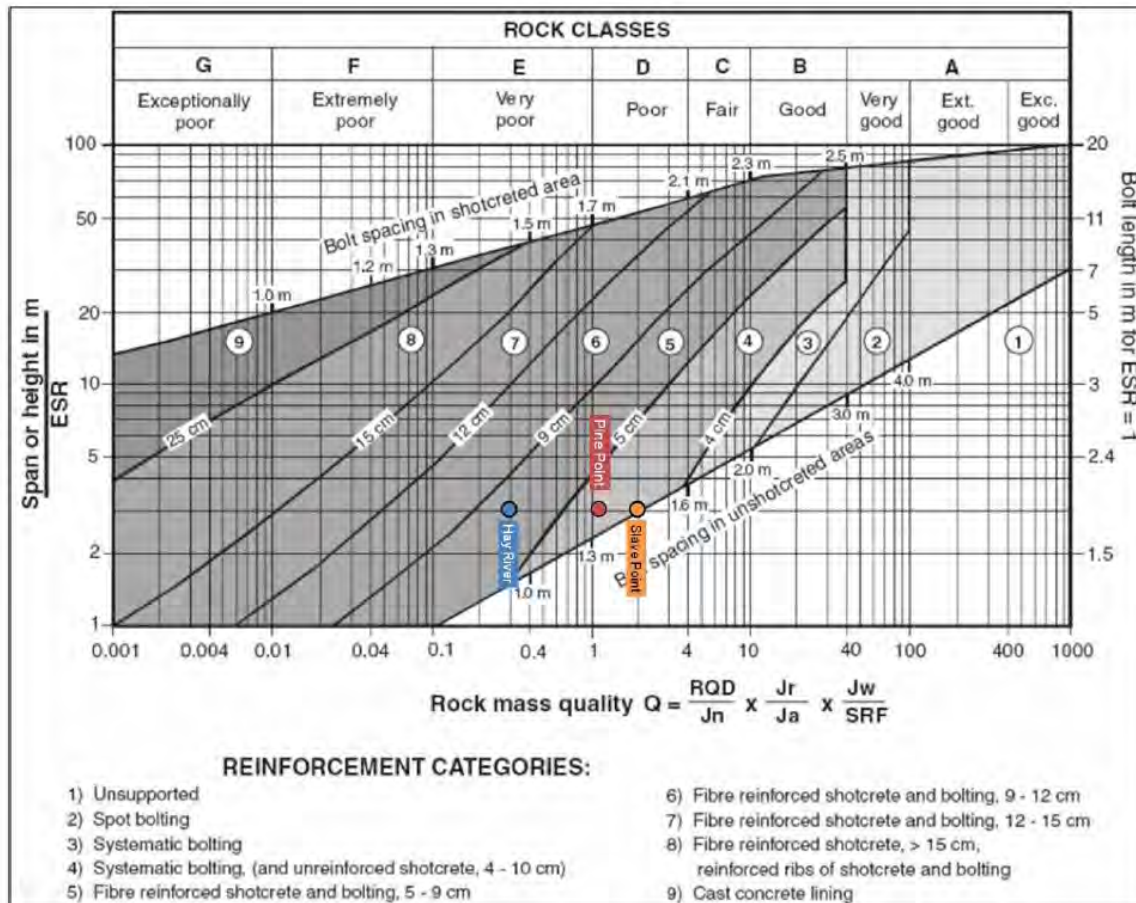


Figure 16-5: Ground support – RMR spacing rating 10

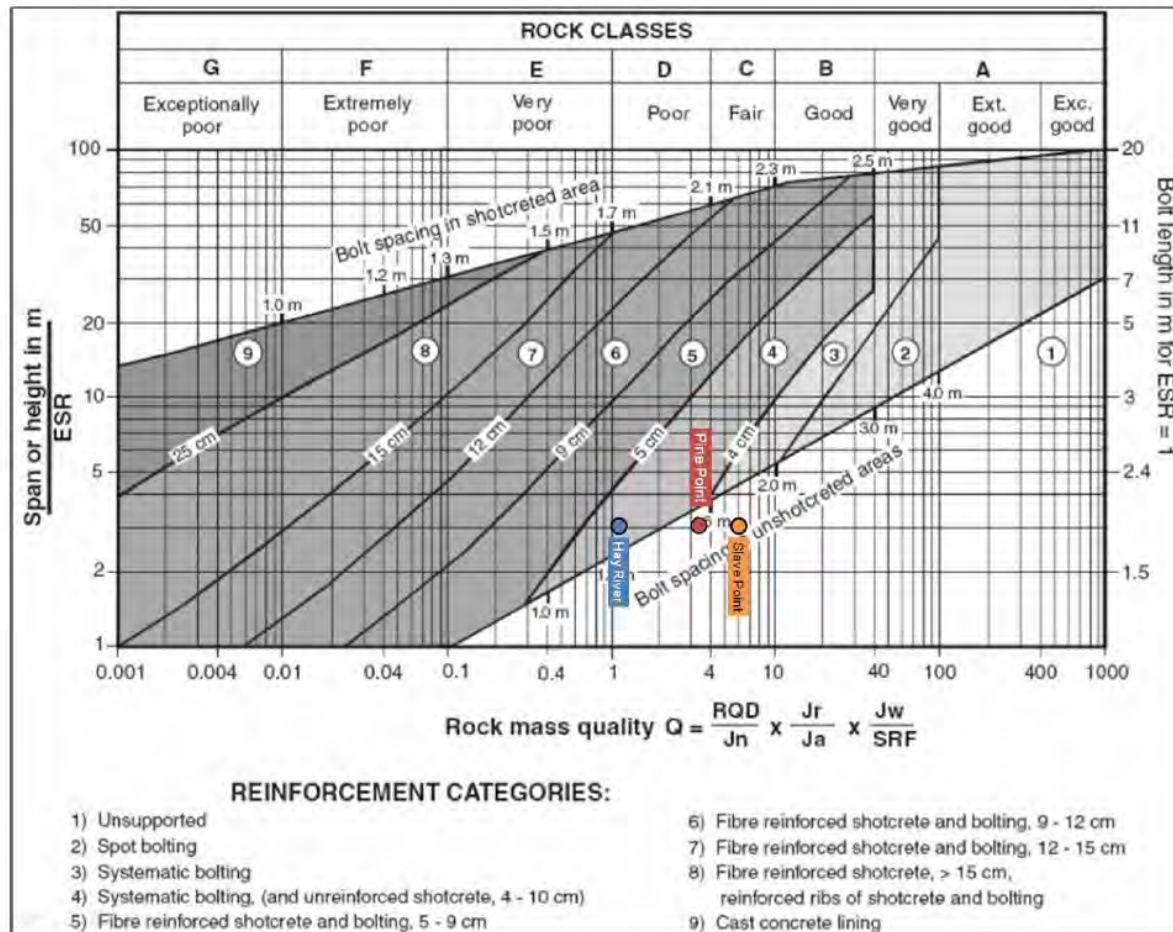


Figure 16-6: Ground support – RMR spacing rating 20

Table 16-2: Excavation type vs. Ground support system

Waste Lateral Development (m)	Dimension	Ground Support
Ramp	5 m x 5 m	8' rebars pattern 1.2 x 1.2 m with mesh on roof and walls
Sublevel access	5 m x 5 m	8' rebars pattern 1.2 x 1.2 m with mesh on roof and walls
Drilling access Drift	5 m x 5 m	8' rebars pattern 1.2 x 1.2 m with mesh on roof and walls
Haulage Drift	5 m x 5 m	8' rebars pattern 1.2 x 1.2 m with mesh on roof and walls
PS Drift	5 m x 5 m	8' rebars pattern 1.2 x 1.2 m with mesh on roof and walls
Room and pillar drift	5 m x 5 m	8' rebars pattern 1.2 x 1.2 m with mesh on roof and walls
Ventilation Access	5 m x 5 m	8' rebars pattern 1.2 x 1.2 m with mesh on roof and walls
Gear Bay	5 m H x 8 m W	8' rebars pattern 1.2 x 1.2 m with mesh on roof and walls + 10' rebar pattern 1.8 x 1.8 m as back secondary support

A typical ground support standard as illustrated on figure 167 was selected for the mine design and costing purposes.as an average ground support system for development shown in Figure 16-7.

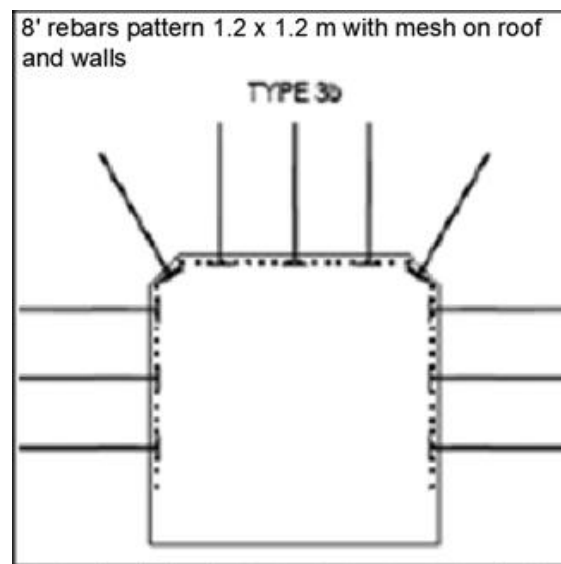


Figure 16-7: Typical development drift ground support layouts

16.2.2.7 Sills Ground Support

Stope entry development is generally deemed a short-term mine opening and an ESR of 2 has been applied to the ground support charts. Depending on the longevity requirements of the drifts and the ground conditions encountered, AMC recommended that planned support of stope entry excavations is achieved using both cement grout single strand cable bolts and split set or Swellex bolts for surface support. Since the width of the stopes has increased from 10 m to 15 m, a re-evaluation of the secondary support at the stopes back will be made. For a width of 15 m and an RMR of 60, 9 m long double-strand cables on a pattern of 2 m x 2 m must be installed at the upper stope access (top sill). Preliminary estimates of ground support for the upper and lower stope access drifts are summarized in Table 16-3.

Table 16-3: Sills ground support

Stope Drift	Primary Support
Upper Stope Access	Split Sets/Swellex L=2.5m; S=1 to 1.5m c/c
Lower Stope Access	Split Sets/Swellex L=2.5m; S=1 to 1.5m c/c

16.3 Hydrogeological Settings

Three principal carbonate bedrock units are present beneath the overburden and Hay River Shale at the Site. These include the Slave Point Formation of limestone, the Sulphur Point Formation (limestone), and the underlying Pine Point dolomite. A highly porous alteration zone is locally present, replacing a portion of the Sulphur Point, the Pine Point, Watt Mountain, and Lower Slave Lake Formations. This zone is referred to as the hydrothermal dolomitization ("HTD") alteration zone. The presence of the HTD alteration zone also greatly enhances porosity and permeability. The HTD alteration zone comprises the known "aquifer" unit in the Main and North Trends.

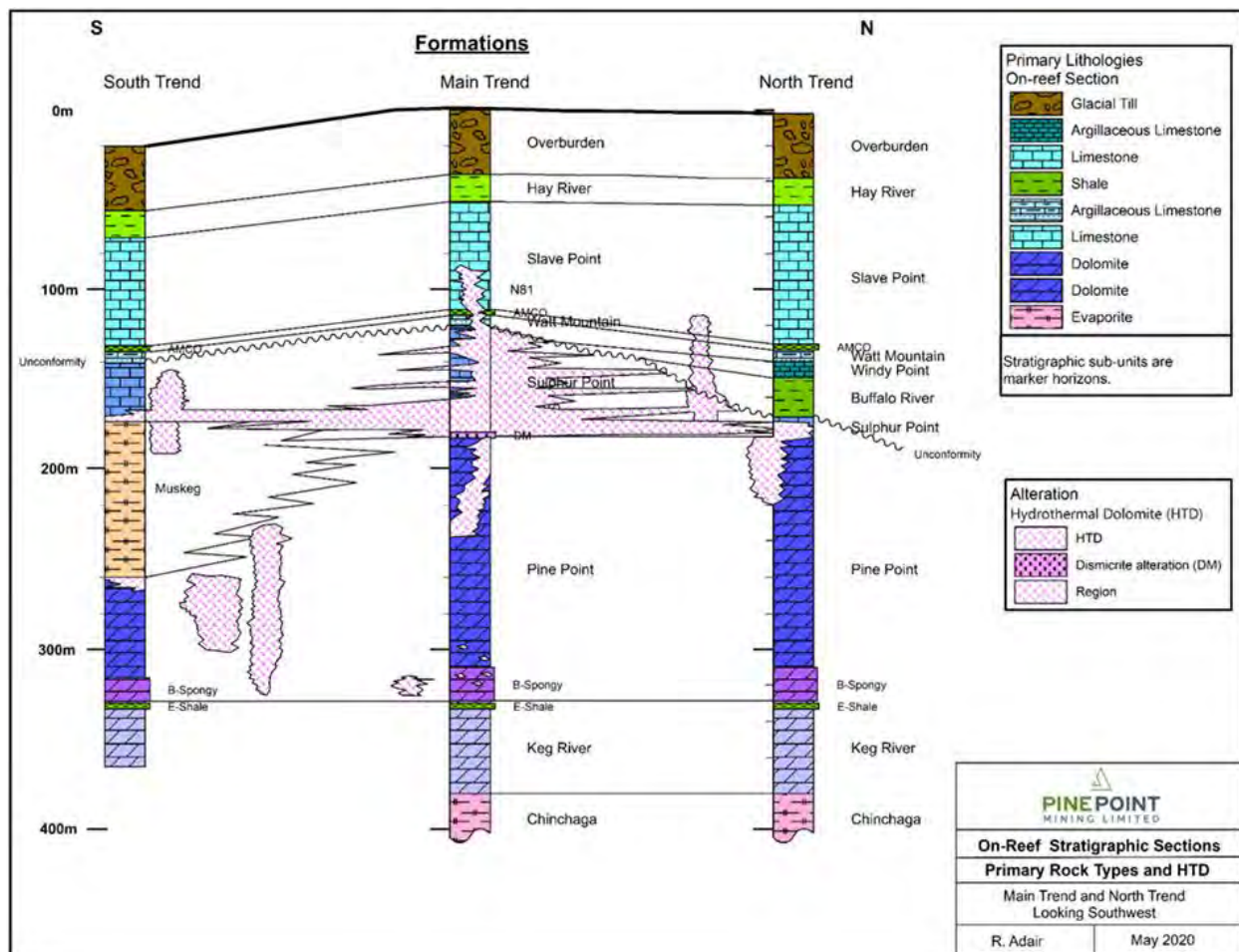


Figure 16-8: Pine Point geological formations

Hydraulic Conductivity ("K") was measured with various testing processes at site. Overall information is suggesting that bulk K values are ranging on the low 10⁻⁶ m/s and the main

water-bearing units (such as faults) are in the mid 10-5 m/s range. There are no obvious variations of bulk K values within all the units, but there are variations with K values in the faults.

Since topography is slightly sloped trending towards the NNW, the groundwater flow direction also follows that trend, discharging groundwater into the Great Slave Lake, and sometimes, transiting through rivers and streams. Since the Prismatic deposits are located adjacent and within faulted/collapsed zones, these geological contexts will likely induce higher inflow because K values associated to faults are higher by approximately one order of magnitude. Historical observations of multiple non-productive wells agree with recent field testing and interpretation: wells located in the bulk aquifer are much less productive.

16.4 Open-pit Summary

Nine areas – namely EM, C1, C2, C3, N1, N2, N204, NE1, and NE2 – were selected to be mined by open-pit methods. In order to do so, a pit optimization process was performed on each separate area to generate pit shells per deposit. A block model was provided for each area with relevant data such as tonnages, grades as well as the most recent updated topographic files from LiDAR. In addition, some economic parameters were updated as an input form Osisko Metals. WSP validated and updated economic parameters to run pit optimizations. Table 16-4 summarizes the updated pit optimization parameters used in this 2022 PEA.

The block models that were used in this 2022 PEA are the same as the block models used in the previous 2020 PEA, without change.

After selecting each final pit shell, a detailed mineralized inventory was drawn up, which was used for the mine production schedule.

16.4.1 Pit Optimization

16.4.1.1 Data Validation

The block models were received from BBA as percent block models. WSP used the block models provided by BBA to conduct pit optimizations in open-pit areas and to run mineable stope optimizer (MSO) tool in underground areas.

For open-pit areas, the mineral resources were constrained into pit shells with parameters suitable for mineral inventory estimations. The block models were adapted and imported into Whittle™ software to assess the Revenue Factors ("RF"). This process slightly modified the initial mineral resource estimates, but differences were less than 5%.



For all open-pit and underground areas, WSP used a sub-block model provided by Osisko Metals, which was built from the original percent block model provided by BBA. The use of a sub-block model was dictated by the practicality to run mine production schedule in Deswik Scheduler and mineable stope optimizer (MSO) tool in Deswik which is more convenient to run on a sub-block model than on a percent model. WSP compared the mineral results on both sub-block and percent models and the difference in less than 1%.

For C1 underground area, no change has been made to the block model since 2020 PEA. WSP Updated Underground workings in C1 based on Updated pit shells from C1 open-pit shells.

16.4.1.2 Pit Optimization Parameters

For mineral inventory, and eventually mineable mineral resources, WSP updated some parameters to perform pit optimization runs in Whittle™. The agreed parameters with Osisko Metals for those runs are presented in Table 16-4.

Table 16-4: Pit optimization parameters

Element	Unit	Criteria
Material Density		
Overburden Mineral (OB)	t/m ³	2.00
Waste rock	t/m ³	2.71
Mineralized material	t/m ³	Variable per block (formula)
Pre-concentrate mineralized material	t/m ³	3.00
Mineral sorter rejects	t/m ³	2.71
Block model names		Block size (X,Y,Z)
East Mill (EM)	m	10m, 10m, 5m
Area C1	m	10m, 10m, 5m
Area C2	m	10m, 10m, 5m
Area C3	m	10m, 10m, 5m
Area N1	m	10m, 10m, 5m
Area N2	m	10m, 10m, 5m
Area NE1	m	10m, 10m, 5m
Area NE2	m	10m, 10m, 5m
Area W1	m	10m, 10m, 5m
Area W2	m	10m, 10m, 5m
Area N204	m	10m, 10m, 5m



Element	Unit	Criteria
Pit Design		
Overburden overall slope	°	26
Rock slope (bench)	°	N/A
Rock slope (inter-ramp)	°	N/A
Rock overall slope - rock (global)	°	45
Ramp width - one lane	m	N/A
Ramp width - double lane	m	N/A
Bench height (pit design)	m	10
Bench height - mining schedule for waste removal	m	10
Bench height - mining schedule for mineralized material removal	m	5
Berm per bench (number of berms per number of benches)	m	N/A
Berm width	m	N/A
Geotechnical berm (likely each 100 m down)	m	N/A
Mining Method		
Mine operations performed by	-	Owner or Contractor
Daily production	tpd	6,000 to 10,000
Maximum daily production (All material - including mineralized material and waste)	tpd	70,000
Major loading equipment - mineralized material	-	7 m ³ bucket capacity
Major loading equipment - waste		12 m ³ bucket capacity
Major hauling equipment	-	100t Class Truck
Dilution	%	12
Dilution width above a block	m	0.4
Dilution width below a block	m	0.4
Powder factor	kg/t	0.30
Dilution grade	% Zn Eq	0.34
Mining recovery	%	95
Swell factor	-	1.30
Material carry-back	%	0
Mineralized material moisture content	%	2
Cut-off grade on ZnEq	%	2.2



Element	Unit	Criteria
Operating Costs/Parameters (for pit optimization only)		
Mining cost (mineralized material)	\$/t	3.85
Mining cost (waste rock)	\$/t	3.58
Mining cost (overburden)	\$/t	2.20
Dewatering	\$/t	0.48
Mineralized material hauling additional cost (from pit exit to mill or mineral sorter)	\$/t-km	0.13
Distance EM to processing plant	km	5
Distance C2 to processing plant	km	5
Distance C1/C3 to processing plant	km	10
Distance N1 to processing plant	km	10
Distance N2 to processing plant	km	20
Distance NE1 to processing plant	km	10
Distance NE2 to processing plant	km	20
Distance W1/W2 to processing plant	km	40
Mineral Sorting cost (\$/t)	\$/t	\$8.50
Cash flow discounted rate	%	8
Processing Costs, Transportation and G&A		
Processing costs	\$/t milled	\$22.52
Processing Cost	\$/t Ore	13.40
General and administration costs (\$/t milled)	\$/t milled	\$14.12
Recoveries		
Overall zinc (all block models except N204)	%	87
Overall lead (all block models except N204)	%	92
Overall zinc (block model N204 only)	%	82
Overall lead (block model N204 only)	%	85
Mineral Sorter mass pull	%	59.5
Zinc concentrate grade	%	59
Lead concentrate grade	%	64
Zinc concentrate moisture content	%	8
Lead concentrate moisture content	%	8
Sales and Revenues		
Zn	USD/lb	1.30
Pb	USD/lb	1.00
Exchange rate USD/CAD	USD/CAD	1.31



Element	Unit	Criteria
Payables and Selling Costs		
Zn payability	%	85
Pb payability	%	96
Royalties (% NSR)	%	3
Transportation losses (%)	%	0.2
Zinc transport from Mill to Pine Point junction	\$/dmt Zn Conc.	33
Lead transport from Mill to Pine Point junction	\$/dmt Pb Conc.	33
Zinc concentrate freight and marketing	\$/wmt Pb Conc.	198
Lead concentrate freight and marketing	\$/wmt Zn Conc.	223
Zinc concentrate smelter cost	\$/wmt Pb Conc.	295
Lead concentrate smelter cost	\$/wmt Zn Conc.	262
Revenue Factor per Area		
EM	-	1
C1	-	1
C2	-	1
C3	-	1
N1	-	1
N2	-	1
NE1	-	1
NE2	-	1
N204	-	1
Dilution Grade		
Waste dilution Zn	%	0.11
Waste dilution Pb	%	0.23

16.4.1.3 Pit Optimization Software

Economic pit shells were determined using Geovia's Whittle™ software that uses the Lerchs-Grossmann ("LG") algorithm and the Pseudoflow algorithm. Pseudoflow uses the same algorithm as Lerchs-Grossman but runs faster than the Lerchs-Grossman. In this PEA, all shells were generated in Whittle™ software with the Pseudoflow algorithm. The Pseudoflow algorithm progressively identifies economic blocks, considering waste rock stripping, that results in the highest possible total value mined within the open-pit shell, subject to the specified pit slope constraints.

For each block model, Whittle™ generated multiple pit shells, considering economic parameters as well as geotechnical slopes and dewatering constraints as shown in Table 16-4: Pit Optimization Parameters.

16.4.1.4 Pit Optimization Geological Block Model

For each area, the block model provided by BBA was a Multi-Ore Percent. In other words, for each deposit, each single block of 10 m * 10 m * 5 m (X*Y*Z) has attributes for overburden, mineralized material, and waste rock. For pit optimization purposes, WSP coded the topography information in each block model per area in Surpac™, before importing the block model to Whittle™.

To maintain data accuracy, Whittle™ 4.7.3 was set up to run pit optimization directly from block model data input without any transformation or re-blocking. Specifically, the multi-ore percent option was set up in Whittle™ to keep the same layer configuration in each block, thus allowing a better approach for dilution than single mineralized material block configuration.

Each geological block model per area was used as an input to Pseudoflow runs. In addition, other economic and operational variables were used, such as pit slope angles per rock type material, mining costs, processing costs, selling costs, metal prices, and other variables listed in Table 16-4. Although these parameters are not necessarily final, a reasonable degree of accuracy is required, since the analysis is an iterative process.

16.4.1.5 Overall Pit Optimization Slope Angle

For pit optimization, WSP set up all rock material at an overall slope angle of 45°, and all overburden material at an overall slope angle of 26.6°. These conservative estimates are based on previous experience. As additional geotechnical data become available, pit slopes could potentially steepen and improve the mine plan and the economic assessment study.

The overall slope angle considers the bench face angle, the berm width, and the ramp.

16.4.1.6 Pit Optimization Operating Costs

Mining costs were set up at \$3.85/t for mineralized material and \$3.58/t for waste rock. Overburden mining cost was estimated at \$2.20/t.

The 2022 PEA mining costs were updated from the 2020 PEA mining costs by an increase of 10%, as an input from Osisko Metals. Mining Cost updates are based on benchmarking analysis and databases for similar types of operations.

16.4.1.7 Pit Optimization Cut-Off Grades

WSP used the cash-flow mode in Whittle to run Pit Optimization. The cash-flow mode interrogates each block and calculates the net income based on costs and revenues. As a result, it maximizes the cash flow. Since two grades (zinc and lead) are included in the optimization process, WSP did not constrain the optimization process on any cut-off, but rather selected the optimal pit shells based on maximum NPV. Imposing a cut-off grade on zinc and lead simultaneously could discard some economic blocks with a Zn equivalent grade that can be processed to the mill.

To make sure mineralized material grades that are contained in selected pit shells can be processed to the mill, WSP recalculated the zinc equivalent grade based on formulae provided by the client. For each selected pit shell, WSP made sure that the average zinc equivalent grade was well above the calculated break-even zinc equivalent cut-off grade, which is derived from the break-even cut-off grades for zinc and the break-even cut-off grades for lead as mentioned in the mineral resource parameters provided by BBA.

16.4.1.8 Pit Optimization Dilution and Mining Recovery

Dilution of 12% on mineralized material was considered in the mineralized material inventory. This is an average estimate determined by WSP for all deposits.

Waste rock dilution grade was set to 0.23% for zinc and 0.11% for lead.

Mining recovery of mineralized material was set to 95% based on experience for these types of deposits.

16.4.1.9 Pit Optimization Results

The pit optimization process resulted in a series of nested pit shells, each corresponding to a RF. The RF scales the metal price only, and no costs are factored by the RF. In this PEA, it was agreed with the client to use a RF of 1, which corresponds to a zinc price of US\$1.30/lb and a lead price of US\$1.00/lb.

Table 16-5 summarizes pit optimization results for N1 Area. For a RF = 1, the pit shell generated by Whittle™ contains roughly 16.4 Mt of mineralized material and 82.3 Mt of waste rock for a total mined tonnage of 98.7 Mt and a strip ratio of 5.03.



Table 16-5: Pit optimization results summary for N1 Area

N1 MC= 3.85\$/t mineralized material, 3.58\$/t waste rock, with 12% dilution on mineralized material, with dewatering cost										
Nested Pits	RF	1.00 Pb	1.30 Zn	Rock (W+OB+MIN.)	Mineralized Material	Strip	PBG	ZNG	ZNEQ	Waste+OB
		US\$/lb	US\$/lb	Tonnes	Tonnes	Ratio	Grade	Grade	Grade	Tonnes
1	0.5	0.5	0.65	5,387,511	893,743	5.03	2.7695	8.7307	11.504	4,493,768
2	0.6	0.6	0.78	23,831,539	3,944,305	5.04	2.3281	5.8468	8.178	19,887,234
3	0.7	0.7	0.91	42,869,224	7,109,657	5.03	1.9329	5.0004	6.936	35,759,567
4	0.8	0.8	1.04	62,028,028	10,390,713	4.97	1.6744	4.4008	6.078	51,637,315
5	0.85	0.85	1.105	74,366,586	12,429,551	4.98	1.5575	4.1262	5.686	61,937,035
6	0.9	0.9	1.17	83,012,893	13,932,702	4.96	1.4706	3.9592	5.432	69,080,191
7	0.95	0.95	1.24	92,088,485	15,266,718	5.03	1.406	3.836	5.244	76,821,767
8	1	1	1.3	98,714,800	16,365,611	5.03	1.3575	3.7272	5.087	82,349,189
9	1.05	1.05	1.37	104,249,131	17,311,187	5.02	1.3186	3.6344	4.955	86,937,944
10	1.1	1.1	1.43	114,947,864	18,668,586	5.16	1.2754	3.5316	4.809	96,279,278
11	1.15	1.15	1.50	125,100,571	20,092,029	5.23	1.2287	3.4216	4.652	105,008,542
12	1.2	1.2	1.56	128,542,749	20,713,654	5.21	1.2074	3.3698	4.579	107,829,095
13	1.3	1.3	1.69	139,818,319	22,199,429	5.3	1.166	3.2667	4.434	117,618,890
14	1.4	1.4	1.82	149,989,884	23,513,516	5.38	1.1268	3.1786	4.307	126,476,368
15	1.5	1.5	1.95	156,112,812	24,427,436	5.39	1.1007	3.1129	4.215	131,685,376

Note: MC - mining cost

Table 16-6 shows a revenue factor table for the N1 Area. The table shows that the maximum discounted cash flow is obtained for a RF of 0.9 (Pit #6) as illustrated on Figure 16-9 in red text. It should be noted that all areas show a maximum discounted cash flow reached for RF equal or close to 1. To remain consistent in the evaluation of mineralized material tonnages, all areas will be set to a RF of 1 as mentioned above.

Table 16-6: Discounted cash flow @ 8% per revenue factor for N1 Area

RF	Final Pit	Cashflow (CAD disc.)			Tonnes		Mine Life (years)		
		Best Case	Specified Case	Worst Case	Mineralized	Waste	Best Case	Specified Case	Worst Case
0.5	1	119,679,097	119,679,097	119,679,097	1,097,901	4,289,610	0.3	0.3	0.3
0.6	2	304,004,147	303,987,492	303,987,492	4,436,798	19,394,741	1.2	1.2	1.2
0.7	3	405,836,038	389,981,720	389,981,720	7,800,388	35,068,835	2.1	2.5	2.5
0.8	4	458,669,198	428,239,959	415,886,383	10,920,768	51,107,260	3.0	3.5	3.7
0.9	5	478,779,974	438,617,381	421,284,983	12,860,303	61,506,282	3.5	4.1	4.3
0.9	6	487,447,530	442,418,001	421,403,038	14,191,604	68,821,290	3.9	4.5	4.7
1	7	491,560,968	440,329,921	416,671,477	15,396,415	76,692,071	4.2	4.8	5.1
1	8	492,460,160	436,270,791	409,324,545	16,365,611	82,349,189	4.5	5.1	5.4
1.1	9	491,448,924	429,268,964	399,197,287	17,191,377	87,057,754	4.7	5.4	5.8
1.1	10	486,960,088	412,321,476	378,774,712	18,382,299	96,565,565	5.0	5.9	6.3
1.2	11	480,778,490	397,582,676	360,609,276	19,644,435	105,456,136	5.4	6.4	6.8
1.2	12	477,836,433	391,571,225	353,294,745	20,128,636	108,414,112	5.5	6.5	6.9
1.3	13	466,885,522	372,470,493	332,207,378	21,291,500	118,526,820	5.8	6.9	7.3
1.4	14	455,235,004	353,785,195	310,259,965	22,219,571	127,770,313	6.2	7.2	7.7
1.5	15	446,775,504	341,160,557	295,727,756	22,771,795	133,341,017	6.4	7.4	7.8

A basic schedule was applied to the nested pit shells to produce a 'pit-by-pit' graph. An objective of the pit-by-pit graph is to illustrate the impact of scheduling on the pit shells and to provide guidance on the selection of an optimum pit shell to use as a guide in the detailed pit design. The optimum pit limit is chosen by estimating the pit size where an incremental increase in pit size does not significantly increase the pit mineral resource and where the economic return starts to decline. The selection of pit shells was based on the entire area generating multiple pits.

Figure 16-9 illustrates the pit-by-pit graph for N1 area as it has the most open-pits and is representative of the general mining area, generated for the Base Case deposit. Three schedules are represented:

- The Best-Case schedule consists of mining nested Pit Shell 1, the smallest nested pits, and then mining each subsequent nested pit shell from the top down, before starting the next pit shell. This schedule is seldom feasible because the pushbacks are usually too narrow. Its usefulness lies in setting an upper limit to the achievable Present Value ("PV");
- The Worst-Case schedule consists of mining each bench completely before starting on the next bench. The usefulness of this schedule lies in setting a lower limit to the Present Value. If, as is sometimes the case, worst-case and best-case schedules differ by only a few percent



points then, for that nested pit, the mining schedule is relatively unimportant from an economic point of view;

- If, as is usually the case, the difference between Worst and Best Case is significant, a more realistic mining schedule can be approximated, between the two extremes, by specifying a schedule of pit outlines to push back to - this is the Specified Case. Chosen pushbacks should satisfy mining constraints and produce a PV curve that is as close as possible to the Best-Case PV curve.

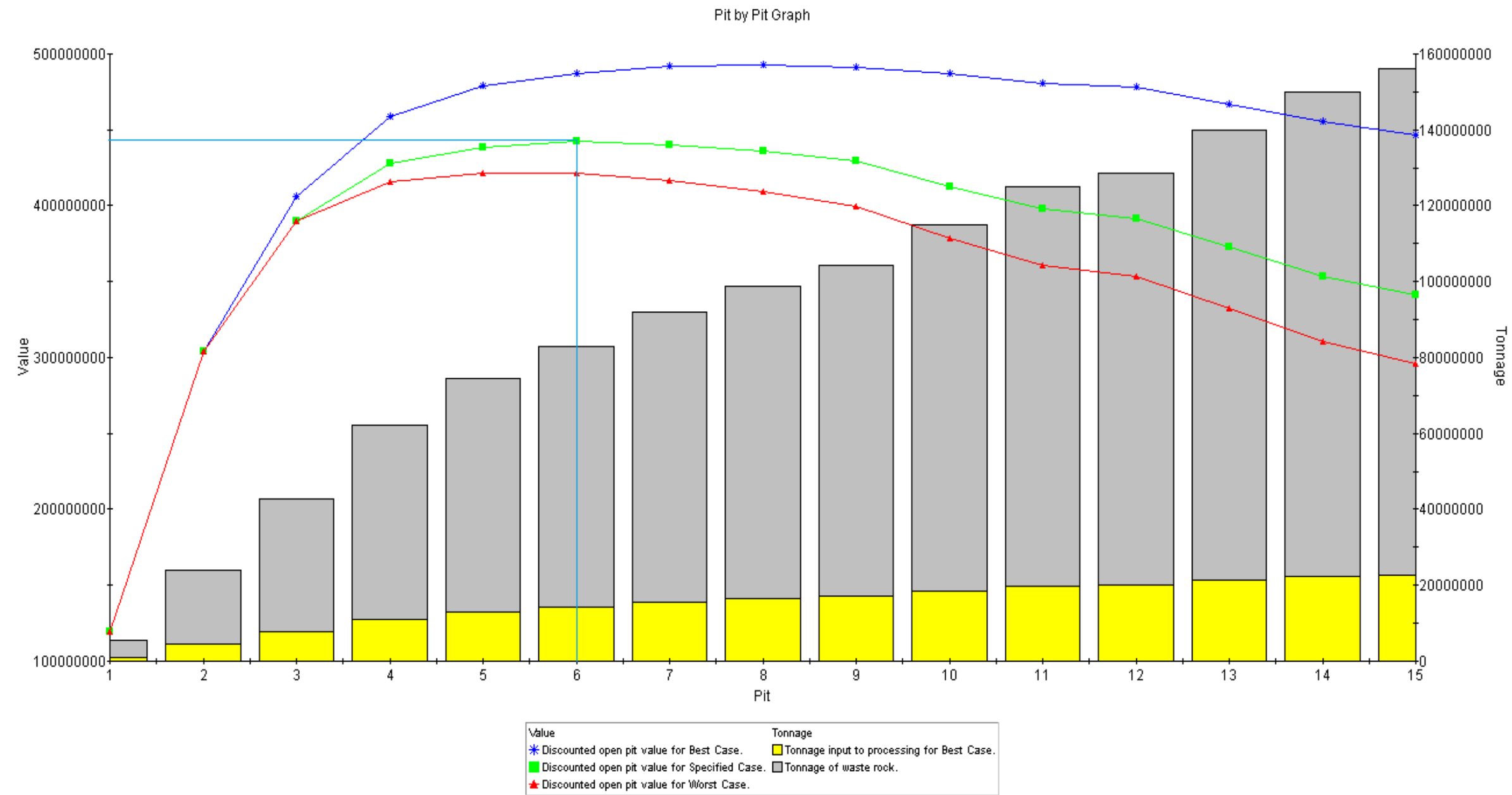


Figure 16-9: Discounted cash flow graph @ 8% for N1 Area





As stated above, all other areas have been run with similar settings. Table 16-7 summarizes pit optimization results for the nine areas at a RF of 1 (same RF as used at MRE phase).

Table 16-7: Pit optimization results for each area @ revenue factor = 1

Pit	RF	Pb US\$/lb	Zn US\$/lb	Rock (ws+ob+mm)	Mineralized Material	Ratio	PBG	ZNG	Waste+OB
				Tonnes	Tonnes	Strip Ratio	Grade	Grade	Tonnes
East Mill	1	1	1.3	68,736,746	12,406,047	4.54	1.38	3.71	56,330,699
C1	1	1	1.3	47,026,821	2,982,919	14.77	1.98	5.40	44,043,902
C2	1	1	1.3	18,414,961	2,110,431	7.73	1.36	4.30	16,304,530
C3	1	1	1.3	2,213,421	463,557	3.77	2.24	7.12	1,749,864
N1	1	1	1.3	98,714,800	16,365,611	5.03	1.36	3.73	82,349,189
N2	1	1	1.3	16,928,926	2,779,038	5.09	2.49	4.38	14,149,888
N204	1	1	1.3	42,729,386	7,663,776	4.58	0.94	3.40	35,065,610
NE1	1	1	1.3	1,873,067	425,083	3.41	2.02	5.65	1,447,984
NE2	1	1	1.3	2,834,463	639,764	3.43	0.66	3.02	2,194,699
Total	1	1	1.3	299,472,591	45,836,226	5.53	1.41	3.88	253,636,365

Table 16-8 summarizes pit optimization results for all nine deposits for a range of RF between 0.5 and 1.5. For each RF, the tonnages are added for the nine deposits.



Table 16-8: Pit optimization results for all areas per revenue factor

MC= 3.85\$/t mineralized material, 3.58\$/t waste rock, with 12% dilution on mineralized material								
Rev Ftr	Pb US\$/lb	Zn US\$/lb	Rock (ws+ob+mm)	Mineralized Material	Strip Ratio	Pb Grade	Zn Grade	Waste+OB
			Tonnes	Tonnes	Cumulative	Cumulative	Cumulative	Tonnes
0.5	0.50	0.65	14,195,658	3,071,814	3.62	3.97	8.28	11,123,844
0.6	0.60	0.78	50,632,726	9,124,658	4.55	2.86	6.19	41,508,068
0.7	0.70	0.91	96,263,894	16,710,676	4.76	2.22	5.22	79,553,218
0.8	0.80	1.04	160,957,305	26,127,375	5.16	1.83	4.65	134,829,930
0.85	0.85	1.11	200,493,874	32,220,437	5.22	1.65	4.35	168,273,437
0.9	0.90	1.17	233,113,189	36,920,389	5.31	1.56	4.17	196,192,800
0.95	0.95	1.24	266,438,704	41,365,376	5.44	1.48	4.02	225,073,328
1	1.00	1.30	299,472,591	45,836,226	5.53	1.41	3.88	253,636,365
1.05	1.05	1.37	327,003,745	49,801,827	5.57	1.35	3.77	277,201,918
1.1	1.10	1.43	372,749,470	55,153,736	5.76	1.29	3.65	317,595,734
1.15	1.15	1.50	414,414,164	59,680,893	5.94	1.24	3.55	354,733,271
1.2	1.20	1.56	440,657,832	62,715,439	6.03	1.21	3.49	377,942,393
1.3	1.30	1.69	489,439,657	68,527,774	6.14	1.16	3.37	420,911,883
1.4	1.40	1.82	533,258,195	73,740,646	6.23	1.11	3.27	459,517,549
1.5	1.50	1.95	572,462,190	78,213,780	6.32	1.07	3.18	494,248,410

16.4.1.10 Pit Shells Overview

The pit optimization process generated a series of pit shells for each area. Figure 16-10 to Figure 16-18 present a plan view of the generated pit shells per area. Figure 16-19 presents a cross-section of East Mill Pit L35.

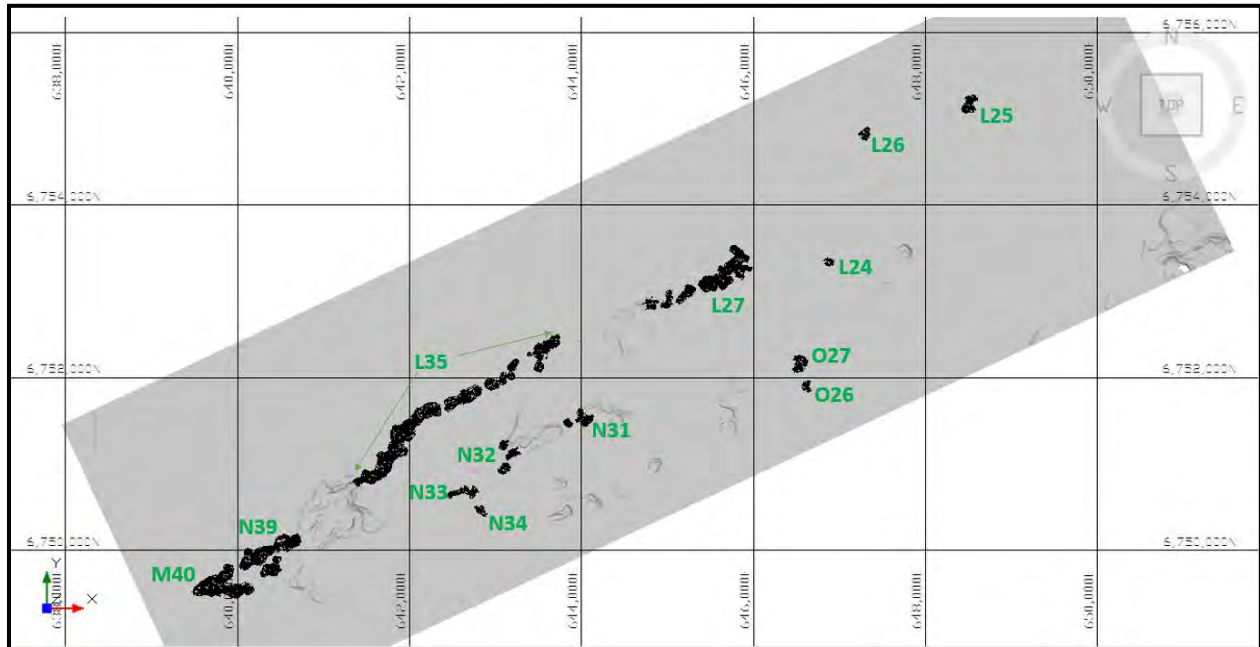


Figure 16-10: East Mill pit shells

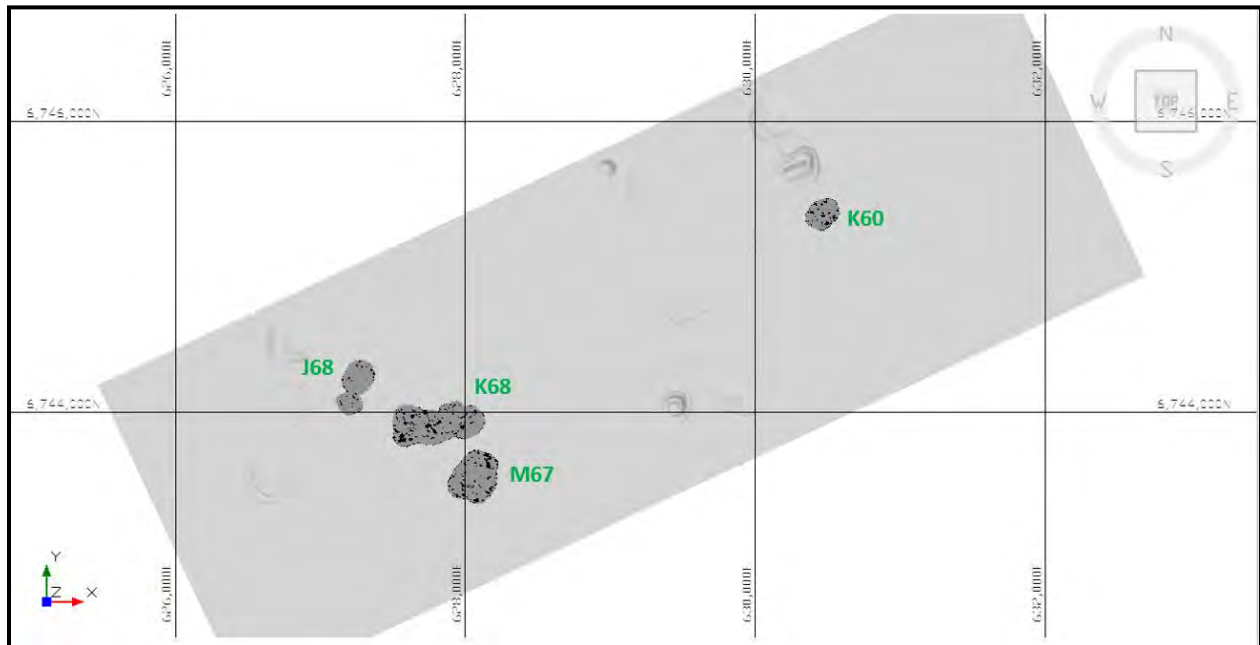


Figure 16-11: C1 Pit shells

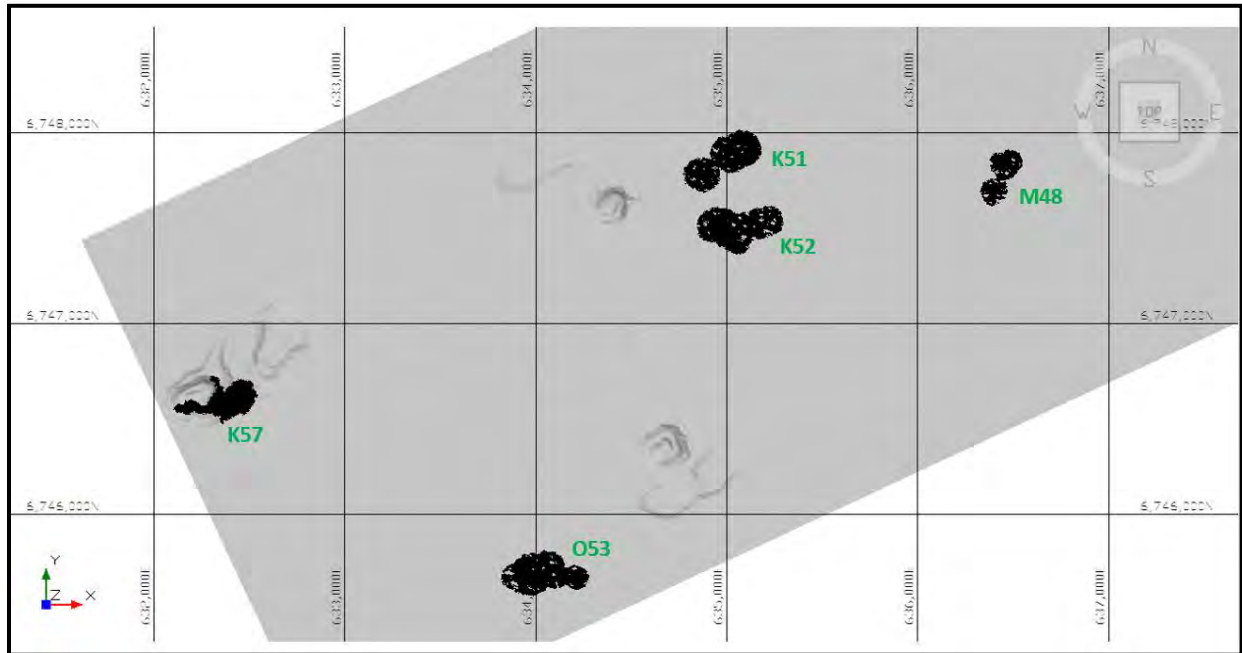


Figure 16-12: C2 Pit shells

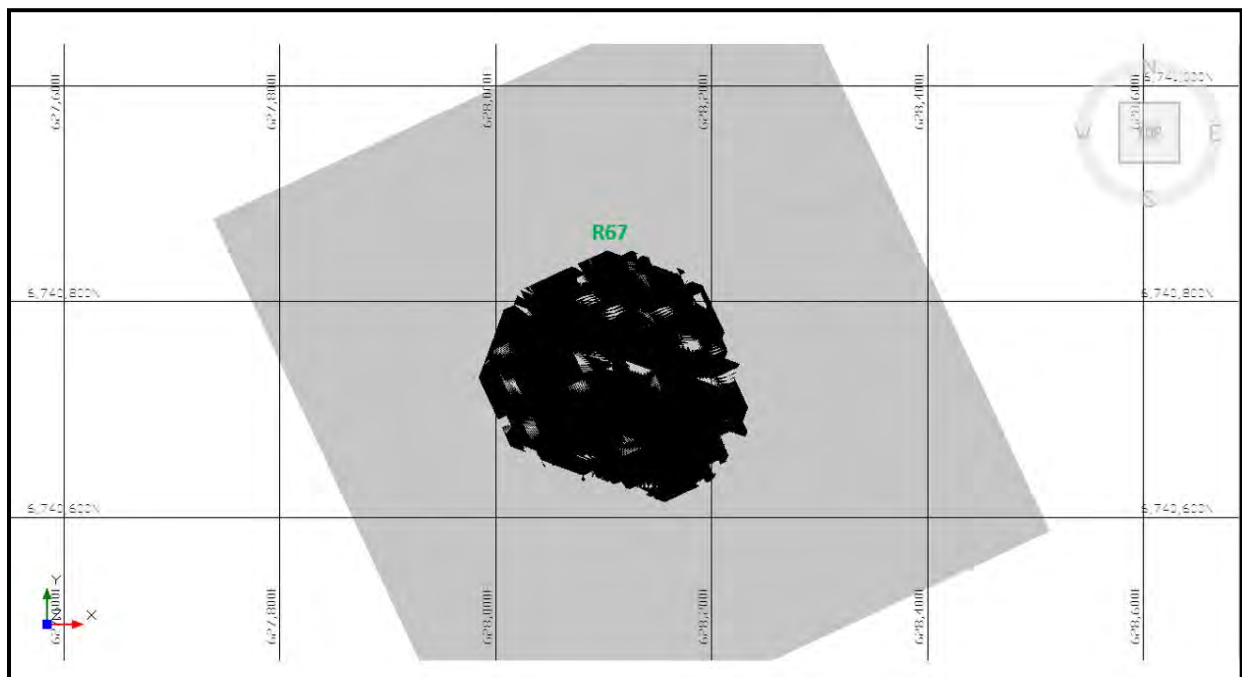


Figure 16-13: C3 Pit shell

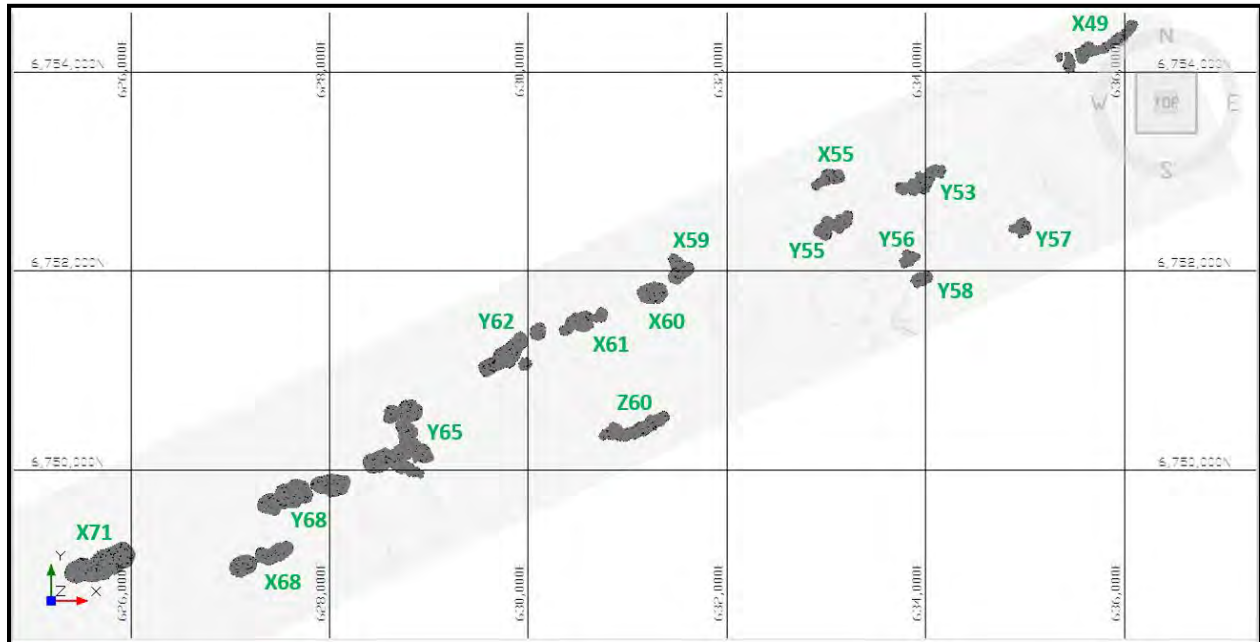


Figure 16-14: N1 Pit shells

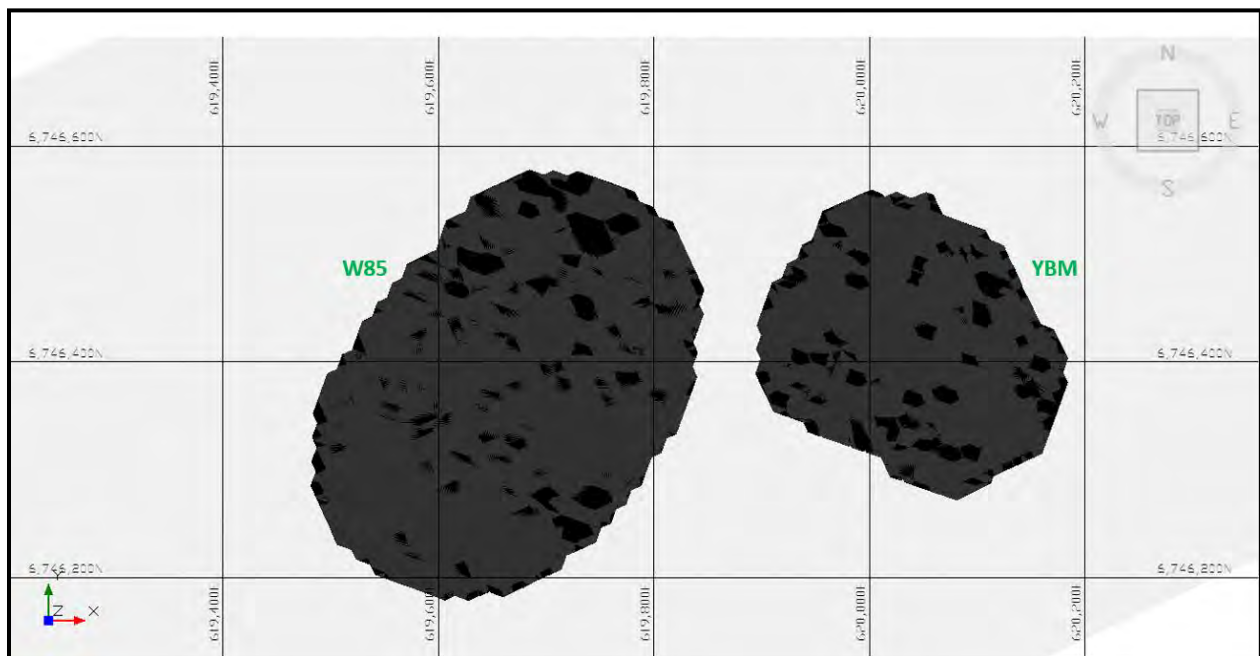


Figure 16-15: N2 Pit shells

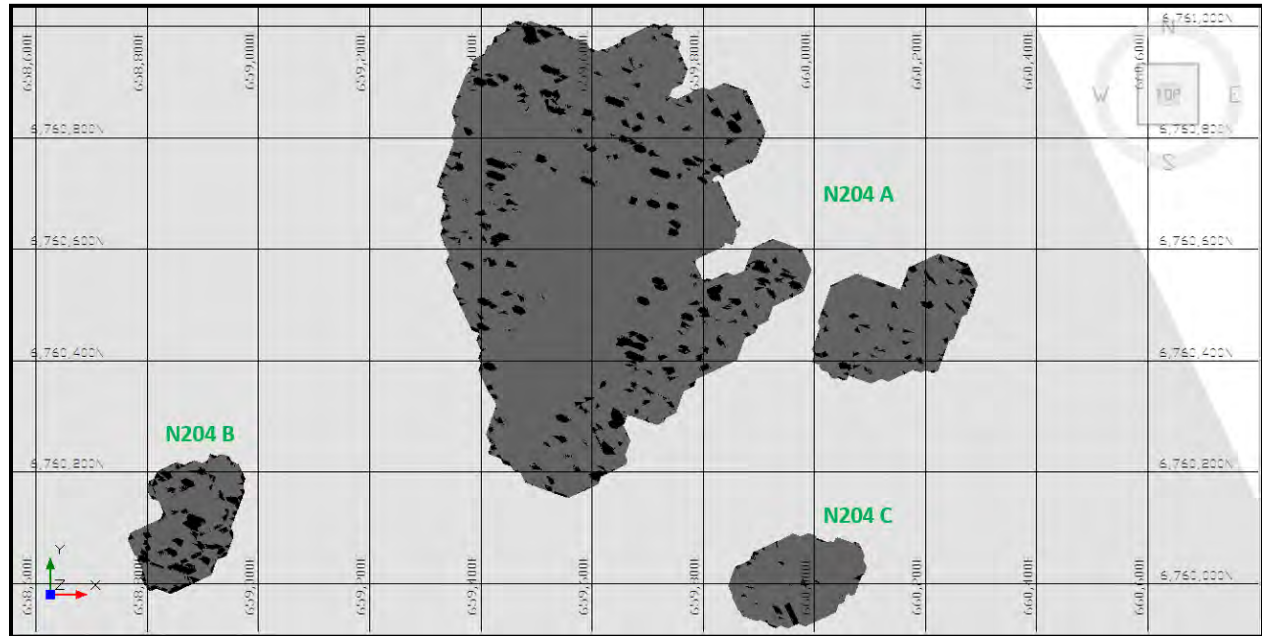


Figure 16-16: N204 Pit shells

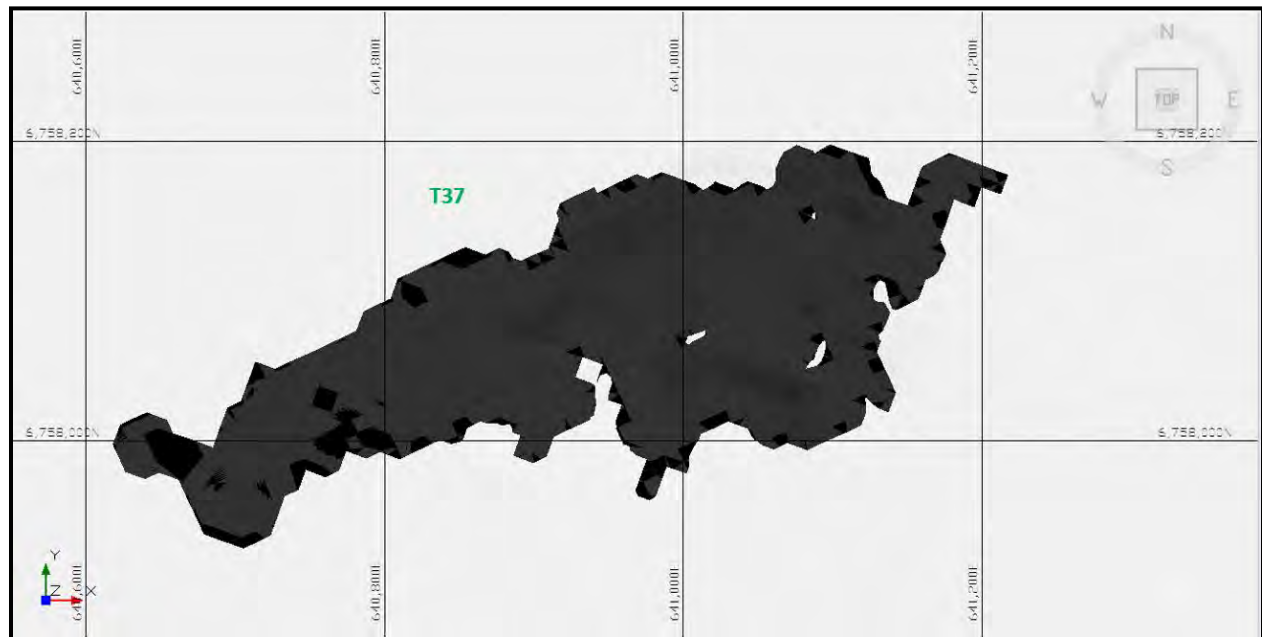


Figure 16-17: NE1 Pit shell

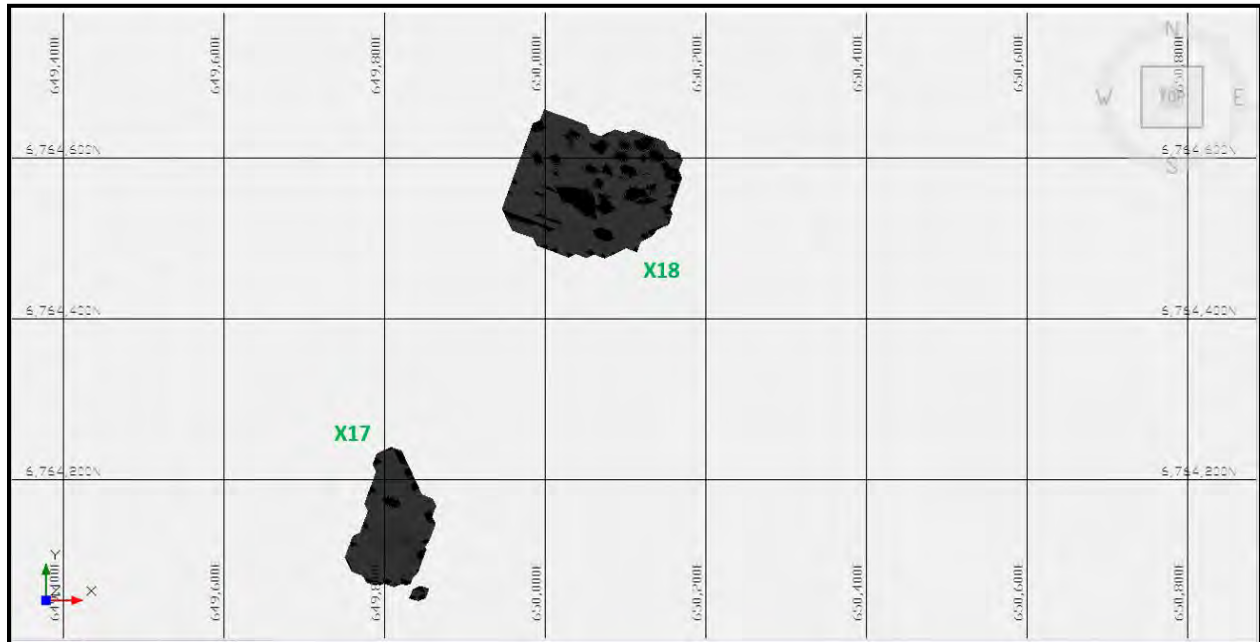


Figure 16-18: NE2 Pit shells

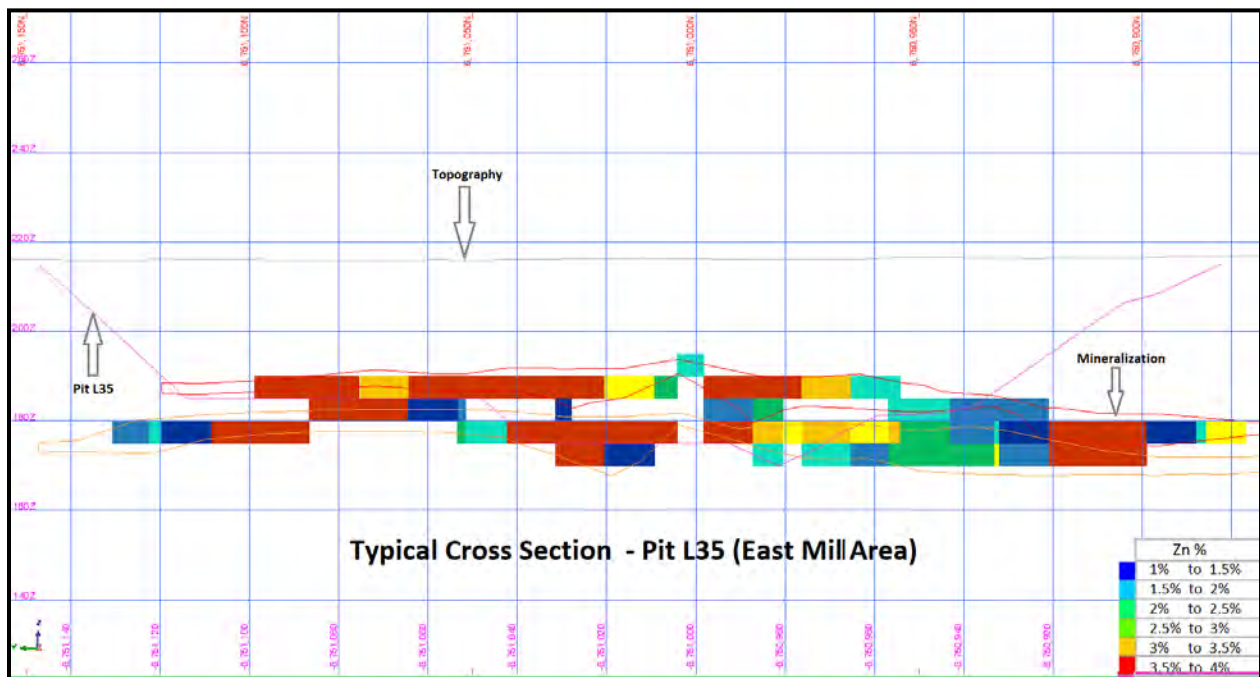


Figure 16-19: Cross-section looking North-East – East Mill pit L35

16.4.1.11 Pit Shell Validation and Open-pit Mineralized Inventory

Since selected pit shells from pit optimizations yielded multiple pits per area (except for Areas C3 and NE1, which have only one pit each) it was important to select those pits which can be effectively mined. Two criteria were used to select mineable pits:

- Mineralized material tonnage per pit should be above 50 000 tonnes due to operational constraints;
- Reasonable strip ratio (below 20).

A total of 47 pits were selected with a total of 37.9 Mt of mineralized material at a dilution of 12%, a mining recovery of 95% and a cut-off grade of 2.2% on ZnEq, as presented in Section 16.4.1.10 (Pit Shells Overview).

A list of the selected pits, and their respective tonnages and grades, is presented in Table 16-9 and Table 16-10 summarizes the mineralized material inventory by area.



Table 16-9: Inventory by pit

Diluted Tonnages and grades per pit using a RF=1 and @ 12% on Mineralized Material and 95% Mine recovery									
Pit	Area	Mineralized Material	Zn (%)	Pb (%)	ZnEq. (%)	Overburden	Waste Rock	Total Waste	Strip Ratio
J68	C1	319,966	7.20	3.39	10.58	1,469,488	1,446,139	2,915,627	9.11
K60	C1	158,983	4.13	6.32	10.46	916,089	1,446,139	2,362,227	14.86
K68	C1	1,009,548	5.25	1.42	6.67	2,817,589	1,406,377	4,223,965	4.18
M67	C1	617,754	5.08	1.61	6.69	2,176,446	10,822,250	12,998,696	21.04
K51	C2	328,737	4.28	1.29	5.57	1,978,624	6,740,692	8,719,317	26.52
K52	C2	319,792	4.60	1.28	5.88	1,902,668	975,982	2,878,651	9.00
K57	C2	412,048	3.08	1.51	4.59	671,002	2,478,703	3,149,705	7.64
M48	C2	135,427	5.36	2.07	7.42	471,004	1,012,001	1,483,004	10.95
O53	C2	596,053	5.43	1.58	7.01	2,045,660	1,118,415	3,164,075	5.31
R67	C3	439,691	7.43	2.36	9.78	1,270,941	1,218,729	2,489,670	5.66
X49	N1	877,937	3.69	1.49	5.18	2,210,243	367,115	2,577,358	2.94
X55	N1	314,874	2.93	0.58	3.51	863,205	549,115	1,412,320	4.49
X59	N1	247,206	4.58	1.13	5.72	1,266,123	682,300	1,948,423	7.88
X60	N1	295,420	5.66	1.38	7.04	1,473,086	1,120,883	2,593,969	8.78
X61	N1	1,103,236	4.06	1.63	5.68	1,319,298	1,415,525	2,734,823	2.48
X68	N1	732,262	3.79	0.78	4.56	2,946,369	1,437,138	4,383,508	5.99
X71	N1	2,861,234	2.84	2.11	4.95	5,208,624	1,980,554	7,189,177	2.51
Y53	N1	340,180	8.67	2.99	11.67	1,650,464	8,050,487	9,700,951	28.52
Y55	N1	249,460	4.10	0.92	5.01	1,639,741	2,247,470	3,887,211	15.58
Y56	N1	147,028	4.12	0.45	4.58	489,311	443,456	932,766	6.34
Y57	N1	166,332	2.44	0.93	3.37	460,617	104,338	564,955	3.40
Y58	N1	147,214	3.06	0.63	3.69	483,785	70,324	554,110	3.76
Y62	N1	919,540	3.56	1.66	5.23	3,722,848	309,014	4,031,862	4.38
Y65	N1	2,185,893	4.58	1.34	5.92	7,917,121	3,057,792	10,974,913	5.02
Y68	N1	1,658,692	3.74	1.48	5.22	6,018,432	3,732,775	9,751,207	5.88
Z60	N1	673,931	4.73	0.92	5.65	2,568,216	2,815,340	5,383,556	7.99
W85	N2	1,649,645	4.93	2.33	7.27	2,412,428	556,052	2,968,480	1.80
YBM	N2	984,559	3.89	3.10	6.99	695,709	7,440,122	8,135,832	8.26
N204A	N204	6,992,380	3.43	0.96	4.39	1,967,619	3,545,839	5,513,457	0.79
N204B	N204	498,732	2.76	0.83	3.59	80,491	29,714,581	29,795,072	59.74
N204C	N204	241,928	3.55	1.03	4.57	215,661	1,176,968	1,392,628	5.76



Diluted Tonnages and grades per pit using a RF=1 and @ 12% on Mineralized Material and 95% Mine recovery									
Pit	Area	Mineralized Material	Zn (%)	Pb (%)	ZnEq. (%)	Overburden	Waste Rock	Total Waste	Strip Ratio
T37	NE1	448,583	5.73	2.03	7.76	448,133	1,207,153	1,655,286	3.69
X17	NE2	102,539	5.61	1.14	6.75	58,532	979,316	1,037,847	10.12
X18	NE2	246,755	3.05	0.82	3.87	63,801	130,532	194,333	0.79
L24	EM	63,552	4.55	1.92	6.46	82,553	1,778,683	1,861,237	29.29
L25	EM	273,749	2.39	0.74	3.13	316,990	217,031	534,021	1.95
L26	EM	115,334	3.03	0.52	3.56	256,174	148,791	404,964	3.51
L27	EM	1,457,672	3.70	0.76	4.46	2,053,951	90,222	2,144,173	1.47
L35	EM	4,064,386	3.67	1.45	5.13	4,941,283	5,378,947	10,320,230	2.54
M40	EM	1,120,732	4.04	1.01	5.06	2,305,386	16,066,494	18,371,879	16.39
N31	EM	202,016	3.28	1.01	4.29	402,108	6,126,428	6,528,536	32.32
N32	EM	370,975	2.32	0.88	3.20	392,001	267,742	659,743	1.78
N33	EM	232,267	3.79	1.14	4.93	378,550	360,810	739,361	3.18
N34	EM	51,109	4.01	0.29	4.30	104,049	308,357	412,406	8.07
N39	EM	1,182,146	4.57	1.16	5.73	1,481,937	138,802	1,620,738	1.37
O26	EM	62,350	4.54	1.43	5.97	139,907	6,066,656	6,206,563	99.54
O27	EM	289,453	3.57	0.77	4.34	319,227	111,282	430,509	1.49
Total		37,909,301	3.98	1.44	5.41	75,073,482	138,859,860	213,933,342	5.64



Table 16-10: Inventory by area

Diluted Tonnages and grades per area using a RF=1 and @ 12% on Mineralized Material and 95% Mine recovery								
Area	Mineralized Material	Zn (%)	Pb (%)	ZnEq. (%)	Overburden	Waste Rock	Total Waste	Strip Ratio
C1	2,106,252	5.41	2.14	7.56	7,379,611	15,120,904	22,500,515	10.68
C2	1,792,057	4.52	1.50	6.02	7,068,959	12,325,793	19,394,752	10.82
C3	439,691	7.43	2.36	9.78	1,270,941	1,218,729	2,489,670	5.66
N1	12,920,439	3.91	1.52	5.43	40,237,484	28,383,626	68,621,110	5.31
N2	2,634,205	4.54	2.62	7.16	3,108,137	7,996,175	11,104,312	4.22
N204	7,733,040	3.39	0.96	4.34	2,263,770	34,437,388	36,701,157	4.75
NE1	448,583	5.73	2.03	7.76	448,133	1,207,153	1,655,286	3.69
NE2	349,293	3.80	0.91	4.71	122,332	1,109,848	1,232,180	3.53
EM	9,485,742	3.74	1.16	4.90	13,174,116	37,060,244	50,234,360	5.30
Total	37,909,301	3.98	1.44	5.41	75,073,482	138,859,860	213,933,342	5.64

It should be mentioned that the tonnages presented in Table 16-10 are a summation of pit tonnages from Table 16-9, which were calculated with Deswik Sched. These numbers might differ slightly from those presented in Table 16-7, which were calculated with Whittle™ software. This difference is due to the calculation method used by two different software programs. In addition, Table 16-9 presents only mineable pits with a minimum of 50,000 tonnes with a strip ratio below 20, while Table 16-7 presents all mineral inventory, including small pit that cannot be mined conventionally.

16.4.2 Pit Design

No pit design work was done for this PEA.

16.4.3 Pit Dewatering

16.4.3.1 Historical Dewatering

Cominco operated the site between 1964 and 1988. There is a lot of information related to dewatering as it is a major component of the operation. Historical data about a similar operating rate of (approximately 10,000 tpd of mineralized material) shows that dewatering ranged between 33,000 m³ and 268,000 m³ per day, and an average of 180,000 m³ per day between 1968 and 1982.

The following items were considered using historical information and recent experience and technology:

Well depths were usually 65 m on average, based on pit depth and an additional 10 m of elevation to create a dewatering gap between the pit bottom and dewatered elevation, this is to ensure that the bench being drilled will be dry.

Each well yielded around 10,900 m³/d; four different types of pumps were planned according to the total head and discharge rates that are forecasted.

16.4.3.2 Pumping Rate Estimates

Initially, the Company and HRI compiled existing hydrogeological reports, reference papers, and used the 3D Geological model interpreted from historical and recent drill logs to define the stratigraphic boundaries, including the Sulfur Point Formation, the main host of Tabular mineralization at Pine Point.

Then, 26 boreholes were surveyed using Profile Tracer Tests (PTT), chemical profiles, slug tests and injection tests. The PTT consists of mixing a tracer in the borehole without inducing any stress, followed by the monitoring of tracer dilution over time within each hole. This allows for the accurate identification of water-bearing sources, including structural discontinuities, and allows for better flow characterization and interpretation. The Chemical profiles that are measured in each of the tested hole gives a full downhole measurement survey of water temperature, electrical conductivity, total dissolved solids and water salinity. The measured changes in water chemistry often highlights isolated flow zones, confirming the contrast between faults and/or bedding planes when compared to low flow zones.

PTT results indicate that groundwater flow is preferentially controlled by discrete discontinuities (i.e., faults and/or fracture zones) contrasting from the historical concept of a majority of the flow originating from formational aquifers. Locally, Prismatic high-grade deposits, that were preferentially mined during the Cominco era, are also located within the porous dolomitized Sulfur Point Formation and are associated to higher inflow rates since these Prismatic deposits are also associated to the same structural discontinuities discussed above.

Recent structural analyses of all available open-pits, combined with 3D drone photography of pit walls, of historical Cominco Ltd. era open-pits, was carried out by Terrane Geosciences in 2021. This allowed for the construction of initial structural geology models which are now being combined with the Pine Point Project, drill hole database and interpreted stratigraphic off-sets from vertical sections, 3D geological modeling, and GIS database that includes regional data sources (Magnetic surveys, Gravity Surveys, LIDAR, regional maps, aerial photographs, etc.) All of

these datasets are being used to interpret and locate specific structural features or lineaments within proposed mining areas in an attempt to correlate with regional structural discontinuities.

This will lead to a better understanding of potential water bearing discontinuities and provide for strategically targeted placement of highly efficient water wells. To date, predictable fault trends were identified in historical open-pits and correlated with stratigraphy that was off-set in drill holes.

Data shows that the water inflow is mainly associated to fault structures, therefore strategically localized water well emplacement around these structures will be more effective, and a far less costly dewatering management strategy.

16.4.4 Site Wide Numerical Model; A Dewatering Overview

The SWNM was built using Feflow V7, a well-known simulation software, the SWNM covers a surface area of approximately 1775 km², including the project area 94 kilometers of strike length in an East North-easterly direction and a 22-kilometers project width in an approximately north south direction.

It includes all of the proposed open-pits and underground mines, as well as all of the historical mines. Two regional faults were also integrated into the SWNM and zones of hydraulic conductivities were integrated. Calibration of the SWNM was achieved by comparing historical inflows to simulated inflows obtained for historical open-pit areas using the SWNM. Following this, the hydraulic conductivity values were compared to those obtained during the ongoing exploration and historical drillhole testing process. Current overall dewatering volumes are approximately in-line with historical dewatering records.

Table 16-11: Hydrogeological model calibration results

Open-pit (Name)	Historical Flow (m ³ /day)	Simulated Flow (m ³ /day)	Percentage Difference (%)
J-44	40,197	41,066	2%
N-42	13,735	11,352	-21%
O-42	29,705	23,720	-25%
X-15	26,707	23,422	-14%
K-57	47,353	52,922	11%
R-61	50,287	70,487	29%
K-62	38,246	40,748	6%
A-70	104,373	93,868	-11%
J-69	78,823	71,411	-10%
T-58	51,617	52,172	1%



Open-pit (Name)	Historical Flow (m ³ /day)	Simulated Flow (m ³ /day)	Percentage Difference (%)
S-65	14,557	34,184	57%
A-55	131,150	138,810	6%
K-77	33,841	32,679	-4%
M-64	30,083	36,951	19%
Sum	690,674	723,792	+5%

The simulated results are in the same order of magnitude as the observed flow rates compiled by Stevenson (1984). Therefore, modeling is slightly overestimating inflow by five percent and, since it considers only the Prismatic type of deposits, the K values are likely to be lower than those simulated, which is supported by recent field test results. Open-pit S-65 shows the biggest difference between simulated estimates and observed results.

Since there are no historical values of inflow rates in the West Zone, located in the area west of the Buffalo River, making comparisons is not possible. Therefore, the historical pump test result was used in the R190 deposit area, reported by Vogwill (1982). This historical test result was used to verify the accuracy of the SWNM in this area: Simulated drawdown estimates are comparable to the observed drawdown values, suggesting the calibration of the SWNM is appropriate for the West Zone. Following recent results in X25, another isolated numerical was prepared to better understand the specific dewatering needs of that potential underground deposit.

The LOM plan that was used for the SWNM, listed monthly production rates that were provided to simulate monthly inflow rates that maintain water levels at 10 m below all operational bench elevations. The evolution of inflow rates is illustrated in the graph below; this considers a grouting strategy in the proposed underground mines located in the West Zone.

The graph below illustrates a summary of the inflow on the whole property.

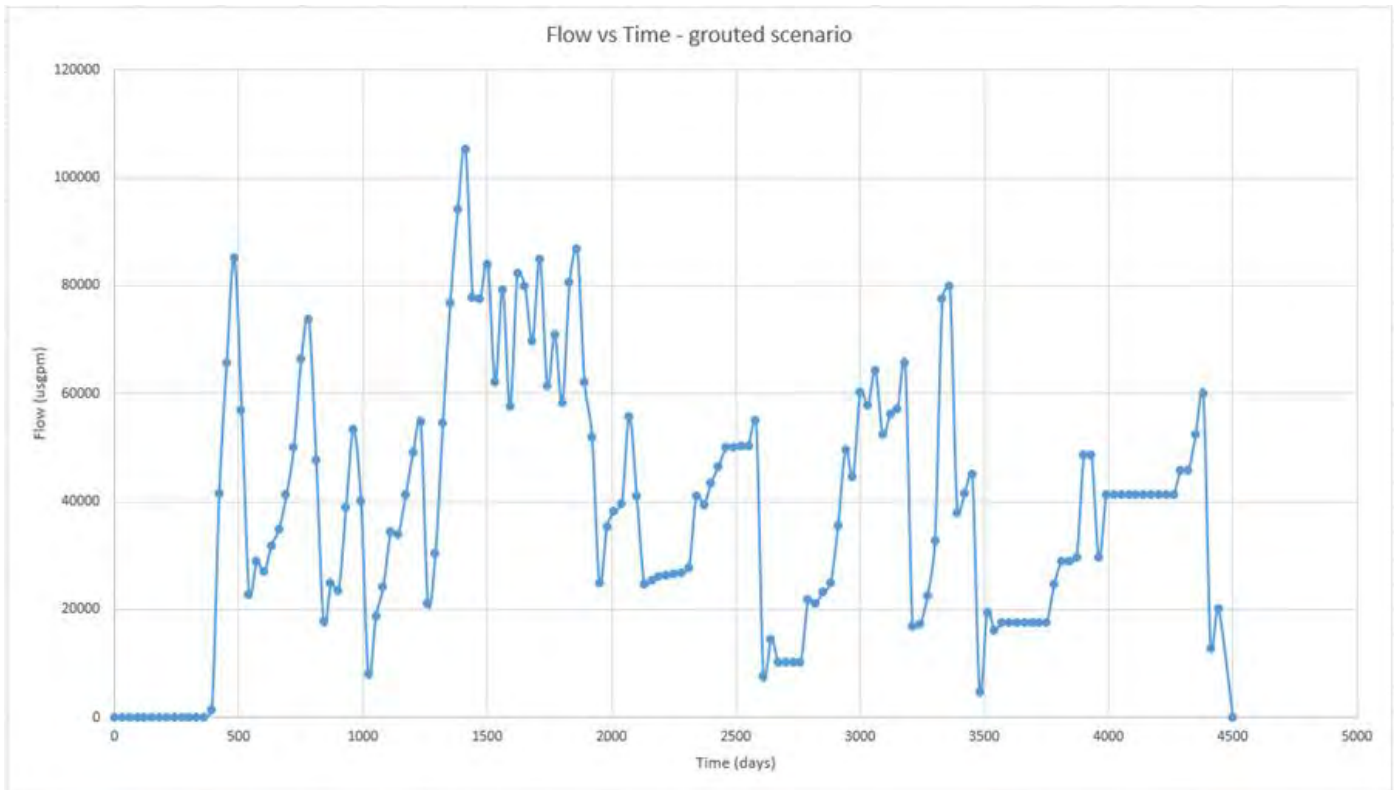


Figure 16-20: Summary of the inflow on the whole property: Inflow Rates vs Time

Peak inflows occur in Year 4 and 8 with an inflow of 105,000 and 80 000 usgpm. The main water producers are the proposed underground mines.

16.4.4.1 Dewatering Methodology

Dewatering wells will be located using various techniques to isolate water bearing zones and ensure higher inflow rates so that dewatering strategies are as efficient as possible around or within the proposed mining areas. Once the dewatering wells are located, they would be drilled to depths that are slightly deeper than the pit bottom to allow for any rock fragments to fall to the bottom of the hole and to also have room for dewatering pumps. Dewatering to an elevation that is 10 m deeper than pit bottom at any stage of operation to ensure that the production drills would be working in dry material. Wells would be 400 mm diameter open holes with casing from surface to the overburden/rock interface. The exact location of the dewatering wells will be determined prior to the proposed mining operations.



16.4.5 Pit Drilling and Blasting

Drilling and blasting requirements are estimated based on assumptions derived from experience from similar projects in similar regions. Table 16-12 shows the drill metrics used for rock material only. It is assumed that overburden material will not require drilling and blasting.

In this PEA, it is assumed that drilling will be performed by the owner whereas blasting will be done by a contractor.

To estimate blasting costs, we assumed that the on-site emulsion manufacturing plant will be in operation starting from Year 2. Explosives will be supplied by a specialized company in the region.

Table 16-12: Open-pit major equipment fleet

Parameter	Mineralized material	Waste rock
Borehole size (mm)	152.4	152.4
Bench height (m)	10	10
Sub-drilling (m)	1.2	1.2
Depth per hole (m)	11.2	11.2
Burden (m)	4.57	4.88
Spacing (m)	4.57	4.88
Volume/hole (m ³)	208.8	238.1
Rock density (g/cc)	2.90	2.71
Tonnage/hole (met. ton.)	605.7	645.4
Collar length (m)	3.0	3.0
Loaded length/hole (m)	8.2	8.2
Bulk emulsion density (g/cc)	1.20	1.20
Charge/linear m. (kg)	21.89	21.89
Charge/hole (kg)	179.50	179.50
Powder factor (kg/m ³)	0.86	0.75
Powder factor (kg/m ³)	0.29	0.25
Drill rate (min/hole of 10 m)	30.00	30.00
Drill rate (hole of 10 m/h)	2.00	2.00
Drill availability	69%	69%
Drill utilization	85%	85%

16.4.6 Material Stockpile and Disposal

16.4.6.1 Mineralized Material

Mineralized material is sent directly to the process plant. For more details refer to the mine production schedule in Table 16-32.

16.4.6.2 Mineral Sorter Rejects

A stockpile for the mineral sorter rejects is planned to be located in proximity to the process plant. A smaller fleet of equipment is planned to re-handle this material to nearby available mined-out pits. The mining production schedule (Section 16.6) shows that a large amount of mineralized material will be produced from underground operations between Year 3 and Year 8. There will therefore be more 100-t class trucks available. These trucks are planned to be used to move the mineral sorter rejects into available mined-out pits further afield, within a radius of about 5 km.

16.4.6.3 Pit Waste Rock and Overburden

Waste rock and overburden stockpile design and management are discussed in Chapters 18 and 20. Waste rock and overburden stockpiles are planned to be located within 2 km of each pit. Some of the stockpiles will be used for many pits resulting in less waste rock and overburden stockpiles than the total volume of material mined from open-pits. Additionally, where appropriate, overburden and waste rock material will be disposed of in available mined-out pits.

16.4.7 Pit Mining Equipment Fleet

Open-pit operating costs are calculated from an owner-operated scenario. Contract mining was reviewed for comparison and benchmarking purposes. The major mining equipment selected for the Project, along with the auxiliary fleet, is summarized in Table 16-13. The mining equipment is selected to match the mine production schedule (Section 16.5).

- The open-pit mining truck size selected is a 100-ton class rigid-frame rear dump truck, for all types of material. These trucks have a payload capacity of 90 tonnes. The average number of 100-t truck required to move the material is twenty from Year 1 to Year 3 due to a high strip ratio, then this number decreases to fifteen 100-t class trucks from Year 4 to Year 8 and finally this number decreases to ten 100-t class trucks from Year 9 to Year 12.
- It can be assumed that the average number of 100-t trucks is fifteen with an additional four mining trucks to be leased from Year 1 to Year 3 to accommodate for early peak production due to high waste stripping.



- On average, four hydraulic shovels with 12 m³ bucket capacity will be used to excavate the waste rock from Year 1 to Year 7, then this number decreases to two hydraulic shovels with 12 m³ bucket from Year 8 to Year 12;
- On average three hydraulic excavators of 7 m³ bucket capacity will be used in the mineralized material for more selectivity from Year 1 to Year 5, then this number decreases to three hydraulic excavators of 7 m³ bucket capacity from Year 6 to Year 12.
- Overburden removal is planned to be achieved with the same smaller sized shovel (7 m³ bucket capacity);
- A dedicated fleet, composed of one small excavator and two 45-t articulated trucks, is planned to move mineral sorter rejects on a continuous basis.
- A front-end loader of 12 m³ bucket capacity is planned as a back-up unit for waste rock removal and mineral sorter rejects handling during early peak production from Year 1 to Year 3;

Table 16-13: Open-pit major equipment fleet

	Units	Y1	Y2	Y3	Y4	Y5	Y6	Y7	Y8	Y9	Y10	Y11	Y12
Loading	#												
PC1250	#	4	5	3	5	4	4	3	3	2	1	2	1
PC2000	#	4	3	4	2	2	2	3	2	2	3	3	1
CAT992K	#	1	2	1	1	1	1	1	1	1	1	0	1
Number of units required	#	9	10	8	8	7	7	7	6	5	5	5	3
Hauling	#												
100t	#	20	22	17	14	15	15	13	15	10	9	12	3
Number of units required	#	20	22	17	14	15	15	13	15	10	9	12	3
Drilling	#												
DM45	#	4	4	4	2	2	2	3	3	2	3	3	1
Number of units required	#	4	4	4	2	2	2	3	3	2	3	3	1
Production Support													
CAT 14M Grader	#	1	2	2	3	3	3	3	3	3	3	3	3
CAT D6 Track Dozer	#	2	2	2	2	2	2	2	2	2	2	2	2
CAT D8 Track Dozer	#	2	3	3	3	3	3	3	3	3	3	3	3
Number of units required	#	5	7	7	8	8	8	8	8	8	8	8	8



	Units	Y1	Y2	Y3	Y4	Y5	Y6	Y7	Y8	Y9	Y10	Y11	Y12
Auxiliary													
Loader 980	#	2	2	2	2	2	2	2	2	2	2	2	2
CAT 336 Ex	#	2	3	3	3	3	3	3	3	3	3	3	3
Articulated truck 40t	#	2	2	2	2	2	2	2	2	2	2	2	2
CAT 775 Water Truck	#	1	2	2	2	2	2	2	2	2	2	2	2
CAT CS78 Compactor	#	1	1	1	1	1	1	1	1	1	1	1	1
VOLVO Utility Truck	#	1	2	2	2	2	2	2	2	2	2	2	2
Tractor	#	1	1	1	1	1	1	1	1	1	1	1	1
Service Truck	#	1	1	1	1	1	1	1	1	1	1	1	1
Personal Carriers	#	1	1	1	1	1	1	1	1	1	1	1	1
Tyre Handler	#	1	1	1	1	1	1	1	1	1	1	1	1
Crane	#	1	1	1	1	1	1	1	1	1	1	1	1
Fuel Lube Truck	#	1	1	1	1	1	1	1	1	1	1	1	1
Fardier	#	1	1	1	1	1	1	1	1	1	1	1	1
Number of units required	#	16	19	19	19	19	19	19	19	19	19	19	19
Total (major and minor production)	#	54	62	55	51	51	51	50	51	44	44	47	34



16.4.8 Open-pit Workforce

Table 16-14 summarizes the workforce required for open-pit operations. For major equipment, it is assumed that one operator per shift is required.

Table 16-14: Open-pit Workforce

Year	Y1	Y2	Y3	Y4	Y5	Y6	Y7	Y8	Y9	Y10	Y11	Y12
Mine Operators												
Production operators (4 shifts)	104	140	120	112	112	124	116	148	132	120	120	24
Auxiliary operator (day shift)	32	38	38	38	38	38	38	38	38	38	38	8
Pumping team	8	8	8	8	8	8	8	8	8	8	8	2
Blasting team	4	4	4	4	4	4	4	4	4	4	4	2
Mine Operators Total	148	190	170	162	162	174	166	198	182	170	170	36
Mine Management												
Superintendent	1	1	1	1	1	1	1	1	1	1	1	1
Assistant superintendent	1	1	1	1	1	1	1	1	1	1	1	0
Pit supervisor	8	8	8	8	8	8	8	8	8	8	8	2
Drilling supervisor	2	2	2	2	2	2	2	2	2	2	2	2
Labourer	8	8	8	8	8	8	8	8	8	8	8	2
Fleet management system	6	6	6	6	6	6	6	6	6	6	6	2
Mine trainer	2	2	2	2	1	1	1	1	1	1	1	0
Clerk	4	4	4	4	4	4	4	4	4	4	4	2
Mine Management Total	32	32	32	32	31	31	31	31	31	31	31	11
Mine Maintenance												
Superintendent	1	1	1	1	1	1	1	1	1	1	1	1
Foreman	5	6	7	7	7	7	7	7	7	7	7	2
Planner	2	2	2	2	2	2	2	2	2	2	2	2
Mechanic	16	22	20	19	19	20	19	23	21	20	20	8
Electrician	5	7	6	6	6	6	6	7	7	6	6	2
Welder	8	8	8	8	8	8	8	8	8	8	8	2
Machinist	2	2	2	2	2	2	2	2	2	2	2	2
Labourer	8	8	8	8	8	8	8	8	8	8	8	2
Fuel/lube technician	4	4	4	4	4	4	4	4	4	4	4	2
Mine Maintenance Total	51	60	58	57	57	58	57	62	60	58	58	23



Year	Y1	Y2	Y3	Y4	Y5	Y6	Y7	Y8	Y9	Y10	Y11	Y12
Technical Services												
Superintendent technical services	1	1	1	1	1	1	1	1	1	1	1	1
Chief engineer	2	2	2	2	2	2	2	2	2	2	2	1
Mining engineer + 1 Civil	3	3	3	3	3	3	3	3	3	3	3	2
Mine technician	4	4	4	4	4	4	4	4	4	4	4	2
Chief surveyor	1	1	1	1	1	1	1	1	1	1	1	1
Surveyor / Rodman	2	2	2	2	2	2	2	2	2	2	2	2
Chief geologist	1	1	1	1	1	1	1	1	1	1	1	1
Senior geologist	2	2	2	2	2	2	2	2	2	2	2	1
Geologist	2	2	2	2	2	2	2	2	2	2	2	2
Senior grade control technician	1	1	1	1	1	1	1	1	1	1	1	1
Grade control technician	2	2	2	2	2	2	2	2	2	2	2	1
Technical Services Total	21	21	21	21	21	21	21	21	21	21	21	15
Total Open-pit Operation	252	303	281	272	271	284	275	312	294	280	280	85



16.5 Underground Mining

16.5.1 Underground Mining Summary

The deposits that are planned to be mined using underground methods are located in two zones: the West Zone (W1 Area) and the Central Zone (C1 Area). These zones and areas are located as shown on Figure 16-21.

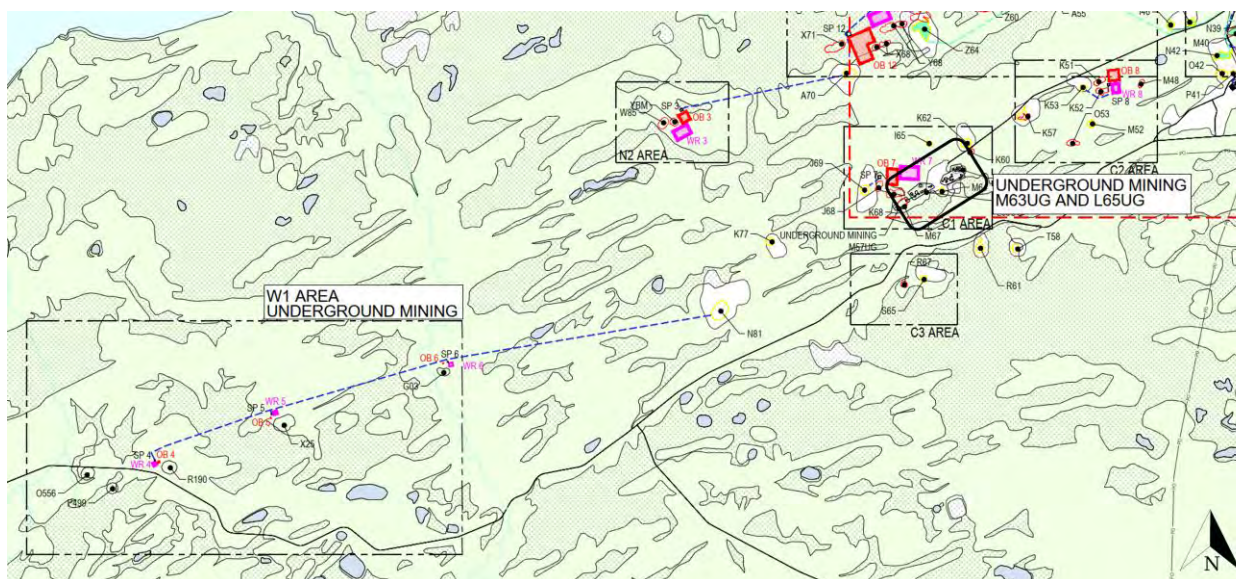


Figure 16-21: Underground workings locations, excerpt from Figure 16-1 (not to scale)

The West Zone MRE contains six deposits of which five are planned to be mined from underground: R190, G03, P499, X25, and O556. Zone Z155 is not included in the mine plan for economic reasons.

The Central Zone includes three deposits that are planned to be mined using underground mining methods: M67UG, L65UG, and M63UG.

The mining plan for the underground deposits consists of a combination of two mining methods: long hole stoping, and room and pillar. Long hole stoping was chosen to suit the geometry and dip of the Prismatic deposits while the room and pillar method was chosen for Tabular style deposits. The mining method in the West Zone will be mainly long hole stoping. The room and pillar method will be used along the fringes of the deposits in the West Zone and in the C1 Area where only Tabular deposits are identified.

The underground MRE was supplied by BBA, provided by WSP with the Mineable Shape Optimizer ("MSO") shapes from Datamine software. WSP reviewed the MSO shapes and did a review of the location, tonnage and grade, based on a cut-off grade of 4.27 ZnEq for the longhole and room and pillar methods, respectively. Underground designs including ramps, drifts, sills, stopes, and other underground infrastructure were provided by the Osisko Metals for the West Zone. WSP validated the design, drift lengths, and methodology. The underground design for the C1 Area was developed by WSP.

For both zones, the mineralized material is accessed via a ramp and transported to surface using 45-t articulated rear-dump trucks. Waste rock material is brought to the surface and used to backfill mined-out stopes after mining completion.

16.5.2 Underground Mining Methods

In the proposed mine plan, and for both zones combined, the longhole method provides 71% of the total tonnage, development will provide approximately 8%, and the room-and-pillar method will account for 21%.

16.5.2.1 Longhole Method

As previously discussed, longhole is the principal mining method for the West Zone. Given the significant width of the deposits, the transverse stoping technique was selected as a variant of long hole stoping. The transverse technique consists of excavating a series of primary and secondary stopes along one axis, usually perpendicular to deposit strike, from hangingwall to footwall. A top access for production drilling and a bottom access for mucking will be driven for each stope (primaries and secondaries). Figure 16-22 shows an isometric view of a typical transverse longhole method.

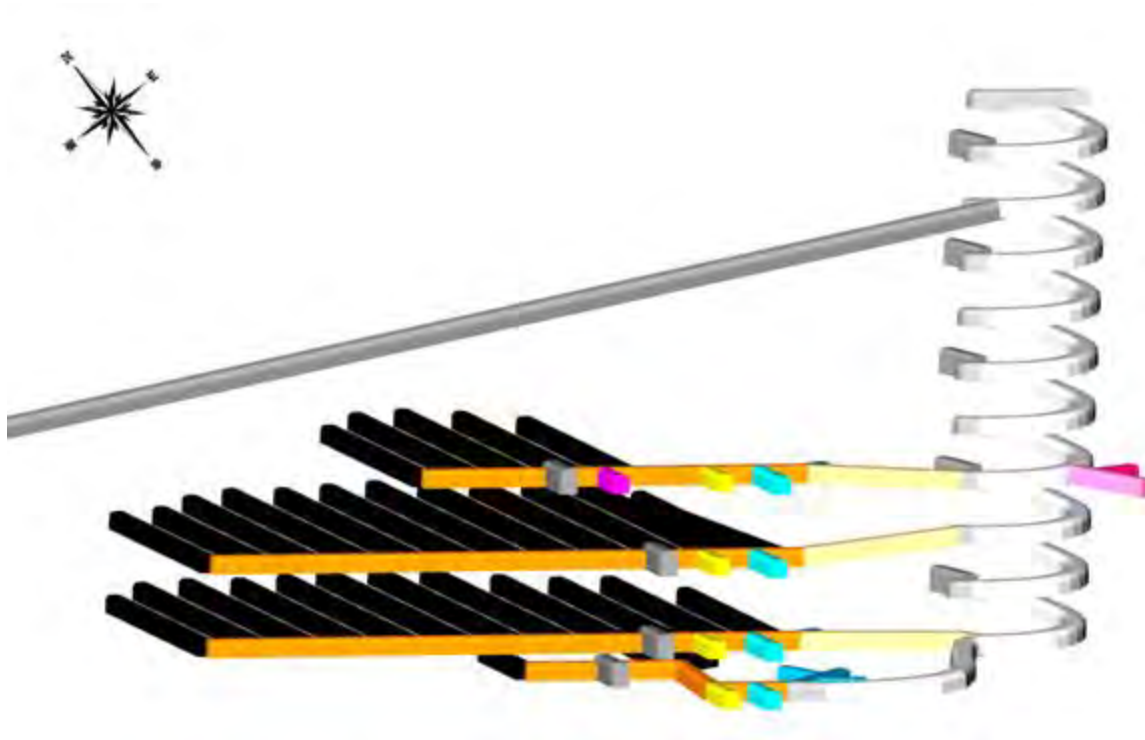


Figure 16-22: Isometric view of typical transverse longhole stope - R190 Deposit (not to scale)

After top access is developed, stope preparation is carried out including cable bolts drilling and installation. Production drilling operations begin with a primary opening, or “slot” Production blastholes of 76mm diameter and 25 m length will be drilled in a typical 3.2m x 3.4m pattern. The drilling design criteria are summarized in Table 16-15.

Table 16-15: Drilling design criteria

Description	Unit	Quantity
Drilling pattern	m	3.4 x 3.2
Number of rows	#	7
Number of holes per row	#	5
Number of holes per stope	#	35
Number of drilled metres per stope	m	975
Number of drilling days/stope	day	5

16.5.2.2 Long Hole Dilution and Recovery

An operational (external) dilution of 10% is added to the mineralized material for the West Zone. Given geomechanical characteristics of the deposits, and stope height, the operational dilution is expected to remain under control during the stope production phase. A mining recovery of 95% has been set for the longhole methodology and is an acceptable level for this mining method.

16.5.2.3 Room and Pillar Method

As discussed above, the room-and-pillar method has been specified for the West Zone to mine the Tabular areas of the deposits (external limits) which represents 8% of the West Zone stope tonnage. Room-and-pillar is to be the main mining method for the C1 UG Area. This method is carried out by a classic development method (jumbo). The height of the rooms and pillars is 5 m and, therefore, the same as that of the development levels which will allow the use of the same mobile equipment as used to develop both the West Zone and the C1 UG Area. Mining will be performed in phases. There are some areas in the Central Zone where the mineral resource height significantly exceeds the original development height of 5 m. In general, these stopes have an average height of 3 m (without exceeding 7 m) and represent less than 2% of the total tonnes of mineralized material. In areas thicker than 5m, ore will be recovered with an “uppers” drilling method (also known as take-down-back or TDB) and drilling will be with the West Zone drill.

16.5.2.4 Room and Pillar Dilution and Recovery

External dilution for the room-and-pillar method is planned to be 5%. During the mining of the room and pillar stopes, the recovered rooms represent eight blocks of 5m x 5m out of nine possible blocks. The last block (1/9) represents the pillar that is left in place. Mining recovery is estimated at 88% for the room-and-pillar method.

16.5.3 Underground Mine Design

16.5.3.1 Primary Underground Access

Two different access strategies will be employed for the West Zone and C1 Areas to allow access to the deposits. For the West Zone, the excavation of overburden is necessary beforehand while for the C1 Area, portals will be excavated into pit walls at the bottom of mined-out and dewatered pits.

16.5.3.2 West Zone Portals

This area consists of five deposits. Three of these deposits (R190, G03, and X25) will be accessed by one surface portal, while P499 and O556 will be accessed by a connection ramp, which will be excavated from the R190 decline. Given the fact that these last three deposits are very close to each other, a connection ramp will save the cost of two extra portal excavations and limit the impact of such excavations on the surface environment. For the R190, G03 and X25 portals, the overburden excavation should be carried out before being able to access the rock (10 m of rock kept above the portal). The estimated overburden and rock volumes to be excavated are shown in Table 16-16.

Table 16-16: Overburden (OB) volume and tonnage for portal excavation – West Zone

Process Plant Site	OB depth (m)	OB volume (m ³)	OB tonnage (t)	Rock volume (m ³)	Rock tonnage (t)
R190	35	1,785,569	3,571,138	21,523	58,112
G03	10	122,548	245,096	21,523	58,112
X25	30	1,241,706	2,483,412	21,523	58,112

To excavate these volumes of material, a preliminary excavation design was made (see Figure 16-23 and Figure 16-24).

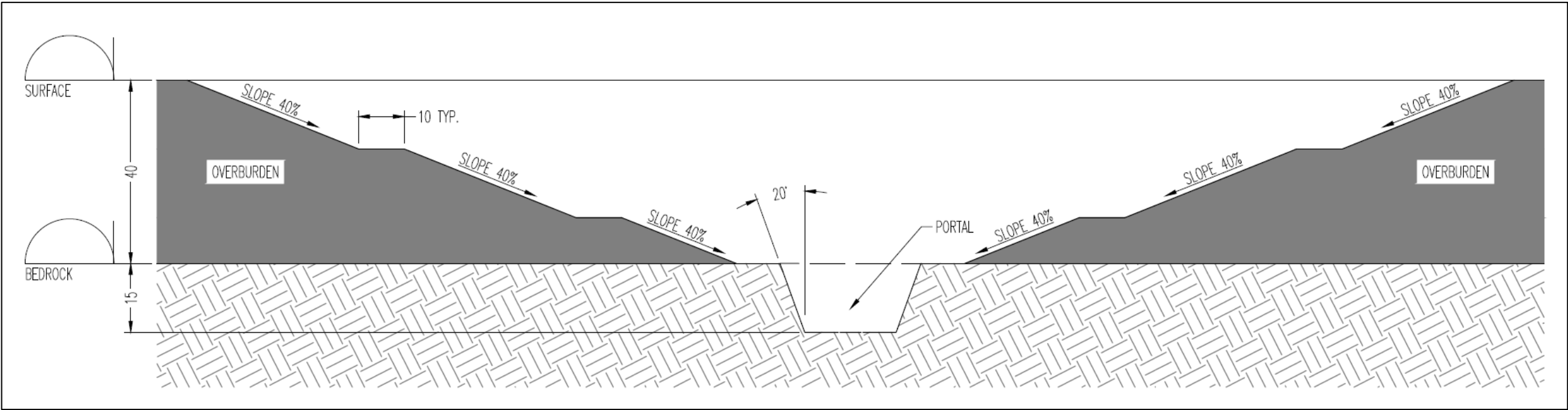


Figure 16-23: Longitudinal view - Portal cut – West Zone

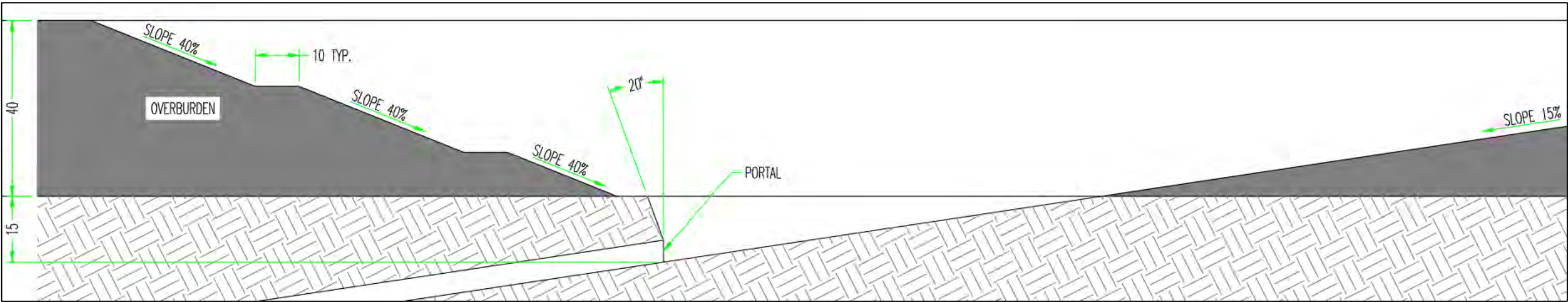


Figure 16-24: Section view - Portal cut – West Zone



16.5.3.3 Connection Ramp R190 – P499 – O556

To allow for the haulage of mined material from deposits P499 and O556, a connection ramp is planned to be excavated between R190, P499, and O556. This 5m x 5m ramp starts from the second loop of the R190 decline, thus at a depth of 30 m from the surface. With an average nominal grade of 4%, this ramp first leads to P499 and then to O556. Figure 16-25 shows an isometric view of the connection ramp.

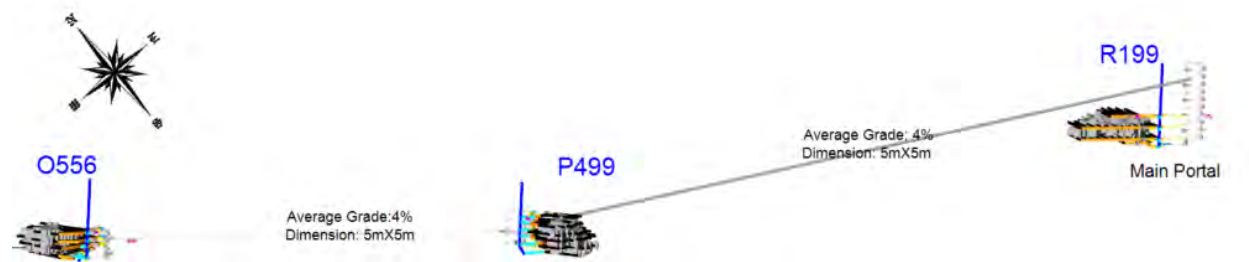


Figure 16-25: Isometric view – Connection ramp - West Zone (not to scale)

16.5.3.4 Central Zone Portals

For the Central Zone (C1 Area), portals will be excavated from the bottom of existing pits: from the M67 pit, mined earlier in the mining schedule, and from which the M67UG and L65UG deposits will be accessed. After being dewatered, the M64 pit, which was mined during the Cominco era, will be used to access M63UG. The C1 portals have the same dimensions as those of the West Zone to allow common underground mobile mining equipment to be used (see Figure 16-26).

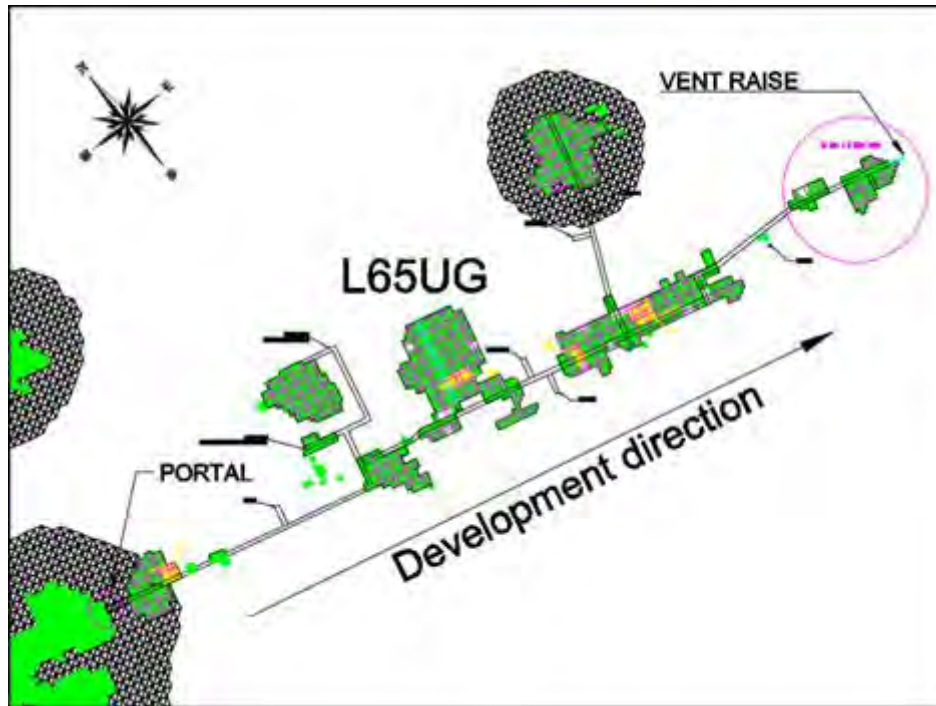


Figure 16-26: Plan view – C1 Area (not to scale)

16.5.3.5 Secondary Underground Egress

Secondary accesses are possible via ventilation raises. After the completion of decline/ramp development, ventilation raises (fresh air) are to be excavated in priority to create a second egress before the start of production. Therefore, these raises will be equipped with an escapeway from the bottom level of the mine to surface.

16.5.3.6 Primary Development

Starting from surface portals or from existing ramps, declines will be excavated by typical drill-and-blast development methods. In order to reach the productivity objectives, a dedicated crew will be assigned to ramp development. The average LOM ramp advance is expected to be 1,800 m/year for single -face excavations, and 3,600 m/year when multi-face excavations are available.

Sublevels will be developed at 25m vertical intervals. Each level or sublevel is accessed using crosscuts (5 m x 5 m) from the main ramp to allow effective loading and haulage. The ore and waste rock are hauled by LHD's from the production area to either a re-muck or a loading bay excavated close to the main ramp. The ore will then be loaded into 45-t trucks and hauled to surface.

16.5.3.7 Secondary Development

All secondary development will be excavated by a second development team. Level infrastructure will typically include:

- Ventilation raise access ;
- Sumps.
- Refuge stations cut out for portable refuge chamber ;
- Material storage bays (gear bays).
- Electrical substations.
- Backfill mixing stations.
- Underground explosives and cap magazines.
- Lunchrooms, if required

16.5.3.8 Stopes Access

When the longhole method is used, the development of the top access and the bottom access will be located in mineralized zones.

The development access for the room and pillar method is established when hitting the mineralized core. The stope access drifts will have a dimension of 5m x 5m, the same dimensions as the main ramp.

16.5.3.9 Underground Mineralized Material Transportation to Mill

Underground ore will be stockpiled near the portals from where it will be rehandled into haul trucks by a contractor for delivery to the Process Plant.

For the more distant West Zone, on-highway trucks will be loaded at the portal and will take Highway 5 across the Buffalo River and then Highway 6 up to the nearest off-highway haul road onwards to the Process Plant.

For the C1 Area, 45-t articulated trucks will transport the mineralized material to the mill.

16.5.4 Development Rates

During the development phase, development rates will vary depending on the number of active headings available to crews. (see Table 16-17).



Table 16-17: Development advance rates

UG Development	Dimensions (m)	Advance rate (m/yr)	Availability
Ramp	5 x 5	1,800	Single face
Lateral development	5 x 5	3,600	Multi-face
O/C & U/C development	5 x 5	3,600	Multi-face
Fresh air raise access	2.4 x 2.4	1,000	Single face

16.5.5 Mine Development Schedule

For each deposit mined from underground, mine development is planned to take less than two years. This period includes surface dewatering, portal excavation and construction, decline development, lateral and vertical development. For costing purposes, ramp, raise and lateral development are included in the (sustaining) CAPEX metres, while sill development (mineralized material) is included in OPEX metres. All development activities are planned to be carried out by contractor teams.

16.5.5.1 West Zone

The development schedule for the West Zone is shown in Table 16-18.

Table 16-18: Development Schedule – West Zone

Dev Teams (West Zone)			1	2	3	1	1	1	1	1
Years	Unit	Total	1	2	3	4	5	6	7	8
G03 Zone										
Ramp CAPEX	m	714	432	283	-	-	-	-	-	-
Lateral Development CAPEX	m	1,814	584	1,160	70	-	-	-	-	-
Vertical Development CAPEX	m	168	25	111	32	-	-	-	-	-
Waste Development OPEX	m	2,878	626	2,192	60	-	-	-	-	-
Ore development OPEX	m	4,113	929	2,896	289	-	-	-	-	-
O556 Zone										
Ramp CAPEX	m	1,216	-	-	-	-	-	219	885	112
Lateral Development CAPEX	m	856	-	-	-	-	-	-	227	629
Vertical Development CAPEX	m	153	-	-	-	-	-	-	18	135
Waste Development OPEX	m	1,709	-	-	-	-	-	-	1,034	675
Ore development OPEX	m	2,347	-	-	-	-	-	-	1,455	892
P499 Zone										
Ramp CAPEX	m	2,713	57	20	640	-	747	1,094	155	-



Dev Teams (West Zone)			1	2	3	1	1	1	1	1
Years	Unit	Total	1	2	3	4	5	6	7	8
Lateral Development CAPEX	m	1,344	-	-	-	-	-	832	513	-
Vertical Development CAPEX	m	121	-	-	-	-	-	-	121	-
Waste Development OPEX	m	1,031	-	-	-	-	-	587	444	-
Ore development OPEX	m	2,089	-	-	-	-	-	1,547	542	-
R190 Zone										
Ramp CAPEX	m	1,139	301	20	809	-	10	-	-	-
Lateral Development CAPEX	m	516	13	-	366	-	137	-	-	-
Vertical Development CAPEX	m	151	-	-	123	-	27	-	-	-
Waste Development OPEX	m	1,609	-	-	947	-	662	-	-	-
Ore development OPEX	m	1,943	-	-	1,078	-	865	-	-	-
X25 Zone										
Ramp CAPEX	m	2,659	242	-	1,003	936	477	-	-	-
Lateral Development CAPEX	m	1,016	-	-	346	515	155	-	-	-
Vertical Development CAPEX	m	194	-	-	-	76	118	-	-	-
Waste Development OPEX	m	4,645	-	-	562	2,922	1,161	-	-	-
Ore development OPEX	m	3,171	-	-	392	2,215	565	-	-	-
Total Lateral Development		39,526	3,183	6,570	6,562	6,588	4,780	4,280	5,254	2,308
Total Vertical Development		787	25	111	155	76	145	-	139	135

16.5.5.2 Central Zone

The development schedule for the Central Zone is shown in Table 16-19.

Table 16-19: Development schedule – Central Zone

Dev Teams (Central Zone)			1	3	2	1
Years	Total	Unit	6	7	8	9
M67UG						
Lateral Development CAPEX	175	m		175		
L65UG						
Lateral Development CAPEX	1,946	m		1,835	110	
M63UG						
Lateral Development CAPEX	12,890	m		2,087	803	
Total Lateral Metres	16,479	m	1,675	8,208	5,682	914
Total Vertical Metres	180	m	120	60	-	-

16.5.6 Production Forecast

The underground mine production rate starts at an average of 1,850 t/day in Year 4 and ramps up to an average of 4,250 t/day from Years 5 to 7. After Year 7, the mineralized material from underground mining comes mainly from the Central Zone with an average productivity around 1,500 t/day until Year 10. The overall underground mine life is expected to be approximately ten years of production. All production activities are to be conducted by contractor teams.

16.5.6.1 West Zone

Production in the West Zone will be by longhole methods, which allows for the production of stopes of nominally 29,925 t (diluted and recovered). The UG mine production rate will start at an average of 2,500 t/day (27 stopes/year) in Year 3 to ramp up to an average of 4,000 t/day (42 stopes/year) from Years 4 to 6. West Zone underground mining will end in Year 7 with the O556 deposit. Table 16-20 shows mineralized material production per deposit and per year.

Table 16-20: Production forecast including development in mineralized zone – West Zone

UG Deposits	Unit	Yr3	Yr4	Yr5	Yr6	Yr7
R190	t	944,831				
G03	t		1,000,000	214,631		
P499	t		460,000	129,484		
X25	t			1,115,885	738,863	
O556	t				721,137	61,920
Total	t	944,831	1,460,000	1,460,000	1,460,000	61,920

16.5.6.2 Central Zone

Since the underground mining production in the Central Zone is mainly carried out by the room and pillar method, the tonnes of mineralized material are linked directly to the development metres. The mineralized tonnes resulting from development metres are summarized in Table 16-21.

Table 16-21: Mineralized tonnes – Central Zone

UG Deposits	Unit	Yr7	Yr8	Yr9
M67UG	t	44,562		
L65UG	t	475,298		
M63UG	t		491,652	79,079

In addition, a portion of the mineralization thicker than 5 m will be mined using the uppers method. Uppers tonnes are summarized in Table 16-22.

Table 16-22: Production forecast for upper method only – Central Zone

UG Deposits	Unit	Yr7	Yr8	Yr9
L65UG	t	27,120		
M63UG	t		55,368	

16.5.7 Underground Mining Equipment

Underground mining equipment is to be provided by the contractor. The recommended mobile equipment fleet is summarized in Table 16-23.

Table 16-23: Underground mine equipment list

Type	Brand	Model	Main spec	Average annual quantity	Utilization (%)
Jumbo	Sandvik	DD311	2 Boom	2	75
LHD scoop	Caterpillar	R1600H	Bucket 8yd	2	90
Truck	Caterpillar	AD 45	45 t	3	90
Bolter	Sandvik	DS311	1 Boom	1	50
Production drill	CMAC	LH II	3" Diameter	1	90
Boom truck	MacLean	MiniMate BT2	1,8m wide platform	1	50
Personnel carrier	Toyota	Landcruiser	Standard	5	50
Fuel truck	MacLean	FL3	6,000 L capacity	1	50
Cement truck	MacLean	TM2	Shipping 14 t	1	50
Grader	Caterpillar	140 M	Blade 3,6m	1	50

16.5.8 Underground Mine Personnel

Underground mining personnel is to be supplied by the contractor. In addition to the open-pit engineering team, a dedicated underground Owner's team will be affected to supervise the contractor. Staff rosters, segmented by contractor and Owner teams, is tabulated in Table 16-24.

Table 16-24: Underground mine personnel list

Operators (included in Contractor Unit Costs)										
Contractor Operators	Units	Yr1	Yr2	Yr3	Yr4	Yr5	Yr6	Yr7	Yr8	Yr9
Construction Phase	#	22	41	19						
Development Phase	#		10	15	5	5	2	10	9	4
Production Phase	#			6	8	7	8	2	2	0
Technical services (payroll)	Units	Yr1	Yr2	Yr3	Yr4	Yr5	Yr6	Yr7	Yr8	Yr9
UG OPS Superintendent	#	1	1	1	1	1	1	1	1	1
UG Mine Foreman	#	0	0	0	0	0	0	0	0	0
Clerk	#	0	0	0	0	0	0	0	0	0
Mining Engineer	#	0	2	2	2	2	2	1	1	1
Mine Technician	#	0	2	4	4	4	4	1	1	1
Surveyors	#	0	0	0	0	0	0	0	0	0
Geologist	#	0	0	0	0	0	0	0	0	0

16.5.9 Mine Support Services

16.5.9.1 Underground Dewatering

Prior to starting any development of the underground workings, dewatering is required. The dewatering methodology for underground workings is similar from the one used for open-pit deposits and is specific for each underground area (W1 and C1).

16.5.9.1.1 West Zone (W1 Area)

The West Zone stratigraphy differs from other zones: the deposits are generally deeper (except for the G03 deposit) and hydraulic conductivity estimates from previous investigations and technical studies (Tetra Tech, 2020) suggest that dewatering in this area with conventional wells and submersible pumps is not economically viable and that groundwater control may be required.

Tamerlane (Tamerlane, 2012) considered using a Freeze Wall. A Freeze Wall consists of installing a ± 3 metres wide Freeze Wall around each deposit. The Freeze Wall would extend from surface down to a layer of rock with low permeability below the deposits. The walls would isolate the deposits from surrounding groundwater in the existing active aquifer. Dewatering would then be limited to the groundwater within the ring plus a relatively small amount of water entering the ring as basal inflow. Once the water level within the frozen ring is pumped lower than the mining levels, internal sumps and pumps would maintain control of water from the basal inflow and mining activities.

This design was reviewed and assessed by Thyssen. Several significant operational constraints were noted following this assessment. Thus, the Freeze Wall approach was not retained as a dewatering method and other options were considered.

16.5.9.1.2 Grouting/till injection

The grouting methodology was proposed by Peter White from Multieurethane, a grouting specialist, who was retained to consult on this area as it showed more potential to significantly reduce the quantity of water to pump as well as require leaner dewatering operational activities (White, 2020).

The methodology consists of building a low-permeability barrier by drilling multiple holes around the selected deposits, with a final distance of 4 m between each. Holes are then grouted using a mixture of bentonite, cement, and local till materials to create a grout curtain. This method is not specified for the X25 deposit since the dewatering needs for this site does not require a grout curtain.

For deposits R190, P499, O556 and G03, this technique requires dewatering wells based on the following criteria:

- Temporary surface casing drilled through overburden;
- 200 mm diameter drillholes to 60 m depth down to appropriate location;
- Steel casing installed at approximately 60 m and externally cemented;
- 150 mm diameter open drillhole drilled to the final depth (~183-200 m).

Once the wells are completed, suspension-based cement-bentonite grout mixtures are injected and will penetrate open fissures and highly porous dolomite with aperture openings greater than 500 microns.

For relatively short-term groundwater control applications, such as the extraction of the underground deposits in the West Zone, a target K value is assumed to be reduced by 1.5 order of magnitude.

The grouting methodology will need to be initiated before the start of production for each underground deposit.

16.5.9.1.3 West Zone Dewatering Methodology

In the W1 Area (except for the X25 deposit), pre-production dewatering will be required only for the water within the grout curtain. A number of dewatering wells will be implemented around the deposit, but inside the grout curtain. The wells will be built the same way as described in the

open -pit section, but will use different pumps due to higher discharge heads. On surface, the pumps will be connected to a network of ditches. The surface infrastructure for water connection and disposal are further described in Chapter 18.

16.5.9.1.4 C1 Area

The underground operations in the C1 Area will be different from the W1 Area, mostly because the access is from the bottom of the historic M64 pit and the proposed M67 pit, once it is mined out. These two pits provide access to the M67UG, L65UG, and M63UG deposits. Dewatering for underground mining in this area is planned to be achieved by pursuing a strategy of dewatering from the bottom of the M67 pit and by dewatering the historic M64 pit.

16.5.9.2 Backfill

Primary stopes are backfilled with cemented rock fill ("CRF") and secondary stopes are planned to be filled with unconsolidated rock fill.

For CRF, given the limited lifetime of each site (one year of production on average), the installation of an underground cement distribution system is not warranted. Cement slurry will be delivered to the underground mixing bay in a trans-mixer truck. Waste rock, from surface waste rock stockpiles or directly from underground waste development will be transported to mixing stations by 45-t trucks. Loading and haul time for the waste was considered in the cycle time calculations for the backfilling phase. LHD units will then mix the slurry with waste and deliver the mixed CRF material to primary mined-out stopes. The CRF used underground will respect the standard practice of using 4% w/w cement content and a curing time of 14 days.

Waste rock from development activities will be used as rockfill in secondary stopes, hauled by 45-t trucks and backfilled directly into the stopes.

16.5.9.3 Ventilation

16.5.9.3.1 Fresh Air Requirement

The fresh air requirements for each of the underground operations is determined in accordance with the Northwest Territories Mining Occupational Health and Safety Regulations (R-125-95).

Paragraph 2 of section 10.62 of this regulation stipulates that the ventilation rate shall be at least 0.06 m³/s for each kilowatt of diesel-powered machinery used in the workplace.

To estimate the overall volume of fresh air to be supplied for each mine, a utilization rate is applied. This utilization rate considers operational time for every piece of equipment. In general, the utilization rates are: 90% for production equipment and 50% for service equipment. A contingency of 10% is applied to the fresh air requirements to allow for vehicles which may enter headings on an irregular basis. This also includes an allowance for any potential leaks in the ventilation network.

Table 16-25 presents the mobile equipment list and air demands in the West Zone.

Table 16-25: Mobile equipment list and air demands for the West Zone

Type of Equipment	Model	Main Spec	Power (KW)	Quantity	Utilization (%)	Air requirement (CFM)
Jumbo	DD311	1 Boom	20	2	50	2,542
LHD Scoop	R1600H	Bucket 8v	208	2	90	47,579
Truck	AD 45	45t	447	3	90	153,373
Bolter	DS311	1 Boom	10	1	50	635
Production Drill	LH II	3" Diameter	75	1	90	8,578
Boom Truck	MiniMate BT2	6'w platform	110	1	50	6,989
Personnel Carrier	Landcruiser	Standard	96	5	50	30,499
Fuel truck	FL3	6,000 L capacity	110	1	50	6,989
Cement truck	TM2	Shipping 14t	110	1	50	6,989
Grader	140 M	Blade 12'	174	1	50	11,056
Sub-total						275,230
Contingency (10%)						27,523
Total						302,753

As seen in Table 16-25, the maximum fresh air demand required for each deposit, including the 10% contingency is estimated at 303,000 CFM (143 m³/s). Depending on the schedule and production rate, the demand for fresh air will vary but never exceed the value above.

Infrastructure and Ventilation Network

The ventilation system for each of the five deposits in the West Zone is planned to perform as follows:

- During the development phase, fresh air is pushed by two 150 hp (112 kW) fans, installed in parallel at the portal. Each fan is connected to a 48" (1,200 mm) rigid ventilation tubing along the ramp. Return air exhausts through the ramp. The air flow required during the development phase is 100,000 cfm (47 m³/s) until the main ventilation raise is established.

Figure 16-27 shows the direction of air flow during the development phase at R190.

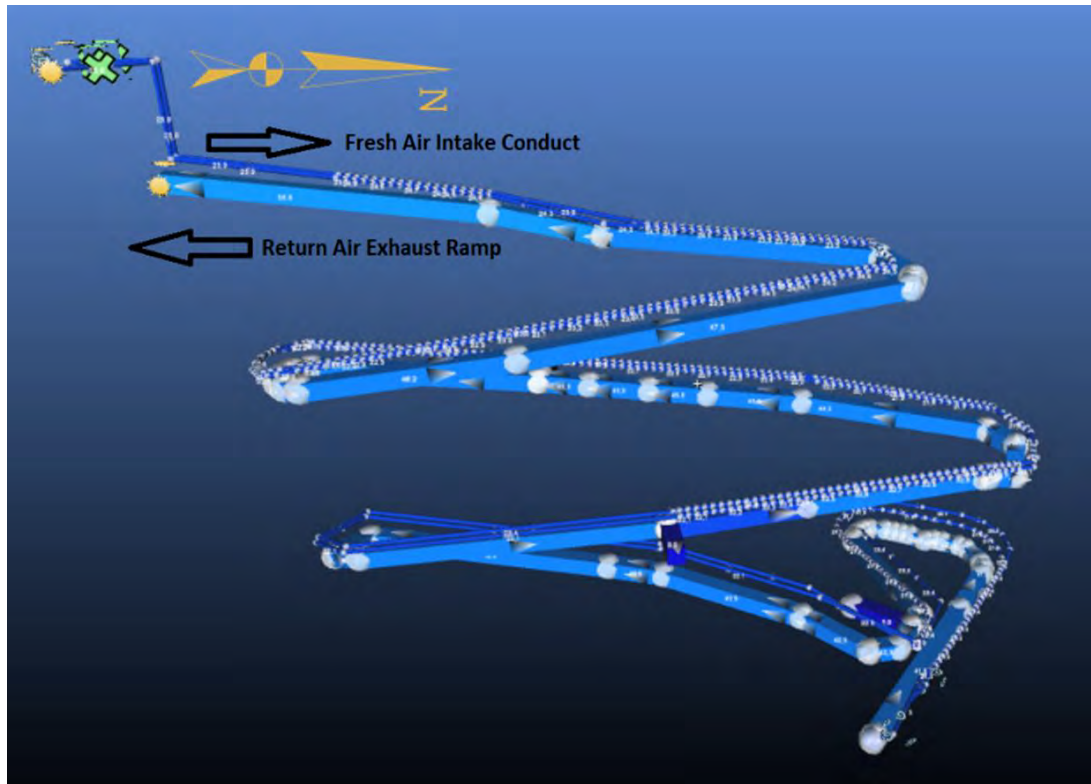


Figure 16-27: Development phase ventilation (R190) isometric view looking west (concept, not to scale)

- During the production phase, fresh air is pushed by two 200 hp (149 kW) fans, installed in parallel above the ventilation raise. Under these conditions, the fresh air flow is 303,000 cfm (143 m³/s) to supply the required volume. Return air exhausts through the ramp towards the surface.

Figure 16-28 shows air flow direction during the production phase at R190.

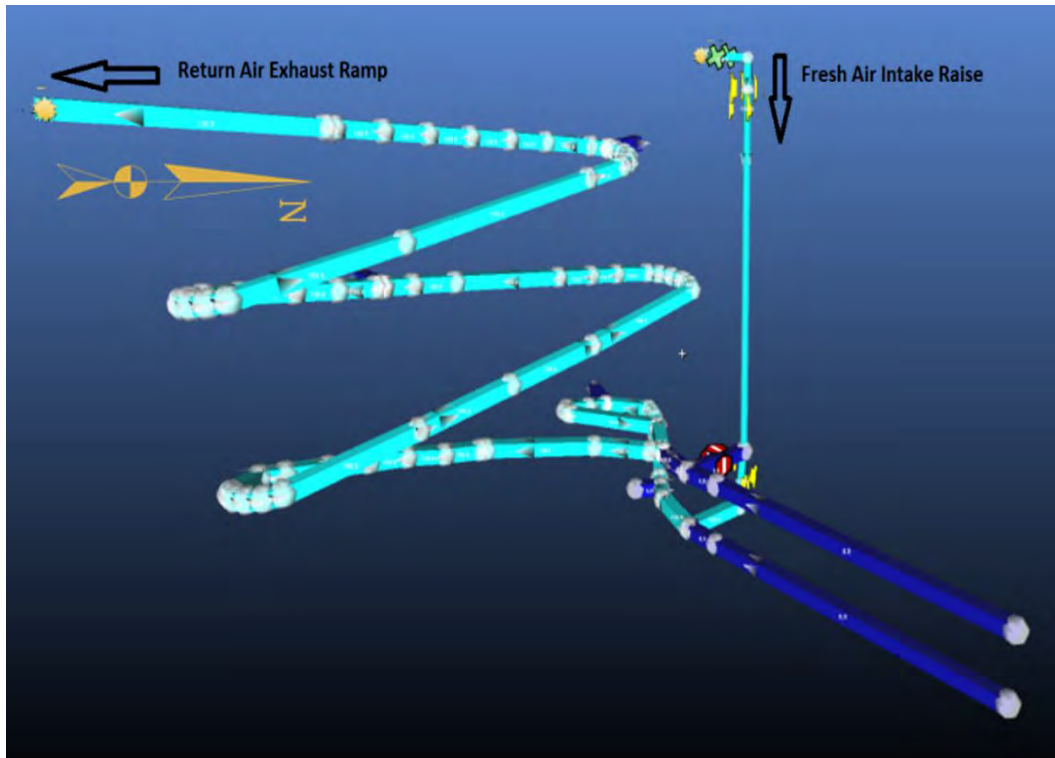


Figure 16-28: Production phase (R190) isometric view looking west (concept, not to scale)

To optimize the mining schedule for the G03 deposit, two ventilation raises are required. These ventilation raises are planned to be excavated immediately after development of the connection ramp is completed. At the top of each raise, a 200 hp (149 kW) fan will supply 150,000 cfm (71 m³/s) of fresh air into the raise.

Figure 16-29 shows the direction of air flow during the production phase at G03.

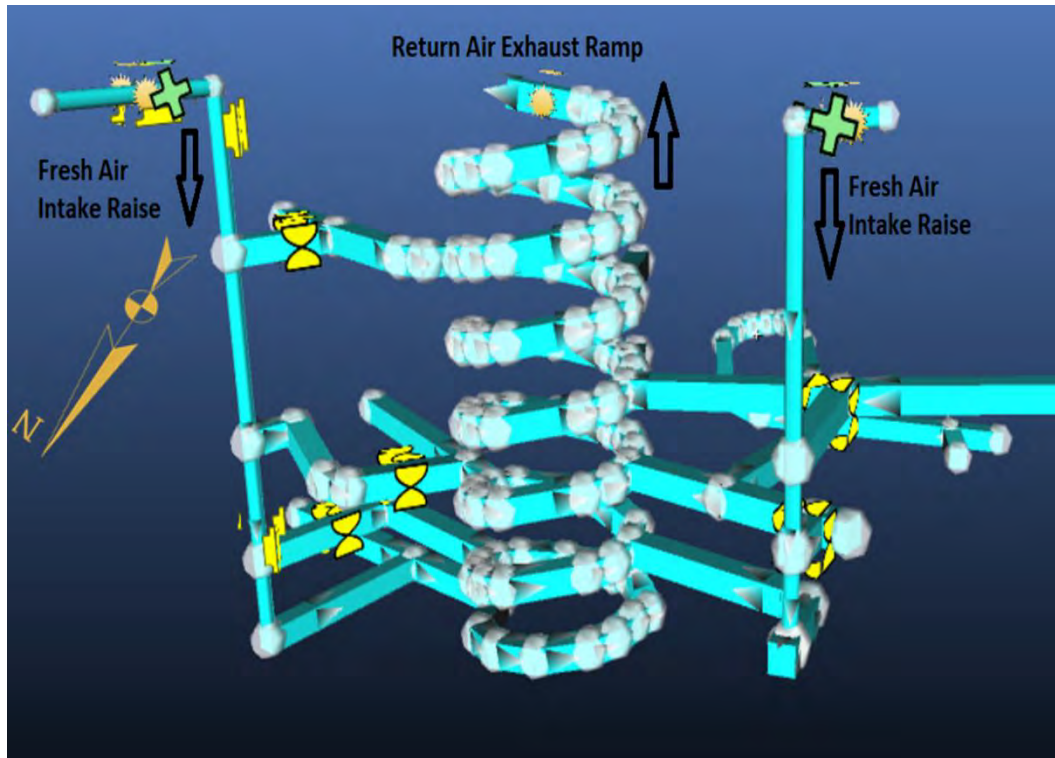


Figure 16-29: Production phase ventilation (G03) isometric view looking east (concept, not to scale)

Fan Models

The selected fan models are presented in Table 16-26.

Table 16-26: Selected fan models

Model	Location	Engine power	Flow required per fan	Total pressure	Ventilation system
HVT 60-30-1800_60Hz Full bladed	Top raise (surface)	200 hp	150 kcfm	9'' water	Main ventilation
HVT 42-26-1800_60Hz Full blade type 3000	Top ramp (surface)	150 hp	50 kcfm	12'' water	Pre-production ventilation
HVT 38-26-1800_60Hz Full bladed	Production level	50 hp	30 kcfm	7'' water	Auxiliary ventilation

Heating system

Ventilating air must be heated during winter months. The heating system will be a direct-combustion propane heater. The selected heating units are 5.84 MMBTU/h, which can heat up to 50,000 cfm (24 m³/s). For 303 000 cfm (143 m³/s), six units are required, for a capacity of 35 MMBTU/h.

Auxiliary ventilation

Auxiliary ventilation, to be installed for delivery of air into dead-end headings, consists of a 50-hp (37 kW) fan coupled with 42" (1,067 mm) diameter flexible ducting. The flow rate delivered to mucking drifts is estimated to be 30,000 cfm (14 m³/s) at total pressure of 7" (18 mm) water gauge.

16.5.9.3.2 Central Zone

Table 16-27 presents the mobile equipment list and air demands for the C1 deposits. Given the small size of the M67UG deposit, the hauling of mineralized material for this site does not require the use of trucks. The equipment used is listed in Table 16-27

Table 16-27: Mobile equipment list for the Central Zone

Type of Equipment	Model	Specs	Main Spec	Power (KW)	Quantity	Utilization (%)	Air requirement (CFM)
Jumbo	Sandvik	DD311	1 Boom	20	1	50	1,271
LHD Scoop	Caterpillar	R1600H	Bucket 8v	208	1	90	23,789
Bolter	Sandvik	DS311	1 Boom	10	1	50	635
Boom Truck	MacLean	MiniMate BT2	6'w platform	110	1	50	6,989
Personnel Carrier	Toyota	Landcruiser	Standard	96	1	50	6,100
Sub-total							38,784
Contingency (10%)							3878
Total							42,662

As shown in Table 16-27, the maximum fresh air demand required at L65UG and M63UG, including the 10% contingency, is estimated at 45,000 cfm (21 m³/s).

- During the development phase, a 300 hp (224 kW) fan will be installed at the entrance of the ramp to supply 75,000 cfm (71 m³/s) through 1,000 m of 48'' (1,200 mm) rigid ventilation tubing. When the ventilation raise is excavated, a 100 hp (75 kW) fan is installed at the top of this raise.
- During the production phase, this fan will supply 100 000 cfm (47 m³/s) to the lowest level.
- For the small M67UG deposit, a 50 hp (37 kW) fan is to be installed at the portal. This setup is sufficient to ventilate the mine during two months of production. Ore and waste will be hauled by LHD.

16.5.9.4 In-mine Dewatering

In-mine dewatering (during production) is planned to be managed by 40 hp (30 kW) submersible pumps installed in dewatering sumps. The sumps are strategically positioned at the bottom of the ramp to collect the water flow from the ramp and the ventilation raise, as well as mine water (drilling water, cleaning water, etc.). The 40 hp (30 kW) pumps are estimated to be used 50% of the time and are moved out after mining completion to allow reuse.

16.6 Mine Production Schedule

The mining production schedule of a multi-deposit project has to take into account several aspects. The proposed schedule for the Project is a balance between the numerous deposits and their respective characteristics, considering various constraints including multiple mining methods, while aiming to optimize the cashflow. The West Zone deposits are high grade but require higher capital investments since they are mined from underground. Dewatering and haulage requirements would promote mining pits by zones or areas, but the nearby deposits can have very variable tonnage, grades and strip-ratios, which impacts their profitability.

The open-pit mining sequence is based on the profit margin per pit. The profit per tonne was estimated for each pit based on preliminary costs including dewatering, haulage and processing. The pits were sorted from the highest profit margin to the lowest and the sequence was developed based on defined constraints such as:

- 25% of the yearly production must come from the East Mill Zone, allowing access to mineralized material near the Process Plant and requiring less dewatering, as a contingency measure for the freshet;



- A maximum of four clusters can be mined simultaneously, to facilitate logistics and benefit from the cumulative effects of dewatering while prioritizing pits with higher profit margins.

Generally, the pits that contain less than 1 Mt of mineralized material are mined within one year. The pits that have more than 1 Mt are mined in two to three years.

There is a maximum of eight pits in operation per year but less than four pits operated simultaneously.

Underground mining has been set to start in Year 3 to delay capital costs. The underground mining sequence is also based on the relative profit margin while considering the proximity (connection ramp) and underground mining methods for the production rate and reuse of equipment (West Zone completed before Central Zone).

The open-pit mining schedule was adjusted to be balanced with the 4,000 tpd delivered from the underground operations between Years 3 to 7, and 1,500 tpd from Years 7 to 9.

The overall Mining Development and Production Schedule is presented in Table 16-28, Table 16-29, Table 16-30 and Table 16-31.



Table 16-28: Mine production schedule for underground area C1

Area C1	Unit	Total	7	8	9	10
M63						
Waste mined	DR tonnes	150,928	111,235	39,693	-	-
Ore mined	DR tonnes	588,281	25,592	13,333	186,449	362,906
Total material mined	DR tonnes	739,209	136,827	53,027	186,449	362,906
M67						
Waste mined	DR tonnes	8,162	8,162	-	-	-
Ore mined	DR tonnes	156,723	3,351	153,373	-	-
Total material mined	DR tonnes	164,885	11,512	153,373	-	-
L65						
Waste mined	DR tonnes	101,165	93,982	7,184	-	-
Ore mined	DR tonnes	600,473	26,296	30,627	543,550	-
Total material mined	DR tonnes	701,639	120,277	37,811	543,550	-
Total						
Waste mined	DR tonnes	260,256	213,379	46,877	-	-
Ore mined	DR tonnes	1,345,478	55,238	197,334	730,000	362,906
Total material mined	DR tonnes	1,605,734	268,617	244,211	730,000	362,906



Table 16-29: Mine production schedule for underground area W1

Area W1	Unit	Total	1	2	3	4	5	6	7	8
G03										
Overburden mined	DR tonnes	469	469	-	-	-	-	-	-	-
Waste mined	DR tonnes	398,366	126,707	260,137	11,522	-	-	-	-	-
Ore mined	DR tonnes	2,032,686	53,287	183,295	42,083	-	-	1,392,329	361,693	-
Total material mined	DR tonnes	2,431,521	180,462	443,432	53,605	-	-	1,392,329	361,693	-
O556										
Overburden mined	DR tonnes									
Waste mined	DR tonnes	273,485	-	-	-	-	-	18,279	165,184	90,021
Ore mined	DR tonnes	1,266,572	-	-	-	-	-	-	443,394	823,178
Total material mined	DR tonnes	1,540,057	-	-	-	-	-	18,279	608,578	913,199
P499										
Overburden mined	DR tonnes									
Waste mined	DR tonnes	408,240	4,712	1,666	53,306	-	62,211	200,760	85,584	-
Ore mined	DR tonnes	838,508	-	-	-	-	-	89,958	748,550	-
Total material mined	DR tonnes	1,246,747	4,712	1,666	53,306	-	62,211	290,718	834,134	-
R190										
Overburden mined	DR tonnes	7,015	7,015	-	-	-	-	-	-	-
Waste mined	DR tonnes	236,202	16,667	1,661	163,492	-	54,383	-	-	-
Ore mined	DR tonnes	1,172,908	-	-	435,983	-	669,255	67,671	-	-
Total material mined	DR tonnes	1,416,126	23,682	1,661	599,475	-	723,637	67,671	-	-
X25										
Overburden mined	DR tonnes	9,496	9,496	-	-	-	-	-	-	-
Waste mined	DR tonnes	601,939	7,255	-	137,811	325,892	130,981	-	-	-
Ore mined	DR tonnes	2,329,483	-	-	191,427	1,228,222	883,885	-	25,949	-
Total material mined	DR tonnes	2,940,918	16,751	-	329,238	1,554,114	1,014,866	-	25,949	-
Total										
Overburden mined		16,980	16,980	-	-	-	-	-	-	-
Waste mined		1,918,232	155,341	263,464	366,132	325,892	247,575	219,039	250,768	90,021
Ore mined		7,640,157	53,287	183,295	669,493	1,228,222	1,553,140	1,549,958	1,579,586	823,178
Total material mined		9,575,369	225,607	446,758	1,035,624	1,554,114	1,800,714	1,768,997	1,830,354	913,199



Table 16-30: Detailed production schedule for Open-pit and Underground

Diluted Tonnages and grades per Pit									Open-pit Production Schedule - Mineralized Material (tonnes)											
Pit	Area	Mineralized Material	Zn (%)	Pb (%)	Overburden	Waste Rock	Total Waste	Strip Ratio	Y1	Y2	Y3	Y4	Y5	Y6	Y7	Y8	Y9	Y10	Y11	Y12
J68	C1	319,966	7.20	3.39	1,469,488	1,446,139	2,915,627	9.11			319,966									
K60	C1	158,983	4.13	6.32	916,089	1,446,139	2,362,227	14.86		158,983										
K68	C1	1,009,548	5.25	1.42	2,817,589	1,406,377	4,223,965	4.18			1,009,548									
M67	C1	617,754	5.08	1.61	2,176,446	10,822,250	12,998,696	21.04		617,754										
K51	C2	328,737	4.28	1.29	1,978,624	6,740,692	8,719,317	26.52									328,737			
K52	C2	319,792	4.60	1.28	1,902,668	975,982	2,878,651	9.00									319,792			
K57	C2	412,048	3.08	1.51	671,002	2,478,703	3,149,705	7.64								39,017	373,031			
M48	C2	135,427	5.36	2.07	471,004	1,012,001	1,483,004	10.95								135,427				
O53	C2	596,053	5.43	1.58	2,045,660	1,118,415	3,164,075	5.31								556,855	39,198			
R67	C3	439,691	7.43	2.36	1,270,941	1,218,729	2,489,670	5.66	439,691											
X49	N1	877,937	3.69	1.49	2,210,243	367,115	2,577,358	2.94				877,937								
X55	N1	314,874	2.93	0.58	863,205	549,115	1,412,320	4.49				314,874								
X59	N1	247,206	4.58	1.13	1,266,123	682,300	1,948,423	7.88					165,832	81,374						
X60	N1	295,420	5.66	1.38	1,473,086	1,120,883	2,593,969	8.78				78	295,342							
X61	N1	1,103,236	4.06	1.63	1,319,298	1,415,525	2,734,823	2.48				749,195	354,040							
X68	N1	732,262	3.79	0.78	2,946,369	1,437,138	4,383,508	5.99								732,262				
X71	N1	2,861,234	2.84	2.11	5,208,624	1,980,554	7,189,177	2.51							1,017,752	1,843,482				
Y53	N1	340,180	8.67	2.99	1,650,464	8,050,487	9,700,951	28.52			240,423	99,756								
Y55	N1	249,460	4.10	0.92	1,639,741	2,247,470	3,887,211	15.58			249,460									
Y56	N1	147,028	4.12	0.45	489,311	443,456	932,766	6.34				147,028								
Y57	N1	166,332	2.44	0.93	460,617	104,338	564,955	3.40				166,332								
Y58	N1	147,214	3.06	0.63	483,785	70,324	554,110	3.76				147,214								
Y62	N1	919,540	3.56	1.66	3,722,848	309,014	4,031,862	4.38					919,540							
Y65	N1	2,185,893	4.58	1.34	7,917,121	3,057,792	10,974,913	5.02					109	2,178,511	7,273					
Y68	N1	1,658,692	3.74	1.48	6,018,432	3,732,775	9,751,207	5.88						65,935	1,592,757					
Z60	N1	673,931	4.73	0.92	2,568,216	2,815,340	5,383,556	7.99				673,931								
W85	N2	1,649,645	4.93	2.33	2,412,428	556,052	2,968,480	1.80	596,378	1,053,268										
YBM	N2	984,559	3.89	3.10	695,709	7,440,122	8,135,832	8.26	524,901	459,658										
N204A	N204	6,992,380	3.43	0.96	1,967,619	3,545,839	5,513,457	0.79									117,888	954,668	5,238,653	681,171
N204B	N204	498,732	2.76	0.83	80,491	29,714,581	29,795,072	59.74									498,732			
N204C	N204	241,928	3.55	1.03	215,661	1,176,968	1,392,628	5.76									241,928			
T37	NE1	448,583	5.73	2.03	448,133	1,207,153	1,655,286	3.69	448,583											
X17	NE2	102,539	5.61	1.14	58,532	979,316	1,037,847	10.12												102,539





Diluted Tonnages and grades per Pit									Open-pit Production Schedule - Mineralized Material (tonnes)											
Pit	Area	Mineralized Material	Zn (%)	Pb (%)	Overburden	Waste Rock	Total Waste	Strip Ratio	Y1	Y2	Y3	Y4	Y5	Y6	Y7	Y8	Y9	Y10	Y11	Y12
X18	NE2	246,755	3.05	0.82	63,801	130,532	194,333	0.79												246,755
L24	EM	63,552	4.55	1.92	82,553	1,778,683	1,861,237	29.29				49,732	13,820							
L25	EM	273,749	2.39	0.74	316,990	217,031	534,021	1.95								273,749				
L26	EM	115,334	3.03	0.52	256,174	148,791	404,964	3.51								115,334				
L27	EM	1,457,672	3.70	0.76	2,053,951	90,222	2,144,173	1.47					148	182,967	635,718	638,839				
L35	EM	4,064,386	3.67	1.45	4,941,283	5,378,947	10,320,230	2.54		646,231	1,718,829	1,699,325								
M40	EM	1,120,732	4.04	1.01	2,305,386	16,066,494	18,371,879	16.39	46,738	1,073,994										
N31	EM	202,016	3.28	1.01	402,108	6,126,428	6,528,536	32.32					202,016							
N32	EM	370,975	2.32	0.88	392,001	267,742	659,743	1.78								370,975				
N33	EM	232,267	3.79	1.14	378,550	360,810	739,361	3.18					232,267							
N34	EM	51,109	4.01	0.29	104,049	308,357	412,406	8.07					50,433	676						
N39	EM	1,182,146	4.57	1.16	1,481,937	138,802	1,620,738	1.37	1,182,146											
O26	EM	62,350	4.54	1.43	139,907	6,066,656	6,206,563	99.54				125	62,225							
O27	EM	289,453	3.57	0.77	319,227	111,282	430,509	1.49					289,453							
Total Open-pit		37,909,301	3.98	1.44	75,073,482	138,859,860	213,933,342	5.64	3,238,436	4,009,889	3,538,227	4,925,529	2,585,225	2,509,463	3,253,500	4,705,940	1,919,306	954,668	5,238,653	1,030,465

Diluted Tonnages and grades per Underground deposit										Underground Production Schedule - Mineralized Material (tonnes)											
UG Deposit	Area	Mineralized Material	Zn (%)	Pb (%)	ZnEq. (%)	Overburden	Waste Rock	Total Waste	Strip Ratio	Y1	Y2	Y3	Y4	Y5	Y6	Y7	Y8	Y9	Y10	Y11	Y12
C1_UG	C1	1,345,478	5.37	1.38	6.76	-	260,256	260,256	0.19	-	-	-	-	-	-	55,238	197,334	730,000	362,906	-	-
G03	W1	2,032,686	5.14	3.95	9.09	469	398,366	398,835	0.20	53,287	183,295	42,083	-	-	1,392,329	361,693	-	-	-	-	-
X25	W1	2,329,483	6.85	2.42	9.27	9,496	601,939	611,435	0.26	-	-	191,427	1,228,222	883,885	-	25,949	-	-	-	-	-
R190	W1	1,172,908	9.93	4.99	14.92	7,015	236,202	243,218	0.21	-	-	435,983	-	669,255	67,671	-	-	-	-	-	-
P499	W1	838,508	6.07	2.74	8.81	-	221,862	221,862	0.26	-	-	-	-	-	89,958	748,550	-	-	-	-	-
O556	W1	1,266,572	3.91	3.40	7.31	-	207,921	207,921	0.16	-	-	-	-	-	-	443,394	823,178	-	-	-	-
Total UG		8,985,635	6.16	3.11	9.27	16,980	1,926,546	1,943,526	0.22	53,287	183,295	669,493	1,228,222	1,553,140	1,549,958	1,634,824	1,020,512	730,000	362,906	-	-
Total OP + UG		46,894,936	9.37	4.28	13.65	75,090,462	140,786,406	215,876,869	4.60	3,291,723	4,193,184	4,207,720	6,153,751	4,138,364	4,059,421	4,888,324	5,726,452	2,649,306	1,317,574	5,238,653	1,030,465





Table 16-31: Mine production table

	LOM	Y1-Q1	Y1-Q2	Y1-Q3	Y1-Q4	Y1	Y2	Y3	Y4	Y5	Y6	Y7	Y8	Y9	Y10	Y11	Y12
Pits Open		6	6	6	6	6	6	5	12	12	5	4	9	7	1	1	3
Open-pit Production																	
Mineralized Material (t)	37,909,301	323,844	809,609	971,531	1,133,453	3,238,436	4,009,889	3,538,227	4,925,529	2,585,225	2,509,463	3,253,500	4,705,940	1,919,306	954,668	5,238,653	1,030,465
Zn Grade	3.98	4.87	4.87	4.87	4.87	4.87	4.56	4.73	3.90	4.04	4.37	3.24	3.61	3.52	3.20	3.48	3.65
Pb Grade (%)	1.44	2.13	2.13	2.13	2.13	2.13	1.94	1.59	1.30	1.39	1.25	1.63	1.38	1.15	0.90	0.97	1.05
Total Waste (t)	212,797,645	3,143,657	7,859,141	9,430,970	11,002,798	31,436,566	31,552,082	23,825,936	15,204,470	17,489,774	17,565,536	16,821,501	15,296,836	14,505,695	15,470,334	11,186,348	2,442,567
Ovb Tonnage (t)	75,073,482	861,453	2,153,633	2,584,360	3,015,087	8,614,534	10,784,793	5,362,879	9,518,600	10,921,033	11,514,576	5,669,968	6,916,151	4,851,157	797,450	10	122,332
Waste Tonnage (t)	137,724,163	2,282,203	5,705,508	6,846,610	7,987,711	22,822,032	20,767,289	18,463,058	5,685,870	6,568,741	6,050,960	11,151,533	8,380,685	9,654,538	14,672,884	11,186,338	2,320,235
Underground Production																	
Mineralized Material (t)	8,985,635	0	0	0	53,287	53,287	183,295	669,493	1,228,222	1,553,140	1,549,958	1,634,824	1,020,512	730,000	362,906		
Zn Grade	5.77				5.19	5.19	4.41	8.72	7.00	8.10	5.68	5.09	3.62	1.29	5.42		
Pb Grade (%)	3.03				2.12	2.12	3.76	4.08	2.67	3.41	3.87	3.02	3.23	0.40	1.49		
Total Production																	
Ore Production (t)	46,894,936	323,844	809,609	971,531	1,186,739	3,291,723	4,193,184	4,207,720	6,153,751	4,138,364	4,059,421	4,888,324	5,726,452	2,649,306	1,317,574	5,238,653	1,030,465



17. RECOVERY METHODS

This chapter describes the process plant that is designed to process 11,250 tpd (4.1 Mtpa) of lead and zinc mineralized material. The Process plant flowsheet, design and capacity remain the same as proposed in the July 2020 Preliminary Economic Assessment ("PEA"). The recovery methods are established on the metallurgical testwork results and historical operating data provided in Chapter 13 and typical design parameters. The resulting flowsheet forms the basis for the plant design and plant capital and operating costs development.

The process plant consists of a three-stage crushing circuit incorporated with mineral sorting followed by grinding, lead and zinc differential flotation, concentrate and tailings dewatering. Historical and new mined out open-pits will be used for tailings disposal.

A schematic process flow diagram is presented in Figure 17-1 and a general 3D schematic of the process plant area is presented in Figure 17-2.



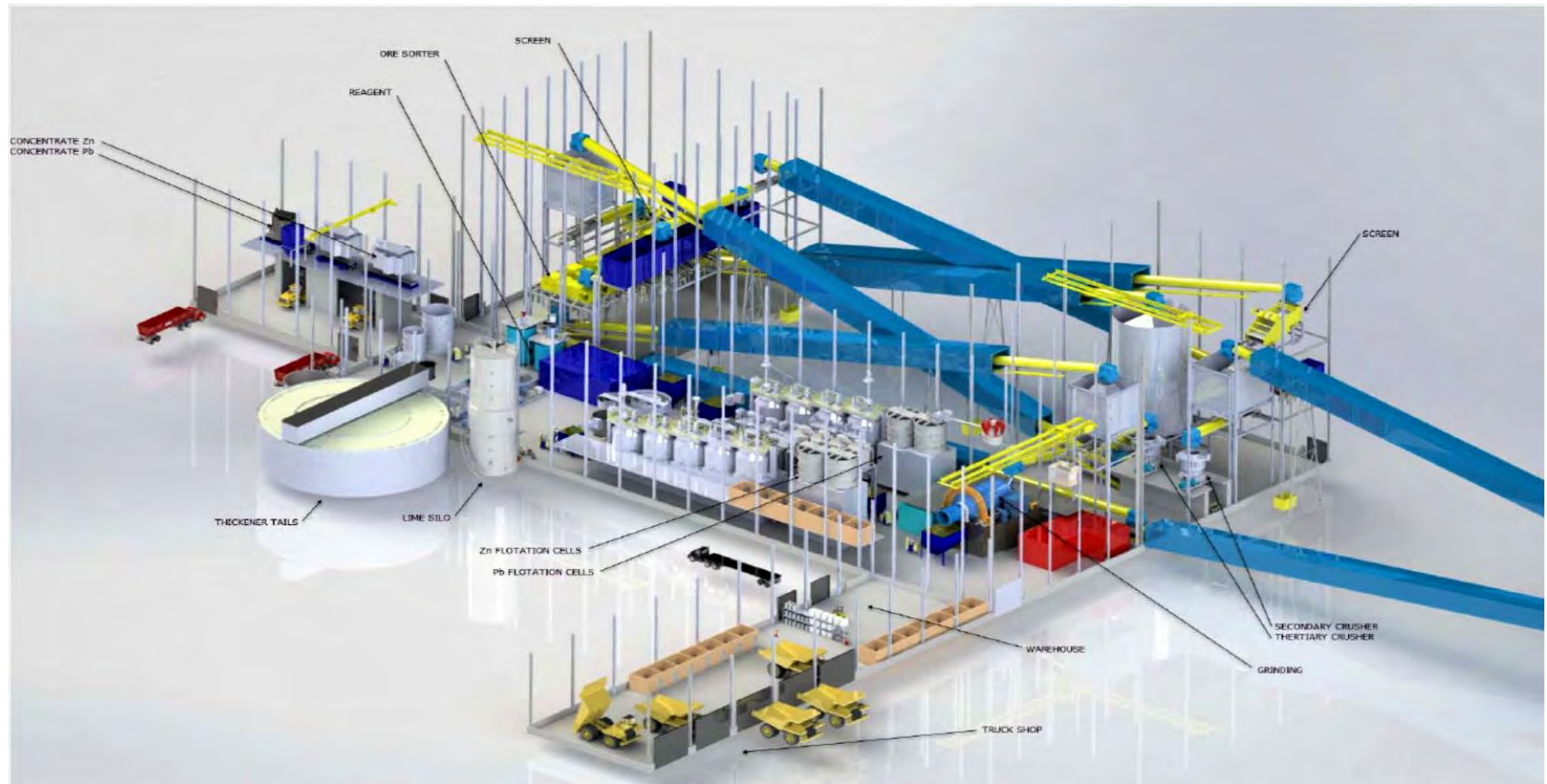


Figure 17-2: Process plant 3D general layout



17.1. Process Plant Design Criteria

The design criteria to determine the equipment sizes are presented in Table 17-1.

Table 17-1: Process plant design basis

Description	Unit	Design value	Source
Crushing throughput	tpd	11,250	Provided by Osisko Metals
Crusher availability	%	75	Recommended by BBA
Design lead grade	%	3.7	Provided by Osisko Metals
Design zinc grade	%	6.9	Provided by Osisko Metals
Mineral sorting throughput	tpd	7,875	Calculation
Mineral sorting concentrate mass pull	%	45.8	Metallurgical testwork
Mineral sorting lead recovery	%	98.8	Metallurgical testwork
Mineral sorting zinc recovery	%	91.9	Metallurgical testwork
Mill throughput	tpd	6,982	Calculation
Mill availability	%	92	Recommended by BBA
Grind size to flotation	µm	100	Metallurgical testwork
Flotation lead head grade - design	%	5.7	Calculation
Flotation zinc head grade - design	%	10.2	Calculation
Lead flotation circuit retention time	min	41	Metallurgical testwork
Lead concentrate grade	%	65	Metallurgical testwork
Lead concentrate	tpd	580	Calculation
Zinc flotation circuit retention time	min	50	Metallurgical testwork
Zinc concentrate grade	%	60	Metallurgical testwork
Zinc concentrate	tpd	1,075	Calculation
Overall lead recovery	%	93	Metallurgical testwork
Overall zinc recovery	%	87	Metallurgical testwork

17.2. Crushing and Mineral Sorting Circuit

The crushing circuit is designed to be fed 4.1 Mt of mineralized material per year. The primary crushing circuit is composed of a vibrating grizzly feeder followed by a 160 kW jaw crusher. The crusher reduces the feed size from a maximum of 500 mm to a P_{80} of approximately 90 mm with a 105 mm closed side setting. The crushed material is transported to the crushed mineralized material bin in the secondary crushing area via a conveyor. Figure 17-3 illustrates the primary crushing arrangement in a 3D model.

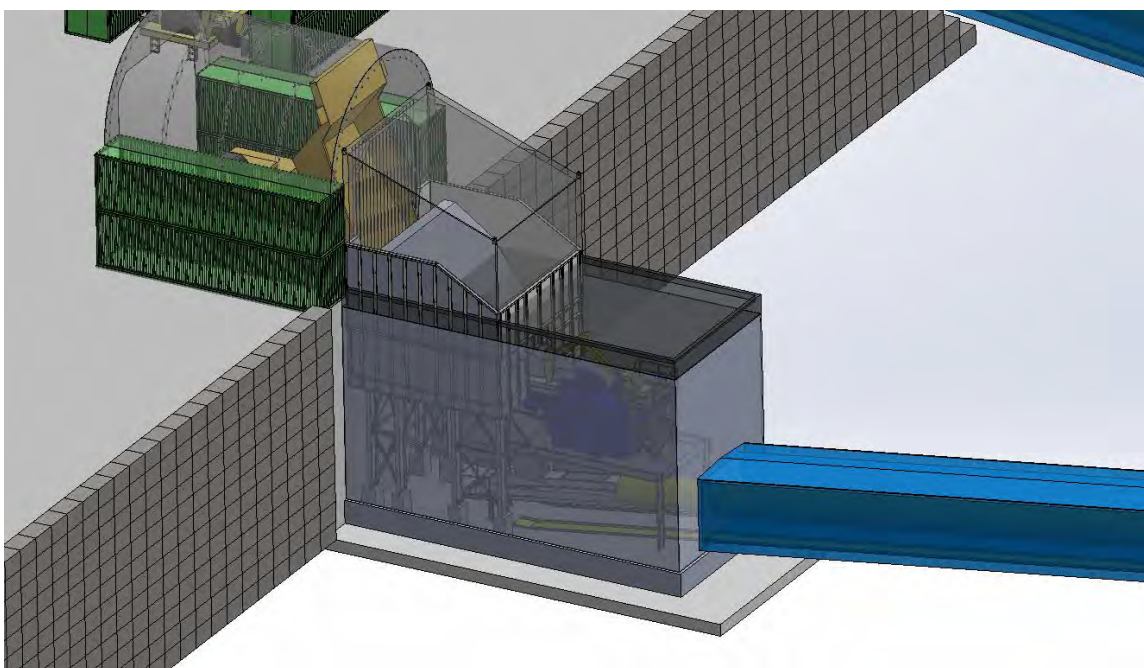


Figure 17-3: Primary crushing arrangement

Primary crushed material will be further crushed by a secondary cone crusher (600 kW) at a 50 mm closed side setting and the discharge will be fed to a classification screen to separate the fine material (<10 mm). The fine material will be transferred directly to the fine mineralized material bin. The screen oversize is transferred to a secondary classification screen.

The secondary screen undersize, sized between 50 mm and 10 mm, will be sent to the mineral sorting circuit consisting of seven parallel mineral sorters at a design rate of 438 tph. The mineral sorting will be done by using X-ray sensing technology. Each sorter is equipped with an air injection system that will eject the detected denser mineralized particles to the concentrate belt. The air for the injection system will be provided by 300 kW air compressors dedicated to each mineral sorter. The mineral sorting concentrate mass pull is designed for 46%. The concentrate will be

conveyed to a tertiary cone crusher (600 kW) to be crushed under 10 mm at a 13 mm closed side setting. The tertiary crusher discharge will be transferred to the primary classification screen to close the circuit. Crushed mineral sorting concentrate is combined with the unsorted fines in the fine mineralized material bin to be conveyed to the grinding circuit. The mineral sorting rejects will be conveyed and stockpiled prior to being transferred by truck for disposal in selected areas near the open-pit operations.

Figure 17-4 illustrates the secondary and tertiary crushing arrangement with mineral sorting from the 3D model.

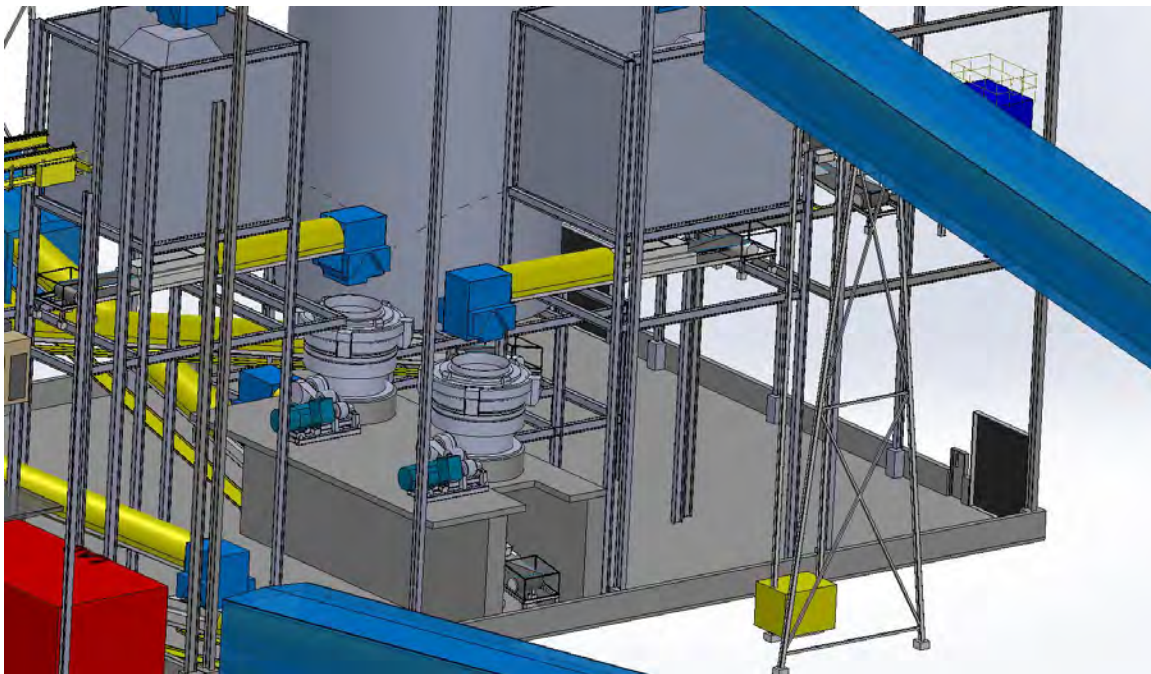


Figure 17-4: Secondary and tertiary crushing arrangement

17.3. Grinding Circuit

The mineral sorted material is ground in a ball mill, Ø4.88 m x 5.80 m (16' diameter x 19' length) with 3,000 hp motor installed, to a P_{80} of 100 μm , which operates in a closed circuit with a cyclone cluster. The production rate is designed to be 316 tph with 92% availability over 24-hr operation. The product fineness of the grinding circuit is monitored by an online particle size analyzer receiving a sample of the cyclone overflow slurry.

Zinc sulphate (ZnSO_4) will be added into cyclone underflow. Cyclone overflow will flow by gravity to the lead flotation circuit. Figure 17-5 illustrates the ball mill cyclone cluster arrangement from the 3D model.

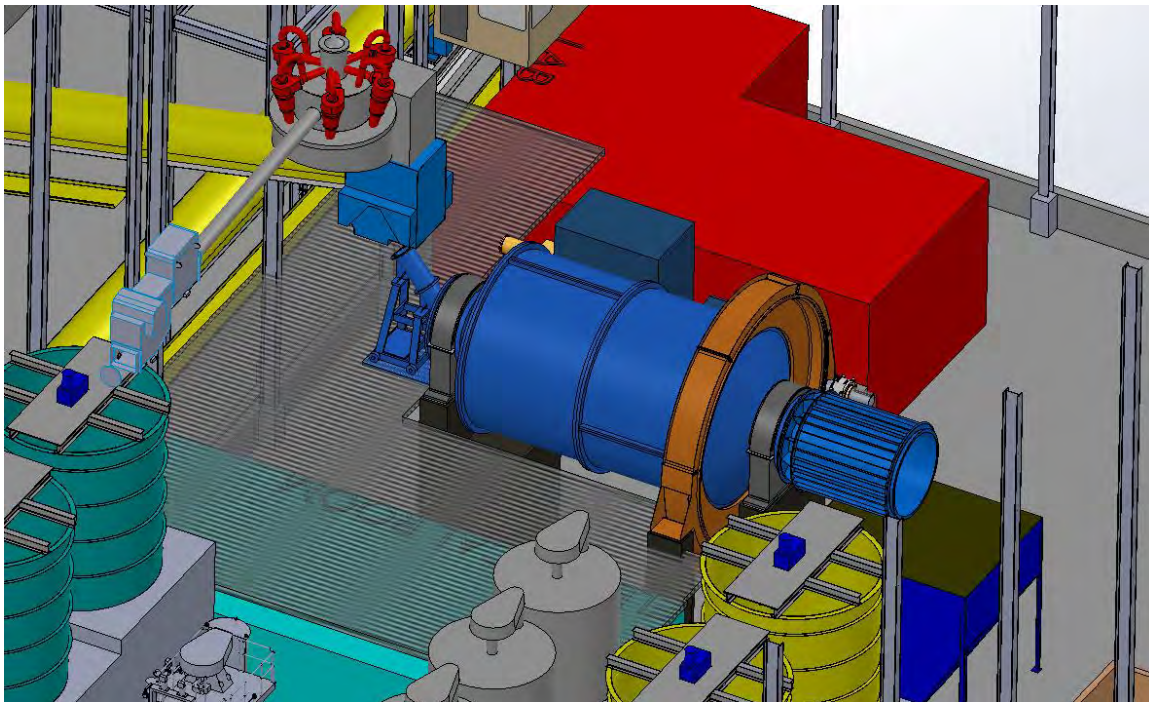


Figure 17-5: Ball mill and cyclone cluster arrangement

17.4. Flotation

The cyclone overflow slurry will feed the conditioning tanks ahead of the lead flotation circuit. The lead flotation circuit will consist of one row of rougher cells and two rows of cleaner cells. The rougher stage will consist of four 100 m³ tank-type cells. Lead rougher concentrate will be pumped to the first cleaner stage consisting of four 20 m³ tank-type cells. While the lead first cleaner concentrate is pumped into the second cleaner flotation cells, the lead first cleaner tailings are combined with lead rougher flotation tailings and transferred to zinc flotation. The lead second cleaner stage consists of two 20 m³ tank-type cells. The concentrate is pumped to the lead concentrate thickener (m diameter) and the tailings of the lead second cleaner stage are recycled to the lead first cleaner stage.

Lead flotation tailings are pumped to the zinc rougher conditioning tanks followed by one row of zinc rougher cells and two rows of zinc cleaner cells. The roughing stage will consist of three 100 m³ tank-type cells. Zinc rougher concentrate will be pumped to the first cleaning stage consisting of four 30 m³ tank-type cells. The concentrate is pumped into the second cleaning stage. The zinc first cleaner tailings are combined with zinc rougher flotation tailings and pumped to the tailings thickener. The zinc second cleaner stage consists of five 20 m³ tank-type cells. The concentrate will be pumped to the zinc concentrate thickener (9 m diameter) and the tailings of the zinc second cleaner row are recycled to the zinc first cleaner row. Figure 17-6 illustrates the flotation circuit arrangement from the 3D model.

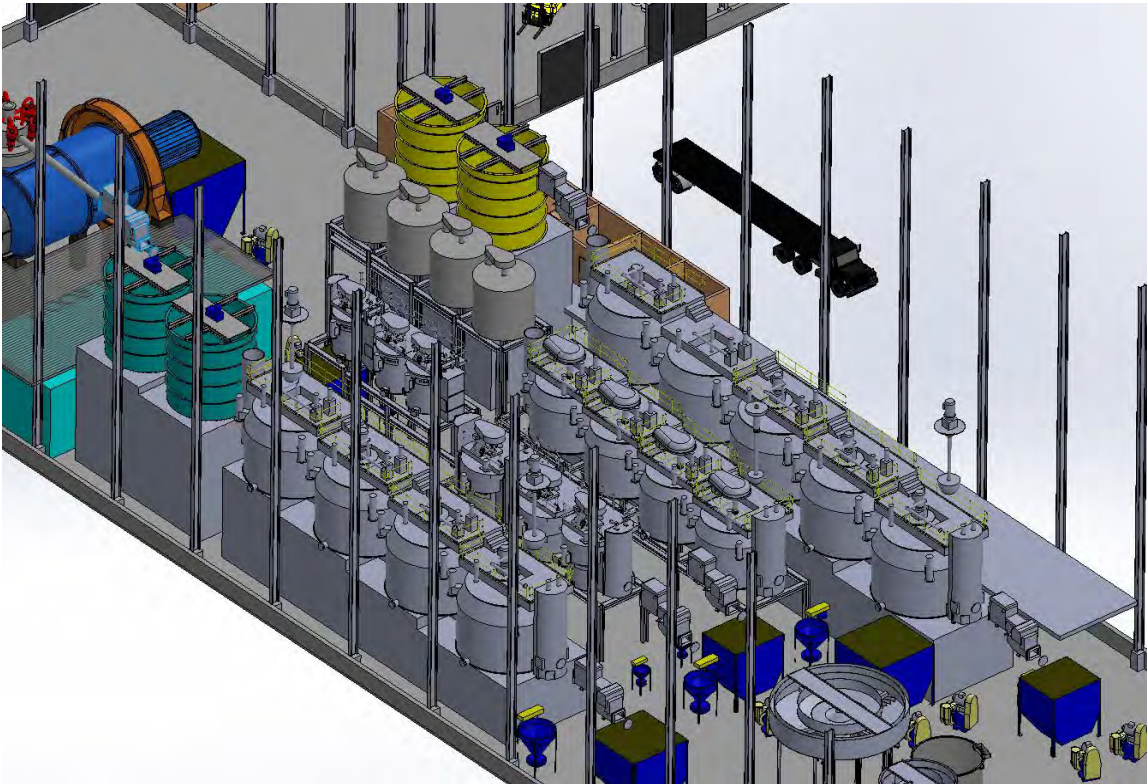


Figure 17-6: Flotation circuit arrangement

The critical streams needed to provide continuous feedback to the operators in terms of grade and recovery will be fitted with samplers. An X-ray diffraction-based on-stream analyzer (“OSA”) will provide assay data for each stream collected through sample pumps.

17.5. Concentrate Dewatering

Both lead and zinc concentrates will be pumped to their dedicated dewatering systems. Concentrate thickeners, followed by filter feed tanks, will provide enough retention time for the operation of a dedicated vertical filter press for each concentrate. The lead concentrate filter is designed to filter 76 m³/h and the zinc concentrate filter is designed to filter 156 m³/h of 62% solids thickened slurry. For operational flexibility, a third filter press will be installed as a stand-by for both lead and zinc concentrate filtration processes. The filtered concentrate cakes hold about 8% moisture and will be transferred by trucks. Figure 17-7 illustrates the concentrate dewatering arrangement from the 3D model.

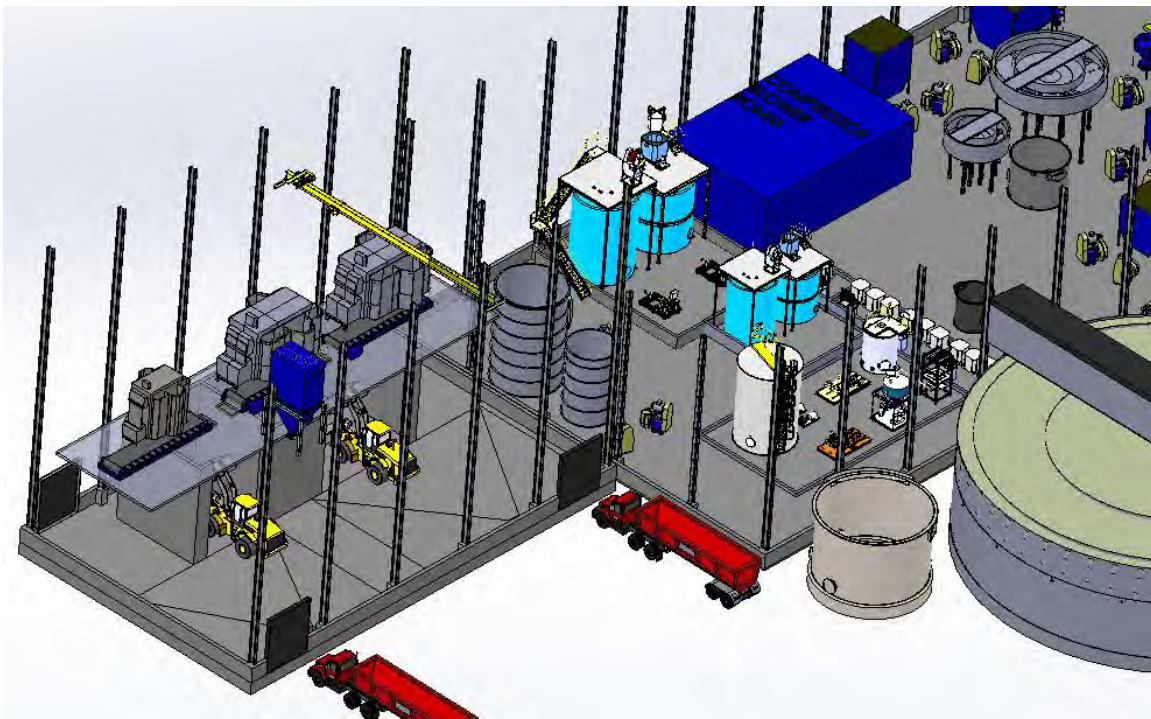


Figure 17-7: Concentrate dewatering arrangement

17.6. Tailings Dewatering and Disposal

Flotation tailings will be pumped to a 30 m diameter conventional high-rate thickener. Tailings will be dewatered to 60% of solids in the underflow for disposal. Thickened tailings will be transferred to an existing or new mined out open-pit. Thickener overflow will be returned to the process plant as process water. The tailings dewatering arrangement is illustrated in Figure 17-8.

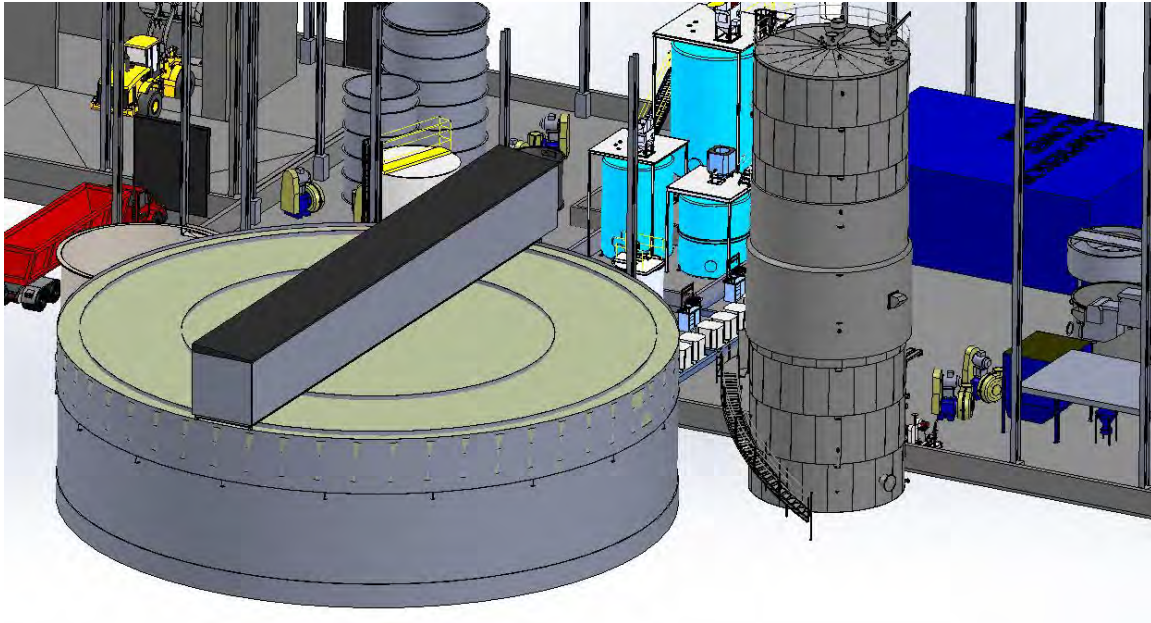


Figure 17-8: Tailings dewatering arrangement

17.7. Reagent Systems

The liquid reagents that do not require dilution will be delivered in one-tonne totes and distributed via dosing pumps to their respective addition point.

For solid reagents, an agitated mixing tank is provided with batch controllers used to mix the required reagent concentration. The reagents will be distributed throughout the plant via metering pumps or, in case of lime pumps, feeding a pressurized distribution loop.

A summary of reagents with the expected form of supply and mixing requirements is presented in Table 17-2.

Table 17-2: Reagent systems

Reagent	Delivery	Preparation
Zinc sulphate	Bags - solid	Mixing tank, water addition
Aero3894	Totes - liquid	No preparation required
CMC	Bags - solid	Mixing tank, water addition
Copper sulphate	Bags - solid	Mixing tank, water addition
Aerofloat 3418A	Totes - liquid	No preparation required
MIBC (methyl isobutyl carbinol)	Totes - liquid	No preparation required
Quick lime ("CaO")	Trucks - solid	Mixing tank, water addition
Flocculant	Bags solid	Eductor, mixing tank, water addition to inline mixer

17.8. Energy, Water and Consumable Requirements

17.8.1. Energy Requirements

The electrical energy requirements for the process plant were derived from the equipment list in which expected motor sizes for all equipment and ancillaries have been provided. Each motorized item of equipment was assigned utilization, efficiency and load factors to derive the data presented in Table 17-3.

Table 17-3: Process plant power demand by area

Area	Connected load (kW)	Power demand (kW)	Yearly consumption (GWh)
Crushing	3,162	1,930	12.68
Mineral sorting	3,714	2,444	16.06
Grinding	3,218	2,785	22.44
Flotation and reagents	3,191	2,229	17.96
Concentrate dewatering	868	525	4.23
Tailings & services	961	700	5.64
Total	15,113	10,613	79.02

17.8.2. Water Requirements

The water requirements for the plant are divided into two main areas, freshwater and process water.

Freshwater is planned to be sourced from Great Slave Lake located north of Pine Point and is used for reagent preparation and gland seal water, and is also stored as fire water. The freshwater requirement for the process plant is estimated at 42 m³/h. The process water is used throughout the process plant and is collected from thickener overflows. The total amount of process water recirculating in the process plant is estimated to be approximately 1,000 m³/h.

17.8.3. Consumable Requirements

The main consumables for the process plant include the crusher liners, the grinding media and liners for the ball mill, filter cloths and the reagents.

The crusher and ball mill liner replacement schedules were based on vendor recommendations and BBA's database.

The grinding media for ball milling will be 2.5" forged steel balls. Annual consumption rate has been estimated at 1,126 tpy.

The reagent names, applications and estimated average annual consumptions are outlined in Table 17-4.

Table 17-4: Reagents – Application and consumption

Description	Application	Consumption (tpy)
Zinc sulphate	Zinc depressant	870
Copper sulphate	Lead activator	1,980
CMC (Carboxymethyl cellulose)	Slimes Depressant	1,050
Aerofloat 3418A	Zinc promoter	40
Aero3894	Lead promoter	70
MIBC (methyl isobutyl carbinol)	Frother	80
Quick lime (CaO)	pH modifier	2,900
Flocculant	Flocculation of solids in thickeners	175

18. PROJECT INFRASTRUCTURE

18.1 Overview

The Pine Point Project benefits from the presence of existing infrastructure from the Cominco era and the proximity of provincial highways (Nos. 5 and 6) and railways (Mackenzie Northern Railway). The Project proposed infrastructure is located at Pine Point Junction, 10 km south of Hay River, and at the Pine Point Process Plant Site, beginning 42 km east from Hay River and extending over about 60 km of strike length parallel to the highways, with its central area being located near the historical and deconstructed Pine Point town. The Project infrastructure at the Process Plant Site and Pine Point Junction is described in Sections 18.2 and 18.3, respectively.

The Project infrastructure includes the following:

Process Plant Site

- Waste rock ("WR") and overburden ("OB") mineral storage facilities including conventional piles, as well as historical and proposed mined-out pits for mine waste management;
- Mine water management infrastructure including collection ponds, ditches, rehabilitated ponds, mobile water treatment units, pumps and pipelines;
- Mine dewatering wells, and associated sumps and/or reinjection wells;
- Historical pits (eight) used as Tailings Disposal Areas ("TDA") and their associated pumps and pipelines;
- Final effluent water treatment plant;
- Gate house, parking, weigh scale, and on-site roads;
- Camp complex;
- Process plant;
- Power generation and distribution facilities;
- Fresh water pumping station;
- Truck shop and warehouse;
- Mobile equipment.

Pine Point Junction

- Railway spur lines;
- Concentrate transloading installation;
- Warehouse, offices, and assay laboratory.



18.1.1 Project Area

Figure 18-1 provides an overview of the Project area including the Process Plant Site and Pine Point Junction locations as well as regional infrastructure.

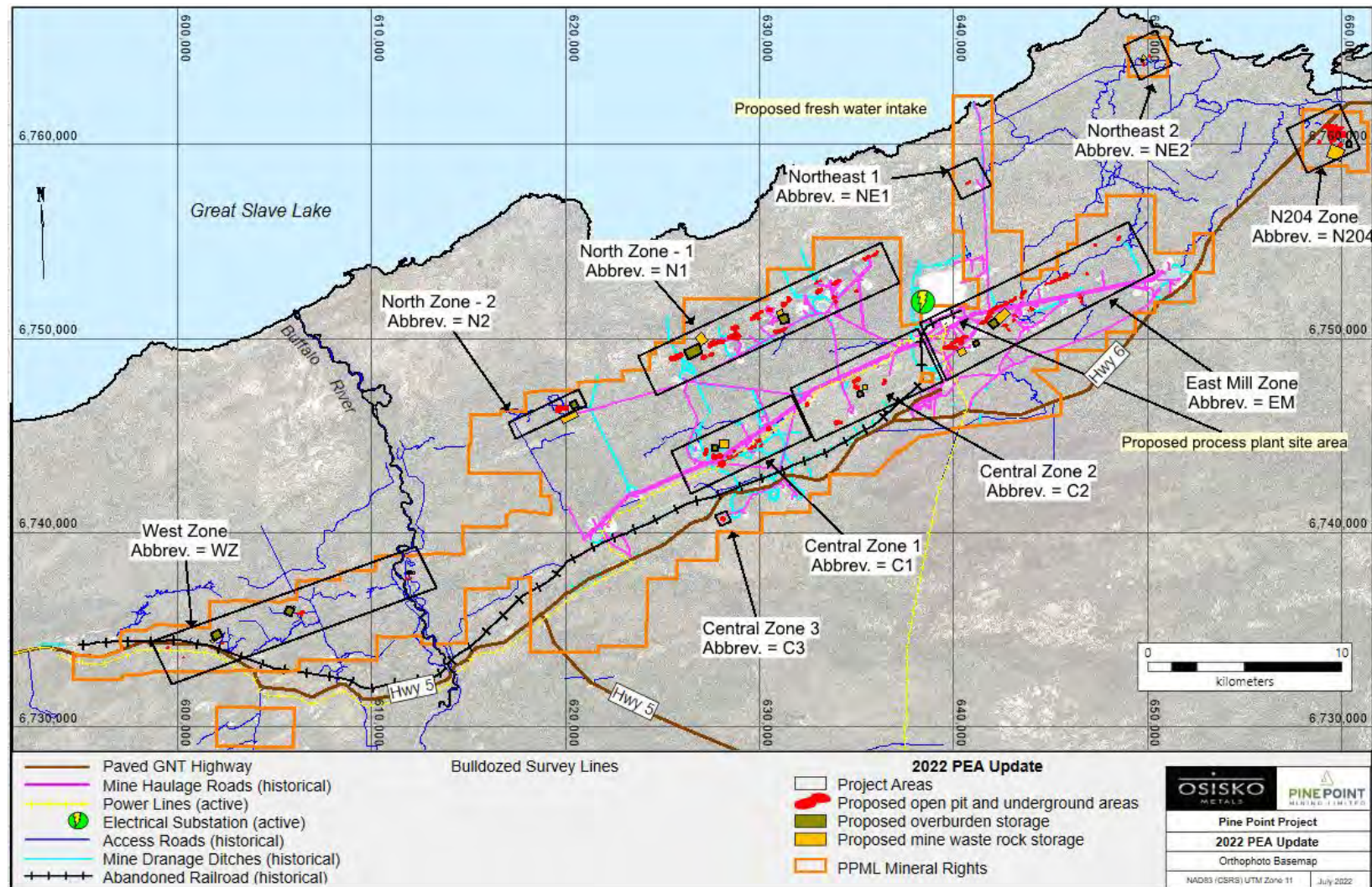


Figure 18-1: Pine Point Project area

18.2 Process Plant Site

18.2.1 Overview

The Process Plant Site will be located near the site of the historical Cominco mine complex, utilizing existing, supplemented by proposed infrastructure. Most of the site infrastructure is located east of the Buffalo River, including the process plant, the camp complex, all the open-pits as well as some underground workings. The main underground workings are located west of Buffalo River, where there is limited existing infrastructure as it was not part of the historical Cominco operation. Portable buildings are planned to be used while operating in this area, which will allow for re-location in other remote areas of the Process Plant Site and meet the varying needs of the Project.

The Cominco historical Plant Site was located approximately 80 km east of the junction of Highways 2 and 5, 30 km east of the Buffalo River. Existing infrastructure from the historical operations that are planned to be reused for the Project includes:

- Historical water ponds (two), for water management once rehabilitated;
- Historical building pads, for the process plant and the camp complex foundations;
- Numerous historical pits that have not been backfilled, either to be used for tailings, waste rock, mineral sorter rejects disposal or for water management;
- A network of paved roads and 100 km+ of all-season historical haul roads, for site access, transport and haulage.

18.2.2 Project Area

Figure 18-2 provides an overview of the Project's Process Plant Site area including existing and newly proposed infrastructure.

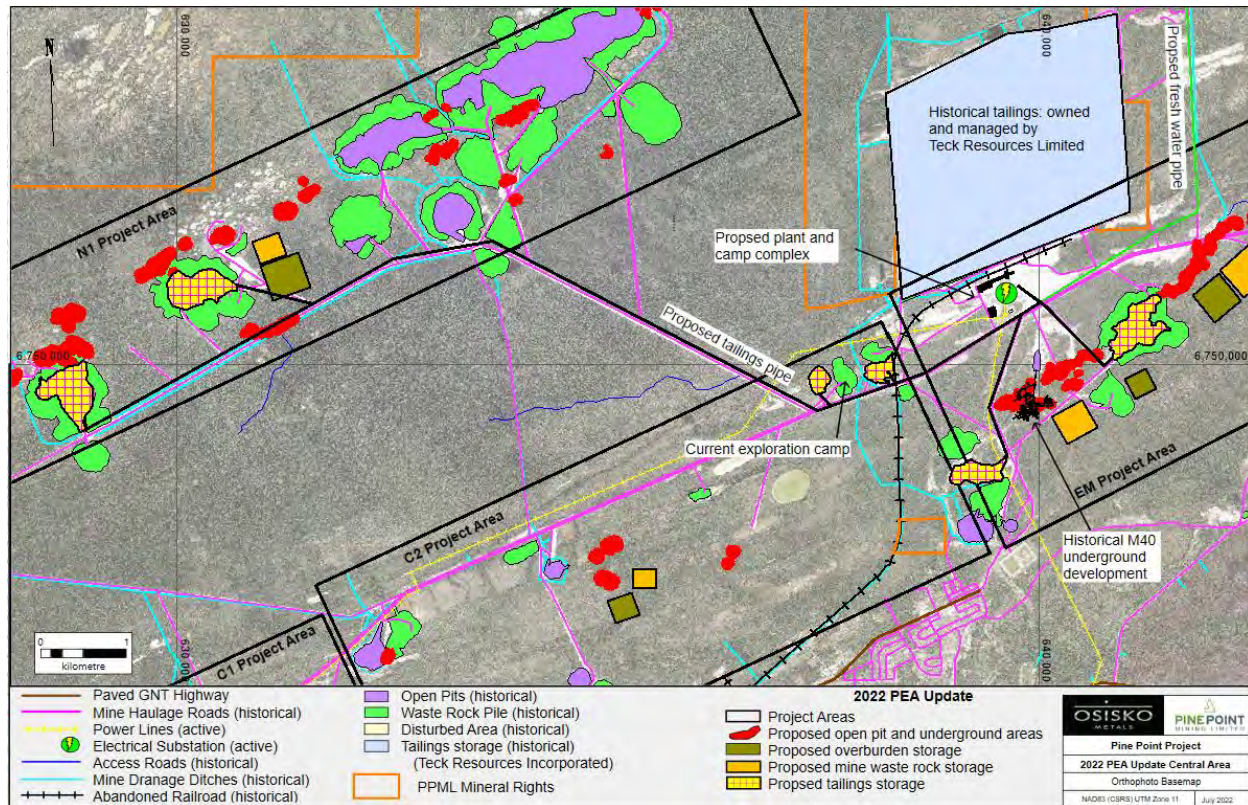


Figure 18-2: Process Plant Site infrastructure overview

18.2.3 Process Plant Site Facilities

Brief descriptions of the Process Plant Site infrastructure for the Pine Point Project are provided in the following section. Table 18-1 summarizes the design characteristics for each of the major Process Plant Site facilities.

Site Access (Primary Roads)

The Pine Point Process Plant Site is easily accessible via Highway 6, a paved road that is maintained 365 days a year. The use of existing historical haul roads and exploration cut-lines will be used wherever possible to minimize the need for additional ground disturbance. WR from historical and proposed mining activities may be used for road construction if geochemical testing demonstrates that it can be used as construction material.

Gate House, Parking, and Weigh Scale

The gate house modular building and parking lot will be installed on the existing access road between Highway 6 and the process plant area. This area will have a gravel surface surrounded by a perimeter fence to restrict and control the Process Plant Site access.

All site security cameras and alarms (fire, safety shower, etc.) will be available to the gatekeeper.

A 100-ton truck scale with a remote monitoring system will be used to gather information on incoming delivery trucks and outgoing concentrate transportation trucks.

On-site Roads

Some of the existing on-site roads will require minor repairs prior to mine operations but overall are considered to be in good to excellent condition. To access the new open-pits and underground mines across the property, new haulage roads will be constructed. In most cases, this will require a short extension of existing haulage roads.

To proceed to borehole drilling for dewatering and pumping system installation, maintenance, and monitoring access, roads will be constructed around the perimeter of the proposed open-pits and underground mines. The road to Great Slave Lake Fresh Water pumping station is existing and is generally in good condition. Trimming and removal of vegetation may be required in localized areas.

On-site road construction will be spread over the life of mine ("LOM").

Truck Shop and Warehouse

The maintenance truck shop will be a conventional construction with four maintenance bays and one wash bay. A central lubrication and fluid distribution and an oil recuperation system will be installed.

The warehouse will be an insulated steel frame, fabric-covered building located near the truck shop (24 m x 40 m).

Another steel frame, fabric-covered building will be dedicated for emergency vehicles.

Camp, Kitchen and Recreational Facilities

The camp complex will be installed near the process plant (described in Chapter 17) and will be connected by a corridor. The camp complex comprises rooms, administrative offices, infirmary,

mine dry, kitchen, lunchroom, entertainment area, potable water treatment, fire water distribution, sewage, and communications. A gymnasium will also be constructed at Year 1.

Figure 18-3 and Figure 18-4 show the general arrangement proposed for a typical dormitory module for the administrative / dry building, and the kitchen.



Figure 18-3: Camp complex - Typical room aisle

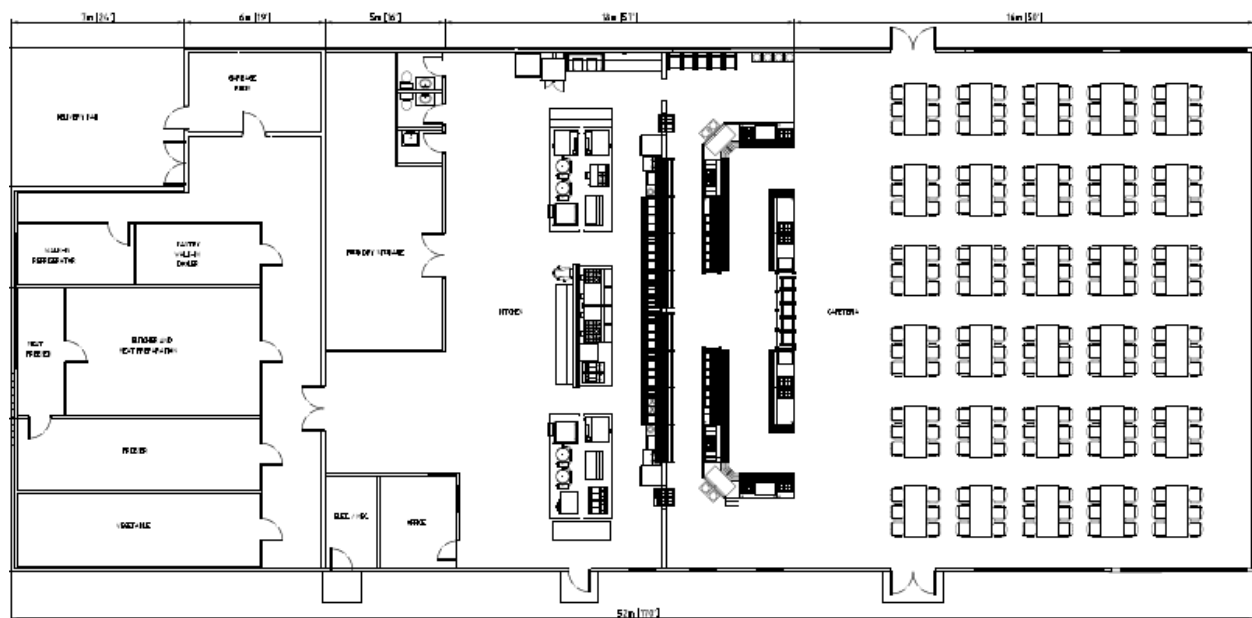


Figure 18-4: Camp complex - Kitchen

Administrative and Dry Building

Part of the camp complex, the Administrative and Dry Building shown in Figure 18-5, will include W/C facilities, work clothes storage, and separated men and women changing rooms sections:

- 160 lockers, 160 baskets and 30 single showers

A dedicated ventilation and dehumidifier system will be installed to allow proper drying of the miner's clothing.

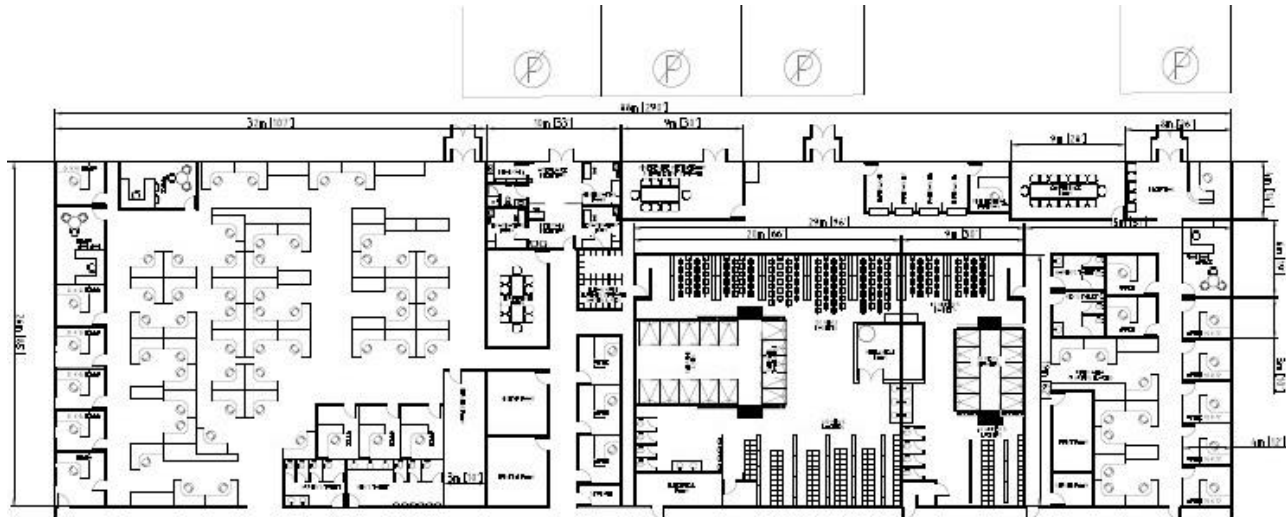


Figure 18-5: Camp complex - Administrative and dry facility building

Fuel Storage and Distribution

For mining operations, a fuel storage area with three double-walled distribution tanks of 50,000 L each will be installed. High flow and low flow dispensers will be installed. Also, a gasoline storage and distribution system of 10,000 L will be installed.

The distribution area will have secondary containments with a concrete slab to ease spill collection. Bollards will be installed to protect equipment and tanks.

Explosive Storage

Explosives will be provided by an external specialized and licenced contractor. In the first year, it is expected that the emulsion will come from an external facility and that the contractor will have their own explosive plant close to the site for the remaining years. The location of the explosive facility has not been selected yet, however, it will comply with the Explosive Act and Explosives Regulation (2013) as regulated by Natural Resources Canada ("NRCan").

Small explosive magazines will be constructed underground for the underground operations.

Fire Protection

The fire protection system covers the camp complex, truck shop, water treatment plant, and process plant. Fire water reserve will be from a dedicated portion of the fresh water tank at the process plant.

- A pumping station capacity of 2,500 USGPM;
- A 300 hp electrical pump with Diesel pump backup;
- A 4x4 fire truck, with a mini pump and foam system;
- Fire extinguishers;
- Fire protection hydrants.

Mobile Equipment

Auxiliary equipment covered in the mining section (see Section 16.4.7 for details) will be used for the whole site. This equipment includes:

- Two graders;
- Small excavators (two): for maintenance of ditches on-site, management of pumps, loading of Mineral Sorter rejects when the big front-end loader is not available;
- Small loaders (two): for management of pumps and pipes everywhere on the Process Plant Site, to transport replacement parts of mining equipment at the warehouse;
- Two water trucks;
- One compactor.

In addition to these shared equipment units, there will be:

- A firetruck;
- An ambulance;
- A forklift;
- A small loader (Cat950 size);
- Four pick-up trucks;
- One Genie boom;
- One 100 kW Genset with trailer.

General Utilities

The sewage system consists of a septic tank of 180,000 L and a pumping station with a drainage field piping of 3,500 m² area. A grease trap of 31 m³ has been added at the kitchen. A drainage field location should be confirmed with soil characterization studies.

Potable water will come from the Great Slave Lake pumping station. The system is sized for a nominal demand of 200 L per person per day. The system will include a 40,000 L reservoir,

distribution pumps and treatment modules (anion exchanger, activated charcoal filter, catalytic filter, and UV disinfection).

Each building will have its own electrical distribution. Fire alarm systems will be installed where fire protection is installed and in buildings where employees are working. These systems will have local annunciator panels, linked to a main panel inside the gate house.

Safety showers and eye wash stations will be installed in the garage, assay lab, wet lab, mechanical shop, and the process plant.

West Zone Mobile Infrastructure

For underground mine crews, offices, a lunchroom and a mine dry module will be installed on surface, at a central location in the West Zone.

The following infrastructures are planned to be installed near the portal and then relocated to the next portal:

- Garage (fold-away type);
- Containers for storage;
- Mine ventilation and air heating system;
- Propane storage;
- Stench gas mine alert system;
- Air compressors;
- Fuel storage and distribution;
- Underground mine electrical distribution and communication system, including mine mobile substation.

Table 18-1: Mine infrastructure - Design criteria summary

Description	Design Criteria
Main access road	NWT Highway 6 paved highway
Gate house, parking, truck scale	■ Modular gate house ■ Parking lot ■ 100t truck scale
Camp Complex Personnel capacity	■ 230 persons
Camp	■ Modular, hotel quality, installed on wood cribs ■ Single rooms with individual W/C & shower ■ 4 aisles with 2 storeys, 25 rooms on each level ■ 1 aisle reserved for couples (1 storey, 16 rooms) ■ 1 laundry room and 1 storage room per level
Administrative building	■ 24 closed office and 56 open space desks ■ 18 and 12-person conference rooms ■ Mine rescue team meeting and storage room ■ Infirmary with examination room ■ 4 Mine supervisor units ■ Printing room ■ W/C facilities ■ Server room for telecom hardware cabinets, complete with dedicated HVAC
Mine dry facility	■ Included in Administrative building ■ W/C facilities ■ Dedicated heating and dehumidifier system ■ Work clothes storage ■ For men: - 120 lockers - 120 baskets - 20 single showers ■ For women: - 40 lockers - 40 baskets - 10 single showers
Kitchen facility	■ 180 persons capacity lunchroom ■ Dedicated chambers for food delivery, walk-in refrigerator, pastry walk-in cooler, meat preparation, walk-in freezer, dry food / vegetable storage, office, and garbage
Entertainment area	■ Fitness room with weights, treadmills and stationary bikes ■ Common area lounge with couches and tables ■ Foosball and ping-pong tables ■ W/C facilities

Description	Design Criteria
Gymnasium	■ Air-supported dome structure ■ Multi-sport court (basketball size) ■ Two running / walking lanes ■ Planned in sustaining capital
Truck shop	■ 4 maintenance bays, 1 wash bay ■ Centralized lubrication and oil recuperation system
Warehouse	■ Insulated steel frame, fabric-covered building (24 m x 40 m)
Fuel Storage	■ Fuel storage and distribution 3 x 50,000 L ■ Gasoline storage and distribution 10,000 L
Telecommunication	■ Microwave communication tower for site Internet link ■ Wi-Fi access point throughout the buildings ■ Television in each room and common area lounge
Potable Water	■ Nominal demand of 200 L / person / day ■ Peak demand: 96 L / minute ■ Potable water reservoir 40,000 L ■ Three distribution pumps ■ Treatment system: anion exchanger, activated charcoal filter, catalytic filter, UV disinfection
Fire Protection	■ System capacity 2,500 usgpm ■ 300 hp electrical pump with diesel pump backup ■ Fire extinguishers ■ Fire protection hydrants
Sewage Treatment	■ Septic tank, capacity 180,000 L ■ 31 m³ kitchen grease trap ■ Pumping station, drainage field piping (3,500 m²)

Process Plant

The process plant will consist of four main areas:

- Primary crusher dump building: 8.2 m wide by 18.2 m long with a height of 12 m;
- Crusher building: 35.4 m wide by 45 m long with a height of 41 m;
- Mineral sorting and screening building: 36 m wide by 36 m long with a height of 41 m;
- Process building: Total of 176 m long with heights adapted for three sections (grinding 28 m, tailings area 19 m, concentrate filtering and load-out area 22 m). The main building will have a width of 36 m; except for the concentrate filtering and load-out area, the width will be reduced to 22 m. It will also include allocated space for the process services such as, electrical room (7 m wide by 16 m long with a height of 5 m), lab (122 m² surface area with a height of 5 m), mechanical shop and HVAC;

A 3D view of the primary crusher, material sorting plant and process plant is presented in Figure 18-6.

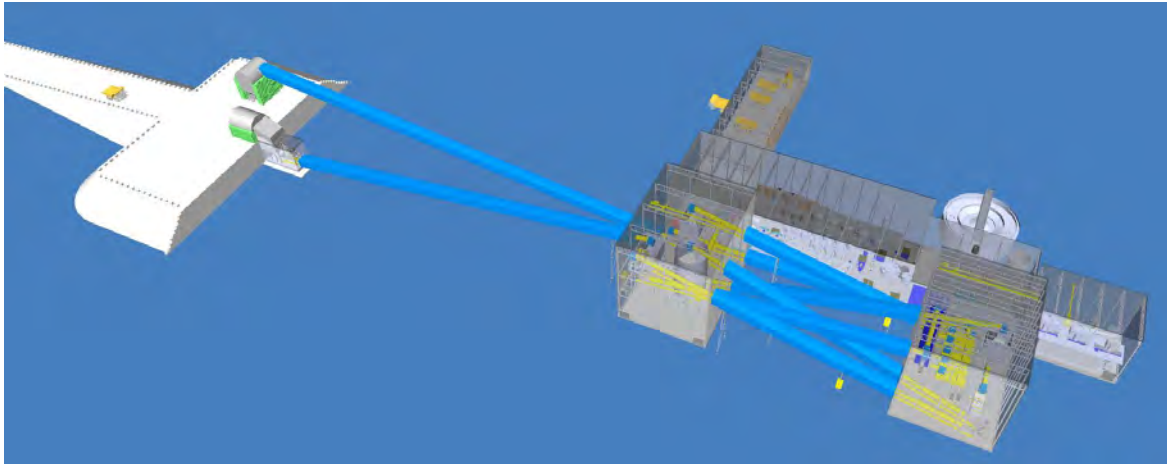


Figure 18-6: 3D view of the process plant

The primary crusher building will contain the vibrating grizzly feeder with the jaw crusher. Added to this building, there will be a light shelter for truck discharge. This area will contain an overhead crane with enough capacity to lift the heaviest crusher parts.

The secondary and tertiary crusher will be housed in the crusher building as well as the fine mineralized material bin. Mineral sorting and screening building will contain the mineral sorters and the vibrating screens of the crushing circuit. Both buildings will contain an overhead crane with enough capacity to lift the heaviest equipment parts.

The process building will consist of three main sections: grinding, flotation, and concentrate dewatering and load-out. Each section will have a dedicated overhead crane. The reagent preparation area and concentrate thickeners will be in the process building. The tailings thickener will be outside of the process building.

18.2.4 Power

The total power demand for the site varies over the years of operation mainly due to the dewatering of the various mines. Much of the power required for operations could come from the local public utility, Northwest Territories Power Corporation ("NTPC"). Between October and March (winter period), approximately 7.27 MW is expected to be available for use by Pine Point Mining Limited ("PPML"); outside of these months it is anticipated that 10 MW will be available. NTPC does not have the production and transmission capacity to supply all the Project's energy needs. A



local power generation strategy has been developed to maintain mining operations throughout the year. The power required for the site could, therefore, be a combination of NTPC-supplied power at an annual average of 8.4 MW, a 2.6 MW Photovoltaic solar farm and a local Liquid Natural Gas ("LNG") power plant.

Project Power Demand

The total power demand varies over the years of operation. See Table 18-2 below for the average total power demand.



Table 18-2: Site power demand

Year		Y1	Y2	Y3	Y4	Y5	Y6	Y7	Y8	Y9	Y10	Y11	Y12
Area	Description	Average power demand (MW)											
200	Underground Mine	-	0.06	0.53	1.01	1.35	0.83	0.88	0.08	-	-	-	-
300	Open-pit Mine	0.00	2.19	1.64	1.93	2.39	1.60	1.75	0.71	2.05	1.62	1.44	1.87
500	Site Infrastructure	0.85	0.85	0.85	0.85	0.85	0.85	0.85	0.85	0.85	0.85	0.85	0.85
600	Process plant	10.95	10.95	10.95	10.95	10.95	10.95	10.95	10.95	10.95	10.95	10.95	10.95
Total power demand (MW)		11.80	14.04	13.96	14.74	15.54	14.23	14.43	12.59	13.85	13.42	13.24	13.67



NTPC Network

Northwest Territories Power Corporation supplies power to South Slave region of the Northwest Territories ("NT") via the NTPC Taltson Hydro Facility. The Taltson facility was brought online in 1965 to supply power to the original Pine Point Mine (Cominco) facilities at Pine Point, which closed in 1988, and to the town of Fort Smith, NT. The facility now supplies power to the towns of Hay River, Fort Smith, Fort Resolution and Enterprise. The dam complex currently has an 18.5 MW capacity with a forecasted capacity increase of 4 MW (power upgrade scheduled to be completed in 2022 at the Taltson Generation Station). Of that, 7.27 MW would be available for PPML during the winter months (October to March) and 10.09 MW unused and available during the summer months (April to September).

Energy cost provided by NTPC is 0.145\$/kwh and, due to the relatively low-cost energy, power supply from NTPC would be favoured over on-site power generation.

Solar PV farm

An extra 2,600 kW would be provided by a solar PV farm built on-site. Having the solar station connected to the local electrical distribution will add more capacity and help increase reliability of the region's power supply. Over the LOM, coupled with power from NTPC, these two energy generation systems will cut NG consumption by 79% compared to an all fossil-fuel based solution.

On-site Power Generation

The Project will seek to benefit from an existing LNG delivery network (virtual pipeline), for industrial clients, sourced out of Fort Nelson BC. This network can supply the southern area of the Northwest Territories through an existing public road network. A supply contract would be established with a local LNG provider to source, transport and transform the natural gas on-site for local generator use.

On-site power generation will consist of two types of setups: a stationary power plant and a fleet of mobile gensets.

Stationary Power Plant

There will be one stationary power plant setup for the centralized process plant and the dewatering loads.

The main purpose of the stationary power plant is to supply supplemental power that is unavailable from NTPC and the Solar PV plant. The power plant will also provide redundancy for the network. The current peak load demand would require an installed power plant of 16 MW

capacity. This power plant will be a combination of procured gensets and rented units that will vary in numbers as dewatering loads vary over the years. As a result, the stationary power plant capacity will be scaled on an annual basis according to the power needs aligned with the forecasted loads shown in Table 18-2.

Mobile Genset Fleet

A fleet of mobile gensets will be rented from a local supplier and will be used to service various open-pit and underground mines throughout the site. As open-pit mines will constantly be opened and closed throughout the mine life, a mobile genset fleet concept will give PPML the flexibility to relocate gensets as needed.

Power Transmission – 12.5KV

As the NTPC currently owns and operates an electrical substation (138 - 12.5 kV) located within the historical Pine Point infrastructure, it was agreed with NTPC that the electricity required for the site could be available at 12.5 kV from two existing main breakers. From the NTPC substation, two full capacity interconnections are required between the existing substation and the process plant facilities; this is to ensure reliability and flexibility for maintenance.

Near the process plant, a main electrical room including main 12.5 kV switchgears, 600 V transformers and motor control centres ("MCCs") has been planned. The main switchgear will distribute power throughout the concentrator building. An allocation for site electrical distribution to the camp, water treatment plant and other service buildings has also been included.

Local distribution at the underground mines and open-pits will be built at 12.5 kV.

Power Distribution – 600 V

Since the pumps operate at a lower voltage (600 V) than the local distribution (12.5 kV), a strategy has been developed to optimize the cost of the step-down transformers and distribution cables required to connect the pumps to the 12.5 kV grid.

The strategy consists of having the 12.5 kV grid reach as close as possible to all major pumping clusters. A set quantity of cables and transformers would then be moved around between pumps according to the dewatering schedule. The same cables would then be displaced and reused as the dewatering loads shift in place and in time.

Emergency Power

Critical loads, such as the concentrator, pumping stations, water treatment plant and camp, will include local emergency diesel power generators in the event that the main power from NTPC is offline. The emergency system will be optimized to ensure that the generation capacity is adequate.

18.2.5 IT and Communications

Telecommunication services will be provided to the Process Plant Site via a microwave communication tower to allow Internet connection. IT and networking equipment required for the Project will be deployed to service the different areas via Wi-Fi, or fibre optic where appropriate, for the administrative, industrial and fire alarm networks. There will be Wi-Fi access points available inside each building.

A fibre optic network will interconnect the main areas of the Project, including the following:

- West Zone;
- Camp complex accommodations and offices;
- Main entrance gate;
- Main electrical substation;
- Concentrator;
- Water treatment plant.

18.2.6 Waste Management

Reclamation Material

Reclamation material stockpiles will be located close to WR and/or OB piles and open-pits to provide storage for topsoil and organics. No design effort was carried out for the current study since the volume of reclamation material is expected to be relatively low. The reclamation material stockpiles will include mainly peat and small trees and will be used for reclamation purposes.

Waste Rock (WR) and Overburden (OB) Material

The WR and OB volumes have been calculated based on the Mine Plan, as described in Chapter 16, where the estimated tonnage of WR and OB material is reported per proposed

open-pit. The WR tonnage was converted to volume using a dry density of 2.71 t/m^3 as specified in the MRE, and a swell factor of 1.3. The OB was converted to volume based on the assumption that it will mainly be composed of till, with a dry density of 1.7 t/m^3 (Rice et al., 2013).

Where possible, the WR will be backfilled in nearby historical pits or in available proposed mined-out pits. The OB and any remaining WR will be stored in separate stockpiles for use during reclamation. The major part of the overburden will be used for reclamation purposes. Considering the number of pits, the design and location of the WR storage facilities ("WRSF") was developed using a maximum hauling distance of 2 km.

The preliminary design of the WRSF is based on typical criteria and considers the Design for Closure concept, such that reshaping and reworking of the WR material during the active reclamation phase is minimized. The WRSF have been designed using the following geometry:

- Maximum height: 30 m;
- General Slope: 2.5 H:1 V;
- Lift thickness: 10 m;
- Berms: 4 m wide berms at each lift;
- Pile width and length: dependent of the site restrictions.

General information on the geotechnical conditions was found in historical documents and will be confirmed with field investigations and stability studies. Observations made of existing WR facilities do not suggest notable issues with stability.

The preliminary WR storage locations have been established with the objective of minimizing the disturbed footprint whilst limiting haulage distances. Site restrictions, such as historical pits and piles as well as transport infrastructure, have also been considered. In total, 14 WRSF and 14 OB stockpiles have been designed to store 52.6 Mm^3 of waste rock and 36.4 Mm^3 of overburden. The total volume of WR produced over the LOM is 48.8 Mm^3 ; the total OB volume is 44.2 Mm^3 . The difference will be stored in proposed or historical mined-out pits. Table 18-3 lists the WRSF capacities with respect to each area, as well as the list of historical and new open-pits proposed to be used for backfill.

Table 18-3: Volumes of waste rock or overburden capacities in each pile or pit group

WR or OB Storage ID	Area	Pit Receiving Waste Rock		WRSF capacity (m³)	OB stockpile capacity (m³)
		Historic Pit	Proposed Mined-Out New Pit		
1	NE2	N/A – External only		914,000	72,000
2	N204			15,382,000	1,332,000
3	N2			5,241,000	1,828,000
4	W1			252,000	13,000
5	W1			306,000	6,000
6	W1			152,000	300
7	C1	J69, M64	J68	5,848,000	3,802,000
8	C2	N/A – External only		263,000	2,560,000
9	EM	O42, P41	-	3,512,000	2,228,000
10	EM	-	L35	7,297,000	3,658,000
11	N1	-	Z60	5,848,000	6,088,000
12	N1	-	Y65	263,000	12,994,000
13	EM	N/A – External only		114,000	337,000
14	EM			2,886,000	1,527,000
Total				52,647,000	36,445,300

Mineral Sorter Rejects

A mineral sorter rejects pad, with a capacity of 5 Mt (approximately 2.4 Mm³), will be located east of the process plant, as shown in Figure 18-2. The pad is designed to have a height of 20 m and will be used to temporarily store mineral sorter rejects before they are transferred to the TDA for levelling or available mined-out pits or WRSF. The pad location will be refined as the process plant design advances.

18.2.7 Water Management

Site water management at the Pine Point Mine has historically been an important aspect of mine operations, which will also be the case for the proposed mine development plan. The substantial historical site-specific dataset related to pit dewatering and pumping tests provided a unique opportunity to calibrate appropriate underground and open-pit dewatering and management systems for the Project. Additional studies are also being conducted under the Confirmation and Exploration Program.

Multiple field tests were carried out in over 26 boreholes at the Pine Point site. Understanding of the Hydrogeology has advanced since the K values and flow are mostly understood. Inflow rates were estimated using groundwater flow simulations and are described in Chapter 16.

Surface Water

Surface water management infrastructure at the Pine Point site includes the following:

- Existing open-pits, in-pit sumps and pumping equipment for runoff water management;
- Ditches and collection ponds for WRSF and OB stockpiles;
- Two reconstructed water collection ponds for storing reclaim and process water surplus;
- A water treatment plant ("WTP") for treating excess contact waters (not reclaimed at the process plant).

Two historic ponds are located 3 km south of the former operation site. These ponds will be reconstructed and used to store reclaim water from the TDA's and process plant site runoff water and used for the process plant needs.

Excess contact water from these reconstructed ponds will be transferred to the WTP and treated to meet applicable requirements before being discharged into the environment, as described in the following sub-section.

Waste Rock Storage Facilities and Overburden Piles

WR and OB disposal strategy consists of building a total of 28 (14 + 14) WRSF and OB stockpiles. Contact water from the WRSF and OB stockpiles will be collected and directed to collection ponds via a network of perimeter collection ditches. These ponds have been sized to withhold a 6-hour 100-year rainfall event while considering a complete discharge of the pond in three days. Climate change affecting rainfall events and snowmelt has also been considered.

Submersible pumps installed on floaters will be used to pump out the contents of the collection ponds. As necessary, excess water in these collection ponds will be transferred to an available open-pit in an adjacent area. Pit runoff and contact water from neighbouring WRSF or OB stockpiles that collect in other nearby collection ponds may be combined, and also sent to the adjacent receiving pit. In some areas, no piles are planned, and runoff collected in the developed open-pits will be the only contact water to be transferred to the receiving open-pit. When required, a booster pumping station will be added for water transfer to the mill area.

If required, any discharge of contact water to the environment will meet applicable discharge requirements. If required, a mobile water treatment unit may be added to ensure water quality meets environmental regulations.

Whenever possible, collection pond pumps, booster pumping stations, and piping are assumed to service the next developed area once a pond is rehabilitated.

Open-pit Dewatering

Prior to mining each pit, ground water will be lowered to a depth of 10.0 m below the current excavation level by pumping from wells into surface channels discharging into historical mined-out pits or surface settling ponds, and discharged once discharge effluent quality criteria are met. This is to be confirmed in more detail in subsequent stages of engineering. The dewatering requirements for each pit, in terms of numbers of boreholes and pumped flows, have been estimated based on numerical modelling by Hydro-Ressources Inc. A range of sizes of submersible, vertical turbine, borehole pumps was selected based on pumping at 2,000 USgpm (454.2 m³/h) at each well. Each pump will discharge through a 200 mm diameter bore-line pipe, with flow regulated by means of a choke valve, at the ground surface, as necessary. Pumps, along with their bore-line pipes, valves and control panels, will be used as they become available from the 73 pumps, of various sizes shown in Table 18-4.

Table 18-4: Summary of pumps required for open-pit dewatering

Pump Size (horsepower)	Number
75	5
150	68
Total	73

This plan, to tailor the sizes of dewatering pumps to suit pit depths, differs from the arrangement presented in the 2020 PEA, which was based on single size pumps for all open-pits. Thus, economies are achieved by providing smaller pumps, where possible, that are designed to operate closer to optimal efficiencies.

In accordance with the mine production schedule the submersible pumps, are planned to be relocated to another pit once an open-pit is no longer in operation.

Rainwater accumulation at the bottom of the open-pits will be pumped out using centrifugal diesel pump mobile systems. A 2-year return period, 6-h rain event has been used as the design basis. The pumping rate is calculated based on an emptying time of 24-h to 48-h. The pumping pressure head is calculated based on pit elevation data and approximate location of the booster pumps.

Rainwater collected from adjacent operating pits is pumped to a common booster pump located on surface that transfers this water to a nearby receiving pond, either directly or combined with runoff from waste rock and overburden piles runoff.

Whenever possible, pit bottom pumps and piping (HDPE DR11) are assumed to be relocated to another pit, according to the mining schedule. For the 47 open-pits, 16 pumps are required. Overall, eight different pump models are being considered with motor powers ranging from 10 hp to 200 hp.

Underground Dewatering

Similar to open-pits, dewatering requirements for the five underground workings have been established by Hydro-Ressources Inc. The same pumping strategy with submersible pumps in wells, with ditches transferring the groundwater to previously exploited open-pits or ponds is being used, as well as the relocation of equipment to the next deposit when an underground mine is no longer in operation.

The underground workings located west of the Buffalo River will, however, require groundwater control by means of grout/till injection prior to dewatering, as described in Chapter 16. Some underground workings in the Central Area are, expected to be fully or partially dewatered as nearby open-pits are dewatered. The wells will be built in the same way as the open-pits but will use 65, 300-hp pumps and 19, 400-hp pumps. Table 18-5 defines shows the number of pumps required for underground dewatering.

Table 18-5: Summary of pumps required for underground dewatering

Pump Size (horsepower)	Number
300	65
400	19
Total	84

Water Treatment

A WTP will be required to treat the excess water from the reconstructed water collection ponds. To size the WTP, a simulation using the PCSWMM software was carried out to define volumes of runoff reporting to the mill complex and the rehabilitated ponds area. These inflows were compared to the pond capacities, estimated to be 40,000 m³ each, the volume of water required by the process plant (2,640 m³/day), and the volume of water inflows from the TDA (3,924 m³/day). The volume in excess of the capacity of the ponds was used to calculate the maximum monthly

pumping rates. The monthly peak pumping flow is estimated to be 135 m³/h and the WTP has to be designed accordingly. The average annual flow rate is 88 m³/h. The WTP will be located at the process plant site and will be connected to the reconstructed ponds by an appropriate network of water pipes and pumps.

For this study, it is assumed that the water will be treated mainly for suspended solids, zinc, and lead. Water quality assessments will be conducted in the next steps of the engineering studies to update the WTP design.

18.2.8 Tailings and Reclaim Water Management

Tailings from the processing plant are pumped into historical and mined-out proposed open-pits via HDPE pipelines. A total of 23.9 Mm³ of tailings will be produced during the LOM. Over the life of the Project, the pipeline network will be expanded to reach the eight pits that have been identified for tailings storage (Figure 18-7). To deposit tailings at the greatest distance from the processing plant, a booster pump station will be required.

Reclaim water is brought back to the plant in a HDPE pipeline installed next to the tailings line. Submersible pumps (200 hp) installed on floaters are used to pump back reclaim water from TDA until pits have been fully filled. The distance between the TDA and the process plant varies from 150 km to 14 km. No booster pumps are required for the reclaim water.

Along with the piping, pumping stations will be moved to the next TDA once operations are completed in one location. Three reclaim water pumping stations will be required for the extent of the Project.



18.2.9 Fresh Water

Water from the Great Slave Lake will be pumped over a 10-km length of buried pipeline from a permanent pumping station to the process plant to meet its fresh water requirements (132 m³/h). A portion of the pumped water will be treated to meet the potable water requirements for the camp.

18.3 Pine Point Junction Infrastructure

18.3.1 Overview

A 2017 Tetra Tech study (Tetra Tech, 2017) concluded that concentrate transloading to the railroad was the best transporting option, and a site located near the railway at the Pine Point Junction (junction of Highways 2 and 5) had been selected (Figure 18-8). The land is owned by CNR and is located where Fort Smith Highway intersects Hay River Highway, between the towns of Hay River and Enterprise.



Figure 18-8: Pine Point Junction transloading site location

18.3.2 Facilities

A 140 m x 25 m building containing conveyors, hoppers, chutes, silos, and dust collectors will be constructed to allow railroad transloading of concentrate to railcars. A ramp will be erected to allow trucks to dump directly into the hopper.

The railway spur lines, totaling 1.21 km of rail, will allow transfer and storage of the railcars. Handling will be executed by a Shuttlewagon SWX315 Railcar mover, while containers will be handled by a 45,000-ton mobile reach stacker. A Caterpillar 950 Loader will be used for material handling on-site.

Buildings on-site will be modular, and will include offices, lunchroom, washroom and an assay laboratory. A 42 m x 15 m fabric structure warehouse mounted on concrete blocks will serve for mobile equipment and material storage.

The site will be secured by a fence and an automatic access gate. Areas will be reserved for container handling and storage. A security camera system and adequate site lighting will allow remote surveillance of the site.

Electrical power will come from a new overhead line connected to a nearby grid (1.2 km) and a communication tower with microwave system that will relay the connection to the Internet.

Figure 18-9 presents the general arrangement for the concentrate transloading site.

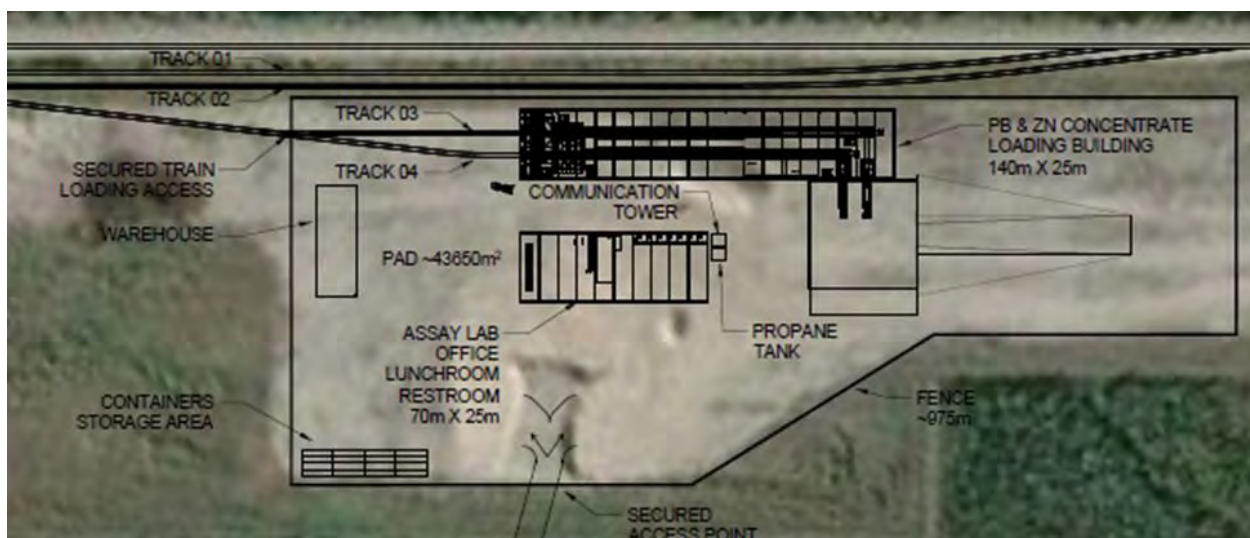


Figure 18-9: Pine Point Junction concentrate transloading general arrangement

19. MARKET STUDIES AND CONTRACTS

The information summarized in this chapter is derived from Stonehouse Consulting Inc., a third-party concentrate marketing specialist company, who was contracted by Osisko Metals to prepare a market study for both the Lead and Zinc concentrates to support the PEA (Vice, 2022).

19.1 Zinc and Lead Price Projections

Project economics were estimated based on a long-term CAD:USD exchange rate of 1.27 and metal prices (two-year trailing average) of US\$1.37/lb for zinc, and US\$0.96/lb for lead, which was established by Osisko Metals (Charles, 2022). Colin Hardie, QP notes that the proposed exchange rate, zinc and lead pricing, used in the cashflow analysis, is aligned with recent studies and BBA's survey of industry-consensus long-term prices.

19.2 Zinc Concentrate

The average annual production of zinc concentrate at Pine Point is projected to be approximately 250,000 tpa (Project LOM total: 2,987 Kt) at a grade ranging from 56 to 62% Zn. The life of mine ("LOM") zinc concentrate grade forecast has been assumed to be 60%.

The projected assays for the zinc concentrate shown in Table 19-1 were based on laboratory flotation testwork studies (New Comp-F5) described in Chapter 13.



Table 19-1: Indicated zinc concentrate assays (ALS)

Element	Symbol	Unit	Reported Concentration	Typical Smelter Penalty Threshold ⁽¹⁾
Arsenic	As	ppm	< 2	2,000
Antimony	Sb	ppm	< 0.5	1,000
Bismuth	Bi	ppm	< 0.1	1,000
Cadmium	Cd	ppm	864	4,000
Cobalt	Co	ppm	3	1,000
Copper + Lead	Cu+Pb	%	0.23	3
Fluorine	F	ppm	< 20	300
Iron	Fe	%	2.6	8.0-9.0
Magnesium	MgO	%	0.36	0.35
Manganese	Mn	ppm	100	12,500
Mercury	Hg	ppm	0.31	50
Silica	SiO ₂	%	< 0.021	4

1. Source: Wood Mackenzie

Analysis of the zinc concentrate indicates that it is relatively clean and free of impurities except for Magnesium Oxide (0.36% MgO), which is at the current smelter penalty threshold (typical: 0.35% MgO).

19.2.1 Zinc Market Overview

Global Supply and Demand

Global supply of refined zinc metal has dwindled, with inventories remaining below critical levels. Zinc mines continue to face significant shutdowns or lowered output due to COVID-19 and heightened geopolitical risk. In Europe, unplanned smelter shutdowns due to rapidly rising energy prices show no sign of abating. This has resulted in unprecedented high zinc prices over the last 12 months. There remains a distinct lack of fully permitted large-scale projects globally and with continued under-investment in the zinc development space by diversified miners compared to copper, chronic under-supply will be further exacerbated and underpin elevated prices over the coming years. Zinc supply fundamentals will continue to rapidly evolve.

Demand fundamentals are also evolving as the world tries to enter a post-pandemic phase. Zinc usage remains dominated by infrastructure and the auto industry. While consumer demand is expected to remain steady, governments have turned toward infrastructure projects to stimulate and prepare their economy for decarbonization. This includes mass transit solutions and it is expected to be highly beneficial for the level of zinc demand. With continued low global

inventories below critical levels (ten days of consumption in inventory) and struggling supply, zinc prices should be well supported into the future.

North American Supply and Demand

Similar to global supply outlooks, North American supply has suffered greatly. Over the last 4 years, two zinc mines were shut down; Teck's Pend'Oreille mine (Regneir, 2019) and Nyrstar's Langlois mine (Schmidt, 2019). Further shutdowns are expected over the coming years. Trevali recently reopened its Caribou mine from under care & maintenance but has struggled to meet previous throughput levels (Trevali Mining Corp., 2021). Glencore has also announced the shutdown of its Matagami mine in 2022 (Gagnon, 2022) and the Kidd-Creek mine may also face closure in 2023 (Grech, 2021). Similarly, Hudbay has moved ahead with plans to close the majority of its Manitoba zinc production in 2022 (Mining.com, 2022).

With no new production expected over this timeframe, North American zinc smelters will face unprecedented challenges in sourcing and securing concentrate supply. Major forecasted zinc projects in North America include South32's Taylor project in Arizona (Mining.com, 2022). However, the elevated manganese content severely restricts potential smelters that can process its zinc concentrate. Therefore, North American smelters will still need to find a clean, high-quality source of North American zinc concentrate to compensate for the forecasted mine closures.

North American demand demonstrates the same themes as global demand. It is expected that consumer demand will be tepid but local, state/provincial and federal governments are all expected to push large infrastructure stimulus legislation to stimulate their economies. This will provide strong underlying support for zinc prices.

19.2.2 Zinc Concentrate Marketability

The expected average grade of the zinc concentrate is estimated at 60% Zn based on the metallurgical testwork and proposed mine plan. The high-grade and clean nature of the Pine Point zinc concentrate is likely to make it attractive to those traders that blend concentrates. No penalties are expected to be imposed.

The preliminary economic analysis in Chapter 22 assumes that the smelter pays for 85% of the zinc content in the concentrate, subject to a minimum deduction of eight units.

The focus of sales of zinc concentrate will be to North American smelters at terms estimated to be at a discount to benchmark treatment charges due to the quality of the concentrate. The balance of the zinc concentrate will be sold at spot treatment charges to markets in Asia.



19.2.3 Zinc Concentrate Sales Terms

For the purposes of Project evaluation, the zinc concentrate sales terms presented in Table 19-2 were used in derivation of the net smelter return ("NSR").

Table 19-2: Zinc concentrate terms used in derivation of the NSR

Payable Metals	
Metal Recovery to Zinc Concentrate (LOM Avg)	87%
Concentrate Moisture	8%
Zinc	85% of the Zn content, subject to a minimum deduction of 8 units
Silver	No payable – below detection limit
Treatment Charge	US\$170.10/dmt of concentrate delivered
Price Participation	Nil
Penalties Allowance	Nil

19.2.4 Zinc Concentrate Transport

The zinc concentrate (8% moisture) will be trucked to the Pine Point Junction railway facility approximately 65 km west of the mine site, where it will be shipped to North American and Asian markets. Trucking and rail car loading costs have been estimated at \$30.56/wmt. The freight and logistics costs for transporting the zinc concentrate from Pine Point Junction to North American and Asian Smelters have been estimated at \$168.87/dmt, based on a preliminary marketing strategy. The transport losses have been estimated at 0.2%.

19.3 Lead Concentrate

The average annual production of lead concentrate at Pine Point is projected to be approximately 100,000 t/yr (Project LOM total: 1,177 Kt) at a grade ranging from 57 to 67% Pb. The LOM lead concentrate grade forecast has been assumed to be 65%.

The projected assays for the lead concentrate shown in Table 19-3 were based on laboratory flotation testwork studies (New Comp-F5), as described in Chapter 13.



Table 19-3: Indicated lead concentrate assays (SGS)

Element	Symbol	Unit	Reported Concentration	Typical Smelter Penalty Threshold ⁽¹⁾
Arsenic	As	%	< 0.001	0.2
Antimony	Sb	%	< 0.02	0.2
Bismuth	Bi	%	< 0.01	0.1
Fluorine	F	%	0.03	0.5
Mercury	Hg	%	< 0.001	0.005
Silica	SiO ₂	%	0.1	6

1. Source: Wood Mackenzie

Analysis of the lead concentrate indicates that it is free of impurities and any measurement elements are well below typical smelter penalty thresholds.

19.3.1 Lead Market Overview

Global Supply and Demand

Lead supply is nearly exclusively sourced from secondary production of zinc and silver. Therefore, lead supply is often more closely correlated to the zinc price or silver price. This can create a supply context divorced from lead prices; this situation is exacerbated by a large amount of lead recycling. Primary lead supply remains difficult to source and lead production has slowly dwindled during the COVID-19 pandemic. Furthermore, lead concentrate quality continues to be challenged and clean sources such as the lead concentrate produced from Pine Point will remain in good demand.

The demand side of lead remains dominated by the battery market. Even with a transition toward electric vehicles, lead demand for the automotive industry is expected to remain important. Electric vehicles still require an alternative power source during vehicle start-up to handle the high amperage from a vehicle's electronic package. As lithium-ion batteries remain expensive, lead is expected to continue to capture this market.

19.3.2 Lead Concentrate Marketability

The expected average grade of the lead concentrate is estimated at 65% Pb based on the metallurgical testwork performed to date and the proposed mine plan.

The preliminary economic analysis in Chapter 22 assumes that the smelter pays for 95% of the lead content in the concentrate, subject to a minimum deduction of three units.



No penalties are expected to be imposed. It is assumed that lead concentrate sales will be made to North American smelters at benchmark treatment charges, with the balance sold to Asian smelters at a mix of benchmark and spot terms. These terms and assumptions are in line with current market conditions.

19.3.3 Lead Concentrate Sales Terms

For the purposes of Project evaluation, the lead concentrate sales terms presented in Table 19-4 were used in derivation of the NSR.

Table 19-4: Lead concentrate terms used in derivation of the NSR

Payable Metals	
Metal Recovery to Lead Concentrate (LOM Avg)	93%
Concentrate Moisture	8%
Lead	95% of lead content in the concentrate, subject to a minimum deduction of three units
Silver	No payable– below detection limit
Treatment Charge	US\$115/dmt of concentrate
Price Participation	Nil
Silver Credit	No payable– below detection limit
Penalties Allowance	Nil

19.3.4 Lead Concentrate Transport

The lead concentrate will be trucked to the Pine Point Junction railway facility approximately 65 km west of the mine site, where it will be shipped to North American and Asian markets. Trucking and rail car loading costs have been estimated at \$30.56/wmt. The freight and logistics costs for transporting the lead concentrate from Pine Point Junction to North American and Asian Smelters have been estimated at \$192/dmt, based on a preliminary marketing strategy. The transport losses have been estimated at 0.2%.



19.4 Royalties

On February 11, 2022, Osisko Metals announced that it entered into a binding letter agreement with Osisko Gold Royalties Ltd. ("Osisko Royalties"), pursuant to which Osisko Royalties will be granted a further 1.0% net smelter returns royalty on the Pine Point Project in exchange for a cash consideration of \$6.5 million. After this NSR Amendment, Osisko Royalties will hold a combined 3.0% NSR royalty on the Pine Point Project. This royalty is not subject to buy-back rights in favour of Osisko Metals.

19.5 Contracts

No contractual arrangements for concentrate trucking, rail freight, port usage, shipping or smelting & refining are currently in place. Furthermore, no contractual sales arrangements have been made for the lead concentrate or zinc concentrate at this time.

19.6 Comments on Chapter 19

Marketable high-grade zinc and lead concentrates would be produced from Pine Point. Initial testing of the zinc concentrate indicates that Magnesium Oxide is at the current smelter penalty threshold. However, no penalties are expected on the zinc or lead concentrates. The indicative terms selected for the economic analysis are in line with current market conditions. As project development continues, it is recommended that additional marketing and logistics studies should be undertaken to confirm the expected high interest in the mine's lead and zinc concentrates.

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

20.1 Environmental Studies

The information summarized in this chapter is in part derived from information provided by Golder, a member of WSP (WSP Golder) in support of the Preliminary Economic Assessment ("PEA"), a third-party consulting firm contracted by Pine Point Mining Limited ("PPML") to conduct baseline and environmental assessment studies.

The Pine Point Project is located primarily on a previously disturbed site resulting from Cominco Ltd.'s historical mining activities in the Northwest Territories ("NWT" or "NT"), near the southern shore of Great Slave Lake. The Project site is accessible by Highways 5 and 6 between Hay River and Fort Resolution, NWT. Existing environmental studies in the area surrounding the Project include:

- Studies completed by EBA Engineering Consultants Ltd. ("EBA") on behalf of Tamerlane Ventures Inc. ("Tamerlane") in 2005-2006 (EBA, 2005a,b,c, 2006a,b,c,d);
- Indigenous Traditional Knowledge study reports conducted in October 2006 with the cooperation of Indigenous groups (Swisher, 2006a,b);
- The environmental assessment of the Pine Point Pilot Project, by Tamerlane in 2007 (environmental assessment EA0607-002 and water licence MV2006L2-0003), in the area known as the West Zone (see also MVEIRB, 2008 and EBA, 2007).

Much of the information presented in these studies remains relevant for describing historical trends that have influenced existing conditions. Recent studies include additional baseline investigations completed by Tamerlane following the approval of the Pine Point Pilot Project and field surveys completed by PPML for the proposed Project between 2018 and 2022. Recent studies include:

- Studies completed by Rescan Environmental Services Ltd. ("Rescan") on behalf of Tamerlane in 2011 (Rescan 2012a-l);
- Field studies were completed by WSP Golder on behalf of PPML in 2018 to 2022).
- Previous studies include:
 - Bathymetry of existing historical open-pits;
 - Noise monitoring;
 - Vegetation surveys;
 - Groundwater quality monitoring;

- Wetland characterizations;
- Heritage resources surveys;
- Hydrology studies;
- Traditional Knowledge and Socioeconomic studies; and
- Demographic data gathering.

Project-specific baseline data collection has been conducted for the following disciplines:

- Air quality;
- Surface water quantity and quality;
- Fish and fish habitat;
- Soils and vegetation;
- Wildlife and species at risk;
- Archaeology.
- Waste rock characterization;

Additional baseline studies are planned for the following disciplines:

- Noise and visual;
- Sediment quality, plankton, and benthic invertebrates;
- Socio-economics;
- Traditional land and resource use;
- The Project is located at the edge of the Boreal Plains and Taiga Plains Ecozones, and within the Slave River and Hay River Lowland Ecoregions. The area is characterized by short, cool summers and long, cold winters. The average monthly temperatures in 2019 at the closest monitoring station (Hay River Airport) ranged from a minimum of -22.7 °C in February to a maximum of 15.5 °C in July. The winter months are typically the driest, with the most precipitation usually occurring in August.

The two nearest major drainages to the site are the Buffalo River, which is located approximately 30 km to the west of the Project Process Plant Site and the Little Buffalo River located about 35 km to the east. These watercourses flow north into Great Slave Lake, which is situated immediately north of the Project boundary and about 1.5 km north of the closest mining lease area. The existing water quality of Buffalo River and Great Slave Lake is typical of natural background values for this area of the NWT, with concentrations of most parameters below the federal water quality guidelines for the protection of aquatic life and drinking water (CCME 1999; Health Canada 2006).

Fish species that occupy the Buffalo River include Inconnu, Whitefish, Northern Pike, Walleye, and Burbot.

The Project is located in an area of sporadic, discontinuous permafrost with generally subdued topography. To date, permafrost was found at one location in 2019 during the baseline soils survey. However, permafrost has not been found during any drilling activity completed by PPML since 2017. It is thus assumed that permafrost conditions are unlikely to occur in the Project area.

The vegetation in the surrounding area is characterized by medium to tall, closed stands of jack pine and trembling aspen. White and black spruce dominate older stands of forest. Poorly drained fens and bogs in this region are covered with low, open stands of larch, black spruce, and ericaceous shrubs. Wildfires have been a common occurrence in the South Slave Region.

Moose, boreal caribou, and occasionally wood bison are the main ungulates found in the Project region. Hunting and trapping occur in the vicinity of the Project. Wildlife identified as being present and harvested include caribou, lynx, wolf, river otter, marten, beaver, muskrat, black bear, snowshoe hare, porcupine, ptarmigan, ruffed grouse, and waterfowl. Migratory songbirds typical of the boreal forest are also present in the area. The shore of Great Slave Lake immediately north of the Project is an important staging area for waterfowl during their annual migrations.

Environmental studies are ongoing to support the advancement of the Project. The data obtained from these studies will be used to identify mitigation, assess Project environmental impacts, and monitor impacts and the effectiveness of mitigation once the Project is underway.

20.2 Social or Community Considerations

PPML is committed to conducting its business responsibly with the communities in areas where it operates, through the company's Code of Ethics, and to making a positive contribution to the well-being and development of these communities. The Engagement Plan for the Confirmation and Exploration Program was approved by the Mackenzie Valley Land and Water Board ("MVLWB") in April 2022 and this plan will be expanded to include further detail and more parties as the Project enters the environmental assessment process. Public engagement is also driven by the duty to consult through the requirements of acts and regulations. These requirements will be managed through the Project's Engagement and Collaboration Plan. An Engagement and Collaboration Framework was developed and submitted with the Environmental Assessment Initiation Package. PPML prepared the Engagement and Collaboration Framework to guide engagement activities and allow for feedback on the document prior to the submission of the Developer's Assessment Report ("DAR"). The Engagement and Collaboration Framework will form the basis for ongoing engagement as the Project advances and will be a living document that is updated over the course of the regulatory process and the Project development.

PPML has, at this time, prioritized potentially affected parties for involvement based on the expected level of impact. The list of potentially affected parties will be updated as the Project advances based on feedback from those engaged and other parties that self-identify throughout Project development.

The Project lies within the South Slave Region of the NWT, and within the traditional territories of the Akaitcho Dene First Nations (including the Deninu Kué First Nation), Dehcho First Nations (including the K'at'l'odeeche First Nation), and the Northwest Territory Métis Nation ("NWTMN"). Engagement has been ongoing in these three communities in relation to the Project since 2017. This engagement has facilitated the early identification of potential issues and concerns which are being considered in Project design and environmental mitigation. PPML has entered into Collaboration Agreements with the Deninu Kué First Nation and the NWTMN, and has also entered into an Exploration Agreement with the K'at'l'odeeche First Nation. Other communities in the region are also being engaged regarding the Project, including Smith's Landing First Nation, West Point First Nation, North Slave Metis Alliance, Deh Gah Gotie First Nation and the municipalities of Hay River, Fort Smith, Fort Resolution and Yellowknife.

20.3 Anticipated Concerns

Sustained engagement with regulatory agencies, Indigenous groups and local communities affected by the Project is key in addressing and mitigating concerns.

A preliminary pathway analysis was undertaken to identify risks to the biophysical and human environments from the Project as part of the Environmental Assessment Initiation Guidelines (MVERB 2018). This analysis identified potential interactions between the Project and the environment, further identifying potential mitigation strategies to eliminate and/or minimize potential adverse effects and resulting impacts.

The following sections lists key concerns that have been identified to this stage in the regulatory process. A list of Project concerns will also be collected during ongoing Project engagement, and, where possible, mitigations will be applied to address those concerns.

20.3.1 Water Management

For the environmental assessment and permitting processes, a detailed Water Management Plan will be provided outlining the overall water balance for the site, the volumes and qualities of water to be managed, and the design of water management structures. Reviewers, such as the Mackenzie Valley Environmental Review Board ("MVEIRB"), MVLWB, and the Government of the Northwest Territories ("GNWT") will require a detailed understanding of the volume and associated quality of mine dewatering and surface water that will require management in the Project area

during all phases of the Project life. In addition, the water management plan will describe how water quantity and quality will be managed without adverse effects on the receiving environment. It is expected that the detailed water management plan will outline water transfers throughout all stages of the mine life, including the plan for discharge and water treatment.

Preliminary site water modelling has been initiated to develop and interlink groundwater, water quantity, and water quality models for the site. These models will be further developed in future engineering studies; a written Water Management Plan will be developed to support regulatory applications.

The water management plan will also include contingencies and adaptive management measures (i.e., how will PPML manage larger than expected volumes and/or poorer quality water than anticipated).

20.3.2 Water Quality

As part of the environmental assessment and permitting processes, it is anticipated that the methods, analysis, results, and related assumptions for water quality and potential effects on the receiving environment for the Project will be evaluated by the MVEIRB, MVLWB, GNWT, Fisheries and Oceans Canada ("DFO"), Environment and Climate Change Canada ("ECCC"), and other reviewers. Water quality is one of the environmental components of interest for Indigenous groups and local communities addressed through engagement activities.

PPML is evaluating water management options, including storing mine water in historical open-pits and underground workings, as well as managing runoff in stormwater ponds. The current proposed plans assume surface water can be managed for the Project in such a way that any discharge to nearby watercourses (e.g., Buffalo River) or waterbodies (e.g., Great Slave Lake) will comply with applicable water quality guidelines, and effects on the receiving environment will not be significantly adverse. The location(s) of discharge and associated water volumes and qualities through all Project phases, as applicable, will need to be determined in future studies including the need for treatment.

Water quality modelling will be completed for all Project phases (i.e., construction, operations, and reclamation and closure), and into post-closure, to compare water quality predictions to baseline conditions and applicable guidelines. Water quality models will be interlinked with the hydrogeological and water quantity models.

Applicable guidelines include Protection of Aquatic Life, the Metal and Diamond Mine Effluent Regulations criteria, protection of water for wildlife consumption, and protection of sources of

drinking water, where applicable. Community-specific criteria or guidelines will also be considered, where available.

20.3.3 Caribou

Woodland caribou (Boreal population) is a threatened species under the federal *Species at Risk Act* and is identified as being present in the general area of the Project through previous studies. It is also an important subsistence, cultural, and economical species for Indigenous groups and local communities.

Construction of the Project is expected to result in a direct loss or alteration of caribou habitat, which may alter caribou abundance and distribution. PPML is proposing a Project primarily developed over previously disturbed land to minimize potential effects on caribou. Availability and distribution of suitable caribou habitat will be mapped and existing versus removed habitat will be compared for the Project, in consideration of habitat loss due to natural occurrences in the region, i.e., natural fires. Protection measures will be proposed per regulatory requirements and industry best practices, based on the adequacy of the baseline data to understand current conditions and future conditions which may or may not be related to the Project. Protection measures will also incorporate Indigenous Traditional Knowledge, as well as information gathered through engagement activities, where appropriate. Residual impacts to Boreal caribou and their habitat following mitigation may require a caribou habitat offset plan.

20.3.4 Indigenous Land and Resource Use

The Project falls within the traditional territories of the Akaitcho Dene First Nations (i.e., Deninu Kue First Nation), the Dehcho First Nations (i.e., K'atl'odeeche First Nation), and the NWTMN. Project-induced effects on the Indigenous land and resource use of these groups will be largely assessed within the study areas defined for potentially affected resources through the environmental assessment. As PPML is proposing a Project, primarily developed over previously disturbed land, it is anticipated that impacts to Indigenous land and resource use can be minimized. Indigenous Traditional Knowledge and the results of engagement with Indigenous groups and communities will be key in determining the potential effects of the Project on Indigenous land and resource use, and are dependent on the extent and availability of Indigenous Traditional Knowledge and engagement participation by Indigenous groups and local communities.

20.3.5 Social and Economic Conditions

Indigenous groups and local communities in the South Slave Region of the NWT may be subject to socio-economic effects from the Project, particularly with regards to territorial economics and population-driven effects. The Project will affect individuals and communities in different ways, depending on their proximity to the Project, their relationship with the area of the Project, and their degree of participation in the Project itself. Population and employment impacts will be assessed quantitatively, though the overall approach to socio-economic impacts will be through a qualitative assessment, incorporating Indigenous Knowledge and the results of engagement activities.

Employment and contracting opportunities will be provided preferentially to qualified candidates from nearby communities with an interest in the Project, and preference will subsequently be given to current residents of the NWT. Communities with signed Impact Benefit Agreements ("IBAs") with PPML will be targeted and prioritized for employment and other contracting opportunities. The Company has already entered into Collaboration Agreements with Deninu Kué First Nation and the NWTMN, and has also entered into an Exploration Agreement with the K'at'l'odeeche First Nation.

20.3.6 Legacy Effects and Expectations for Closure and Reclamation

The Project is located on a previously disturbed site resulting from Cominco Ltd.'s historical mining, milling, and working accommodation activities. The Terms of Reference for the DAR indicate the need to consider the legacy effects of Cominco Ltd.'s former mining operations in the environmental assessment, and consider the biophysical and human environment prior to any historical mining at the site. Plans are underway to collect baseline data from field work and/or Indigenous Traditional Knowledge to support the characterization of the reference (i.e., predevelopment) conditions for individual disciplines.

As described in Section 20.6, the historical Pine Point Mine was considered closed by the territorial and federal governments. PPML has had ongoing discussions with the Governments of Canada and the NWT to determine environmental liabilities with regard to existing impacts from operations under Cominco Ltd. Furthermore, PPML has had discussions with other regulators (i.e., MVEIRB and MVLWB) and Indigenous groups and communities regarding the previously disturbed nature of the site and PPML's liabilities. PPML will submit a conceptual closure and reclamation plan as part of DAR and Interim Closure Plans as part of the environmental assessment and permitting process that will address liability and securities. PPML will continue to engage with communities on the closure and reclamation planning for the Project with the aim to manage the expectations of communities regarding PPML's responsibilities and the overall status of the site.

20.4 Mine Waste and Water Management Strategy

The mine waste and water management components are described in Chapter 18. The following paragraphs present a summary of the proposed conceptual water and waste management plans. These plans will be updated in future studies.

20.4.1 Geochemistry

According to the available information, waste rock ("WR") and tailings are considered non-potentially acid generating ("Non-PAG"), but tailings and mineral sorter rejects could be metal leaching. Additional sampling and testing programs are required, and are planned to be carried out in the next steps of the Project development, especially to verify the tailings and mineral sorter rejects metal leaching potential and for developing source terms for water quality models.

20.4.2 Waste Rock (WR) and Overburden (OB)

Where possible, the WR will be backfilled in nearby historic pits or in available proposed mined-out pits. The remaining WR will be stored in waste rock storage facilities ("WRSF"). The overburden ("OB") will be stockpiled separately from the WR as a large proportion of the OB will be used for reclamation purposes. Considering the number of pits, the design and locations of the WRSF were developed based on proximity, limiting the overall footprint and limiting, as much as possible, hauling distances. Building several WRSF close to active open-pits was preferred to hauling WR for longer distances to consolidate in a single WRSF.

Considering the performance of existing WR piles at the site and in agreement with the preliminary geochemical assumptions, no improvement of the foundation for groundwater protection is deemed necessary. However, geochemical assessments and water quality modelling are planned to verify the need for groundwater protection measures or alternative WR storage options.

20.4.3 Tailings

Process tailings will be backfilled in available mined-out open-pits (Tailings Deposition Areas or "TDA's"). Tailings will be delivered as thickened slurry and will be deposited in a way to maximize space in each open-pit within the bedrock limits. Survey and bathymetries conducted for the existing pits in the Project area have confirmed there is sufficient available space for the entire life of mine ("LOM"). Geochemical assessments and water quality or seepage modelling are also planned to confirm the proposed in-pit tailings storage option.

No surface tailings facility is necessary for the Project. All the TDA's will be levelled with the existing topography and thus will have no visual impact in the long term.

20.4.4 Mineralized Material and Mineral Sorter Rejects

Mineralized material will be sorted at the process plant site. The ore from the mineral sorter will be sent to the process plant for processing, while the mineral sorter rejects will be temporarily stockpiled at the designated pad prior to hauling them to a TDA to continue backfilling in historic or mined-out pits above the bedrock interface.

No on-surface permanent disposal area is necessary for the mineral sorter rejects.

20.4.5 Contact Water Management – WRSF, TDA, and Surface Water

The overall Water Management Plan requires that all contact water reporting from various elements of the Project is directed and managed so that all water discharged to the receiving environment meets discharge criteria as schematically illustrated in Figure 20-1. The Water Management Plan includes the following elements:

- Management of contact and seepage water from the Waste Rock Piles and Overburden Piles;
- Management of contact and/or seepage water from the Tailings and Mineral Sorter Rejects disposal areas;
- Management of water from mine dewatering;
- Water Balance Modelling to quantify flows and optimize water reclaim to meet process needs.

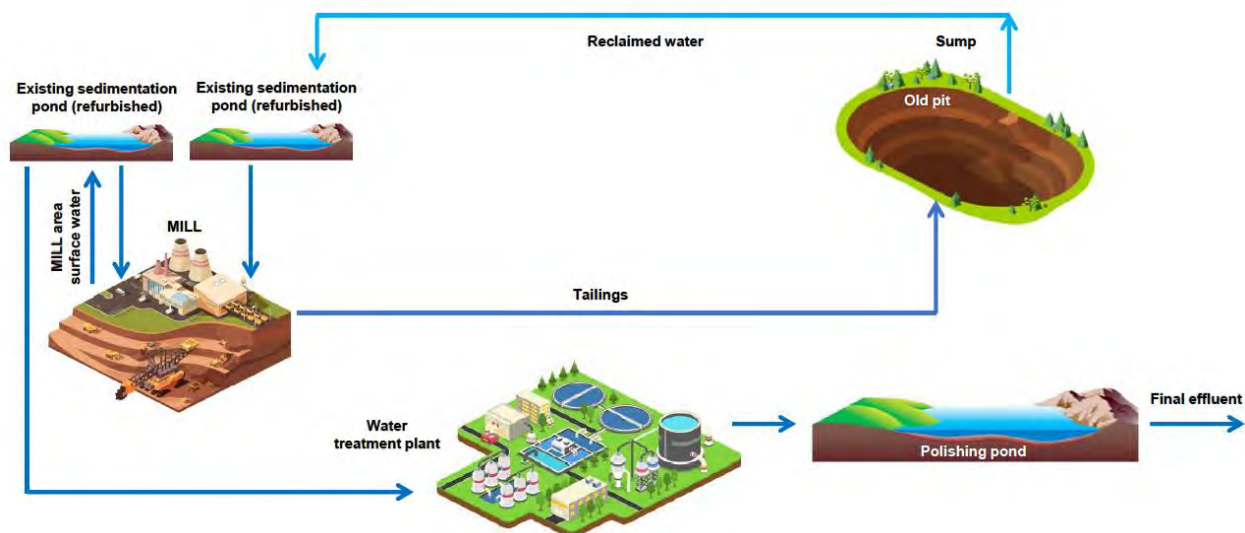
Waste Rock Storage Facilities and Overburden Stockpiles

The water management system is designed to limit the number of points of discharge of effluent to the environment. Therefore, stockpile contact water is collected by a network of ditches surrounding each facility, directed into lined collection ponds for control and management of the Project flood event and then pumped into a nearby historical pit by a water pipe network. This water management system for OB stockpiles and WRSF, as described in Chapter 18, has been designed assuming metal leaching is not an issue for the runoff from these piles. Potential for metal leaching of the WR will be verified by geochemical assessments planned in future studies. The lined water collection ponds provide an opportunity to control the discharge, including the eventual potential for metal leaching.

Monitoring during the LOM will be done according to the requirements of the regulating authorities and determined during the Project permitting phase.

Tailings, Mineral Sorter Rejects, and Process Plant Area

Tailings and mineralized material contact water consists of supernatant water from the TDA, runoff water from the sorter rejects temporary stockpile, and runoff water from the process plant sub-watershed. The water will be pumped back to the process plant as reclaim water. Excess water not needed for the process will be directed to a series of two sedimentation ponds, where sedimentation, treatment and polishing can occur prior to discharging. The sedimentation ponds will consist of a series of rehabilitated existing ponds, located 3 km south of the process plant area, and each have a potential storage capacity of 40,000 m³. The need for a water treatment plant has been considered. The plant has been located in the vicinity to provide for control mainly of Total Suspended Solids ("TSS"), and possibly zinc and lead. Figure 20-1 illustrates the Tailings Water Management flowsheet. The water balance and water quality models to be developed in future engineering studies will determine the volumes of water to be managed during all phases of the Project life and the quality of the water, and if additional water storage and/or treatment are required.



Note: A water treatment plant has been included in consideration of potential Project needs.

Figure 20-1: Tailings and reclaim water management overview

20.4.5.1 Water Balance and Water Quality

The volume of water generated by a rainfall event and/or snowmelt over the process plant site sub-watershed, as well as direct precipitation on the sedimentation pond area, was determined by performing a preliminary simulation with the PCSWMM software. A simple mass balance was then performed to establish the relationship between inflows of surface water and ground water, sedimentation ponds capacity, volumes of water required to feed the process plant, and the water reclaim volume from the TDA. The mass balance was used to establish preliminary effluent volumes and needs in terms of water treatment. Assumptions using available information were made in terms of the water quality. A more detailed water balance for the Project and water quality modelling will be undertaken in the future engineering studies.

20.4.5.2 Water Treatment

The volume exceeding the capacity of the restored sedimentation ponds was used to calculate the maximum monthly treatment rates required. The peak monthly treatment rate was estimated to be 135 m³/h, and this rate was used for preliminary sizing of the water treatment plant. The expected average annual treatment capacity is estimated to be of 88 m³/h. More detailed design of the water treatment facility will be undertaken in future engineering studies.

20.4.6 Mine Dewatering

The mine dewatering plan builds on experience from the historical operations, as well as recent hydrogeological modeling, as described in Chapter 16. Dewatering wells and submersible pumps will be used to lower the groundwater to levels below the proposed pit bottoms or underground mines lowest levels. The water is then pumped to nearby available historical or mined-out pits or ponds for management and final discharge.

Dewatering methods, calculations, and estimates are described in Chapter 16. Additional pumping tests and groundwater monitoring programs are recommended to improve the accuracy of water flow rate estimates and the overall water management planning for mine dewatering water, including the potential need for discharge and treatment of groundwater.

20.5 Regulatory Context

The regulatory process in the NWT includes a combination of both federal and territorial jurisdiction, with environmental assessment and regulatory processes overseen by NWT boards. The Project lies primarily within non-federal lands, with some areas of exception. The Project will be assessed and regulated primarily under the federal *Mackenzie Valley Resource Management Act*

("MVRMA"), which governs the environmental assessment process. Applications for a Land Use Permit, Water Licence, and other regulatory approvals will be initiated following the completion of the environmental assessment. The Land Use Permit will be issued by the Mackenzie Valley Land and Water Board ("MVLWB") under the authority of the MVRMA, while the Water Licence will be issued by the MVLWB and approved by the Government of the Northwest Territories ("GNWT") under the NWT *Waters Act*.

20.5.1 Environmental Assessment Process

In PPML's initial discussions related to the Project, community and regulators indicated that the Project "might cause significant public concern." As a result, the Project was directly referred to the MVEIRB for an environmental assessment under the MVRMA.

PPML developed an Environmental Assessment ("EA") Initiation Package for the Project, per the MVEIRB *Draft Environmental Assessment Initiation Guidelines for Developers of Major Projects* (EA Initiation Guidelines; MVEIRB 2018). The EA Initiation Package was submitted to MVEIRB in February 2021 and included the following:

- A Project Description;
- A summary of the existing biophysical and human environment;
- A summary of impacts and mitigations;
- A Developer's Assessment Proposal;
- Framework versions of environmental management plans.

The EA Initiation Package was reviewed by parties who provided comments and PPML provided written responses. Following public scoping sessions led by the MVEIRB, a draft Terms of Reference were issued in August 2021, which allowed for parties and PPML to provide comments. The final Terms of Reference for the Developer's Assessment Report were issued by the MVEIRB in November 2021, which list the requirements to complete a fulsome environmental and socio-economic assessment. PPML will submit a Developer's Assessment Report based on the final Terms of Reference, initiating the review phase of the environmental assessment process.

The environmental assessment review process will include receipt of and addressing information requests, participation in technical sessions, and a public hearing. Throughout the review process, PPML will provide information to regulatory authorities, parties to the review process, and most specifically the MVEIRB so they can issue a decision on the assessment to the GNWT Minister of Lands for approval. It is expected that MVEIRB will specify measures in its Report of Environmental Assessment that must be implemented by PPML and regulatory agencies to avoid significant impacts during the construction and operation of the Project.

20.5.2 Regulatory Process

Following approval of the environmental assessment, PPML will proceed through the regulatory process for issuance of a Water Licence and a Land Use Permit. This process will be led by the MVLWB, who have the authority to recommend the issuance of a Land Use Permit and Water Licence that requires approval of the GNWT Minister of Lands. The MVLWB will also play a role in the implementation of the Report of Environmental Assessment measures. As part of this process, PPML will be required to engage with regulatory authorities as it completes detailed engineering and other designs for the Project, including plans for mitigation and environmental control features, and updates to management and monitoring plans.

The primary contents for the Land Use Permit and Water Licence applications include the following:

- A Project Description, including term and location of operations and Project component descriptions;
- Use and quantities of water required;
- Types and disposal methods for waste rock and overburden;
- Equipment required;
- Reclamation plans and reclamation security deposit;
- Predicted environmental impacts and proposed mitigations;
- Environmental management plans, including spill contingency measures;
- Public engagement plan and records;
- Technical support documents (e.g., water balance, modelling predictions);
- Applicable application forms, questionnaires, and fees.

Multiple environmental approvals and permits are anticipated to be required for the Project. A non-exhaustive list of required permits and authorizations (Table 20-1) has been developed based on the known components of the Project and typical components of a mining project. The mining leases, which provide the fundamental mineral and mining rights required for the Project, are already held by PPML.



Table 20-1: Authorizations likely to be required for the Project

Authorization, Permit, or Licence	Act and/or Regulation	Permitting Board, Agency, or Organization
Type A Land Use Permit (for mine construction and operation)	<i>Mackenzie Valley Resource Management Act</i> <i>Mackenzie Valley Land Use Regulations</i>	MVLWB
Type A Water Licence (for mining and milling)	<i>Waters Act</i> <i>Waters Regulations</i>	MVLWB
Quarry Permit	<i>Northwest Territories Lands Act</i> Quarrying Regulations	GNWT-Lands
Explosives Permit	<i>Explosives Act/Regulations 2013</i> <i>Explosives Use Act/Regulations</i>	Natural Resources Canada Workers' Safety and Compensation Commission
Approval to transport dangerous goods	<i>Transportation of Dangerous Goods Act/Regulations</i>	Transport Canada
Permit to Burn and Fire Preparedness Plan	<i>Forest Protection Act</i>	GNWT- Environment and Natural Resources
Minor Works Order	<i>Canadian Navigable Waters Act</i>	Transport Canada
<i>Fisheries Act</i> Review / Authorization	<i>Fisheries Act</i>	Fisheries and Oceans Canada
Wildlife Research Permit	<i>Wildlife Act</i>	GNWT - Environment and Natural Resources
Wildlife Management and Monitoring Plan approval	<i>Wildlife Act</i> (if the Project triggers Section 95 of the Act)	GNWT- Environment and Natural Resources
Archaeology Permit	<i>Archaeological Sites Act/Regulations</i>	GNWT- Education, Culture and Employment
Camp Sanitation Approval	<i>Public Health Act/Camp Sanitation Regulations</i>	GNWT-Health and Social Services
Waste Disposal Approval (for any off-site waste disposal)	<i>Mackenzie Valley Land Use Regulations</i>	List land use permits for off-site disposal facilities

Note: GNWT = Government of Northwest Territories; MVLWB = Mackenzie Valley Land and Water Board.

Depending on the Project footprint and infrastructure design, there is potential that the Project will cause a harmful alteration, disruption or destruction of fish habitat as defined under the *Fisheries Act* (i.e., from placement of watercourse crossings or intake/outfall structures). Project details will be provided to Fisheries and Oceans Canada for review, and if required, an application for authorization under Section 35 (2) (b) of the *Fisheries Act* will be submitted. If required, PPML will work with Fisheries and Oceans Canada on the details of the offsetting requirements for the Project.

Moreover, using a natural water body frequented by fish for mine waste disposal (i.e., tailings, waste rock, or mine water) requires an amendment to the Metal and Diamond Mining Effluent Regulations (“MDMER”), which is a federal legislative action, to add the water body to Schedule 2 of the MDMER. Subsection 36(3) of the *Fisheries Act* prohibits the deposit of deleterious substances into water frequented by fish, unless authorized by regulation and is administered by Environment and Climate Change Canada. As a result, PPML must also:

- Prepare an assessment of alternatives for mine waste disposal for consideration (according to the federal guidelines);
- Prepare a fish habitat compensation plan for consideration as part of the environmental assessment.

As the Project would be anticipated to cause significant disturbance to large game, substantially alter or damage wildlife habitat, or significantly contribute to cumulative effects, the Project will require a Wildlife Management and Monitoring Plan under the NWT *Wildlife Act*. This document will require approval by the Minister of GNWT Environment and Natural Resources.

An authorization under the Minor Works Order of the *Canadian Navigable Waters Act* is anticipated to be required in relation to the construction of the Project footprint, particularly for water intake and discharge infrastructure.

Several permits and authorizations are required per NWT acts and regulations for the purposes of protecting heritage and biophysical resources, as well as health and safety requirements. The NWT *Mine Health and Safety Act* and its regulations require PPML to submit annual reports for the mine site to the Chief Inspector of Mines, specifying the mine plan, mineral leases, surface leases, mineral claim boundaries, affected areas and site infrastructure information.

20.5.3 Regulatory Timeline

The final Terms of Reference describe the detailed requirements for the Developer’s Assessment Report (DAR), a document that typically takes 8 to 10 months to prepare for a project of this scale, following the completion of baseline surveys and further engineering studies.

Once the DAR has undergone a conformity and/or adequacy review by the MVEIRB, the MVEIRB will set a work plan for the review process which will outline the timelines. It is expected that a hearing will be required and, therefore, the MVEIRB process has a time limit of 16 months excluding PPML’s response times. The GNWT Ministerial review is within three months after which the Minister must either:

- Accept MVEIRB’s report and recommendations;

- Refer it back to MVEIRB or further consideration; or
- Accept the report with modifications, after consulting with MVEIRB.

The Minister will distribute the ministers' decision to all Indigenous groups and organizations, local governments, regulatory authorities, and federal and territorial government agencies affected by the decision.

The MVEIRB has 30 days to issue an amended development certificate after it receives the ministers' decision. Each regulatory authority must then incorporate the new or amended development certificate conditions in any licence, permit or other authorization that it issues, amends or reviews, to the extent of its authority to do so.

Once the Land Use Permit and Water Licence application has been submitted, the MVLWB will provide PPML with a work plan that outlines the timelines for the review process. The MVLWB is required to decide on a Water Licence application within nine months, excluding applicant time, after the application has been deemed complete. Any time that the applicant is providing information required by the MVLWB (e.g., responses to review comments, information requests, interventions) is not counted as part of the nine-month time period. In the case of a Type A Water Licence, the GNWT Minister must make a decision within 45 days, but this timeline may be extended by the Minister by a further 45 days.

20.6 Mine Closure Requirements

The Project is located on a previously disturbed site resulting from Cominco Ltd.'s historical mining, milling, and working accommodation activities. The historical Pine Point Mine is considered closed by the territorial and federal governments. Currently, the territorial government is liable for existing infrastructure, excluding the railbed, which is a federal government liability, and the historical tailings impoundment for the Pine Point Mine, which is undergoing closure activities by Teck Metals Ltd.

The regulating authorities have established requirements for mine closure for projects in the Northwest Territories, which are presented in the Guidelines for the Closure and Reclamation of Advanced Mineral Exploration and Mine Sites in the Northwest Territories (MVLWB/AANDC, 2013) (hereinafter, the "Guidelines"). For the Pine Point Project, a framework version of a Closure and Reclamation Plan ("CRP") was submitted with the EA Initiation Package. An updated CRP will be developed during the permitting phase of the Project, or potentially earlier if required, based on feedback through the EA process. The CRP framework details closure plans for the Project only and does not include activities or monitoring associated with historical mining activities at or near the Project, outside of developments directly associated with the Project. As indicated in the CRP

framework, the closure goal for the Project is similar to that shown in the Closure Guidelines (MVLWB/AANDC, 2013) and comprises two parts to reflect the historical disturbance that has already been experienced by the site:

"For previously undisturbed areas, the goal is to return the affected areas of the site developed by the Project to viable and, wherever practicable, self-sustaining ecosystems that are compatible with a healthy environment and human activities. Where areas of the Project have been previously disturbed through historical mining activities, the goal is to return the areas of the site affected by the Project to an equivalent environmental state that they were left by the Government of Canada prior to the Project."

An updated CRP will be prepared and submitted with the application for the Land Use Permit and Water Licence. The CRP that will be developed to support the Water Licence and Land Use Permit applications that will be submitted to the MVLWB for review and approval. The CRP for permitting will incorporate relevant feedback and commitments made by PPML during the EA review process. An Interim Closure and Reclamation Plan ("ICRP") is generally required within 12 months of the issuance of the Water Licence.

Mining companies must also provide the regulating authorities with a financial security to ensure that they are able to bear the cost of the reclamation work including care, maintenance, and monitoring (MVLWB/AANDC, 2013). The amount due is set by the MVLWB and depends on the scale of the Project and its level of detail. The Guideline recommends that companies use the RECLAIM model developed as described in the LWB/GNWT CIRNAC Guidelines for Closure and Reclamation Cost Estimates for Mines (2022) to estimate the closure cost and, thus, the financial security required.

The following mine closure concepts are preliminary (at a PEA level) and are subject to change as the mine design is advanced.

20.6.1 Closure Concept

Mine site reclamation is carried out with the intent to limit, as much as possible, the impacts of the site on its environment after closure. The general closure concept presented considers the Design for Closure practice, upon which designing a mine site whilst considering its closure allows for anticipation and mitigation of potential environmental and social risks. This concept also helps reduce closure costs as it reduces replicated work.

A formal process of engagement with local communities is required to develop the objectives for the closure and reclamation plan. Where possible, affected areas will be reclaimed progressively

during mine operation, which will provide information regarding site-specific results with respect to the efficacy of the methods used. The progressive reclamation work completed during operations will be presented in ICRPs and in a final CRP, as required by the Guidelines. When the closure objectives are met for the completed progressive reclamation, an application to reduce the financial security can be made. Progressive reclamation reduces the time required for the final reclamation works, as the partial completion of the reclamation works will be demonstrated in every interim CRP.

20.6.2 Reclamation Objectives

For the study, the objectives adopted for the reclamation plan for the mine are to minimize the lasting environmental impacts of operations to the extent practical and to allow disturbed areas to return to a similar productive state that existed prior to disturbance. The reclamation objectives will be developed in consultation with local communities.

Short-term reclamation objectives are to:

- Progressively reclaim disturbed areas as soon as they are no longer active;
- Minimize the risk and impact of water erosion and sediment transportation;
- Stabilize slopes; and
- Prevent soil drifting/dust.

Long-term objectives are to:

- Return disturbed areas to similar state as existed prior to the disturbance;
- Maintain the level of wildlife habitat.

Considering the current available geochemical information, the concepts, short- and long-term reclamation objectives are mostly pertinent to the WRSF and TDA of the Project.

20.6.3 Closure Activities

The tailings from the process plant will be deposited in historical pits, along with the mineral sorter rejects, thereby avoiding the construction of a conventional tailings management facility, reducing the Project footprint and long-term management of the tailings. Upon closure, the open-pits used for tailings deposition would have been filled to ground surface with mineral sorter rejects, and WR where necessary. The open-pits will be covered with 300 mm of till stored in the OB piles and contoured to restore the natural drainage. Measures to stabilize the surface and limit erosion will be implemented in consultation with local communities. These measures may include

revegetation. Note that the tailings, mineral sorter rejects, and WR are not foreseen to be acid generating, but could be metal leaching; thus, further testing is necessary to address this potential concern.

A portion of the WR will be backfilled into historical or proposed mined-out pits during the operation to limit the Project's footprint and double handling. The surface of the backfilled pits and the remaining WR piles will be covered by 300 mm of till stored in the OB piles. Note that the effects of the deposition of tailings, mineral sorter rejects, and WR in the pits that will be water covered require further testing to understand the potential to influence the water quality of the overlying pit water. Where WR cannot be stored in nearby pits, the final design of the WR piles will be developed in consultation with local communities.

Part of the remaining OB material will be used to backfill WR and OB water management network, such as ditches and collection ponds. Pumps and pipelines will be removed.

At the process plant site, the water management system components will be reclaimed by breaching the sedimentation ponds, removing pumps and pipelines, and dismantling any other water management infrastructure.

The remaining open-pits and underground mine accesses will be blocked and potential risks to the safety of local populations and wildlife will be identified and mitigated. The buildings, infrastructure and equipment on the site will be dismantled and disposed of properly. The plan for the reclamation of haul roads will be determined through engagement with local communities.

20.7 Greenhouse Gas ("GHG") Emissions Assessment

PPML has requested an assessment of the projected carbon footprint of the proposed mine to facilitate the identification of the major sources of GHG emissions and where efforts should be focused to find alternative methods, processes, or equipment to reduce the overall emissions. as the project definition improves throughout its design phases. In 2021, BBA Inc. ("BBA") conducted a GHG and Energy Consumption (i.e., Carbon Footprint) assessment of the Project, largely based on the Project's proposed design outlined in the initial Pine Point Lead-Zinc Project PEA Technical Report (2020). Subsequent project design alterations, including the addition of a solar energy farm concept as well as the extension of mining from 10 to 12 years, have prompted a re-assessment of the Project's Carbon Footprint. This section highlights the key findings of the revised GHG assessment.

20.7.1 Methods

This GHG assessment was drafted in accordance with the key GHG project accounting concepts, policies, and principles prescribed within the World Resources Institute's "The Greenhouse Gas Protocol for Project Accounting" as recommended by the Towards Sustainable Mining ("TSM") Guiding Principles of the Mining Association of Canada.

BBA utilized a Scope 1 GHG Emissions approach which quantified the predicted emissions from all owned operations within the battery limit of the mine site during the Project's production phase. Emissions from the production and supply of on-site consumables (e.g., NTPC electricity, cooling water, diesel, etc.) as well as the transportation of concentrate from Pine Point Junction to various smelters, were excluded from this assessment since they are attributed to 3rd party emitters as per the definition of a Scope 1 assessment. Emissions from the construction and closure phases, as well as intermittent backfilling operations, were also excluded since these activities aren't defined or sufficiently developed to assess their GHG footprint and will also be partly performed by 3rd parties who would fall under a Scope 3 assessment.

As per a Scope 1 assessment, the energy usage and emissions were categorized into five accounts: Open-pit Production, Underground Production, Process Plant, Power Generation, and Transport of Material to Pine Point. The Underground Production account was further divided into three sub-accounts: Central Zone, West Zone, and Sustaining CAPEX. Each account/sub-account was split into "operations" that typically emit GHGs, such as: hauling, drilling, heating, and dewatering.

Input data for this assessment, as volume of fuel consumed in each operation, was sourced from the initial Pine Point Lead-Zinc Project PEA Technical Report (2020) as well as updated engineering assessments conducted by BBA, WSP, and Nahanni Construction (2022). Diesel, natural gas, and propane are assumed to be the three main fuel types to be used within all proposed operations at the Scope 1 level. GHG emissions from each operation were quantified by converting the volume of fuel combusted into CO₂ equivalent factors ("CO₂eq") using emission factors sourced from scientific and recognized sources. The overall emissions from the operation of the mine have been divided into five distinct activities to facilitate understanding of the sources and enable potential measures to reduce them.

20.7.2 Results

The Project is anticipated to produce a combined total of 4,167,715 tonnes ("t") of zinc and lead concentrate over its 12-year LOM. Throughout the production phase, the Project is estimated to consume 142,213,975 L of diesel, 14,107,000 L of propane, and 175,424,753 m³ of natural gas, which will result in the total emission of 806,470 tonnes of CO₂ equivalent ("tCO₂eq") over the LOM

(Table 20-2). The annual GHG intensity will vary between 0.15 and 0.24 tCO₂eq per tonne of concentrate produced ("tCO₂eq/t") with an average of 0.19 tCO₂eq/t throughout the LOM.

Table 20-2: Fuel consumption during the LOM

Fuel Type	Unit	Quantity LOM	tCO ₂ eq LOM	%
Diesel	L	142,213,975	398,700	49
Natural Gas	M ³	175,424,753	385,934	48
Propane	L	14,107,000	21,835	3
Total			806,470	100

Power Generation will be the largest source of GHG emissions over the LOM (272,956 tCO₂eq) out of the five assessed accounts. Open-pit mining will be the second largest source of GHG emissions (240,499 tCO₂eq) followed by Underground mining (159,300 tCO₂eq). The Process Plant (Mill) will be a relatively small source of GHG emissions (78,975 tCO₂eq), while the Transport of Material to Pine Point will be the smallest source of GHG emissions over the LOM (50,048 tCO₂eq) out of the five assessed accounts.

The Northwest Territories Power Corporation ("NTPC") grid will be unable to supply the total power required for the mine's infrastructure (i.e., process plant, camp facilities, administrative buildings). Supplemental power will be generated by an on-site solar production to shave the peak load as well as stationery, natural gas generators. While the solar farm will emit 0 GHGs, the natural gas generators will be the single largest source of GHG emissions over the LOM (272,956 tCO₂eq).

The loading and hauling of raw material from the Open-pit to the Mill will be the main driver (57%) of total Open-pit GHG emissions. Ancillary equipment will account for 37% of total Open-pit emissions, while drilling will account for 6%. All Open-pit operations will use diesel, except explosives which will use an emulsion mix consisting of approximately 3% diesel. Assuming there are no other carbon-based fuels in the emulsion mix, explosives will contribute only marginally to the total Open-pit emissions.

The largest source of GHG emissions from Underground Production will be the ventilation and de-watering of the two underground sites. Mobile, natural gas generators powering de-watering pumps will emit 56% of total GHGs, while grouting equipment (fueled by diesel) will emit 21%. Additional, natural gas gensets powering air movers (ventilation) will account for 13% of emissions while propane-fueled heating will account for 10%.

Out of the two Underground zones, the West Site will emit over seven times (7x) more GHGs than the Central Site due to several differences among the sites. First, the Central Site is located only

5 km from the mill, whereas the West Site portal is 40 km away, requiring longer hauling by diesel trucks. Additionally, since the Central Site has a smaller underground volume than the West Site, its propane-fueled heating requirements are significantly lower. It is also important to note that the emissions from drilling operations for water pumping for the Central Site were accounted for in the Open-pit emissions (data are lumped and undifferentiated) while those of the West Site were not.

Natural gas burners used for heating the material concentration will emit 82% of total Process Plant GHG's. The rest will be emitted from ancillary, diesel trucks and equipment. 84% of GHG's from this account will be emitted by diesel-powered hauling trucks transporting wet concentrate from the Mill to Pine Point Junction along a 65 km route from the mill. The rest will be emitted by diesel equipment/machinery, at the mill and Pine Point Junction, used for loading and unloading the material.

20.7.3 Conclusions and Recommendations

The key findings from this assessment should be used to target GHG sources with the largest impact on air quality and to explore different solutions for limiting emissions as the project design progresses and gains in definition. Using alternative fuels to power mobile equipment, such as hydrogen cell, biofuels, or natural gas, instead of diesel, could be a potential solution worth assessing in the future phases of the project definition.

The use of natural gas generators will reduce the GHG emissions (30%) compared to using conventional diesel generators. Because power generation is the largest GHG emission source, the use of LNG will enable the reduction of the total LOM GHG footprint by 10% compared to using diesel generators. To further offset emissions from the off grid (on-site) natural gas generators, PPML have optimized the power generation plan as per the variation of the load throughout during the LOM and availability of renewable source (hydroelectric) through the interconnection with NTPC. The addition of on-site solar energy will further reduce the emissions by enabling peak shaving of the load. Overall, the latest optimization of the operations, namely the dewatering plan and the power generation strategy, including solar energy, has enabled reducing emissions (3%) while increasing the overall production (10%) over the 12-year LOM. These reductions of GHG also translate into important cost savings related to the NWT carbon tax.

As per the Terms of Reference issued by MVEIRB, further analyses and evaluation of GHGs and effects on climate change will be completed to support the DAR.

21. CAPITAL AND OPERATING COSTS

The capital and operating cost estimates presented in this PEA for the Pine Point Lead-Zinc Project are based on the construction of numerous open-pit and underground mines as well as a process plant and site infrastructure. The process plant is designed to have a capacity of 11,250 tpd (4.1 Mtpy) over the life of mine ("LOM").

All capital and operating cost estimates cited in this Report are referenced in Canadian dollars as of Q2 2022.

21.1 Capital Costs

21.1.1 Summary

The total pre-production capital cost for the Pine Point Project is estimated to be \$653M (including contingencies and indirect costs). The cumulative LOM capital expenditure including costs for pre-production, sustaining, site reclamation and closure is estimated to be \$1,319M (Table 21-1). Figure 21-1 provides an overview of the capital costs (pre-production and sustaining) on a cumulative basis for the life of the Project.

Table 21-1: Project LOM capital cost summary

WBS	Cost area	Pre-production capital cost (\$M)	Sustaining capital cost (\$M)	Total cost (\$M)
000	General Administration (Owner's costs)	22.8	0.0	22.8
200	Underground Mine	0.0	183.3	183.3
300	Open-pit Mine	15.7	80.6	96.3
400	Electricity and Communications	45.7	19.3	64.9
500	Site Infrastructure	59.7	11.8	71.5
600	Process Plant	297.3	0.0	297.3
800	Tailings, Mine Waste and Water Management	47.7	123.6	171.3
900	Indirect Costs	76.6	0.0	76.6
999	Contingency	87.8	23.7	111.5
	Capitalized Operating Costs ¹	0.0	174.5	174.5
	Total	653.3	616.8	1,270.1
	Site Reclamation and Closure	0.0	68.0	68.0
	Salvage Value	0.0	-19.6	-19.6
	Total - Forecast to spend	653.3	665.1	1,318.5

1. The operating costs attributable to overburden and waste stripping that is in excess of the average strip ratio over the life of mine have been transferred to capital expenditures.

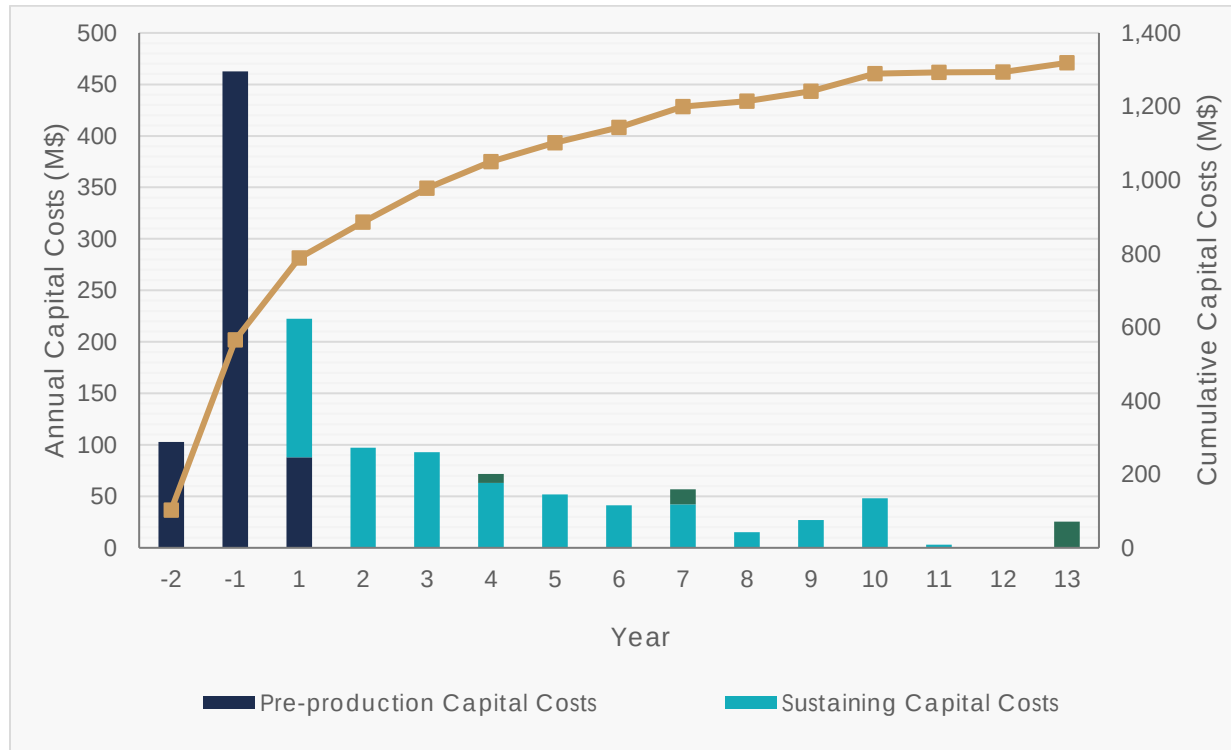


Figure 21-1: Annual and cumulative project capital costs

21.1.2 Scope and Structure of Capital Cost Estimate

The overall capital cost estimate developed in this Preliminary Economic Assessment Study generally meets the AACE Class 4 requirements and has an accuracy range of between -30% and +30%. The capital cost estimate for this study forms the basis for the approval of further development of the Project by means of further technical studies. Generally, engineering performed to date is between 1% to 10% of full project definition.

The capital cost estimate abides by the following criteria:

- Reflects general accepted practices in the cost engineering Standards;
- Assumes contracts will be awarded to reputable contractors on a cost reimbursable basis;
- Labour costs are based on Western Canadian Industrial construction rates;
- Winter conditions are expected between the months of October and April. This is incorporated within the productivity factors of the Project;
- Pre-production and sustaining capital costs are expressed in constant Q2 2022 Canadian dollars ("CAD"); with an exchange rate of 1.27 CAD for 1.00 US Dollar ("USD").

The Project schedule, from the feasibility study ("FS"), detailed engineering to start-up, was also used in the estimate preparation; refer to Chapter 24 for the preliminary execution plan and schedule. The decision to proceed with construction of the Project is expected to be made after the completion of the feasibility study and reception of permits. The cost estimate is divided into the following elements:

- **Pre-production Capital Costs:**
 - **Owner's costs** (WBS 000 General Administration): costs associated with specific personnel, management, support infrastructure, safety and environmental, community relations, administration and finance, human resources, training and others;
 - **Direct costs** (WBS 200 to 800): costs for mine development and permanent infrastructure. Includes services and equipment required for the surface infrastructure, open-pit mines, underground mines and the process plant;
 - **Indirect costs** (WBS 900): costs needed to support the construction of the facilities included in the direct costs. Includes engineering, procurement and construction management ("EPCM") services, EPCM temporary facilities (infrastructure) and construction management, capital spare parts, freight and logistics;
 - **Contingency** (WBS 999): includes variations in quantities, differences between estimated and actual equipment and material prices, labour costs and site-specific conditions. Also accounts for variation resulting from uncertainties that are clarified during detail engineering, when basic engineering designs and specifications are finalized.
- **Sustaining Capital Costs:**
 - **Capital expenditures** after the start of operations: surface tailings and reclaim water pipelines, underground mine extensions and associated infrastructure, production equipment replacement, and closure costs. These costs are included in the financial analysis in Chapter 22 in the year in which they are incurred. Capital costs expensed in Year 1 (2028) and beyond are classified as sustaining capital.

21.1.3 Work Breakdown Structure (WBS) and Estimate Responsibilities

The capital cost estimate was developed in accordance with Osisko Metals' work breakdown structure ("WBS") with the estimate responsibilities summarized in Table 21-2:



Table 21-2: CAPEX estimate responsibilities by WBS

WBS area	WBS description	Responsible entity
000	General Administration (Owner's cost)	Osisko Metals and BBA
200	Underground Mines	WSP
300	Open-pit Mines	WSP
400	Electricity and Communications	BBA
500	Site Infrastructure	WSP
600	Process Plant	BBA
800	Tailings, Mine Waste and Water Management	WSP
900	Indirect costs	Osisko Metals and BBA
999	Contingency	Osisko Metals, WSP and BBA
	Salvage Value	BBA
	Site Closure and Reclamation	WSP

21.1.4 Exclusions

The following items were excluded from the capital cost estimate:

- Risk related costs;
- Certain land acquisitions;
- Licensing and financing costs;
- Project development costs incurred to date, including studies and early works;
- Taxes (included in the financial model);
- Geotechnical anomalies (must be considered as a risk);
- Operating costs (except capitalized overburden and waste stripping);
- Changes to design criteria;
- Work stoppages;
- Scope changes or an accelerated schedule;
- Hazardous waste issues.

21.1.5 Pre-production Capital Costs

The pre-production capital cost summary of the Project is outlined in Table 21-3 and shown as a pie chart in Figure 21-2. The capital cost breakdown descriptions are outlined in the following sections.

Table 21-3: Project pre-production capital cost summary

Area	Cost area description	Pre-production capital cost (\$M)	CAPEX (%)
000	General Administration (Owner's costs)	22.8	3
200	Underground Mine	0.0	0
300	Open-pit Mine	15.7	2
400	Electricity and Communications	45.7	7
500	Site Infrastructure	59.7	9
600	Process Plant	297.3	45
800	Tailings, Mine Waste and Water Management	47.7	7
900	Indirect Costs	76.6	12
999	Contingency	87.8	13
	Total	653.3	100

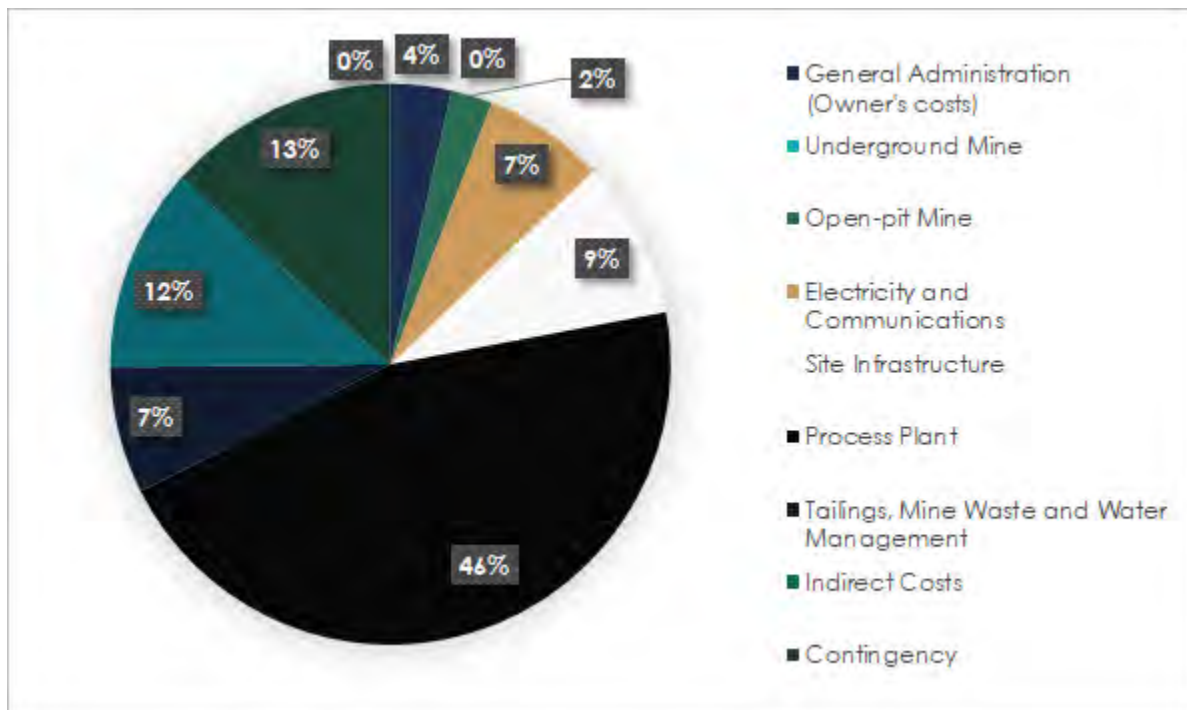


Figure 21-2: Distribution of pre-production capital costs

21.1.5.1 General Administration (Area 000) – Owner's Costs

The following items are part of the General Administration area, representing the Owner's costs during pre-production:

- Employee salaries until production begins;
- Energy, consumables and maintenance costs during open-pit mine development;
- Insurance during preproduction;
- Surface mobile equipment for the site and the warehouse;
- Environmental management and mitigation;
- Security;
- Pre-investment costs;
- Personnel training;
- Administration, financial and human resources costs;
- Community relations.

General Administration (Owner's) costs were estimated by Osisko Metals and total \$23M. Table 21-4 summarizes the General Administration pre-production capital costs.

Table 21-4: General administration (Owner's costs) pre-production capital cost summary (Area 000)

Cost area description	Total cost (\$M)	CAPEX (%)
General Management Expenses	10.2	45
Mine Pre-production	7.8	34
Process Plant Pre-production	2.1	9
Insurance	2.1	9
Mobile Equipment	0.6	2
Total	22.8	100

21.1.5.2 Open-pit Mine (Area 300)

WSP provided estimates for all open-pit mine capital costs. The total open-pit mine pre-production capital cost is \$15.7M. Open-pit dewatering equipment costs are not included as they are described in Section 21.1.5.6 (Area 800). Open-pit mine Owner's costs (\$7.8M) are included in the General Administration area described in Section 21.1.5.1. Development activities (clearing and grubbing) for the first two open-pits will start in Year -1.

Table 21-5 summarizes the pre-production open-pit capital costs and provides a breakdown per item. Pre-production capital costs for the open-pit mine consist primarily of mine preparation costs, mobile equipment down-payments and the purchase of a dispatch system. Open-pit dewatering costs are included in the water management costs.

Table 21-5: Open-pit pre-production capital costs (Area 300)

Cost Description	(\$M)	(%)
Site Preparation	0.4	3
Primary Mobile Equipment (15% Down payment)	9.5	61
Support Equipment	3.4	22
Dispatch System and Software	2.3	15
Total	15.7	100

21.1.5.3 Electrical and Communications (Area 400)

The electrical and communication capital costs for the Pine Point site were estimated by BBA based on project requirements. The total capital cost of the site electrical and communication infrastructure is estimated to be \$45.7M (Table 21-6).

Assumptions used to determine the capital cost of electrical infrastructure include the following:

- The current site electrical substation does not require a refurbishment to meet project requirements;
- The process plant will be located in the same area as the historical Cominco process plant, a short distance from the electrical substation;
- All site electrical distribution and communication infrastructure costs are included in the site infrastructure capital cost estimate (Area 500);
- Supplementary electricity for the open-pit mine dewatering, underground mine ventilation and dewatering as well as emergency backup power will be supplied by mobile LNG based electrical generators on an as needed basis, which will be leased from a reputable supplier;
- The mobile power generation infrastructure includes pads, switchgear, E-buildings, transformers, a heat recovery system as well as overhead lines to the various surface and underground facilities.

A summary of electrical and communications capital costs is provided in Table 21-6.

Table 21-6: Electrical and communication pre-production capital costs (Area 400)

Cost description	Total cost (\$M)	CAPEX (%)
Liquified natural gas gensets	20.3	44
Electrical infrastructure for power system	4.5	10
High voltage lines	7.8	17
Low voltage lines and transformers	7.3	16
Solar power plant	5.3	12
General	0.5	1
Total	45.7	100

21.1.5.4 Site Infrastructure (Area 500)

Site surface infrastructure capital costs for both the Pine Point site and the Rail Depot at Pine Point Junction were estimated by WSP. The capital costs for the site infrastructure were determined by performing preliminary level engineering and architectural design to generate factored estimates based on building surface area. The total capital cost of all the infrastructure facilities is estimated to be \$59.7M.

Assumptions used to determine the capital cost of site infrastructure include the following:

- The Project site has existing utility/access roads that implies no major additional cost for site road construction. New haul roads will be built to support the first year of production from the open-pits;
- The Plant Site infrastructure location is assumed to have a typical geotechnical capacity with no rock blasting required;
- Costs for the camp, service building, administration building, and site access control building are all based on factored estimates and similar projects;
- Recreational facilities will be built in Year 1;
- Fire water system costs are included in the capital costs for each building;
- Underground piping costs were estimated using a site layout and each of the building's requirements;
- Pine Point Junction infrastructure and railway spur lines were estimated using a preliminary level design;

A summary of the site surface infrastructure capital costs is provided in Table 21-7.



Table 21-7: Site infrastructure pre-production capital costs (Area 500)

Cost description	Total cost (\$M)	CAPEX (%)
Pine Point junction buildings	2.0	3
Railroad	1.8	3
Admin building	7.0	12
Camp building	9.7	16
Truck shop	12.4	21
Cafeteria building	4.5	8
Service Buildings	1.0	2
Fuel Storage Facilities	1.0	2
Fire Protection	1.4	2
Water pumping and distribution	2.7	5
Buildings	5.3	9
Railcar loadout	5.1	9
Haulage roads	2.3	4
Mobile equipment	3.4	6
Total	59.7	100

21.1.5.5 Process Plant (Area 600)

The design of the process plant was largely based on BBA's and Osisko Metals' experience on recent projects. This design included the production of a 3D model of the process plant by BBA and Osisko Metals during the PEA. An equipment list was developed with equipment sizes, capacities, and motor power from the process flow diagrams developed for the PEA. The cost estimate of the process plant was calculated from factors based on the purchase cost of process equipment and the process plant layout. Quantities were summarily developed for concrete and steel to validate the factoring methodology.

The total capital cost of the process plant facility is estimated to be \$297.3M as summarized in Table 21-8:



Table 21-8: Process plant pre-production capital costs (Area 600)

Cost description	Total cost (\$M)	CAPEX (%)
Building Structure, Piping and HVAC	92.4	31
Surface Primary Crushing	21.6	7
Secondary Crushing	48.3	16
Preconcentration	54.7	18
Grinding	18.2	6
Flotation	22.5	8
Flotation Reagent Distribution	1.4	0
Concentrate Dewatering & Pumping	19.8	7
Concentrate Storage	2.6	1
Tailings Dewatering & Pumping	5.8	2
Reagent Storage and Distribution	3.1	1
Process Services	2.2	1
Process Plant Laboratory	2.7	1
Mobile Equipment	2.0	1
Total	297.3	100

The following sections provide the basis for the capital cost estimates for the major component costs of constructing the process infrastructure.

Mechanical

An equipment list, including platework, was developed from the process flow diagrams. Budget pricing was obtained for the primary process equipment while the remainder of the equipment was factored based on recent prices obtained on a project with similar equipment. The Installation cost is factored from the purchase value.

Concrete

Preliminary design sketches were used to develop the concrete quantities to validate the cost factoring methodology.

Structural

Preliminary design sketches were used to develop the steel quantities to validate the cost factoring methodology.

Other Disciplines

Supply and installation costs of architectural works, (i.e. siding, roofing, doors, plant offices, etc.), HVAC (units and ducting), electrical distribution (e-rooms equipment, cable tray and power cables), and automation (instruments, control and communication cabling) were also factored from the supply cost of process equipment.

Construction Labour

Construction Labour Force was inferred from the factored installation cost and crew rates and productivity loss factors developed on a recent similar project in the same geographical area.

21.1.5.6 Tailings, Mine Waste and Water Management (Area 800)

The tailings, mine waste and water management capital costs for the Pine Point Project were estimated by WSP.

No tailings management facility will be built for the Pine Point Project. All process plant tailings will be piped to historical and mined out pits as they become available. Water needs for the process plant will be reclaimed from the pits once settling has occurred and a clear supernatant becomes available. The pre-production capital cost estimate includes all piping and pumps required for the first two years of operations.

The dewatering costs includes the drilling cost of dewatering wells, pumps, installation and piping. The selected well diameter has been set to 16 inches, which will allow submersible pumps of 150 hp and 400 hp to be installed. There will be several wells around each pit that will be connected to a network with booster pumps to draw down the groundwater table and reduce pit dewatering. Underground mines in the West Zone will be mainly dewatered inside the grouting ring installed at the perimeter of each deposit with the exception of the G03 deposit, which is not expected to be grouted. Underground mines in C1 Area will be dewatered from pit M67 once depleted and from the historical pit M64. Additional dewatering will be carried out during operations for the remaining infiltration water.

The pumps for the first open-pits will be purchased in Year -1 and whenever possible, pumps and piping will be transferred to other open-pits for the remaining years of the LOM, according to the mining sequence. For the purpose of the cost estimate, the established distance discharge point has been set to 5 km. When possible, mined out pits will be used for water discharge.

Site preparation and water collection systems surrounding the Waste Rock and Overburden stockpiles are included for six initial WR & OB storage facilities as well as a Mineral Sorter Rejects material pad. Contact water from the Process Plant Site and surplus process water from the

Process Plant will be collected and treated by a final effluent Water Treatment Plant ("WTP"). Existing site polishing ponds located north of the former town of Pine Point will be refurbished and used as the final stage of this treatment.

The total pre-production capital cost of tailings, mine waste and water management infrastructure for the Pine Point Project is estimated to be \$47.7M and is summarized in Table 21-9.

Table 21-9: Tailings, waste rock and water management infrastructure pre-production capital costs (Area 800)

Cost description	Total cost (\$M)	CAPEX (%)
Waste Management Infrastructure	13.8	29
Surface Water Management	4.9	10
Settling Pond Rehabilitation	2.2	5
Mine Dewatering Infrastructure	24.3	51
Contact Water Treatment Plant	2.5	5
Total	47.7	100

21.1.5.7 Direct Cost Summary (Area 000 to 800)

The overall pre-production capital direct costs (WBS Areas 000 to 800) for the Pine Point Project total \$488.9M.

21.1.5.8 Indirect Costs (Area 900)

Indirect costs for the Pine Point Project include all costs needed to carry out the engineering, procurement, and construction management services. These costs were calculated by Osisko Metals' estimating group and BBA. The main costs in this category are EPCM services, temporary facilities, third-party services, spare parts, freight, and customs.

For the Project's indirect costs included within the pre-production capital cost estimate, an itemized list of elements has been used to generate factored estimates. The following have been covered:

- EPCM;
- Costs associated with permitting and public consultations;
- Construction of temporary facilities both for erection and operation;
- Land and ocean freight for process and major electrical equipment;
- Pre-operational verifications;



- Commissioning support;
- Vendor representatives;
- Capital spares;
- Commissioning spares;
- First fills;
- Waste disposal;
- Sanitary blocks;
- Construction of temporary power.

The indirect costs were calculated using various sources of information, including the construction execution plan and information provided by Osisko Metals. Indirect costs, excluding Owner's costs (WBS 000) total \$76.6M and are summarized in Table 21-10.

Table 21-10 Indirect pre-production capital costs (Area 900)

Area	Cost description	Total cost (\$M)	CAPEX (%)
905	Construction Temporary Facilities	21.5	28
910	Construction Equipment / Maintenance and Tools	3.1	4
920	Permitting	2.8	4
925	Detailed Engineering	21.2	28
930	Construction Management	16.5	21
935	Freight	5.6	7
945	Initial Fill	1.3	2
950	Capital Spares	2.4	3
955	Commissioning	2.2	3
	Total	76.6	100

21.1.5.9 Contingency (Area 999)

Contingency is an allowance included in the pre-production capital cost estimate that is expected to be spent to cover unforeseeable items within the scope of the estimate. These can arise due to currently undefined items of work or equipment, or to uncertainty in the estimated quantities and unit prices for labour, equipment and materials. Contingency does not cover scope changes or project exclusions.

The pre-production cost contingency for the Project was calculated as a whole by Osisko Metals using a deterministic approach based on their experience, execution philosophy, historical data,



assessment of major risks/opportunities, level of project definition and advancement of engineering as well as contributions from the various firms according to their scope of work.

The total amount calculated for the pre-production contingency has been estimated at \$87.8M representing 5% of the mine mobile equipment costs and 15% of all other pre-production capital costs.

It is expected that in order to meet the budget for the Project, sufficiently developed engineering, adequate project management and tight construction cost controls will be implemented.

21.1.6 Sustaining Capital Costs

The sustaining capital costs incurred over the 12 years of production (Year 1 to Year 12) and closure for the Pine Point Project are estimated to total \$665.1M of project-related capital expenditures, including end-of-mine salvage value and site reclamation and closure costs. The breakdown of LOM sustaining capital expenditures by area is provided in Table 21-11 and Figure 21-3, while a detailed sustaining capital schedule is provided in Table 21-12. A contingency is \$23.7M for all sustaining capital costs apart from capitalized stripping costs.

Table 21-11: Project sustaining capital cost summary

Area	Description	Sustaining capital cost (\$M)	CAPEX (%)
200	Underground Mine	183.3	28
300	Open-pit Mine	80.6	12
400	Electricity and Communication	19.3	3
500	Site Infrastructure	11.8	2
800	Tailings, Mine Waste and Water Management	123.6	19
999	Contingency	23.7	4
	Capitalized stripping costs ¹	174.5	26
	Total	616.8	93
	Site Reclamation and Closure	68.0	10
	Salvage Value	-19.6	-3
	Total	665.1	100

1. The operating costs attributable to overburden and waste stripping that is in excess of the average strip ratio over the life of mine have been transferred to capital expenditures.

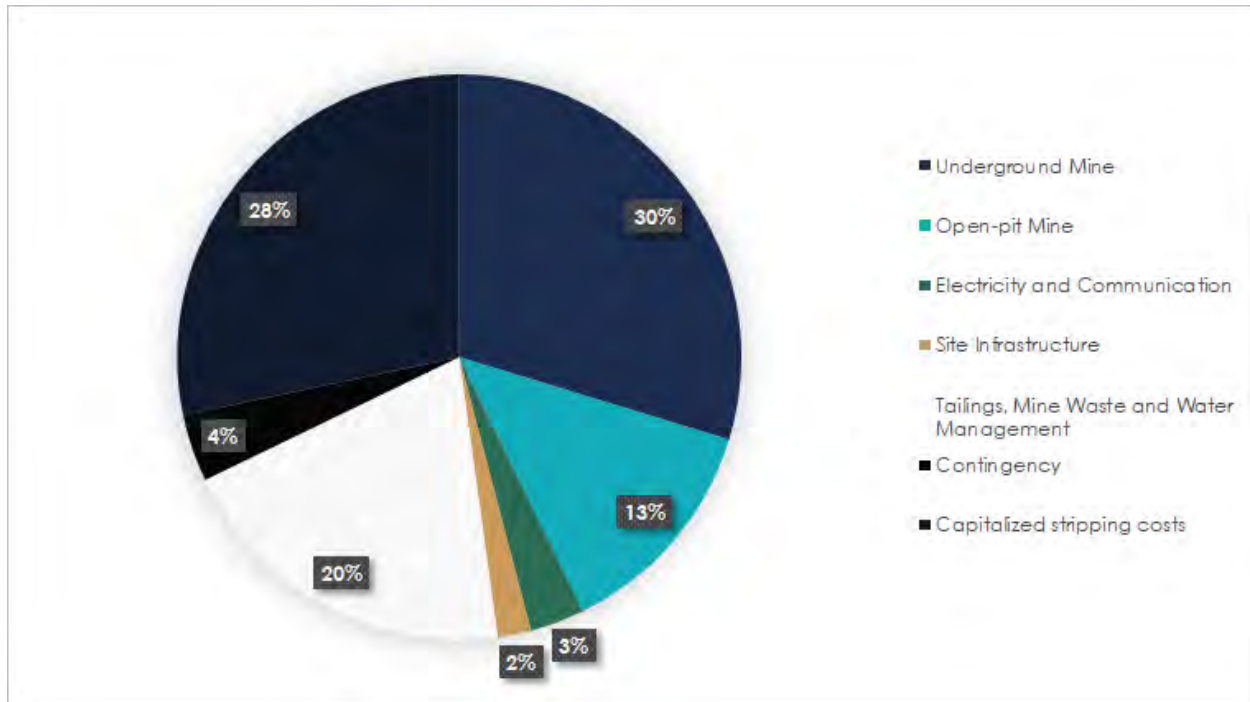


Figure 21-3 Project sustaining capital cost summary



Table 21-12: Sustaining capital cost breakdown

Area	Description	Year	1	2	3	4	5	6	7	8	9	10	11	12	13	Total	CAPEX (%)
		Sustaining Capital Cost (\$M)															
200	Underground Mine		41.0	36.7	39.5	8.0	8.4	10.3	28.6	10.7	0.1	0.0	0.0	0.0	0.0	183.3	28
300	Open-pit Mine		15.6	15.5	15.5	15.5	15.5	2.9	0.1	0.0	0.0	0.0	0.0	0.0	0.0	80.6	12
400	Electricity and Communication		10.6	0.8	1.1	0.7	0.9	1.0	0.7	0.9	0.7	1.0	0.5	0.5	0.0	19.3	3
500	Site Infrastructure		1.6	0.5	1.0	1.3	4.0	0.9	1.8	0.6	0.0	0.0	0.0	0.0	0.0	11.8	2
800	Tailings, Mine Waste and Water Management		25.6	13.3	14.6	34.5	2.0	3.7	9.0	2.2	11.9	4.4	2.3	0.0	0.0	123.6	19
999	Contingency		5.0	5.8	3.7	3.0	1.5	0.9	2.0	0.7	0.6	0.3	0.1	0.0	0.0	23.7	4
	Capitalized stripping costs		35.2	24.8	17.4	0.0	19.5	21.7	0.0	0.0	13.6	42.4	0.0	0.0	0.0	174.5	26
	Total		134.6	97.3	92.9	63.1	51.8	41.3	42.3	15.1	26.9	48.0	3.0	0.5	0.0	616.8	93
	Site Reclamation and Closure		0.0	0.0	0.0	8.7	0.0	0.0	14.4	0.0	0.0	0.0	0.0	0.0	44.9	68.0	10
	Salvage Value		0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	-19.6	-19.6	-3
	Total		134.6	97.3	92.9	71.7	51.8	41.3	56.7	15.1	26.9	48.0	3.0	0.5	25.3	665.1	100



21.1.6.1 Underground Mining (Area 200)

The largest cost area for sustaining capital costs is associated with underground mining operations at Pine Point. Mine development and operations will be performed by a qualified contractor. The underground mine related sustaining capital costs are estimated to be \$183.3M and are broken down by activity in Table 21-13. Significant sustaining capital is required as mining progresses. Sustaining capital includes drifts, ventilation raises, ramp extension, underground infrastructure, contractor development, and dewatering. No additional contingency is assumed to be included in these costs.

Sustaining capital cost of underground mines will start on Year 1. It is anticipated to take two years for each deposit to complete the following tasks that are included in the development costs:

- Grouting around mineralized deposit;
- Dewatering inside the grouting ring;
- Portal excavation;
- Decline development;
- Underground infrastructure excavation and construction;
- Development drift and stope preparation.

Table 21-13: Underground sustaining capital costs (Area 200)

Item	Total cost (\$M)	CAPEX (%)
Mine Development	90.0	49
Infrastructure	11.7	6
Ventilation	8.3	5
Water Management	65.0	35
Portal	7.9	4
Owner's Costs	0.6	0
Total	183.3	100

21.1.6.2 Open-pit Mining (Area 300)

The sustaining capital costs for open-pit mining are estimated to be \$80.6M and are broken down by category in Table 21-14. Significant sustaining capital for the open-pit mine includes the purchase of primary production equipment and ongoing lease payments. Open-pit dewatering costs are included in the water management cost area (Area 800).

Table 21-14: Open-pit mining sustaining capital costs (Area 300)

Item	Total cost (\$M)	CAPEX (%)
Major Equipment	70.0	87
Support Equipment	10.6	13
Total	80.6	100

21.1.6.3 Electricity and communication sustaining capital costs (Area 400)

The sustaining capital costs for the electricity and communication are estimated to be \$19.3M and are broken down by activity in Table 21-15.

Table 21-15: Electricity and communication sustaining capital costs (Area 400)

Item	Total cost (\$M)	CAPEX (%)
Liquified natural gas gensets	10.1	52
Low voltage lines and transformers	9.2	48
Total	19.3	100

21.1.6.4 Site Infrastructure (Area 500)

The sustaining capital costs for the surface facilities are estimated to be \$11.8M and are broken down by activity in Table 21-16. Significant sustaining capital for the site includes the construction of additional haul roads to support mining operations as well as the installation of a recreational facility in Year 1.

Table 21-16: Site infrastructure sustaining capital costs (Area 500)

Cost item	Total cost (\$M)	CAPEX (%)
Haul Roads	10.5	89
Recreational Facility	1.3	11
Total	11.8	100

21.1.6.5 Tailings, Mine Waste and Water Management (Area 800)

The sustaining capital costs for the tailings, mine waste rock piles and water management infrastructure were estimated by WSP. It is estimated to be \$123.6M as shown in Table 21-17.

Significant sustaining capital for this area includes the construction of seven overburden and waste rock piles, extensions to the tailings/water reclaim pipelines as well as drilling and equipment related to mine dewatering.

Site preparation (grubbing and landscaping) as well as the construction of ditches and collection ponds network are included in the costs of the overburden and waste rock infrastructure. Infrastructure construction is based on the mine plan and will generally be completed the year before mining begins.

Water reclaim equipment that recirculates process water back to the process plant that was purchased in the pre-production period will be re-used and adapted as mined out pits are filled with tailings over the life of the mine.

Dewatering pumps and pipelines will also be re-used and adapted to the needs of operations as mining moves from pit to pit.

Table 21-17: Tailings, mine waste and water management sustaining capital costs (Area 800)

Cost item	Total cost (\$M)	CAPEX (%)
Site preparation	17.8	14
Drainage ditches and water basin excavation	8.0	6
Dewatering drilling	1.6	1
Dewatering pumps	56.5	46
Dewatering pump installation	39.7	32
Total	123.6	100

21.1.6.6 Rehabilitation and Mine Closure Capital Costs

Reclamation and closure costs for the Pine Point Project was provided by WSP and estimated to total \$68M. A phased financial security of approximately the same amount will be required by the regulating authorities to obtain the water licences and land use permits for mine operations. The financial security required by the regulating authorities will be adjusted in accordance with Project phasing as agreed with regulators and completed progressive reclamation.

The mine closure cost estimate includes direct and indirect costs of site restoration as well as post-restoration monitoring. The cost estimate has been determined using the RECLAIM 7.0 model, as recommended by the GNWT and the MVLWB (MVLWB/AANDC, 2013), which includes a 25% contingency. These costs considered the preliminary geochemical assumptions that the tailings are non-potentially acid generating ("NPAG") and metal leaching while waste rock and mineral sorter rejects are NPAG and non-metal leaching.

Table 21-18 provides a breakdown of the costs associated with site rehabilitation and closure.

Table 21-18: Site rehabilitation and closure

Cost item	Total cost (\$M)	CAPEX (%)
Progressive site closure activities	23.1	34
Final site closure activities	44.9	66
Total	68.0	100

21.1.6.7 Salvage Value

The salvage value (sustaining capital credit) for the Pine Point Project was estimated to total \$19.5M. The salvage value was estimated for major mechanical and electrical equipment from the process plant. No salvage value was estimated for mobile equipment from the underground or open-pit operations. Salvage value ranged from 15% to 40% of original purchase cost depending on equipment type and application.

Table 21-19 provides a breakdown of the credit associated with the salvage value of equipment from the Pine Point Project.

Table 21-19: Salvage value

Cost Area Description	Total credit (\$M)	CAPEX (%)
Process Plant – Mechanical Equipment	17.2	88
Process Plant – Electrical Equipment	2.3	12
Total	19.6	100

21.2 Operating Costs

21.2.1 Summary

The average operating cost over the 12-year mine life of the Pine Point Project is estimated to be \$191M/y or \$48.9/t milled. Table 21-20 provides the breakdown of the mine site operating costs.

Table 21-20: Pine Point Project operating cost summary

Cost area	LOM (\$M)	Annual average cost (\$M)	Average LOM (\$/tonne milled)	OPEX (%)
Open-pit Mining ⁽¹⁾	843.1	70.3	17.98	37
Underground Mining (West and Central Zones) ⁽²⁾	376.4	31.4	8.03	16
Processing	575.3	47.9	12.27	25
Power Generation and Supply	216.3	18.0	4.61	9
Tailings, Waste and Water Management	76.4	6.4	1.63	3
General and Administration	380.6	31.7	8.11	17
Capitalized Operating Costs ³	-174.5	-14.5	-3.72	-8
Total (Mine Site)	2,293.6	191.1	48.91	100

1. Open-pit mining = \$3.36 /t mined
2. West Zone underground mining (Years 3 to 8) = \$40.10 /t mined, Central Zone underground mining (Years 8 to 10) = \$52.07 /t mined
3. The operating costs attributable to overburden and waste stripping that are in excess of the average strip ratio over the life of mine have been transferred to capital expenditures.

Basis of Operating Cost Estimate

The operating cost estimate was based on Q2 2022 assumptions and pricing. The estimate has an accuracy of $\pm 30\%$. All operating cost estimates are in CAD. Estimates are based on budgetary quotations, in-house data and typical allowances. Salaries are based on Osisko Metals' projected salary chart.

The operating cost estimate is based on the open-pit and underground mine production schedule produced by WSP and inclusive of all on-site costs to final Project closure (LOM). Costs up to and including C4 commissioning (completion of wet commissioning) are excluded from operating costs and are included in the capital cost estimate as summarized in Section 21.1.

Assumptions and Exclusions

The following items were assumed:

- All equipment and materials will be new;
- The labour rate build-up will be based on the statutory laws governing benefits to workers that were in effect at the time of the estimate;
- There will be a mix of local and fly-in / fly-out ("FIFO") roles;

- All employee transport, operations camp and catering costs are included in the G&A cost category;
- Freight estimates are based on vendor supplied freight quotations or in-house data. Freight for reagents and steel consumables is included in the price of those commodities. Freight for spare parts is calculated as a percentage of equipment costs expected to be used annually;
- All underground mining will be performed by contractors;
- Open-pit mining costs are estimated on an owner operated basis except for blasting and the drilling of dewatering wells, which will be performed by specialized contractors.

The following items were specifically excluded from the operating cost estimate:

- Contingency;
- Cost escalation or de-escalation;
- Cost of financing and interest;
- Commissioning and start-up;
- Pre-operations and maintenance training;
- Transport and handling of concentrate from the process plant. A description of transport costs is provided in the Market Studies chapter (Chapter 19) and included within the Financial Analysis chapter (Chapter 22).

Estimate Responsibilities

The operating cost estimate combined inputs from a number of sources including BBA, WSP and Osisko Metals as summarized in Table 21-21.

Table 21-21: OPEX estimate responsibilities

Cost area	Responsible entity
Open-pit Mining	WSP
Underground Mining (West and Central Zones)	WSP
Processing	BBA
Power Generation and Supply	BBA
Tailings, Waste and Water Management	WSP
General and Administration	BBA and Osisko Metals
Capitalized Operating Costs	BBA and Osisko Metals

General Unit Rates

General rates used in the operating cost estimate are summarized in Table 21-22.

Table 21-22: General rate and unit cost assumptions

Parameter	Unit	Value
Average Daily Processing Tonnage	tpd	11,250
Years of Operations	years	12
Open-pit - Mineralized Material	Mt	37.9
Underground West Zone - Mineralized Material	Mt	7.6
Underground Central Zone - Mineralized Material	Mt	1.3
Total LOM - Mineralized Material	Mt	46.9
LOM Lead Grade	%	1.76
LOM Zinc Grade	%	4.39
LOM Zinc Equivalent Grade	%	5.87
Grid Power	\$/kWh	0.145
Liquified Natural Gas	\$/GJ	18.15
Diesel Fuel	\$/L	1.41

21.2.2 Mining

Overview

WSP provided estimates for all open-pit and underground mine operating costs. The overall mining operating cost is \$1,219.6M over the life of mine. The mining unit cost per tonne of mineralized material is \$22.20/t for open-pit mining, \$40.10/t for underground mining of the West Zone and \$52.10/t for underground mining of the Central Zone.

Table 21-23: Mining operating cost summary

Cost Area / Year	Mineralized material mined (Mt)	LOM total (\$M)	Average LOM (\$M/year)	Average LOM (\$/t of mineralized material mined)	OPEX (%)
Open-pit Mining ⁽¹⁾	37.9	843.1	70.3	22.24	69
Underground Mining- West Zone	7.6	306.4	25.5	40.10	25
Underground Mining- Central Zone	1.3	70.1	5.8	52.07	6
Total	46.9	1,219.6	101.6	26.01	100

1. Inclusive of mineralized material, overburden and waste rock, LOM average = \$3.36 /t mined.

Table 21-27, Table 21-28 and Table 21-29 represent the operating costs by year for open-pit mining and underground mining of both the West Zone and Central Zone. The next three sections provide the basis of estimate for how the open-pit and underground operating costs were developed.

Open-pit Mining

Table 21-24 summarizes the open-pit mining operating costs and provides a breakdown per item. The LOM costs are based on mining a total of 250.7 Mt including 37.9 Mt of mineralized material, 137.7 Mt of waste rock and 75.1 Mt of overburden. The LOM average unit cost is \$3.36/t inclusive of mineralized material, waste rock and overburden.

Table 21-24: Open-pit mining cost summary

Cost Area / Year	LOM Total (\$M)	Annual (\$M/year)	Average LOM (\$/tonne mined)	OPEX (%)
Loading and Hauling	363.3	30.3	9.58	43
Rehandling	6.7	0.6	0.18	1
Auxiliary Equipment	132.8	11.1	3.50	16
Workforce	136.7	11.4	3.61	16
Drill and Blast	187.4	15.6	4.94	22
Aggregates	14.4	1.2	0.38	2
Equipment Rental	1.9	0.2	0.05	0
Total	843.1	70.3	22.24	100

Drill and blast costs are based on drill productivity and powder factor. Drilling cost is calculated from the number of hours and the hourly costs from the WSP cost database. The blasting cost estimate is based on an explosive's service supplier quotation.

Loading costs are estimated based on hydraulic shovel and hydraulic excavator productivity to complete the task of the loading of the trucks.

Hauling costs were calculated using the cycle time from each open-pit to the different destination locations (overburden and waste rock stockpiles or to the process plant). Availability and productivity factors were applied to assess the number of hours and trucks required. These factors include mechanical availability, seasonal downtime, shift delay, and operational efficiency. From these parameters, operating hours to complete each operational process were calculated. Based on 100 t class trucks (90 metric tonnes capacity), typical costs were applied and include:

- Overhaul parts and labour costs;
- Regular maintenance parts and labour;

- Fuel and lubricant consumptions;
- Tires;
- Operator salaries.

The costs are based on a combination of information from databases and suppliers.

A small, dedicated fleet has been added to the operation to transport the mineral sorter rejects.

Auxiliary equipment costs were calculated with the hourly costs coming from Costmine and the number of hours required for each piece of equipment.

An allowance for aggregates has been added for a total of \$1.2M per year.

Workforce costs excludes operators that are included in equipment costs. What is included in the workforce costs are the following services:

- Mine management;
- Pumping and blasting team;
- Maintenance management and labourers;
- Technical services.

The total yearly average number of employees for open-pit operations is 285.

Underground Mining

Table 21-25 and Table 21-26 summarize the underground mining operating costs for the West Zone and Central Zone. Underground mining will be performed by a reputable mining contractor.

Table 21-25: Underground mining operating costs - West Zone

Operating costs		Total LOM cost	Average LOM cost	OPEX
Activity	Sub-activity	(\$M)	(\$/t mined)	(%)
Long Hole Mining ⁽¹⁾	Definition drilling	5.6	0.74	2
	Mine Development OPEX	101.9	13.34	33
	Production - Stope Preparation	19.4	2.54	6
	Production - Drilling & Blasting	23.9	3.12	8
	Production - Mucking & Hauling	51.4	6.73	17
	Production - Backfill	46.1	6.03	15
	Services	11.9	1.55	4
	Maintenance	1.3	0.17	0
	Energy Cost - Propane	10.3	1.35	3
	Transportation cost from UG mine to mill	34.6	4.52	11
	Total	306.4	40.10	100

⁽¹⁾ Represents 94% of the total tonnage in the stopes.

Development cost is on a metre basis while drilling and blasting is calculated per tonne of mineralized material. Mucking and hauling has been calculated also based on contractor cost per tonne from information provided by WSP.

Backfilling is from a supplier's quotation and the energy cost is only for propane required to heat air pushed underground. All electrical costs are covered by the power generation cost category (Section 21.2.4)

Transportation cost represents the cost to send the mineralized material from the West Zone to the process plant.

The total yearly average number of employees for the underground mine area is 41 (29 miners and 12 Staff).

Table 21-26: Underground mining operating costs - Central Zone

Cost Area	LOM total (\$M)	Average LOM (\$M/year)	Average LOM (\$/t of mineralized material mined)	OPEX (%)
Definition drilling	1.4	0.1	1.01	2
UG Maintenance	0.9	0.1	0.69	1
UG Services	5.1	0.4	3.82	7
UG Energy Cost	1.0	0.1	0.74	1
Production - Development RP	43.5	3.6	32.31	62
Production - Mucking & Hauling RP	7.5	0.6	5.59	11
Development OPEX LH	4.2	0.3	3.10	6
Production - Stope Preparation LH	0.8	0.1	0.61	1
Production - Drilling & Blasting LH	0.9	0.1	0.68	1
Production - Mucking & Hauling LH	2.0	0.2	1.48	3
Production - Backfill LH	1.9	0.2	1.44	3
Transportation cost from UG mine to mill	0.8	0.1	0.60	1
Total	70.1	5.8	52.07	100

The methodology for cost calculation of the Central Zone operating costs is the same as the West Zone. However, this will be mainly mined by room and pillar methods with some uppers used to remove mineralization in the back, and the access will be through portals from mined-out pits, resulting in different average unit costs. It is assumed that the same contractor will be mining the Central Zone after the completion of the West Zone.



Table 21-27: Open-pit mine operating costs per year

Operating costs per year (\$M)		1	2	3	4	5	6	7	8	9	10	11	12	Total cost
Activity	Sub-activity	(\$M)	(\$M)	(\$M)	(\$M)	(\$M)	(\$M)	(\$M)	(\$M)	(\$M)	(\$M)	(\$M)	(\$M)	(\$M)
Open-pit Mining	Loading and Hauling	45.8	48.9	39.2	31.6	32.8	32.4	29.6	31.1	21.6	20.6	24.4	5.3	363.3
	Rehandling	0.2	0.0	0.1	1.0	0.0	0.0	0.4	0.8	0.7	1.4	0.6	1.5	6.7
	Auxiliary Equipment	8.5	10.9	10.9	11.4	11.4	11.4	11.4	11.4	11.4	11.4	11.4	11.4	132.8
	Workforce	11.0	11.2	11.3	11.3	11.2	11.2	11.2	11.2	11.2	11.2	12.4	12.4	136.7
	Drill and Blast	25.3	24.2	22.4	12.2	10.8	10.2	15.5	14.4	12.9	16.5	17.5	5.5	187.4
	Aggregates	1.2	1.2	1.2	1.2	1.2	1.2	1.2	1.2	1.2	1.2	1.2	1.2	14.4
	Equipment Rental	0.0	1.0	0.0	0.0	0.0	0.0	0.0	0.0	1.0	0.0	0.0	0.0	1.9
	Total	92.0	97.3	85.0	68.8	67.4	66.5	69.2	70.0	59.9	62.2	67.5	37.3	843.1





Table 21-28: Underground mine operating costs per year - West Zone

Operating costs per year (\$M)		1	2	3	4	5	6	7	8	9	10	11	12	Total cost
Activity	Sub-activity	(\$M)	(\$M)	(\$M)	(\$M)	(\$M)	(\$M)	(\$M)	(\$M)	(\$M)	(\$M)	(C\$M)	(C\$M)	(\$M)
Underground Mining (West Zone)	Definition Drilling	0.0	0.0	0.0	0.6	1.2	1.5	1.5	0.8					5.6
	Mine Development OPEX	6.2	20.3	13.3	20.5	13.0	8.5	13.9	6.3					101.9
	Production - Stope Preparation	0.0	0.0	1.6	3.1	4.2	4.2	4.2	2.2					19.4
	Production - Drilling and Blasting	0.0	0.0	1.9	3.9	5.1	5.1	5.1	2.7					23.9
	Production - Mucking and Hauling	0.0	0.0	4.2	8.3	11.1	11.1	11.1	5.7					51.4
	Production - Backfill	0.0	0.0	3.7	7.5	9.9	9.9	9.9	5.1					46.1
	UG Services	1.1	1.6	1.6	1.6	1.6	1.6	1.6	0.9					11.9
	UG Maintenance	0.0	0.0	0.3	0.1	0.2	0.2	0.3	0.1					1.3
	UG Energy Cost	0.0	0.0	0.0	0.5	1.4	1.4	0.7	6.4					10.3
	Transportation Cost	0.0	0.0	2.8	5.6	7.4	7.4	7.4	3.9					34.6
	Total	7.3	21.9	29.4	51.8	55.1	51.0	55.8	34.0					306.4

Table 21-29: Underground mine operating costs per year - Central Zone

Operating costs per year (\$M)		1	2	3	4	5	6	7	8	9	10	11	12	Total cost
Activity	Sub-activity	(\$M)	(\$M)	(\$M)	(\$M)	(\$M)	(\$M)	(\$M)	(\$M)	(\$M)	(\$M)	(\$M)	(\$M)	(\$M)
Underground Mining (Central Zone)	Definition Drilling	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.2	0.8	0.4			1.4
	UG Maintenance	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.4	0.4	0.2			0.9
	UG Services	0.0	0.0	0.0	0.0	0.0	0.0	0.9	1.6	1.6	0.9			5.1
	UG Energy Cost	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.1	0.6	0.3			1.0
	Room and Pillar OPEX	0.0	0.0	0.0	0.0	0.0	0.0	0.0	8.0	27.8	15.1			51.0
	Longhole OPEX	0.0	0.0	0.0	0.0	0.0	0.0	3.4	1.1	3.9	1.4			9.8
	Transportation Cost	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.1	0.5	0.2			0.8
	Total	0.0	0.0	0.0	0.0	0.0	0.0	4.3	11.6	35.5	18.6			70.1



21.2.3 Process Plant

The average process plant operating costs were calculated over the LOM. The annual operating cost was estimated to be \$47.9M or \$12.27 per tonne processed.

The steady-state operating costs include reagents, equipment consumables and maintenance, grinding media, personnel (including contractors) as well as laboratory assays and an allowance for special projects for plant optimization. Excluded are the electrical power costs for the process plant which are included in Section 21.2.4 and concentrate transport costs as described in Chapter 19. The process consumables include grinding media, screen panels, filter press cloths as well as mill and crusher liners. A breakdown of the steady-state process plant operating costs, without contingency, is presented in Table 21-30.

Table 21-30: Process plant operating costs

Cost area	LOM cost (\$M)	Average annual cost (\$M)	Cost per tonne milled (\$/t)	OPEX (%)
Reagents	252.0	21.0	5.37	44
Maintenance and Parts	98.8	8.2	2.11	17
Major Equipment and Consumables	54.1	4.5	1.15	9
Grinding Media	17.8	1.5	0.38	3
Personnel	116.4	9.7	2.48	20
Utilities	28.8	2.4	0.61	5
Miscellaneous	7.3	0.6	0.16	1
Total	575.3	47.9	12.27	100

Reagents

Numerous reagents are required for the Pine Point process flowsheet to operate the flotation circuits as well as the thickeners. The reagent consumptions were estimated based on testwork results, industrial references and literature, and are presented in Chapter 17.

Budgetary prices, including delivery to site, were obtained for all reagents.

A summary of the average annual cost for each of the reagents is presented in Table 21-31.

Table 21-31: Average reagent costs

Reagent	LOM cost (\$M)	Average annual cost (\$M)	Cost per tonne milled (\$/t)	OPEX (%)
Zinc sulphate (ZnSO_4)	60.4	5.0	1.29	24
Cytec 3894	8.7	0.7	0.19	3
Aerofloat 3418A	9.3	0.8	0.20	4
MIBC	6.0	0.5	0.13	2
CMC	136.3	11.4	2.91	54
Copper sulphate ($\text{CuSO}_4 \cdot 5\text{H}_2\text{O}$)	17.6	1.5	0.38	7
Lime	13.6	1.1	0.29	5
Flocculant	0.0	0.0	0.00	0
Total	252.0	21.0	5.37	100

The average annual cost of reagents was calculated to be \$21.0M, or \$5.37 per tonne milled. Nearly 54% of the reagent costs are for CMC alone, while 24% are for zinc sulphate.

Personnel

A total of 66 workers are required in the process plant, including 19 salaried staff and 47 hourly workers divided amongst management and technical services, operations and maintenance departments. The list of personnel, along with the salaries and benefits, including bonuses where applicable, associated with each position was provided by Osisko Metals. The personnel costs including an allowance for maintenance contractors was estimated at an average cost of \$9.7M per year or \$2.50 per tonne processed.

Equipment Consumables, Spares and Maintenance

The replacement costs of major equipment consumables such as the crusher and ball mill liners, the screen panels and the filter press cloths were calculated based on recommended change-out schedules and budgetary quotations and using BBA's internal database.

The general maintenance of the process plant equipment was calculated by applying fixed percentages to the indicated capital cost of a given area. The processing areas and percentages applied to the mechanical, electrical and instrumentation equipment, as well as the piping materials in the process plant, are presented in Table 21-32.

Table 21-32: Process plant maintenance costs by area

Process plant equipment	Percentage of capital costs applied (%)
Mechanical	
■ Crushing, Conveying and Sorting	9%
■ Grinding	7%
■ Flotation, Reagents and Concentrate Handling	3%
■ Tailings Dewatering and Pumping	3%
Plant Services	3%
Piping	10%
Electrical and Communication	2%

The average annual maintenance costs were calculated to be \$8.2M or \$2.10 per tonne milled, including an allowance of \$1.1M per year for mobile equipment fuel and maintenance costs.

Grinding Media

The Pine Point process flowsheet includes single size of steel media for the ball mill. The consumption rate for the 2.5" media was calculated using MolyCop tools and the Bond method. The input data considered the average operating conditions for the ball mill, in terms of power draw, rotational speed and media loading. Budgetary quotations were obtained for the type of media used. The estimated annual media consumption rate is presented in Table 21-33.

Table 21-33: Media consumption rate

Area	Type	Size (in)	Consumption (tpy)
Ball Mill	Forged steel	2.5	700

The average annual cost of media was estimated to be \$1.5M or \$0.40 per tonne processed, which represents 3% of the process plant operating costs.

Miscellaneous

The miscellaneous costs for the process plant include items such as special projects and assays. An analytical laboratory located in Hay River will perform the analysis of the solid, slurry and liquid samples collected from the process plant and mine operations. The analysis will be used for metallurgical accounting, on-stream analyzer calibration or development testwork as well as for environmental impact monitoring and mine plan development.

The samples collected include slurry samples from various stages of the flowsheet; as well as solid and liquid samples from the mine site. Based on the estimated sample collection frequency and the unit cost from BBA's internal database of similar projects, an annual cost of \$0.4M was considered for assay consumables.

An allowance of \$0.2M per year was made for special projects realized to enhance the metallurgical performance or maintainability of the processing facility.

21.2.4 Power Generation and Supply

The annual average power operating costs for the entire project were determined to be \$18.0M per year or \$4.61 per tonne milled including grid power, LNG power generation equipment rental and natural gas.

Supplementary power is required by the Pine Point Project due to capacity limitations of the existing power grid, which varies according to the season. Between October and March (6 months/year during winter period), approximately 7.3 MW is expected to be available for use by Pine Point, outside of these months it is anticipated that 10.0 MW will be available (see Chapter 18 for more details). In addition to a 2.6MW of installed solar photovoltaic, electrical power will be generated by a mix of rented and procured mobile generators (natural gas powered) for open-pit and underground mine dewatering and ventilation. The number of generators and their locations vary over the life of the mine due to variations in dewatering and ventilation requirements. Based on the mine production schedule, ventilation and dewatering requirements the underground power demand ranges from 350 kW to 5,200 kW while the open-pit dewatering demand will vary from 950 kW to 7,500 kW.

Budgetary quotations for both equipment rental and Liquefied Natural Gas fuel were obtained from reputable regional suppliers and used as the basis of the operating cost estimate. The employee total for the Power generation and supply group is eight.

This area includes the following operating costs:

- Power sourced from the existing grid.
- Mobile Generator Rental (on demand basis);
- Fuel Supply, Transformation and Storage;
- Equipment Mobilization and Demobilization Fees;
- Labour costs for operations and maintenance of the generators and on-site electrical network;
- Electrical supplies;



A breakdown of the power operating costs is presented in Table 21-34.

Table 21-34: Power Generation and Supply operating cost summary

Cost Area	LOM Total (\$M)	Average LOM (\$M/year)	Average LOM (\$/tonne milled)	OPEX (%)
Equipment Rental	10.1	0.8	0.22	5
Mobilization and Demobilization	7.5	0.6	0.16	3
Energy (LNG)	59.2	4.9	1.26	27
Grid Power	128.1	10.7	2.73	59
Labour	11.4	1.0	0.24	5
Total	216.3	18.0	4.61	100

21.2.5 Tailings, Waste and Water Management

The annual average operating costs for tailings, waste and water management were determined to be \$6.4M per year or \$1.63 per tonne milled. Water management operating costs include WTP operating all-inclusive costs based on treatment flowrate, as well as ditch and pond maintenance which are based on ditch length and pond capacity. WTP operating costs will be detailed in further studies, once a detailed WTP plant has been designed.

Labour costs includes salary and benefits for the workforce required for all waste-related operations such as infrastructure construction, maintenance, monitoring, etc. The employee total for the Tailings, Waste and Water Management group is 18.

Engineering, monitoring, and reporting are estimated to represent 10% of tailings, waste, and water management OPEX and include water sampling and monitoring, geotechnical studies, monitoring, and reporting, infrastructure design, etc.

A breakdown of the tailings, waste and water management operating costs is presented in Table 21-35.

Table 21-35: Tailings, Waste and Water Management operating cost summary

Cost Area	LOM Total (\$M)	Average LOM (\$M/year)	Average LOM (\$/tonne milled)	OPEX (%)
Water Management	44.6	3.7	0.95	58
Labour	24.9	2.1	0.53	33
Engineering, Monitoring and Reporting	6.9	0.6	0.15	9
Total	76.4	6.4	1.63	100

21.2.6 General and Administration

General and Administrative ("G&A") costs are expenses not directly related to the production of goods and encompass items not included in the mining, processing, and transportation costs of the Project. These costs were developed based on the Osisko Metals' past project experience, similar sized operations, and BBA's in-house database.

The G&A area includes the following items:

- Site administration and management labour;
- Human Resources, Information Technology ("IT") and Health Services labour;
- Office furniture and supplies;
- Computer hardware and software costs/license fees;
- Camp Operations (accommodations and food for 230 persons);
- Transportation costs for fly-in / fly-out staff and employees;
- Infrastructure heating;
- Health and Safety supplies;
- Building Insurance (including loss of production);
- Security, maintenance, cafeteria, laundry, snow removal and janitorial service contracts;
- Warehouse administration and supplies;
- Waste collection and recycling services;
- Telecommunications and data service fees;
- Training.

Employment roles included in the General and Administration Area will be split between the Process Plant Site and the offices located in Pine Point Junction. The employee total for the overall General and Administration services group is 65.

On an annual basis, the General and Administration costs are estimated to be \$32.5M/y or approximately \$380.6M over the mine's planned 13 years of operations. The G&A cost on a per tonne milled basis is \$8.11/tonne milled (LOM).

The major costs broken down by item within the General and Administration category are shown in Table 21-36. The largest cost within the G&A category is operating the operations camp and cafeteria, representing approximately 25% (\$8.1M/y), while management and administration salaries is the second greatest cost accounting for approximately 21% (\$6.7M/y). Employee transport represents the 3rd largest G&A cost for the Project at 13% (\$4.1M/y).



Table 21-36: Average general and administrative costs

Item	LOM Total (\$M)	Average LOM (\$M/year)	Average LOM (\$/tonne milled)	OPEX (%)
General Management and Administration Salaries	78.4	6.7	1.67	21
Insurance, Taxes and Fees	24.3	2.1	0.52	6
Information and Technology	18.6	1.6	0.40	5
Accounting and Finance	2.6	0.2	0.06	1
Engineering and Projects	2.8	0.2	0.06	1
Community Relations	8.0	0.7	0.17	2
Supply Chain and Warehouse	3.4	0.3	0.07	1
Employee Transport (FIFO)	48.4	4.1	1.03	13
Freight, Handling and Duties	14.8	1.3	0.32	4
Camp Operations (accommodation and food)	94.9	8.1	2.02	25
Health and Safety	9.6	0.8	0.20	3
Environment	17.1	1.5	0.36	4
Security Services	9.3	0.8	0.20	2
Human Resources	8.8	0.8	0.19	2
Organizational Development	3.4	0.3	0.07	1
Road Maintenance	7.0	0.6	0.15	2
Buildings Maintenance	9.7	0.8	0.21	3
Surface Services	11.2	1.0	0.24	3
Power and Heating	2.5	0.2	0.05	1
Mobile Equipment Fuel and Maintenance	5.7	0.5	0.12	2
Total	380.6	32.5	8.11	100

21.2.7 Site Personnel Summary – All Areas

On average over the life of mine, a total workforce of 485 employees is estimated for the Pine Point Project. A breakdown of positions by area is shown in Table 21-37. Local based employees are expected to total 153 while 332 positions are expected to be filled on a fly-in/fly out basis (2 weeks on / 2 weeks off rotation). Included in the total workforce estimate are 36 contract positions mostly for underground mining and power generation maintenance.



Table 21-37: Summary of personnel – All areas

Facility Area	Role	FIFO	Local	Total
General & Administration	Management	0	3	3
	Administration	2	17	19
	Human Resources and Community Relations	3	0	3
	Health and Safety	2	4	6
	Environment	4	0	4
	Site Services	10	20	30
	Subtotal	21	44	65
Open Pit Mining	Staff and Supervision	31	0	31
	Operations	113	53	166
	Maintenance	41	18	59
	Technical Services (Mine and Geology)	21	0	21
	Dewatering	5	3	8
	Subtotal	211	74	285*
Underground Mining	Operations (Contractor)	29	0	29
	Technical Services (Mine and Geology)	10	2	12
	Subtotal	39	2	41
Power Generation and Supply	Staff and Supervision	2	1	3
	Operations and Maintenance (Contractor)	6	1	7
	Subtotal	8	2	10
Process Plant	Staff and Supervision	16	3	19
	Operations	16	12	28
	Maintenance	8	8	16
	Laboratory	3	0	3
	Subtotal	43	23	66
Water and Waste Management	Staff and Supervision	6	2	8
	Operations and Maintenance	4	6	10
	Subtotal	10	8	18
Overall Project		332	153	485
Contractors		35	1	36
Pine Point Employees		297	152	449

*Open Pit workforce was calculated using an average of Years 2 to 11

22. ECONOMIC ANALYSIS

22.1 Overview

The economic/financial assessment of the Pine Point Lead Zinc Project for Osisko Metals was carried out using a discounted cash flow approach on a pre-tax and after-tax basis, based on consensus equity research long-term commodity price projections in United States currency and cost estimates in Canadian currency. An exchange rate of 0.79 USD per 1.00 CAD was assumed to convert USD market price projections and particular components of the capital cost estimates into Canadian Dollars ("CAD"). No provision was made for the effects of inflation. Current Canadian tax regulations were applied to assess the corporate tax liabilities, while the most recent territorial regulations were applied to assess the Northwest Territories' tax liabilities. Mining royalties were also included as stipulated by the Northwest Territories Lands Act, as well as carbon taxes as stipulated by the Northwest Territories Petroleum Products and Carbon Tax Act.

The internal rate of return ("IRR") on total investment was calculated based on 100% equity financing, even though Osisko Metals may decide in the future to finance part of the Project with debt financing. The net present value ("NPV") was calculated from the cash flow generated by the Project, based on a discount rate of 8%. The payback period, based on the undiscounted annual cash flow of the Project, is also indicated as a financial measure. Furthermore, a sensitivity analysis has been performed for the after-tax base case to assess the impact of variations in commodity prices, exchange rates, grades, initial capital costs and operating costs on NPV and IRR.

22.2 Cautionary Statement

Certain information and statements contained in this section and in the Report are "forward looking" in nature. Forward-looking statements include, but are not limited to, statements with respect to the economic and study parameters of the Project; Mineral Resource estimates; the cost and timing of any development of the Project; the proposed mine plan and mining methods; dilution and extraction recoveries; processing method and rates and production rates; projected metallurgical recovery rates; infrastructure requirements; capital, operating and sustaining cost estimates; the projected life of mine and other expected attributes of the Project; the net present value (NPV) and internal rate of return (IRR) and payback period of capital; capital; future metal prices; the timing of the environmental assessment process; changes to the Project configuration that may be requested as a result of stakeholder or government input to the environmental assessment process; government regulations and permitting timelines; estimates of reclamation

obligations; requirements for additional capital; environmental risks; and general business and economic conditions.

All forward-looking statements in this Report are necessarily based on opinions and estimates made as of the date such statements are made and are subject to important risk factors and uncertainties, many of which cannot be controlled or predicted.

Material assumptions regarding forward-looking statements are discussed in this Report, where applicable. In addition to, and subject to, such specific assumptions discussed in more detail elsewhere in this Report, the forward-looking statements in this Report are subject to the following assumptions:

- There being no significant disruptions affecting the development and operation of the Project;
- The availability of certain consumables and services and the prices for power and other key supplies being approximately consistent with assumptions in the Report;
- Labor and materials costs being approximately consistent with assumptions in the Report;
- The timelines for prior consultation and baseline data collection being generally consistent with PEA assumptions, and permitting and arrangements with stakeholders being consistent with current expectations as outlined in the Report;
- All environmental approvals, required permits, licenses and authorizations will be obtained from the relevant governments and other relevant stakeholders;
- Certain tax rates and royalties, including the allocation of certain tax attributes, being applicable to the Project;
- The availability of financing for Osisko Metals' planned development activities;
- The timelines for exploration and development activities on the Project;
- Assumptions made in Mineral Resource estimate and the financial analysis based on that estimate, including, but not limited to, geological interpretation, grades, commodity price assumptions, extraction and mining recovery rates, geotechnical, hydrological and hydrogeological assumptions, capital and operating cost estimates, and general marketing, political, business and economic conditions.

The production schedules and financial analysis annualized cash flow table are presented with conceptual years shown. Years shown in these tables are for illustrative purposes only. If additional mining, technical, and engineering studies are conducted, these may alter the Project assumptions as discussed in this Report and may result in changes to the calendar timelines presented.

The preliminary economic analysis is partly based on Inferred Mineral Resources that are considered too speculative geologically to have the economic considerations applied to them that would enable them to be categorized as Mineral Reserves, and there is no certainty that the PEA based on these Mineral Resources will be realized.

22.3 Assumptions and Basis

The economic analysis was performed using the following assumptions and basis:

- The Project Executive Schedule developed in Chapter 24, taking into consideration key Project milestones;
- Commercial production start-up is scheduled to begin in the second quarter ("Q2") of Year 1 (2028). The first full year of production is therefore Year 2. Operations are estimated to span a period of approximately twelve years;
- The base case lead and zinc prices are 0.96 USD/lb, and 1.37 USD/lb respectively;
- The long-term prices of lead and zinc were estimated on the basis of the average monthly price over the trailing two years (June 2020 to May 2022). The forecasts used are meant to reflect the average metal price expectation over the life of the Project. No price inflation or escalation factors were taken into account. It is understood that commodity prices can be volatile and that there is the potential for deviation from the LOM forecasts.
- The United States to Canadian dollar exchange rate has been assumed to be 0.79 USD : 1.00 CAD over the life of mine (CAD:USD exchange rate of 1.27);
- All cost estimates are in constant Q2 2022 Canadian dollars with no inflation or escalation factors taken into account;
- For a given year, the operating costs attributable to overburden and waste stripping that is in excess of the average strip ratio for the life of mine have been transferred to capital expenditures. These capitalized operating costs amount to \$174.5M over the life of mine;
- All concentrates and metals are assumed sold in the same year they are produced;
- Class specific Capital Cost Allowance rates are used for the purpose of determining the allowable taxable income;
- Final rehabilitation and closure costs are assumed to be incurred in 2040 (Year 13);
- Project revenue is derived from the sale of concentrates into the international marketplace. No contractual arrangements for concentrate smelting or refining exist at this time; however, preliminary market studies on the potential concentrate sales were completed to



provide an indication of transportation costs and market smelter terms, as noted in Chapter 19.

- Osisko Gold Royalties Ltd ("OGR") retains a 3.0% net smelter return royalty on all metals produced.

This financial analysis was performed on both a pre-tax basis and after-tax basis with the assistance of an external tax consultant. The general assumptions used for this financial model, LOM plan tonnage and grade estimates are summarized in Table 22-1, and are outlined in Table 22-2.

Table 22-1: Financial model parameters

Description	Unit	Value
Long Term Lead Price	USD/lb	0.96
Long Term Zinc Price	USD/lb	1.37
Exchange Rate	USD:CAD	0.79
Discount Rate	%	8%
Mine Life	years	12
Total Material Processed (LOM)	Mt	46.90
Lead Grade (LOM)	%	1.76
Zinc Grade (LOM)	%	4.39
ZnEq Grade (LOM)	%	5.87
Overall Lead Recovery (LOM)	%	92.9
Overall Zinc Recovery (LOM)	%	87.0
Total Lead Production (in concentrate)	Mlb	1,687
Total Zinc Production (in concentrate)	Mlb	3,951
Total Zinc Equivalent Production (in concentrate)	Mlb	5,366
Average Annual Lead Production (in concentrate)	Mlb per year	141
Average Annual Zinc Production (in concentrate)	Mlb per year	329
Open-pit Mining Operating Cost	\$/t mined	3.36
Underground Mining Operating Cost (West Zone)	\$/t mined	40.10
Underground Mining Operating Cost (Central Zone)	\$/t mined	52.07
Processing Operating Cost	\$/t processed	12.27
Power Production Operating Cost	\$/t processed	4.61
Tailings, Waste & Water Management Operating Cost	\$/t processed	1.63
General and Administration Operating Cost	\$/t processed	8.11
Royalties	% NSR	3.0
Pre-production Capital Cost	\$M	653.3
Sustaining Capital Cost	\$M	616.8

Description	Unit	Value
Reclamation and Closure Cost	\$M	68.0
Salvage Value	\$M	19.6

22.4 Metal Production

Over the life of the Project, the process plant will produce of a total of 1,687 Mlb of Lead (Average annual: 141 Mlb) in concentrate and 3,951 Mlb of Zinc (Average annual: 329 Mlb) in concentrate. On an equivalent Zinc (ZnEq) basis, a total of 5,366 Mlb will be produced in concentrate. Figure 22-1 provides a summary of the lead and zinc production by year.

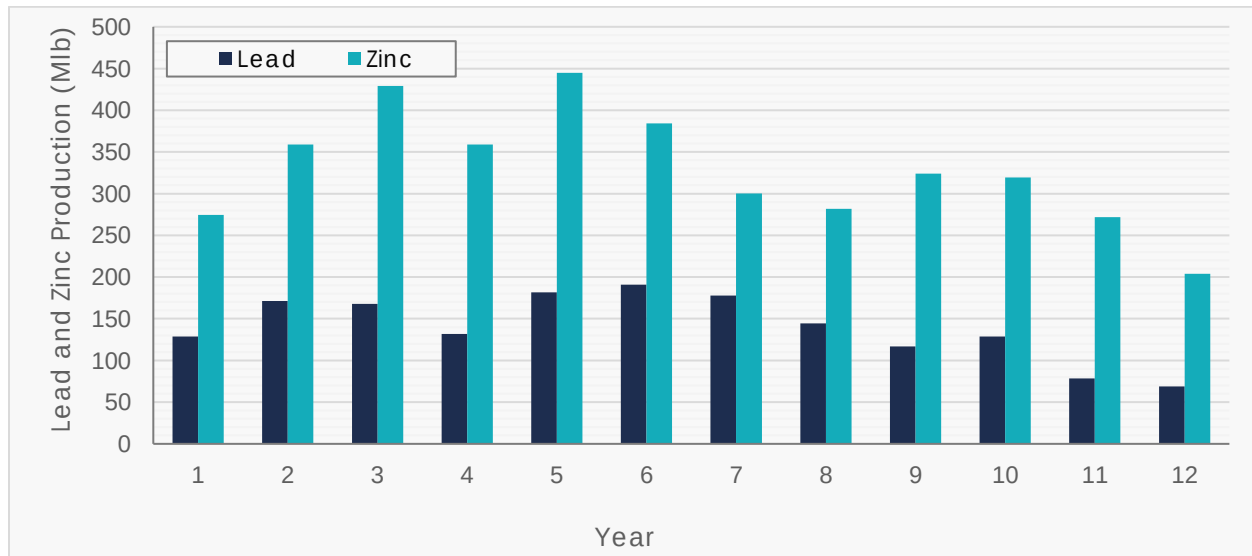


Figure 22-1: Annual lead and zinc production (Mlb)

22.5 Capital and Sustaining Costs

All capital costs (pre-production, sustaining, reclamation and closure) for the Project have been distributed against the development schedule to support the economic cash flow model. Figure 22-2 presents the planned annual and cumulative LOM capital cost profile.

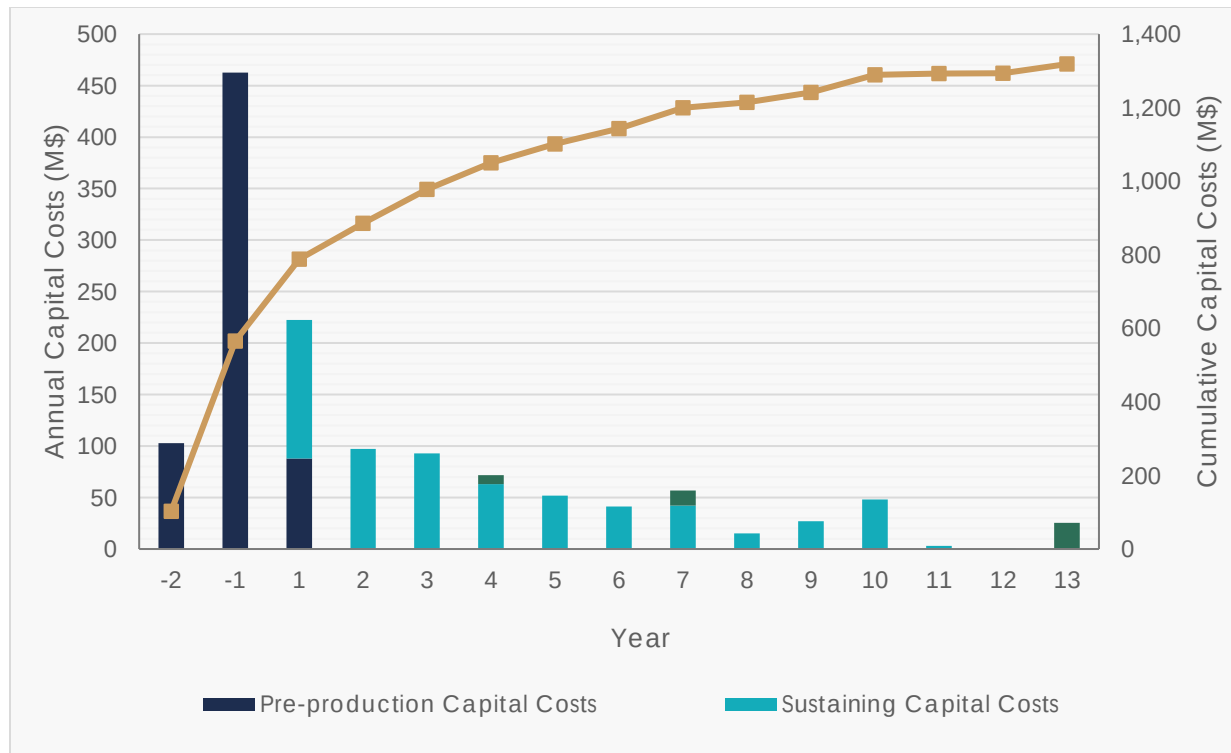


Figure 22-2: Overall Pine Point Project capital cost profile

22.6 Royalties

OGR retains a 3.0% net smelter return ("NSR") royalty on all metals produced by the Pine Point Project. Pursuant to the terms of the NSR Agreement, OGR has a right of first offer on any future sales by the Company of any additional royalties, streams or similar interests on the Pine Point Project. Over the life of the Project, approximately \$183.7M in royalties is expected to be paid based on the base case metal prices and Project assumptions.

22.7 Taxation

The Pine Point Project is subject to federal income tax, territorial income tax, a territorial mining tax ("Crown royalty") and carbon taxes. Osisko Metals compiled the taxation calculations for the Pine Point Project with assistance from third-party taxation experts.

The current Canadian tax system applicable to Mineral Resource Income was used to assess the annual tax liabilities for the Project. This consists of federal and territorial corporate taxes, as well as the Crown royalty. The federal corporate tax currently applicable over the operating life of the Project is 15.0% of taxable income while the territorial corporate tax is 11.5%. The marginal tax rates applicable under the Northwest Territories Mining Regulations are levied on a mine-by-mine basis and is equal to the lesser of 13% of the value of the output of the mine and the amount determined using escalating rates that are a function of annual mine profits ranging from 0% (0 to \$10,000) to 14% (over \$45 million).

The current Northwest Territories Carbon Tax Rate Schedule, along with diesel, natural gas and propane consumption estimates was used to assess carbon taxes for the Project.

The tax calculations are underpinned by the following key assumptions:

- The Project is held 100% by a corporate entity and the after-tax analysis does not attempt to reflect any future changes in corporate structure or property ownership;
- Assumes 100% equity financing and therefore does not consider interest and financing expenses;
- Class specific Capital Cost Allowance rates are used for the purpose of determining the allowable taxable income;
- Payments projected relating to NSR royalties are allowed as a deduction for federal and territorial income tax purposes, but are added back for mining tax purposes;
- Actual taxes payable will be affected by corporate activities, and current and future tax benefits have not been considered.

The combined effect on the Project of all taxes, including the elements described above, is an approximate cumulative effective tax rate of 38%, based on Project Earnings. It is anticipated, based on the Project assumptions, that Osisko Metals will pay approximately \$892.6M in tax payments (including \$37.4M in carbon taxes) over the life of the Project.



22.8 Financial Analysis Summary

An 8% discount rate was applied to the cash flow to derive the NPV for the Project on a pre-tax and after-tax basis. Cash flows have been discounted to Q1 2026 under the assumption that major Project financing and that the start of construction would be carried out at this time. The summary of the financial evaluation for the base case of the Project is presented in Table 22-2.

Table 22-2: Financial analysis summary (pre-tax and after-tax)

Description		Unit	Base Case
Pre-Tax	Net Present Value (0% disc)	\$M	2,328.6
	Net Present Value (8% disc)	\$M	1,098.7
	Internal Rate of Return	%	35.1
	Simple Payback Period	years	4.8
	Payback Period (after start of operations)	years	2.8
After-Tax	Net Present Value (0% discount rate)	\$M	1,436.0
	Net Present Value (8% discount rate)	\$M	602.3
	Internal Rate of Return	%	24.8
	Simple Payback Period	years	5.8
	Payback Period (after start of operations)	years	3.8

The pre-tax base case financial model resulted in an internal rate of return of 35.1% and an NPV of \$1,099M with a discount rate of 8%. The simple pre-tax payback period is 4.8 years. On an after-tax basis, the base case financial model resulted in an internal rate of return of 24.8% and an NPV of \$602M with a discount rate of 8%. The simple after-tax payback period is 5.8 years.

The summary of the Pine Point Project discounted cash flow financial model (pre-tax and after-tax) is presented in Table 22-3.



Table 22-3: Pine Point Project financial model summary

Description / Period	Unit	LOM (Total / Average)	-2	-1	1	2	3	4	5	6	7	8	9	10	11	12	13	14	15
			Pre-production		Production												Closure		
Mine Production																			
Mineralized Material	Mt	46.9	-	-	3.3	4.2	4.2	6.2	4.1	4.1	4.9	5.7	2.6	1.3	5.2	1.0	-	-	-
Open-pit	Mt	37.9	-	-	3.2	4.0	3.5	4.9	2.6	2.5	3.3	4.7	1.9	1.0	5.2	1.0	-	-	-
Underground	Mt	9.0	-	-	0.1	0.2	0.7	1.2	1.6	1.5	1.6	1.0	0.7	0.4	-	-	-	-	-
Overburden	Mt	75.1	-	-	8.6	10.8	5.4	9.5	10.9	11.5	5.7	6.9	4.9	0.8	0.0	0.1	-	-	-
Waste	Mt	140.0	-	-	23.0	21.0	18.8	6.0	6.8	6.3	11.6	8.5	9.7	14.7	11.2	2.3	-	-	-
Total Tonnes Mined	Mt	261.9	-	-	34.9	36.0	28.4	21.7	21.9	21.8	22.2	21.2	17.2	16.8	16.4	3.5	-	-	-
Process Plant Production																			
Material sent to Process Plant	Mt	46.9	-	-	2.9	4.1	4.1	4.1	4.1	4.1	4.1	4.1	4.1	4.1	4.1	2.9	-	-	-
Lead Feed Grade	%	1.76	-	-	2.13	2.02	1.99	1.57	2.15	2.25	2.10	1.72	1.40	1.54	0.97	1.20	-	-	-
Zinc Feed Grade	%	4.39	-	-	4.88	4.55	5.36	4.52	5.56	4.87	3.88	3.64	4.11	4.07	3.48	3.72	-	-	-
Zinc Equivalent Feed Grade	%	5.87	-	-	6.68	6.26	7.01	5.82	7.35	6.77	5.69	5.11	5.28	5.36	4.27	4.72	-	-	-
Lead Recovery (overall)	%	92.9	-	-	93.6	93.5	93.3	92.5	93.6	93.8	93.6	92.9	91.9	92.3	89.9	89.5	-	-	-
Zinc Recovery (overall)	%	87.0	-	-	87.4	87.0	88.4	87.7	88.4	87.2	85.4	85.4	87.1	86.8	86.3	85.3	-	-	-
Lead Concentrate Grade	%	65.0	-	-	65.0	65.0	65.0	65.0	65.0	65.0	65.0	65.0	65.0	65.0	65.0	65.0	-	-	-
Zinc Concentrate Grade	%	60.0	-	-	60.0	60.0	60.0	60.0	60.0	60.0	60.0	60.0	60.0	60.0	60.0	60.0	-	-	-
Lead Production (in concentrate)	Mlb	1,686.7	-	-	128.6	171.2	167.8	131.7	181.7	190.7	177.8	144.4	116.7	128.5	78.6	69.0	-	-	-
Zinc Production (in concentrate)	Mlb	3,951.0	-	-	274.5	358.7	429.1	359.0	444.9	384.0	300.2	281.7	323.8	319.4	271.7	203.9	-	-	-
Lead Concentrate Production (after transport losses)	kt	1,175.3	-	-	89.6	119.3	116.9	91.8	126.6	132.9	123.9	100.6	81.3	89.6	54.7	48.1	-	-	-
Zinc Concentrate Production (after transport losses)	kt	2,978.0	-	-	206.9	270.4	323.4	270.6	335.4	289.4	226.3	212.3	244.1	240.8	204.8	153.7	-	-	-
Payable Metals																			
Payable Lead	Mlb	1,625.3	-	-	123.9	165.0	161.7	126.9	175.1	183.8	171.3	139.2	112.4	123.8	75.7	66.5	-	-	-
Payable Zinc	Mlb	3,348.3	-	-	232.6	304.0	363.6	304.2	377.1	325.4	254.4	238.7	274.4	270.7	230.2	172.8	-	-	-
Metal Payable Value	\$M	7,807.2	-	-	555.8	730.1	829.7	684.1	869.6	790.3	651.5	585.1	614.6	622.0	492.8	381.7	-	-	-
Lead Payable Value	\$M	1,981.5	-	-	151.1	201.2	197.1	154.8	213.5	224.0	208.9	169.7	137.1	151.0	92.3	81.0	-	-	-
Zinc Payable Value	\$M	5,825.7	-	-	404.8	528.9	632.6	529.3	656.1	566.2	442.7	415.4	477.5	471.0	400.5	300.7	-	-	-
Transport Cost	\$M	867.8	-	-	62.1	81.6	91.8	75.5	96.4	88.6	73.7	65.7	67.7	68.9	53.8	42.0	-	-	-
Lead Concentrate	\$M	264.8	-	-	20.2	26.9	26.3	20.7	28.5	29.9	27.9	22.7	18.3	20.2	12.3	10.8	-	-	-
Zinc Concentrate	\$M	603.0	-	-	41.9	54.7	65.5	54.8	67.9	58.6	45.8	43.0	49.4	48.8	41.5	31.1	-	-	-
Treatment and Refining Costs	\$M	815.0	-	-	57.8	75.8	86.9	71.9	90.9	81.9	67.0	60.6	64.6	65.1	52.2	40.2	-	-	-
Lead	\$M	171.7	-	-	13.1	17.4	17.1	13.4	18.5	19.4	18.1	14.7	11.9	13.1	8.0	7.0	-	-	-
Zinc	\$M	643.3	-	-	44.7	58.4	69.9	58.5	72.4	62.5	48.9	45.9	52.7	52.0	44.2	33.2	-	-	-



Description / Period	Unit	LOM (Total / Average)	-2	-1	1	2	3	4	5	6	7	8	9	10	11	12	13	14	15
			Pre-production		Production												Closure		
Revenue																			
Exchange Rate	\$CAD : \$USD	1.27	1.27	1.27	1.27	1.27	1.27	1.27	1.27	1.27	1.27	1.27	1.27	1.27	1.27	1.27	1.27	1.27	1.27
Net Smelter Return	\$M	6,124.4	-	-	436.0	572.6	651.0	536.8	682.2	619.8	510.8	458.8	482.2	488.0	386.8	299.5	-	-	-
Lead	\$M	1,545.0	-	-	117.8	156.8	153.7	120.7	166.5	174.7	162.9	132.3	106.9	117.7	72.0	63.2	-	-	-
Zinc	\$M	4,579.4	-	-	318.2	415.7	497.3	416.1	515.7	445.1	348.0	326.5	375.3	370.3	314.9	236.3	-	-	-
Royalty Payments	\$M	183.7	-	-	13.1	17.2	19.5	16.1	20.5	18.6	15.3	13.8	14.5	14.6	11.6	9.0	-	-	-
Operating Expenditures																			
Mine Site Operating Costs	\$M	2,293.6	-	-	157.1	202.9	206.2	230.2	216.7	204.6	238.2	221.2	187.8	140.3	167.5	118.7	2.2	-	-
Open-pit Mining	\$M	843.1	-	-	92.0	97.3	85.0	68.8	67.4	66.5	69.2	70.0	59.9	62.2	67.5	37.3	-	-	-
Underground Mining	\$M	376.4	-	-	7.3	21.9	29.4	51.8	55.1	51.0	60.1	45.7	35.5	18.6	0.0	0.0	-	-	-
Processing	\$M	575.3	-	-	39.8	50.1	50.8	49.7	51.1	50.6	49.6	49.0	49.1	49.2	48.2	38.2	-	-	-
Energy Production	\$M	216.3	-	-	14.3	19.4	19.5	19.9	22.4	19.0	18.2	15.1	18.9	16.6	16.1	16.8	-	-	-
Tailings, Waste & Water Manag.	\$M	76.4	-	-	6.3	6.3	6.5	7.7	7.7	6.8	8.6	8.9	5.4	3.7	3.2	3.2	2.2	-	-
General and Administration	\$M	380.6	-	-	32.5	32.5	32.5	32.5	32.5	32.5	32.5	32.5	32.5	32.5	32.5	23.1	-	-	-
Capitalized Operating Costs	\$M	-174.5	-	-	-35.2	-24.8	-17.4	0.0	-19.5	-21.7	0.0	0.0	-13.6	-42.4	0.0	0.0	-	-	-
Net Operating Earnings	\$M	3,647.1	-	-	265.8	352.5	425.2	290.5	445.0	396.6	257.3	223.9	280.0	333.0	207.7	171.9	-2.2	-	-
Capital Expenditures																			
Total Capital Costs	\$M	1,318.5	102.9	462.6	222.4	97.3	92.9	71.7	51.8	41.3	56.7	15.1	26.9	48.0	3.0	0.5	25.3	-	-
Pre-production	\$M	653.3	102.9	462.6	87.8	-	-	-	-	-	-	-	-	-	-	-	-	-	-
Sustaining	\$M	616.8	-	-	134.6	97.3	92.9	63.1	51.8	41.3	42.3	15.1	26.9	48.0	3.0	0.5	-	-	-
Reclamation and Closure	\$M	68.0	-	-	-	-	-	8.7	-	-	14.4	-	-	-	-	-	44.9	-	-
Salvage Value	\$M	-19.6	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-19.6	-	-
Changes in Working Capital	\$M	0.0	-	-	3.5	1.1	0.6	-0.9	1.2	-0.5	-0.9	-0.4	0.2	0.0	-0.8	-0.7	-2.4	-	-
Pre-Tax Cash Flow																			
Pre-Tax Cash Flow	\$M	2,328.6	-102.9	-462.6	39.9	254.2	331.7	219.6	392.1	355.8	201.5	209.2	252.8	284.9	205.5	172.1	-25.1	-	-
Taxes and Duties																			
Total Taxes Payable	\$M	892.6	-	-	14.2	73.8	104.3	60.7	125.2	113.7	59.4	59.7	82.8	100.6	60.9	50.6	-10.0	-1.9	-1.3
Federal Corporate Income Tax	\$M	301.8	-	-	6.7	23.6	34.9	18.8	41.9	38.4	20.1	19.7	29.2	37.0	20.9	18.1	-5.7	-1.1	-0.8
Territorial Corporate Income Tax	\$M	231.4	-	-	5.1	18.1	26.8	14.4	32.1	29.5	15.4	15.1	22.4	28.4	16.0	13.9	-4.3	-0.8	-0.6
Mining Taxes	\$M	322.0	-	-	0.0	28.7	38.7	23.5	46.6	42.2	20.5	22.1	28.7	32.9	21.6	16.6	-	-	-
Carbon Taxes	\$M	37.4	-	-	2.3	3.3	3.9	4.0	4.7	3.5	3.4	2.9	2.5	2.3	2.4	2.1	-	-	-
After-Tax Cash Flow																			
After-Tax Cash Flow	\$M	1,436.0	-102.9	-462.6	25.7	180.4	227.4	159.0	266.9	242.1	142.0	149.5	170.0	184.3	144.6	121.5	-15.1	-	-

Figure 22-3 shows the cumulative cash flows for the Project projected for the life of the mine on a pre-tax and after-tax basis.

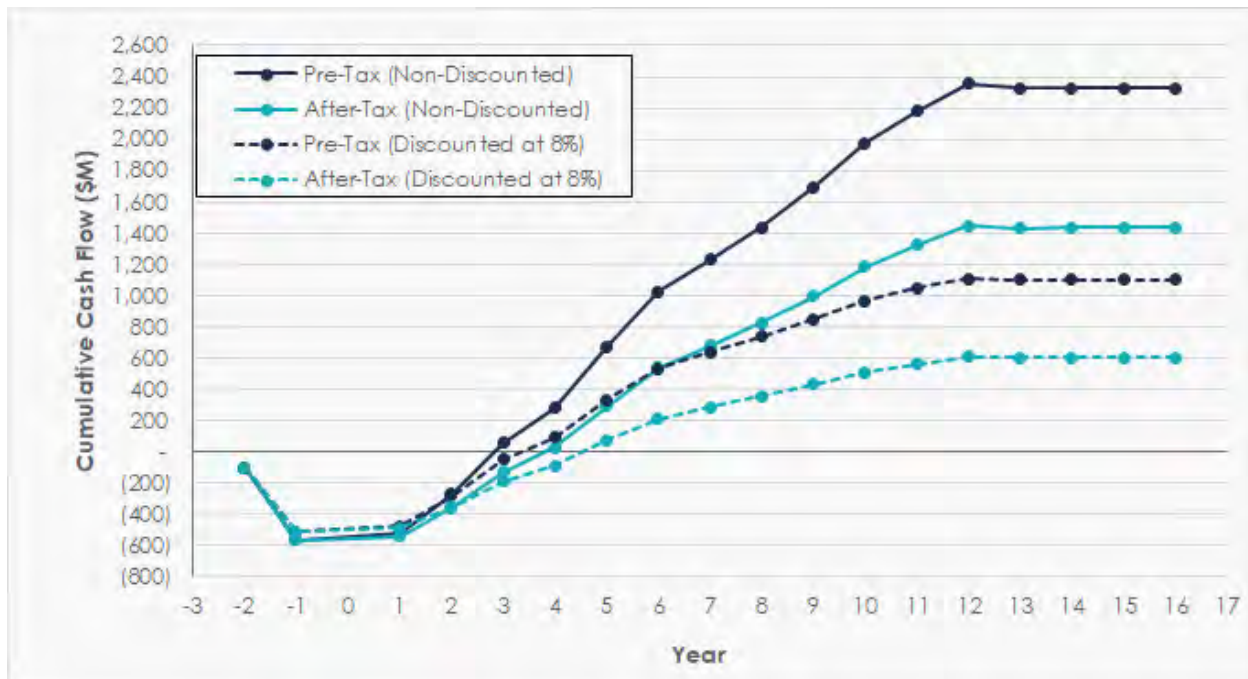


Figure 22-3: Life of mine cash flow projection (cumulative, pre-tax and after-tax)

22.9 Production Costs

A summary of the Project's production costs is provided in Table 22-4. All costs are in USD and are calculated on a Zinc equivalent (ZnEq) basis.

The LOM mine site operating cost per lb is 0.34 USD/lb. C1 Cash costs (Mine Site + Transport and Refining + Royalties) are estimated at 0.61 USD/lb while the LOM all-in sustaining cost ("AISC") is 0.71 USD/lb derived from the total cash costs plus sustaining capital, and closure costs. All in costs including on-site, off-site and required project capital is estimated at 0.80 USD/lb.



Table 22-4: Production cost summary (USD)

Description	Unit	LOM
Metal Production		
Zinc in concentrate	Mlb	3,951
Lead in concentrate	Mlb	1,687
Zinc Equivalent in concentrate	Mlb	5,366
Costs, Deductions and Royalties		
Mining	USD M	960
Processing	USD M	453
Power Generation	USD M	170
Tailings, Waste & Water Management	USD M	60
General and Administration	USD M	300
Capitalized Operating Costs	USD M	-137
Total Mine Site OPEX	USD M	1,806
Initial Capital	USD M	514
Sustaining Capital and Closure Costs	USD M	524
Total Capital Costs	USD M	1,038
Transport	USD M	683
Treatment and Refining	USD M	642
Royalties	USD M	145
Total Off-site Costs	USD M	2,354
Costs (per lb Zinc Equivalent)		
Mine Site Operating Cost	USD/ZnEq lb	0.34
Cash Cost (C1)	USD/ZnEq lb	0.61
All in Sustaining Costs (AISC)	USD/ZnEq lb	0.71
Estimated All-In Costs	USD/ZnEq lb	0.80

22.10 Value Drivers

Figure 22-3 presents the primary project value drivers discounted at 8%. The main value drivers (over \$350M) of the Project are the initial capital costs, zinc processing losses, zinc smelting payability factors/deductions, zinc concentrate TC+RC, open-pit mining, and taxes.

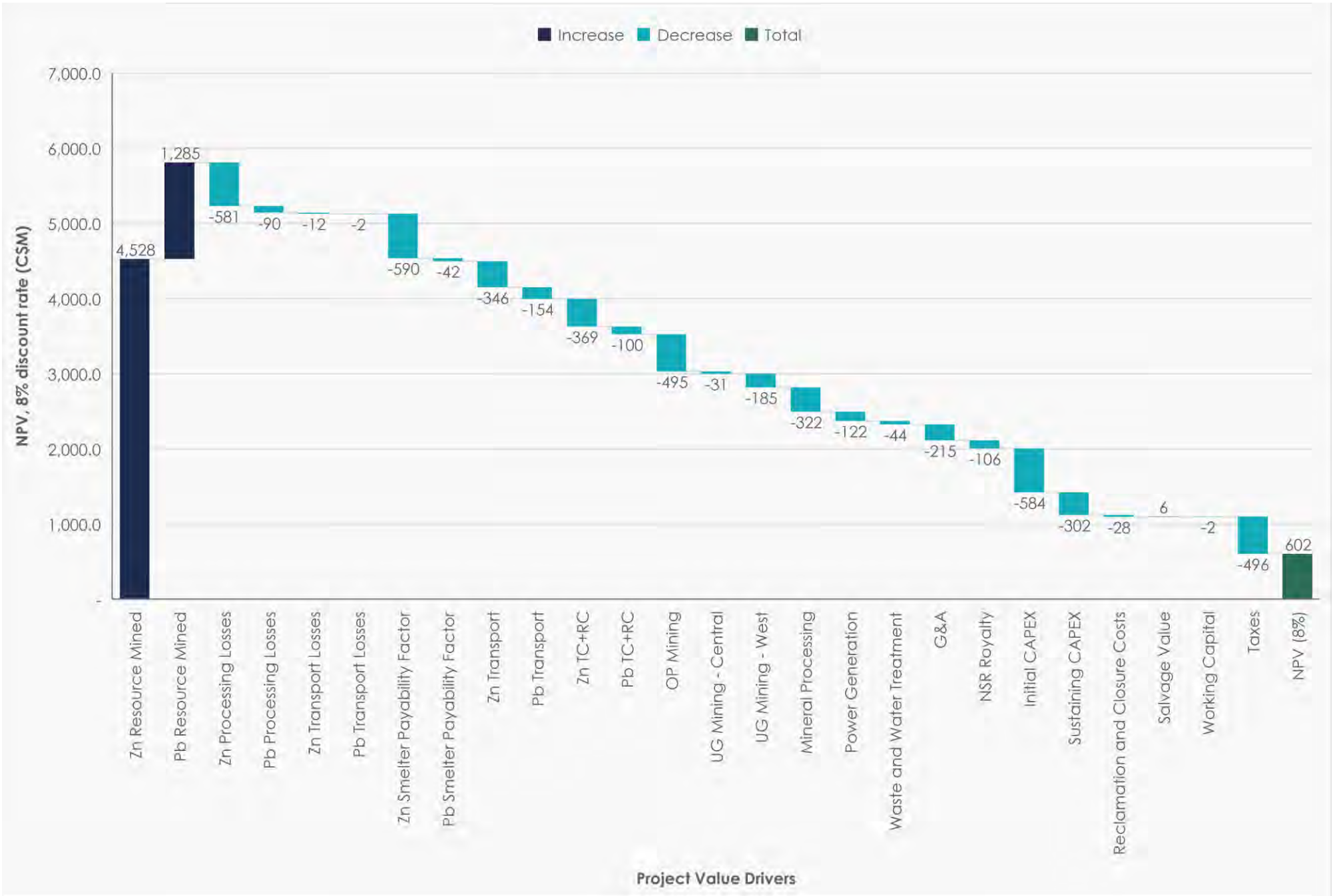


Figure 22-4: Main cost value drivers (CAD, discounted at 8%)





22.11 Sensitivity Analysis

A financial sensitivity analysis was performed on the base case after-tax cash flow NPV (8%) and IRR of the Project, considering variations in metal prices, feed metal grades, initial capital costs, operating costs and exchange rate. For the base case assumptions refer to Table 22-1. The after-tax results for the Project IRR and NPV based on the sensitivity analysis are summarized in Table 22-5 through Table 22-12 and shown graphically in Figure 22-5 and Figure 22-6. Table 22-13 presents the Project NPV at a range of discount rates from 0% to 14%. The NPV (8%), which is the base case, is bolded in the table.

Table 22-5: NPV sensitivity results (after-tax) for zinc price and exchange rate variations

Exchange Rate	Zinc Price (USD/lb)						
	1.01	1.13	1.25	1.37	1.49	1.61	1.73
-30%	-692.3	-498.6	-310.7	-157.8	-23.8	105.1	232.7
-20%	-397.3	-202.2	-46.9	100.6	246.5	392.5	538.4
-10%	-147.2	22.8	187.3	351.5	515.7	680.0	844.6
0%	55.0	237.5	420.0	602.3	785.1	968.4	1,151.8
10%	251.2	451.9	652.6	853.8	1,055.5	1,257.3	1,459.1
20%	447.4	666.4	885.9	1,106.0	1,326.1	1,546.3	1,766.5
30%	643.5	881.4	1,119.8	1,358.3	1,596.8	1,835.4	2,073.9

Table 22-6: IRR sensitivity results (after-tax) for zinc price and exchange rate variations

Exchange Rate	Zinc Price (USD/lb)						
	1.01	1.13	1.25	1.37	1.49	1.61	1.73
-30%	-	-12.1%	-2.8%	2.8%	7.2%	11.2%	14.9%
-20%	-6.7%	1.2%	6.5%	11.1%	15.3%	19.3%	23.1%
-10%	3.1%	8.7%	13.7%	18.2%	22.6%	26.7%	30.6%
0%	9.7%	15.1%	20.1%	24.8%	29.3%	33.6%	37.8%
10%	15.5%	20.9%	26.0%	30.9%	35.6%	40.2%	44.6%
20%	20.8%	26.4%	31.7%	36.8%	41.7%	46.5%	51.2%
30%	25.9%	31.6%	37.1%	42.5%	47.7%	52.7%	57.7%



Table 22-7: NPV sensitivity results (after-tax) for lead price and exchange rate variations

Exchange Rate	Lead Price (USD/lb)						
	0.66	0.76	0.86	0.96	1.06	1.16	1.26
-30%	-353.5	-276.6	-215.3	-157.8	-101.8	-46.9	6.4
-20%	-82.4	-20.5	40.3	100.6	160.4	220.3	280.2
-10%	149.4	216.7	284.1	351.5	418.8	486.1	553.5
0%	377.9	452.7	527.5	602.3	677.3	752.3	827.3
10%	606.3	688.8	771.3	853.8	936.5	1,019.3	1,102.0
20%	835.3	925.5	1,015.7	1,106.0	1,196.3	1,286.6	1,376.9
30%	1,064.9	1,162.7	1,260.5	1,358.3	1,456.1	1,553.9	1,651.8

Table 22-8: IRR sensitivity results (after-tax) for lead price and exchange rate variations

Exchange Rate	Lead Price (USD/lb)						
	0.66	0.76	0.86	0.96	1.06	1.16	1.26
-30%	-4.4%	-1.2%	0.8%	2.8%	4.7%	6.5%	8.2%
-20%	5.3%	7.3%	9.3%	11.1%	12.9%	14.6%	16.3%
-10%	12.5%	14.5%	16.4%	18.2%	20.1%	21.8%	23.6%
0%	18.9%	20.9%	22.9%	24.8%	26.7%	28.5%	30.4%
10%	24.8%	26.9%	28.9%	30.9%	32.9%	34.9%	36.8%
20%	30.4%	32.6%	34.7%	36.8%	38.9%	40.9%	43.0%
30%	35.8%	38.0%	40.3%	42.5%	44.7%	46.8%	49.0%

Table 22-9: NPV sensitivity results (after-tax) for zinc and lead feed grade variations

Lead Grade (%)	Zinc Grade (%)						
	-30%	-20%	-10%	0%	10%	20%	30%
-30%	-353.3	-64.6	178.5	386.9	595.2	803.9	1,013.1
-20%	-252.9	8.9	250.4	458.7	667.1	876.0	1,085.3
-10%	-171.8	81.3	322.3	530.5	739.0	948.1	1,157.5
0%	-95.1	153.2	394.1	602.3	811.1	1,020.3	1,229.7
10%	-20.8	224.9	465.9	674.3	883.2	1,092.6	1,302.0
20%	51.8	296.9	537.7	746.3	955.4	1,164.8	1,374.2
30%	123.7	368.7	609.5	818.3	1,027.6	1,237.0	1,446.5



Table 22-10: IRR sensitivity results (after-tax) for zinc and lead feed grade variations

Lead Grade (%)	Zinc Grade (%)						
	-30%	-20%	-10%	0%	10%	20%	30%
-30%	-4.5%	5.9%	13.4%	19.2%	24.5%	29.6%	34.5%
-20%	-0.5%	8.3%	15.4%	21.1%	26.3%	31.4%	36.2%
-10%	2.3%	10.5%	17.4%	22.9%	28.1%	33.1%	37.9%
0%	4.9%	12.7%	19.4%	24.8%	29.9%	34.8%	39.5%
10%	7.3%	14.8%	21.3%	26.6%	31.6%	36.5%	41.2%
20%	9.6%	16.8%	23.2%	28.4%	33.3%	38.1%	42.8%
30%	11.8%	18.8%	25.0%	30.1%	35.0%	39.8%	44.4%

Table 22-11: NPV sensitivity results (after-tax) for operating and capital cost variations

CAPEX	OPEX						
	-30%	-20%	-10%	0%	10%	20%	30%
-30%	1,153.6	1,069.0	984.6	900.2	816.0	731.8	647.5
-20%	1,054.3	969.7	885.3	800.9	716.7	632.5	548.2
-10%	955.0	870.4	786.0	701.6	617.4	533.2	448.9
0%	855.7	771.1	686.7	602.3	518.1	433.9	349.6
10%	756.4	671.8	587.4	503.0	418.8	334.6	250.3
20%	657.1	572.5	488.1	403.7	319.5	235.3	151.0
30%	557.8	473.2	388.8	304.4	220.2	136.0	51.7

Table 22-12: IRR sensitivity results (after-tax) for operating and capital cost variations

CAPEX	OPEX						
	-30%	-20%	-10%	0%	10%	20%	30%
-30%	49.5%	46.9%	44.3%	41.7%	39.0%	36.3%	33.5%
-20%	41.9%	39.6%	37.2%	34.8%	32.4%	29.9%	27.3%
-10%	35.8%	33.7%	31.5%	29.3%	27.0%	24.7%	22.4%
0%	30.8%	28.8%	26.8%	24.8%	22.7%	20.5%	18.3%
10%	26.7%	24.8%	22.9%	21.0%	19.0%	16.9%	14.8%
20%	23.1%	21.3%	19.5%	17.7%	15.8%	13.9%	11.8%
30%	20.0%	18.3%	16.6%	14.9%	13.0%	11.2%	9.2%

Table 22-13: NPV sensitivity results (after-tax) for discount rate

	Discount Rate						
	0%	4%	6%	8%	10%	12%	14%
NPV	1,436.0	937.5	753.8	602.3	476.7	372.1	284.4

The sensitivity analysis reveals that the USD:CAD exchange rate, zinc price and zinc feed grade have the most significant influence on both NPV and IRR compared to the other parameters, based on the ranges evaluated. The Project is less sensitive to changes in operating costs, capital costs, lead price and lead feed grades.

Overall, the NPV and IRR of the Pine Point Project is positive over the range of values used for the sensitivity analysis when the sensitivities are analyzed individually. The only exceptions occurs when the USD:CAD exchange rate approaches parity and the zinc grade and price is at -30%.

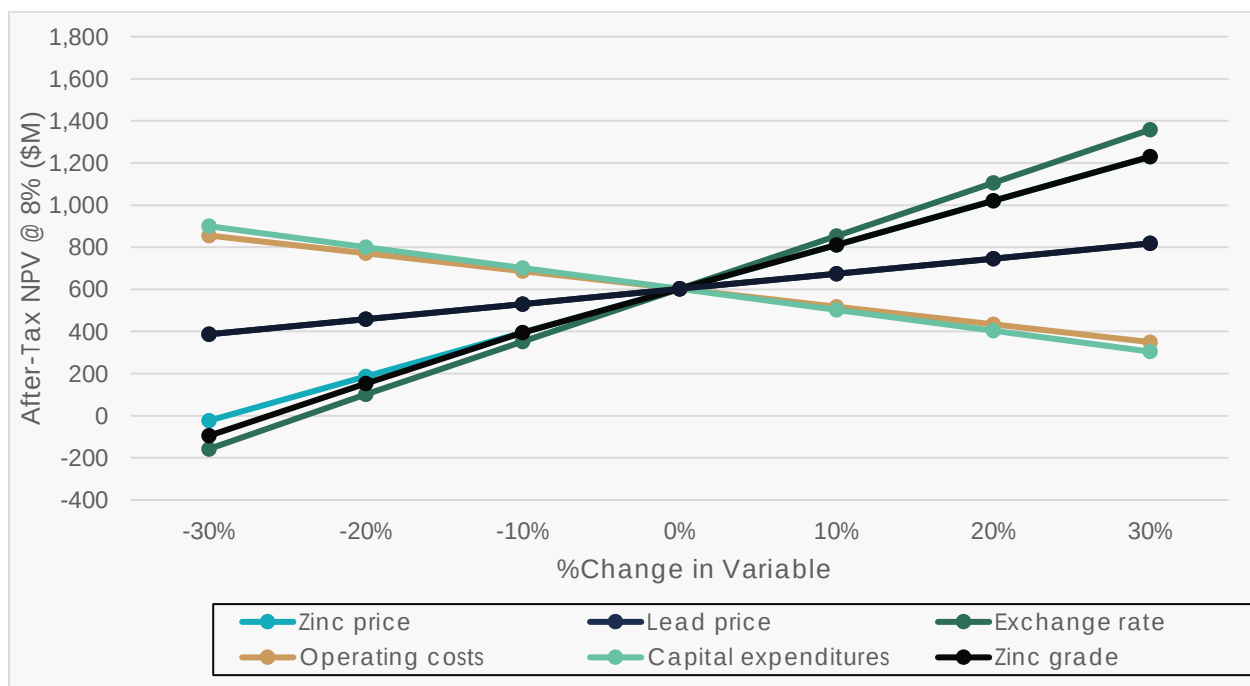


Figure 22-5: Sensitivity of the net present value (after-tax) to financial variables

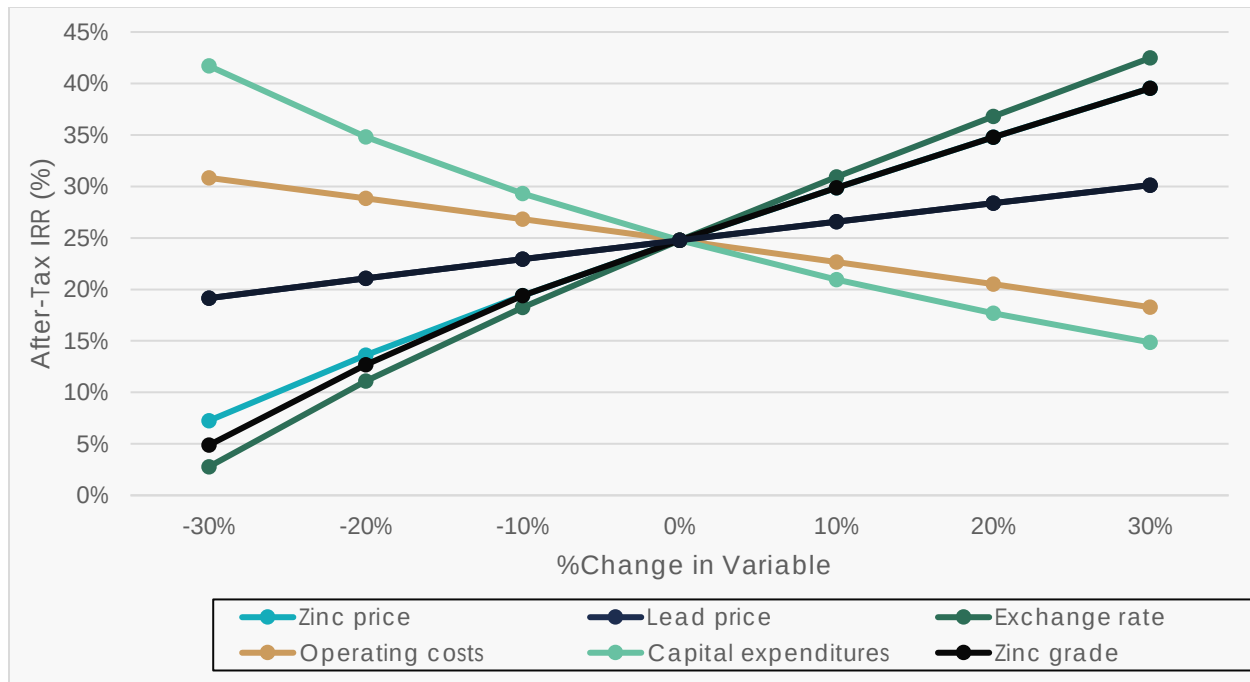


Figure 22-6: Sensitivity of the internal rate of return (after-tax) to financial variables

22.12 Comments on Chapter 22

Under the assumptions in this Report, and based on the available data, the Project shows positive economics. Using an 8% discount rate, the Project has an after-tax NPV of \$602M, an IRR of 24.8% and a payback period of 5.8 years.

The Project economics are very sensitive to the USD:CAD exchange rate, Zinc price and Zinc grade. It should be noted that the economic viability of the Project will not be significantly negatively impacted by variations in the capital cost within the margins of error associated with the capital cost estimate. There is upside potential for the Project if the zinc prices increase from the base assumption of 1.37 USD/lb used in the Report, since the sensitivity analysis shows that the Project is very sensitive to zinc price.

23. ADJACENT PROPERTIES

Historically, several mining and exploration companies have held and explored grounds outside the current area controlled by Pine Point Mining Limited, i.e., 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 Westmin Resources, who led the Great Slave Reef joint venture.

Cominco Ltd. explored an area of roughly 60 km by 18 km, covering 1,182 km² 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 ("IP") geophysics and follow-up drilling. Unfortunately, the IP 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 IP geophysics and geological concepts. Westmin turned over much of its 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.

Currently, apart from one claim (registered to a third party), there are no adjacent mineral claims or leases to the Project as the surrounding area is withdrawn from staking.

24. OTHER RELEVANT DATA AND INFORMATION

The Pine Point Project is composed of two geographical areas where mine development and construction activities will occur:

- **Process Plant Site:**
 - Underground and open-pit mines;
 - Process plant;
 - Accommodations & office complex;
 - Mine garage;
 - Power generation and distribution;
 - Waste rock and overburden piles;
 - Tailings and process water reclaim systems;
 - Dewatering infrastructure.
- **Pine Point Junction:**
 - Concentrate trans loading facilities;
 - Railway spur lines;
 - Office and laboratory.

The Project organization and execution philosophy reflects this subdivision.

24.1 Project Organization

24.1.1 Management

Osisko Metals will manage the technical studies and construction management.

All Project phases including detailed engineering, procurement, pre-production and construction activities will be under the direction of the Osisko Metals Vice-President, Engineering and Construction.

Project financing will also be led by the Osisko Metals financial team.

The Osisko Metals team is an experienced mine project development team and will be in charge of the Project management functions for the Pine Point Project. The team and their network consists of highly experienced individuals with knowledge of the local construction conditions and

contractors. They have successfully managed projects in difficult conditions and remote environments from the engineering and planning phases through construction to commissioning and operations.

Osisko Metals will supervise the feasibility study ("FS"), the environmental and social impact assessment study and the Project detailed engineering. The requirement for an early works program will be evaluated and planned during the early stages of the FS phase. Specialized engineering firms will be selected for each portion of the Project to assemble a strong integrated design and execution team. They will be responsible for the following functions in the respective phases:

■ **Feasibility study:**

- Preliminary technical specification and scope of work documents;
- Bid selection for FS;
- 3D modelling, drawings production and material take-offs;
- Cost estimating of direct cost components.

■ **Execution Phase:**

- Definite specification and scope of work documents;
- Technical and economical evaluations;
- Short list meetings;
- Purchase order requisition preparation;
- Drawing management and approval.

Osisko Metals is responsible for the following activities in the respective phases:

■ **Feasibility study:**

- Budgetary/firm bid request;
- Addenda;
- Reception of bids;
- Indirect cost estimate;
- Project execution plan.

■ **Project execution:**

- Definite bid request;
- Addenda;
- Bid reception;



- Final negotiation;
- Contract award;
- Purchase order release;
 - o Progressive payment;
 - o Shop visits;
 - o Site logistics.

Due to the complexity of major process equipment transportation, Osisko Metals will retain the services of a specialized company to develop the project logistics plan during the feasibility study and for its implementation during the project execution phase.

24.1.2 Construction Management

In the feasibility study phase of the Pine Point Project, construction management will contribute to the project design with constructability reviews. In the project execution phase, Construction Management will be performed by Osisko Metals with the support of third-party construction and contract administration specialists under the supervision of the construction manager. The Construction Management Team ("CMT") will include the following services:

■ Constructability reviews;	■ Reporting;
■ Site supervision;	■ Health, safety and environment;
■ Project cost control;	■ Contract administration;
■ Scheduling;	■ Construction progress measurement.

It is recognized that an effective health and safety program during the Project is a necessity. The success of the construction safety program is contingent upon its enforcement at all phases of the Project, including design, construction planning, construction execution, and start-up and commissioning.

The CMT will also follow the Osisko Metals procedures and work methods to ensure the protection of the environment. Furthermore, the CMT will work closely with each department of the operations group to ensure proper installation and functional results. During the construction phase, personnel from operations will be integrated into the construction team as coordinators and supervisors.



The Osisko Metals and Pine Point Mining Limited operations group will support the CMT for the following services during the construction phase:

- Staff payroll;
- Accounting support;
- IT support;
- Site security;
- Public relations;
- Environmental and permitting;
- Medical and first aid;
- Site logistics.

24.2 Project Execution Schedule

The preliminary project execution plan is developed to a PEA level and is therefore conceptual in nature. The execution plan and schedule will be further developed and detailed during the feasibility study and detailed engineering phase. The preliminary project execution schedule, developed in this PEA and described herein, covers the period from the end of the PEA Update (Q3 2022) up to the achievement of commercial operation in the third quarter (Q3) of 2028. For the purposes of this conceptual schedule and alignment with the mining and CAPEX/OPEX expenditure schedules, 2028 is considered year one (YR 1) of the twelve-year LOM.

Major project milestones for the Project activities are shown in Table 24-1.

Table 24-1: Key milestones (preliminary)

Activity	Start Date	Completion Date
Complete PEA Update Study		Q3 2022
Developers Assessment Report	Q3 2022	Q2 2024
Complete Feasibility Study	Q2 2023	Q3 2024
Process plant detailed engineering	Q1 2025	Q2 2026
Project EAC* application and reception of Certificate	Q2 2024	Q4 2026
Process plant construction	Q4 2026	Q1 2028
Pre-production mine development	Q2 2027	Q3 2027
End of process plant construction/plant commissioning		Q1 2028
Process plant fully operational (commercial production)		Q3 2028

*EAC – Environmental Assessment Certificate

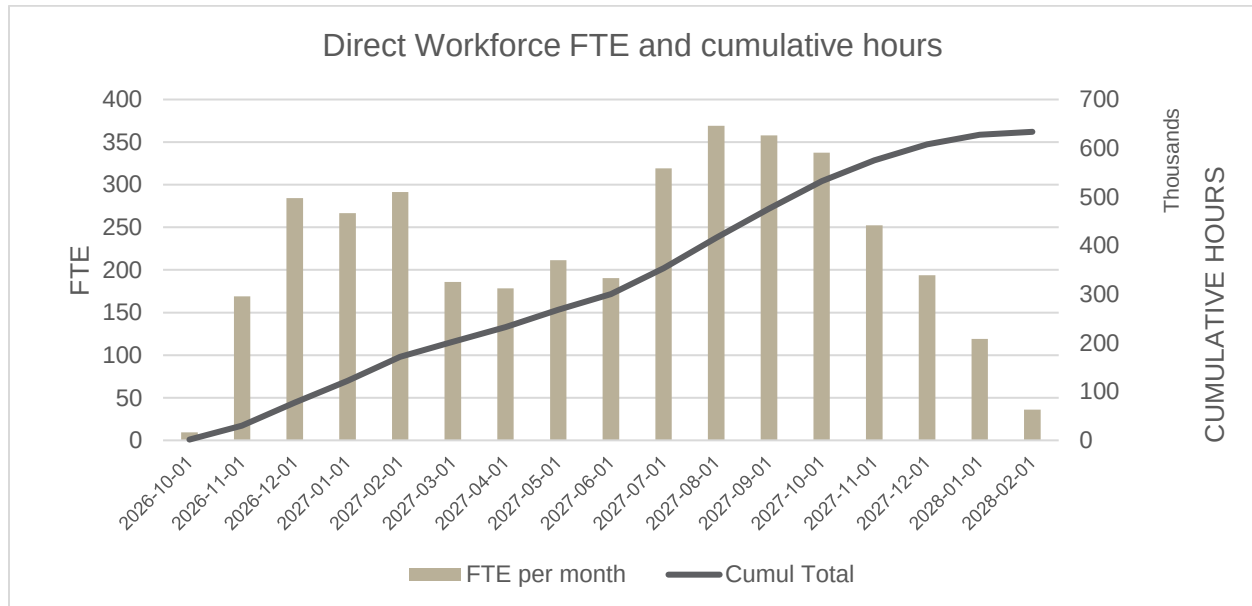
The Project critical path runs through the completion of the developer's assessment report, the Environmental Assessment Certificate ("EAC") application process and the reception of the EAC, the detail engineering, the procurement and construction of the process plant facility.



All mining projects located in the Northwest Territories are assessed in accordance with the *Mackenzie Valley Resource Management Act* ("MVRMA"). Environmental assessments are conducted by the Mackenzie Valley Environmental Review Board ("MVEIRB") and includes all relevant federal agencies, such as Environment and Climate Change Canada ("ECCC") and Fisheries and Oceans Canada ("DFO"), Indigenous Governments and community groups as parties to the process. At the completion of the environmental assessment ("EA"), if the board recommends the Project be approved, the Mackenzie Valley Land and Water Board ("MVLWB") will process the proponents' applications for a Water License and Land Use Permit through a public process (the Permitting Phase). It is expected that the EA process will take approximately 40 months, inclusive of an Initiation stage to determine the scope of the EA, followed by the Permitting Phase, which is expected to take approximately 12 to 15 months. A feasibility study will be developed in parallel with the EA Process. The developer's assessment report is expected to be completed in 2024.

Detailed engineering is expected to begin in the first quarter of 2025 and be completed over a 15-month period. Construction of the process plant will begin in the fall of 2026 with site preparation and foundation work. Work will continue through the winter aiming for construction to be completed in the first quarter of 2028. Commissioning will be completed in Q1 of 2028 and ramp up to commercial operation will be achieved in Q2/Q3 2028.

Over the estimated 16-month construction period, an average of 280 construction personnel, peaking at approximately 350, will be present on-site (Figure 24-1). This personnel count includes direct construction labour force for both underground and surface facilities, contractor supervision, owner and seconded construction management team, third party testing technicians, vendor representatives for installation and commissioning support, and underground construction crews.



FTE - The calculation of full-time equivalent ("FTE") is an employee's scheduled hours divided by the employer's hours for a full-time workweek

Figure 24-1: Construction total labour requirements

The Project is located in a relatively unpopulated area and will require that the workforce be lodged at the site. The workforce is expected to come mostly from the Northwest Territories and northern Alberta. The construction workforce will be lodged in owner supplied camps. At the onset, the exploration camp with an expanded capacity of approximately 150, will be used to lodge the permanent operations camp construction, civil and concrete crews. As soon as it is completed, the operations camp, capacity of 280, will then be used to lodge construction personnel.

25. INTERPRETATION AND CONCLUSIONS

25.1 Overview

BBA, PLR Resources, WSP and Hydro-Ressources have been mandated by Osisko Metals ("OM") to carry out a Preliminary Economic Assessment ("PEA") in accordance with NI 43-101 standards in order to assess the economic viability of the Pine Point Project.

This NI 43-101 compliant technical report provides a summary of the results and findings from the main aspects evaluated, including mineral resources, mine design, metallurgy, process design, infrastructure, environmental management, capital and operating costs and economic analysis.

This PEA was prepared by experienced and competent independent consultants using recognized engineering standards and methods. The mutual conclusion is that the Pine Point Project PEA contains adequate details and information to support the positive economic outcome presented. The QPs are not aware of any fatal flaws, and consider the Project technically viable based on the assumptions considered. In Chapter 26, recommendations are made to mitigate the risks and benefit from potential opportunities identified during the study. In conclusion, the QP's recommend that Osisko Metals proceed to the next phase of project development for the Pine Point Project that is the initiation of a Feasibility Study.

25.2 Mineral Tenure, Surface Rights, Agreements and Royalties

The information provided by Osisko Metals supports 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 the Mineral Resource Estimate.

25.4 Data Verification and Mineral Resources

The 2022 Pine Point Mineral Resource Estimate (the "2022 MRE") was prepared by Pierre-Luc Richard, P. Geo., using all available information including historical and recent diamond drillholes as of December 31st 2020.

The mineral resources in the 2022 MRE are not mineral reserves as they do not have demonstrated economic viability. The estimate is categorized as Inferred and Indicated Mineral Resources based on data density, search ellipse criteria, drillhole density, specific interpolation parameters, geological continuity and grade continuity above the cut-off grade. The effective date of the estimate is March 10th, 2022 based on the compilation status and cut-off grade parameters.

The QP considers the 2022 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, the following conclusions can be made:

- The MRE is divided into five geographic zones and includes 47 open-pits as well as 8 underground deposits;
- Using a variable cut-off grade, the Indicated Mineral Resources amount to 15.8 Mt grading 4.17% zinc and 1.53% lead;
- Using a variable cut-off grade, the Inferred Mineral Resources amount to 47.2 Mt grading 4.43% zinc and 1.68% lead;
- At the deposit scale the grade and tonnage show very little variability at lower cut-off grades.

25.5 Exploration Potential

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

- The exploration potential remains high at the property scale, justifying further geological compilation and continuing exploration target generation programs;
- The Pine Point Project hosts many mineralized intercepts that merit follow-up work;
- The potential is high for adding additional mineral resources to Pine Point Project by drilling lateral extensions of the numerous currently identified zones;
- The potential is high to upgrade Inferred Mineral Resources to the Indicated category with additional drilling.

25.6 Mining Methods

The mining methods used for the extraction of mineralized material of the Project is 85% by open-pit mining method and 15% by underground mining methods (based on the mineralized material produced). Most of the deposits are mined as open-pits, prioritizing the ones with the highest profit margin while deeper deposits with higher strip ratios are mined from underground. Indicated and Inferred Mineral Resources account for 21% and 79% of the open-pit and underground mined tonnes, respectively.

Underground mining methods are longhole mining for W1 and room and pillar mining for C1.

The LOM plan and mining production schedule is the result of many iterations, including combination sequencing with different mining rates and methods which aimed to optimize the profitability and result in a 12-year mine life with 4.1M tons of mine mineralized material per year. Other iterations will be required as new information becomes available. The Pine Point Project provides much flexibility and many opportunities due to the number of shallow deposits.

25.6.1 Mine Dewatering

Mine Dewatering is an important part of the Project and is required to commence prior to mine development. The 2022 mine dewatering plan is based on historical data, new field information and flow simulation that is beyond PEA level. Further investigation and more advanced modelling and studies will be required to better estimate the required dewatering rates for future studies (FS). The mining schedule will be optimized to potentially reduce total inflow and power consumption.

Pumping is planned to be performed in a similar fashion as was performed during the Cominco era, using wells and submersible pumps, but located through water-bearing faults. The current number of wells, which assumes that some deposits will be dewatered from dewatering nearby active pits, are considered to be appropriate based on the current assumptions, but the pump capacity and the borehole spacing and depths are defined to a level that is similar to an operational level. The re-use of Pumps and piping equipment will be optimized once a more detailed mine production schedule is completed.

The West Zone dewatering requirements are based on limited data but are expected to be significantly higher than in the eastern Zones. Grouting and till injection are therefore considered an appropriate groundwater inflow control method and could possibly be used in other areas. Currently, the X25 deposit results suggest a low inflow and the NE trend suggests a high inflow due to the calibration approach. Further information in each of these areas will help to better estimate inflow rates for the project and will be used during the FS.



Water management infrastructure includes the use of existing open pits, in-pit sumps and pumping equipment, ditches and collection ponds. If required, any direct discharge of water to the environment will meet applicable discharge requirements. Further studies will be conducted to inform water management planning.

A system of ditches and ponds will be used to convey mine dewatering water to Great Slave Lake. PPML is conducting further studies evaluating water management options, taking into account environmental and permitting considerations. As per the above, any discharge to the receiving environment will meet applicable criteria. On the North Trend, current simulations are considering re-injection in the old pits, inducing higher inflow in this area. The water management strategy is to be validated with field testing and further simulations. There could be a significant inflow reduction based on variable scenarios. Water chemistry needs to be defined further for the FS.

25.6.2 Open-pit Mining

Forty-seven deposits located within four zones (East Mill, Central, North, and N204) are mined as open pits. The total mineralized material mined from open-pit mining is 37.9 Mt grading 3.98% Zn and 1.44% Pb for 5.18 % ZnEq

Open-pit mining is planned to be conducted using conventional methods with trucks and shovels. Mining costs are considered accurate based on the high mining rate. It assumes that no drilling and blasting is required in the overburden and the use of a large open-pit mining fleet used on a leased-to-purchase plan that is distributed throughout the LOM. The mining costs also include waste rock and overburden transportation to nearby disposal areas (< 2km) as well as mineral sorter rejects handling, that has a dedicated fleet but also plans on using available open-pit mining equipment when the surface production rate is reduced when underground production rates are scheduled to increase.

No detailed pit design engineering was conducted at this phase of the Project and further engineering may allow for some optimization. Some nearby pits may benefit from being mined and/or dewatered as clusters. The slopes used to generate the Whittle™ pit shells are considered appropriate at this phase of the Project, and though conservative, the design considers that the pits are shallow and that some historical pits have steeper slopes. Additional characterization may provide opportunities to optimize the pit slopes and reduce the strip ratio.

An average dilution percentage was used for all the open-pit deposits for the PEA but the dilution variation from one deposit to another should be analyzed on a pit-by-pit basis in the next phases of the Project development and be considered in the mining production schedule. Higher dilution might impact mining costs as well as operational efficiencies.



The grade distribution by bench and along tabular shaped deposits may have an impact on the mining schedule.

The volume of overburden to strip could also be an issue if significantly higher than expected. Additional definition drilling is required and recommended for the next project phases.

25.6.3 Underground Mining

Deeper deposits with a higher strip ratio are proposed to be mined by using underground mining methods. A total of eight deposits, located in two zones (West and Central) require underground development, which is initiated in Year 1 to alleviate initial capital expenditures. The total mineralized material mined from underground is 9.0 Mt grading 6.16% Zn and 3.11% Pb for 9.27 % ZnEq. The underground mining costs assume contract mining, using mainly the longhole stope method in the West Zone as well as the room and pillar mining method in both the West and Central zones.

Ground conditions are expected to be "Fair" to "Good" in the mineralized areas, and "Fair" in the other rock formations. A stability pillar will be set in the middle of the mineralized material body to support the crown pillar span. Geotechnical analyses support the planned open stope designs and all underground workings are planned to be backfilled. Further characterization may allow increasing stope dimensions and room and pillar recovery.

For certain deposits in the West zone, dewatering is used to reduce the use of grouting beforehand. However, grouting requirements are based on high-level estimates and dewatering for the connection ramp was not assessed at this phase of the Project. Dewatering requirements and design, including the grouting, will have to be refined during the next phase of the Project development.

The presence of several deposits independent from one another, represents a moderate risk to the Project schedule. However, the use of two mining methods provides significant flexibility to mining operations. The longhole and room and pillar mining methods are proven methods and are suitable for the prismatic and tabular deposits respectively.

The planned equipment fleet is appropriate for the mine as planned; ventilation benefits may be attained through the use of LNG units instead of diesel units.

The amount of overburden and waste rock to be displaced during the excavation of the underground deposit portals are high-level estimates. Trenchless methods should be explored during the next Project study.

The underground mining fleet and crew are considered appropriate based on the current mining production schedule but vary over the LOM. Up to three development teams are necessary in Year 3, in order to reach the production schedule targets considering the development needs to be executed in three underground deposits during the same year. A more detailed assessment of the mine fleet and crew requirements is recommended for future studies.

The hauling of mineralized material from the West zone towards the process plant will also have to be further assessed during the next phase.

25.7 Metallurgy

In 2018, Osisko Metals initiated testwork to investigate the potential of using mineral sorting technology. Preliminary results indicate that the 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. Based on an average of the samples tested, mineral sorting of the coarse fraction was able to increase the grade of both Pb and Zn in the concentrate by approximately two times, achieve recoveries of 98% for Pb and 93% for Zn while rejecting approximately 58% of the mass in the coarse fraction to a waste product. Including the fine fraction, the expected waste rejection is estimated at approximately 40%.

In 2019, flotation testwork based on a conventional Pb/Zn flowsheet (roughing and two stages of cleaning) using mineral sorting testwork concentrate from the East Mill, Central and North Zones was performed. The testwork program demonstrated that zinc recoveries of 93% to 95% are possible at concentrate grades of 63% to 64% Zn while lead recoveries of 87% to 91% are achievable at concentrate grades ranging from 68% to 72% Pb. Subsequent to the 2020 Preliminary Economic Assessment ("PEA"), a small flotation testwork program was completed in December 2020. The primary findings from the test program indicate that both zinc and lead recoveries from the N204 zone material will likely be lower than originally estimated due to its fine grain mineralization.

Elemental analysis of the zinc concentrate during the laboratory testing program indicates that it contains low levels of deleterious elements except for Magnesium Oxide, which is at the typical smelter penalty threshold. Based on the preliminary testwork completed to date, it is not anticipated that there will be any smelter or refinery penalties for the zinc or lead concentrates. These recovery factors, concentrate grades and typical smelter terms have been applied to the Mineral Resource Estimate and the PEA and are considered acceptable and appropriate.

Additional material sorting, grinding, flotation and dewatering testwork on all zones at Pine Point is recommended to confirm metallurgical performance and variability as well as for gathering design data for flowsheet development and future economic studies.

25.8 Recovery Methods

Metallurgical testwork has shown that the mineralized material is amenable to preconcentration by mineral sorting ("XRT") in the crushing circuit with 99% lead and 93% zinc recovery at 42% weight recovery and to froth flotation at a primary grind size of P80 100 µm achieving a 65% lead concentrate with 93% overall recovery and a 60% zinc concentrate with 87% overall recovery. Lead and zinc will be recovered in dedicated standard rougher and two-stage cleaner flotation circuits. The proposed equipment is conventional. No regrind circuit is included based on preliminary testwork results.

The processing plant will consist of the following unit operations:

- Primary jaw crushing;
- Secondary and tertiary crushing and mineral sorting;
- Fine mineralized material storage and handling;
- Ball mill grinding in a closed circuit with hydrocyclone classification;
- Lead flotation cells including rougher and two cleaning stages;
- Zinc flotation cells including rougher and two cleaning stages;
- Concentrate dewatering (zinc and lead concentrates);
- Flotation tailings thickening and pumping;
- Fresh and reclaim water supply;
- Reagent preparation and distribution.

Two types of final concentrates will be produced: a zinc concentrate and a lead concentrate. Tailings will be disposed of in already mined-out open-pits. Process plant energy requirements are expected to average about 79 GWh/a (excluding heating) over the planned life of mine. The average installed power for these facilities will be about 10.8 MW (excluding heating). Fresh water requirements are estimated to be 42 m³/h. The major reagents are anticipated to be collectors (Aerofloat 3418A and Cytec 3894), frother (MIBC), CMC, lime, copper sulphate activator, zinc sulphate depressant, and flocculant.

25.9 Environmental Studies

The Project has entered the environmental assessment process with the submission of the Environmental Assessment Initiation Package and the receipt of the final Terms of Reference from the Mackenzie Valley Environmental Impact Review Board. Historical baseline data are available for the area and environmental baseline studies to support the Project have been initiated. Engagement with local Indigenous communities has been ongoing.

The environmental baseline studies are ongoing. Available data support the concepts in the PEA. Further work is underway to support environmental assessment and permitting, as well as closure planning for the Project.

25.10 Infrastructure

The information and assumptions used in the design of the various Project infrastructure components are sufficient to support a PEA. Field work and further design studies are underway or recommended to continue refining the Project and to support subsequent design phases.

The historical brownfield site is a benefit to the Project and has reduced required road and surface infrastructure capital expenditures for the proposed project.

25.11 Market Studies and Contracts

Osisko Metals engaged Stonehouse Consulting Inc, a third-party marketing specialist company to prepare a market study (Vice, 2022) to support the PEA. Project economics were estimated based on a long-term exchange rate (CAD:USD) of 1.27 and metal prices of US\$1.37/lb for zinc, and US\$0.96/lb for lead (Charles, 2022).

The project will produce very clean zinc and lead concentrates with very low deleterious elemental content. The zinc concentrates are expected to be high-grade averaging 60% Zn. Analysis of the zinc concentrate indicates that the level of MgO is near the typical smelter penalty threshold of 0.35% MgO. The focus of sales of zinc concentrate will be to North American smelters at terms estimated to be at a discount to benchmark treatment charges due to the quality of the concentrate. The smelter is expected to pay for 85% of the zinc content in the concentrate subject to a minimum deduction of eight units. No smelter penalties for the zinc concentrate is expected.

The lead concentrate grade is expected to range from 57% to 65% Pb. The smelter is assumed to pay for 95% of lead content in the concentrate subject to a minimum deduction of three units. No smelter penalties for the lead concentrate is expected.



Osisko Metals has no current contracts for project development, mining, concentrating, smelting, refining, transportation, handling, sales and hedging, forward sales contracts or arrangements.

25.12 Capital Costs

The capital cost estimate was prepared by BBA based on inputs from WSP, Hydro-Ressources and Osisko Metals. The capital cost estimate for the PEA has an expected accuracy range of $\pm 30\%$ and is expressed in Q2-2022 CAD.

Table 25-1 shows the capital costs breakdown for the Project. Initial capital costs are \$ 653.3 million, including an \$87.8 million contingency.

Sustaining capital costs are \$616.8 million excluding reclamation and closure.

Reclamation and closure costs are \$68 million.

The estimate includes direct and indirect costs (such as engineering, procurement, construction and start-up of facilities), as well as the Owner's costs and contingency associated with mine and processing facilities, and on-site and off-site (Pine Point Junction) infrastructure.

Table 25-1: Project LOM capital cost summary

WBS	Cost area	Pre-production capital cost (\$M)	Sustaining capital cost (\$M)	Total cost (\$M)
000	General Administration (Owner's costs)	22.8	0.0	22.8
200	Underground Mine	0.0	183.3	183.3
300	Open-pit Mine	15.7	80.6	96.3
400	Electricity and Communications	45.7	19.3	64.9
500	Site Infrastructure	59.7	11.8	71.5
600	Process Plant	297.3	0.0	297.3
800	Tailings, Mine Waste and Water Management	47.7	123.6	171.3
900	Indirect Costs	76.6	0.0	76.6
999	Contingency	87.8	23.7	111.5
	Capitalized Operating Costs ¹	0.0	174.5	174.5
	Total	653.3	616.8	1,270.1
	Site Reclamation and Closure	0.0	68.0	68.0
	Salvage Value	0.0	-19.6	-19.6
	Total - Forecast to spend	653.3	665.1	1,318.5

1. The operating costs attributable to overburden and waste stripping that is in excess of the average strip ratio over the life of mine have been transferred to capital expenditures.



25.13 Operating Costs

The operating cost estimate ("OPEX") is based on a combination of experience, reference projects, budgetary quotes and factors as appropriate for a preliminary study. The target accuracy of the operating cost is +/-30%. No cost escalation or contingency has been included within the operating cost estimate.

The operating cost estimate in this Study was prepared by BBA based on inputs from WSP, Hydro-Ressources and Osisko Metals. The operating costs include the costs to mine, process the mineralized material to produce lead and zinc concentrates, supplementary power generation as well as general and administration expenses ("G&A"). Underground mining will be performed on a contract basis while all supplementary power is supplied using rented generators.

The average operating cost over the 12-year mine life is estimated to be \$191.1M/y or \$48.90/t processed. Total LOM and unit operating cost estimates are summarized in Table 25-2.

Table 25-2: Project operating costs

Cost Area	Total cost (\$M)	Annual average cost (\$M)	Average cost (\$/tonne milled)	OPEX (%)
Open-pit Mining ⁽¹⁾	843.1	70.3	18.0	37
Underground Mining (West and Central Zones) ⁽²⁾	376.4	31.4	8.0	16
Processing	575.3	47.9	12.3	25
Power	216.3	18.0	4.6	9
Tailings, Waste and Water Management	76.4	6.4	1.6	3
General and Administration	380.6	31.7	8.1	17
Capitalized Operating Costs ⁽³⁾	-174.5	-14.5	-3.7	-8
Total (mine site)	2,293.6	191.1	48.9	100

1. Open pit mining: \$3.36/ tonne mined
2. West Area underground mining: \$40.32/tonne mined (years 3 to 8)
Central Area underground mining: \$53.92/tonne mined (years 7 to 10)
3. Operating costs attributable to overburden and waste stripping that is in excess of the average strip ratio over the life of mine have been transferred to capital expenditures.



25.14 Indicative Economic Results

The Preliminary Economic Assessment is partly based on Inferred Mineral Resources that are considered too speculative geologically to have the economic considerations applied to them that would enable them to be categorized as Mineral Reserves, and there is no certainty that the PEA based on these mineral resources will be realized.

The financial analysis uses a DCF methodology and an NPV8% as the base case. The zinc price and lead price that was used are US\$1.37/lb and US\$0.96/lb, respectively. The CAD:USD exchange rate was 1.27. All monetary amounts are presented in constant Q2-2022 Canadian dollars. The Preliminary Economic Assessment is based on 100% equity financing. No escalation or inflation has been applied. All amounts are in real terms.

Indicated and Inferred Mineral Resources account for 21% and 79% of total mill feed, respectively.

Based on a preliminary marketing plan developed by Stonehouse Consulting (Vice, 2022), the lead and zinc concentrates will be sold to a mix of North American and Asian smelters and on-the-spot markets.

A construction period of approximately 16 months was considered (starting in Q4 2026) for the overall project implementation based on a preliminary execution plan and schedule. Commissioning and ramp-up of the process plant to 75% capacity is planned to begin in Q3 2028. 2028 is considered year 1 of the 12-year LOM plan.

Taxation and royalties modelled within the financial analysis are based on the tax rates and taxation schemes validated by an outside Taxation expert.

The project is forecasted to generate a pre-tax NPV of \$1098.7M at an 8.0% discount rate, an IRR of 35.1%, and a payback period of 2.8 years. The financial analysis results showed an after-tax NPV of \$602M at an 8.0% discount rate, an IRR of 25%, and a payback period of 3.8 years.

Cash costs (C1) are estimated at US\$0.61 per payable zinc equivalent pound. All-in sustaining costs ("AISC") are estimated at US\$3,799M over the LOM, which equates to \$0.71 per payable zinc equivalent pound.

A sensitivity analysis was performed considering variations in commodity prices, exchange rate, feed grades, initial capital and operating costs on the after-tax NPV8% and on IRR. The project is most sensitive to fluctuations in USD:CAD exchange rate, zinc price, and, then to zinc feed grade. The project is less sensitive to changes in operating costs, capital costs, lead price and lead feed grades.

25.15 Project Risks and Opportunities

BBA, PLR Resources, WSP and Hydro-Ressources are 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 information discussed herein or the right or ability to perform future work on the Pine Point Project. Nonetheless, as with most mining projects, there are some inherent risks that could affect the economic viability of the Project. Many of these are due to a lack of information and can be managed as more sampling, testing, modelling, design and engineering are conducted. There are also significant opportunities that could mitigate some of the risks and potentially improve the Project economics, schedule, and environmental and social impacts.

Table 25-3 and Table 25-4 present the main risks and opportunities specific to the Project identified for this phase of the Project, excluding the external risks and opportunities such as the metal prices, exchange rate, political situation and government legislation that typically apply to all mining projects. The table lists the 2020 risks & opportunities (R/O), as well as 2022 update comments, and is described as follows:

“Retired”: the R/O is no longer relevant or has been eliminated via the implementation of the 2020 mitigation plan.

“Reduced”: changes in context and/or implementation of the 2020 mitigation plan led to a reduction of likelihood and/or severity of the risk. However, the risk still exists.

More detailed updates and comments are included, when relevant.

If no addition is made, the R/O remains unchanged since the 2020 PEA.

The PEA results 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 study results 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 and mine dewatering requirements estimates and approach;
- 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.



Additional work recommended to mitigate the risks and benefit from the opportunities identified so far are presented in Chapter 26.



Table 25-3: Project risks (preliminary risk assessment)

Area	Risk Description and Potential Impact	Mitigation Approach
Geology and Mineral Resources	- The M40 deposit area was depleted using historical plans for the underground voids. It is possible that unreported mined-out stopes are to be found. Also, remnant pillars were kept in the MRE and might present a challenge during future operation. - Local possible displacement due to unrecognized faults.	- Find additional historical information about past mining at M40 and/or plan additional investigation if needed. - A geo-structural model would help mitigate such risks. It must be noted that such possible local displacements are unlikely to have a material impact on the MRE and on eventual open-pit mining, but could be more challenging in underground operations.
Geotechnical and Mine Stability	- Rock mass characterization may be different than expected and influence pit slope, stope dimensions, pillar recovery, ground support and mining production sequence. - Geological structures are not fully known in some areas and have not been taken into account in the PEA mine design, which may impact dewatering capacity and mining production sequence.	- Gather additional geotechnical data, especially in areas where there has been no historical mining, further assess rock mass characterization and review mine design. - Gather additional geological and structural data and review the dewatering estimates and mine design. 2022: Initial investigations took place (from 2021 to 2022) into the impact of geological structures on hydrogeological characteristics and mineralization trends. Further investigations will have to be conducted during FS.
Hydrogeology and Mine Dewatering	Dewatering requirements may be underestimated due to oversimplification of dewatering estimates and lack of geological and hydrogeological information. 2022: Retired.	- Refine dewatering estimates by conducting: - Additional geological, geotechnical and hydrogeological investigation; 2022: Completed, with further investigation planned during FS.



Area	Risk Description and Potential Impact	Mitigation Approach
		■ More detailed methods for simulating local conditions and integration of more detailed and locally specific data; 2022: Completed, with further investigation planned during FS. ■ 3D numerical modelling to reduce uncertainty related to complex hydrogeologic settings; 2022: Completed, with further investigation planned during FS. ■ Additional pumping tests, particularly in areas where no mine dewatering has previously been performed; Groundwater monitoring to provide or confirm groundwater elevations and improve aquifer thickness estimates; 2022: Completed, with further investigation planned during FS. ■ More refined recharge estimates to reduce uncertainty due to recharge from precipitation. 2022: Cancelled ■ 2022 update: - Hydro-Ressources Inc has been hired by Osisko Metals to replace hydrogeological estimates with actual specific measurements. - Led by Michael Verreault, this new approach has brought a 21% power reduction for dewatering



Area	Risk Description and Potential Impact	Mitigation Approach
	2. The impacts of dewatering at the surface may not be fully captured due to the analytical model not considering groundwater-surface interaction. 2022: Retired due to new simulation approach. 3. Dewatering water becoming more saline over time due to groundwater flux from underlying geological units bearing water of higher salinity/TDS. 2022: Retired. Water not saline, as proven by data. 4. The number of dewatering wells may be underestimated due to the heterogeneous nature of the aquifer. 2022: Reduced. 5. Pumped groundwater disposal capacity may not be sufficient due to the re-infiltration rate in the available nearby mined-out pits and re-injection boreholes. Re-infiltration capacity may reduce over time due to regional groundwater level increase caused by the infiltration of water pumped into nearby mined-out pits and boreholes and/or conveyance and disposal of high TDS/TSS water in pits that may lead to sedimentation and/or solute precipitation in the pits. 2022: Retired: water would be discharged after treatment to the environment	2. Develop a groundwater management plan to assess baseline conditions including low-flow conditions prior to operation. 2022: Retired. 3. Perform detailed groundwater quality analysis as part of the groundwater monitoring plan 2022: data collection completed. Further analysis to come for FS and EA (Environmental Assessment) 4. Develop a dewatering contingency plan including contingency equipment and pumps. 2022: Field tested has proven preferential flow through faults increasing the potential pumping yield of future wells. Monitor the reinjection capacity and develop a groundwater disposal contingency plan. 2022: Using ditches and settling ponds/pits for dewatering infrastructure prior to discharging to the environment is planned to be investigated further. The water will meet water quality requirements before discharging. Artificial recharge is expected only on the North-East trend. 5. Pipe the pumped groundwater farther from the pits to be dewatered and plan for contingency reinjection wells and pumps. 2022: Simulation results suggest an absence of pore pressure due to high permeability.



Area	Risk Description and Potential Impact	Mitigation Approach
	6. Elevated pore pressure may cause geotechnical instability leading to potential slope failures and resulting injuries and damages. 2022: Retired. 7. Dewatering may be delayed or extended in areas requiring groundwater control measures prior to dewatering due to grouting not being as efficient and/or encountering unexpected voids or conditions and requiring more time, materials and/or equipment than estimated. 8. An additional water treatment plant could be required to ensure the water discharged to the environment meets the applicable water quality requirements.	6. Develop a contingency plan including contingency materials and equipment 7. Develop a detailed water balance and water management plan that considers water quality within the next steps of engineering. 8. Continue water quality sampling and baseline studies during the FS.
Open-pit Mining	1. Changes to current geotechnical and hydrogeological parameters may affect mine design and capacity to reach production targets and the mining production schedule. 2022: Reduced. 2. Overburden and waste rock volumes may be underestimated due to a lack of geological data. 3. Mineralized material may be overestimated due to higher dilution and lower mining recovery than estimated.	1. Conduct additional geotechnical and hydrogeological investigations, especially in areas where there has been no historical mining, perform additional modelling and simulations, and review mine design. 2022: Hydrogeological investigations completed. Additional geotechnical, modeling simulations and review of mine design are required during feasibility. 2. Conduct additional geological and geotechnical investigations. 3. Review dilution and mining recovery for each pit and adjust mine plan and mine equipment selection (use a smaller shovel for selectivity).
Underground Mining	1. Changes to current geotechnical and hydrogeological parameters may affect mine design and capacity to reach production targets and the mining production schedule. 2022: Reduced.	1. Conduct additional geotechnical and hydrogeological investigation, especially in areas where there has been no historical mining, perform additional modelling and simulations and review mine design.



Area	Risk Description and Potential Impact	Mitigation Approach
	2. Grout curtain pattern and grout estimate failure to reduce water infiltration to design level or higher grouting cost. 2022: Reduced. 3. Grout curtain failure during mine development and operations, resulting in groundwater inflows and potential delays and mining interruption. 4. Mineralized material may be overestimated if the dilution is higher/or and mining recovery is lower than estimated.	2022: Hydrogeological survey completed. Further analysis of geotechnical aspects will be required during FS. 2. Test the grout pattern and recipe on a small scale on site. 2022: Further analysis required during FS. 3. Conduct continuous water level monitoring during operations. 4. Review stope design, cut-off grade, and add diamond drilling if required.
Mine Site Infrastructure	1. National highways disturbance, due to proximity of some pits and shallow underground workings. 2022: Retired. 2. Foundation design and preparation may be underestimated due to limited geotechnical data available.	1. Ensure safe operating procedures, especially for blasting, and develop control and monitoring plans, including regular inspections and use of road signs. 2022: Retired. 2. Carry out site-specific geotechnical investigations and review design accordingly.
Mineral Processing and Metallurgy	1. Grinding circuit sizing due to lack of comminution testing. Grinding power requirements could be higher than expected. 2. Limited testwork on N204. The metal recovery estimates could be lower than the current assumptions. 3. Production of fines by the crushing circuit could be higher than estimated. Fines from the crushing circuit will bypass the mineral sorting stage and higher tonnage will report directly to milling.	1. Perform comminution testwork to confirm the grindability estimations. 2. Perform further testwork on N204. 3. Perform blasting and crushing simulations to identify the operating parameters to minimize fines production.



Area	Risk Description and Potential Impact	Mitigation Approach
	- Mineral sorting equipment capacity was based on an atypical feed size range. Additional capacity might be required. - The impact of bitumen/oil residue presence on flotation performance was not evaluated. The flowsheet may need to be modified. - The process plant capacity is based on a certain grade range. Large variations in feed grade will result in poor metallurgical control and production variation (quality and tonnage).	- Perform mineral sorting tests on the atypical feed size range. - Perform flotation testwork to simulate the impact of bitumen/oil presence on flotation performance. - Develop a strategy to control the feed grade variations. Review design factors to be in line with mine plan variability.
Process Plant	- Main Process building may be undersized if a larger Ball Mill is required. Piping has been sized and costed based on similar projects. Piping requirements and building space may be underestimated. - Total equipment electrical load may be underestimated.	- Actual allocated space for equipment is conservative. The actual budget allocated for minor changes, should not impact the global building sizes. - See the electrical Infrastructure mitigation solutions.
Electrical Infrastructure	- Size of mobile gensets may be small in comparison to pit power demand, increasing the footprint of the pads and project costs. 2022: Retired (LNG tailored to requirements). - Heavy LNG trailer traffic on the pit roads may increase the risk of accidents. - The existing NTPC substation on the Pine Point Project may need additional upgrades to accommodate the Project's estimated loads. 2022: Realized – cost born by NTPC - LNG distribution dependant on one supplier.	- Engage multiple vendors to identify the most appropriate setup. 2022: Completed. - Ensure dedicated lanes for tractor-trailer traffic, strict traffic rules, trained tractor-trailer drivers, etc. - Assess the existing infrastructure to identify adaptability and possible upgrade requirements of the sub-station. (Already discussed with NTPC representatives and to be finalized during next phase of the Project.) 2022 action to upgrade. - Discuss with other local LNG consumers to create a list of potential suppliers



Area	Risk Description and Potential Impact	Mitigation Approach
	5. Power supply is dependant on public road access. Risk of road damage or blockage due to the environment may impact LNG distribution. 6. The quantity of available NTPC power for the Pine Point Project is based on ongoing NTPC upgrade projects at the Taltson generation station 2022: Realized – NPTC will provide 8.5 MW average 7. NTPC power supply is dependant on 1 power generation station. 2022: Retired. 8. Delivery time for generators have increased significantly (not risk – fact).	5. Ensure proper LNG storage on-site or at Hay River will be considered to limit this risk and provide multiple days of fuel supply independence. 6. Follow the Project closely and size the on-site stationary power generation according to NTPC power availability. 7. Develop a pre-arranged agreement with the mobile genset supplier to be prepared for a possible long-term loss of NTPC power supply.
Mine Waste and Water Management	1. WR and Overburden Pile ("OB") capacity might be inaccurately estimated. Provided space for WR management may be insufficient. 2. OB piles could require lower slopes and thus larger footprints. 3. Location of WR piles and OB piles might need to be modified with respect to volumes, and/or geochemical performance. 4. WR and mineral sorter rejects could be metal leaching, which may impact surface and groundwater quality and limit the potential to reuse this material as construction and cover material. 5. Abundance of water (groundwater and surface) could lead to an overall decrease in a flood management capacity for surface water	1. Site-specific geotechnical investigations will allow for establishing and confirming individual overburden and WR volumes. 2. Site-specific geotechnical investigation (test pits) will help in defining overburden composition and confirming parameters. 3. Site-specific investigations will also confirm volumes and quality. Historical data does not show major issues with metal leaching or acid rock drainage ("ARD"). 4. Carry out WR and Mineral Sorter geochemical characterization studies to determine expected long-term performance as construction and cover material. Review stockpile designs and water treatment requirements if needed. Investigation for alternative borrow sources will decrease this uncertainty. 5. Integrated water balance and sensitivity analyses will determine if water storage and treatment should be increased. An integrated water management plan



Area	Risk Description and Potential Impact	Mitigation Approach
	management. 6. Areas of discontinuous permafrost could be detected and impact the location of waste rock or overburden piles and also the quality of the overburden material. 7. Underestimation of concentration of contaminants and volume to be treated may lead to underestimating water treatment unit size (CAPEX) and treatment cost (OPEX).	will be developed in the next phase of Project development. 6. Field investigations will allow for the detection of any permafrost areas, other locations around open-pits will also be investigated, if need be, to find better individual locations. 7. Complete a comprehensive geochemical characterization program and a contaminant transport model specific to surface water and refine estimates using a detailed site water balance as the Project proceeds.
Tailings Disposal Areas (TDA)	1. Tailings could be metal leaching and impact reclaim water quality. 2. TDA's capacity may temporarily be reduced due to surface water (recharge), especially during freshet, ice lenses, and/or difficulties to reach the tailings density target. 3. Freezing of the tailings or reclaim water pipelines could cause a temporary stoppage of processing in winter conditions. 4. TDA development schedule may be impacted by slower settlement or difficulties to reach tailings density targets. Extending the tailings' transport pipeline and water reclaim lines could be required earlier, affecting the sustaining CAPEX.	1. Perform tailings geochemical characterization and studies to determine pore water quality. Sample and analyze water in each of the open-pits identified for tailings disposal. Perform water quality modelling and review water treatment requirements if needed. 2. Review water and mass balance and develop a detailed tailings deposition plan including monitoring and contingency plans and equipment. 3. Winter operation has been taken into account in costing. Careful planning of the deposition sequence and allowing for the contingency of the temporary disposal of tailings in areas closer to the process plant would avoid disruption of operations. Appropriate protection of the pipelines is provided and needs to be confirmed in the next design phase. 4. Detailed scheduling will confirm the filling sequence, and contingency plans will be put in place to allow preparedness in terms of cost and materials.



Area	Risk Description and Potential Impact	Mitigation Approach
	5. Potential Fish habitat in historical open-pits may require further characterization and could lead to delays in the permitting process.	5. Complete the baseline studies and work on mitigation and compensation opportunities.
Construction (Costs and Schedule)	1. Possible shortage of lodging if delays or excessive additional hoarding is required to support schedule pushing through winter of 2024. 2022: Retired due to schedule delay. 2. Additional costs related to increased heating and hoarding for winter works. 2022: Retired due to schedule delay. 3. Excessively inclement weather. 4. New normal after COVID 19 pandemic resolution resulting in reduced production level. 5. Heating up of equipment manufacturing market causing delays in long lead equipment availability.	1. Improve permitting schedule by optimally at least a quarter. 2022: Retired. 2. Heating and hoarding costs can be attenuated if either permitting delays are improved or construction start is delayed to Q2 of 2024. This would delay plant construction completion to Q3 2025. 2022: Retired. 3. Advance planning process to support schedule risk analysis and incorporate results in the plan. 4. Track construction market and reaction to pandemic to incorporate results in the plan and capital cost estimate. 5. Prepare to incorporate early (during FS) the procurement of long lead equipment.
Environmental, Permitting and Social Licenses	1. EA and Permitting Schedule may be longer than planned. This would delay the start-up of construction.	1. Prepare key documents in advance of receiving specific direction from MVEIRB. This may require some rework once the Terms of Reference are available. Assume Project is going to EA (95% likely) and skip screening stage. Ensure the Project team is appropriately resourced to respond rapidly to Information Requests ("IRs") and project changes. 2022: Terms of Reference (TOR) have been released by the MVEIRB. The TOR contains additional requirements in comparison to previous projects which will potentially extend the time required to prepare the DAR. Ensure the project team is resourced to address the DAR requirements and



Area	Risk Description and Potential Impact	Mitigation Approach
	2. EA process may be delayed if there are significant changes to the Project requiring to be addressed in the DAR. 3. Delay in negotiating Impact and Benefit Agreements ("IBA") with nearest impacted communities may result in failure of majority of communities to support EA or Permitting Public Hearings. 4. While the Project is not expected to be liable for the existing disturbances from historical mining, there are local areas where the Project overlaps previously disturbed sites and PPML may be required to reclaim the site to its natural productive capacity requiring additional costs.	respond rapidly to information requests and project changes. Advance water management planning takes into account environmental and permitting considerations, including potential for mine water discharge to the environment and treatment requirements. Continue engagement with impacted Indigenous parties and regulators to address issues as early as possible. 2. Ensure that the EA team works closely with the engineering team to get advance warning of likely changes and provide advice on minimizing impacts to the DAR, including on water management planning. Freeze design as early as possible to provide basis for environmental assessment of impacts. 3. Commence IBA discussions as soon as possible and develop an IBA framework with impacted communities. 4. PPML is in the process of obtaining clarification from the GNWT that lands that are not directly disturbed by PPML's proposed activities will not be PPML's responsibility. 2022: PPML is continuing to work with the GNWT on existing liabilities. Maintain the position that the extent of reclamation required for previously disturbed areas that are reused by the project will be negotiated similar to that previous to the Project development. Continue engagement with Indigenous parties and regulators during the development of the Closure & Reclamation Plan for the EA & Permitting phases.



Area	Risk Description and Potential Impact	Mitigation Approach
	5. Significant public opposition to the Project may require additional engagement and a possible redesign of parts of the Project and this may result in delays and additional costs.	5. Regular engagement with communities and regulators to explain project, listen to concerns and propose mitigations.
Rehabilitation and Closure	1. WR, mineral sorter rejects and tailings are not foreseen to be acid generating but could be metal leaching. A review of the rehabilitation and closure plan might be necessary. 2. Organic soil suitable for revegetation may be limited and impact the rehabilitation plan. 3. Additional environmental liabilities related to historic mining operations could result in additional closure costs.	1. Perform WR, sorter rejects and tailings geochemical characterization and conduct studies to predict potential contaminant migration. Provide mitigation measures at the onset of the deposition. 2. Investigate if there are organic soil borrow sources locally and conduct revegetation studies. Develop relationships with research groups investigating re-vegetation for locations in the NWT. 2022: investigate other sources of organics e.g. municipal composting programs. 3. Identify and discuss liabilities and closure expectations with the regulatory authorities early in the permitting stage.
Land Tenure	1. Pine Point Junction land not currently owned by OM/PPML.	1. Conclude an agreement with CNR (current owner of the land). To be reviewed during FS and perform trade off study between Pine Point Junction and the Enterprise site.



Table 25-4: Project opportunities

Area	Opportunity Explanation	Benefit
Geology and Mineral Resources	1. Potential to add to the Mineral Resource Estimate by drilling deposit extensions and exploration targets. 2. Potential to upgrade inferred mineral resources to the indicated category by additional drilling and database upgrades.	1. Increase in mineral resources. 2. Increase in mineral resource confidence possibly allowing to conduct an economic study on a larger Indicated Mineral Resource.
Geotechnical and Mine Stability	1. Steeper overburden and pit wall slopes by better characterization, including use of drone footage to measure structural geology in historical pits. 2. Better rock mass classification for underground design. 3. Overburden characterization of West Zone	1. Potential reduction in strip ratio resulting in decreased OPEX. 2. Potential bigger long hole stope and higher recovery in room and pillar stopes. 3. Optimization and localization of portal design and cost.
Hydrogeology and Mine Dewatering	1. Mining Production Schedule optimization to further account for dewatering requirements. 2022: Simulation results are showing significant effect of optimizing the mine plan based on dewatering strategy. 2. Additional groundwater control measures prior to dewatering (consider grouting in other areas and/or other approaches). 3. Advanced groundwater 3D modelling to assess re-injection and aquifer capacity to reduce pumping distances for groundwater discharge. 4. 2022 update from Andrew Williams (Environment): Proposed methods for management of ground water are still largely conceptual and require further work to confirm. This will be a key component of the environmental assessment.	1. Possible reduction of dewatering requirements due to cumulative drawdown effect and potentially reduce OPEX and CAPEX. 2022: Expecting 15% dewatering reduction during FS. 2. Potential to reduce dewatering requirements in some areas, possibly resulting in lower OPEX and CAPEX. 3. Potential reduction of piping and reinjection pumping equipment possibly resulting in lower CAPEX. Potential reduction in grouting and pumping requirements for underground. 4. 2022 update from Andrew & MV: Continue to conduct proposed hydrogeological testing at site.



Area	Opportunity Explanation	Benefit
	2022 (from MV - Hydrogeology): Current simulation results are considering re-injection in the North trend. 5. 2022: High simulated in-flow on the North trend is suggesting an over-estimation of the predicted flow rates.	5. 2022: Additional field testing may lead to significant reduction on the predicted flow rate on the North trend and west area.
Open-pit Mining	1. Automation equipment and technology. 2. Trade-off study evaluating the use of mining contractors vs owner operated fleet. 3. Mining Production Schedule optimization to further account for pit proximity, consider combining cluster pits. 2022: Reduced 4. Use smaller equipment in some areas to reduce dilution, especially for thin tabular deposits. 5. Purchase of used equipment.	1. Potential reduction in OPEX. 2. Possible savings in OPEX and CAPEX. 3. Possible reduction in OPEX. 2022: hydrogeological measurements and simulations completed during 2022 PEA. Sizeable reduction of dewatering and related power. Further development required during FS. 4. Potential to increase revenue by reducing dilution. 5. Possible reduction in CAPEX.
Underground Mining	1. Automation equipment and technology. 2. Trade-off study to evaluate mining lower grade mineral resources later in the LOM.	1. Possible reduction in OPEX. 2. Potential to increase mineral resources and revenue.
Mineral Processing and Metallurgy	1. Additional testwork on mineral sorting and flotation to optimize circuits. 2. Trade-off study evaluating pre-concentration by mineral sorting vs dense media separation.	1. Possible reduction in OPEX and CAPEX, improved metal recoveries. 2. Potential savings in CAPEX and OPEX.
Process Plant	1. Investigate alternative processing flowsheets and technologies. 2. Layout optimization to reduce building space. 3. Integrated operations/automation study to identify project improvement opportunities.	1. Possible reduction in OPEX and CAPEX. Reduction in building size requirements. Improved metal recoveries. 2. Possible reduction in CAPEX. 3. Possible reduction in OPEX. Improved control of operations.



Area	Opportunity Explanation	Benefit
Plant site Infrastructure	1. Consider purchasing used permanent camp modules. 2. 2022: Consider transportation of workers daily to site from local communities to reduce permanent camp requirements. Benefit: reduced costs, carbon footprint and site disturbance.	1. Reduce capital costs and material delivery time 2. Conduct a trade-off study or permanent camp vs. daily transportation. Consult with local communities to discuss the options and determine their preference.
Electrical Infrastructure	1. Stationary power plants to be owned and operated by the public utility to support the public network as well as the Project. 2022: retired 2. PPML to own and operate the stationary power plants, rather than rent. 3. Heat recovery from stationary power plants. 4. Competitive cost analysis of the various LNG energy supplier options.	1. Potential to share CAPEX and OPEX cost. 2022: retired 2. Reduce OPEX costs and overall NPV of the electrical infrastructure costs. 3. Possible to offset use of boilers at the process plant and underground mines. 4. Potential savings in OPEX.
Mine Waste and Water Management	1. Sequenced progressive closure using WR, overburden and mineral sorter rejects material.	1. Potential to reduce the number and/or the size of stockpiles and water drainage systems, resulting in lower OPEX and CAPEX.
Tailings Disposal Areas (TDA)	1. Use closer and/or higher capacity proposed pits once mined-out if Mining Production Schedule allows.	1. Potential reduction in pumping and hydraulic transport equipment, resulting in lower CAPEX.
Construction (Costs and Schedule)	1. Investigate possibilities to reduce the environmental permitting duration to allow construction to begin earlier. 2. FS phase project execution and constructability reviews to optimise plan and costs. 3. Investigate the use of pre-assembled buildings or modules.	1. Minimize winter construction, improve efficiency and lower CAPEX. 2. Reduce the construction duration and minimize work during winter. Lower CAPEX. 3. Reduce the onsite construction effort and minimize work during winter.
Environmental, Permitting and Social Licenses	1. Investigation of the possibilities to further benefit from the Project's brownfield or pre-disturbed site status	1. Reduce reclamation and closure costs as the Project only has to return sites to their current condition if



Area	Opportunity Explanation	Benefit
	which enables development on previously disturbed areas.	they are previously disturbed.
	2. Consult with local communities on re-vegetation approach.	2. May reduce reclamation and closure costs.
Rehabilitation and Closure	1. Progressive closure tightly sequenced with Mine Production Schedule.	1. Potential to reduce material handling and water diversion requirements and duration resulting in decreased OPEX.
	2. Investigate the possibility of using historical WR and OB stockpiles as construction and closure material.	2. Possible to reduce transportation distances in some areas.

26. RECOMMENDATIONS

This NI 43-101 Preliminary Economic Assessment ("PEA") report was prepared by experienced and qualified independent consultants using recognized engineering standards. Based on the results of this study, the QP's recommend that the Pine Point Project proceeds to the next phase of the Project development in the form of a feasibility study and that additional exploration and engineering work be conducted. Fieldwork programs should be planned in order to gather data for both geotechnical and hydrogeological studies as well as to include sampling for metallurgical testing and environmental characterization so that refined design data is available to support future economic studies.

This section provides a summary of the QP's recommendations based on the results of this PEA and findings for each major area of investigation.

26.1 Geology and Mineral Resources

The activities related to geology and Mineral Resources are recommended in the following sections.

26.1.1 Geological Modelling and Target Evaluation

Complete detailed 3D models of the geological setting, alteration, mineralization trends, structural influences, collapse structures and incorporate both airborne and ground-based geophysical interpretations and inversion models. In addition, complete detailed digital 3D models for each deposit area, pit design and underground engineering design models. Use these models to facilitate exploration target generation and prioritization.

26.1.2 Additional In-fill Definition Drilling

To achieve a drillhole spacing of approximately 30 m, or better, and to upgrade the entire pit constrained and underground Inferred Mineral Resources across the PPMC to the Indicated category, approximately 1,900 drillholes totalling approximately 70,000 m is required. Note that approximately 30,000 m in approximately 450 holes of this proposed 70,000 m program has already been drilled as of the effective date of this Report.

26.1.3 Exploration

The following activities related to exploration are recommended:

- Collect petrophysical data on three to five drillholes from tabular and prismatic deposit types to optimize reconciliation between ground and airborne geophysical surveys;
- Investigation of structural controls on mineralization using pit mapping, photogrammetry, LiDAR topographic elevation data, and historical airborne electromagnetic and magnetic survey datasets;
- Fly a project wide electromagnetic survey optimized for detection of mineralization;
- Complete airborne gravity gradiometry over the eastern end of the Project area;
- Conduct Induced Polarization ("IP") geophysical surveys using modern equipment and referencing current results over in situ tabular and prismatic mineralization to define targets identified by the 2019 airborne gravity gradiometry survey, features identified in the LiDAR and compiled data (orphan holes with unconstrained grade intersected);
- A brownfield exploration drilling program totalling approximately 20,000 m in 250 drillholes is recommended to follow up on priority targets developed from the initial targets generated by the airborne gravity gradiometry, and LiDAR depressions potentially representing favorable collapse breccia features and compilation work conducted in 2019;

A total program of approximately 1,700 drillholes for a total of approximately 60,000 metres is proposed using an average cost of \$300 per metre drilled. The per metre rate is based on all costs related to drilling (i.e., drilling, moving between holes, drilling additives, mob/demob, accommodation, meals, transportation, fuel, assay, core logging, sample preparation, reclamation as well as environmental health and safety).

26.2 Geotechnical and Hydrogeological Studies

Additional and geotechnical and hydrogeological characterization and studies are required to reduce uncertainties and refine the design parameters and assumptions to support future economic analysis and mine operations.

Dewatering and water management has been identified as being a key factor. More detailed knowledge of the hydrogeological conditions of the site as well as for each deposit area is required. The Project benefits from an extensive hydrogeological database including real time dewatering measurements. The knowledge base will increase by completing the tasks as follows:

- Implementing a field investigation program, particularly in the areas where no mine dewatering had previously been performed. The field work program should include

additional pumping tests to improve the hydraulic conductivity estimates across the site, reduce uncertainties and help to refine dewatering estimates. A thorough groundwater monitoring program would also provide updated groundwater elevations and improve aquifer thickness estimates.

- Building and running 3D numerical model to better simulate local conditions using locally specific data and aquifer properties from pumping tests to predict more reliable groundwater flow estimates. Run various dewatering sequences for several mine schedules to identify the most attractive schedule from economical and technical points of view. This model will also allow for the evaluation of depressurization impacts from dewatering upon the environment and assess the re-infiltration capacity for water disposal.

Additional efforts should also include geotechnical work regarding Project infrastructure and the open-pit and underground geotechnical designs. Given the brownfield nature of the Project, a large project database is available and will continue to be fully used. Based on the current understanding, no particular issues are anticipated. There could be opportunities to steepen the pit slopes and increase the underground mine stopes and recovery. Additional work recommended includes:

- 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 and underground mine design will require complementing oriented core drilling, especially in areas where there has been no historical mining. Results will be used to define the appropriate slopes for overburden excavations, verify stability for all impoundments and provide or confirm parameters for the open-pit and underground designs.
- Photogrammetry surveys of all pit wall exposures should also 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.

Fieldwork programs should be coordinated in order to gather data for both geotechnical and hydrogeological studies as well as include sampling for metallurgical testing and environmental characterization. The QP's recommend the following to be included in the geotechnical and hydrogeological field programs to confirm design criteria for the site surface infrastructure, open-pit mines and underground mines:

26.2.1 Mine Dewatering

- A three-dimensional numerical model using aquifer properties from pumping tests at a given proposed pit is recommended to predict more reliable estimates for the individual mineral resources of the Pine Point Project;
- Additional pumping tests performed within the Sulphur Point Formation are recommended to improve hydraulic conductivity estimates across the site, reduce uncertainties, and help refine dewatering estimates;
- A thorough groundwater monitoring program is recommended to provide updated groundwater elevations at all proposed pit locations and improve aquifer thickness estimates;
- Integration of surface recharge in the three-dimensional groundwater flow model;
- Open-pit and underground mine dewatering using a water reinjection technique back into the aquifer should be integrated in the hydrogeology model. Combined with the dewatering model optimization, this could potentially lead towards a significant reduction to pumping flows and lengths. The model should also determine aquifer capacity for reinjection and consider salinity aspects;
- Refine the hydrogeological model for underground deposits (both zones) with a more detailed hydrogeological study to validate the groundwater inflow rates;
- In the Central Zone, water inflow estimates should be updated in the next phase of work to consider inflow from the open-pit into the underground mines;
- Develop a hydrogeological model for the connection ramp between R190, P499, and O556;
- Refine the grouting design for pre-production dewatering;
- Include water quality modelling (contaminant transport) for environmental impact assessment.

26.2.2 Open-pit Deposits

- Perform a geotechnical drilling campaign to assess open-pit geotechnical domains and to confirm geotechnical rock properties.

26.2.3 Underground Deposits

- Review the geotechnical section of the UG deposits by performing:
- A geotechnical drilling campaign that will confirm the geomechanical characteristics of all geological formations relevant to underground mining;
- A geotechnical investigation for the portal location and for the first 500 m of the ramp;
- A geotechnical investigation in the form of geotechnical pilot holes for all major raises from surface to the targeted level underground;
- A geotechnical study to confirm the following points:
 - Dimensions of longhole stopes;
 - Dimensions of room and pillar stopes and pillar design;
 - Crown pillar dimension;
 - Overburden excavations design for portals;
 - Definition of rock support systems.

26.2.4 Waste Rock, Mineral Sorter Reject and Overburden Storage Facility

- A geotechnical investigation for waste and overburden stockpiles is recommended in order to assess soil conditions such as soil stratigraphy, depth to bedrock, permafrost, groundwater levels, and soil properties. Three to Five boreholes per dump going down 6-20 m below the bedrock surface is proposed. Tests pits are also recommended for soil characterization and complete foundation investigation. Boreholes and test pits will provide geotechnical and hydrogeological data that are required to adapt and optimize the design (geometry, stability and location) of infrastructure. Permafrost needs to be considered in these investigations;
- Optimize the in-pit management sequence of overburden in order to prioritize waste rock disposal and preserve enough OB material for other purposes, such as closure.

26.2.5 Infrastructure

- A specific geotechnical study must be performed to ensure stability of the on-site roads and highway near underground mines in W1 Area. Procedure during operation will be required when blasting;
- A complete geotechnical study is recommended to verify foundation conditions for the Camp Complex, Administration, Truck Shop and Warehouse buildings. The study should

include bearing capacity, soil characterization, bedrock depth, excavation slope, foundation type, and backfill material requirements.

26.3 Mining

26.3.1 Open-pit Mining

The following technical aspects should be addressed and are recommended for open-pit mining:

- Confirm historical drill patterns, if possible, to better assess/update production drill parameters;
- Map historical faces and inter-ramp slopes from existing mined deposits to better assess/update overall slope angles for main rock types. This will help to design optimized slope walls and potentially lead to more mineralized material access in pit bottom;
- Perform density measurements for main rock types (waste, mineralized material and overburden) in different areas to confirm densities;
- Conduct a trade-off study for a secondary crusher unit in N1 area and a conveyor belt to transport crushed mineralized material to the mill. This scenario should significantly reduce hauling costs since N1 area is located approximately 10 km from the mill and the total mineralized material tonnage from N1 accounts for 1/3 of total open-pit mineralized material tonnage.

26.3.2 Underground Mining

The following technical aspects should be addressed:

- Develop a detailed operational mine plan/schedule for pre-production and for the first years of underground mining;
- Conduct a scoping study/preliminary economic assessment to evaluate the possibility of applying a mineralized material sorting technology locally at the West Zone;
- Evaluate backfill binder content in the cemented rock fill recipe (laboratory testing);
- Re-assess the current haulage system (trucking with 45 t trucks), considering the short life span of underground deposits. The benefit of using smaller equipment allows for a reduction of the dimensions of the haulage drift, and a quicker development rate.

26.4 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 N204 and West Zones) of the PPMC to determine the most suitable process flowsheet:

- Pre-concentration:
 - Additional XRT mineral sorting and DMS testing;
 - Process technology selection trade off studies;
 - Investigate and confirm the percentage of fines which will be sent to the process plant.
- 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;
 - Locked cycle testing.
- 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.

26.5 Processing Plant

The following additional activities are recommended to adjust the process plant design to the refined mineral processing and metallurgy parameters as well as to mitigate the associated risks and benefit from potential opportunities:

- Confirm geotechnical conditions for the proposed site;
- Review the equipment sizing and the main building size and layout;
- Investigate alternative processing flowsheets and technologies;
- Review the total equipment electrical load requirements;

- Conduct integrated operations and automation studies.

26.6 Infrastructure

The following additional activities are recommended to complete the next phase for the following infrastructure items.

26.6.1 On-site Roads

- Conduct a survey of the existing road to identify and prioritize rehabilitation work to be executed;
- Identify non-PAG waste rock to be used for road rehabilitation and construction;
- Confirm Highway 6 crossing requirements, in terms of signs and visibility, with NWT authorities;
- Evaluate bearing capacity of foundation materials along the length of the new on-site roads to optimize construction.

26.6.2 Sewage System

- Conduct soil characterization near the Camp complex to confirm the drainage field location and efficiency.

26.6.3 Transloading Site

- Initiate a trade-off study to select the preferred site for the transload facility;
- Include the selected transloading infrastructure in the environmental impact assessment;
- Conclude an agreement with the owner of the land for the transloading site.

26.6.4 Communications

- A site survey should be conducted by an Internet supplier to confirm the point of connection at Hay River and optimal tower locations (line of sight).

26.6.5 Potable Water

- In order to optimize the treatment system for potable water, Great Slave Lake water characterization is required.

26.6.6 Power

- Track progress of the NTPC Taltson Dam power upgrade project on a regular basis and safeguard the NTPC power availability for the Pine Point Project;
- Establish the technical criteria with NTPC for the electrical interconnexion limits on their network (power factor, harmonics, flicker, current unbalance, etc.);
- Conduct additional trade-off studies to identify the optimized energy sourcing options for the on-site power generation:
 - Own/operate model compared to a rental/lease strategy;
 - Fuel source and procurement models, including evaluating a turn-key virtual pipeline contract or taking part ownership of the distribution network such as fuel transportation.
- Conduct a trade-off study to determine the size of the individual mobile gensets for a fleet best suited for the various pit energy needs;
- Conduct a trade-off study for the thermal energy production and distribution system optimizing the heat recovery options and minimizing energy costs.

26.7 Tailings, Mine Waste and Water Management

- A comprehensive geochemical investigation is required at the next step of engineering in order to confirm the water management, water treatment, tailings, mineral sorter rejects, and waste rock stockpiles design and strategies. The existing static analysis must be completed and enhanced with kinetic testing for all types of material;
- The estimated reclamation costs are highly dependent on the acid generating potential and metal leaching potential of tailings and waste rock. The existing geochemical studies focussed mainly on Waste rock geochemistry and showed 80 Non-PAG results over 82 samples. Additional geochemical analysis including kinetic testing is recommended for tailings, mineral sorter and Waste rock. In the case of a significant change to the acid potential or metal leaching results, the closure and reclamation concept will need to be adapted and reviewed, and the closure costs could be significantly modified;
- Additional testing for specific tailings density could lead to a change in the overall tailings volume requirement;
- Review of liner requirements in collection ponds beside the waste rock and overburden stockpiles;
- Water balance for the overall site and specifically around the Process Plant should be detailed;

- Integration of refined climate data analysis is recommended.

26.8 Environment and Permitting

Additional environmental activities are required to complete the ongoing baseline studies and to initiate and complete the Environmental Assessment ("EA") and permitting process, as per the MVEIRB *Draft Environmental Assessment Initiation Guidelines for Developers of Major Projects* (EA Initiation Guidelines; MVEIRB 2018).

Multiple environmental approvals and permits are also anticipated to be required following approval of the EA, as described in Chapter 20.

It is recommended to continue discussions and engagement throughout the Project with the local Indigenous and other communities to understand and respond to their concerns in the Project through mitigations.

In order to validate the in-pit disposal strategy for tailings and part of the Waste Rock, it is recommended to investigate the targeted historical pit lakes to determine which of them may have fish habitat.

For the site closure, a formal process of engagement with local communities is required to develop the objectives and evaluate options.

26.9 Market Studies and Contracts

Osisko Metals should continue to initiate and review Market Studies as the Project evolves so that the economics are in line with the current market conditions and the updated metallurgical testwork

Logistic and transport studies should be conducted to confirm costs and determine optimum locations for the zinc and lead concentrates.



26.10 Proposed Work Program and Budget

The costs for the recommended work program are estimated to be approximately \$62 M based on past project experience and current site costs. Table 26-1 provides a cost estimate summary for the required field work and studies to support the next phase of project development.

Table 26-1: Proposed Work Program Budget

Activities	Estimated Cost (\$)
Additional definition and exploration drilling	20,000,000
Geophysics, Assaying, relogging, compilation, and surveying	1,000,000
Geotechnical investigation and studies for open-pit	1,000,000
Geotechnical investigation and studies for underground	1,000,000
Geotechnical investigation for waste and water management	1,000,000
Geotechnical investigation for infrastructure	250,000
Hydrogeological investigation and studies	1,000,000
Metallurgical testwork	1,000,000
Power system trade-off studies	500,000
Feasibility study	8,000,000
Environmental baseline studies & Geochemical Characterization	1,000,000
Environmental Assessment (EA) & permitting	18,000,000
Market studies and contracts	100,000
Sub-total	53,850,000
Contingency (15%)	8,077,500
Total	61,927,500

27. REFERENCES

- Adair, R., 2020. Document "2020_06_30_Pine Point Geology Summary Report", dated July 30, 2020, 26 p.
- Advanced Terra Testing, Inc., 2006. Testing report. Includes lab tests for (bulk) density, (particle) specific gravity, moisture content, porosity, and compressive strength on core samples from drill holes W-85-TV02, W85-TV03, R-190-TV05, R-190-TV07 and R190-TV10.
- AeroQuest Limited, 2005. Report on a helicopter-borne AeroTEM II electromagnetic and magnetic survey, Consulting report prepared for Tamerlane Ventures Inc.
- Allen, C.F. and Walker, G.B., 1949. Beneficiation of Industrial Minerals by Heavy-media Separation. AIME, January 1949 Mining Transactions.
- AMC Mining Consultants Canada, 2012. Pine Point Geotechnical Review Phase 2 R190 Deposit. AMC Report #AMC711018.
- Babrock, E.A., 1973. Regional jointing in southern Alberta. Canadian Journal of Earth Sciences, v. 10, no. 12 p. 1769-1781.
- BBA Engineering, 2021. Pine Point Lead-Zinc Project Carbon Footprint Accounting. Report prepared for Pine Point Mining Limited.
- Beales, P., Woodward, R. and Campbell, C., 2002. Great Slave Reef, in A Guide to Mineral Deposits of the Northwest Territories. Minerals, Oil and Gas Division, Resources, Wildlife and Economic Development, Government of Northwest Territories, p. 82.
- Bell, J.M., 1929. The lead-zinc deposits near Pine Point, Great Slave Lake. Transactions of the Canadian Institute of Mining and Metallurgy, v. 32, p. 122-139.
- Bell, R., 1900. Mackenzie District. Summary Report of the Geological Survey Department for the Year 1899. p. 105-109.
- Brown, Erdman & Associates Ltd., 1982. Western Mines Ltd., Great Slave Lake Project. R-190 Zone Aquifer Test Analysis and Preliminary Dewatering Design.
- Burns, R. F. and Kent, M.M., 2001. The Great Slave Reef Lead-Zinc Deposits, Pine Point, N.W.T. Private Report for Kent Burns Group.
- Calver, B. and Farnsworth, D.J.M., 1969. Open Pit Dewatering at Pine Point Mines. Canadian Mining and Metallurgical Bulletin (CIM). December, p. 1290-1298.

- Cameron, A.E., 1917. Reconnaissance on Great Slave Lake, North West Territories. Geological Survey of Canada, Summary Report, 1916, p. 66-76.
- Cameron, A.E., 1918. Explorations in the vicinity of Great Slave Lake. Geological Survey of Canada, Summary Report, 1917, Part C, p. 21-28.
- Campbell, N., 1957. Stratigraphy and Structure of Pine Point Area, NWT. Structural Geology of Canadian Ore Deposits, Volume II, Sixth Commonwealth Mining and Metallurgical Congress.
- Camsell, C., 1915. An exploration of the region between Athabaska and Great Slave Lakes, Alberta and Northwest Territories. Geological Survey of Canada, Summary Report 1914, p. 55-60.
- Carmode, D.A. 1977. Reduction in Dolomite Contamination in Pine Point Zinc Concentrate. Canadian Mineral Processors Conference, Ottawa, January 1977.
- Carmode, 1978. Pine Point Mines Limited, Milling Practice in Canada: Milling of Copper, Lead and Zinc Ores, 1978.
- Cassoff, J., Hardie, C., Richard, P-L., Latulippe, H., Latulippe, S. and Poirier, E. 2020: Technical Report titled "Preliminary Economic Assessment, Pine Point Lead-Zinc Project, Hay River, Northwest Territories, Canada", (2020 PEA) prepared by BBA Inc. for Osisko Metals, effective date June 11, 2020;
- Cassoff, J., Hardie, C. and Richard, P-L. 2019: Technical Report titled "Mineral Resource Estimate Update, Pine Point Lead-Zinc Project, Hay River, Northwest Territories, Canada", (2019 MRE) prepared by BBA Inc. for Osisko Metals, effective date November 10, 2019;
- Cassoff, J., Hardie, C. and Richard, P-L. 2018: Technical Report titled "Mineral Resource Estimate for the Pine Point Lead-Zinc Project, Hay River, Northwest Territories, Canada", (2018 MRE) prepared by BBA Inc. for Osisko Metals, effective date December 20, 2018..
- CCME,. 1999. Canadian Water Quality Guidelines for the Protection of Freshwater Aquatic Life (Updated October 2005). Available at: <http://st-ts.ccme.ca/en/index.html>. Accessed March 2020.
- Charles, K., 2022. Pine Point Zinc and Lead Pricing, Osisko Metals, August 15, 2022, 1 p.
- Chlumsky, Armbrust & Meyer, LLC. (CAM). 2007. NI 43-101 Technical Report, Pine Point Project, Northwest Territories, Canada. Prepared for Tamerlane Ventures, Inc. Prepared by F. Barnard, S. Milne and R.L. Sandefur.

- Clemmer, S. and Lesnikov K., 2018. Assessment Report for the D1 to D6 and D9 to D13 Claims. A Summary of the 2017 Gravity Survey and Exploration Drilling Program. CMS, Pine Point Project, 2006. Metallurgical Report. Consulting report prepared by Confidential Metallurgical Services (Mississauga, ON) for Tamerlane Ventures.
- Collins, S.E., Conner, J.R., Horlacher, C.F., Larsen, D.M., Stone, B.G., 2008. NI 43-101 Technical Report Update, Pine Point Project Northwest Territories, Canada. Prepared for Tamerlane Ventures Inc. by Pincock, Allen & Holt (PAH).
- Cominco Ltd., 1984. Evaluation of Great Slave Reef Project. Internal report by R.C. Armstrong of Cominco subsidiary Pine Point Mines Ltd.
- Continental Metallurgical Services, 2006. Metallurgical Testwork Report – R-190 Deposit, November 23, 2006.
- Dawson, C.B., 1963. Pioneers at Pine Point. The Cominco Magazine, January 1963.
- Douglas, R.J.W., 1959. Great Slave Lake and Trout River map-areas, Northwest Territories, 85 S1/2 and 95 A, H; Geological Survey of Canada, Paper 58-11, 57 p.
- Durston, K.J., 1979. Open Pit Dewatering at the Pine Point Operations of Cominco Limited. First International Mine Drainage Symposium, Denver, Colorado, May 20-23, 1979.
- Dziuba, F, Geoph, P., 2020. Technical Report prepared by Aurora Geosciences Ltd., Pine Point Induced Polarization Survey, NTS 085B/15 Pine Point Project, Northwest Territories, Canada. March 4th to 23rd, 2020, 26 pages.
- EBA (EBA Engineering Consultants Ltd). 2005a. Tamerlane Pine Point Project: Water Quality and Stream Assessment Baseline Studies. Report prepared by EBA Consultants Ltd. for Tamerlane Ventures Inc.
- EBA. 2005b. Tamerlane Pine Point Project: Vegetation Ecosystem Baseline Studies. Report prepared by EBA Consultants Ltd. for Tamerlane Ventures Inc.
- EBA. 2005c. Tamerlane Pine Point Project: Wildlife Baseline Studies. Report prepared by EBA Consultants Ltd. for Tamerlane Ventures Inc.
- EBA. 2006a. Tamerlane Pine Point Project: 2006 Water Quality Sampling Program Pine Point, Northwest Territories. Report prepared by EBA Consultants Ltd. for Tamerlane Ventures Inc.
- EBA. 2006b. 2006 Rare Plant Survey, Tamerlane Pine Point Project, Northwest Territories. Report prepared by EBA Consultants Ltd. for Tamerlane Ventures Inc.

- EBA. 2006c. Tamerlane Pine Point Project: 2006 Wildlife Surveys, Pine Point, NT. Report prepared by EBA Consultants Ltd. for Tamerlane Ventures Inc.
- EBA. 2006d. Desktop Evaluation of Natural Groundwater Flow Velocities – Pine Point Mine Ground Freezing Project. Letter Report submitted to Tamerlane Ventures Inc. September 2006.
- EBA. 2007. Developers Assessment Report: Pine Point Pilot Project. Report prepared by EBA Consultants Ltd. for Tamerlane Ventures Inc.
- ECG (Ecosystem Classification Group). 2009. Ecological Regions of the Northwest Territories – Taiga Plains. Department of Environment and Natural Resources, Government of the Northwest Territories, Yellowknife, NT, Canada. viii + 173 pp. + folded insert map.
- Evans, 1998. Metal studies of water, sediments and fish from the Resolution Bay area of Great Slave Lake: studies related to the decommissioned Pine Point Mine. National Hydrology Research Institute. NWRI Contribution Series No. 98-87.
- Fish, R., 1981. Pine Point Orebodies Dictate Unusual Approaches. In The Canadian Mining Journal, May 1981.
- G&T Metallurgical Services, 2011. KM2984. Metallurgical Testing on Samples from the Pine Point Project. - R-190 Deposit.
- G&T Metallurgical Services Ltd., 2012. KM3195. Flotation Testing Dense Media Product N-204 Deposit – N-204 Deposit.
- Gagnon, J.-M. G. (2022, April 05). *Money Enterprises*. Retrieved from le Journal de Quebec: <https://www.journaldequebec.com/2022/04/05/matagami-fragilisee-par-le-depart-definitif-dune-miniere-etrangere>
- Gann P., 2016. NI 43-101 Summary Technical Report Update of the Pine Point Mine Development Project, Northwest Territories, Canada for Darnley Bay Resources Ltd.
- Gann, 2017. NI 43-101 Summary Technical Report Update of the Pine Point Mine Development Project, Northwest Territories, Canada.
- Garven, G., 1985. The role of regional fluid flow in the genesis of the Pine Point deposit, Western Canada Sedimentary Basin. *Economic Geology*, v. 80, p. 307-324.
- Gibbins, W.A., Seaton, J.B., Laporte, P.J., Murphy, J.D., Hurdle, E.J., and Padgham, W.A., 1977. Mineral industry report 1974, Northwest Territories. Indian and Northern Affairs Canada, EGS 1977-5, 267 p.

- Giroux, 1982a. Global Ore Reserve Estimates - O556 deposit. Consulting report prepared October 1982 for Westmin Resources Ltd. by Sinclair, A.J., and Giroux, G.H, 50 p.
- Giroux, 1982b. Global Ore Reserve Estimates - P499 deposit. Consulting report dated November 1982 for Westmin Resources Ltd. by Sinclair, A.J., and Giroux, G.H., 60 p.
- Giroux, 1982c. Global Ore Reserve Estimates - X25 deposit. Consulting report dated November 1982 for Westmin Resources Ltd. by Sinclair, A.J., and Giroux, G.H., 75 p.
- Giroux, G.H. and McCartney, I., 2001. Report on the Great Slave Reef lead-zinc deposits, Pine Point, NWT. Consulting report prepared for Kent Burns Group LLC, 37 p.
- Giroux, G.H. and McCartney, I., 2004. Report on the Great Slave Reef Lead-Zinc Deposits, Pine Point, NWT, for Tamerlane Ventures, November 2001, amended May 13, 2004.
- Golder, 2018. Pine Point Project 2018 Wildlife Studies. Prepared for Pine Point Mining Limited by Golder Associates Ltd. Yellowknife, Northwest Territories.
- Golder. 2019a. Pine Point Mine Soil and Vegetation 2019 Post Field Interim Findings. Submitted to: Pine Point Mining Limited. v + 12 pp.
- Golder. 2019b. Pine Point Mine – Water Quality and Fish Site Reconnaissance Survey Field Memo, October 2019. Submitted to: Pine Point Mining Limited. v + 3 pp.
- Golder, 2020. Doc030_19125747_Pine Point Project Environmental Input to PEA, May 15, 2020, 11 p.
- Government of Northwest Territories, 2017. Reclaim 7.0 User Manual Mining Version, dated November 2017, 19 p.
https://www.enr.gov.nt.ca/sites/enr/files/resources/reclaim_user_manual_mining.pdf
- Grech, R. (2021, June 04). News. Retrieved from The Timmins Daily Press:
<https://www.timminspress.com/news/local-news/glencore-approves-further-drilling-in-hopes-of-extending-kidd-mine-life>
- GTC Geologic Testing Consultants, 1983. Hydrogeologic Evaluation of the Pine Point – Great Slave Lake Region. Report to National Hydrology Research Institute, Environment Canada.
- Hannigan, P., 2007. Metallogeny of the Pine Point Mississippi Valley-Type Zinc-Lead District, Southern Northwest Territories. Mineral Deposits of Canada, ed. W.D. Goodfellow, pp. 609 to 632.

Health Canada, 2006. Health Canada. 2006. Guidelines for Canadian Drinking Water Quality Summary Table. Prepared by the Federal-Provincial-Territorial Committee on Drinking Water of the Federal-Provincial-Territorial Committee on Health and the Environment. www.healthcanada.gc.ca/waterquality

Horlacher, C.F., 2011. Technical Report for Resources Pine Point N204 Deposit, Northwest Territories, Canada. Prepared for Tamerlane Ventures Inc. by Consultant Pincock, Allen & Holt in March 18, 2011, 134 p.

Hutchinson D.J and Diederichs M.S, 1996, Cable bolting in underground mines (Library and Archives Canada)

Infomine Mining Intelligence And Technology, 2017. Costmine MME 2017 Surface Equipment, 2017, 47p.

Krebs, W. and Macqueen, R., 1984. Sequence of diagenetic and mineralization events, Pine Point lead-zinc property, Northwest Territories, Canada. Bulletin of Canadian Petroleum Geology, v. 32, no. 4, p. 434-464.

Kyle, J.R., 1981. Geology of the Pine Point lead-zinc district, Chapter 11; in Wolf, K.H., ed., Handbook of Strata-bound and Stratiform Ore Deposits. Elsevier, p. 643-741.

Lafontaine, C. 1997. Fort Resolution Fish Monitoring Program (1992 – 1997): Concentrations of metals and trace elements in muscle and liver of fish collected from Great Slave Lake, Fort Resolution Area, NWT. Final Report. Prepared for Department of Indian Affairs and Northern Development, Water Resources Division, Yellowknife, NWT.

Lajoie, J. J. and Klein, J., 1979. Geophysical exploration at the Pine Point Mines Ltd. zinc-lead property, Northwest Territories, Canada; in Hood, P.J., ed., Geophysics and Geochemistry in the Search for Metallic Ores. Geological Survey of Canada, Economic Geology Report 31, p. 653-664.

Leach, D.L., Taylor, R.D., Fey, D.L., Diehl, S.F., and Saltus, R.W., 2010. A Deposit Model for Mississippi Valley-Type Lead-Zinc Ores. Scientific Investigations Report 2010–5070–A, U.S. Geological Survey.

Leach, D.L. and Sangster, D.F., 1993. Mississippi Valley-type lead-zinc deposits, in Kirkham, R.V., Sinclair, W.D., Thorpe, R.I., and Duke, J.M., eds., Mineral Deposit Modeling. Geological Association of Canada, Special Paper 40, p. 289-314.

Leach, D.L., Viets, J.B., Foley-Ayuso, N. and Klein, D.P.. 1995. Mississippi Valley-Type Pb-Zn Deposits. USGS Bulletin 1693.

- Macdonald, G., McLeod, K., Pilotto, D., Embree, K., Siega, A.D., and Gann, P., 2017. NI 43-101 Preliminary Economic Assessment Technical Report on The Pine Point Zinc Project, Northwest Territories, Canada. Prepared for Darnley Bay Resources Limited by Consultant JDS Energy & Mining Inc. in June 2017, 250 p.
- Meijer Drees, N. C., 1988. The Middle Devonian sub-Watt Mountain unconformity across the Tathlina uplift; District of Mackenzie and northern Alberta, Canada: Canadian Society of Petroleum Geologist International Symposium on the Devonian System, 2nd, August 1987, Calgary, Alberta, Proceedings, v. 2, p. 477-494. Meijer Drees, N. C., 1993. The Devonian succession in the subsurface of the Great Slave and Great Bear plains, Northwest Territories. Geological Survey of Canada, Bulletin 393, 222 p.
- Meikle, M., 1930a. Report on Northern Lead Zinc Company Property, Great Slave Lake, North West Territories. Department of the Interior, Supervisory Mining Engineer's Office, The Pas, Manitoba, February 28, 1930. (National Archives of Canada, RG 85, Volume 262, Box 2, File 999-240).
- Meikle, M., 1930b. Report on Prospecting and Mining Operations, Great Slave Lake Area, North West Territories. Department of the Interior, Supervisory Mining Engineer's Office, Fort Smith, NWT, November 7, 1930.
- MineTech International Ltd., 2012. Technical Report on the Pine Point N-204 Lead-Zinc Deposit Latitude N60° 57', Longitude W 114° 02' Near Fort Resolution, Northwest Territories, Canada. Prepared for Tamerlane Ventures Inc. Authors: Wm. Douglas Roy, M.A.Sc., P. Eng., Patrick J. Hannon, M.A.Sc., P.Eng., Ian M. Flint, PhD., P. Eng. Effective Date of Report: March 23, 2012.
- MineTech International Ltd., 2012. Technical Report on the Pine Point Project Deposits: R-190, P-499, O-556, X-25, Z-155, G-03 Near Pine Point, Northwest Territories, Canada NTS 85B/(10, 11, 14, 15, 16)N 60° 44', W 115° 07'. Prepared for Tamerlane Ventures Inc. Authors: Patrick J. Hannon, M.A.Sc., P.Eng., William Douglas Roy, M.A.Sc., P.Eng., Ian M. Flint, PhD., P.Eng., Bradford Ernst, B.Eng., P.Eng., Edward Thornton, P. Eng. Effective Date of Report: April 2, 2012.
- Mining.com. (2022, January 17). *Exploration Top Companies USA Lead Manganese Silver Zinc*. Retrieved from Mining.com: <https://www.mining.com/south32s-pfs-for-taylor-supports-multi-decade-critical-metals-operation-in-arizona/>
- Mining.com. (2022, June 22). *News - Copper Zinc*. Retrieved from Mining: <https://www.mining.com/hudbay-marks-end-of-flin-flon-mining-era-with-closure-of-777-operations/>

- MVEIRB, 2008. Mackenzie Valley Environmental Impact Review Board. Report of environmental assessment and reasons for decision on Tamerlane Ventures Inc.'s Pine Point Pilot Project. EA0607-002. 127 p.
- MVLWB/AANDC, 2013. Guidelines for the Closure and Reclamation of Advanced Mineral Exploration and Mine Sites in the Northwest Territories. Mackenzie Valley Land and Water Board. November 2013, 88 p.
- MVLWB/INAC/GNWT, 2017. Guidelines for Closure and Reclamation Cost Estimates for Mines. Mackenzie Valley Land and Water Board; Gwich'in Land and Water Board; Sahtu Land and Water Board; Wek'èezhii Land and Water Board; Government of the Northwest Territories; Indigenous and Northern Affairs Canada. November 2017, 17 p.
- Nagle, E.H., Zinovich, J., 1989. The Prospector North of Sixty. Edmonton. Lone Pine Publishing. Northwest Territories and Nunavut Mining Regulations (C.R.C., c. 1516), Schedule 1, accessed from http://laws-lois.justice.gc.ca/PDF/C.R.C.,_c._1516.pdf on July 22, 2013.
- Nwt-reg-125-95, Mine Health and Safety Act, Nwt-reg-125-95, 2018. Mine Health and Safety Regulations, Sep 21, 2018, 286 p.
- Pickett D.E., Hall W.S., Smith G.W., 1978. Milling Practice in Canada, Canadian Institute of Mining and Metallurgy, p. 222-227.
- Pincock, Allen & Holt, 2008. NI 43-101 Technical Report Update, Pine Point Project Northwest Territories, Canada. Prepared for Tamerlane Ventures Inc. July 30, 2008. Authors: Stuart E. Collins, P.E., J. Ross Conner, P.Geo., Craig F. Horlacher, Don M. Larsen, Ph.D., P.E., Barton G. Stone, C.P.G.
- Pincock, Allen & Holt, 2011. Technical Report for Resources Pine Point N204 Deposit, Northwest Territories, Canada. DRAFT Report Prepared for Tamerlane Ventures Inc. February 18, 2011. Prepared by Craig F. Horlacher, MAusIMM.
- Porter, J.W., Price, R.A., and McCrossan, R.G., 2001. The Western Canada Sedimentary Basin; Philosophical Transactions of the Royal Society of London, A 305, p. 169-192.
- Qing, H., 1991. Diagenesis of Middle Devonian Presqu'ile dolomite, Pine Point NWT and adjacent subsurface. PhD thesis, McGill University, 292 p.
- Qing, H., 1998a. Petrography and geochemistry of early-stage, fine- and medium-crystalline dolomites in the Middle Devonian Presqu'ile Barrier at Pine Point, Canada. Sedimentology, v. 45, p. 433-446.

- Qing, H., 1998b. Geochemical constraints on the origin and timing of paleofluid flow in the Presqu'ile barrier reef, Western Canada Sedimentary Basin; in Parnell, J., ed., *Dating and Duration of Fluid Flow and Fluid-Rock Interaction*. Geological Society, London, Special Publications 144, p. 173-187.
- Qing, H. and Mountjoy, E.W., 1992. Large-scale fluid flow in the Middle Devonian Presqu'ile barrier, Western Canada Sedimentary Basin. *Geology*, v. 20, p. 903-906.
- Qing, H. and Mountjoy, E.W., 1994a. Formation of coarsely crystalline, hydrothermal dolomite reservoirs in the Presqu'ile barrier, Western Canada Sedimentary Basin. *American Association of Petroleum Geologists, Bulletin*, v. 78, no. 1, p. 55-77.
- Qing, H. and Mountjoy, E.W., 1994b. Origin of dissolution vugs, caverns, and breccias in the Middle Devonian Presqu'ile barrier, host of Pine Point Mississippi Valley-type deposits. *Economic Geology*, v. 89, p. 858-876.
- Regneir, S. (2019, July 23). *News*. Retrieved from Today in BC:
<https://www.todayinbc.com/news/slowng-market-has-teck-closing-pend-oreille-mine/>
- Rescan (Rescan Environmental Services Ltd.). 2012a. Pine Point Project: 2011 Terrestrial Ecology Baseline Study — X-25, P-499, O-556, Z-155, and G-03. Prepared for Tamerlane Ventures Inc. by Rescan Environmental Services Ltd. Vancouver, British Columbia.
- Rescan. 2012b. Pine Point Project: 2011 Terrestrial Ecology Baseline Study — N-204. Prepared for Tamerlane Ventures Inc. by Rescan Environmental Services Ltd. Vancouver, British Columbia.
- Rescan. 2012c. Pine Point Project: 2011 Meteorology, Air Quality, and Noise Baseline Study – R-190, X-25, P-499, O-556, Z-155, and G-03. Prepared for Tamerlane Ventures Inc. by Rescan Environmental Services Ltd. Vancouver, British Columbia.
- Rescan. 2012d. Pine Point Project: 2011 Terrestrial Ecology Baseline Study — N-204. Prepared for Tamerlane.
- Rescan. 2012d. Pine Point Project: 2011 Meteorology, Air Quality, and Noise Baseline Study – N-204. Prepared for Tamerlane Ventures Inc. by Rescan Environmental Services Ltd. Vancouver, British Columbia.
- Rescan. 2012e. Pine Point Project: 2011 Fish and Fish Habitat Baseline Studies. Prepared for Tamerlane Ventures Inc.
- Rescan. 2012f. Pine Point Project: 2011 Aquatic Resources Baseline Study – N-204. Prepared for Tamerlane Ventures Inc.

- Rescan. 2012g. Pine Point Project: 2011 Aquatic Resources Baseline Study – X-25, P-499, O-556, Z-155, and G-03. Prepared for Tamerlane Ventures Inc.
- Rescan. 2012h. Pine Point Project 2011 Wildlife Baseline Studies — X-25, P-499, O-556, Z-155, AND G-03.
- Rescan. 2012i. Pine Point Project: 2011 Wildlife Baseline Studies – N-204. Prepared for Tamerlane Ventures Inc.
- Rescan. 2012j. Pine Point Project: 2011 Heritage Baseline Studies – X-25, P-499, O-556, Z-155, and G-03. Prepared for Tamerlane Ventures Inc. by Rescan Environmental Services Ltd. Vancouver, British Columbia.
- Rescan. 2012k. Pine Point Project: 2011 Heritage Baseline Studies – N-204. Prepared for Tamerlane Ventures Inc. by Rescan Environmental Services Ltd. Vancouver, British Columbia.
- Rescan. 2012l. Pine Point Project: 2011 Socio-economic Baseline Study. Prepared for Tamerlane Ventures Inc. by Rescan Environmental Services Ltd. Vancouver, British Columbia.
- Rhodes, D., et al., 1975. Pine Point Ore bodies and Their Relationship to the Stratigraphy, Structure, Dolomitization, and Karstification of the Middle Devonian Barrier Complex. *Economic Geology*. Vol. 79, pp. 991-1055.
- Rhodes, D., Lantos, E.A., Lantos J. A., Webb, R.J. and Owens, D.C., 1984. Pine Point ore bodies and their relationship to the stratigraphy, structure, dolomitization, and karstification of the Middle Devonian Barrier Complex. *Economic Geology*. 79(5): 991-1055.
- Rice, J.M., Paulen, J.M., McClenaghan, M.B., and Oviatt, N.M., 2013. Glacial stratigraphy of the Pine Point Pb-Zn mine site, Northwest Territories Geological Survey of Canada. *Current Research* 2013-5.
- Roedder, E., 1968. Temperature, salinity, and origin of the ore-forming fluids at Pine Point, Northwest Territories, Canada, from fluid inclusion studies. *Economic Geology*, 63(5): 439-450.
- Sangster, D.F., 1990. Mississippi Valley-type and sedex lead-zinc deposits: a comparative examination; *Institution of Mining and Metallurgy, Transactions, (Sect. B: Applied earth science)*, v. 99, p. B21-B42. SGS Lakefield Research Ltd., 2007a. Project No. 11574-001 "Sulphide Ore Upgrading Using Gravity Separation"—R-190 Deposit.
- Sangster, D.F., 1996. Mississippi Valley-type Lead-Zinc. *Geology of Canadian Mineral Deposit Types*, (ed) O.R. Eckstrand, W.D. Sinclair, and R.I. Thorpe; Geological Survey of Canada, *Geology of Canada*, no. 8, p. 253-261.

- Schmidt, D. (2019, October 16). *Asset Management > Life-cycle-end-of-life-management*. Retrieved from Mining Magazine: <https://www.miningmagazine.com/life-cycle-end-of-life-management/news/1373654/langlois-closing-its-doors>
- SGS Lakefield Research Ltd., 2007. Projection No. 11633-001 “The Recovery of Lead and Zinc from Pine Point Ore Samples Using a Non-cyanide Flotation Method”– R190 Deposit.
- SGS Lakefield Research Ltd., 2008. Project 11790-001 “The Recovery of Lead and Zinc Using a Non-cyanide Flotation Method on the Pine Point Ore Samples”– O-556/Z-155 Deposit.
- SGS Canada Inc., 2011. Project No. 13001-002 “Bulk Concentrate Production from Heavy Liquid Separation on a Sample from the Pine Point N-204 Deposit”– N-204 Deposit.
- Siege, A. D. and Gann, P., 2014. NI-43-101 Summary Technical Report Update of the Pine Point Mine Development Project, Northwest Territories, Canada. Prepared for Tamberlane Ventures Inc.
- Siege, A.D. and Gann, P., 2017. NI43-101 Technical Report Resource Summary, Pine Point, Northwest Territories, Canada for Darnley Bay Resources Ltd.
- Silke, R., 2009. Pine Point. The Operational History of Mines in The Northwest Territories, Canada. p 376-391.
- Sinclair, A.J. and Giroux, G.H., 1982a. Global Ore Reserve Estimates - O556 deposit. Consulting report prepared October 1982 for Westmin Resources Ltd. 50 p.
- Sinclair, A.J. and Giroux, G.H., 1982b. Global Ore Reserve Estimates - P499 deposit. Consulting report dated November 1982 for Westmin Resources Ltd. 60 p.
- Sinclair, A.J. and Giroux, G.H., 1982c. Global Ore Reserve Estimates - X25 deposit. Consulting report dated November 1982 for Westmin Resources Ltd. 75 p.
- Skall, H., 1975. The Paleoenvironment of the Pine Point Lead-Zinc District. *Economic Geology*. 70(1): 22-47.
- Steinert USA Inc, 2018. Project No. TRE-10981-XRT-646, Rev 01 “XRT Test Work Report – Osisko –Pine Point Waste Rock Sorting Test Work Opportunity No 10981”.
- Stevenson International Groundwater Consultants Ltd., 1984. A Study of the Great Slave Reef Pine Point Mines Aquifer, Based on Analyses of Selected Pine Point Mines Pumping Test Data. Prepared for: Mr. A.W. Randall, Westmin Resources Limited.
- Swisher, S. 2006a. Traditional Knowledge Summary Report. Pine Point Pilot Project. Fort Resolution, NWT. December 2006.

- Swisher, S. 2006b. Traditional Knowledge Summary Report. Pine Point Pilot Project. Hay River, NWT. December 2006.
- Tamerlane Ventures Inc. 2006. Project Description Report Pine Point Pilot Project. Report prepared by Tamerlane Ventures Inc. for the Mackenzie Valley Land and Water Board. May 2006.
- Tetra Tech, 2017. Pine Point Project, Concentrate Transload Location and Equipment Trade-Off Studies, File: 704-Min.Vmin03108-041, October 19, 2017, 23 p.
- Tetra Tech, 2018. PP Tetra Tech Geochem Review Report Draft, Summary of Geochemical Characterization Data for the Pine Point Project, NWT, July 3, 2018, 76 p.
- Tetra Tech, 2020. Pine Point Mine Dewatering Estimates. Technical Memo prepared for Pine Point Mining Limited by consultants Gutmann, C., Hsieh, A., July 20, 2020, 30 p.
- Thorpe, R.I., 1972. Mineral exploration and mining activities, mainland Northwest Territories, 1966 to 1968 (excluding the Coppermine River area). Geological Survey of Canada, Paper 70-70, 204 p.
- Trevali Mining Corp. (2021, January 15). Caribou. Retrieved from trevali.com:
<https://trevali.com/operations/operations/caribou/>
- Turner, W.A., Pierce K.L., and Cairns K.A., 2002. Great Slave Reef (GSR) Project Drill Hole Database. A compilation of the drill hole locations, drill logs, and associated geochemical data for the Great Slave Reef Joint Venture Project; Interior Platform, Northwest Territories, Canada (NTS 85B11 to 14). NWT Open Report 2002-001, C.S. Lord Northern Geoscience Centre, Yellowknife, Northwest Territories.
- Turner, W.A. and Gall L.P., 2003. Regional structural data from the Hay River area, Northwest Territories, with emphasis on the Pine Point mining camp; Geological Survey of Canada, Current Research 2003-C10, 10 p.
- Turner, W.A., 2006. Microthermometric study of fluids associated with Pb-Zn mineralization in the vicinity of the Pine Point mining camp; *in* Potential for Carbonate-hosted Lead-zinc Mississippi Valley-type Mineralization in Northern Alberta and Southern Northwest Territories: Geoscience Contributions, Targeted Geoscience Initiative, (ed.) P.K. Hannigan; Geological Survey of Canada, Bulletin 591, p. 221-240.
- U.S. Geological Survey, 2013. Mineral commodity summaries 2013. U.S. Geological Survey, Technical Report. 198 p.
- Vice, J., 2022. Pine Point Marketing Report, Stone House Consulting, August 18, 2022, 2 p.

- Vogwill, R.I.J., 1976. Some Practical Aspects of Open-Pit Dewatering at Pine Point. Bull. Can. Inst. Min. Met. 69(768): 76-88.
- Vogwill, R.I.J., 1981. R-190 Zone Aquifer Test Analysis and Preliminary Dewatering Design. Prepared for Western Mines Ltd. Great Slave Reef Project.
- Webb, R.J., 1981. Mineralization: Pine Point, N.W.T., Pine Point Mines, un-pub. report., 14 p.
- Westmin Resources Ltd., 1981. Great Slave Reef Project - Summary report of 1981 drilling program. Internal report by A.W. Randall. May 1981. 20 p plus maps of O-555 and O-556.
- Westmin Resources Ltd., 1983a. Special Studies Group: internal geological report by J.L. Hardy dated April 1983, Approx. 100 p.
- Westmin Resources Ltd., (undated, 1980's). Great Slave Reef Project, West Reef, list of diamond drill holes. Internal report, handwritten and typed, 60 p.
- Westmin Resources Ltd., 1983b. Great Slave Reef - R190, Review of mine planning. Internal report by C.T. Penney dated Nov 1983, 18 p.
- Westmin Resources Ltd., 1983c. Great Slave Reef Project, Review of Mine Planning - Phase I. Internal report by C.T. Penney dated 8 Dec 1983, Approx. 25 p.
- Westmin Resources Ltd., 1984. Great Slave Reef - Exploration meeting. Internal report by A.W. Randall dated November 20-22, 1984, 30 p.
- White, P., 2020. Pine Point PEA - Ground Water Control Measures - Rev #2, July 6, 2020, 14 p.
- Williams, G.K., 1990. Tectonics and structure, Mackenzie Corridor, Northwest Territories. Geological Survey of Canada, Open File 2248, 36 p.
- Wright, G.N., McMechan, M.E., and Potter, D.E.G., 1994. Structure and architecture of the Western Canada Sedimentary Basin; in Mossop, G.D. and Shetsen, I., comp., Geological Atlas of the Western Canada Sedimentary Basin: Canadian Society of Petroleum Geologists / Alberta Research Council, p. 25-40.