Murray Brook Project
NI 43-101 Technical Report & Preliminary Economic Assessment
Bathurst, New Brunswick, Canada

Effective Date: June 20, 2025
Report Date: July 7, 2025

Prepared for:
Canadian Copper Inc.
The Canadian Venture Building
82 Richmond Street East
Toronto, ON M5C 1P1

Prepared by:
Ausenco Engineering Canada ULC
11 King St. West, 15th Floor
Toronto, Ontario, M5H 4C7

List of Competent Persons:
Tommaso Roberto Raponi, P. Eng., Ausenco Engineering Canada ULC
Glenn LeBlanc, P.Eng., Ausenco Engineering Canada ULC
Jonathan Cooper, P. Eng., Ausenco Sustainability ULC
James Millard, P.Geo., Ausenco Sustainability ULC
Eugene Puritch, P.Eng., P&E Mining Consultants Inc.
Andrew Bradfield, P.Eng., P&E Mining Consultants Inc.
William Stone, P.Geo., P&E Mining Consultants Inc.
Jarita Barry, P.Geo., P&E Mining Consultants Inc.
Yungang Wu, P.Geo., P&E Mining Consultants Inc.
Pierre Lacombe, P.Eng., Independent Consultant
Marcello Locatelli, P.Eng., Inteloc, Inc.
Jeff Gilchrist, P.Eng., Stantec Consulting Ltd.

Important Notice

This report was prepared as National Instrument 43-101 Technical Report for *Canadian Copper Inc. (Canadian Copper)* by Ausenco Engineering Canada ULC (Ausenco), P&E Mining Consultants, Inc. (P&E), Pierre Lacombe, Inteloc, Inc. (Inteloc) and Stantec Consulting Ltd. (Stantec), 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 Snowline subject to terms and conditions of its contracts with each of the Report Authors. Except for the purposed legislated under Canadian provincial and territorial securities law, any other uses of this report by any third party are at that party's sole risk.

CERTIFICATE OF QUALIFIED PERSON
Tommaso Roberto Raponi

I, Tommaso Roberto Raponi, P.Eng., as a Principal Metallurgist with Ausenco Engineering Canada ULC (Canada), ("Ausenco"), with an office address of Suite 1550 - 11 King St West, Toronto, ON M5H 4C7, do certify that:

1. This certificate applies to the technical report titled "Murray Brook Project NI 43-101 Technical Report & Preliminary Economic Assessment, New Brunswick, Canada," with an effective date of June 20, 2025 (the "Technical Report").
2. I graduated from the University of Toronto with a Bachelor of Applied Science degree in Geological Engineering, with specialization in Mineral Processing in 1984.
3. I am a Professional Engineer registered with the Professional Engineers Ontario (No. 90225970), Engineers and Geoscientists British Columbia (No. 23536) and NWT and Nunavut Association of Professional Engineers and Geoscientists (No. L4508).
4. I have practiced my profession continuously for over 40 years with experience in the development, design, operation and commissioning of mineral processing plants, focusing on gold projects, both domestic and internationally. Examples of projects I have worked on include: Generation Mining Marathon PGM Project FS, Adriatic Metals Vareš & Rupice FS, Meridian Mining UK Societas Cabaçal Gold-Copper Project PFS, and Discover Silver Corp. Cordero Silver Project FS.
5. I have read the definition of "Qualified Person" set out in the National Instrument 43-101 Standards of Disclosure for Mineral Projects ("NI 43-101") and certify that by virtue of my education, affiliation to a professional association and past relevant work experience, I fulfil the requirements to be a "Qualified Person" for those sections of the Technical Report that I am responsible for preparing.
6. I have not visited the Murray Brook project property.
7. I am responsible for Sections 1.1, 1.2, 1.11, 1.12.1, 1.13, 1.15, 1.16, 1.18, 1.19, 2.1 to 2.8 excluding 2.4, 2.6, 3.3, 15, 17, 18.1, 18.5, 18.6, 18.11, 18.13, 19, 21 (excluding the following: 21.2.2.1, 21.2.2.3, 21.2.2.4, 21.2.3.1, 21.2.3.2.1, 21.2.5.1.1, 21.2.5.2.2, 21.2.5.2.4, 21.2.6.1, 21.2.6.3, 21.2.7.1 to 21.2.7.4, 21.2.7.5.1 to 21.2.7.5.3), 22, 24, 25.11, 25.12.1, 25.14 to 25.16, 25.18.1.4, 25.18.1.5, 25.18.2.3 to 25.18.2.5, 26.1, 26.5, 26.6, and coauthor of Section 27 of the Technical Report.
8. I am independent of Canadian Copper Inc as independence is defined in Section 1.5 of NI 43-101.
9. I have had no previous involvement with the Murray Brook Project.
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 that Instrument. As of 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 those sections of the Technical Report not misleading.

Dated: July 7, 2025

"Signed and sealed"

Tommaso Roberto Raponi, P.Eng.

CERTIFICATE OF QUALIFIED PERSON
Glenn LeBlanc, P.Eng.

I, Glenn LeBlanc, P.Eng., as a Senior Project Manager with Ausenco Engineering Canada ULC (Ausenco), with an office address of 11 King Street West, Suite 1500, Toronto, Ontario M5H 4C7, do certify that:

1. This certificate applies to the technical report titled "Murray Brook Project NI 43-101 Technical Report & Preliminary Economic Assessment, New Brunswick, Canada," with an effective date of June 20, 2025 (the "Technical Report").
2. I graduated from the Technical University of Nova Scotia with a Bachelor of Engineering Degree in Civil Engineering in 1988.
3. I am a Professional Engineer registered and in good standing with Association of Professional Engineers & Geoscientists of New Brunswick (APEGNB, Member Number M6730), and with Engineers Nova Scotia (APEGNS, Member Number M10994).
4. I have practiced my profession for continuously for over 37 years with experience in the development, design, operation, maintenance and commissioning of infrastructure and mining projects. Previous projects that I have worked on that have similar features to the Murray Brook Project are the Brunswick Mining Operations with Noranda located around Bathurst, New Brunswick, the Moose River Consolidated Project for St. Barbara located in Nova Scotia, and the Goldboro Project for Signal Gold in Nova Scotia.
5. I have read the definition of "Qualified Person" set out in the National Instrument 43-101 Standards of Disclosure for Mineral Projects ("NI 43-101") and certify that by virtue of my education, affiliation to a professional association and past relevant work experience, I fulfil the requirements to be a "Qualified Person" for those sections of the Technical Report that I am responsible for preparing.
6. I have visited the Property that is the subject of this Technical Report on October 4th, 2024.
7. I am responsible for Sections 2.4.1, 18.2, 18.3.1, 18.3.2, and 18.3.5 of the Technical Report.
8. I am independent of Canadian Copper Inc as independence is defined in Section 1.5 of NI 43-101.
9. I have had no previous involvement with the Murray Brook Project.
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 that Instrument. As of 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 those sections of the Technical Report not misleading.

Dated: July 7, 2025

"Signed and sealed"

Glenn LeBlanc, P.Eng.

CERTIFICATE OF QUALIFIED PERSON
Jonathan Cooper, P.Eng.

I, Jonathan Cooper, P.Eng., as a water resources engineer with Ausenco Sustainability ULC, a wholly owned subsidiary of Ausenco Engineering Canada (Ausenco), with an office address of 11 King Street West, Suite 1500, Toronto, Ontario M5H 4C7, do certify that:

1. This certificate applies to the technical report titled "Murray Brook Project NI 43-101 Technical Report & Preliminary Economic Assessment, New Brunswick, Canada," with an effective date of June 20, 2025 (the "Technical Report").
2. I graduated from the University of Western Ontario with a Bachelor of Engineering Science in Civil Engineering in 2008, and University of Edinburgh with a Master of Environmental Management in 2010.
3. I am a Professional Engineer registered and in good standing with Order of Engineers of Quebec (temporary engineer permit #6067376), Professional Engineers Ontario (registration #100191626), Engineers and Geoscientists British Columbia (registration #37864) and Northwest Territories and Nunavut Association of Professional Engineers and Geoscientists (registration # L4227).
4. I have practiced my profession for continuously for over 15 years with experience in the development, design, operation, and commissioning of surface water infrastructure. Previous projects that I have worked on that have similar features to the Novador Project are the Kwanika-Stardust for NorthWest Copper located in British Columbia, Colomac Gold Project located in the Northwest Territories, and the Crawford Project located in Ontario.
5. I have read the definition of "Qualified Person" set out in the National Instrument 43-101 Standards of Disclosure for Mineral Projects ("NI 43-101") and certify that by virtue of my education, affiliation to a professional association and past relevant work experience, I fulfil the requirements to be a "Qualified Person" for those sections of the Technical Report that I am responsible for preparing.
6. I have not visited the Murray Brook site.
7. I am responsible for Sections 18.14, 21.2.2.3, and 25.12.2 of the Technical Report.
8. I am independent of Canadian Copper Inc as independence is defined in Section 1.5 of NI 43-101.
9. I have had no previous involvement with the Murray Brook Project.
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 that Instrument. As of 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 those sections of the Technical Report not misleading.

Dated: July 7, 2025

"Signed and sealed"

Jonathan Cooper, P.Eng.

CERTIFICATE OF QUALIFIED PERSON
James Millard, P. Geo.

I, James Millard, P. Geo., as a Director, Strategic Projects with Ausenco Sustainability ULC., a wholly owned subsidiary of Ausenco Engineering Canada (Ausenco), with an office address of Suite 100, 2 Ralston Avenue, Dartmouth, NS, B3B 1H7, Canada, do certify that:

1. This certificate applies to the technical report titled "Murray Brook Project NI 43-101 Technical Report & Preliminary Economic Assessment, New Brunswick, Canada," with an effective date of June 20, 2025 (the "Technical Report").
2. I graduated from Brock University in St. Catharines, Ontario in 1986 with a Bachelor of Science in Geological Sciences, and from Queen's University in Kingston, Ontario in 1995 with a Master of Science in Environmental Engineering.
3. I am a member (P.Geo.) of the Association of Professional Geoscientists of Nova Scotia, Membership No. 021.
4. I have practiced my profession for 25 years. I have worked for mid- and large-size mining companies where I have acted in senior technical and management roles, in senior environmental consulting roles, and provided advice and/or expertise in a number of key subject areas. These key areas included: feasibility-level study reviews; NI 43-101 report writing and review; due diligence review of environmental, social, and governance areas for proposed mining operations and acquisitions, and directing environmental impact assessments and permitting applications to support construction, operations, and closure of mining projects.

   In addition to the above, I have been responsible for conducting baseline data assessments, surface and groundwater quantity and quality studies, mine rock geochemistry and water quality predictions, mine reclamation and closure plan development, and community stakeholder and Indigenous peoples' engagement initiatives.

   Recently, I acted in the following project roles: Qualified Person for the environmental/sustainability aspects for "Puquios Project, Feasibility Study Report, La Higuera, Coquimbo Region, Chile", "Volcan Project, NI 43-101 Technical Report on Preliminary Economic Assessment, Tierra Amarilla, Atacama Region, Chile" and, "Colomac Gold Project, NI 43-101 Technical Report and Preliminary Economic Assessment, Northwest Territories, Canada"; and principal author for the environmental/sustainability sections for the "Kwanika-Stardust Project, NI 43-101 Technical Report and, Preliminary Economic Assessment, British Columbia, Canada".
5. I have read the definition of "Qualified Person" set out in the National Instrument 43-101 Standards of Disclosure for Mineral Projects ("NI 43-101") and certify that by virtue of my education, affiliation to a professional association and past relevant work experience, I fulfil the requirements to be a "Qualified Person" for those sections of the technical report that I am responsible for preparing.
6. I have not visited the Murray Brook Project.
7. I am responsible for Sections 1.14 to 1.14.2, 1.14.3 – paragraphs related to permitting and liability, 3.2, 20.1, 20.2, 20.3, 20.4 except for 20.4.1 25.13, 26.7 and coauthor of Section 27 of the Technical Report.
8. I am independent of the Company as independence is described by Section 1.5 of NI 43-101 and in Section 1.5 of the Companion Policy to NI 43-101
9. I have had no previous involvement with the Murray Brook Project.

10. I have read NI 43-101, Form 43-101F1 and the sections of the technical report for which I am responsible have been prepared in compliance with NI 43-101, Form 43-101F1. As of 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 those sections of the technical report not misleading.

Dated: July 7, 2025

"Signed and sealed"

James Millard, P.Geo.

CERTIFICATE OF QUALIFIED PERSON
Jarita Barry, P.Geo.

I, Jarita Barry, P.Geo., as an independent geological consultant contracted by P&E Mining Consultants Inc., and with an office address of 9052 Mortlake-Ararat Road, Ararat, Victoria, Australia, 3377, do certify that:

1. This certificate applies to the technical report titled "Murray Brook Project NI 43-101 Technical Report & Preliminary Economic Assessment, New Brunswick, Canada," with an effective date of June 20, 2025 (the "Technical Report").

2. I am a graduate of RMIT University of Melbourne, Victoria, Australia, with a B.Sc. in Applied Geology. I have worked as a geologist for over 19 years since obtaining my B.Sc. degree. I am a geological consultant currently licensed by Engineers and Geoscientists British Columbia (License No. 40875) and Professional Engineers and Geoscientists Newfoundland & Labrador (License No. 08399) and Northwest Territories and Nunavut Association of Professional Engineers and Geoscientists (License No. L3874). I am also a member of the Australasian Institute of Mining and Metallurgy of Australia (Member No. 305397).

3. I have read the definition of "Qualified Person" set out in National Instrument 43-101 ("NI 43-101") and certify that by reason of my education, affiliation with a professional association (as defined in NI 43-101) and past relevant work experience, I fulfil the requirements to be a "Qualified Person" for the purposes of NI 43-101.

4. My relevant experience for the purpose of the Technical Report is:

- Geologist, Foran Mining Corp. 2004
- Geologist, Aurelian Resources Inc. 2004
- Geologist, Linear Gold Corp. 2005-2006
- Geologist, Búscore Consulting 2006-2007
- Consulting Geologist (AusIMM) 2008-2014
- Consulting Geologist, P.Geo. (EGBC/AusIMM) 2014-Present

   I have specialized in GIS and have also been heavily involved in and ran quality control programs for various drilling exploration projects. I have also built, managed and refined small and large exploration databases.

5. I have not visited the Property that is the subject of this Technical Report.

6. I am responsible for Sections 1.7, 11, all of 12 except 12.1.5, 25.7, and coauthor of Section 27 of the Technical Report.

7. I am independent of the Issuer applying the test in Section 1.5 of NI 43-101.

8. I have had prior involvement with the Project that is the subject of this Technical Report. I was a "Qualified Person" for a Technical Report titled "Technical Report and Updated Mineral Resource Estimate of the Murray Brook Zn-Pb-Cu-Ag Project New Brunswick, Canada" for Canadian Copper Inc. with an effective date of October 3, 2023.

9. I have read NI 43-101 and Form 43-101F1 and the Technical Report has been prepared in compliance therewith.

10. As of the effective date of this Technical Report, to the best of my knowledge, information and belief, the Technical Report contains all scientific and technical information that is required to be disclosed to make the Technical Report not misleading.

Dated: July 7, 2025

"Signed and sealed"

Jarita Barry, P.Geo.

CERTIFICATE OF QUALIFIED PERSON
Eugene Puritch, P.Eng., FEC, CET

I, Eugene J. Puritch, P.Eng., FEC, CET, as an independent mining consultant and President of P&E Mining Consultants Inc. and with an office address of 44 Turtlecreek Blvd., Brampton, Ontario, L6W 3X7, do certify that:

1. This certificate applies to the technical report titled "Murray Brook Project NI 43-101 Technical Report & Preliminary Economic Assessment, New Brunswick, Canada," with an effective date of June 20, 2025 (the "Technical Report").

2. I am a graduate of The Haileybury School of Mines, with a Technologist Diploma in Mining, as well as obtaining an additional year of undergraduate education in Mine Engineering at Queen's University. In addition, I have also met the Professional Engineers of Ontario Academic Requirement Committee's Examination requirement for a Bachelor's degree in Engineering Equivalency. I am a mining consultant currently licensed by the: Professional Engineers and Geoscientists New Brunswick (License No. 4778); Professional Engineers, Geoscientists Newfoundland and Labrador (License No. 5998); Association of Professional Engineers and Geoscientists Saskatchewan (License No. 16216); Ontario Association of Certified Engineering Technicians and Technologists (License No. 45252); Professional Engineers of Ontario (License No. 100014010); Association of Professional Engineers and Geoscientists of British Columbia (License No. 42912); and Northwest Territories and Nunavut Association of Professional Engineers and Geoscientists (No. L3877). I am also a member of the National Canadian Institute of Mining and Metallurgy.

3. I have read the definition of "Qualified Person" set out in National Instrument 43-101 ("NI 43-101") and certify that, by reason of my education, affiliation with a professional association (as defined in NI 43-101) and past relevant work experience, I fulfil the requirements to be a "Qualified Person" for the purposes of NI 43-101.

4. I have practiced my profession continuously since 1978. My summarized career experience is as follows:

- Mining Technologist - H.B.M.& S. and Inco Ltd., ... 1978-1980
- Open Pit Mine Engineer – Cassiar Asbestos/Brinco Ltd., ... 1981-1983
- Pit Engineer/Drill & Blast Supervisor – Detour Lake Mine,.......... 1984-1986
- Self-Employed Mining Consultant – Timmins Area, ... 1987-1988
- Mine Designer/Resource Estimator – Dynatec/CMD/Bharti,.......... 1989-1995
- Self-Employed Mining Consultant/Resource-Reserve Estimator, ... 1995-2004
- President – P&E Mining Consultants Inc,.......... 2004-Present

5. I have visited the Property that is the subject of this Technical Report on March 18, 2013.

6. I am responsible for Sections 2.4.2, 14.14, 14.15, 14.16 and coauthor of Section 27 of the Technical Report.

7. I am independent of Canadian Copper Inc. as independence is defined in Section 1.5 of NI 43-101.

8. I have had prior involvement with the Project that is the subject of this Technical Report. I have had prior involvement with the Project that is the subject of this Technical Report. I was a “Qualified Person” for a Technical Report titled “Technical Report and Updated Mineral Resource Estimate of the Murray Brook Zn-Pb-Cu-Ag Project New Brunswick, Canada” for Canadian Copper Inc. with an effective date of October 3, 2023; and “Technical Report and Preliminary Economic Assessment of the Murray Brook Project New Brunswick, Canada” for Votorantim Metals Canada Inc. and El Nino Ventures Inc., with an effective date of June 4, 2013; and “Technical Report on the Murray Brook Property, Restigouche County New Brunswick, Canada” for El Nino Ventures Inc., dated April 13, 2013.

9. I have read NI 43-101 and Form 43-101F1. This Technical Report has been prepared in compliance therewith.

10. As of the effective date of this Technical Report, to the best of my knowledge, information and belief, the Technical Report contains all scientific and technical information that is required to be disclosed to make the Technical Report not misleading.

Dated: July 7, 2025

“Signed and sealed”

Eugene Puritch, P.Eng., FEC, CET

CERTIFICATE OF QUALIFIED PERSON
William Stone, Ph.D., P.Geo.

I, William Stone, Ph.D., P.Geo., as an independent geological consultant contracted by P&E Mining Consultants Inc. and with an office address of 4361 Latimer Crescent, Burlington, Ontario, do certify that:

1. This certificate applies to the technical report titled "Murray Brook Project NI 43-101 Technical Report & Preliminary Economic Assessment, New Brunswick, Canada," with an effective date of June 20, 2025 (the "Technical Report").

2. I am a graduate of Dalhousie University with a Bachelor of Science (Honours) degree in Geology (1983). In addition, I have a Master of Science in Geology (1985) and a Ph.D. in Geology (1988) from the University of Western Ontario. I have worked as a geologist for a total of 35 years since obtaining my M.Sc. degree. I am a geological consultant currently licensed by the Professional Geoscientists of Ontario (License No 1569).

3. I have read the definition of "Qualified Person" set out in National Instrument 43-101 ("NI 43-101") and certify that, by reason of my education, affiliation with a professional association (as defined in NI 43-101) and past relevant work experience, I fulfil the requirements to be a "Qualified Person" for the purposes of NI 43-101.

4. My relevant experience for the purpose of the Technical Report is:

- Contract Senior Geologist, LAC Minerals Exploration Ltd. 1985-1988
- Post-Doctoral Fellow, McMaster University 1988-1992
- Contract Senior Geologist, Outokumpu Mines and Metals Ltd. 1993-1996
- Senior Research Geologist, WMC Resources Ltd. 1996-2001
- Senior Lecturer, University of Western Australia 2001-2003
- Principal Geologist, Geoinformatics Exploration Ltd. 2003-2004
- Vice President Exploration, Nevada Star Resources Inc. 2005-2006
- Vice President Exploration, Goldbrook Ventures Inc. 2006-2008
- Vice President Exploration, North American Palladium Ltd. 2008-2009
- Vice President Exploration, Magma Metals Ltd. 2010-2011
- President & COO, Pacific North West Capital Corp. 2011-2014
- Consulting Geologist 2013-2017
- Senior Project Geologist, Anglo American 2017-2019
- Consulting Geoscientist 2020-Present

5. I have not visited the Property that is the subject of this Technical Report.

6. I am responsible for Sections 1.3 to 1.6, 1.17, 2.6, 3.1, 4 to 10, 23, 25.1 to 25.6, 25.17, 26.2, and coauthor of Section 27 of the Technical Report.

7. I am independent of the Issuer applying the test in Section 1.5 of NI 43-101.

8. I have had prior involvement with the Property that is the subject of this Technical Report. I was a "Qualified Person" for a Technical Report titled "Technical Report and Updated Mineral Resource Estimate of the Murray Brook Zn-Pb-Cu-Ag Project New Brunswick, Canada" for Canadian Copper Inc., with an effective date of October 3, 2023. I was an executive with El Nino Ventures Inc. from 2011 to 2013 and a geological consultant to El Nino Ventures Inc. from 2014 to 2017 and from 2019 to 2022.

9. I have read NI 43-101 and Form 43-101F1 and this Technical Report has been prepared in compliance therewith.

10. As of the effective date of this Technical Report, to the best of my knowledge, information and belief, the Technical Report contains all scientific and technical information that is required to be disclosed to make the Technical Report not misleading.

Dated: July 7, 2025

"Signed and sealed"

William E. Stone, Ph.D., P.Geo.

CERTIFICATE OF QUALIFIED PERSON
Yungang Wu, P.Geo.

I, Yungang Wu, P.Geo., as an independent consulting geologist contracted by P&E Mining Consultants Inc., and with an office address of 3246 Preserve Drive, Oakville, Ontario, L6M 0X3, do certify that:

1. This certificate applies to the technical report titled "Murray Brook Project NI 43-101 Technical Report & Preliminary Economic Assessment, New Brunswick, Canada," with an effective date of June 20, 2025 (the "Technical Report").

2. I am a graduate of Jilin University, China, with a Master's degree in Mineral Deposits (1992). I have worked as a geologist for 33 years. I am a geological consultant and a registered practising member of the Association of Professional Geoscientists of Ontario (Registration No. 1681).

3. I have read the definition of "Qualified Person" set out in National Instrument 43-101 ("NI 43-101") and certify that, by reason of my education, affiliation with a professional association (as defined in NI 43-101) and past relevant work experience, I fulfil the requirements to be a "Qualified Person" for the purposes of NI 43-101.

4. My relevant experience for the purpose of the Technical Report is as follows:

- Geologist –Geology and Mineral Bureau, Liaoning Province, China 1992-1993
- Senior Geologist – Committee of Mineral Resources and Reserves of Liaoning, China 1993-1998
- VP – Institute of Mineral Resources and Land Planning, Liaoning, China 1998-2001
- Project Geologist–Exploration Division, De Beers Canada 2003-2009
- Mine Geologist – Victor Diamond Mine, De Beers Canada 2009-2011
- Resource Geologist– Coffey Mining Canada 2011-2012
- Consulting Geologist 2012-Present

   I specialize in resource modelling using Gemcom and Datamine software, grade control and mineral exploration, including project management, planning, desk top studies, sampling, logging, drilling and interpretation.

5. I have visited the Property that is the subject of this Technical Report on September 7 and 8, 2023.

6. I am responsible for Sections 1.9, 2.4.3, 12.1.5, all of 14 except 14.14 to 14.16, 25.9, 25.18.1.1 and coauthor of Section 27 of the Technical Report.

7. I am independent of the Issuer applying the test in Section 1.5 of NI 43-101. I am independent of the Vendor and the Property.

8. I have had prior involvement with the Project that is the subject of this Technical Report. I was a "Qualified Person" for a Technical Report titled "Technical Report and Updated Mineral Resource Estimate of the Murray Brook Zn-

Pb-Cu-Ag Project New Brunswick, Canada" for Canadian Copper Inc. with an effective date of October 3, 2023; and "Technical Report and Preliminary Economic Assessment of the Murray Brook Project New Brunswick, Canada" for Votorantim Metals Canada Inc. and El Nino Ventures Inc., with an effective date of June 4, 2013; and "Technical Report on the Murray Brook Property, Restigouche County New Brunswick, Canada" for El Nino Ventures Inc., dated April 13, 2013.

9. I have read NI 43-101 and Form 43-101F1 and the Technical Report has been prepared in compliance therewith.

10. As of the effective date of this Technical Report, to the best of my knowledge, information and belief, the Technical Report contains all scientific and technical information that is required to be disclosed to make the Technical Report not misleading.

Dated: July 7, 2025

"Signed and sealed"

Yungang Wu, P.Geo.

CERTIFICATE OF QUALIFIED PERSON
Andrew Bradfield, P.Eng.

I, Andrew Bradfield, P. Eng., as an independent mining engineer contracted by P&E Mining Consultants Inc., and with an office address of 5 Patrick Drive, Erin, Ontario, Canada, N0B 1T0, do hereby certify that:

1. This certificate applies to the technical report titled "Murray Brook Project NI 43-101 Technical Report & Preliminary Economic Assessment, New Brunswick, Canada," with an effective date of June 20, 2025 (the "Technical Report").
2. I am a graduate of Queen's University, with an honours B.Sc. degree in Mining Engineering in 1982. I have practiced engineering continuously since 1982 (43 years). I am a Professional Engineer of Ontario (License No.4894507). I am also a member of the National CIM.
3. I have read the definition of "Qualified Person" set out in National Instrument 43-101 ("NI 43-101") and certify that, by reason of my education, affiliation with a professional association (as defined in NI 43-101) and past relevant work experience, I fulfil the requirements to be a "Qualified Person" for the purposes of NI 43-101.
4. My summarized career experience is as follows:
   - Various Engineering Positions – Palabora Mining Company,.. 1982-1986
   - Mines Project Engineer – Falconbridge Limited,.. 1986-1987
   - Senior Mining Engineer – William Hill Mining Consultants Limited,.. 1987-1990
   - Independent Mining Engineer,.. 1990-1991
   - GM Toronto – Bharti Engineering Associates Inc,.. 1991-1996
   - VP Technical Services, GM of Australian Operations – William Resources Inc,.................................. 1996-1999
   - Independent Mining Engineer,.. 1999-2001
   - Principal Mining Engineer – SRK Consulting,... 2001-2003
   - COO – China Diamond Corp, ... 2003-2006
   - VP Operations – TVI Pacific Inc,.. 2006-2008
   - COO – Avion Gold Corporation, ... 2008-2012
   - Independent Mining Engineer,...2012-Present

   My experience includes mine start-up, development and operations, technical and financial evaluations, and royalty transactions. Prior to joining P&E, I was the Vice President Operations or Chief Operating Officer at several junior mining companies, including Avion Gold Corporation.
5. I have not visited the Property that is the subject of this Technical Report.
6. I am responsible for authoring Sections 1.10, 16, 18.3.4, 18.4, 18.9, 18.10, 18.12, 21.2.2.1, 21.2.2.4, 21.2.6.1, 21.2.7.1, 25.10, 25.18.1.3, 25.18.1.6, 25.18.2.2, 26.4, and coauthor of Section 27 of this Technical Report.

7. I am independent of the Issuer applying the test in Section 1.5 of NI 43-101. I am independent of the Vendor and the Property.
8. I have had no prior involvement with the Project that is the subject of this Technical Report.
9. I have read NI 43-101 and Form 43-101F1. This Technical Report has been prepared in compliance therewith.
10. As of the effective date of this Technical Report, to the best of my knowledge, information and belief, the Technical Report contains all scientific and technical information that is required to be disclosed to make the Technical Report not misleading.

Dated: July 7, 2025

"Signed and sealed"

Andrew Bradfield, P.Eng.

CERTIFICATE OF QUALIFIED PERSON
Pierre Lacombe, P.Eng.

I, Pierre Lacombe, P.Eng., as an independent consultant of Canadian Copper Inc., and with an office address of 450 Gouin St., St-Bruno, QC, J3V 6C8, do certify that:

1. This certificate applies to the technical report titled "Murray Brook Project NI 43-101 Technical Report & Preliminary Economic Assessment, New Brunswick, Canada," with an effective date of June 20, 2025 (the "Technical Report").
2. I graduated from Ecole Polytechnique of Montreal with a B. Eng. in Mining Engineering in 1984.
3. I am a professional engineer registered with the Ordre des Ingénieurs of Québec (No. 39496).
4. I have practiced my profession continuously for 40 years with experience in the testwork program elaboration and interpretation, design and operations of base metals processing facilities. The operating experience was notably acquired over 12 years, in total, with some spent with Cominco (Pine Point Mines, Polaris Mine) as metallurgist; BP Minerals' Selbaie Mines, as assistant plant superintendent; Aur Resources' Louvicourt Mines as client's representative during testwork and design phases of the processing facilities and then plant superintendent. The plant design and optimization experience is related to four years as Director, Metallurgy for Cambior Chile; 11 years as Principal Process Engineer with the consulting firm AMEC; a further two in a similar role with BBA and, most recently, seven years as Group Metallurgist for Lundin Mining Corp. Executive positions held with Pershimco Resources and Scorpio Mining, over a span of three years, plus work as an independent engineering consultant complete my profile.
5. I have read the definition of "Qualified Person" set out in the National Instrument 43-101 Standards of Disclosure for Mineral Projects ("NI 43-101") and certify that by virtue of my education, affiliation to a professional association and past relevant work experience, I fulfil the requirements to be a "Qualified Person" for those sections of the Technical Report that I am responsible for preparing.
6. I have visited the project site on October 3, 2024.
7. I am responsible for Sections 1.8, 2.4.4, 13, 25.8, 25.18.1.2, 25.18.2.1, 26.3, and coauthor of Section 27 of the Technical Report.
8. I am independent of Canadian Copper Inc. as independence is defined in Section 1.5 of NI 43-101.
9. I have not been previously involved with the Murray Brook Project.
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 that Instrument. As of 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 those sections of the Technical Report not misleading.

Dated: July 7, 2025

"Signed and sealed"

Pierre Lacombe, P.Eng.

CERTIFICATE OF QUALIFIED PERSON
Marcello Locatelli, P.Eng.

I, Marcello Locatelli, P.Eng., certify that:

1. I am employed as a VP Projects with Inteloc Inc., with an office address of 17 Stargell Drive, Whitby, L1N 7X4, ON, Canada.
2. This certificate applies to the technical report titled "Murray Brook Project NI 43-101 Technical Report & Preliminary Economic Assessment, New Brunswick, Canada," with an effective date of June 20, 2025 (the "Technical Report"). I graduated from University of Johannesburg with a bachelor's degree in mechanical engineering in 2007.
3. I am a professional engineer registered with the Professional Engineers Ontario (No. 100196154).
4. I have practiced my profession continuously for 17 years with experience in brownfield and green field mineral processing and infrastructure projects.
5. I have read the definition of "Qualified Person" set out in the National Instrument 43-101 Standards of Disclosure for Mineral Projects ("NI 43-101") and certify that by virtue of my education, affiliation to a professional association and past relevant work experience, I fulfil the requirements to be a "Qualified Person" for those sections of the Technical Report that I am responsible for preparing.
6. I have visited the project site (8-10 December 2024).
7. I am responsible for Sections 2.4.5, 18.3.3, 21.2.3.1, 21.2.3.2.1, 21.2.5.1.1, 25.2.5.2.2, 21.2.5.2.4, 21.2.6.3, and 21.2.7.2 and 21.2.7.5.1 of the Technical Report.
8. I am independent of Canadian Copper Inc. as independence is defined in Section 1.5 of NI 43-101.
9. I have not been previously involved with the Murray Brook Project.
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 that Instrument. As of 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 those sections of the Technical Report not misleading.

Dated: July 7, 2025

"Signed and sealed"

Marcello Locatelli, P.Eng.

CERTIFICATE OF QUALIFIED PERSON
Jeff Gilchrist, P.Eng.

I, Jeffrey Gilchrist, P.Eng., as a Geotechnical Engineer with Stantec Consulting Ltd. (Stantec), with an office address of 845 Prospect St., Fredericton, New Brunswick, do certify that:

1. This certificate applies to the technical report titled “Murray Brook Project NI 43-101 Technical Report & Preliminary Economic Assessment, New Brunswick, Canada,” with an effective date of June 20, 2025 (the “Technical Report”).
2. I graduated from University of New Brunswick with a B.Sc. in Engineering in 2008.
3. I am a professional engineer registered with the Association of Professional Engineers and Geoscientists of New Brunswick (M7478).
4. I have practiced my profession continuously for 16 years with Stantec. I have experience in tailings dam planning and design, and construction monitoring for tailings dams.
5. I have read the definition of “Qualified Person” set out in the National Instrument 43-101 Standards of Disclosure for Mineral Projects (“NI 43-101”) and certify that by virtue of my education, affiliation to a professional association and past relevant work experience, I fulfil the requirements to be a “Qualified Person” for those sections of the Technical Report that I am responsible for preparing. READ ONLY
6. I have not visited the Murray Brook Mine site but have visited the Caribou Mine Site (the location of the proposed TSFs) in May of 2025.
7. I am responsible for Sections 1.12.2, 1.14.3 - portions related to Caribou Mine closure, 2.4.6, 18.7, 18.8, 20.4.1, 21.2.7.3, 21.2.7.4, 21.2.7.5.2, 21.2.7.5.3 of the Technical Report.
8. I am independent of Canadian Copper Inc. as independence is defined in Section 1.5 of NI 43-101.
9. I have not been previously involved with the Murray Brook Project.
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 that Instrument. As of 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 those sections of the Technical Report not misleading.

Dated: July 7, 2025

“Signed and sealed”

Jeffrey Gilchrist, P.Eng.

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1 SUMMARY

1.1 Introduction

Canadian Copper Inc. (Canadian Copper) commissioned Ausenco Engineering Canada ULC (Ausenco) to compile a preliminary economic assessment (PEA) of the Murray Brook Project. The PEA was prepared in accordance with the Canadian disclosure requirements of National Instrument 43-101 (NI 43-101) and the requirements of Form 43-101 F1. The responsibilities of the engineering consultants and firms who are providing qualified persons are as follows:

- Ausenco managed and coordinated the work related to the report. Ausenco also developed the PEA-level design and cost estimate for the process plant restart, design criteria for processing the Murray Brook mineralized material, copper circuit installation completion, Murray Brook site water management infrastructure, environmental planning, assessment, licensing and permitting. Ausenco conducted the economic analysis based on the capital and operating costs compiled for the PEA.
- P&E Mining Consultants Inc (P&E) designed the open pit, accessibility, local resources, geological setting, deposit type, exploration work, drilling, exploration works, sample preparation and analysis, data verification, mineral resource estimate, mining methods, mine production schedules, and mine capital and operating costs.
- Pierre Lacombe reviewed the metallurgical testwork and developed the recovery models used in the economic assessment.
- Inteloc Inc. (Inteloc) reviewed the condition of the existing process plant and assessed the plant for maintenance, replacements and repairs required to restart the process plant to develop a cost estimate for restarting the existing plant.
- Stantec Consulting Ltd. (Stantec) completed the sections related to tailings and water management at the Caribou site including closure costs related to the tailings facility.

1.2 Terms of Reference

The report supports disclosures by Canadian Copper in a news release dated May 22, 2025, entitled, “Canadian Copper’s Combined Strategy PEA Delivers After-tax C$171M NPV7%, 36% IRR”.

All measurement units used in this report are International System (SI) units and all currencies are expressed in Canadian dollars (symbol: C$; currency abbreviation: CAD) unless otherwise stated.

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 (CIM, 2014) and the CIM Estimation of Mineral Resources and Mineral Reserves Best Practice Guidelines (CIM, 2019).

The Murray Brook Project consist of two sites, Caribou and Murray Brook. The process plant, tailings management facility, and other infrastructure are located at the Caribou site. The deposit and mine are located at the Murray Brook site.

1.3 Location, Land Tenure, Access, Physiography

The property is approximately 60 km west of the City of Bathurst in the Parish of Balmoral, Restigouche County, New Brunswick, Canada, and consists of surveyed Mineral Lease No. 252, which covers approximately 505 ha, and Murray Book East Claim 4925, which covers an area of 5,082 ha.

Canadian Copper acquired the Murray Brook Joint Venture Property through separate agreements with Votorantim Metals Canada (VMC) and MetalQuest Mining (MQM) for 100% ownership of the property. On August 2, 2023, Canadian Copper successfully executed a definitive purchase agreement to acquire VMC's 72% interest in the Murray Brook Joint Venture. Canadian Copper and VMC agreed to the conditions for the transaction under the Letter of Intent signed February 13, 2023.

Canadian Copper announced on September 12, 2023 its intention to acquire MQM's (previously El Nino Ventures Inc.) 28% interest in the Murray Brook Joint Venture. Canadian Copper and MQM agreed to considerations under a Letter of Intent signed September 11, 2023, subject to exchange approvals and the execution of a definitive purchase agreement. When Canadian Copper satisfied the conditions of the purchase agreement, it completed its purchase of the remaining 28% interest in the Murray Brook Joint Venture. Canadian Copper now controls 100% of the Murray Brook Joint Venture Property.

On October 28, 2024, Canadian Copper announced the signing of a term sheet and exclusivity agreement providing it the exclusive right to acquire the Caribou processing plant complex, which is 10 km east of Murray Brook. The Caribou complex would be used to produce copper, lead, and zinc concentrates with recoverable silver from the Murray Brook mineralized feed material. The closing date for the transaction is October 1, 2025.

The property is located in the Miramichi Highlands, which is characterized by rounded and glacially scoured hills. Land use in the area is mainly for tourism, forestry, and mining. The property is accessible for exploration work and project development year-round and water is available. Although the site has been reclaimed after the 1990s silver-gold mining activities and the equipment has been sold off, part of the historical open pit mine, leach pads, and the tailings management area are still evident. The massive sulphide deposit has never been mined.

A 10 kV powerline links Murray Brook to the power station at the Caribou mine, 10 km to the east. However, service was discontinued in 1996.

1.4 Geology and Mineralization

The Murray Brook area is located in the Bathurst mining camp in northern New Brunswick. The Bathurst mining camp is hosted in an Ordovician back-arc complex of poly-deformed sedimentary, felsic volcanic, and mafic volcanic rocks that are collectively referred to as the Bathurst Supergroup. The sedimentary and volcanic rocks have been intruded by gabbro, diabase, and quartz porphyritic rocks of Ordovician age. The Bathurst mining camp includes at least 46 volcanogenic massive sulphide (VMS) deposits, including the world-renowned Brunswick No. 12 mine.

The Murray Brook deposit is hosted by sedimentary rocks in the lower part of the Mount Brittain Formation. The upper felsic volcanic member of the Mount Brittain Formation is host to the Restigouche deposit, 10 km to the west. The

Mount Brittain Formation is considered to be equivalent of the Spruce Lake Formation, which hosts the Caribou mine 10 km to the east.

The Murray Brook deposit dips moderately to the northwest, plunges shallowly to the north and appears to pinch out at depth and to the east. The geometry of the deposit was probably lens-shaped, but the up-dip portion of the body has been eroded and pre-Pleistocene weathering has produced a gossan. Whereas the deposit is a single body of massive sulphide, drilling by VMC indicates that it consists of two connected thick lenses or lobes; the western lens is richer in zinc and lead, and the eastern lens is richer in copper.

The sulphides are massive to semi-massive, and include copper, pyrite, and lead-zinc zones. Sulphides are mainly fine-grained, massive, laminated pyrite with disseminated and banded sphalerite, chalcopyrite, and galena, with minor tetrahedrite, covellite, marcasite, and arsenopyrite. The Murray Brook sulphide mineralization is classified as a sedimentary rock-hosted volcanogenic massive sulphide deposit.

1.5 History

Kennco Exploration Ltd. staked what is now the Murray Brook area in 1955 to follow up on airborne conductors. The conductors are due to graphitic shale, but mineral prospecting led to the discovery of gossan boulders and a stream sediment survey led to the discovery of the massive sulphide deposit in 1956. The deposit was systematically drilled by Kennco, and they produced a historical mineral resource estimate.

Several companies carried out drilling programs and metallurgical testwork between 1970 and 1989, at which time NovaGold Resources Ltd. began production on the gossan to recover gold and silver. The underlying massive sulphide deposit was not mined. The mining operation was discontinued in 1992 and the open pit was reclaimed.

Further work on the sulphide deposit, including ground surveys, diamond drilling, and tonnage and grade calculations continued intermittently until 2010, when Votorantim Metals Canada Inc. (VMC) acquired the Murray Brook property. Later in 2010, VMC signed a participation agreement for the Murray Brook property with El Nino Ventures Inc. (ELN). VMC carried out drilling, mineral processing, and metallurgical studies, and in 2013, completed a Mineral Resource Estimate and a preliminary economic assessment of the Murray Brook Project in accordance with NI 43-101.

In 2016, Puma Exploration Inc. (Puma) signed a deal with VMC and ELN for the Murray Brook property. Puma completed an updated mineral resource estimate, drilling, and mineral processing and metallurgical studies as part of a strategic agreement with Trevali Mining Corp. (Trevali), who was the owner and operator of the nearby Caribou mining and process plant operation 10 km to the east of the Murray Brook deposit. The Murray Brook property reverted back to VMC and ELN in 2020. Canadian Copper acquired VMC’s ownership interest (72%) and ELN’s ownership interest (28%) in the Murray Brook property in 2023.

1.6 Exploration and Drilling

Mineral prospecting, geological mapping, trenching, soil geochemistry, ground geophysical surveys, and airborne geophysical exploration surveys were completed by VMC and Puma between 2010 and 2020. The ground and airborne geophysical surveys included magnetics, gravity and electromagnetics, which, in combination with ongoing geological and geochemistry work, led to the development of targets for drill testing. Since 2010, 190 drill holes for 37,070 m have been completed on the Murray Brook property: 162 drill holes for 29,938.7 m were completed by VMC between 2010 and 2016; and 28 drill holes totalling 7,131 m were completed by Puma between 2017 and 2019. As of the effective date of this report, Canadian Copper has not conducted any drilling on the Murray Brook property.

1.7 Sampling, Analyses and Data Verification

Robust quality assurance and quality control (QA/QC) programs have been used since the start of exploration activities on the property in 2010. In the author's opinion that the sample preparation, analytical and security procedures, and QA/QC program meet industry standards, and that the data are of good quality and satisfactory for use in the mineral resource estimate reported in this technical report.

Mr. Yungang Wu, P.Geo., of P&E, an independent qualified person under the terms of NI 43-101, conducted a site visit to the Murray Brook property on September 7 and 8, 2023. The site visit included an inspection of the property, drill sites, drill collars, and drill core storage facilities. A data verification sampling program was completed as part of the on-site review. Drill core samples were collected to independently confirm the presence and grades of base and precious metal mineralization. Previously, Mr. Eugene Puritch, P.Eng., of P&E, an independent qualified person under the terms of NI 43-101, conducted a site visit to the property on March 18, 2013. A data verification sampling program was conducted as part of the on-site review.

The authors consider that there is good correlation between gold and silver assay values in the project database and the independent verification samples. In the authors' opinion, the data are of good quality and appropriate for use in the current mineral resource estimate.

1.8 Mineral Processing and Metallurgical Testwork

Three phases of metallurgical testwork were completed. Two are historical and benefitted from the availability of fresh core material. This work highlighted the presence of an oxidized layer sitting above the mineralized zones, in zinc activation. This zinc activation has proven inhibitive to differential flotation of the sulphide minerals. The samples exhibiting this response were excluded from further development work. Most of the tests carried out at that time involved samples requiring either the production of a copper-zinc or lead-zinc concentrate combinations, with limited samples exhibiting sufficient grades of these metals at once to allow for a full sequential flotation scheme to be developed.

The third phase was commissioned by Canadian Copper and completed in early 2025 at SGS Lakefield. One objective of the testwork program was to match, as best as possible, the configuration of the nearby Caribou mine processing facility. The earlier sequential flotation of copper and then lead was replaced by a bulk flotation scheme, which involved cleaning this bulk product and then separating it into a lead concentrate as the floated product, with the copper concentrate representing the tailings of the separation stage. This was followed by zinc flotation on the bulk rougher tails and rejects from the bulk cleaning circuit.

The samples for the 2025 testwork program, however, were from 2011-2012 cores and the availability of intercepts was limited to what could be retrieved. One of the four samples selected included material from the oxide cap, with its typical in-situ zinc activation preventing further work. With the other three samples, bulk and zinc roughing optimization work proceeded and provided interesting insights as to the role of the various depressants, pH modifiers, and collectors. Not until the testwork efforts were transferred towards the cleaning stages was it realized that the age of the samples rendered the pyrite sufficiently activated to prevent further advance towards the production of upgraded final concentrates. A similar response had been outlined through aging tests, performed from a previous testwork program, where all the economic minerals were still amenable to flotation in a selective manner but pyrite would inflate the mass pulls achieved from the roughing stages and prevented proper cleaning. Liberation analysis confirmed that all the economic minerals were properly liberated, with the pyrite in particular clearly floating because of chemical activation.

Enough evidence has been accumulated from the rougher tests from the last testwork campaign and historical testwork to garner confidence in the capability of recovering copper, lead, and zinc from the Murray Brook material. The complex and fine-grained mineralogy of this massive sulphide deposit requires sufficient efforts to be expended on grinding and regrinding to liberate the minerals of interest from the abundant pyrite to which they are intimately tied. This characteristic will limit the maximum recovery achievable, especially for lead and copper, while achieving marketable final concentrate grades.

Cleaning the zinc proved to be relatively straightforward, while lead managed final concentrate grades in the 45% to 50% Pb range, which is acceptable for smelters. Copper upgrading was more elusive but typically entertained with feed grades that did not warrant the production of a separate copper concentrate when it was preferable to combine it with the lead-bearing product to improve payable lead and silver recoveries. The samples gathered for the recent program were meant to assess the optimum mean of producing three concentrates from samples carrying better copper grades and carry through the proof of an efficient copper-lead separation scheme from the bulk copper-lead concentrate. The aging issue encountered with the samples prevented the program from assessing this capability, but it can be re-investigated with future samples taken from fresh material.

Expected recovery levels at the roughing stages for all economic metals have been fairly well defined by the testwork to date. Floating a bulk concentrate Cu-Pb, instead of the sequential route, is also ensuring a high rougher recovery levels for both metals as it is not necessary to introduce depressant against one or another at the roughing stage. All rougher recovery predictions can then be aligned with an expectation of minimum remaining metal grade in the rougher tails, as incorporated in the methodology adopted for projecting the metal recoveries and typical of plant operations. The main area requiring further work remains the upgrading capability of separate copper and lead concentrates in the bulk cleaning circuit and the separation circuit as the limitations brought by the intimate mineralogical assemblage of copper and lead with pyrite will dictate the extent to which recovery has to be sacrificed to achieve the minimum level of upgrading required to produce saleable concentrates.

1.9 Mineral Resource Estimate

This technical report includes a mineral resource estimate prepared in accordance with NI 43-101 for sulphide and oxide mineralization at a C$23/t net smelter return (NSR) cut-off (Table 1-1). The mineral resource estimate was prepared by the authors in accordance with the Canadian Institute of Mining, Metallurgy and Petroleum’s (CIM) Definition Standards and National Instrument 43-101 – Standards of Disclosure for Mineral Projects (“43-101”) (2014) and Best Practices Guidelines (2019). The effective date of the mineral resource is October 3, 2023.

Table 1-1: Murray Brook In-Pit Mineral Resource Estimate on at C$23/t NSR Cut-off

Zone	Classification	Tonnes (k)	Cu (%)	Cu (Mlb)	Zn (%)	Zn (Mlb)	Pb (%)	Pb (Mlb)	Au (g/t)	Au (koz)	Ag (g/t)	Ag (Moz)	ZnEq (%)	CuEq (%)	NSR (C$/t)
Oxide	Measured	1,641	1.05	37.9	2.2	79.6	0.73	26.6	0.36	19	38	2	5.94	1.85	156
	Indicated	373	0.97	7.9	2.31	19	0.78	6.4	0.51	6	44.7	0.5	6.02	1.88	158
	Measured + Indicated	2,014	1.03	45.9	2.22	98.6	0.74	32.9	0.39	25	39.2	2.5	5.95	1.86	157
Sulphide	Measured	15,830	0.43	150.8	2.6	908.3	0.92	322.2	0.52	264	39	19.8	4.83	1.51	115
	Indicated	5,275	0.52	60.9	2.14	248.9	0.85	98.9	0.67	114	37.3	6.3	4.58	1.43	114
	Measured + Indicated	21,105	0.45	211.7	2.49	1,157.2	0.91	421.1	0.56	378	38.6	26.2	4.77	1.49	115
	Inferred	110	0.41	1	1.82	4.4	0.68	1.6	0.62	2	30.4	0.1	3.75	1.17	92
Total	**Measured**	**17,471**	**0.49**	**188.7**	**2.56**	**987.9**	**0.91**	**348.8**	**0.5**	**283**	**38.9**	**21.8**	**4.93**	**1.54**	**119**
	Indicated	**5,648**	**0.55**	**68.9**	**2.15**	**267.8**	**0.85**	**105.3**	**0.66**	**120**	**37.8**	**6.9**	**4.68**	**1.46**	**117**
	Measured + Indicated	**23,119**	**0.51**	**257.5**	**2.46**	**1,255.70**	**0.89**	**454.1**	**0.54**	**403**	**38.6**	**28.7**	**4.87**	**1.52**	**118**
	Inferred	**110**	**0.41**	**1**	**1.82**	**4.4**	**0.68**	**1.6**	**0.62**	**2**	**30.4**	**0.1**	**3.75**	**1.17**	**92**

Notes: **1.** Mineral resources, which are not mineral reserves, do not have demonstrated economic viability. **2.** The estimate of mineral resources may be materially affected by environmental, permitting, legal, title, taxation, socio-political, marketing, or other relevant issues. **3.** The inferred mineral resource in this estimate has a lower level of confidence than that applied to an indicated mineral resource and must not be converted to a mineral reserve. It is reasonably expected that the majority of the inferred mineral resource could be upgraded to an indicated mineral resource with continued exploration. **4.** The mineral resources in this report were estimated using the Canadian Institute of Mining, Metallurgy and Petroleum (CIM), CIM Standards on Mineral Resources and Reserves, Definitions and Guidelines (2019) prepared by the CIM Standing Committee on Reserve Definitions and adopted by the CIM Council. **5.** Totals of tonnage and contained metal may differ due to rounding. **6.** NSR $/t = (Cu % x 81) + (Pb % x 12) + (Zn % x 13) + (Ag g/t x 0.90). **7.** CuEq = Cu%/0.30; and ZnEq = Zn%/0.52

The mineral resource estimate for the sulphide mineralization consists of 15.8 Mt grading 2.60% Zn, 0.43% Cu, 0.92% Pb, 0.52 g/t Au, and 39.0 g/t Ag (4.83% ZnEq or 1.51% CuEq) in measured mineral resources; 5.3 Mt grading of 2.14% Zn, 0.52% Cu, 0.85% Pb, 0.67 g/t Au, and 37.3 g/t Ag (4.58% ZnEq or 1.43% CuEq) in indicated mineral resources; and 0.1 Mt grading 1.82% Zn, 0.41% Cu, 0.68% Pb, 0.62 g/t Au, and 30.4 g/t Ag (3.75% ZnEq or 1.17% CuEq) in inferred mineral resources. Oxide mineralization consists of 1.6 Mt grading 2.2% Zn, 1.05% Cu, 0.73% Pb, 0.36 g/t Au, and 38.0 g/t Ag (5.94% ZnEq or 1.85% CuEq) in measured mineral resources and 0.4 Mt grading 2.31% Zn, 0.97% Cu, 0.78% Pb, 0.51 g/t Au, and 44.7 g/t Ag (6.02% ZnEq or 1.88% CuEq) in indicated mineral resources.

Mineral resources, which are not mineral reserves, do not have demonstrated economic viability.

The drilling database for the Murray Brook Project contains 12,900 samples, all of which were analysed for copper (%), lead (%), zinc (%), gold (g/t), and silver (g/t). A total of 10,300 assays from 165 drill holes have been used for the mineral resource estimate; 146 drill holes were completed prior to 2013.

Grade capping was investigated on 1 m composite values within the constraining domains to ensure that the possible influence of erratic high values did not bias the database. Based on the log-normal histogram performance, lead was capped at 16%, zinc at 24%, and silver at 410 g/t, whereas capping was not applied to copper and gold.

The Murray Brook mineral resource block model was constructed using Gemcom™ GEOVIA GEMS™ modelling software. The block model is oriented with an x-axis at 110° azimuth with 3 m x 3 m x 3 m blocks. Inverse distance squared (ID^2) grade interpolation was utilized for the copper, lead, and zinc grade estimates, whereas inverse distance cubed (ID^3) was used for the gold and silver grade estimates, both with the capped composites. The average block-model mineralized bulk density was calculated to be 4.24 t/m^3.

The mineral resource model classification was determined from the zinc interpolation due to zinc generating the highest proportionate contribution to the NSR value in the block model. Based on the semi-variogram performance and density of the drilling data, the measured resource classification was justified for blocks interpolated using at least four drill holes with average spacing less than 25 m. Indicated mineral resource blocks were classified with the second pass and inferred mineral resources were classified for all remaining constrained blocks.

The NSR cut-off sensitivities to the pit-constrained mineral resource estimate are presented in Table 1-2.

Table 1-2: In-Pit Mineral Resource Estimate Sensitivity

Zone	Classification	Cut-off NSR (C$/t)	Tonnes (k)	Cu (%)	Cu (Mlb)	Zn (%)	Zn (Mlb)	Pb (%)	Pb (Mlb)	Au (g/t)	Au (koz)	Ag (g/t)	Ag (Moz)
Oxide	Measured	100	1,115	1.31	32.3	2.81	69.1	0.94	23.1	0.38	14	46.9	1.7
		50	1,540	1.10	37.4	2.31	78.5	0.77	26.2	0.36	18	39.6	2.0
		45	1,563	1.09	37.5	2.29	78.9	0.76	26.3	0.36	18	39.2	2.0
		40	1,587	1.08	37.7	2.26	79.2	0.76	26.4	0.36	19	38.9	2.0
		35	1,608	1.07	37.8	2.24	79.4	0.75	26.5	0.36	19	38.5	2.0
		30	1,627	1.06	37.8	2.22	79.5	0.74	26.5	0.36	19	38.2	2.0
		25	1,638	1.05	37.9	2.20	79.6	0.74	26.6	0.36	19	38.0	2.0
		23	1,641	1.05	37.9	2.20	79.6	0.73	26.6	0.36	19	38.0	2.0
	Indicated	100	260	1.20	6.8	2.78	15.9	0.93	5.3	0.54	4	56.3	0.5
		50	358	1.00	7.9	2.37	18.7	0.80	6.3	0.52	6	46.1	0.5
		45	362	0.99	7.9	2.36	18.9	0.79	6.3	0.52	6	45.7	0.5
		40	366	0.98	7.9	2.35	18.9	0.79	6.4	0.52	6	45.4	0.5
		35	367	0.98	7.9	2.34	19.0	0.79	6.4	0.51	6	45.3	0.5
		30	369	0.97	7.9	2.33	19.0	0.78	6.4	0.51	6	45.1	0.5
		25	372	0.97	7.9	2.32	19.0	0.78	6.4	0.51	6	44.8	0.5
		23	373	0.97	7.9	2.31	19.0	0.78	6.4	0.51	6	44.7	0.5
Sulphide	Measured	100	7,910	0.56	97.6	3.80	662.3	1.40	243.3	0.68	172	57.2	14.5
		50	13,995	0.46	142.1	2.85	878.1	1.02	313.9	0.56	251	42.6	19.2
		45	14,524	0.45	144.9	2.78	889.3	0.99	317.0	0.55	255	41.5	19.4
		40	14,982	0.45	147.2	2.72	897.5	0.97	319.3	0.54	259	40.6	19.6
		35	15,349	0.44	148.9	2.67	902.9	0.95	320.7	0.53	261	39.9	19.7
		30	15,633	0.44	150.1	2.63	906.3	0.93	321.7	0.52	263	39.4	19.8
		25	15,795	0.43	150.7	2.61	908.0	0.93	322.1	0.52	264	39.0	19.8
		23	15,830	0.43	150.8	2.60	908.3	0.92	322.2	0.52	264	39.0	19.8
	Indicated	100	2,720	0.70	41.9	2.94	176.3	1.22	73.0	0.94	82	53.0	4.6
		50	4,707	0.56	58.3	2.30	239.1	0.92	95.9	0.73	110	40.6	6.1
		45	4,861	0.55	59.1	2.26	242.5	0.90	96.9	0.71	112	39.7	6.2
		40	5,009	0.54	59.9	2.22	245.2	0.89	97.8	0.70	113	38.8	6.3
		35	5,112	0.54	60.3	2.19	246.8	0.87	98.3	0.69	113	38.3	6.3
		30	5,202	0.53	60.7	2.16	248.0	0.86	98.7	0.68	114	37.8	6.3
		25	5,258	0.53	60.9	2.14	248.7	0.85	98.9	0.68	114	37.4	6.3
		23	5,275	0.52	60.9	2.14	248.9	0.85	98.9	0.67	114	37.3	6.3
	Inferred	100	45	0.37	0.4	2.70	2.7	1.08	1.1	0.92	1	45.1	0.1
		50	98	0.42	0.9	1.98	4.3	0.74	1.6	0.67	2	33.1	0.1
		45	101	0.42	0.9	1.94	4.3	0.73	1.6	0.65	2	32.5	0.1
		40	103	0.42	0.9	1.92	4.4	0.72	1.6	0.65	2	32.0	0.1
		35	105	0.41	1.0	1.89	4.4	0.71	1.6	0.64	2	31.6	0.1
		30	107	0.41	1.0	1.86	4.4	0.69	1.6	0.63	2	31.0	0.1
		25	110	0.41	1.0	1.82	4.4	0.68	1.6	0.62	2	30.4	0.1
		23	110	0.41	1.0	1.82	4.4	0.68	1.6	0.62	2	30.4	0.1

Note: Highlighted rows are the base case. All sensitivity NSR cut-off values are greater than the base case, and are therefore reasonable prospects for eventual economic extraction (RPEEE).

1.10 Mining Methods

The Murray Brook Project will consist of an open pit mining operation. One year of pre-production mining of waste material is planned, followed by 14 years of production. Sulphide feed to the Caribou process plant is planned at 0.9 million tonnes (Mt) in the first year of production, followed by 1.2 Mt/a (3,300 t/d) for the remaining production years. The total open pit process plant feed is planned at 15.5 Mt over the life of mine and consists of 79.1% measured mineral resource and 20.9% indicated mineral resource. Metals from oxide material cannot be economically recovered in the Caribou process plant so the material is planned to be placed on the PAG waste rock storage facility.

A mining contractor will be engaged to mine the Murray Brook open pit. The contractor will undertake all drilling and blasting, loading, hauling, and mine site maintenance activities. The Owner will provide overall mine management and technical services. Mining will typically be done on 6 m height benches using conventional equipment such as hydraulic excavators, front-end loaders, and 65 t haulage trucks. Dilution in mineralized material is estimated at 3% and mining losses are estimated at 5%. Over the life of mine, the pit will produce 15.5 Mt of process plant feed material grading 41.1 g/t Ag, 0.47% Cu, 0.96% Pb, and 2.67% Zn. Gold is not considered to be recoverable/payable in this PEA. The average life-of-mine NSR is estimated at $103.5/t. The process feed will be truck hauled to the Caribou plant, located 17 km east of the open pit.

The open pit will produce 5.9 Mt of overburden, 60.1 Mt of NAG waste rock, and 11.8 Mt of PAG waste rock. Total waste material mined is 77.7 Mt with a life-of-mine strip ratio of 5:1. The overburden, NAG waste, and PAG waste materials will be stored in separate storage locations in proximity of the open pit. There will be no temporary process feed stockpile created near the open pit. All mineralized material will be transported directly to the Caribou plant and placed into a run-of-mine stockpile near the primary crusher.

1.11 Recovery Methods

The proposed processing strategy involves processing mineralized material from the Murray Brook deposit at the existing Caribou processing complex approximately 10 km to the east. The process flowsheet design is based on the current Caribou concentrator process flow design, with minimal modification to enable a 10% increase in mill feed design capacity (3,000 to 3,300 t/d) and the recovery of a copper concentrate (in addition to the zinc and lead concentrates recovered in the historical operation).

The design operating availability for grinding and flotation is 92% or 8,059 operating hours per year. The concentrate filter operating availability is 84% or 7,358 hours per year.

The design is supported by preliminary historical testwork and operating data, industrial standard practices, and financial evaluations to optimize concentrate recovery while minimizing capital expenditure and life-of-mine operating costs. The unit operations are standard technologies widely used in polymetallic concentrators.

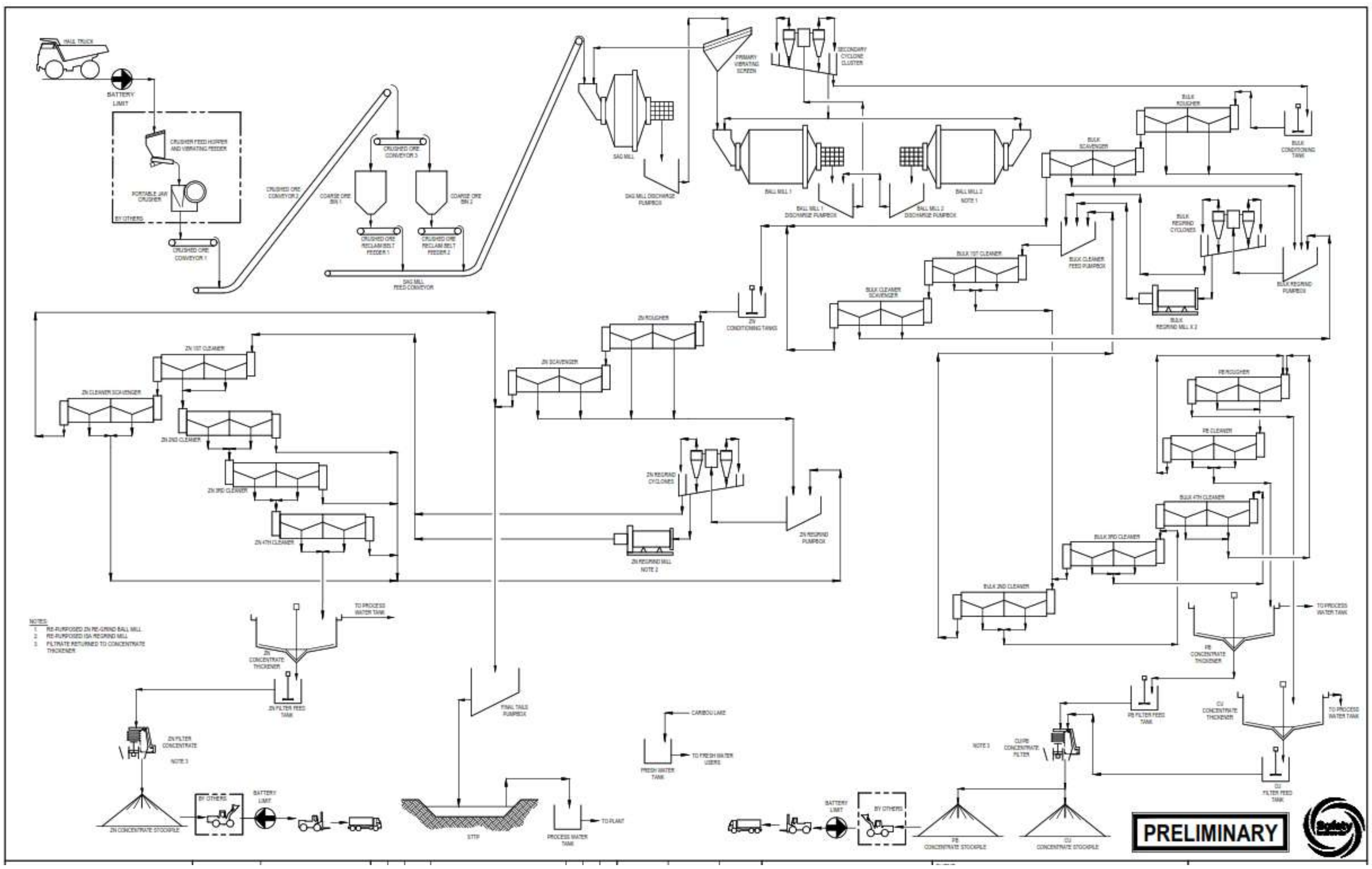
The major circuits are as follows:

- jaw crushing of run-of-mine material (by service provider)

- SAG mill with trommel and classification screens, followed by a ball mill with cyclone classification
- lead and copper bulk flotation with regrinding prior to cleaner flotation
- lead and copper separation flotation
- zinc flotation with regrinding prior to cleaner flotation
- thickening, filtration and loading of copper, lead, and zinc concentrates
- tailings pumping and disposal.

A preliminary simplified process flow diagram is shown on Figure 1-1. Key design criteria are summarized in Table 1-3.

Figure 1-1: Simplified Overall Process Flowsheet



Source: Ausenco (2025).

Table 1-3: Key Process Design Criteria

Design Parameter	Units	Value
Throughput	Mt/a	1.2
	t/d	3,300
Run-of-Mine head Grade – Design		
– Copper	%	0.57
– Lead	%	0.90
– Zinc	%	3.02
Operating Availabilities		
– Grinding and Flotation	%	92
– Filtration	%	84
Semi-autogenous Grind Mill Feed Storage Bins Residence Time	h	16
Run-of-Mine Production Specific Gravity	-	4.42
JK SMC Axb	-	63.6
Bond Rod Mill Work Index, Design	kWh/t	15.7
Bond Ball Mill Work Index, Design	kWh/t	12.3
Bond Abrasion Index, Design	g	0.40
Crushing Circuit Product Size, P_{80}	mm	102
Primary Grind Size, P_{80}	µm	30
Design Copper and Lead Flotation Residence Times		
– Bulk Rougher and Scavenger	min	25
– Bulk Cleaner 1	min	32
– Bulk Cleaner 1 Scavenger	min	14
– Bulk Cleaner 2	min	18
– Bulk Cleaner 3	min	14
– Bulk Cleaner 4	min	18
– Lead Rougher	min	21
– Lead Cleaner	min	14
Copper and Lead Bulk Flotation Regrind Product Size, P_{80}	µm	12
Bulk Flotation Regrind Mill Specific Energy	kWh/t	24
Design Zinc Flotation Residence Times, Design		
– Zinc Rougher and Scavenger	min	18
– Zinc Cleaner 1	min	18
– Zinc Cleaner 1 Scavenger	min	18
– Zinc Cleaner 2	min	12
– Zinc Cleaner 3	min	18
– Zinc Cleaner 4	min	18
Zinc Flotation Regrind Product Size, P_{80}	µm	20
Bulk Flotation Regrind Mill Specific Energy	kWh/t	9
Metal Recoveries to Concentrate		
– Zinc to Zinc Concentrate	%	83
– Copper to Copper Concentrate	%	72
– Lead to Lead Concentrate	%	43
Target Concentrate Grades		
– Zinc Concentrate	%	48
– Copper Concentrate	%	22
– Lead Concentrate	%	45
Target Concentrate Filter Cake Moistures		
– Zinc Concentrate	% w/w	8.5
– Copper Concentrate	% w/w	8.0
– Lead Concentrate	% w/w	9.2

1.12 Project Infrastructure

1.12.1 Project Infrastructure

Infrastructure to support the Murray Brook Project consists of existing infrastructure related to the Caribou process plant as well as new infrastructure to support the Murray Brook mine site.

The existing project infrastructure at the Caribou site includes:

- a site access road, and other facility roads
- process facilities include the process plant, crusher facilities, process plant workshop, assay laboratory, and tailings management facility
- an administration office, warehouse, maintenance shop
- water treatment systems and building
- high-voltage powerline and on-site electrical substation
- diversions, ditches, ponds, and effluent treatment for management of contact and redirection of non-contact water
- potable water, fire water, compressed air, power, diesel, communication, and sanitary systems.

The Murray Brook mine site project infrastructure will include:

- site civil work
- diversions, ditches, ponds, and effluent treatment to manage contact water
- mine facilities include offices, maintenance shop, and storage facilities
- electrical power provided by on-site generation.

1.12.2 Tailings Infrastructure

Processing for the Murray Brook mine will be completed at the Caribou milling complex. Tailings will be piped as slurry and deposited subaqueously in the tailings management facilities at the Caribou mine. During the first three years of production, tailings will be deposited in the existing South Tributary tailings pond, which will be raised by others prior to the start of mining. After Year 3, the remaining life-of-mine tailings will be stored in a new facility contained by a geomembrane-faced rockfill dam constructed in a valley along the north tributary to Forty Mile Brook. This new facility—the North Tributary tailings pond—will include diversion structures, an emergency spillway, and a polishing pond dam. Locally sourced materials will be used to construct the dams and geomembranes will be included for low-permeability elements due to the uncertainty of available low-permeability construction materials. The conceptual dam design includes layers of rockfill, filter materials, native borrow material, and geomembranes, with upstream slope protection.

Due to the likelihood of the tailings having acid-generating potential, a 1 m water cover above the tailings has been assumed for this preliminary design. In addition, 1 m of freeboard was also allowed above the operational water level.

1.13 Market Studies

Project economics were estimated based on long-term metal prices of US$27.00/oz Ag, US$4.25/lb Cu, US$1.1/lb Pb, and US$1.30/lb Zn. These metal prices were based on long-term market forecasts listed on the May 14, 2025 Scotiabank Metals and Mining Research Daily and are consistent with historical prices for these commodities.

No contracts for transportation are currently in place but if negotiated, are expected to be within industry norms. Similarly, no contracts are currently in place for the supply of reagents, utilities, or other bulk commodities required to construct and operate the project.

An off-take agreement for one-third of the concentrate is currently in place, with terms to be determined.

Indicative pricing for the transportation and sale of concentrates was obtained from established concentrate marketing specialists for all concentrates.

1.14 Environmental, Permitting and Community

The project will consist of the Murray Brook mine site and the Caribou milling complex connected to each other via a 10 km haul road. The process plant, tailings management facility, and other infrastructure are located at the Caribou site. The deposit and mine are located at the Murray Brook site and which has not yet been permitted. The Caribou complex has been operating intermittently since the 1970s, with the concentrator commissioning in 1990. The New Brunswick government has managed the care and maintenance of the Caribou complex since January 2023. The Murray Brook site operated as a small open pit and was closed in 1992 after less than two years in operation, when the open pit was reclaimed.

The property is approximately 60 km west of the City of Bathurst in the Parish of Balmoral, Restigouche County, New Brunswick, Canada, and consists of combined lease and claim area of 5,082 ha. The property is located in the Miramichi Highlands, which is characterized by rounded and glacially scoured hills. Land use in the area is mainly for tourism, forestry, and mining. The property is accessible for exploration work and project development year-round and water is plentiful in nearby streams and creeks.

1.14.1 Environmental Considerations

Field studies at the Murray Brook site have been initiated in 2025 including bird, bat and terrestrial wildlife surveys. Field studies are ongoing. Desktop studies assumed a study area that consisted of a five km buffer area around the proposed project. Sources of information for the desktop studies included publicly available information obtained primarily from provincial government sources. Some minor information that was utilized included a groundwater seepage study and some assumptions regarding geochemical characteristics of the deposit.

The study area has a typically continental climate with cold winters with a mean temperature of -13°C in January and summer average daytime highs between 20°C and 28°C. The Murray Brook area generally receives an abundance of snow (300 and 400 cm) and approximately 1,200 mm average rainfall per year.

The provincial database indicates numerous watercourses, waterbodies, and wetlands throughout the study area, many of which are adjacent (less than 30 m) to the proposed project development footprint. The study area does not contain any protected watersheds. Until such time as site-based fisheries investigation is conducted, the identified watercourses should be considered as fish habitat and frequented by fish.

Regarding Species at Risk (SAR), the desktop study revealed 11 species at risk from a total of 40 observations consisting of one mammal, nine birds, and one vascular plant. The study area includes a portion of the project footprint that intersects with an area of critical habitat for Bicknell’s thrush which is a bird listed as threatened and imperilled. Field based studies to develop a deeper understanding of how the bird interacts with the habitat and implications for development within the project area are required, and have been initiated, along with early engagement with federal and provincial authorities.

A search of the Canadian Protected and Conserved Areas Database revealed no protected areas or parks within the study area, no Important Bird Areas (IBAs), and no Key Biodiversity Areas (KBAs) within the study area.

A desktop study related to socio-economic features of the study area revealed a relatively uninhabited area with small communities within an area of industrial forestry use. Small-scale fishing, hunting, and recreational ATV and snowmobile use occurs within the study area; there is a network of ATV trails and shelters in the area. Commercial infrastructure within the study area includes some minor commercial (restaurant/inn) and a number of recreational cottages.

The nearest First Nation community to the Project Area is the Pabineau First Nation (PFN) community, located near Bathurst and about 55 km west of the project site. In October 2024, Canadian Copper signed a non-binding memorandum of understanding with the PFN that allows for the development of open lines of communication and mutual collaboration and thereby facilitating economic benefits for the PFN through job creation, contracting opportunities, and also through potential business and investment partnerships.

1.14.2 Permitting Considerations

The Murray Brook Project, based on the current development strategy, would be subject to mostly provincial regulatory requirements. The project will not likely trigger a federal impact assessment under the *Impact Assessment Act* based on planned production levels and near-term increases in footprint. It is possible that some of the proposed project related activities will trigger a HADD (harmful alteration, disruption, or destruction of fish habitat) and Fish Habitat Compensation Plan. In addition, there may be a requirement for a wetland offset plan based on provincial requirements. However, it may be possible to adjust the project infrastructure footprint to minimize these risks. It is unlikely that the project as envisioned currently will initially trigger a Schedule 2 Amendment under the Metal and Diamond Mining Effluent Regulations, however, in the longer term, a Schedule 2 Amendment may be triggered due to the requirement later in the mine life for a new tailings management facility.

It is anticipated that a SARA permit will be required should project activities interact negatively with critical Bicknell's thrush habitat. Early engagement with regulators is recommended to reduce delays to the project schedule as well as site-based field work and data collection. It may be possible to adjust the project infrastructure footprint to minimize risks.

The anticipated provincial acts and regulations that regulate mining projects in New Brunswick include:

- Environmental Impact Assessment Regulation – *Clean Environment Act*
- Water Quality Regulation – *Clean Environment Act*
- *New Brunswick Species at Risk Act*
- *New Brunswick Clean Water Act*
- *New Brunswick Heritage Conservation Act*
- New Brunswick Mining Act.

At a minimum, it is likely that an EIA registration will be required for the Murray Brook Project. An EIA registration will require environmental field studies followed by several months to prepare and submit an EIA Registration document and then iterative discussions with the provincial government and requests for additional information. The most favourable outcome of the EIA is a Certificate of Determination (approval of the project with conditions).

Following the fulfilment of the EA Regulation requirements, pursuant to the Water Quality Regulation (Regulation 82-126) under the *Clean Environment Act*, the construction and operation of the TMF requires an approval to construct/operate to be obtained to construct and operate the Murray Brook and Caribou mine sites. The approval to operate defines terms and conditions (such as discharge limits, testing and monitoring requirements, reporting requirements, and the like) that a facility must comply with as part of its operation in order to remain in compliance with the regulations

Currently, the Caribou mine site is operating under Approval to Operate I-12715 issued to NB DNRED, which is valid until March 29, 2028. The Murray Brook site is operating under Approval to Operate I-12407, issued to Canadian Copper, which is valid until March 31, 2029. It is unclear if the project could be approved as a revision to these existing approvals to operate or if a separate consolidated approval to construct/operate would be required. Early discussions with NBDELG will help to clarify this point.

1.14.3 Closure and Reclamation Planning

In New Brunswick, the exploration and development of Crown-owned minerals falls under the *New Brunswick Mining Act*. In accordance with the New Brunswick Regulation 86-98-General Regulation of the *Mining Act*. Mining developments require an approved program for the protection, reclamation, and rehabilitation of the environment (also known as a reclamation plan).

Reclamation for the Canadian Copper assets at the Caribou mine site include demolition of the existing water treatment plant, milling complex, laboratory and administration building as well grading and revegetation of these areas. It is understood that other areas and liabilities are the responsibility of others as part of Canadian Copper's limited liability agreement discussions with the Province of New Brunswick.

Due to the acid-generating nature of the tailings, and in alignment with the existing closure strategy for the Caribou mine, the proposed closure concept for the tailings management facility includes raising the dam to provide an enhanced water cover over the tailings and to submerge historical liabilities such as waste rock, legacy tailings, and the underground mine portal to reduce impacts from these sources.

Reclamation activities related to the TMFs at this stage include various upgrades to dams and sludge cells at the site which are related to water treatment as well as long-term monitoring. Perpetual water treatment has been assumed as part of the closure plan, unless future monitoring and site conditions demonstrate that treatment is no longer necessary.

The Murray Brook mineral lease is the site of past production from open-pit mining of the oxidized surficial portion of the Murray Brook deposit. The current liability associated with the Murray Brook property consists of short-term monitoring liability and long-term restoration liability. Additional liability will be incurred once the Murray Brook site is developed including flooding of the open pit, water treatment, pit water overflow via a closure spillway, management of PAG waste rock, vegetative covers on disturbed land and waste rock covers, and dismantling and removal of site buildings, and re-contoured and revegetating.

1.15 Capital and Operating Costs

The capital and operating cost estimates presented in this PEA provide substantiated costs that can be used to assess the preliminary economics of the Murray Brook Project. The estimates are based on the development of an open pit mine and related mining infrastructure at the Murray Brook site, and the recommissioning of an existing process plant, tailings management facility, and related infrastructure, as well as Owner's costs and contingency.

All capital and operating cost estimates are reported in Canadian dollars (C$, or CAD), with no allowance for escalation or exchange rate fluctuations. An exchange rate of 0.746 (CAD:USD) and 0.65 (CAD:EUR) has been applied as necessary.

The capital cost estimate conforms to Class 5 guidelines for a preliminary economic assessment level estimate with a ±50% accuracy according to the Association for the Advancement of Cost Engineering International (AACE International). The capital cost estimate was developed in Q2 2025 Canadian dollars based on Ausenco's in-house database of projects and studies as well as experience from similar operations.

The total initial capital for the Murray Brook Project is C$63.7 million and the life-of-mine sustaining cost is C$48.9 million. The initial capital cost summary by WBS is presented in Table 1-4.

The operating cost estimate has been developed in Canadian dollars with a base date of Q2 2025. No allowance has been included for escalation. The estimate includes mining, processing, and general and administrative (G&A) costs.

Key information used to support the cost estimates include the following:

- production rate of 1.2 Mt/a
- for unit costs quoted in US dollars, an exchange rate of 1.34 USD/CAD
- reagent consumption based on the project mass balance, supported by metallurgical testwork
- grinding media consumption rates estimated based on the mineralized material characteristics
- unit cost estimates for reagents and consumables made by vendor quotation or in-house data from similar projects in the area
- annual maintenance cost estimated as a factor of the total capital cost estimate
- power consumption based on the project mechanical equipment list, with the power unit cost obtained from the local utility provider (NB Power)
- the staffing plan, including positions, provided by Canadian Copper to determine the labour cost
- salaries estimated by Ausenco based on in-house data from reference projects in the area
- mobile equipment cost provides for fuel and maintenance, not for purchase or vehicle lease
- crushing cost estimated on a per tonne processed basis, provided by a local rental equipment provider
- water treatment costs provided by Stantec
- G&A costs that were benchmarked against previous similar projects.

The overall life-of-mine operating cost is C$1,228 million over 13.2 years for an average of C$79.3/t processed. Table 1-5 provides a summary of the project operating costs.

Table 1-4: Summary of Capital Cost Estimate

WBS	WBS Description	Capital Cost ($M)	Sustaining Cost ($M)	Total Cost ($M)
1000	Mine	29.1	5.4	34.5
2000	Water Treatment	0.2	0.0	0.2
3000	Process Plant	7.2	2.8	10.0
4000	On-Site Infrastructure	0.9	0.0	0.9
5000	Off-Site Infrastructure	0	29.9	29.9
Total Direct Costs		**37.4**	**38.1**	**75.5**
6000	Indirects	5.8	0.1	5.9
7000	Project Delivery	4.2	4.5	8.7
8000	Owner's Costs	2.1	0.0	2.1
Total Indirect Costs		**12.1**	**4.6**	**16.7**
Total Direct + Indirect Costs		**49.5**	**42.6**	**92.2**
9000	Contingency	8.2	6.3	14.5
Total Capital Cost Excluding Mill Purchase		**57.7**	**48.9**	**106.6**
Mill and Mill Infrastructure Purchase Cost		**6.0**	**0.0**	**6.0**
Total Capital Cost Including Mill Purchase		**63.7**	**48.9**	**112.6**

Table 1-5: Operating Cost Summary (Life-of-Mine Average)

Cost Area	Total (C$M/a)	C$/t Processed	% of Total
Mining	48.9	41.7	52.5
Processing	36.6	31.2	39.3
G&A	7.6	6.5	8.2
Total	**93.1**	**79.3**	**100.0**

1.16 Economic Analysis

The 2025 PEA is preliminary in nature and there is no certainty that the 2025 PEA will be realized. Mineral resources that are not mineral reserves do not have demonstrated economic viability.

The results of the economic analyses represent forward-looking information as defined under Canadian securities law. The results depend on inputs that are subject to known and unknown risks, uncertainties, and other factors that may cause actual results to differ materially from those presented here.

The project was evaluated using a discounted cash flow analysis based on a 7% discount rate. Cash inflows consisted of annual revenue projections. Cash outflows consisted of capital expenditures, including pre-production costs, operating costs, taxes, and royalties. These were subtracted from the inflows to arrive at the annual cash flow projections. Cash flows were taken to occur at the mid-point of each period. It must be noted that tax calculations involve complex variables that can only be accurately determined during operations and, as such, the actual post-tax results may differ from those estimated. A sensitivity analysis was performed to assess the impact of variations in metal prices, discount rate, operating costs, initial capital costs, metal recovery, and head grade.

The capital and operating cost estimates are presented in Section 21 of this report. The economic analysis was carried out on a constant dollar basis with no inflation and a discount rate of 7%.

The economic analysis included in this technical report have been updated from the press release issued on May 22, 2025 by Canadian Copper. The updates included:

- the addition of a 0.67% royalty, payable to MQM (combined VMC and MQM royalties increased from 0.58% to 1.25%)
- the refinement of the corporate tax rate from 30% to 29% to reflect New Brunswick corporate tax rates.

These refinements result in changes to the pre-tax and post-tax NPV and IRR. The updated NPV and IRR are as follows:

- Pre-tax NPV discounted at 7% is $256.1 million, the IRR is 47.6%, and the payback period is 1.6 years. On a post-tax basis, the NPV discounted at 7% is $169.0 million, the IRR is 35.4%, and payback period is 2.0 years. Project economics are summarized in Table 1-6.

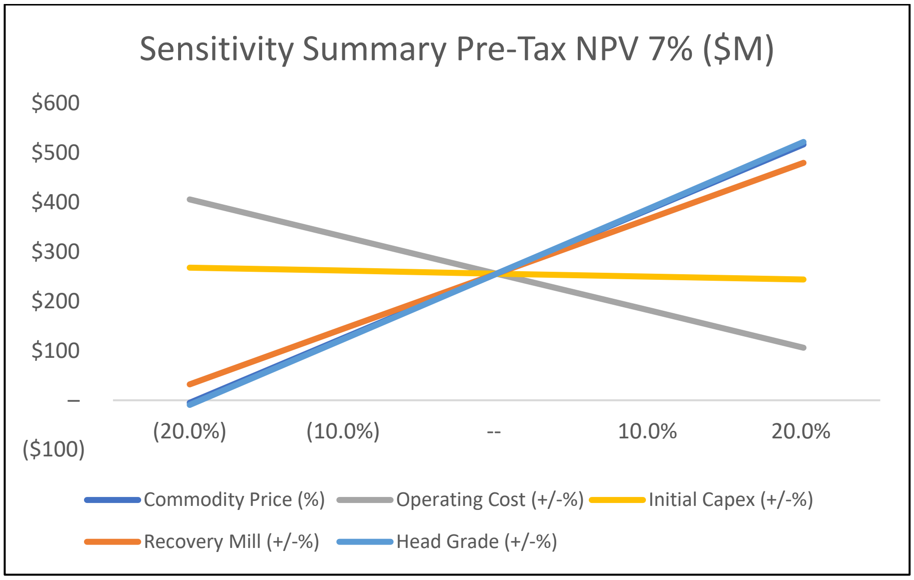
Table 1-6: Economic Analysis Summary

Area	Description	Unit of Measure	Total/Average
General	Copper Price	US$/lb	4.25
	Silver Price	US$/oz	27.00
	Lead Price	US$/lb	1.10
	Zinc Price	US$/lb	1.30
	FX rate	CAD:USD	0.746
Production	Mine Life	years	13.2
	Total Resource Mined	**kt**	**15,486**
	Life-of-Mine Copper Equivalent Grade	%	1.91
	Total Waste Mined	kt	77,725
	Strip Ratio	Waste: Mineral	5.02:1
	Average Annual Mined Resource	kt/a	1,173
	Total Recovery Copper	%	68.3
	Total Recovery Silver	%	55.4
	Total Recovery Lead	%	44.7
	Total Recovery Zinc	%	82.3
	Total Payable Copper	**Mlbs**	**103.8**
	Total Payable Copper Equivalent	**Mlbs**	**396.0**
	Average Annual Payable Copper	Mlbs/a	8
	Average Annual Payable Copper Equivalent	Mlbs/a	30
	Total Payable Zinc	**Mlbs**	**626**
	Total Payable Zinc Equivalent	**Mlbs**	**1,296**
	Average Annual Payable Zinc	Mlbs/a	47
	Average Annual Payable Zinc Equivalent	Mlbs/a	98
Revenue/Costs	Total Revenue	$M	2,258.1
	Average Annual Revenue	$M	171.1
	Copper Revenue (as % of Gross Revenue)	%	26
	Zinc Revenue (as % of Gross Revenue)	%	48
	EBITDA	$M	676.8
	Average Annual EBITDA	$M	51.3
	Total On-Site Operating Costs (Mining, Process, G&A)	**$M**	**1,228**
	Average Annual On-Site Operating Cost	$M	93.05
	Mining Unit Cost	$/t mined	7.20
	Mining Unit Cost	$/t milled	41.66
	Process Unit Cost	$/t milled	31.18
	G&A Unit Cost	$/t milled	6.49
	Total Off Site Operating Costs (Transport, Treatment & Refining)	**$M**	**325.8**
Cash Costs	Total Cash Cost*	US$/lb CuEq	3.0
	All-in Sustaining Cost**	US$/lb CuEq	3.2
	Total Cash Cost*	US$/oz ZnEq	0.9
	All-In Sustaining Cost**	US$/oz ZnEq	1.0
Capital Costs	Initial Capital Cost (Total)	$M	64.00
	Initial Capital Cost – Mining	$M	29.12
	Initial Capital Cost – Other	$M	34.58
	Sustaining Capital Cost	$M	48.9
	Sustaining Capital Cost – Mining	$M	1.0
	Sustaining Capital Cost – Other	$M	47.9
	Closure Cost	$M	52.6
Pre-Tax Economics	NPV (7%)	$M	256.1
	IRR	%	47.6
	Payback	years	1.6
	NPV/Initial Capital	-	4.0
Post-Tax Economics	NPV (7%)	$M	169.0
	IRR	%	35.4
	Payback	years	2.0
	NPV/Initial Capital	-	2.7

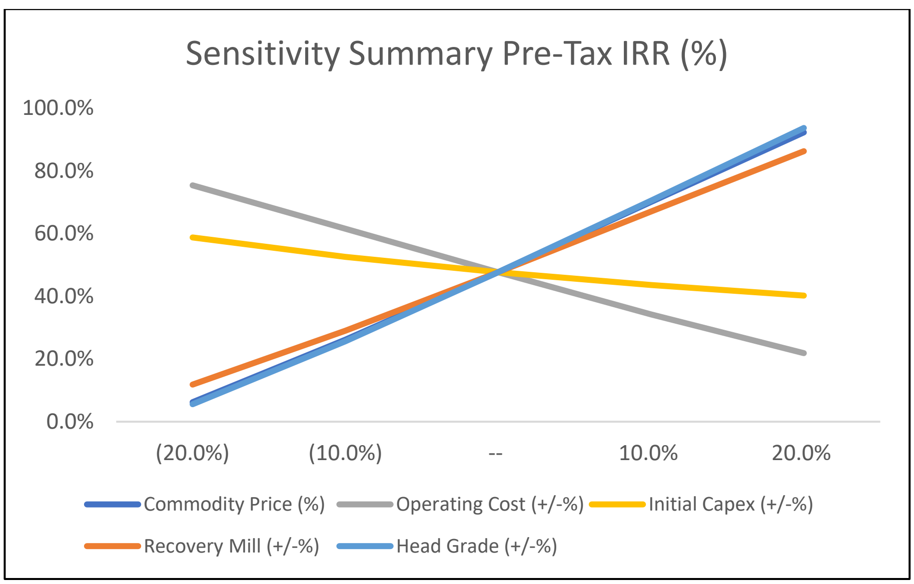
Notes: [1]Total cash costs consist of mining costs, processing costs, mine-level G&A, royalties, and off-site charges. [2]AISC includes cash costs plus sustaining capital and closure costs.

A sensitivity analysis was conducted on the pre-tax and post-tax NPV and IRR of the project using metal prices, discount rate, operating costs, initial capital cost, metal recovery, and head grade as variables. As presented in Figures 1-2 and 1-3, the analysis showed that the project is most sensitive to changes commodity price, head grade, and recovery.

Figure 1-2: Pre-Tax NPV, IRR Sensitivity Results

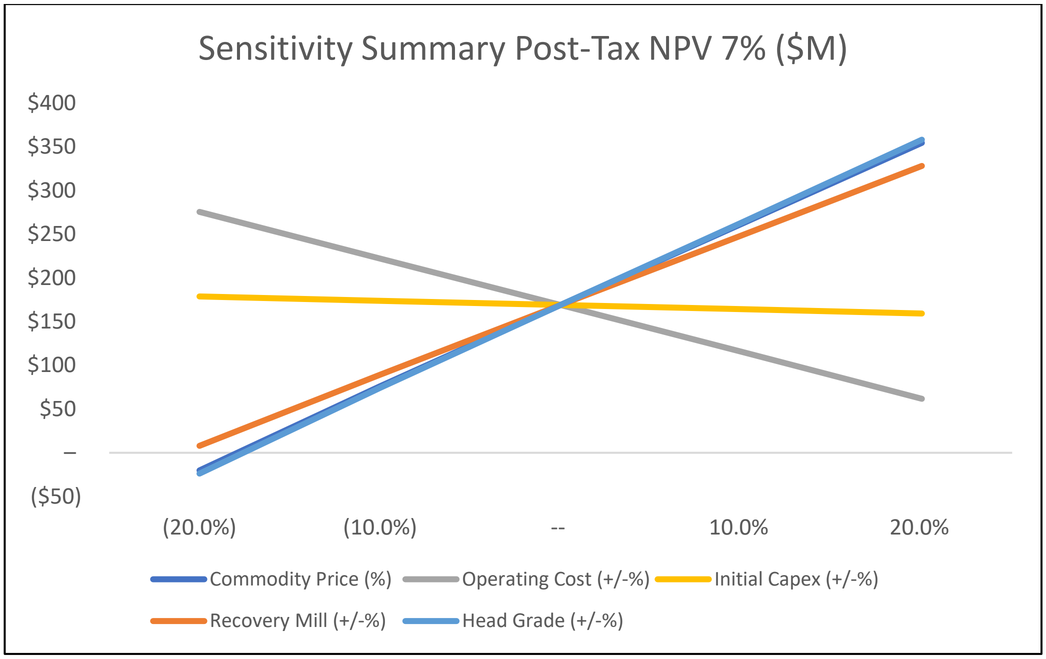


Note: Series lines for commodity price and head grade overlap in the above figures. Source: Ausenco, 2025.

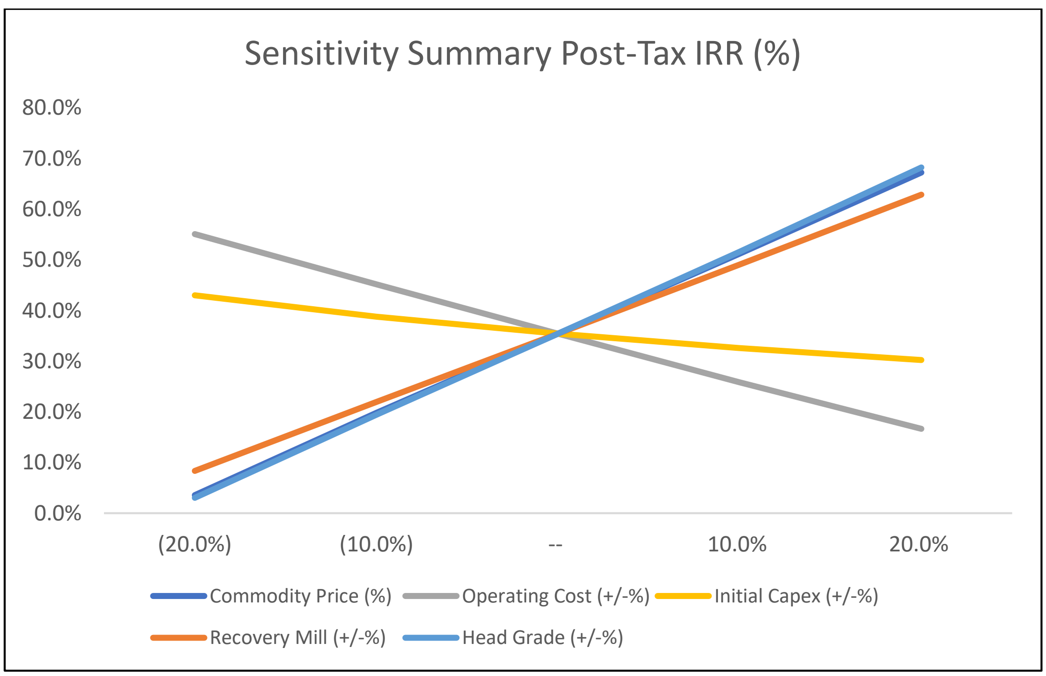


Note: Series lines for commodity price and head grade overlap in the above figures. Source: Ausenco, 2025.

Figure 1-3: Post- Tax NPV, IRR Sensitivity Results



Note: Series lines for commodity price and head grade overlap in the above figures. Source: Ausenco, 2025.



Note: Series lines for commodity price and head grade overlap in the above figures. Source: Ausenco, 2025.

1.17 Adjacent Properties

The two most important properties adjacent to Murray Brook are the Caribou mine property 10 km to the east and the Restigouche mine property 10 km to the west. Excluding the Caribou property (see Section 4), the most important property near Murray Brook is the Restigouche mine property. The geology and mineralization at Restigouche are broadly similar to the Murray Brook and Caribou deposits, both of which are part of the fertile Caribou Horizon that has been traced on surface for a total distance of 18 km in the northwest region of the Bathurst mining camp.

The Restigouche massive sulphide deposit consists of at least two separate lenses of massive sulphide underlain by a chlorite-pyrite stringer zone. Historically, the Restigouche mine produced a total of 756,000 tonnes grading 6.45% Zn, 4.87% Pb, and 107 g/t Ag in the late 1990s and in 2008. Trevali acquired the Restigouche mine in July 2017 and released an updated mineral resource statement for Restigouche in December 2021, in their press release dated March 31, 2022. The historical measured and indicated mineral resources consist of 1.08 Mt grading 5.00% Zn, 3.30% Pb, 0.22% Cu, 46.30 g/t Ag, and 0.52 g/t Au and the historical inferred mineral resources consist of 0.58 Mt grading 6.10% Zn, 4.30% Pb, 0.28% Cu, 67.83 g/t Ag, and 0.81 g/t Au.

Magna Terra Minerals Inc. announced in their press release (February 27, 2025) that they had acquired an option on the Restigouche property and began exploration work in May 2025.

The QP has been unable to verify the information and the information is not necessarily indicative of the mineralization on the property that is the subject of this technical report.

1.18 Interpretation and Conclusions

Based on the assumptions and parameters presented in this report, the PEA shows positive economics (e.g., $169 million post-tax NPV (7%) and 35.4% post-tax IRR). The PEA supports a decision to carry out additional detailed studies. There is a recommended work program totalling C$5.2 million, including recommendations pertaining to geological investigations, metallurgical testwork, geotechnical investigations, renewable energy generation investigations, mining engineering, environmental and community impact studies, and the execution of a pre-feasibility study.

The economic analysis included in this technical report have been updated from the press release issued on May 22, 2025 by Canadian Copper. The updates included:

- the addition of a 0.67% royalty, payable to MQM (total royalties increased from 0.58% to 1.25%)
- the refinement of the corporate tax rate from 30% to 29% to reflect New Brunswick corporate tax rates.

These refinements result in changes to the pre-tax and post-tax NPV and IRR. The updated NPV and IRR are as follows:

- Pre-tax NPV discounted at 7% is $256.1 million, the IRR is 47.6%, and the payback period is 1.6 years. On a post-tax basis, the NPV discounted at 7% is $169.0 million, the IRR is 35.4%, and payback period is 2.0 years. Project economics are summarized in Table 1-6.

1.19 Recommendations

The work carried out to date has justified the continued exploration and development of the project. Work is recommended to advance the project with additional drilling and a pre-feasibility study, to allow for additional metallurgical testwork and to improve the mineral resource estimate. The estimated cost of the recommended program is $5.2 million. For more details, refer to Section 26.

2 INTRODUCTION AND TERMS OF REFERENCE

2.1 Introduction

Canadian Copper Inc. (Canadian Copper) commissioned Ausenco Engineering Canada ULC (Ausenco) to compile a preliminary economic assessment (PEA) of the Murray Brook Project. The PEA was prepared in accordance with the Canadian disclosure requirements of National Instrument 43-101 (NI 43-101) and the requirements of Form 43-101 F1.

The responsibilities of the engineering consultants and firms who are providing qualified persons are listed below.

- Ausenco managed and coordinated the work related to the report. Ausenco also developed the PEA-level design and cost estimate for the process plant restart, design criteria for processing the Murray Brook mineralized material, copper circuit installation completion, Murray Brook site water management infrastructure, environmental planning, assessment, licensing and permitting. Ausenco conducted the economic analysis based on the capital and operating costs compiled for the PEA.
- P&E Mining Consultants Inc. (P&E) designed the open pit, accessibility, local resources, geological setting, deposit type, exploration work, drilling, exploration works, sample preparation and analysis, data verification, mineral resource estimate, mining methods, mine production schedules, and mine capital and operating costs.
- Pierre Lacombe reviewed the metallurgical testwork and developed the recovery models used in the economic assessment.
- Inteloc Inc. (Inteloc) reviewed the condition of the existing process plant and assessed the plant to determine the maintenance, replacements, and repairs required to restart the process plant as well as a cost estimate.
- Stantec Consulting Ltd. (Stantec) completed the sections related to tailings and water management at the Caribou site, including estimating the closure costs related to the tailings facility.

2.2 Terms of Reference

SThe report supports disclosures by Canadian Copper in a news release dated May 22, 2025, entitled, “Canadian Copper’s Combined Strategy PEA Delivers After-tax C$171M NPV7%, 36% IRR”.

All measurement units used in this report are International System units (SI) and all currencies are expressed in Canadian dollars (C$ or CAD) unless otherwise stated.

Mineral resources and mineral reserves are reported in accordance with the Canadian Institute of Mining, Metallurgy and Petroleum’s (CIM) Definition Standards for Mineral Resources and Mineral Reserves (CIM, 2014) and the CIM Estimation of Mineral Resources and Mineral Reserves Best Practice Guidelines (CIM, 2019).

The Murray Brook Project consist of two sites, Caribou and Murray Brook. The process plant, tailings management facility, and other infrastructure are located at the Caribou site. The deposit and mine are located at the Murray Brook site.

2.3 Qualified Persons

The qualified persons for the report are listed in Table 2-1. By virtue of their education, experience, and professional associations, they meet the standard of a qualified person as defined in the NI 43-10 guidelines.

Table 2-1: Report Contributors

Qualified Person	Professional Designation	Position	Employer	Independent of Issuer
Tommaso Roberto Raponi	P.Eng.	Principal Metallurgist	Ausenco Engineering Canada ULC	Yes
Glenn LeBlanc	P.Eng.	Senior Project Engineer	Ausenco Engineering Canada ULC	Yes
Jonathan Cooper	P.Eng.	Water Resources Engineer	Ausenco Sustainability ULC	Yes
James Millard	P.Geo.	Director, Strategic Projects	Ausenco Sustainability ULC	Yes
Jarita Barry	P.Geo.	Professional Geologist	P&E Mining Consultants Inc.	Yes
Andrew Bradfield	P.Eng.	Professional Mining Engineer	P&E Mining Consultants Inc.	Yes
Eugene Puritch	P.Eng., FEC, CET	President, Professional Mining Engineer	P&E Mining Consultants Inc.	Yes
William Stone	Ph.D., P.Geo.	Professional Geologist, Economic Geologist	P&E Mining Consultants Inc.	Yes
Yungang Wu	P.Geo.	Professional Geologist	P&E Mining Consultants Inc.	Yes
Pierre Lacombe	P.Eng.	N/A	Independent consultant	Yes
Marcello Locatelli	P.Eng.	VP Projects	Inteloc Inc.	Yes
Jeff Gilchrist	P.Eng.	Principal, Geotechnical Engineering	Stantec Consulting Ltd.	Yes

Source: Ausenco (2025).

2.4 Site Visits and Scope of Personal Inspection

2.4.1 Glenn LeBlanc Site Visit

Glenn LeBlanc visited the site on October 4, 2024 to review the general condition of the process plant buildings including aspects of the process plant equipment.

2.4.2 Eugene Puritch Site Visit

Eugene Puritch visited the site on March 18, 2013 to visit the Caribou mine site.

2.4.3 Yungang Wu Site Visit

Yungang Wu visited the site on September 7th and 8th 2023 to inspected the property, drill sites, drill collars, and drill core storage facilities. A data verification sampling program was completed as part of the on-site review.

2.4.4 Pierre Lacombe Site Visit

Pierre Lacombe visited the site on October 4, 2024 to review the process plant equipment and condition.

2.4.5 Marcello Locatelli Site Visit

Marcello Locatelli visited the site on February 12, 2025 and February 13, 2025 to review the condition and costs related to process plant restart.

2.4.6 Jeff Gilchrist Site Visit

Jeff Gilchrist visited the site in May 2025 to visit the Caribou mine site.

2.5 Effective Dates

The effective date of this technical report is June 20, 2025. This report also has the following significant dates:

- Mineral resource estimate: October 3, 2023
- Financial analysis: June 20, 2025.

2.6 Information Sources & References

This technical report is based on existing company reports, maps, published government reports, and public information listed in Section 27, as well as information cited in Section 3.

Several sections from historical and previous reports authored by other consultants have been directly quoted or summarized in this report as indicated in the appropriate sections. The authors held discussions with technical personnel from the company regarding pertinent aspects of the project. The authors have not conducted detailed land status evaluations, and has relied on previous qualified reports, public documents, and statements by the prior owners regarding property status and legal title.

The authors have reviewed and interpreted the historical documentation of data and observations of past activities by previous claim holders and exploration personnel who operated in the vicinity of the Murray Brook property. The majority of this information is located within internal reports and memorandums of historical claim holders for this property. The information in Section 23, Adjacent Properties, is in the form of published NI 43-101 technical reports. The list of information used to complete this technical report is located herein under Section 27.

Although selected copies of the tenure documents, operating licenses, permits, and work contracts were reviewed, an independent verification of land title and tenure was not performed. The authors have not reviewed or verified the legality of any underlying agreement(s) that exist concerning the claims, leases and licenses or other agreement(s) between third parties. Information on tenure and permits was obtained from Canadian Copper. Selected information was verified by the authors.

All tenure documents, operating licenses, permits, and work contracts were not reviewed. Information relating to tenure was reviewed on May 22, 2025 by means of the public information available on the Government of New Brunswick website at: https://nbeclaims.gnb.ca/nbeclaims. The authors of this technical report have relied on this public information and tenure information from Canadian Copper, and have not undertaken an independent detailed legal verification of title and ownership of the Murray Brook property. The authors have not verified the legality of any underlying agreement(s) that may exist concerning the licenses or other agreement(s) between third parties, but have relied on, and considers that it has a reasonable basis to rely on Canadian Copper to have conducted the proper legal due diligence.

Draft copies of this technical report have been reviewed for factual errors by Canadian Copper. Any changes made as a result of these reviews did not involve any alteration to the conclusions made. Hence, the statement and opinions expressed in this document are given in good faith and in the belief that such statements and opinions are not false and misleading at the effective date of this technical report.

2.7 Previous Technical Reports

The Murray Brook site has been the subject of one previous technical report under Canadian Copper:

P&E Mining Consultants Inc.; (October 3, 2023). Technical Report and Updated Mineral Resource Estimate of the Murray Brook Zn-Pb-Cu-Ag Project, New Brunswick, Canada.

The Murray Brook site has been the subject of previous technical repots under other companies, including the following reports:

P&E Mining Consultants Inc.; (February 20, 2017). Amended and Restated Technical Report and Updated Mineral Resource Estimate on the Murray Brook Project, New Brunswick, Canada.

P&E Mining Consultants Inc.; (July 19, 2013). Technical Report and Preliminary Economic Assessment (PEA) of the Murray Brook Project, New Brunswick, Canada.

The Caribou process plant has been the subject of a previous technical report, which includes aspects of both the Caribou process plant and the Caribou underground mine:

Roscoe Postle Associates, Inc. (December 31, 2017). Technical Report on the Caribou Mine, Bathurst, New Brunswick, Canada.

2.8 Currency, Units, Abbreviations and Definitions

All units of measurement in this report are metric and all currencies are expressed in Canadian dollars (symbol: C$ or currency: CAD) unless otherwise stated. Contained silver metal is expressed as troy ounces (oz), where 1 oz = 31.1035 g. All material tonnes are expressed as dry tonnes (t) unless stated otherwise. A list of abbreviations and acronyms is provided in Table 2-3, and units of measurement are listed in Table 2-4.

Table 2-2: Abbreviations and Acronyms

Abbreviation	Description
AA	Atomic absorption spectroscopy
AACE	Association for the Advancement of Cost Engineering International
AC CDC	Atlantic Canada Conservation Data Centre
AGG	Airborne gravity gradiometry
agl	Above ground level
AHB	Archaeological and Heritage Branch
amsl	Above mean sea level
Au	Gold
Az	Azimuth
BIF	Banded iron formation
BWi	Ball mill work index
C$	Canadian dollar
CAD	Canadian dollar
CAD/USD	Canadian-American exchange rate
CIM	Canadian Institute of Mining, Metallurgy and Petroleum
CIM Definition Standards	CIM Definition Standards for Mineral Resources and Mineral Reserves 2014
CMIF	Critical minerals infrastructure fund
CoG	Cut-off grade
CRM	Certified resource material
CuEq	Copper equivalent
CWi	Bond crusher work index
DCIP	Direct current resistivity and induced polarization
DDH	Diamond drill hole
E-GRG	Extended gravity recoverable gold
EIA	Environmental impact assessment
ELN	El Nino Ventures Inc.
EM	Electromagnetic
FA	Fire assay
FET	Federal excise tax
FS	Feasibility study
G&A	General and administration
GPR	Gross production royalty

Abbreviation	Description
GQCV	Greenstone-hosted quartz-carbonate vein deposits
GRAV	Gravimetric finish method
HADD	Harmful alteration, disruption, or destruction
HLEM	Horizontal-loop electromagnetic
IBAs	Important Bird Areas
ICP	Inductively coupled plasma
ICP-OES	Inductively coupled plasma - optical emission spectrometry
ID^2	Inverse distance squared
ID^3	Inverse distance cubed
IOCG	Iron oxide copper gold
IP	Induced polarization
IRGS	Intrusion-related gold system
ISO	International Organization for Standardization
KBAs	Key Biodiversity Areas
LCT	Locked cycle test
LIDAR	Light detection and ranging
LUP	Land use permit
MCF	Mechanized cut and fill
MDMER	Metal and Diamond Mining Effluent Regulations
MOU	Memorandum of Understanding
MQM	MetalQuest Mining
NAD 83	North American Datum of 1983
NAG	Non-acid-generating
NB DNR	New Brunswick Department of Natural Resources
NBDELG	New Brunswick Department of Environment and Local Government
NI 43-101	National Instrument 43-101 (Regulation 43-101 in Quebec)
NN	Nearest neighbour
NSR	Net smelter return
NTS	National topographic system
OK	Ordinary kriging
P&E	P&E Mining Consultants Inc
PAC	Polyaluminum chloride
PAG	Potentially acid generating
PEA	Preliminary economic assessment
PFN	Pabineau First Nation
PFS	Prefeasibility study
PGE	Platinum group elements
PVC	Polyvinyl chloride
QA/QC	Quality assurance/quality control
QP	Qualified person (as defined in National Instrument 43-101)
RF	Revenue factor

Abbreviation	Description
RQD	Rock quality designation
RWi	Rod mill work index
SAG	Semi-autogenous grinding
SAR	Species at Risk
SCC	Standards Council of Canada
SD	Standard deviation
S_d-BWI	Micro hardness or bond ball mill work index on SAG ground material
SEDEX	Sedimentary exhalative deposits
SG	Specific gravity
SI	International System
SIPX	Sodium isopropyl xanthate
SMBS	Sodium metabisulphite
SMU	Selective mining unit
SOCC	Species of Conservation Concern
TMP	Tailings management plan
TRC	Technical Review Committee
UG	Underground
UTM	Universal Transverse Mercator coordinate system
UV	Ultraviolet
VMC	Votorantim Metals Canada
VMS	Volcanogenic massive sulphide
VWAP	Volume-weighted average price
WAWA	Watercourse and Wetland Alteration
ZnEq	Zinc equivalent

Table 2-3: Units of Measurement

Abbreviation	Description
%	percent
% solids	percent solids by weight
CAD	Canadian dollar (currency)
C$	Canadian dollar (as symbol)
$/t	dollars per metric ton
°	angular degree
°C	degree Celsius
µm	micron (micrometre)
cm	centimetre
cm^3	cubic centimetre
ft	foot (12 inches)
g	gram

Abbreviation	Description
g/cm^3	gram per cubic centimetre
g/L	gram per litre
g/t	gram per metric ton (tonne)
h	hour (60 minutes)
ha	hectare
kg	kilogram
kg/t	kilogram per tonne
km	kilometre
km^2	square kilometre
kW	kilowatt
kWh/t	kilowatt-hour per tonne
L	litre
lb	pound
m, m^2, m^3	meter, square meter, cubic meter
M	million
Ma	million years (annum)
masl	meters above mean sea level
mm	millimetre
Moz	million (troy) ounces
Mt	million tonnes
MW	megawatt
oz	troy ounce
oz/t	ounce (troy) per tonne
oz/ton	ounce (troy) per short ton (2,000 lbs)
ppb	parts per billion
ppm	parts per million
t	metric tonne (1,000 kg)
ton	short ton (2,000 lbs)
t/d	tonnes per day
USD	US dollars (currency)
US$	US dollar (as symbol)

3 RELIANCE ON OTHER EXPERTS

3.1 Introduction

The authors have assumed that all the information and technical documents listed in Section 27, References, are accurate and complete in all material aspects. Although the authors have carefully reviewed the available information presented, they cannot guarantee its accuracy and completeness. The authors reserve the right, but will not be obligated to revise the technical report and conclusions, if additional information becomes known to them subsequent to the effective date of this technical report.

3.2 Environmental, Permitting, Closure, and Social and Community Impacts

The QPs have fully relied upon information supplied by Canadian Copper and experts retained by Canadian Copper for information related to environmental (including tailings and water management), permitting, closure planning and related cost estimation, and social and community impacts as follows:

- Stantec Consulting Ltd., March 6, 2025: Environmental Constraints Analysis and Regulatory Framework Review for the Proposed Murray Brook Mine Project, Draft Repot: prepared for Canadian Copper Inc, March 6, 2025, 68 pp.

This information is used in Sections 1, 20, 25, and 26.

3.3 Markets

The QPs have not independently reviewed the marketing or transportation cost information. The QPs have fully relied upon information derived from Canadian Copper and experts retained by Canadian Copper for this information through the following documents:

- Email received from Simon Quick on May 13, 2025, "Re: Marketing Terms - Murray Brook PEA": The email contained marketing and transportation costs for copper, zinc and lead concentrates, with the information received by Canadian Copper from a third party and forwarded to Ausenco.
- Email received from Simon Quick on May 14, 2025, "FW: [External] Marketing Terms - Murray Brook PEA": The email contained additional information regarding Zinc concentrate treatment costs, with the information received by Canadian Copper from a third party and forwarded to Ausenco.

The information in the emails was considered to be reasonable to rely upon based on the knowledge of Simon Quick of Canadian Copper with respect to the marketing and transportation costs.

The marketing and transportation costs received in the emails are within expected ranges based on historical payabilities and minimum deductions, and transportation costs are within expectations based on similar projects.

Risks associated with relying upon the information would include the increase or decrease in the economic feasibility of the study, related to a decrease or increase in costs, these risks are discussed in Section 22 of the report under sensitivity analyses.

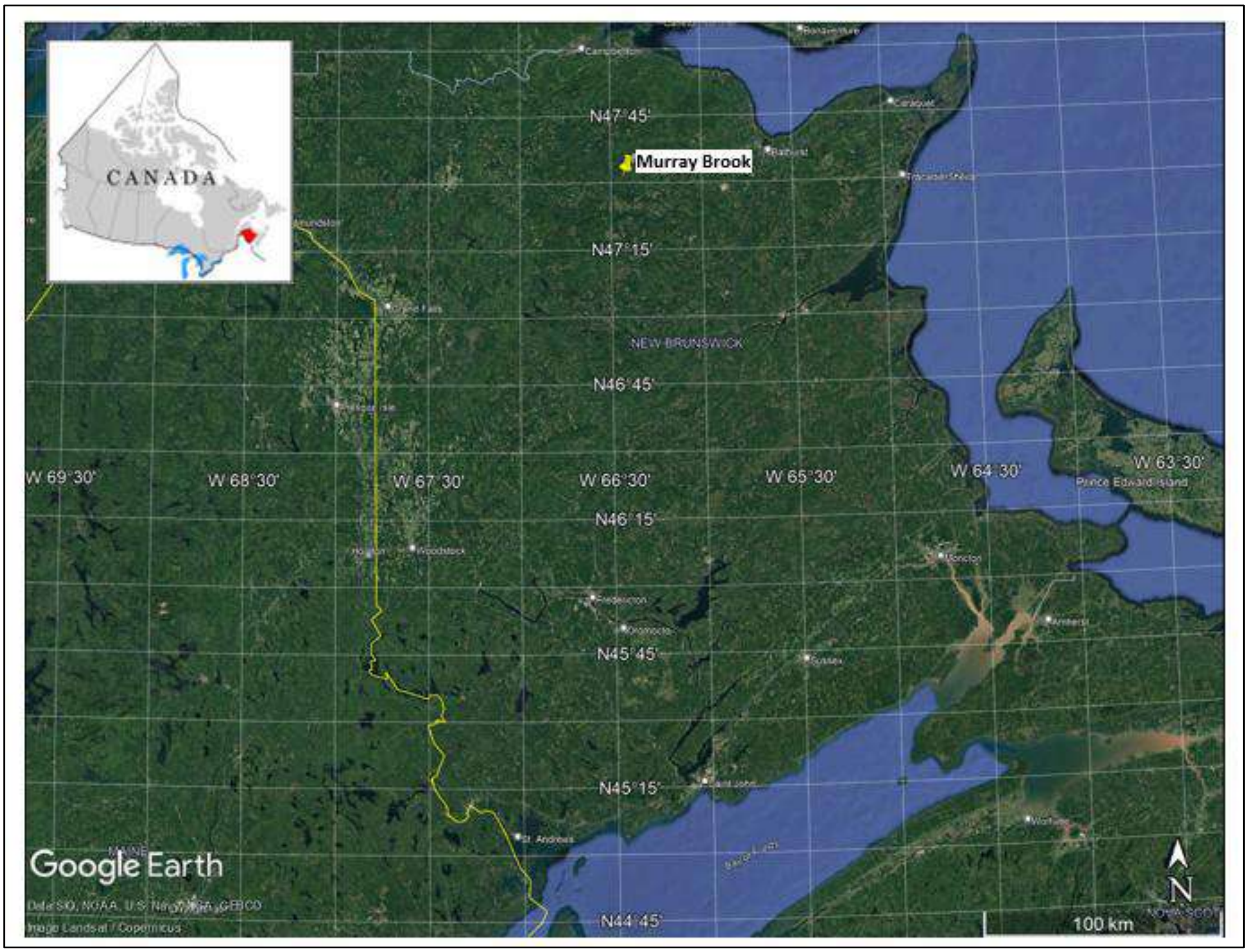
This information is used in Sections 1.13, 1.16, and 19 of the report. The information is also used in support of the financial analysis in Section 22.

4 PROPERTY DESCRIPTION AND LOCATION

4.1 Property Location

The Murray Brook property is located approximately 60 km west of the City of Bathurst in the Parish of Balmoral, Restigouche County, Province of New Brunswick, Canada (Figure 4-1). The centre of the property is located at latitude 47° 31’ 30’’ N and longitude 66° 26’ 00’’ W, and UTM NAD83 Zone 19N 5,266,700 m N and 6,932,00 m E.

Figure 4-1: General Location Map, Murray Brook Property

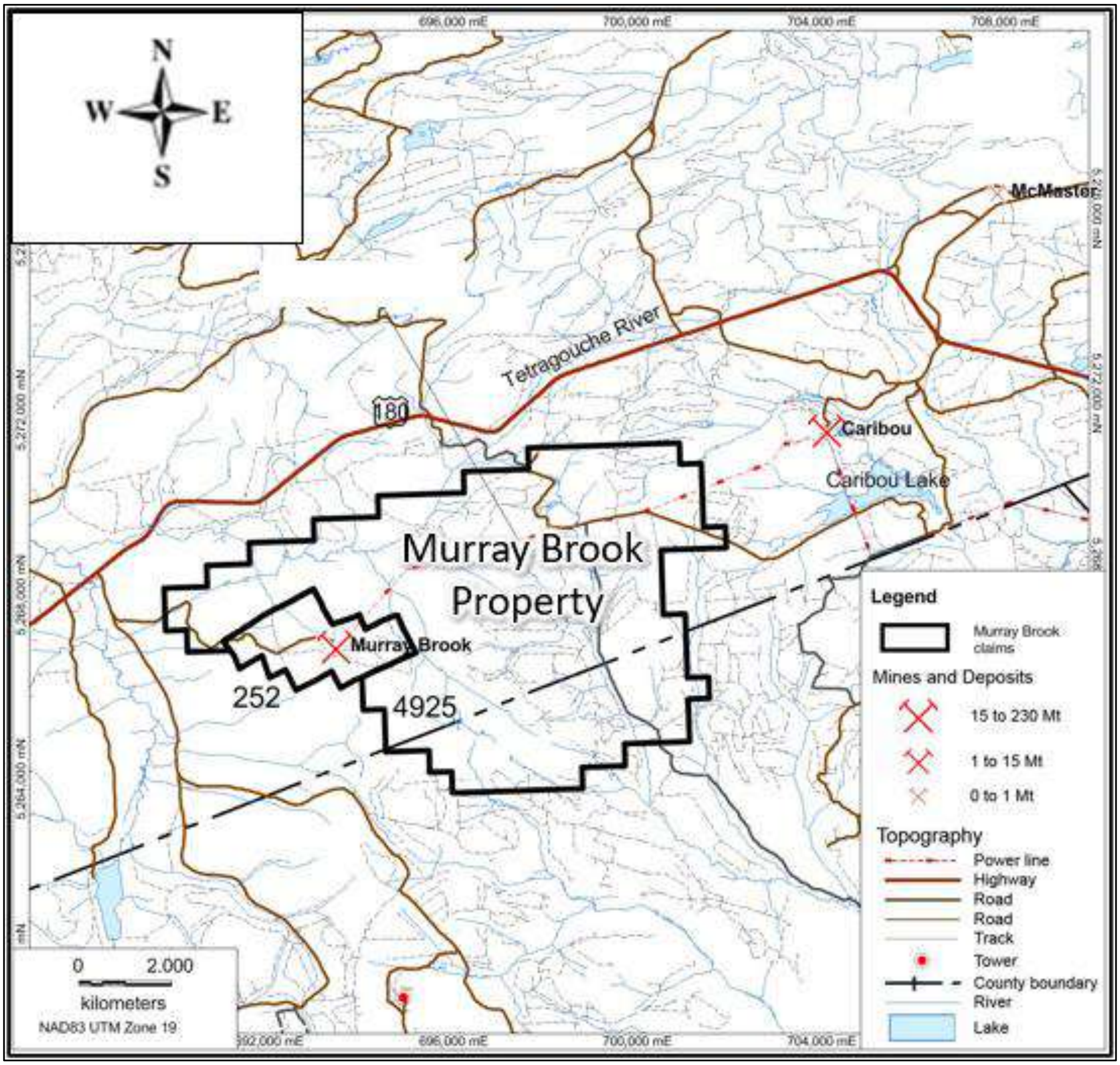


Source: Google Earth (October 3, 2023).

4.2 Property Description and Tenure

The property consists of surveyed Mineral Lease No. 252 covering an area of 505 ha and mineral claim 4925 covering an area of 5,341 ha, for a total area of 5,846 ha. The location of the lease and claim is shown in Figure 4-2.

Figure 4-2: Murray Brook Property Map



Source: P&E (2023).

Mineral Lease No. 252 was issued on October 17, 1989 to Murray Brook Resources Inc. The initial term was for 20 years with three automatic 20-year renewals. The current expiry date is October 16, 2029. The annual fee is C$3,030 and the rental fees are current. According to the information on the New Brunswick website www.nbeclaims.gnb.ca, the Mineral Lease is owned 100% by Votorantim Metals Canada Inc. (VMC) and is in good standing as of the effective date of this report (Table 4-1). As noted in Section 4.3 below, Canadian Copper acquired the Murray Brook Joint Venture property through separate agreements with VMC and MetalQuest Mining in 2023. The surface rights are held by the Crown.

Murray Brook East claim block 4925 (a.k.a. "Camelback") was staked on September 7, 2006 (Table 4-1) and consisted of 215 units. The claim block was converted to the map-staked format in January 2012 and now consists of 245 larger units covering an area of 5,082 ha (Figure 4-3). Note that the apparent infringement of the claim units on Mining Lease No. 252 in Figure 4-3 is an artifact of the map-staked unit system. The original surveyed boundaries of Lease No. 252 mark the legal boundary between the lease and the claims. According to the information on the New Brunswick website www.nbeclaims.gnb.ca, the claim units are owned 100% by VMC and in good standing as of the effective date of this report (Table 4-1).

Table 4-1: Murray Brook Property Land Tenure

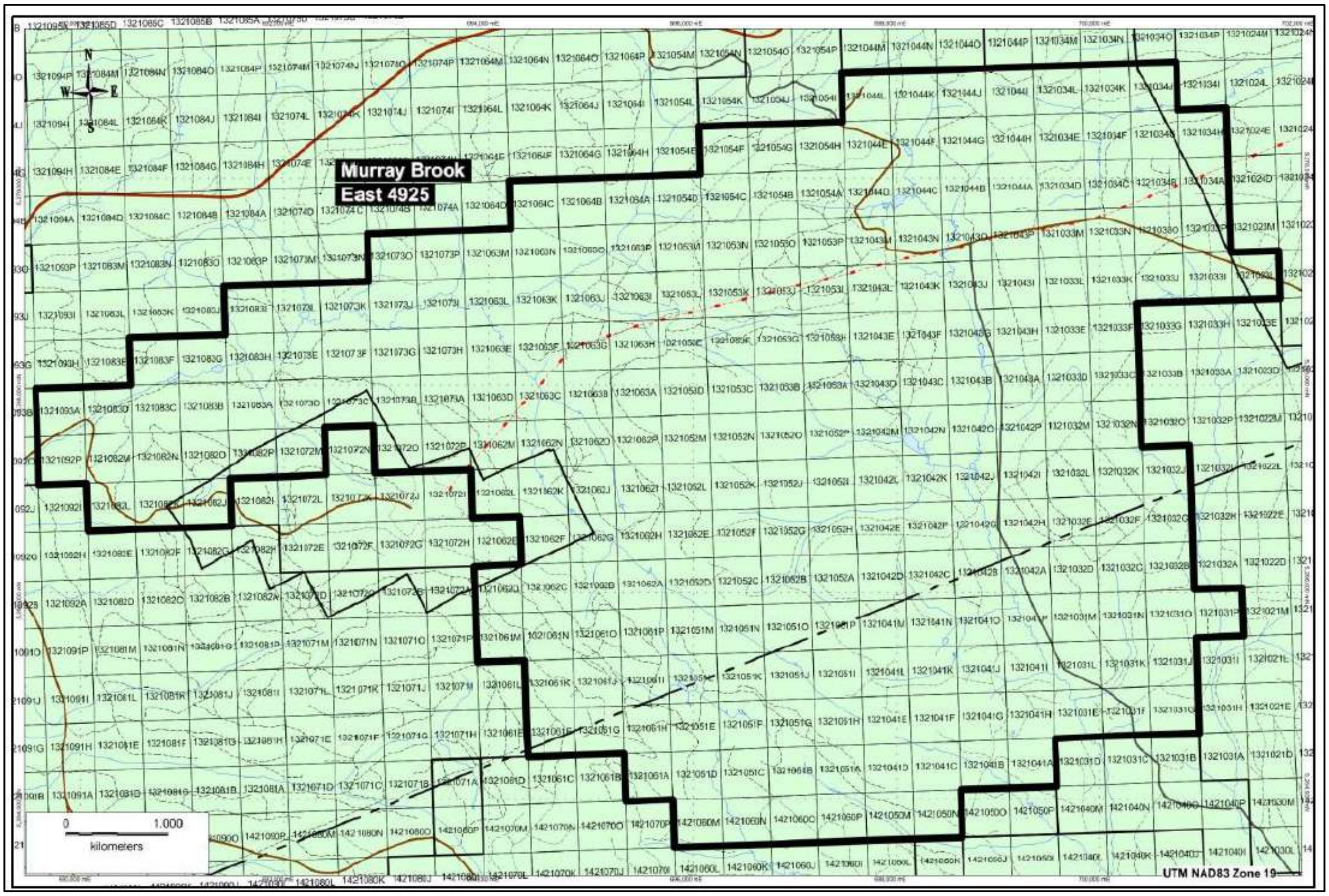
Claim Number	Title Type	Expiry Date	Issue Date	Claim Name	Area (ha)	Status	NTS Sheet	Owner (100%)
252	Mineral Lease	2029-10-16	1989-10-17	Murray Brook	505	Active	21 O/09	Canadian Copper Inc.
4925	Mineral Claim	2025-09-07	2006-09-07	Murray Brook East	5,082	Active	21 O/08 & 21 O/09	Canadian Copper Inc.
Total					**5,587**			

Note: Land tenure information effective May 22, 2025.

Claims in the 4925 block are as follows:

1321023L (1)	1321031E-G (3)	1321031J-P (7)	1321032B-G (6)
131032J-N (5)	1321033C-F (4)	1321033I-P (8)	1321034A-H (8)
1321034 (3)	1321041A-P (16)	1321042A-P (16)	1321043A-P (16)
1321044A-L (12)	1321051A-P (16)	1321052A-P (16)	1321053A-P (16)
1321054A-D (4)	1321054F-H (3)	1321061A (1)	1321061F-K (6)
132106M-P (4)	1321062A-D (4)	1321062F-P (11)	1321063A-P (16)
1321064A-C (3)	1321072M (1)	1321072O-P (2)	1321073A-L (12)
1321073O-P (2)	1321082J-P (7)	1321083A-D (4)	1321083F-I (4)
1321092P (1)	1321093A (1)	1421050M-N (2)	1421060M-P (4)

Figure 4-3: Claim Units of Claim Block 4925



Note: The claim units of claim block 4925 overlapping Mineral Lease 252 is an artifact of the map-staked unit system. The boundaries of Mineral Lease 252 are surveyed and mark the legal boundary between the lease and the adjacent contiguous claims. Source: Gagné and Hupé (2017).

The New Brunswick Energy and Resource Development had provided approval for VMC to access the Murray Brook property, maintain service roads, and carry out exploration activities, such as drilling and trenching. A recent Approval to Operate (I-10445), for operation of the reclaimed Murray Brook mine site, was issued on April 8, 2024. It is valid from April 1, 2024 to March 31, 2029. Canadian Copper maintains a C$2,000,000 rehabilitation bond on the property.

4.3 Acquisition Agreements

Canadian Copper acquired the Murray Brook Joint Venture property through separate agreements with VMC and MetalQuest Mining (MQM) in 2023. The terms and conditions of each of these agreements are summarized in the following subsections.

4.3.1 Acquisition Agreement with VMC

Canadian Copper announced on August 2, 2023 that it had successfully executed a definitive purchase agreement to acquire VMC's 72% interest in the Murray Brook Joint Venture. The Company and VMC agreed to the following conditions for the transaction under the Letter of Intent signed February 13, 2023:

1. A C$250,000 deposit paid to VMC on expiration of Right of First Refusal;
2. The execution of a definitive purchase agreement;
3. A C$750,000 instalment to be paid by the Company to VMC;
4. The issuance of 2,000,000 units of Canadian Copper. Each unit to consist of one common share priced using the 30-day volume-weighted average price (VWAP) ending on the date immediately prior to the closing date of the Purchase Agreement ("Unit Price") with a twelve-month hold period, and one full warrant exercisable for five years at an exercise price that is a 50% premium to the Unit Price;
5. A 0.25% net smelter return (NSR) royalty on the MB asset. 50% of NSR can be repurchased by the Company for C$1.0 million. The NSR has a zinc price sliding scale defined as: <US$1.50/lb = 0.25%, US$1.50-1.59/lb = 0.50%, US$1.59/lb to 1.68/lb = 0.75%, >US$1.68/lb = 1%;
6. The replacement of the Seller's bond provided to the Government of New Brunswick totalling C$2 million within three months of closing the transaction (completed November 2023); and
7. A final instalment of C$2 million to be paid by the Company to the Seller within 31 days of commercial production.

Conditions 1 to 6 have been satisfied as of the effective date of this report and Canadian Copper has completed its purchase of 72% interest in the Murray Brook Joint Venture.

4.3.2 Acquisition Agreement with MQM

Canadian Copper announced on September 12, 2023 its intention to acquire MQM's (previously El Nino Ventures Inc.) 28% interest in the Murray Brook Joint Venture. With the completion of this acquisition, Canadian Copper controls 100% of the Murray Brook property.

The Company and MQM have agreed to the following considerations under a Letter of Intent (LOI) signed September 11, 2023, which was subject to TSX Exchange approvals and the execution of a definitive purchase agreement:

1. A C$100,000 deposit paid to MQM on signing LOI and commencement of five-month exclusivity arrangement ending January 31, 2024;
2. A C$200,000 to be paid by the Company to MQM on purchase agreement execution;
3. The issuance of 2,500,000 units of Canadian Copper. Each unit to consist of one common share priced using the 30-day volume-weighted average price (VWAP) ending on the date immediately before the closing of the purchase agreement ("Unit Price") with a four-month hold period plus one day after which, 25% of the total units shall be released to MQM every three months (a "quarter"), resulting in 100% of the units being released to MQM after

four quarters from the conclusion of the initial hold period, and one full warrant exercisable for five years at an exercise price that is a 50% premium to the Unit Price;

4. A 0.33% net smelter return (NSR) royalty on the Murray Brook asset, 50% of which can be repurchased by the Company for C$1.0 million; and
5. A final instalment of C$1 million to be paid by the Company to MQM within 31 days of commercial production.

After the Company satisfied conditions 1 to 4 above, Canadian Copper completed its purchase of the remaining 28% interest in the Murray Brook Joint Venture. MQM retains an existing 0.67% NSR royalty on the Murray Brook property from an earlier transaction.

4.3.3 Potential Acquisition of Caribou Processing Plant Complex

On October 28, 2024, Canadian Copper announcing the signing of a term sheet and exclusivity agreement providing the Company the exclusive right to acquire the Caribou processing plant complex (Caribou complex), 10 km to the east of Murray Brook. The Caribou complex would be utilized to produce copper, lead, and zinc concentrates with recoverable silver from Murray Brook mineralized feed material.

The transaction summary terms and conditions are as follows:

- Purchase Price: C$6,225,000
- Term Sheet Signature Date: October 28, 2024
- Transaction Closing Date: July 11, 2025, subsequently changed to October 1, 2025
- Deposit: C$225,000, of which $125,000 is refundable against the purchase price.

The Caribou complex includes a 3,000 tonne per day (t/d) milling facility consisting of a primary grinding circuit with one semi-autogenous (SAG) mill and one ball mill, two regrinding circuits with three IsaMills and one ball mill. It also contains a differential sulphide flotation plant and associated reagents and addition systems, metallurgical and geochemical laboratories, a tailings management facility, an underground mine with historical mineral reserves, and other infrastructure, such as connected grid power and water supply for operations. When the acquisition is closed, the associated Caribou complex mining lease and mineral claims will be transferred to Canadian Copper.

The New Brunswick government has managed the care and maintenance of the Caribou complex since January 2023. The Caribou complex operated until August 2022, at which point a concentrator care and maintenance shutdown was instigated by the previous operator.

4.4 Tenure Maintenance

In New Brunswick, the holder of the mineral claim has the right of free access by any reasonable means to and from the claim area, and the exclusive right to prospect for minerals and carry-on mining in or on the claim area and to remove minerals from the claim area for purposes of sampling and testing (*Mining Act*, SNB 1985, c M-14.1).

Retention of claims in good standing from year to year requires payment of a renewal fee for each claim plus submission of documentation to the government describing work programs and associated costs applicable to the property during the reporting year. The work commitments and renewal fees are summarized in Table 4-2.

Table 4-2: Mineral Assessment Work Requirements in New Brunswick

Year of Issue	Required Work per Claim (C$)[1]	Renewal Period	Renewal Fees per Claim (C$)
Year 1	100	1 to 5	10.00
Year 2	150	6 to 10	20.00
Year 3	200	11 to 15	30.00
Year 4	250	16 and more	50.00
Years 5 to 10	300		
Years 11 to 15	500		
Years 16 to 25	600		
Years 26 and Over	800		

Note: Per mineral claim unit per year.

Reports of work (mineral assessment reports) are received and processed by the New Brunswick Department of Natural Resources (NB DNR) and Energy Development, Mineral and Petroleum Branch (NBDNRED). The reports are kept for a confidential period of two years from the date of submission. The reports are made public when the confidential period is finished or when all claims in a report have lapsed or were surrendered. The work can be performed on any one or more claims. Mineral claims must be contiguous, are held in the name of one person or company and have the same recording date.

4.5 Permitting

The Company will be required to obtain the following permits and licences to conduct mineral exploration in New Brunswick:

- A prospecting licence is required to prospect or register mineral claims. Application is made through NB e-CLAIMS and is valid for a lifetime.
- Notification requirements prior to performing exploration work and general prospecting must notify private landowners; Recorder, DNR; District Forest Ranger, DNR; Work Safe NB; and Offices of the Recorder (Bathurst in this case).
- Prior to commencing work that would cause actual damage to or interference with the use and enjoyment of Crown lands; the following procedures must be followed:
  - Submit to the Recorder the completed Notice of Planned Work on Crown Land-Form 18.1, listing the proposed work and enclosing a map showing the area of work and the claims.

- The Recorder will review the submitted form and give permission on behalf of the DNR for the work to proceed.
- In some cases, the Recorder will advise the person planning the work that a reclamation plan and security are required before the work commences.
- Obtain the consent of the lessee if work is done on a Crown land lease.

- A lease or a right to occupy as issued under the *Crown Lands and Forests Ac*t is required to erect a permanent camp, building or other structure on Crown Land.
- Review the *Mining Act* for standard conditions for mineral exploration.
- Claim holders wishing to conduct advanced exploration on mineral claims may require additional approvals beyond a Form 18 under the *Mining Act* depending on the scope of work involved.

Anyone with a mineral claim in New Brunswick who has decided to produce minerals from the mineral claim can apply for a mining lease. A mining lease allows mineral production and requires an application fee, rent per hectare per of $6.00 and a minimum dollar value of work required per hectare per year of $60.00. Guides to the Mine Approval Process, and Development of a Mining and Reclamation Plan are provided by the DNRED at the following website:

https://welcomenb.ca/content/gnb/en/departments/erd/energy/content/minerals/content/Minerals_exploration.html

4.6 Environmental Liability

The Murray Brook mineral lease is the site of past production from open-pit mining of the oxidized surficial portion of the Murray Brook deposit. A site assessment by Stantec (2012) for VMC reports that the liability associated with the Murray Brook property consists of short-term monitoring liability and long-term restoration liability. The current liability is for monitoring rather than restoration of fish habitat in nearby Gossan Creek and Copper Creek. First VMC and now Canadian Copper maintain a C$2 million environmental cash security with the New Brunswick government as security for this liability.

4.7 QP Comments

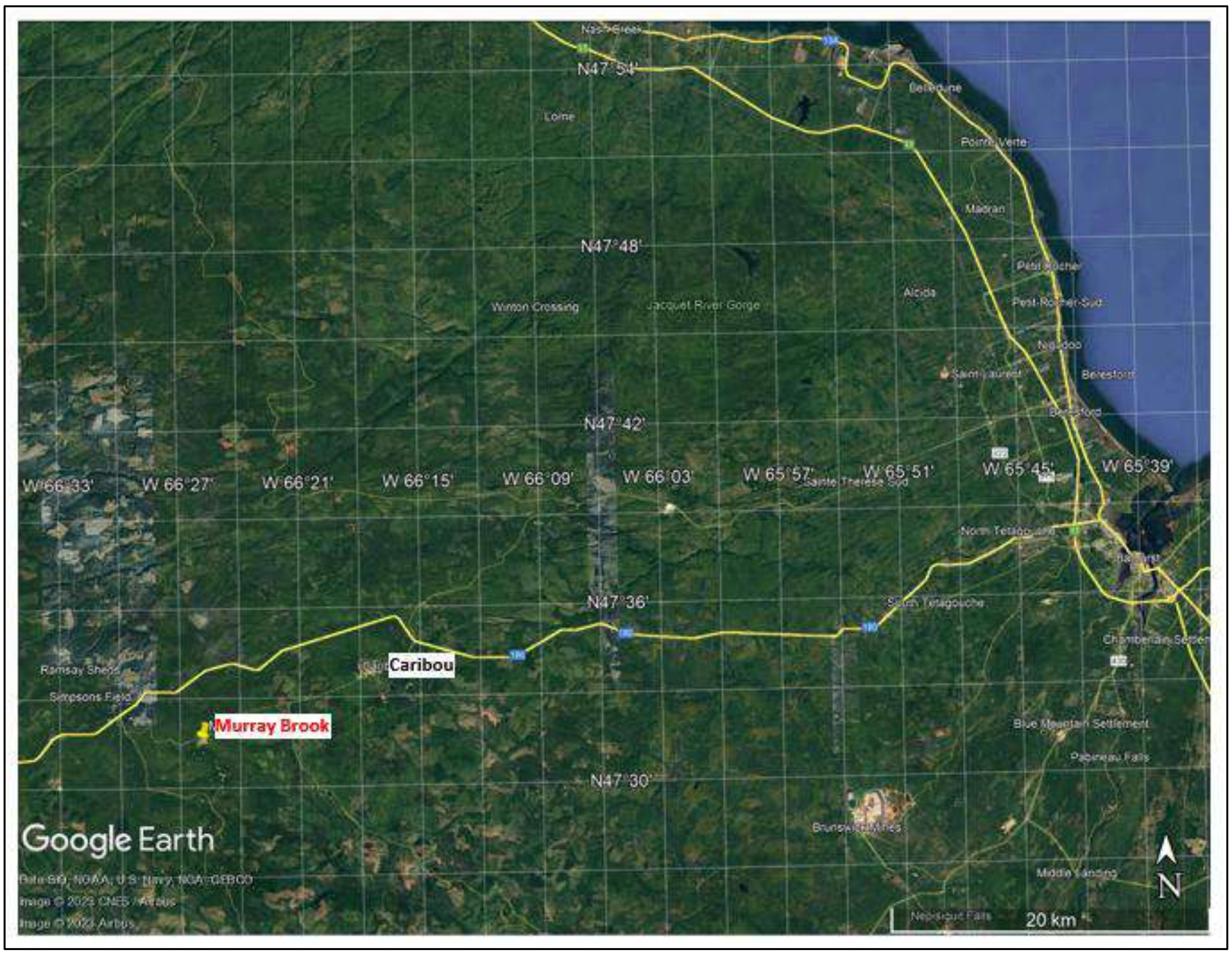
To the extent known, and apart from the aforementioned land encumbrances, the author is not aware of any other significant factors or risks that may affect access, title or right or ability to perform work on the Murray Brook property.

5 ACCESSIBILITY, CLIMATE, LOCAL RESOURCES, INFRASTRUCTURE & PHYSIOGRAPHY

5.1 Access

The Murray Brook property is located approximately 60 km west of the City of Bathurst, New Brunswick. At KM 60 from Bathurst, a 5 km gravel road extends southward from Highway 180 to the Murray Brook mine site. The City of Bathurst provides access to rail and ocean shipping facilities into the Gulf of St. Lawrence, the Atlantic Ocean, and globally (Figure 5-1).

Figure 5-1: Murray Brook Property Access



Source: Google Earth (October 3, 2023).

5.2 Local Resources and Infrastructure

Several communities in the region, particularly the City of Bathurst, offer commercial goods, social, educational, and financial amenities, and a pool of skilled labour. Bathurst is the closest important city, with a population of 12,157 (Census Canada, 2021) has been an important centre for mining, forestry, fishing, and tourism and has the closest accommodations to the Murray Brook property. The Town of Saint-Quentin, with a population of 2,141 (Census Canada, 2021) and located 65 km west of Murray Brook along Highway 180, is another potential source of goods, amenities, skilled labour, and accommodation.

Although the site has been reclaimed and the equipment sold off, part of the historical open pit mine (Figure 5-2), leach pad and tailings management area are still evident. A 10 kV powerline links Murray Brook to the power station at the Caribou mine, 10 km to the east (Figure 5-1). However, service was discontinued in 1996.

Figure 5-2: View of the Historical Open Pit Mine

Source: El Nino Ventures Inc. (autumn, 2011). View looking southwesterly.

5.3 Climate

Located south of Chaleur Bay, the Bathurst mining camp has a typically continental climate. January is the coldest month and July is the warmest. The area experiences very cold winters with a mean temperature of -13˚C in January. Frigid temperatures are not infrequent in the Murray Brook area with extreme low temperatures of -30˚C to -35˚C reported every winter. The Murray Brook area generally receives between 300 and 400 cm of snow annually for approximately 33% of its annual total of precipitation. In summer, average daytime highs vary between 20°C and 28˚C. The highest temperature ever recorded in the area is 39˚C. Spring and early summer are generally dry, but there is ample water during the growing season. The area records approximately 1,200 mm of rainfall a year, with the heaviest amounts falling during the summer months.

Winds generally blow predominantly from the west and northwest in the cold months and from the south and southwest in the warm months. Wind speeds average 15 to 20 km/h in winter and 12 to 15 km/h in summer.

The Murray Brook property is accessible for exploration year-round.

5.4 Physiography

The property is located in the Miramichi Highlands, characterized by rounded, glacially scoured hills. Topographic maps show a broad plateau in the east at approximately 630 masl with deeply incised watercourses reaching down to approximately 470 masl in the western portion of the area. Drainage in the area is eastward towards the Atlantic Ocean. Land use in the region is mainly for tourism, forestry, and mining.

Vegetation on the property is characterized by poplar and birch trees that dominate the higher elevations, mixed with pines and firs. Much of the Murray Brook Project area was historically logged, and re-vegetated with secondary growth.

6 HISTORY

The history of the Murray Brook property is summarized below from mainly Derosiers (2008) and P&E (2017).

6.1 Early History

Prior to 1955, very little prospecting was carried out in the Murray Brook area, mainly due to the lack of access. The gossan had been discovered at the end of the 19th Century and an old shaft had been sunk in it to a depth of 7.6 m. After the construction of the New Brunswick Lumber Co. gravel road, which follows the east side of the Upsalquitch River, some prospectors and mining companies started to explore the area.

6.2 Kennecott (1955 to 1980)

The Murray Brook property was staked originally by Kennecott Copper Exploration Limited (Kennco) in 1955 to cover seven airborne electromagnetic anomalies. Ground follow-up of the anomalies, however, proved that the electromagnetic responses were caused by graphitic sedimentary rocks rather than sulphide mineralization.

In 1956, an intermediate lava float assaying 1.35% Cu was discovered in the western part of the property (Perusse, 1957), which stimulated further exploration. Ground geophysical surveys missed the Murray Brook deposit, because there was no airborne survey immediately over it. Field determinations of heavy metal contents of active steam and bank sediments pinpointed an anomaly source at the head of Gossan Creek. Subsequent trenching outlined an area of gossan measuring 760 m by 120 m. Packsack drilling failed to intersect fresh sulphides below the gossan. A horizontal-loop electromagnetic (HLEM) survey was carried out to determine if any part of the gossan was underlain by massive sulphides. Results indicated that massive sulphide lenses were present.

In 1956, a drill hole intersected 89 m of massive sulphides under 16 m of gossan. By 1958, Kennco had sufficient drilling to complete an initial mineral resource estimate (Perusse, 1958).

In 1970, the property was optioned to Cominco who drilled three holes that did not increase the tonnage.

In 1973, the property was optioned to Gowganda Silver Mines Limited. In 1974, Canex Placer Explorations Ltd. gained control of the deposit through exploration expenditures. An extensive drilling program was carried out to obtain material for metallurgical testing.

The property reverted to Kennco Explorations in 1979.

6.3 Northumberland and NovaGold (1980 to 1996)

In 1985 Northumberland Mines Ltd. optioned the property to examine the precious metals content of the gossan. Thirty-six drill holes (NM-1 to NM-36) were completed to test the economic potential of the gossan zone. A feasibility study of an open pit mine using an indoor vat leaching process using a Merrill-Crowe system for gold and silver production was submitted and approved by the Department of Natural Resources and Energy in 1986.

In 1988, NovaGold Resources Ltd. (NovaGold) acquired Northumberland Mines and the Murray Brook deposit. NovaGold completed a mineral resource estimate and vat leaching of the gossan zone commenced commercial production in September 1989. The process facility operated year-round and yielded recoveries in the 85% range. The mining and vat leaching activities were discontinued in 1992, leaving the sulphide deposit unmined. The pit and property were reclaimed in 1996. Production from 1989 to 1992 totalled 2.7 Mt for 1,384 kg Au and 9,829 kg Ag (Desrocher, 2008).

In 1998, a drilling campaign and a mineral resources calculation were completed by NovaGold.

6.4 Murray Brook Resources (1996 to 2010)

In July 1995, Murray Brook Resources Inc. (MBR) signed an agreement with the Sheridan Platinum Group Ltd. (SPG) for the development of the copper mineral resource of the deposit. SPG was able to acquire 60% of the massive sulphide deposit. SPG and MBR intended to restart the leaching operation utilizing the 50,000 tonnes which were placed on the leach pads and the technical operation of the copper leach recovery plant. MBR also expected to start a drilling program to determine the extent and potential of developing underground mineral reserves of high-grade copper. This drilling campaign was finally undertaken in 1998, but the agreement was terminated.

In 2007, MBR resampled 645.65 m of NovaGold drill core for copper, lead, zinc, gold, and silver. The assays indicated comparable copper and lead values, slightly elevated zinc values, and a 10% decrease in silver values compared to those previously reported. In January 2008, GEOSTAT Systems International Inc. completed a study of open pit exploitation of the copper mineralization for MBR (Desrosiers, C., 2008). It was noted that copper grade decreases with depth, and lead, zinc and gold values increase with depth.

In 2008, MBR carried out a 42 line-km magnetic survey and a 20.8 line-km induced polarization/resistivity survey (IP/RES). The magnetic survey delineated the volcanic rocks of the Boucher Brook Formation. The IP/RES survey delineated a 900 m long conductive anomaly with a positive chargeability and a low resistivity response. The response is comparable with the response of massive sulphides below the open pit, and appears to be a southwest extension of the known massive sulphide zone. The IP/RES response is validated by the presence of a gravimetric anomaly and a soil geochemical copper anomaly.

6.5 VMC-El Nino Option-Joint Venture Agreement (2010 to 2016)

Under an Option and Joint Venture Agreement with Murray Brook Minerals Inc. and Murray Brook Resources Inc., both privately held companies, VMC earned 50% interest in the property by funding C$2,250,000 in exploration expenditures and making payments totalling C$300,000 over a three-year period commencing November 1, 2010. VMC earned an additional 20% interest in the property by funding an additional C$2,250,000 over an additional two-year period. On January 3, 2011, VMC and El Nino Ventures Inc. (ELN) entered into a participation agreement, wherein the latter could earn 50% of VMC's interest by paying 50% of the costs incurred by VMC in the Option and Joint Venture Agreement.

VMC completed drill programs in 2010, 2011 and 2012. In 2010, VMC completed four due diligence drill holes for 595.2 m to confirm results of the historical drilling. These holes were consistent with historical results with significant intersections of zinc, copper, lead, gold, and silver reported. VMC duly finalized its agreement with the property owners.

VMC's 2011 drill program consisted of 63 vertical drill holes totalling 10,499.45 m. The results were announced in news releases dated August 30, 2011, November 28, 2011, January 16, 2012, and January 23, 2012. The objectives of the drilling were two-fold: (1) infill drilling to close gaps of up to 100 m in the drill coverage; and (2) step-out drilling to define the size of the Murray Brook deposit. A new mineral resource estimate was released in 2012 (Harron, 2012), now an historical mineral resource.

The objective of the 2012 drilling was to upgrade the inferred and indicated mineral resources to measured resources, define additional near-surface mineral resources along the northwest margin of the deposit, and provide material for completion of preliminary metallurgical testing on selected portions of the deposit. The drill program commenced in February 2012 and consisted of 99 vertical drill holes totalling approximately 18,264 m. Therefore, between 2010 to 2012, 166 drill holes were completed for 29,718 m.

Analysis of the drilling results identified two distinct north-trending massive sulphide zones with different mineralogical characteristics and thicknesses. The western zone appears to be thicker and richer in zinc, lead, and silver, whereas the eastern zone is thinner and richer in copper-gold mineralization.

In July 2013, VMC and El Nino released a positive NI 43-101 preliminary economic assessment and mineral resource estimate based on open pit mining and an on-site process facility (P&E, 2013).

6.6 Puma Exploration (2017 to 2020)

On October 13, 2016, Puma Exploration Inc. executed an asset purchase agreement with VMC and on October 25, 2016, Puma executed an asset purchase agreement with ELN to acquire, respectively, approximately 67.9% and 32.1% beneficial interest in the Murray Brook deposit. Puma signed the acquisition agreement in order to evaluate the Murray Brook property for potential underground development.

In February 2017, Puma released an amended and restated technical report and updated mineral resource estimate based on underground mining at Murray Brook. Puma completed drilling programs in late 2017 and early 2018 on the Murray Brook property (see Section 10).

In April 2018, Puma entered into a strategic exploration and development alliance with Trevali for the development of the Murray Brook deposit and exploration of the surrounding properties. The development was to be completed on the basis of 75% Trevali and 25% Puma ownership of Murray Brook. Trevali was the owner and operator of the Caribou mine and process plant, 10 km east of the Muray Brook deposit. The alliance completed drilling, trenching, and metallurgical testwork programs in 2018 and 2019 (see Sections 9, 10 and 13). Trevali, however, terminated the option agreement with Puma for the Murray Brook property in March 2019.

On February 26, 2020, Puma terminated the asset purchase agreement with VMC. Puma's decision was based on unfavourable equity market conditions that prevented completion of the financing required to complete the transaction and attracting a partner to the project. On August 4, 2020, Puma terminated the asset purchase agreement with ELN.

6.7 VMC-El Nino Option Joint Venture (2020 to 2023)

The Murray Brook property reverted back to VMC and ELN in the summer of 2020. In June and August of 2023, Canadian Copper signed agreements with VMC and ELN to acquire their interests in the property.

7 GEOLOGICAL SETTING AND MINERALIZATION

7.1 Regional Geology of the Bathurst Mining Camp

The Murray Brook deposit is located in the Bathurst mining camp in northern New Brunswick. The Bathurst mining camp is an Ordovician back-arc complex of polydeformed sedimentary, felsic volcanic, and mafic volcanic rocks formed in separate sub-basins within the back-arc basin. These rocks have been juxtaposed by five periods of folding and thrusting, and are collectively referred to as the "Bathurst Supergroup." The sedimentary and volcanic rocks are intruded by gabbro, diabase, and quartz porphyritic rocks of Ordovician age.

The Bathurst mining camp is underlain by Cambro-Ordovician age rocks of the Bathurst Supergroup. The Bathurst Supergroup consist of mafic volcanic and sedimentary rocks of the Fournier, California Lake, Tetagouche and Sheephouse Brook Groups, in order of highest to lowest structural level (Figure 7-1). The various groups were juxtaposed by thrusting and internally imbricated into thrust nappes during successive incorporation into the Brunswick Subduction Complex.

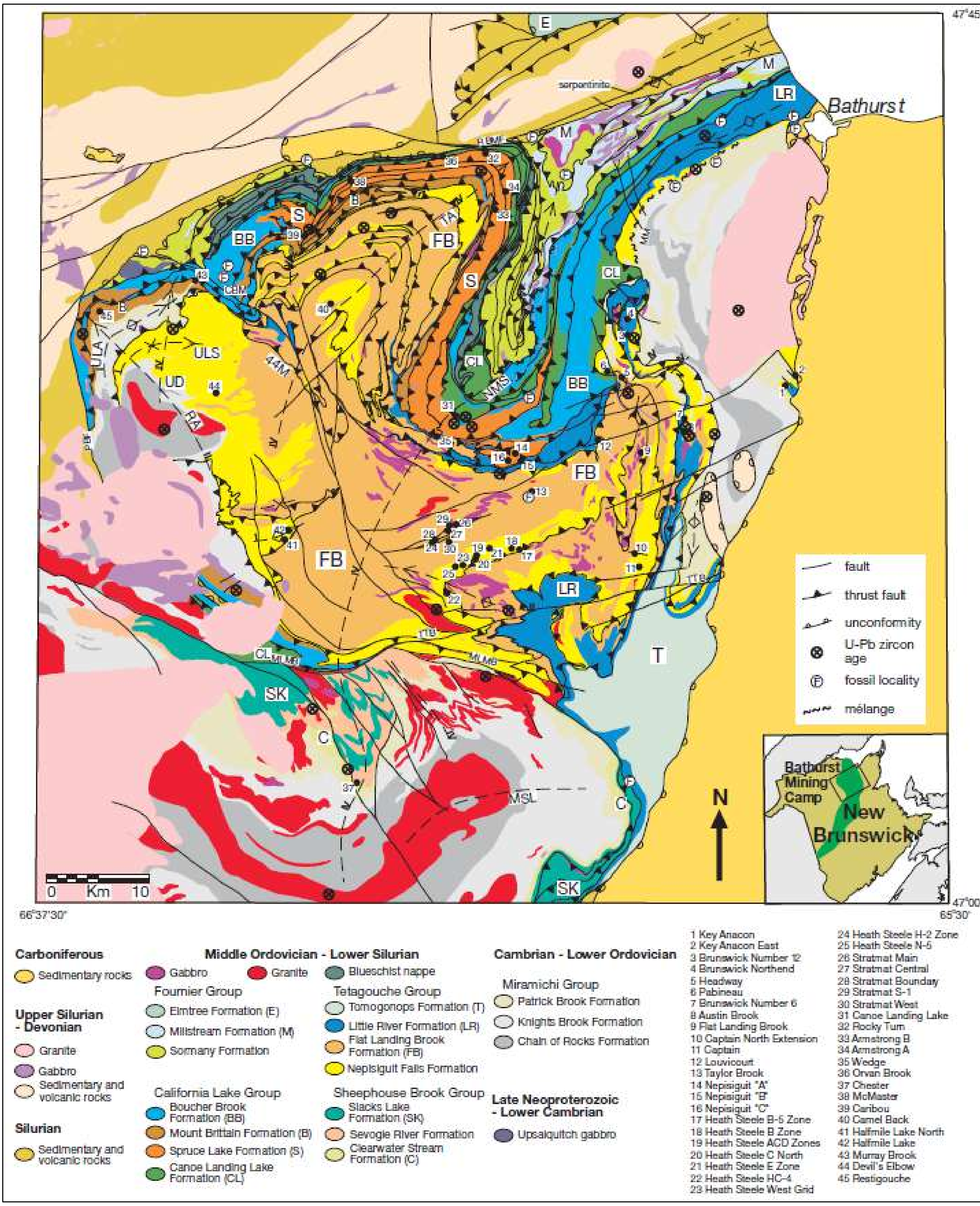
The Bathurst Supergroup encompasses all Ordovician volcanic and sedimentary rocks overlying the Miramichi Group in the Bathurst mining camp, a roughly circular area 70 km in diameter in the northern Miramichi Highlands, and Ordovician rocks of the Elmtree Inlier, an elliptical area measuring approximately 25 x 15 km on the shore of Chaleur Bay.

Rocks of the Bathurst Supergroup lie conformably to disconformably on the Miramichi Group, and are unconformably overlain by, or in fault contact with, Silurian rocks of the Chaleurs Group to the north and west, and the Silurian Kingsclear Group and Carboniferous Mabou and Pictou Groups to the east.

Cambro-Ordovician aged rocks in the Bathurst camp have undergone five episodes of regional deformation. Two structural domains are recognized: (1) a flat-lying belt in the south and west parts of the camp characterized by recumbent or overturned F2 folds; and (2) a steeply dipping belt in the north and east, in which F2 folds are upright. Thrusting related to closure of the Iapetus back-arc basin and regional faulting has also affected the present distribution of major stratigraphic units (van Staal, 1987).

The Bathurst mining camp hosts 46 known volcanogenic massive sulphide deposits with a total sulphide resource of over 500 Mt (McCutcheon and Walker, 2009). The camp also hosts the world-renowned Brunswick No. 12 mine, which from 1964 to 2013 (49 years) produced 136.6 Mt grading 3.44% Pb, 8.74% Zn, 0.37% Cu and 102.2 g/t Ag. The Brunswick No. 12 mine has since been de-commissioned. The QP has been unable to verify the information and the information is not necessarily indicative of the mineralization on the property that is the subject of the technical report.

Figure 7-1: Geological Map of the Bathurst Mining Camp Geology Map, Murray Brook Property



Note: Generalized geology of the Bathurst mining camp with the distribution of the major structures and massive sulphide deposits (modified after Rogers and van Staal, 1996; Rogers et al., 1997; van Staal and Rogers, 2000). Exposure is very poor in the southeastern part of the Bathurst mining camp and structural relationships here are not well understood. The generation of major folds is indicated by Roman numerals on their axial surfaces. Abbreviations: CBM = Camel Back Mountain. Structures: 44M = Forty Four Mile Brook thrust, MM = Miramichi mélange, MLMB = Moose Lake-Mountain Brook fault, MSL = Mullin Stream Lake structure, NMS = Nine Mile synform, PBF = Portage Brook fault, RA = Restigouche antiform, RBMF = Rocky Brook Millstream fault system, TA = Tetagouche antiform, TTB = Tomogonops-Tozer Brook fault, UD = Upsalquitch dome, ULA = Upsalquitch Lake antiform, ULS = Upsalquitch Lake synform. Source: van Staal et al. (2003).

The 46 massive sulphide deposits of the Bathurst camp occupy more than one stratigraphic position; 32 are in the Tetagouche Group and 13 occur in the probably coeval California Lake Group. Within the Tetagouche Group in the southern part of the Bathurst mining camp, massive sulphide deposits occur largely in the first volcanic cycle, represented by crystal tuffs of the Nepisiguit Falls Formation (e.g., Brunswick 12 and 6, Heath Steele and Half Mile Lake deposits), which may be related to a volcanic centre in the southeastern part of the Bathurst mining camp (Helmstaedt, 1973; Franklin et al., 1981. Most of the deposits are hosted by chloritic mudstones at or near the top of this formation ("Brunswick Horizon"), in association with oxide facies iron formation. Within the California Lake Group, in the northern part of the Bathurst mining camp, massive sulphide deposits occur in local sedimentary units within the volcanic sequence (Murray Brook and Caribou deposits) and in the volcanic sequence (Restigouche deposit). These massive sulphide deposits lack associated iron formation and may be related to a second volcanic centre in this area of the Bathurst mining camp, probably near Restigouche (Helmstaedt, 1973).

7.2 Local Geology of the Murray Brook Deposit Area

The geology of the Murray Brook deposit area was mapped and incorporated into a regional geological map of the Bathurst mining camp (Staal et al., 2003) (Figure 7-2). The Murray Brook property area overlies a structurally compressed area of juxtaposed formations and members of the Miramichi, Tetagouche, California Lake, and Fournier Groups.

The Murray Brook deposit is hosted by sedimentary rocks of the Charlotte Brook Member in the lower part of the Mount Brittain Formation. The upper felsic volcanic member of the Mount Brittain Formation hosts the Restigouche deposit, 10 km to the west of Murray Brook. The Mount Brittain Formation is considered to be equivalent of the Spruce Lake Formation, which hosts the Caribou mine, 10 km to the east.

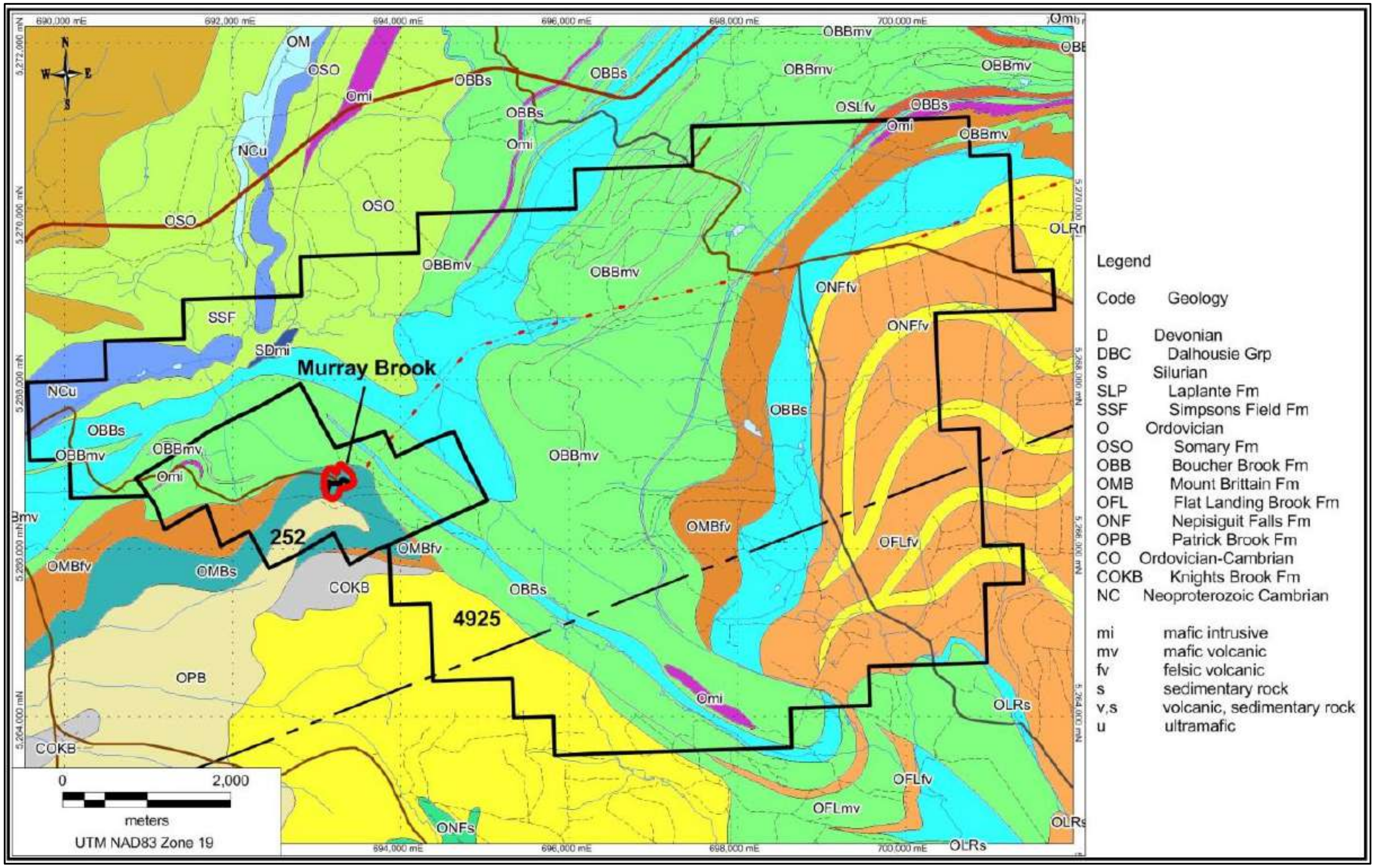
7.3 Deposit Geology, Alteration and Mineralization

The Murray Brook deposit is elliptical in plan with a strike length of approximately 350 m and a maximum thickness of up to 100 m. The deposit dips approximately 40° to the northwest with a dip extent of >350 m. It plunges moderately to the north and appears to pinch out at depth and to the east. The geometry of the deposit was probably lens-shaped, but the up-dip portion of the body has been eroded and pre-Pleistocene weathering produced the gossan that was historically mined for silver and gold.

Structurally, the massive sulphides occupy the core of an F1/F2 synform (sheath fold) that is deformed by F3 folds, such that the hangingwall and footwall are part of the same unit. Although the Murray Brook deposit is a single body of massive sulphide with good continuity, in-fill drilling indicates that it consists of two connected lenses or lobes. The western, deeper lobe is richer in zinc and lead and the eastern, shallower lobe is richer in copper (Figure 7-3). Given that zoned volcanogenic massive sulphide (VMS) deposits tend consist of copper-rich lower zones and zinc-lead-rich upper zones (see Section 8 of this report), it appears possible that the Murray Brook deposit may be overturned.

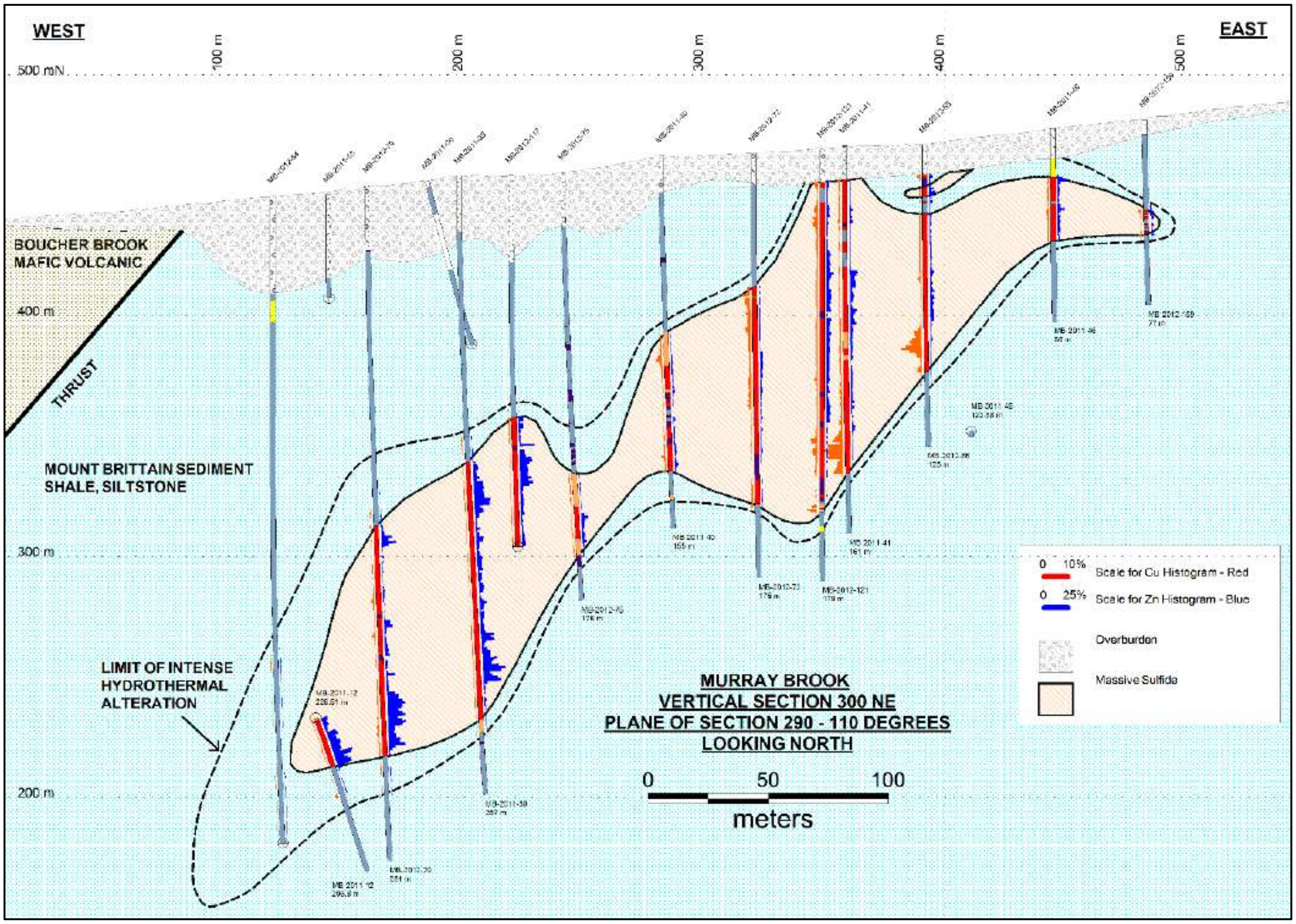
The Murray Brook deposit is enclosed in a 1 to 3 m wide halo of chloritized sedimentary rocks containing disseminated pyrite. The hangingwall is moderately chloritic and locally intensely deformed. The footwall consists of fine grained-felsic tuff and tuffaceous sedimentary rocks with moderate to strong chlorite and sericite alteration.

Figure 7-2: Geology of the Murray Brook Property Area

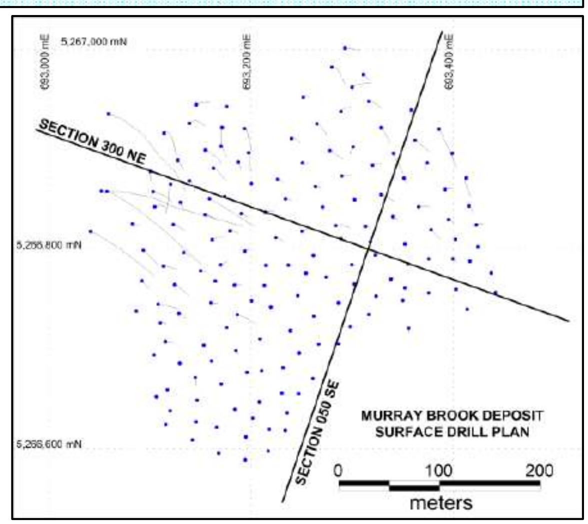


Source: Van Staal et al. (2003).

Figure 7-3: Vertical Cross-section of the Murray Brook Deposit



Source: P&E (2013)

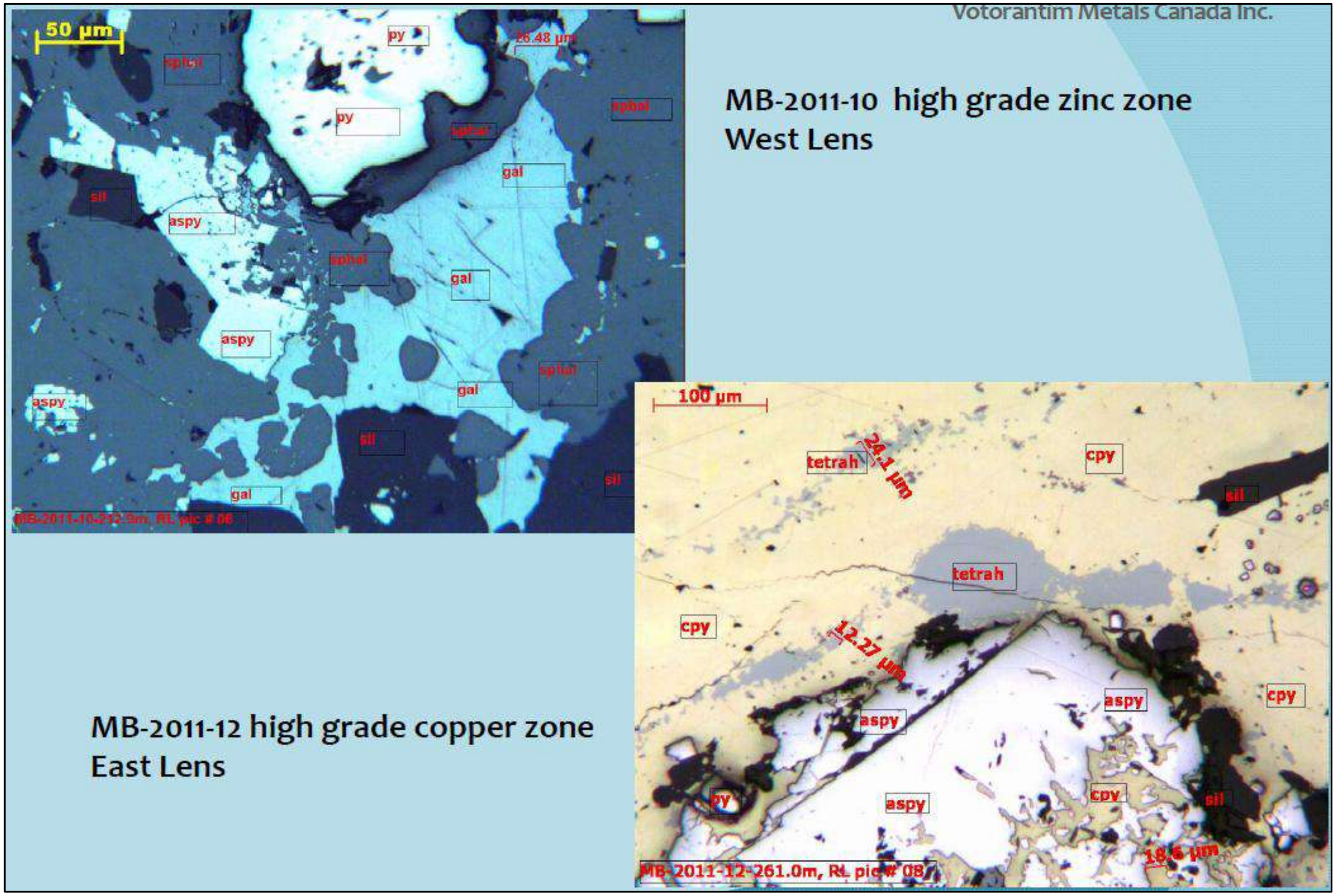


The Murray Brook deposit sulphides are massive to semi-massive, locally banded, and pyrite-rich. The sulphides are mainly fine-grained, massive, weakly laminated pyrite with disseminated and banded sphalerite, chalcopyrite, and galena, and minor tetrahedrite, covellite, marcasite and arsenopyrite (Figures 7-4 to Figure 7-7).

Figure 7-4: Massive Sulphides from Murray Brook Drill Core

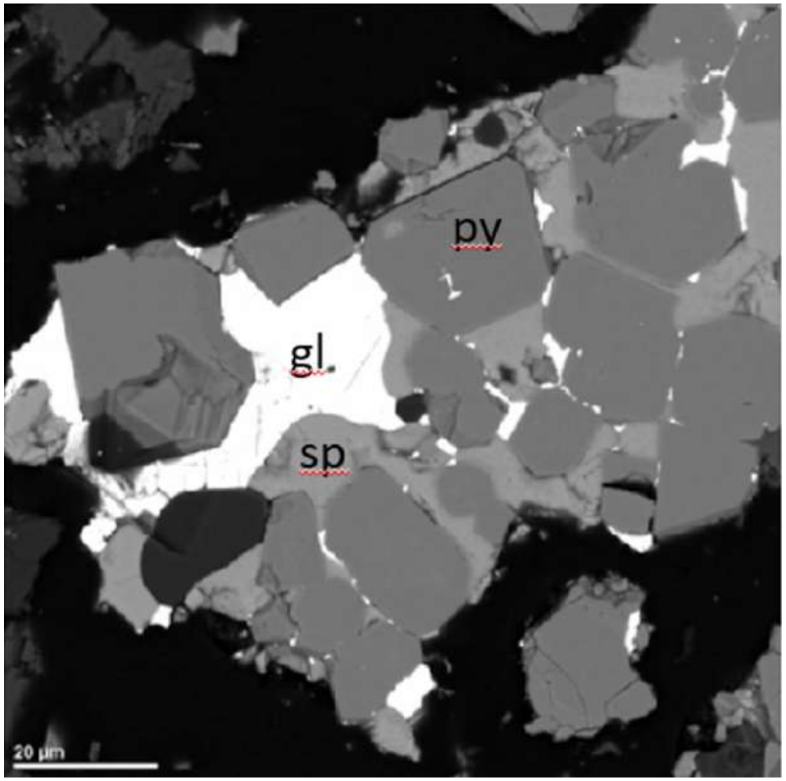
Source: VMC (PDAC 2013).

Figure 7-5: High-Grade Zn and High-Grade Cu Mineral Assemblages

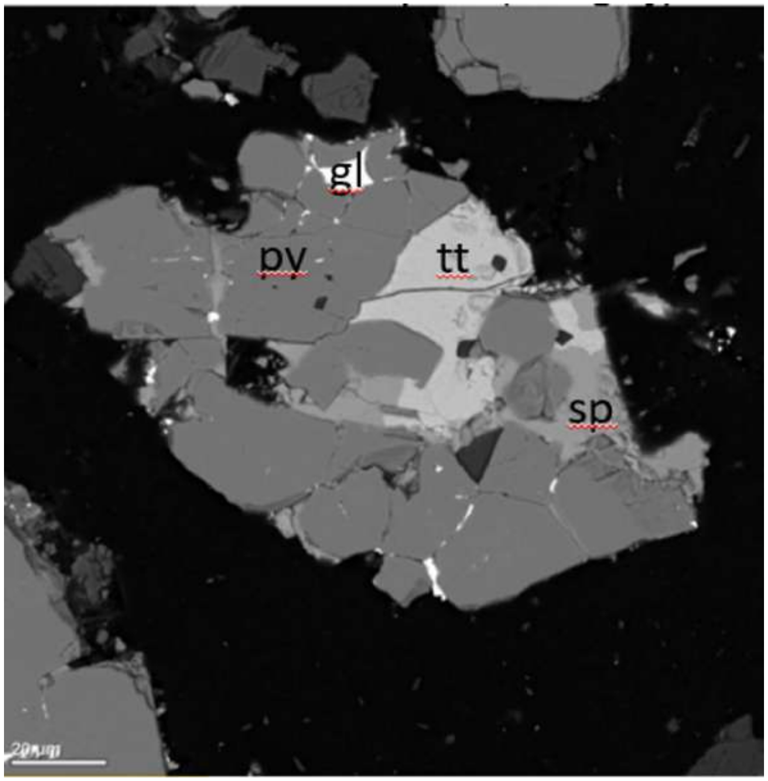


Source: VMC (CIM 2012).

Figure 7-6: Lead-Zinc Mineralization

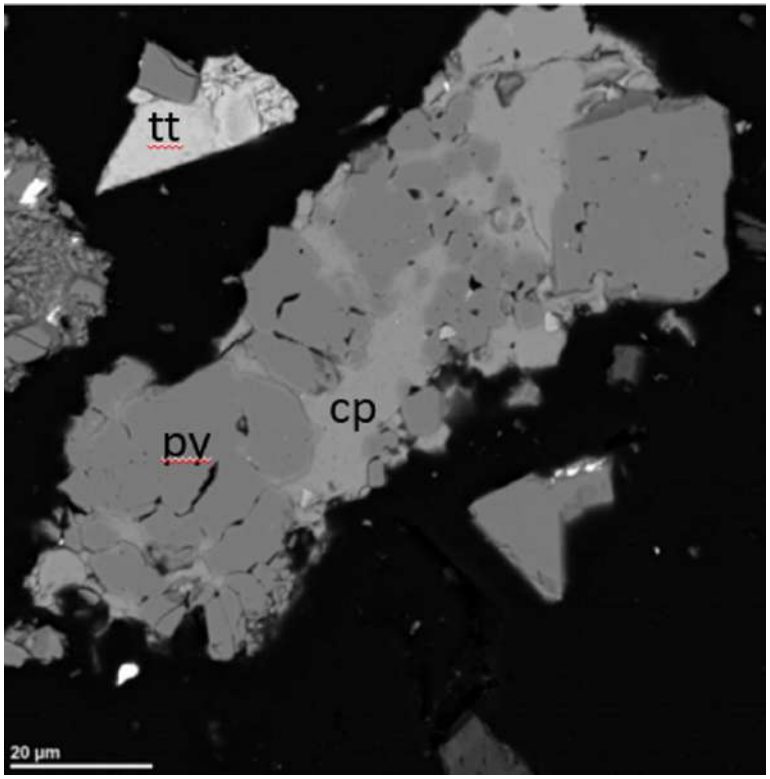


Note: MB-121-0006: Equant subhedral-anhedral pyrite (medium grey) with interstitial sphalerite (light grey) galena (white), and quartz (dark grey) Source: Gilders and Cheung (2012).

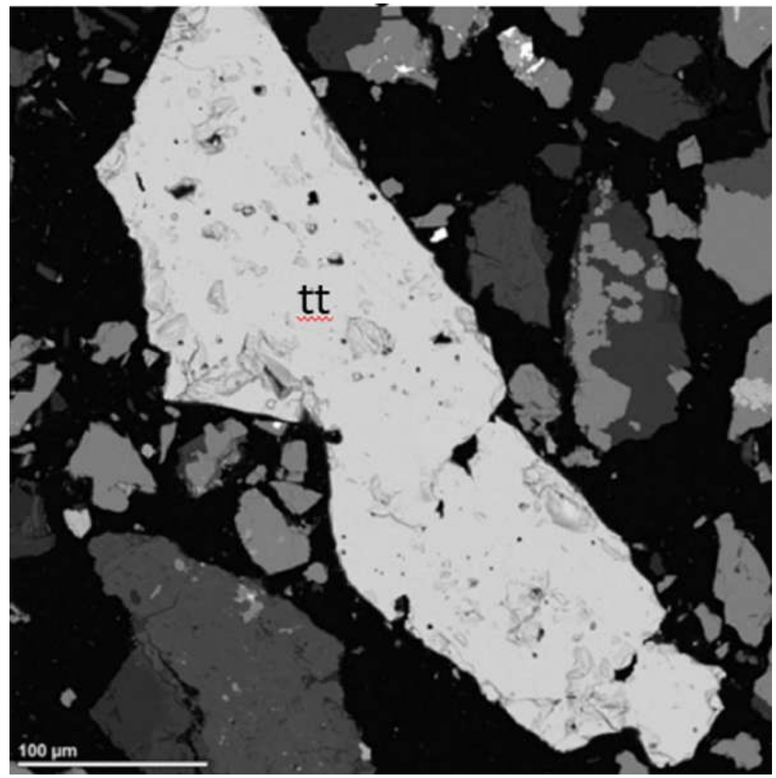


Note: MB-121-0002: Pyrite (Medium dark grey) with sphalerite (medium grey), tetrahedrite (light grey) and galena (white). The fragment also contains minor fine-grained calcite (dark grey). Source: Gilders and Cheung (2012).

Figure 7-7: Tetrahedrite (Ag) Mineralization



Note: MB-121-0003: Pyrite (medium grey) with chalcopyrite (light grey). The very fine-grained (<5 µm) equant grains (lighter grey) are arsenopyrite. The angular fragment at upper left consists of tetrahedrite (light grey) and pyrite. Source: Gilders and Cheung (2012).



Note: MB-121-0010: Coarse fragment of tetrahedrite. This grain contains 0.9 wt % Ag. Source: Gilders and Cheung (2012).

8 DEPOSIT TYPES

8.1 Classification

The Murray Brook sulphide mineralization is classified as a volcanogenic massive sulphide (VMS) deposit hosted in sedimentary rocks. VMS deposits are well-studied and documented, and are a major global source of zinc, lead, copper, silver, gold, and other metals (Galley, 2007; Franklin, 2007; Shanks et al., 2009). Although VMS deposits tend to be hosted volcanic rocks, some of those in the Bathurst mining camp are hosted in sedimentary rocks proximal to felsic volcanic-sedimentary interfaces (Franklin et al., 1981). However, at Murray Brook and elsewhere in the Bathurst mining camp, the original depositional and stratigraphic relationships are obscured by the overprinting effects of post-depositional metamorphism and polyphase deformation.

8.2 VMS Deposits

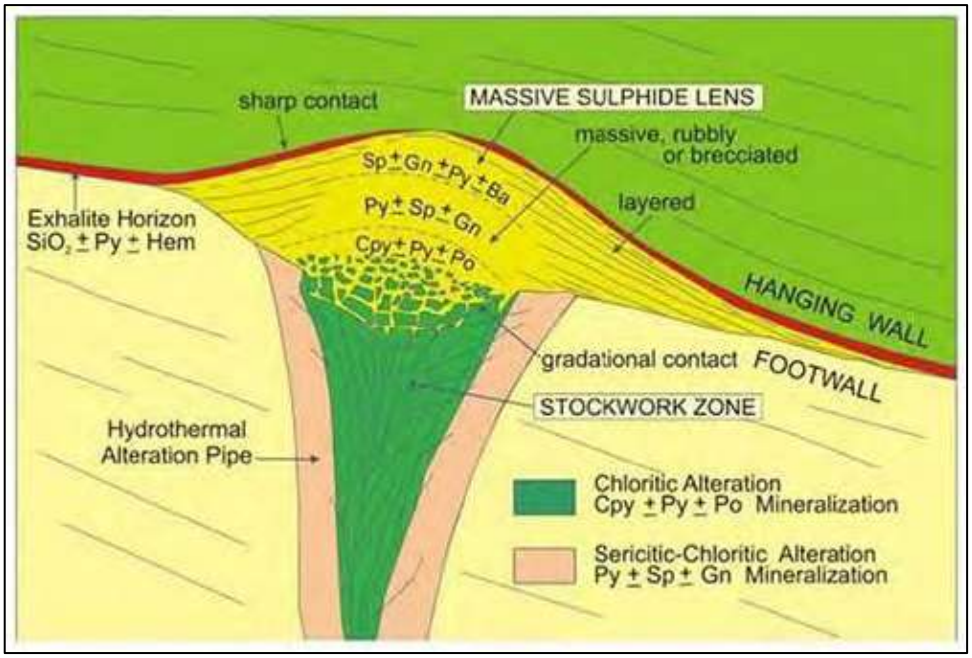
VMS deposits are characterized by mound-shaped to tabular, stratabound bodies composed of massive to semi-massive sulphides, quartz, phyllosilicate minerals, iron-oxides and altered wall rock (Galley et al., 2007) (Figure 8-1). Individual massive sulphide lenses can be >100 m thick, hundreds of metres in length, and tens of metres wide, and they form proximal to the active hydrothermal vent. The vents tend to be faults or fissures active during volcanism. The stratabound sulphide bodies are typically underlain by a “stockwork zone” or “pipe” composed of stockwork veins and disseminated sulphides surrounded by altered wall rocks.

The most common sulphide minerals in VMS deposits are pyrite, pyrrhotite, chalcopyrite, sphalerite, and galena (Lydon, 1988). Magnetite, hematite, and cassiterite are common non-sulphide metallic minerals. The mineral deposits are typically zoned, with decreasing values of chalcopyrite/(sphalerite + galena) and copper-to-zinc ratios upward and downward from the core of the alteration pipe and base of the massive sulphide lens.

VMS deposits are typically hosted in submarine mafic-felsic volcanic rock sequences formed during periods of rifting and volcanism along volcanic arcs, fore arcs, and in back-arc basins (Franklin et al., 2005). The deposits form at or near the seafloor by the reaction of metal-rich hydrothermal fluids with wall rock and seawater during volcanism (Lydon, 1988; Franklin, 2007) (Figure 8-2). Heat sources driving the hydrothermal systems are subvolcanic intrusions at depth.

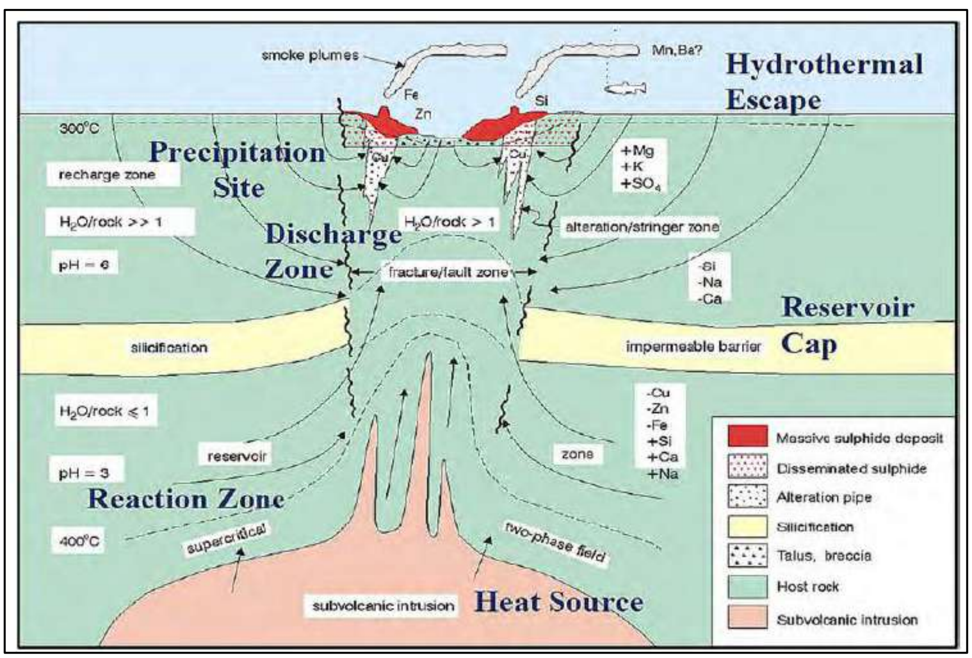
VMS deposits are characterized by a distinctly zoned alteration halo, which consists of an inner chloritized central zone and an outer potassic distal zone (Lydon, 1988). In VMS deposits of the Archean Abitibi Greenstone Belt in Ontario and Quebec, the predominant alteration mineral in the outer potassic zone is sericite. In higher-grade regional metamorphosed environments, such as the Flin Flon and Snow Lake VMS camps, lower temperature alteration minerals are replaced by higher temperature metamorphic mineral assemblages (Galley et al., 2007).

Figure 8-1: Idealized Characteristics of a VMS Deposit



Source: Lydon (1988) and Galley et al. (2007b)

Figure 8-2: Idealized Schematic of VMS Deposit Forming Hydrothermal Fluid System



Source: Franklin (2007)

VMS deposits are classified based on base metal content, gold content, and host rock lithology (Galley et al., 2007). The deposits are divided into copper-zinc, zinc-copper, and zinc-lead-copper groups based on metal ratios (Large, 1992, Franklin et al., 2005). Gold-rich VMS deposits are defined as those in which the amount of gold in ppm numerically exceeds that of the combined base metals in weight percent (Poulsen and Hannington, 1995). The Murray Brook deposit, along with the Bathurst mining camp deposits, belongs to the zinc-lead-copper group classification, which have little or no mafic rocks in their footwall sequence (McCutcheon, 1992). Other examples of zinc-lead-copper group deposits are found in the Sudbury Basin (Ontario), Omineca crystalline belt and enclaves in the Coast Batholith (British Columbia) (Franklin, 1993), and in the Iberian Pyrite Belt in Spain and Portugal (Baily and Hudson, 1991). Modern examples of the zinc-lead-copper group have been discovered in the Lau Basin (Von Stakelburg and Brett, 1990; and the Okinawa Trough (Urabe et al., 1990).

Most of VMS deposits occur in clusters that define a mining camp. Canadian examples are Bathurst, Flin Flon (Manitoba), and Noranda (Québec) (Galley et al., 2007). These clusters of deposits typically occur in linear rifts or caldera features. Regional scale alteration reflects a fluid convection system developed above subvolcanic intrusions, in the order of 5 to 15 km long and 1 to 3 km thick, and reacting with the surrounding country rocks and seawater to form hydrothermal minerals and massive copper-zinc-lead (silver, gold) sulphide deposits (Dimroth et al., 1983; Gibson and Watkinson, 1990; Galley et al., 1993; Powell et al., 1993; Franklin, 2007).

VMS deposits occur worldwide and range in age from 3.4 Ga to actively forming deposits of modern seafloor hydrothermal vents in bimodal mafic-dominated volcanic settings (Galley et al., 2007). More than 800 VMS mineral deposits are known globally, ranging from <0.2 to >150 Mt in size (Galley et al., 2007).

9 EXPLORATION

9.1 Introduction

Canadian Copper have yet to carry out any exploration on the Murray Brook property. The work summarized in this section was conducted by VMC from 2010 to 2016 and Puma from 2017 to 2019. As exemplified by the discovery history of the Murray Brook deposit, heavy mineral and soil geochemical surveys are effective exploration tools. Geophysical surveys, such as magnetic, electromagnetic, and gravity surveys, are also effective exploration tools for concealed deposits. Trenching and stripping work has also been completed.

9.2 Geophysical Surveys

In 2010 and 2011, VMC carried out a Fugro airborne gravity gradiometry survey (line spacing = 200 m, tie line spacing = 2,000 m) and a Fugro HeliTEM electromagnetic survey (line spacing = 100 m, tie line spacing = 1,000 m) over their various properties and areas of interest, including the Murray Brook deposit. The ground geophysical portion of the exploration consisted of a gravimetric survey, which detected the massive sulphide deposit at depth.

In addition to the reports listed above, the Murray Brook area has been covered during the regional government-industry geological and geophysical surveys, such as the 1994-1999 Extech II program, the 2004 Bell geospace airborne full tensor gravity survey, the 2004 MegaTEM II airborne electromagnetic (EM) and magnetic survey, and the 2013 Falcon airborne gravity gradiometry (AGG) survey.

9.3 Mineral Prospecting and Geological Mapping

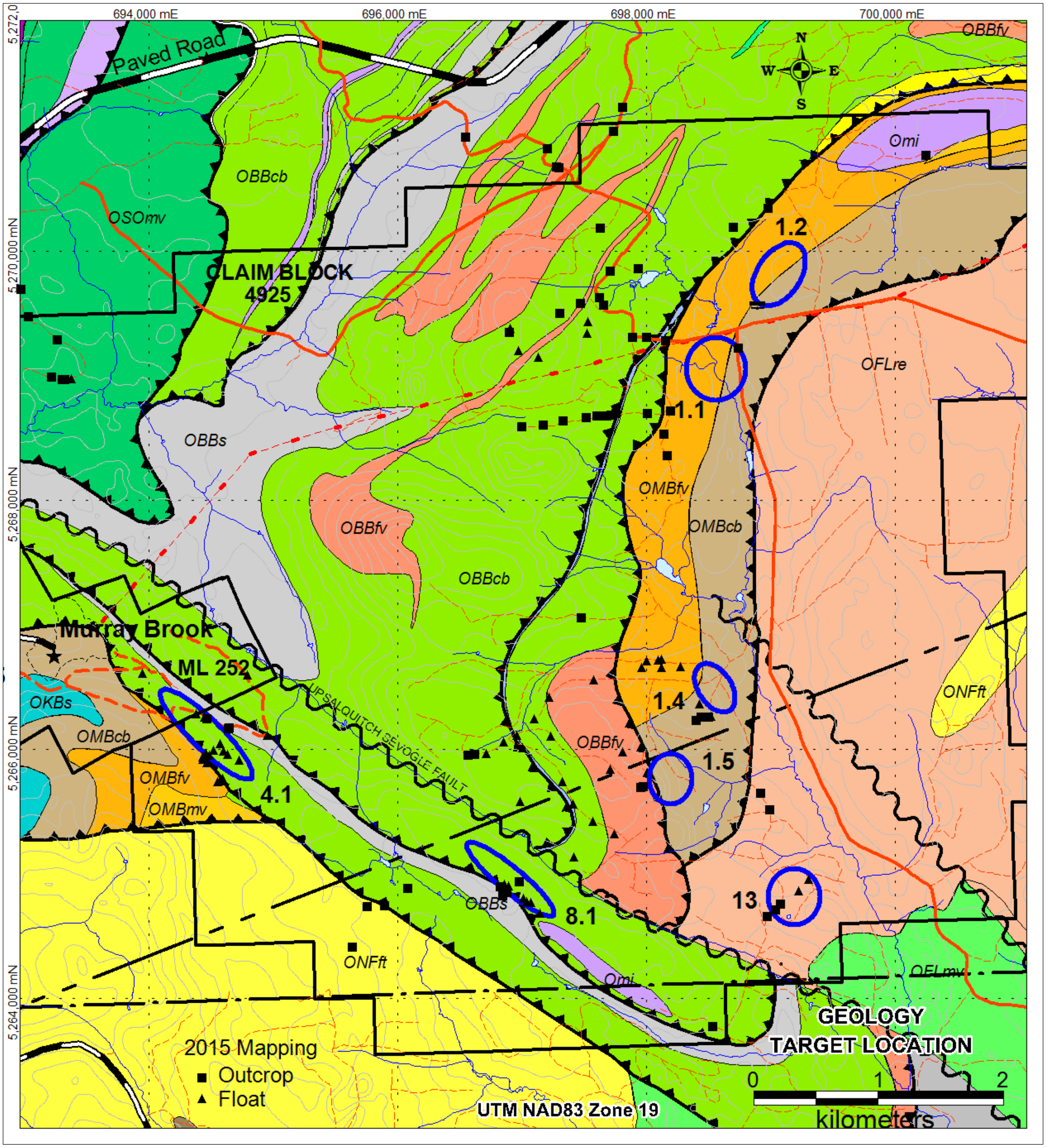
The geophysical surveys resulted in the definition of seven drill targets. In July and August 2015, mineral prospecting and geological mapping and soil geochemical surveys were completed over sections of the eastern part of Mining Claim 4925 by Garth Graves and GeoXplore Surveys Inc. of Bathurst, New Brunswick. The targets are shown on Figure 9-1.

Regional geological mapping and prospecting were conducted on the Murray Brook East claim 4925 in July and August, 2015. The prospecting and mapping were carried out on soil sampling grids that covered the exploration targets and on a more regional scale outside the target areas.

The geological mapping program confirmed results of previous survey work that defined the regional geological setting of the Murray Brook East claim block. The felsic volcanic rocks of the Mount Brittain Formation are more extensive in some areas, specifically Target 1.4 and Target 1.5, than previously indicated. Gravity anomalies associated with Targets 4.1 and 8.1 may be the result of gabbro intruding basalt. However, the area is structurally complex and the possibility that Mount Brittain rocks have been thrust into the geological sequence is indicated by previous drilling at Target 4.1.

Prospecting and mapping in the area of Targets 1.1, Target 1.2, Target 1.4, and Target 1.5 found no obvious explanation for the gravity and EM anomalies. These targets are underlain by the Mount Brittain Formation and are in a geological environment similar to the Murray Brook deposit. They were considered to be valid targets for drill follow-up.

Figure 9-1: Geology and Target Locations on the Murray Brook Property

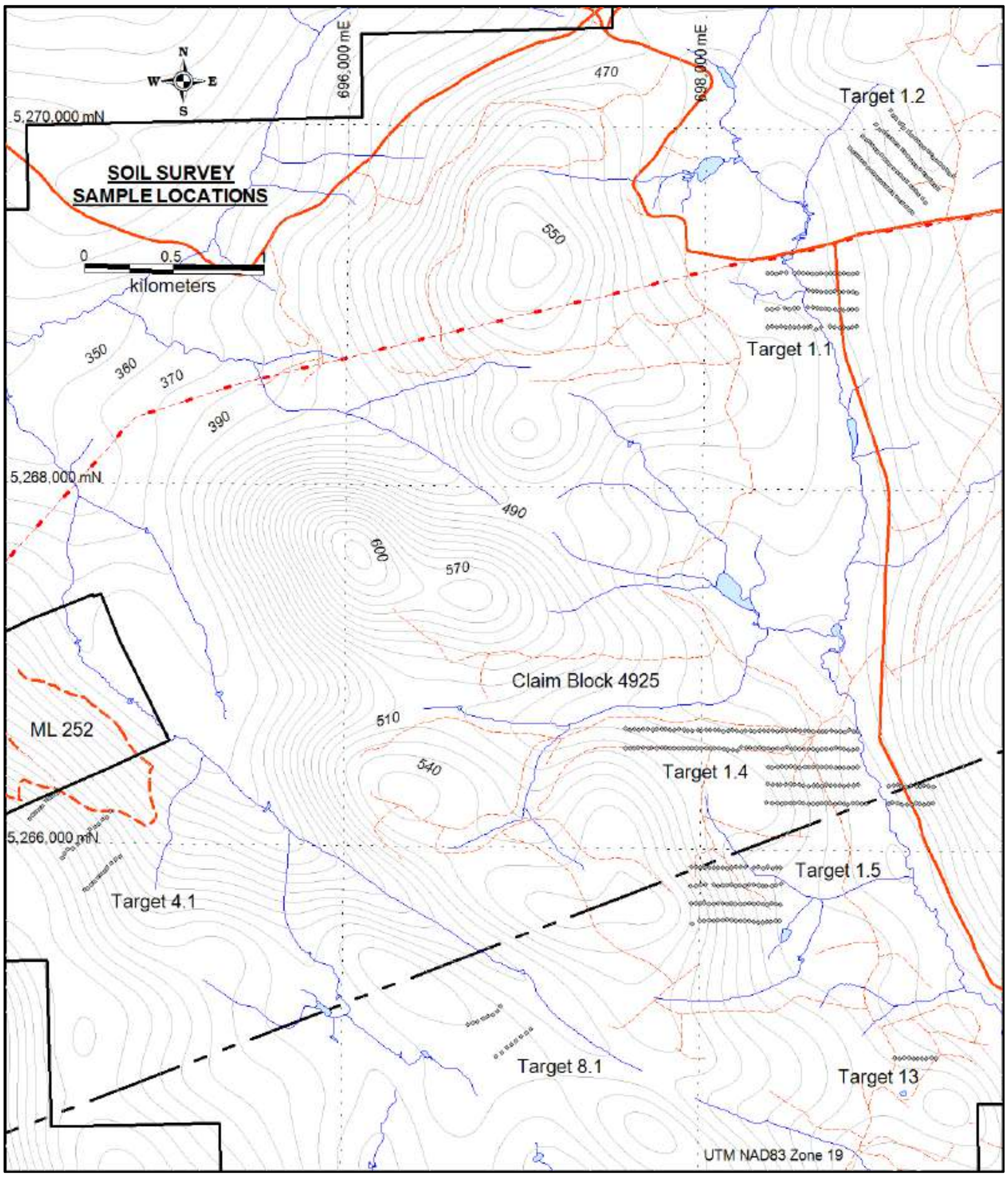


Source: VMC (2015)

9.4 Soil Surveys

During July and August, 2015, 488 B-horizon soil samples across sections of the Murray Brook east claim were collected. These samples covered the seven target areas selected for detailed follow-up. Samples were collected at 25 m intervals on uncut grids along flagged lines. Lines were spaced at various distances depending on the target, and were 100 m apart on Targets 1.1, 1.2, and 1.5 (Figure 9-2). Locations were recorded using a handheld GPS.

Figure 9-2: Soil Survey Sample Locations



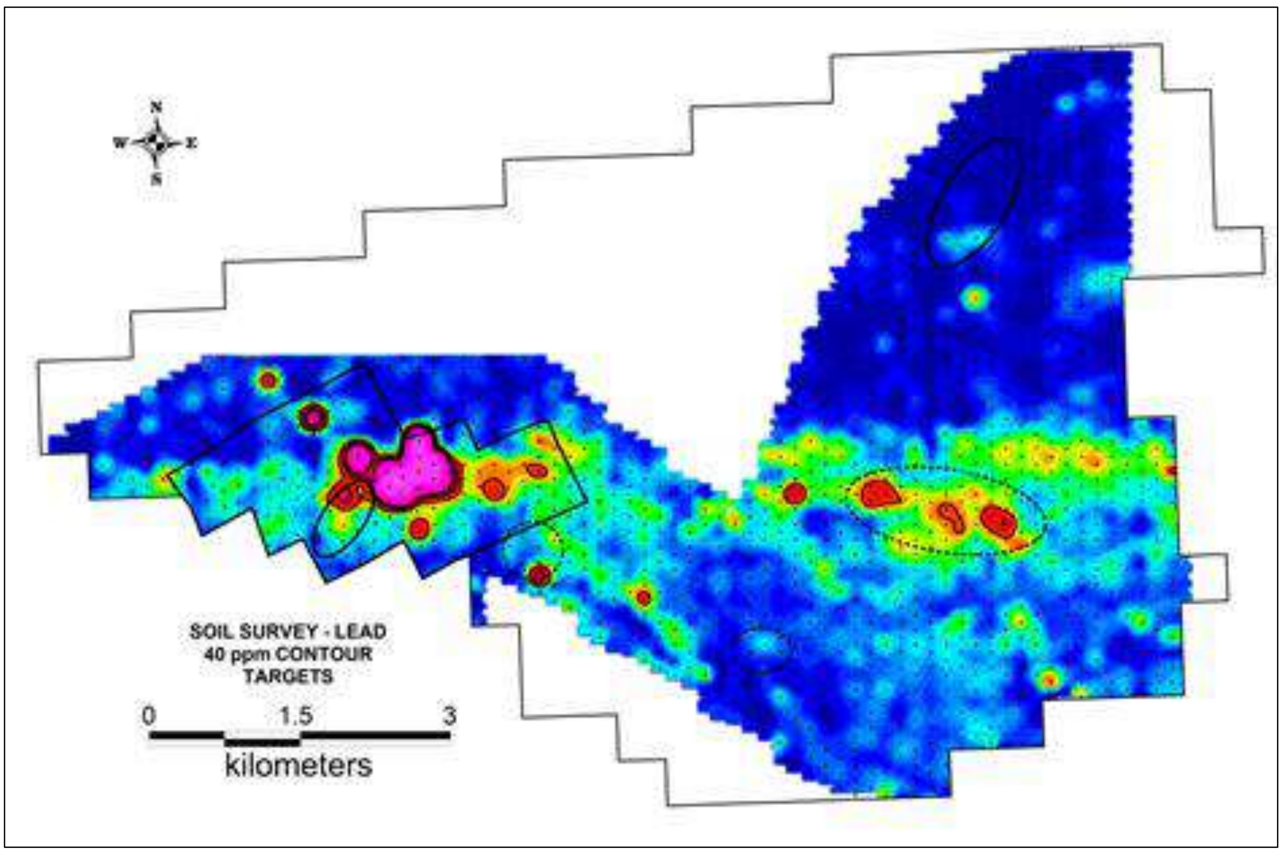
Source: VMC Assessment Report (2015).

The soil profile is well-developed throughout the area and the B-horizon was sampled at a depth of 10 to 20 cm for >90% of the samples. Some samples were light grey-brown in areas of steep terrain with rocky soil and a few slightly organic, brown-coloured samples were collected in low ground. Samples were dried and shipped to TSL Labs in Saskatoon, Saskatchewan. Samples were sieved to -80 mesh, dissolved using an aqua regia digestion, and analysed by ICP-AES for 29 elements.

The soil survey has provided detail over the selected targets areas. Target 1.4 and Target 1.5 are associated with strongly anomalous lead and arsenic and elevated copper, zinc, antimony, and molybdenum in soil. Targets 1.1 and Target 1.2 have sporadic elevated copper and zinc values, but they are probably affected by thick till cover. Target 4.1, Target 8.1, and Target 13 do not have associated soil anomalies, which indicates an extensive hydrothermal system that is not exposed at surface. All of the soil results are affected by glacial transport that has dispersed till in an eastward direction.

The soil surveys, along with the prospecting and mapping surveys, further defined the prospective stratigraphy for several kilometres on strike from the Murray Brook deposit and identified base metal soil anomalies for additional exploration. Soil geochemistry results for lead are shown in Figure 9-3.

Figure 9-3: Contoured Results for Analysis of Lead in Soil



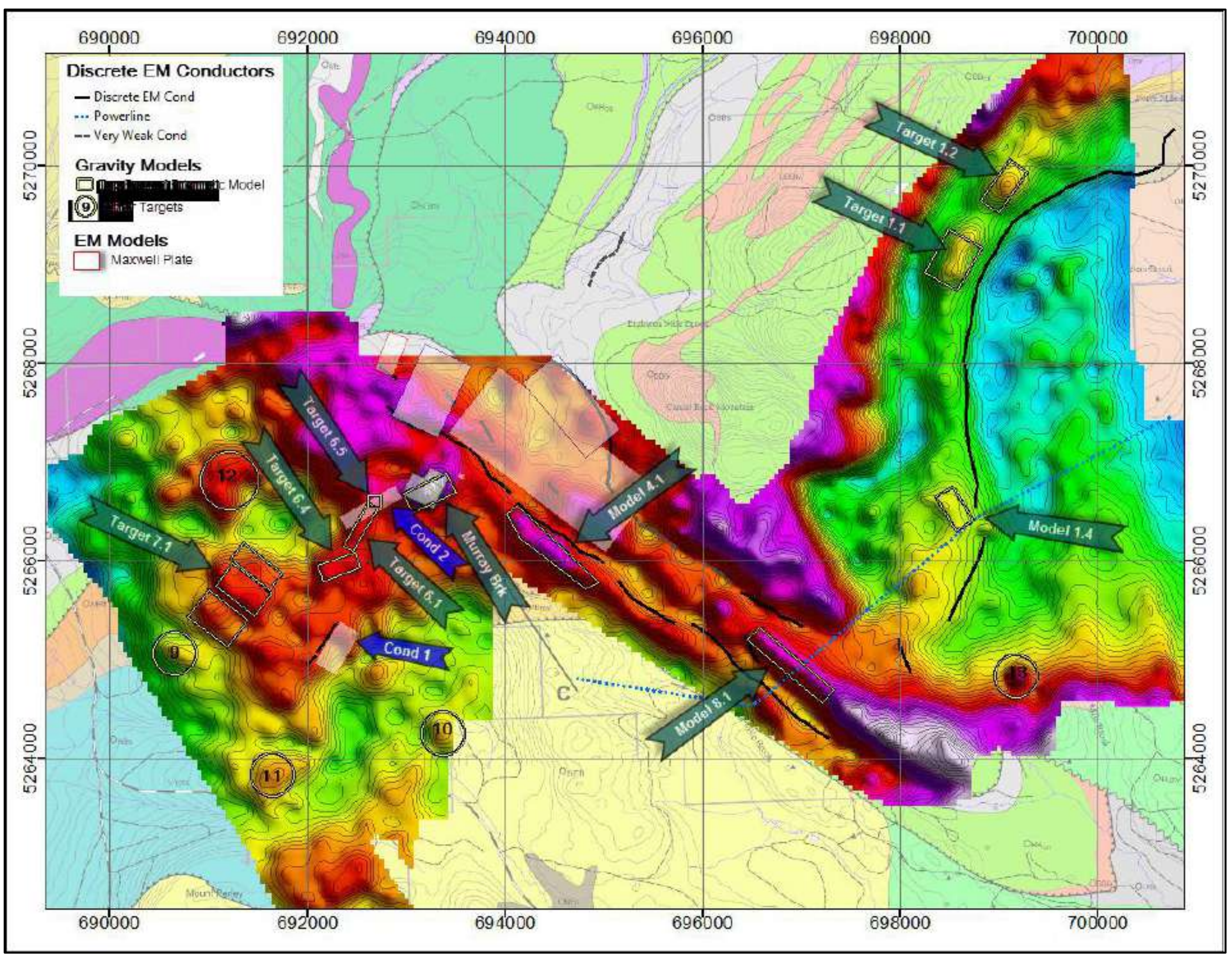
Source: Puma Exploration (2017).

9.5 Trenching

The trenching and prospecting work completed by Puma on the eastern part of claim block 4925 from May to July 2017 is summarized here from the assessment reports of Gagne and Hupe (2017 and 2019). The Mount Brittain Formation was targeted to locate and delineate the sedimentary unit of the Charlotte Brook Member, which hosts the Murray Brook deposit 5 km to the west.

The 2017 program was focused around gravity anomalies (Figure 9-4) and along the contact between the felsic volcanic and the sedimentary rock of the Mount Brittain Formation. A previous ground gravity survey identified density anomalies 1.1, 1.2 and 1.4. Anomalies 1.1 and 1.2 are located between the felsic volcanic and sedimentary rocks of the Mount Brittain Formation, which is a similar geological setting to the Murray Brook deposit. Anomaly 1.4 is within sedimentary rocks (Charlotte Brook Member) of the Mount Brittain Formation, near the thrust contact with volcanic rocks of the Flat Landing Brook Formation.

Figure 9-4: Gravity Target Areas Residual Gravity Image



Source: Gagné and Hupé (2017).

The Charlotte Brook Member had been previously mapped over 6.5 km on the eastern part of the property. Outcrops are very rare, but trenching with an excavator successfully reached bedrock (Figure 9-5). A previous LiDAR survey was helpful in locating old access roads and define areas for trenching. All access roads, trails, and streams in this area of claim 4925 were prospected to locate the favourable horizon (sediment unit). Twenty prospecting samples were taken, but only four samples were analysed with no significant results. However, a new outcrop of black shale was found in the north part of the area.

Figure 9-5: Photographs of Trenches 17-07 and 17-26

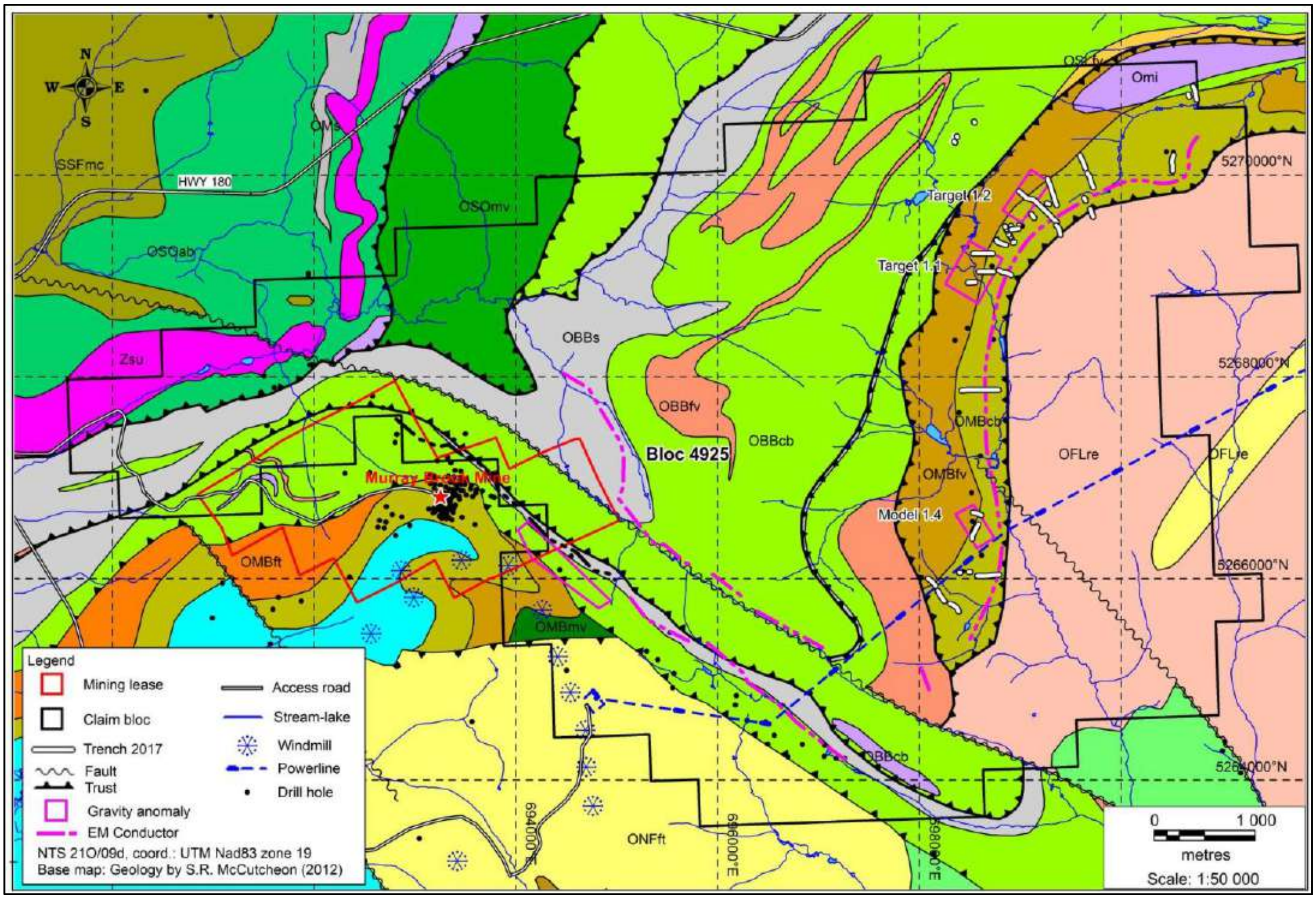
Source: Gagné and Hupé (2017).

Trenches were excavated over 6.5 km along the Charlotte Brook Member. The 34 trenches totalling 3,831 m are located on Figure 9-6. Most of the trenches succeeded in exposing sedimentary rock in contact with felsic volcanic unit. Some of the trenches were too deep to reach bedrock with thick overburden, because the excavator boom was only approximately 3 m long.

On completion, the geology of each of the trenches was mapped and a total of 127 samples were collected for assay. All samples were analysed for gold by AA and 41 elements by ICP-41 at ALS Minerals Laboratory. The black shales (sediment unit) were analysed (64 samples) for whole rock by ICP-09.

The contact between the felsic volcanic rock (OMBfv) and the sedimentary rock (OMBcb) was observed in Trenches 1, 2, 3, and 33. It seems gradually interbedded from black shale, sandstone, tuff to rhyolite cross-cut by gabbro. Contacts are rarely well-exposed, but some are faulted. In Trench 4, the black shale is directly in fault contact with gabbro.

Figure 9-6: 2017 Trenches on Geological Map



Source: Gagné and Hupé (2017).

The contact between the sediment (OMBcb) and felsic volcanic of the Flat Landing Formation (OFLre) was observed in Trenches 25 and 29. Overburden around Trenches 15, 16 and 17, near the Forth Miles stream, is >3 m thick.

Anomalies in As, Cu, Zn >100 ppm and Fe or Fe_2O_3 >10% are observed in the black shale of Trenches 4, 5, 10, 26, 27, 33 and 34. They are located mainly near the gravity Target 1.2 and toward northeast, where the sedimentary unit seems to thicken. Precious metal results are below the detection limit. Typical disseminated to small, massive bands or veins of sulphide (mainly pyrite) were observed in Trenches 26, 27, and 34.

Similar disseminated sulphide (pyrite) has been seen in Trenches 4 and 10 with SiO_2 >80% probably associated with silicification or quartz vein around those samples. The black shale in Trenches 4 and 5 is also graphitic with irregular quartz veins.

Although sulphide mineralization was found in the area of gravity anomaly 1.2 and toward northeast, no indication of mineralization was found around Targets 1.1 and 1.4, due to thick overburden.

The 2017 assay results for the trench samples were subject to an alteration study in 2019. Specifically, a series of geochemical discrimination diagrams were generated to determine mineralized and altered areas. Many of the samples present coincident alteration in some diagrams and most of the anomalous or altered samples are located in the northeastern part of the claim. The northeastern part of claim 4925 should be prioritized for future exploration.

9.6 Geophysical Inversion and Re-Modelling Studies

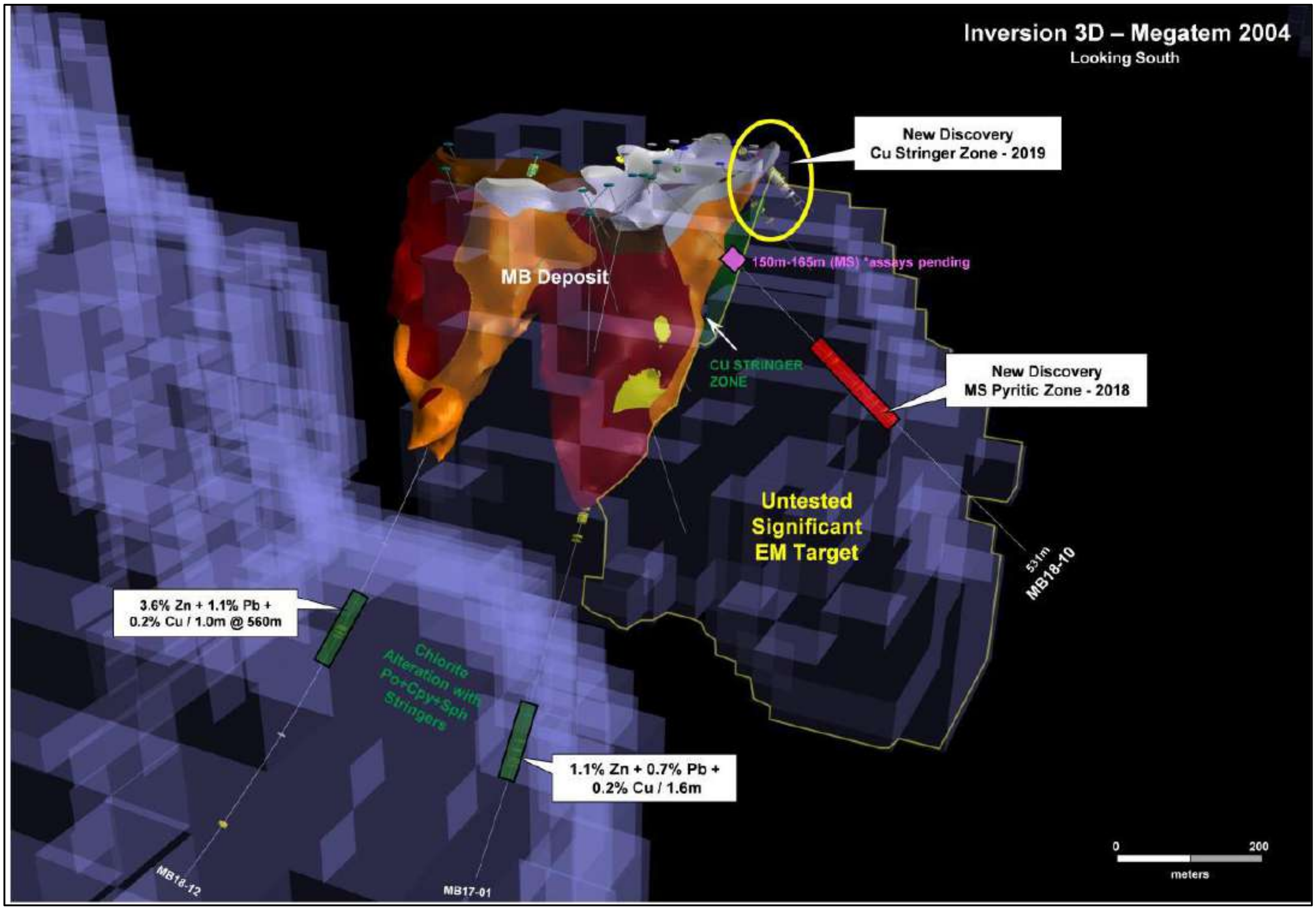
The information in this section is taken largely from Gagné and Hupé (2019).

The first geophysical re-modelling study was completed by Fullagar Geophysics and consisted in 1D TEM inversion using VPem1D software and 3-D TEM inversion using VPem3D software. The database used was the Noranda 2004 MegaTEM survey (Block 2). The VPemID software performs layered earth inversion of airborne and ground transient electromagnetic (TEM) data, both "unconstrained" and geologically constrained. The VPem3D software performs approximate 3-D inversion (both "unconstrained" and geologically constrained) of airborne, ground, and downhole transient electromagnetic (TEM) data. The VPem1D (channels 6 to 20) method suggests the conductive unit (shale) dips mainly to the northwest. The VPem3D method gives a similar result, except with less accuracy on surface.

The second geophysical re-modelling study was performed by Fathom Geophysics and consisted in 3-D inversion and structure detection of magnetic and gravity data. The database used was the Noranda 2004 MegaTEM survey for the magnetic data and extra data from the Murray Brook Project. Two magnetic inversions were calculated, including one coarse cell dimension (250 m x 250 m x 125 m) and one that is more detailed (75 m x 75 m x 37.5 m). Both the magnetic and gravity inversions can be used to plan the next phases of the exploration program. Note that this geophysical interpretation was performed after the trenching and drilling programs. In general, the magnetic interpretation corelate well with the mafic volcanic rocks of the Boucher Brook Formation and the conductivity correlate well with the sedimentary rocks of the Mont Brittain Formation.

Geophysical data were afterward interpreted in closer integration with the geological data by Mira Geoscience of Quebec (Mira) in 2018, by combining the geophysical modelling and inversion with 3-D geological modelling. Mira completed 3-D inversion modelling, integration, and visualization of airborne gravity, magnetic, electromagnetic analysis, and integrated the results with the geological data for the area. The objective of this work was to provide useful 3-D physical property products that can be employed in regional exploration to target prospective ground. The products are 3-D inversion models of density contrast, magnetic susceptibility, and electrical conductivity, and integrated products with combined geology. Target features around the Murray Brook deposit are shown in Figure 9-7.

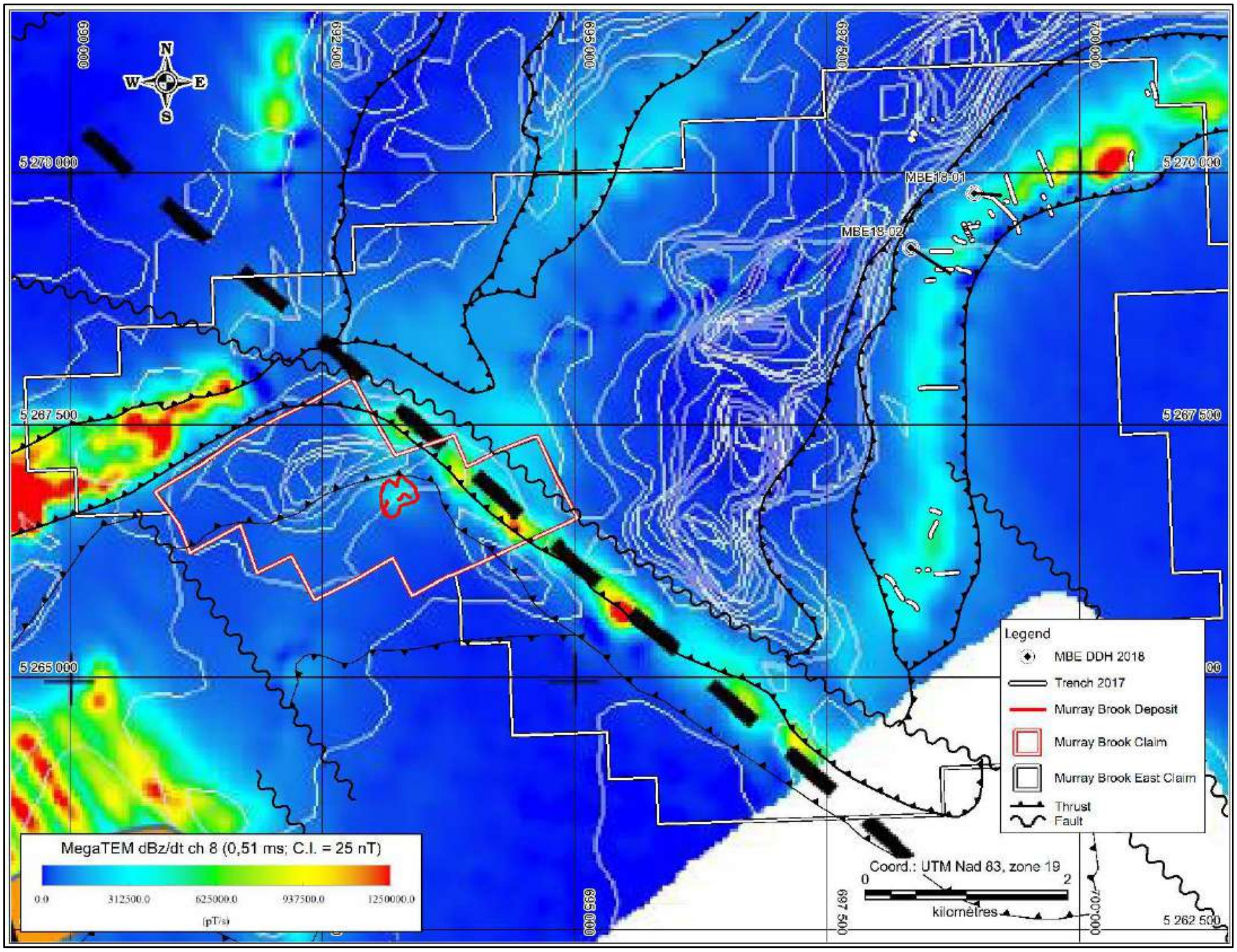
Figure 9-7: MegaTEM Inversion 3-D Model Target West of Murray Brook Deposit



Source: Puma press release dated April 10, 2019.

Geological features like the major Upsalquitch-Sevogle Fault and the upper and lower contacts of the Mount Brittain Formation were also interpreted. Compared with the Murray Brook deposit, which has coincident gravity and conductivity (EM) features, similar proximate gravity and conductivity (EM) anomalies are evident to the east on claim 4925 that may warrant further exploration (Figure 9-8).

Figure 9-8: Modelled Magnetic and MegaTEM Features



Source: Gagné and Hupé (2019).

10 DRILLING

As of the effective date of this report, Canadian Copper has not conducted any drilling on the Murray Brook property. The drilling work summarized in this Section was previously conducted by VMC and Puma and forms the basis for the mineral resource estimate described in Section 14 of this report.

The drilling was completed using a skid-mounted diamond drill with NQ size drill core. Most of the drill holes were completed vertically and cross-cut the mineralization at angles of 40° to 45°.

10.1 2010 and 2011 Drill Programs – VMC

VMC's drilling at Murray Brook commenced in 2010 with the drilling of four due diligence holes totalling 595.2 m (drill hole MB-10-14 was abandoned at 39 m). These holes were consistent with historical results, with significant intersections of zinc, copper, lead, gold, and silver were reported. VMC duly finalized its Agreement with the Owners.

In 2011, 63 vertical drill holes totalling 10,499.4 m were drilled. The results were announced in ELN news releases (August 30, 2011, November 28, 2011, January 16, 2012, and January 23, 2012). The drill hole specifications for the 2010-2011 drilling are presented in Table 10-1. The composite assay results are presented in Table 10-2.

Three objectives of the Phase I and Phase II drilling program were realized: (1) infill drilling to close large (100 m) gaps in the historical drill coverage; (2) step-out drilling to define the margins of the deposit, and (3) due diligence drilling (595.2 m) to confirm results from historical drill programs. The results of the 2011 drilling provided additional data for use in estimating indicated and inferred mineral resources.

An analysis of the 2011 drill program indicated that approximately 18,000 m of additional infill and definition drilling was warranted in a 2012 drill program.

Table 10-1: Diamond Drill Specifications Phase I and Phase II

Drill Hole ID	Easting (m)	Northing (m)	Elevation (masl)	Azimuth (°)	Inclination (°)	Length (m)
MB-10-15	693,051.7	5,266,858	449.32	90	-60	276
MB-10-16	693,213.0	5,266,763	469.33	360	-90	164.2
MB-10-17	693,301.8	5,266,765	470.14	360	-90	116
MB-2011-01	693,246.1	5,266,753	469.45	360	-90	125
MB-2011-02	693,186.2	5,266,734	469.74	110	-75	197
MB-2011-03	693,328.6	5,266,854	471.69	360	-90	206
MB-2011-04	693,300.8	5,266,903	467.10	360	-90	233.6
MB-2011-05	693,283.6	5,266,824	468.26	360	-90	177
MB-2011-06	693,416.3	5,266,788	479.24	360	-90	107
MB-2011-07	693,395.2	5,266,829	477.57	360	-90	167
MB-2011-08	693,351.8	5,266,761	471.35	360	-90	92
MB-2011-09	693,158.6	5,266,850	455.81	120	-75	179.1
MB-2011-10	693,197.5	5,266,897	456.49	360	-90	245
MB-2011-11	693,193.2	5,266,921	451.01	360	-90	277
MB-2011-12	693,059.1	5,266,935	443.60	110	-70	295.8
MB-2011-13	693,210.4	5,266,678	470.09	360	-90	153
MB-2011-14	693,057.4	5,266,858	449.61	110	-70	305
MB-2011-15	693,266.7	5,266,705	470.65	360	-90	98
MB-2011-16	693,041.1	5,266,817	457.23	110	-75	272
MB-2011-17	693,160.9	5,266,688	473.44	360	-90	182
MB-2011-18	693,170.3	5,266,665	474.56	360	-90	131
MB-2011-19	693,146.6	5,266,649	478.82	360	-90	100
MB-2011-20	693,195.8	5,266,676	471.16	360	-90	152
MB-2011-21	693,167.9	5,266,595	482.54	360	-90	63.5
MB-2011-22	693,208.2	5,266,647	472.68	360	-90	119
MB-2011-23	693,208.6	5,266,710	469.78	360	-90	155
MB-2011-24	693,247.6	5,266,696	470.16	360	-90	98
MB-2011-25	693,261.0	5,266,670	471.11	360	-90	74
MB-2011-26	693,153.2	5,266,704	473.13	360	-90	173
MB-2011-27	693,238.6	5,266,718	469.58	360	-90	86
MB-2011-28	693,261.8	5,266,733	470.03	360	-90	72
MB-2011-29	693,286.3	5,266,746	470.35	360	-90	86
MB-2011-30	693,236.7	5,266,784	470.64	360	-90	128
MB-2011-31	693,160.2	5,266,746	479.84	360	-90	218
MB-2011-33	693,169.6	5,266,784	475.80	360	-90	243
MB-2011-34	693,113.1	5,266,783	477.11	360	-90	245
MB-2011-37	693,161.5	5,266,809	470.49	360	-90	251
MB-2011-38	693,261.4	5,266,794	469.11	360	-90	152.5
MB-2011-39	693,173.7	5,266,853	457.96	360	-90	257
MB-2011-40	693,250.3	5,266,818	466.82	360	-90	155
MB-2011-41	693,322.8	5,266,794	470.61	360	-90	161
MB-2011-42	693,298.0	5,266,722	470.79	360	-90	75
MB-2011-43	693,340.3	5,266,832	473.05	360	-90	170
MB-2011-44	693,351.9	5,266,805	472.78	360	-90	143
MB-2011-45	693,375.7	5,266,790	475.00	360	-90	125
MB-2011-46	693,402.5	5,266,760	477.75	360	-90	80
MB-2011-47	693,413.6	5,266,740	479.76	360	-90	75
MB-2011-48	693,193.6	5,266,798	468.89	360	-90	182
MB-2011-49	693,183.5	5,266,764	473.80	360	-90	227
MB-2011-50	693,150.0	5,266,778	476.79	360	-90	251
MB-2011-51	693,133.2	5,266,797	472.07	360	-90	245
MB-2011-52	693,123.0	5,266,824	464.54	360	-90	257
MB-2011-53	693,120.2	5,266,865	450.28	200	-85	233
MB-2011-54	693,200.2	5,266,866	455.92	360	-90	251
MB-2011-55	693,263.4	5,266,863	463.54	360	-90	191
MB-2011-56	693,349.9	5,266,749	471.23	360	-90	75
MB-2011-57	693,162.7	5,266,881	451.73	360	-90	266
MB-2011-58	693,129.3	5,266,685	474.86	360	-90	140
MB-2011-59	693,183.9	5,266,639	472.84	360	-90	116
MB-2011-60	693,230.5	5,266,656	469.12	360	-90	141
MB-2011-61	693,128.5	5,266,735	484.12	360	-90	209
MB-2011-62	693,124.3	5,266,764	477.20	360	-90	233
MB-2011-63	693,170.1	5,266,903	452.60	360	-90	272

Notes: 1. Coordinates are in UTM NAD83 Zone 19; 2. Drill holes MB-2011-32, MB-2011-35 and MB-2011-36 were abandoned.

Table 10-2: Significant Phase I and Phase II Drill Intercepts

Drill Hole ID	From (m)	To (m)	Width (m)	Cu (%)	Pb (%)	Zn (%)	Au (g/t)	Ag (g/t)
MB-2010-15	163.8	167.7	3.9	0.10	2.00	9.16	1.6	77.00
and	201.0	218.0	17.0	0.10	4.24	7.51	0.96	84.0
and	232.0	235.0	3.0	0.13	2.65	6.25	1.02	58.0
MB-2010-16	44.0	71.0	27.0	0.47	3.39	9.56	0.21	122.0
and	79.0	91.0	12.0	0.08	2.22	6.89	0.10	89.0
MB-2010-17	15.0	95.0	80.0	0.33	0.63	1.28	0.65	26.32
MB-2011-01	53.0	77.8	24.8	0.11	1.22	3.34	0.20	41.00
MB-2011-02	42.5	164.3	121.8	0.26	1.07	3.32	0.26	34.61
MB-2011-03	92.0	172.6	80.6	0.83	0.98	1.89	0.80	47.06
MB-2011-04	141.0	217.0	76.0	0.53	1.21	2.35	1.31	45.28
MB-2011-05	73.5	155.0	81.5	1.12	0.44	1.02	0.46	20.97
MB-2011-06	47.4	77.7	30.3	0.26	0.99	2.13	0.90	46.31
MB-2011-07	112.5	138.1	25.6	0.34	1.05	2.28	0.92	46.52
MB-2011-08	11.0	64.2	53.2	0.30	0.80	1.74	1.04	39.66
MB-2011-09	101.0	176.1	75.1	0.13	1.43	3.84	0.36	59.37
MB-2011-10	179.5	216.0	36.5	0.25	1.90	4.65	1.00	69.86
MB-2011-11	187.6	201.8	14.2	0.20	1.52	3.73	0.79	52.94
MB-2011-13	27.0	126.0	99.3	0.22	1.16	3.38	0.62	40.59
MB-2011-14	160.5	225.0	64.5	0.23	0.78	3.87	0.66	35.5
MB-2011-15	29.0	35.3	6.3	0.16	1.21	3.84	0.11	8.38
MB-2011-17	24.1	126.65	102.6	0.65	0.47	1.84	0.20	23.65
MB-2011-18	47.0	107.0	60.0	1.01	0.04	0.19	0.20	11.95
MB-2011-19	23.0	77.0	54.0	0.40	0.43	1.14	0.86	22.29
MB-2011-20	15.0	125.0	110.0	0.32	0.71	2.41	0.25	27.34
MB-2011-21	19.65	31.6	12.0	0.90	0.04	0.15	0.15	10.72
MB-2011-22	17.6	95.2	77.6	0.29	0.81	2.42	0.44	32.96
MB-2011-23	31.5	107.0	75.5	0.38	0.68	2.16	0.30	24.69
MB-2011-24	38.0	55.9	17.9	0.08	0.43	0.68	0.03	8.56
MB-2011-25	nsv							
MB-2011-26	29.0	142.7	113.7	0.31	0.26	1.19	0.26	18.94
MB-2011-27	38.0	69.5	31.5	0.51	0.20	0.63	0.04	7.82
MB-2011-28	38.0	42.5	4.5	0.34	0.20	0.63	0.04	7.82
MB-2011-29	21.0	57.3	36.3	0.19	0.92	1.90	0.80	33.39
MB-2011-30	44.0	103.0	59.0	0.14	1.55	4.58	0.51	68.15
MB-2011-31	53.0	193.3	140.3	0.32	1.03	3.73	0.27	43.24
MB-2011-33	59.0	215.1	156.1	0.23	0.85	2.64	0.41	29.94
MB-2011-34	129.6	212.0	82.4	0.13	1.19	5.05	0.3	44.03
MB-2011-37	88.0	234.4	146.4	0.16	1.33	3.83	0.45	49.2
MB-2011-38	46.10	111.6	65.54	0.59	0.40	0.84	0.78	21.5
MB-2011-39	118.9	222.0	103.1	0.11	1.81	5.45	0.51	65.7
MB-2011-40	83.0	131.0	48.0	0.33	0.41	0.93	0.69	21.2
MB-2011-41	14.0	136.0	122.0	0.89	0.73	1.58	0 .94	37.8
MB-2011-42	15.0	18.0	3.0	0.21	0.32	2.65	0.73	22.8
MB-2011-43	76.0	143.8	67.8	0.41	0.60	0.97	0.71	36.1
MB-2011-44	36.0	60.0	24.0	0.53	0.59	1.00	0.61	32.9
and	65.8	110.7	44.9	0.66	0.79	1.49	0.90	40.4
MB 2011-45	20.2	33.6	13.4	0.31	1.14	2.73	0.97	47.0
and	41.0	95.0	54.0	0.42	0.74	1.78	0.63	35.1
MB-2011-46	20.15	46.80	26.6	0.39	0.63	1.79	0.26	33.9
MB-2011-47	nsv							
MB-2011-48	60.5	161.0	100.5	0.16	1.71	4.65	0.36	56.5
MB-2011-49	35.0	181.0	146.0	0.59	1.40	3.85	0.63	56.1
MB-2011-50	55.0	223.1	168.1	0.28	1.12	3.62	0.38	41.6
MB-2011-51	83.7	220.8	137.1	0.35	0.73	2.23	0.53	28.5
MB-2011-52	134.5	145.0	10.5	0.19	0.06	0.53	0.07	3.8
and	159.5	231.7	72.2	0.26	2.33	5.61	0.71	77.7
MB-2011-53	170.0	204.8	34.8	0.47	0.20	0.59	0.13	13.2
MB-2011-54	156.2	201.0	44.8	0.17	1.55	4.26	0.70	60.7
MB-2011-55	102.0	149.2	47.2	0.99	0.39	0.79	0.37	16.0
and	153.2	155.7	2.47	0.67	0.05	0.09	0.09	5.3
MB-2011-56	15.2	43.0	27.8	0.22	0.35	1.08	0.29	17.0
MB-2011-57	143.3	231.0	87.7	0.14	2.77	7.23	0.61	103.3
MB-2011-58	23.0	72.0	49.0	0.45	0.31	2.02	0.38	23.2
and	98.1	105.0	6.9	1.09	0.05	0.17	0.13	6.9
MB-2011-59	24.50	88.0	63.5	0.47	0.26	1.03	0.21	19.5
MB-2011-60	21.5	54.0	32.5	0.89	0.09	0.44	0.08	6.2
MB-2011-61	80.2	178.0	98.8	.3	0.22	0.80	1.15	15.8
MB-2011-62	118.9	201.0	82.1	0.15	0.98	3.17	0.31	39.8
MB-2011-63	168.4	240.0	71.6	0.18	1.89	4.98	0.91	79.9

10.2 2012 Drilling – VMC

The objective of the 2012 drilling was to upgrade the inferred and indicated mineral resources to measured mineral resources, define additional near-surface mineral resources along the northwest margin of the Murray Brook deposit, and complete preliminary metallurgical testing on selected portions thereof. The drill program commenced in February 2012 and consisted of 99 vertical drill holes totalling 18,264 m. The drill hole specifications are presented in Table 10-3. From 2010 to 2012, 166 drill holes were completed for a total of 29,718 m.

An analysis of the drilling results identified two distinct north-trending massive sulphide zones with different mineralogical characteristics and thicknesses. The deeper, western zone appears to be thicker and richer in zinc, lead, and silver mineralization, whereas shallower, eastern zone is thinner and richer in copper-gold mineralization.

The location of VMC, historical and current drill collars are shown in Figure 10-1. The dimensions of the Murray Brook deposit are illustrated in Figure 10-2. The massive sulphide portion of the deposit measures approximately 320 m north-south by approximately 300 m east-west with a thickness of 120 to 150 m as two north-south lobes. Typical vertical cross-section projections, normal and perpendicular to the axes of the deposit, are presented in Figure 10-3 to Figure 10-5.

The objective of the 2012 Phase III diamond drilling was to upgrade the inferred and indicated mineral resources to measured mineral resources, and define additional near-surface mineral resources along the northwest margin of the deposit. In addition, three HQ size diamond drill holes—MB-2012-121, MB-2012-124, and MB-2012-132—were completed to yield an approximately 3-tonne sample for metallurgical testing.

The diamond drill specifications for the Phase III program are listed in Table 10-3, which commenced in February 2012 and ceased June 2012. A total of 99 NQ size vertical holes totalling 18,624 m were completed during this program, which was designed to fill in the gaps in the drill data and to better define the shape and size of the mineralized zone (ELN News Release: August 14, 2012). Significant intercepts for the Phase III drilling program are listed in Table 10-4.

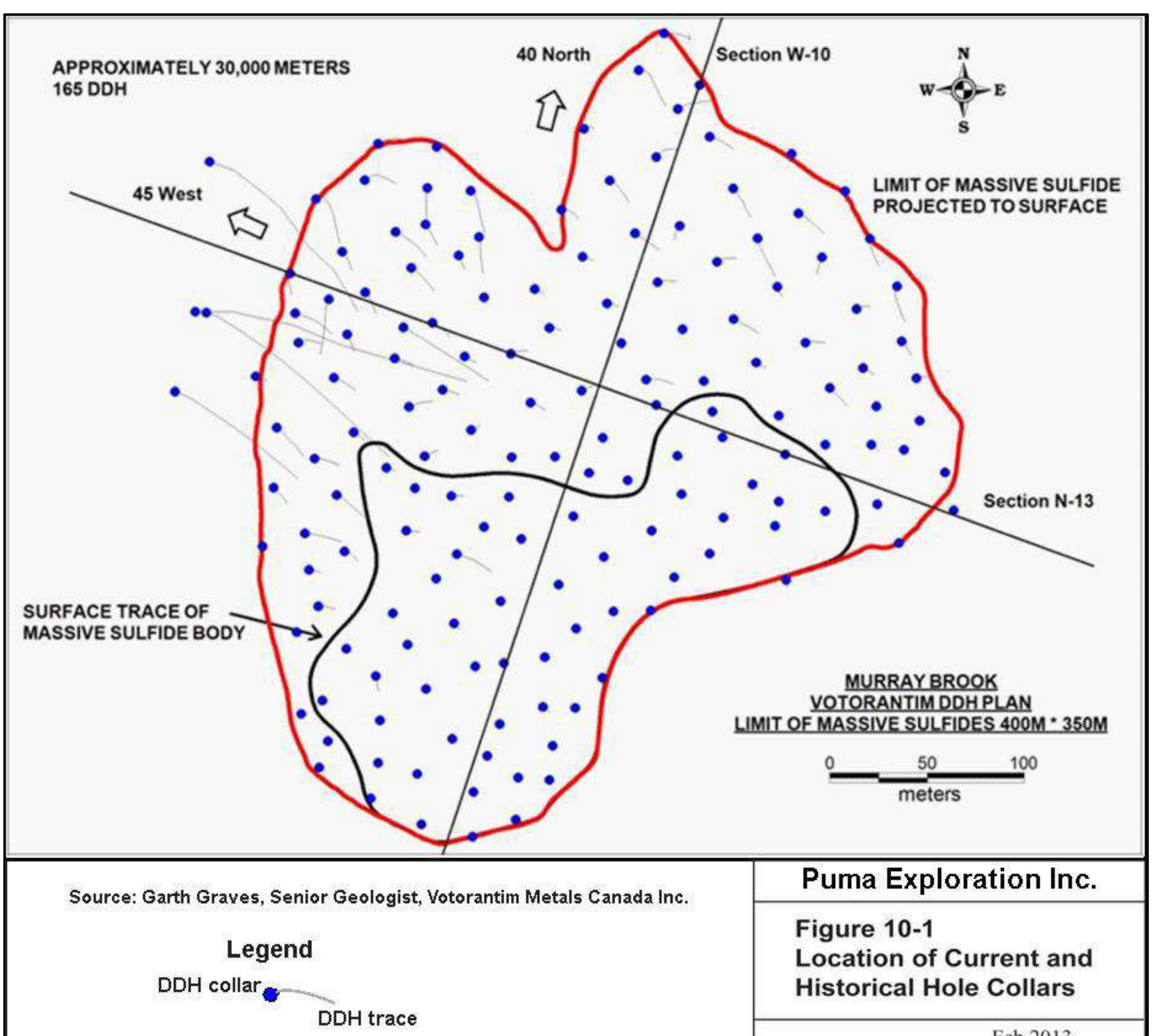
Table 10-3: Diamond Drill Specifications for Phase III Drill Program

Drill Hole ID	Easting (m)	Northing (m)	Elevation (masl)	Azimuth (°)	Inclination (°)	Length (m)
MB-2012-64	693,100.5	5,266,878	449.82	360	-90	275
MB-2012-65	693,355.0	5,266,785	470.78	360	-90	125
MB-2012-66	693,338.2	5,266,770	469.33	360	-90	104
MB-2012-67	693,323.5	5,266,753	470.01	360	-90	98
MB-2012-68	693,127.3	5,266,889	451.97	360	-90	275
MB-2012-69	693,316.3	5,266,734	469.12	360	-90	74
MB-2012-70	693,139.0	5,266,868	454.40	360	-90	281
MB-2012-71	693,299.6	5,266,784	468.80	360	-90	146
MB-2012-72	693,288.7	5,266,810	467.27	360	-90	176
MB-2012-73	693,224.2	5,266,812	462.77	360	-90	176
MB-2012-74	693,274.1	5,266,772	467.54	360	-90	125
MB-2012-75	693,213.9	5,266,837	459.59	360	-90	178
MB-2012-76	693,254.2	5,266,776	466.78	360	-90	125
MB-2012-77	693,187.2	5,266,887	455.01	360	-90	269
MB-2012-78	693,375.7	5,266,756	473.29	360	-90	77
MB-2012-79	693,145.7	5,266,945	443.49	360	-90	300
MB-2012-80	693,378.0	5,266,819	474.53	360	-90	161
MB-2012-81	693,399.5	5,266,790	476.30	360	-90	120
MB-2012-82	693,138.9	5,266,926	444.52	360	-90	302
MB-2012-83	693,401.9	5,266,810	476.32	360	-90	152
MB-2012-84	693,422.7	5,266,824	478.10	360	-90	152
MB-2012-85	693,113.7	5,266,916	444.47	360	-90	350
MB-2012-86	693,424.3	5,266,802	478.31	360	-90	137
MB-2012-87	693,437.3	5,266,776	479.84	360	-90	99
MB-2012-88	693313.1	5,266,823	468.78	360	-90	200
MB-2012-89	693,292.8	5,267,001	462.23	360	-90	308
MB-2012-90	693,154.6	5,266,899	448.85	360	-90	275
MB-2012-91	693,302.2	5,266,849	467.69	360	-90	215
MB-2012-92	693,175.8	5,266,943	446.92	360	-90	299
MB-2012-93	693,270.7	5,266,842	465.70	360	-90	188
MB-2012-94	693,289.5	5,266,873	463.79	360	-90	230
MB-2012-95	693,233.8	5,266,850	460.27	360	-90	200
MB-2012-96	693,316.4	5,266,948	465.51	360	-90	259
MB-2012-97	693,320.0	5,266,884	468.45	360	-90	242
MB-2012-98	693,226.2	5,266,870	457.84	360	-90	203
MB-2012-99	693,299.8	5,266,962	463.99	360	-90	269
MB-2012-100	693,250.8	5,266,886	459.09	360	-90	192
MB-2012-101	693,351.0	5,266,871	471.61	360	-90	202
MB-2012-102	693,164.6	5,266,768	474.18	360	-90	242
MB-2012-103	693,391.7	5,266,860	475.13	360	-90	191
MB-2012-104	693,288.8	5,266,938	464.29	0	-90	287
MB-2012-106	693,365.5	5,266,842	473.92	360	-90	200
MB-2012-107	693,214.4	5266,784	473.14	360	-90	179
MB-2012-108	693,251.6	5,266,952	458.64	360	-90	302
MB-2012-109	693,277.8	5,266,899	462.61	360	-90	233
MB-2012-110	693,154.3	5,266,834	461.57	360	-90	275
MB-2012-111	693,264.9	5,266,926	461.34	360	-90	251
MB-2012-112	693,328.4	5,266,922	466.24	360	-90	251
MB-2012-113	693,219.4	5,266,742	468.49	360	-90	152
MB-2012-114	693,200.2	5,266,748	469.99	360	-90	206
MB-2012-115	693,201.7	5,266,631	473.95	360	-90	107
MB-2012-116	693,340.8	5,266,896	470.95	360	-90	224
MB-2012-117	693,190.3	5,266,835	458.23	360	-90	245
MB-2012-118	693,231.5	5,266,681	468.93	360	-90	123
MB-2012-119	693,239.9	5,266,911	454.80	360	-90	260
MB-2012-120	693,235.4	5,266,636	469.17	360	-90	74
MB-2012-121	693,317.8	5,266,807	468.98	360	-90	179
MB-2012-122	693,217.8	5,266,619	473.36	360	-90	101
MB-2012-123	693,194.8	5,266,612	477.09	360	-90	86
MB-2012-124	693,175.6	5,266,721	467.39	360	-90	188
MB-2012-125	693,165.8	5,266,621	477.04	360	-90	101
MB-2012-126	693,184.9	5,266,698	468.49	360	-90	176
MB-2012-127	693,145.6	5,266,627	478.64	360	-90	92
MB-2012-128	693,119.9	5,266,638	482.33	360	-90	101
MB-2012-129	693,144.6	5,266,671	473.28	360	-90	152
MB-2012-130	693,117.0	5,266,659	481.79	360	-90	125
MB-2012-131	693,362.1	5,266,909	467.89	360	-90	227
MB-2012-132	693,178.9	5,266,818	466.11	360	-90	227
MB-2012-133	693,103.0	5,266,858	452.17	360	-90	251

Drill Hole ID	Easting (m)	Northing (m)	Elevation (masl)	Azimuth (°)	Inclination (°)	Length (m)
MB-2012-134	693,373.9	5,266,886	471.52	360	-90	224
MB-2012-135	693,093.5	5,266,799	472.15	360	-90	258
MB-2012-136	693,358.5	5,266,939	467.31	360	-90	251
MB-2012-137	693,107.9	5,266,745	484.63	360	-90	224
MB-2012-138	693,129.9	5,266,847	455.72	360	-90	269
MB-2012-139	693,110.1	5,266,726	486.42	360	-90	200
MB-2012-140	693,114.9	5,266,707	483.92	360	-90	185
MB-2012-141	693,104.7	5,266,842	458.367	360	-90	245
MB-2012-142	693,103.8	5,266,694	485.23	360	-90	152
MB-2012-143	693,082.6	5,266,825	460.87	360	-90	224
MB-2012-144	693,412.6	5,266,871	475.41	360	-90	200
MB-2012-145	693,398.7	5,266,896	472.16	360	-90	200
MB-2012-146	693,086.2	5,266,738	488.97	360	-90	176
MB-2012-147	693,385.8	5,266,920	470.42	360	-90	200
MB-2012-148	693,091.9	5,266,768	479.47	360	-90	230
MB-2012-149	693,415.1	5,266,843	477.21	360	-90	167
MB-2012-150	693,141.8	5,266,609	483.86	360	-90	74
MB-2012-151	693,279.7	5,266,982	460.75	360	-90	290
MB-2012-152	693,247.1	5,266,655	468.81	360	-90	74
MB-2012-153	693,286.0	5,266,705	470.14	360	-90	74
MB-2012-154	693,233.8	5,266,618	472.96	360	-90	75
MB-2012-155	693,216.6	5,266,598	474.17	360	-90	74
MB-2012-156	693,194.2	5,266,589	477.11	360	-90	74
MB-2012-157	693,115.3	5,266,624	483.82	360	-90	74
MB-2012-158	693,106.2	5,266,652	484.12	360	-90	101
MB-2012-159	693,441.7	5,266,757	481.44	360	-90	77
MB-2012-160	693,355.6	5,266,721	477.45	360	-90	74
MB-2012-161	693,171.1	5,266,922	449.35	360	-90	302
MB-2012-162	693,311.0	5,266,975	463.92	360	-90	287

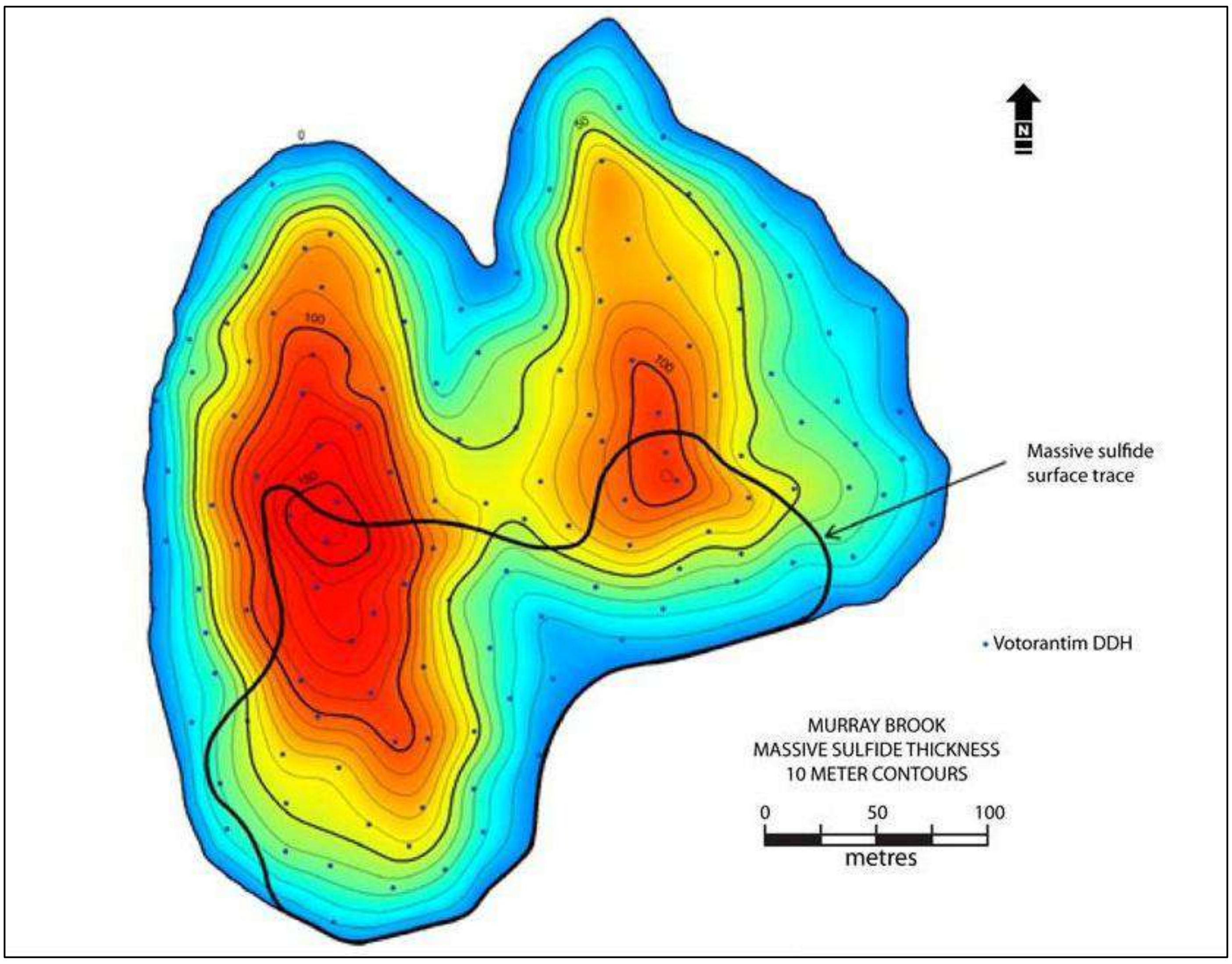
Notes: 1. Coordinates are in UTM NAD83 Zone 19. 2. Drill hole MB-2012-105 was abandoned.

Figure 10-1: Location of Current and Historical Drill Hole Collars



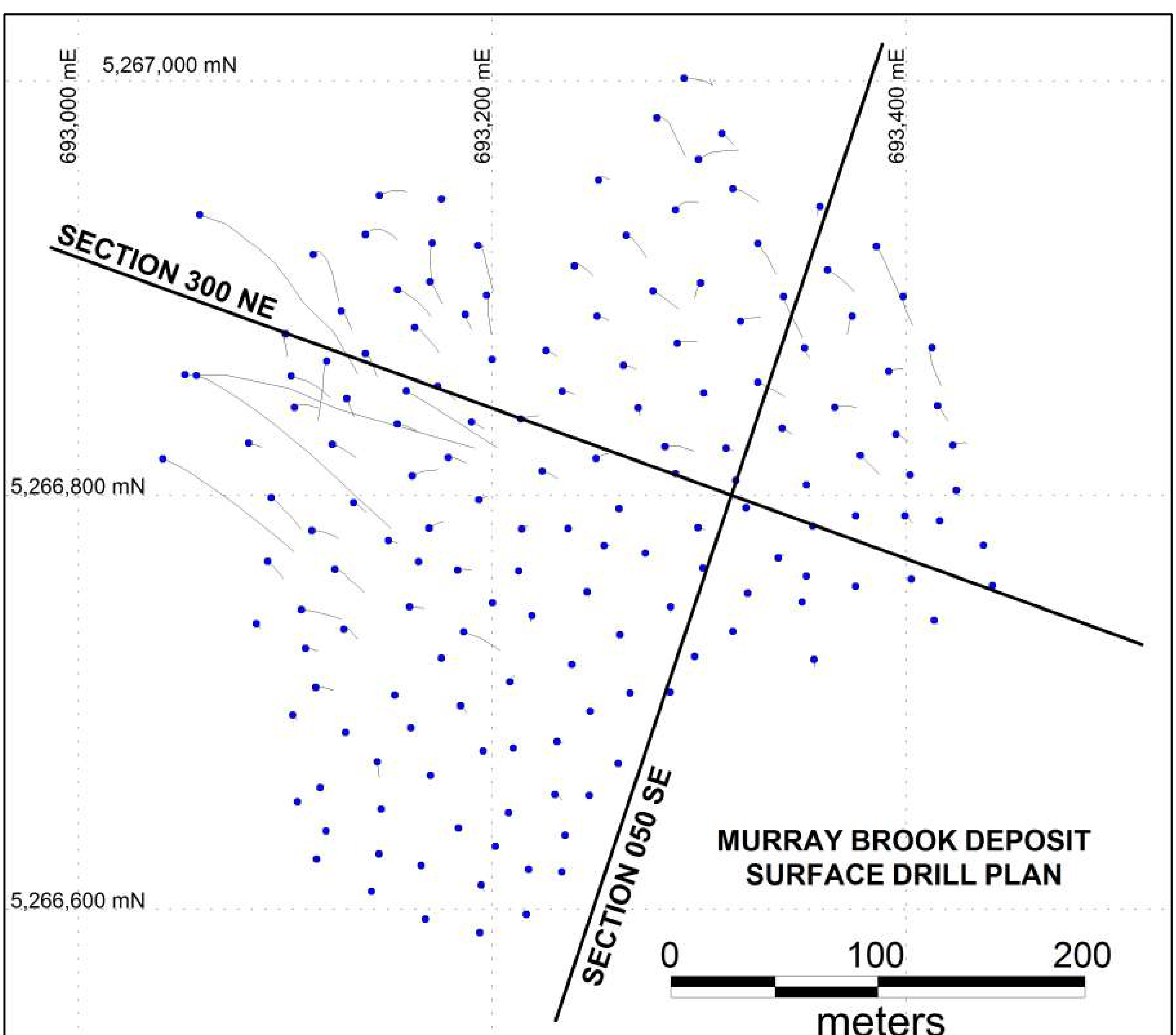
Source: P&E (2017).

Figure 10-2: Deposit Thickness Variations



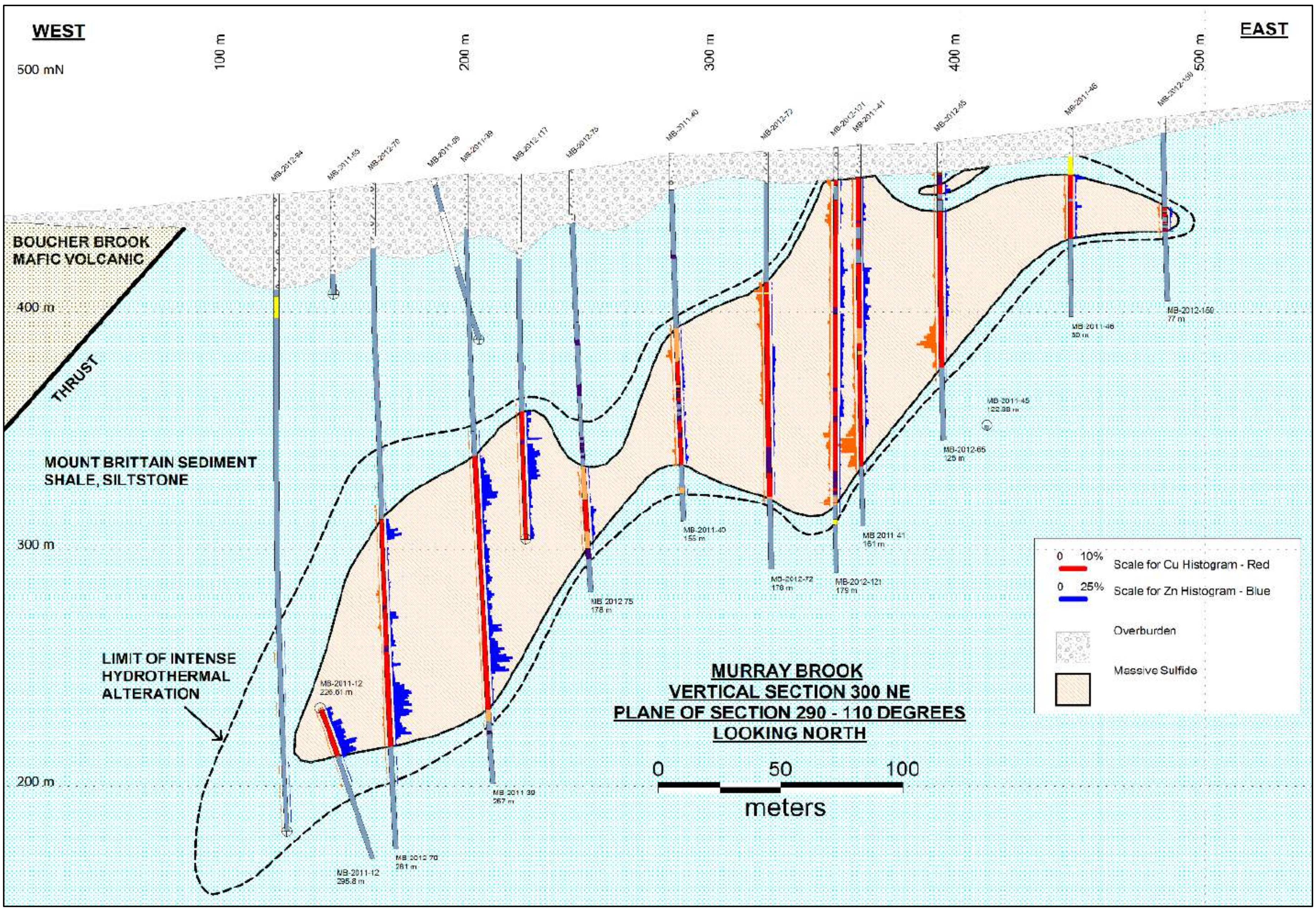
Source: P&E (2017).

Figure 10-3: Orientation of Murray Brook Vertical Cross-section Projections 300NE and 050SE



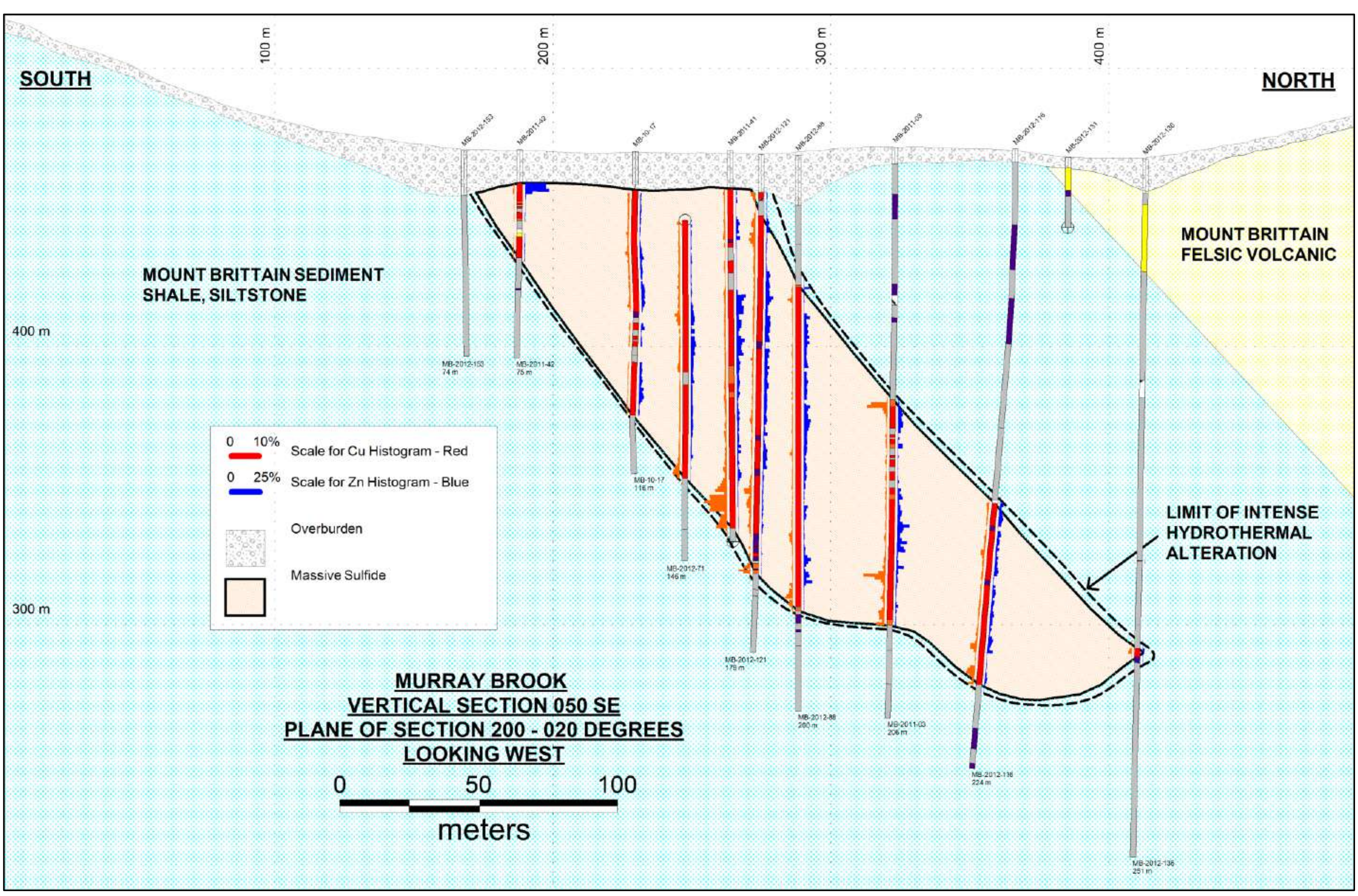
Source: P&E (2017).

Figure 10-4: Typical Vertical Cross-section Projection 300 NE, Murray Brook Property



Source: P&E (2017).

Figure 10-5: Typical Vertical Cross-section Projection 050SE, Murray Brook Property



Source: Puma (2017)

Table 10-4: Significant Phase III Drill Intercepts (2012)

Drill Hole ID	From (m)	To (m)	Width (m)	Cu (%)	Pb (%)	Zn (%)	Au (g/t)	Ag (g/t)
MB-2012-65	17.5	23.9	6.4	0.09	0.89	2.75	0.56	31.4
and	43.0	47.0	4.0	0.29	1.10	2.05	1.01	47.4
and	51.0	54.0	3.0	0.28	1.75	2.92	1.10	62.7
and	57.0	73.0	16.0	0.11	1.09	2.79	0.50	41.8
MB-2012-66	32.0	48.5	16.5	0.37	1.50	3.03	1.41	65.2
and	51.0	56.0	11.0	0.17	1.69	3.40	1.22	65.4
MB-2012-67	15.0	41.0	26.0	0.32	1.41	3.35	1.07	62.0
and	77.65	83.0	5.35	0.11	1.26	2.92	0.60	61.8
MB-2012-68	207.22	216.0	8.78	0.06	2.24	7.29	0.16	62.5
and	220.0	236.0	16.0	0.10	2.34	6.83	0.64	85.3
MB-2012-69	17.0	19.15	2.15	0.31	1.03	2.13	1.63	53.5
MB-2012-70	141.75	150.40	8.65	0.16	3.67	6.43	0.60	85.5
and	181.0	184.0	3.0	0.22	2.82	6.60	0.31	69.4
and	191.0	195.75	4.75	0.05	0.86	3.08	0.22	24.2
and	201.0	211.0	10.0	0.09	1.59	4.07	0.61	54.7
and	211.0	235.0	24.0	0.49	4.55	11.58	1.53	147.8
MB-2012-71	56.00	61.25	5.25	1.18	1.29	2.59	0.79	50.3
and	65.3	76.0	10.7	0.41	1.57	2.80	0.97	56.1
MB-2012-72	82.0	93.0	11.0	0.43	1.24	2.59	0.92	51.2
MB-2012-74	55.0	62.0	7.0	0.96	1.42	2.52	0.82	58.0
MB-2012-75	142.0	154.0	12.0	0.17	0.93	2.58	0.64	36.0
MB-2012-76	32.1	80.80	48.7	0.49	0.63	1.53	0.49	22.1
MB-2012-77	166.0	226.0	60.0	0.25	1.61	4.43	1.24	69.8
MB-2012-78	12.0	28.0	16.0	0.44	0.83	1.89	1.27	38.1
MB-2012-80	70.40	76.2	5.8	0.42	1.16	2.75	1.13	60.5
and	90.6	120.35	29.75	0.32	0.98	2.02	1.14	47.3
MB 2012-81	38.6	42.3	3.7	0.27	1.90	4029	1.06	71.5
and	48.35	85.10	36.75	0.50	1.16	2.90	0.71	56.6
MB-2012-82	249.15	265.0	15.85	0.13	1.96	5.02	0.82	99.9
MB-2012-83	67.65	76.0	8.35	0.46	1.11	2.46	1.09	53.2
and	90.95	93.6	2.65	0.37	1.14	2.48	1.60	49.4
and	97.70	116.45	18.75	0.41	0.82	2.01	0.87	42.1
MB-2012-84	100.45	103.3	2.85	0.66	1.35	2.85	1.35	56.9
MB-2012-86	82.0	105.2	23.2	0.28	1.15	2.46	1.19	51.0
MB-2012-87	64.4	67.25	2.85	0.27	1.42	3.41	0.49	90.1
MB-2012-88	47.5	162.45	114.95	0.39	0.96	2.06	0.74	38.2
MB-2012-89	227.95	229.90	1.95	0.90	0.94	1.91	1.15	45.8
MB-2012-90	151.16	198.28	47.12	0.12	2.97	7.94	0.68	114.6
and	205.64	240.0	34.36	0.34	3.41	7.02	2.0	1.07
MB-2012-91	73.65	184.65	111.0	0.68	0.73	1.53	0.72	35.71
MB-2012-92	208.06	210.60	2.54	0.10	1.16	2.86	0.51	39.29
MB-2012-93	86.6	141.05	54.45	0.96	0.47	0.84	0.48	20.44
MB-2012-94	118.0	196.1	78.1	0.52	0.86	1.66	1.04	36.64
MB-2012-95	153.45	156.4	2.95	0.09	0.65	2.27	0.34	19.45
MB-2012-96	189.9	228.3	38.4	0.52	1.03	2.12	1.14	48.82
MB-2012-97	126.0	201.0	75.0	0.46	1.13	2.66	1.05	56.84
MB-2012-98	167.0	174.05	7.05	0.29	1.55	4.66	1.45	57.38
MB-2012-99	195.9	240.5	44.6	0.93	0.90	1.92	1.73	44.95
MB-2012-101	118.0	171.72	53.72	0.49	1.07	2.36	0.95	50.36
MB-2012-102	68.0	118.2	50.2	0.83	1.05	4.15	0.32	42.5
Incl.	83.0	104.0	21.0	1.07	1.56	6.12	0.27	60.8
and	122.5	174.00	51.5	0.09	0.89	3.39	0.20	34.7
MB-2012-103	133.1	150.3	17.2	0.79	1.06	2.24	1.38	47.8
and	159.0	162.25	3.25	0.32	1.25	2.11	1.31	30.20
MB-2012-104	165.0	182.0	17.0	1.37	0.65	1.32	0.58	33.7
and	183.0	241.0	58.0	0.44	1.06	1.89	1.38	49.3
and	246.51	254.57	8.06	2.52	0.10	0.18	0.38	17.3
MB-2012-106	93.95	104.45	10.5	0.33	1.57	3.86	1.30	74.2
and	129.0	149.5	20.5	0.32	1.23	2.99	1.11	54.5
MB-2012-107	54.80	112.0	57.2	0.15	1.82	5.89	0.32	79.9
incl.	62.0	93.0	31.0	0.18	2.58	9.23	0.34	108.7
MB-2012-109	128.0	202.95	74.95	1.29	0.27	0.67	0.67	22.3
MB-2012-110	108.0	233.0	125.0	0.26	1.27	4.56	0.60	47.1
incl.	108.0	145.0	37.0	0.14	1.64	7.92	0.24	61.88
MB-2012-111	145.30	162.5	17.2	0.91	0.13	0.38	0.25	16.1
MB-2012-112	167.2	220.15	52.95	0.49	0.87	1.96	0.85	45.7
MB-2012-113	53.0	94.25	41.25	0.61	0.26	1.24	0.36	16.4
MB-2012-114	56.0	135.5	79.5	0.53	0.98	3.45	0.32	46.5
incl.	98.0	126.0	28.0	0.18	2.48	7.59	0.56	102.2
and	153.6	169.0	15.5	0.10	1.63	4.12	0.53	51.9

Drill Hole ID	From (m)	To (m)	Width (m)	Cu (%)	Pb (%)	Zn (%)	Au (g/t)	Ag (g/t)
incl.	153.6	163.0	9.4	0.12	2.30	5.41	0.75	71.0
MB-2012-115	23.0	75.0	52.0	0.59	0.24	1.55	0.21	22.4
MB-2012-116	127.0	193.6	66.6	0.35	0.81	1.63	0.91	42.8
MB-2012-117	100.4	185.0	84.6	0.15	1.82	4.62	0.52	69.8
incl.	170.0	183.0	13.0	0.29	4.11	10.34	1.39	126.0
MB-2012-118	40.0	59.0	19.0	4.10	0.03	0.12	0.12	12.0
MB-2012-120	14.0	29.0	15.0	3.40	0.08	0.51	0.12	18.5
MB-2012-121	24.0	113.1	89.1	0.43	1.12	2.42	1.14	55.5
incl.	42.0	58.0	16.0	0.33	1.19	3.13	2.21	66.0
MB-2012-122	17.0	29.3	12.3	2.60	0.24	1.81	0.12	24.9
incl.	14.4	22.0	7.6	4.79	0.10	0.62	0.20	35.6
and	34.7	50.4	15.7	1.92	0.06	0.18	0.02	9.4
MB-2012-123	18.0	54.9	36.9	0.80	0.13	1.22	0.21	19.8
MB-2012-124	29.0	110.0	81.0	0.23	1.35	4.27	0.24	54.0
incl.	29.0	38.0	10.0	0.80	2.55	7.39	0.28	114.8
incl.	56.0	67.0	11.0	0.03	2.00	5.33	0.13	61.6
incl.	84.9	92.0	7.1	0.07	1.10	5.41	0.28	46.6
and	128.0	137.0	9.0	0.18	1.08	4.22	0.33	56.7
MB-2012-125	22.0	60.3	38.3	0.54	0.03	0.10	0.17	10.4
MB-2012-126	22.5	94.0	71.5	0.55	0.89	3.65	0.38	38.5
incl.	22.5	34.0	11.5	2.41	2.13	7.10	0.44	87.6
incl.	39.0	74.0	35.0	0.18	0.75	3.60	0.49	32.2
MB-2012-127	26.0	62.3	36.3	0.62	0.36	1.18	0.85	21.5
MB-2012-128	27.0	49.4	22.4	1.05	0.30	0.82	0.13	20.3
MB-2012-129	71.0	77.0	6.0	0.78	0.80	2.67	0.59	28.9
MB-2012-130	46.0	50.6	4.6	1.12	0.06	0.34	0.08	8.7
and	52.7	67.7	15.0	1.19	0.14	0.56	0.20	18.1
MB-2012-131	152.0	173.0	21.0	0.73	83.0	1.64	1.12	33.6
MB-2012-132	89.0	201.0	112.0	0.10	1.92	6.15	0.64	70.6
incl.	167.0	200.0	33.0	0.10	3.60	10.50	1.37	126.7
MB-2012-133	183.0	210.2	27.2	0.93	0.03	0.08	0.07	8.9
incl.	191.0	210.2	19.2	1.10	0.02	0.10	0.10	7.8
MB-2012-134	139.0	157.4	18.4	0.41	1.54	3.21	1.16	61.9
MB-2012-135	144.8	147.0	2.2	0.42	0.14	0.54	0.03	7.5
MB-2012-136	nsv							
MB-2012-137	nsv							
MB-2012-138	197.6	225.0	45.4	0.18	4.58	8.49	0.59	152.2
incl.	214.0	225.0	11.0	0.2	9.6	13.7	1.10	269.7
MB 2012-139	88.0	114.0	26.0	0.47	0.36	2.06	0.28	21.4
MB 2012-140	64.6	91.6	27.0	0.60	0.64	2.73	0.75	36.7
incl.	67.0	77.0	10.0	0.10	1.10	4.90	0.18	50.5
MB-2012-141	178.3	180.7	2.4	0.15	1.93	6.27	0.47	99.3
MB-2012-142	53.0	56.9	3.9	0.16	3.69	8.89	0.58	86.4
MB-2012-143	nsv							
MB-2012-144	147.4	159.0	11.6	0.65	1.78	2.82	1.57	63.9
MB-2012-145	nsv							
MB-2012-146	nsv							
MB-2012-147	nsv							
MB-2012-148	nsv							
MB-2012-149	123.3	133.6	10.3	0.51	1.18	2. 30	1.55	45.9
MB-2012-150	26.0	30.15	4.15	4.18	0.21	0.40	0.42	24.3
MB-2012-151	200.8	225.1	24.3	1.4	0.38	1.13	0.61	27.4
and	244.8	248.6	3.8	4.65	0.19	0.41	0.72	30.3
MB-2012-152	21.25	22.75	1.5	0.09	0.67	3.38	0.04	9.00
MB-2012-153	nsv							
MB-2012-154	nsv							
MB-2012-155	14.7	30.05	15.35	1.91	0.05	0.18	0.04	8.3
MB-2012-156	20.25	30.0	9.75	5.26	0.03	0.07	0.18	9.9
MB-2012-157	18.0	27.7	9.7	3.94	0.38	1.82	0.44	45.8
MB-2012-158	24.8	53.0	28.2	1.27	0.22	0.82	0.17	22.7
MB-2012-159	37.45	48.0	10.55	0.36	0.67	1.52	0.75	26.8
MB-2012-160	nsv							
MB-2012-162	219.1	238.15	19.05	1.94	1.04	1.48	2.16	47.0

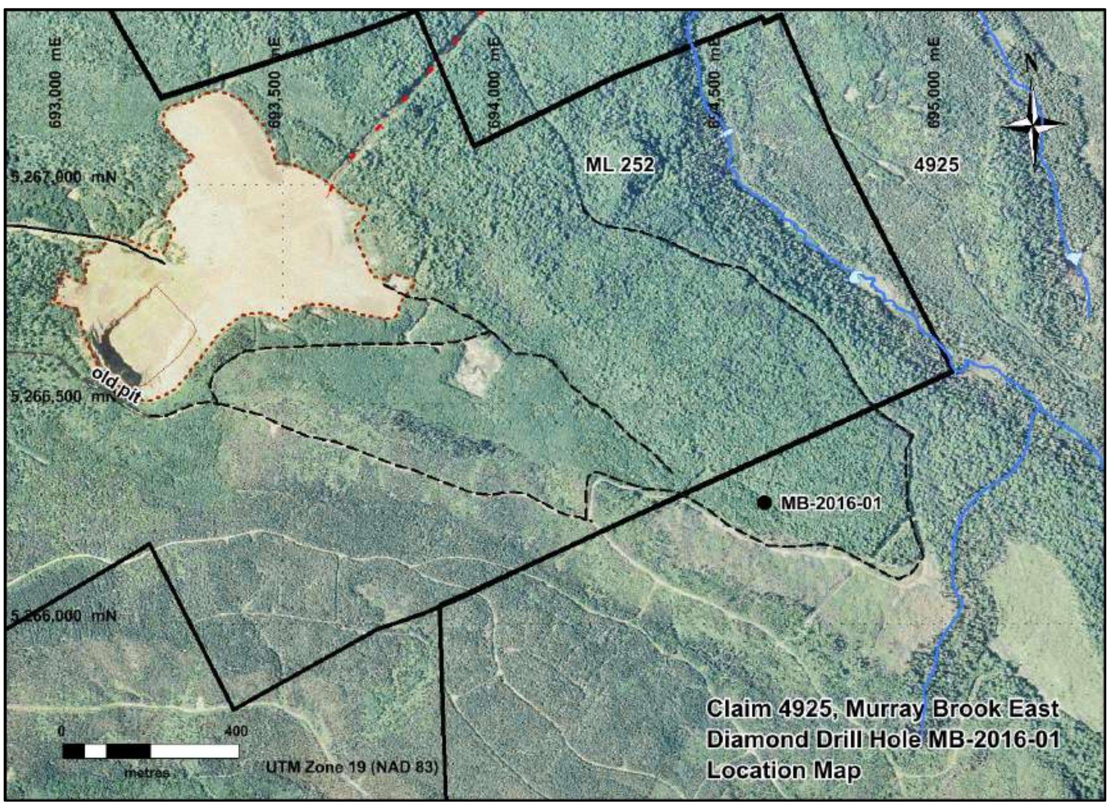
Note: nsv = no significant values.

10.3 2016 Exploration Drilling – VMC

During June 2016, a diamond drilling program consisting of a single drill hole was completed on Claim Block 4925 immediately east of Mining Lease 252. The drilling was carried out by Maritime Diamond Drilling based in Truro, NS, and supervised by geological consultant Garth Graves. GeoXplore Surveys Inc. of Bathurst, New Brunswick, prepared the site and cleared some access roads. A handheld GPS with an implied accuracy of ±4 m was used to determine the field location for the drill hole. A water source for the drilling operation was located 100 m east of the drill hole location.

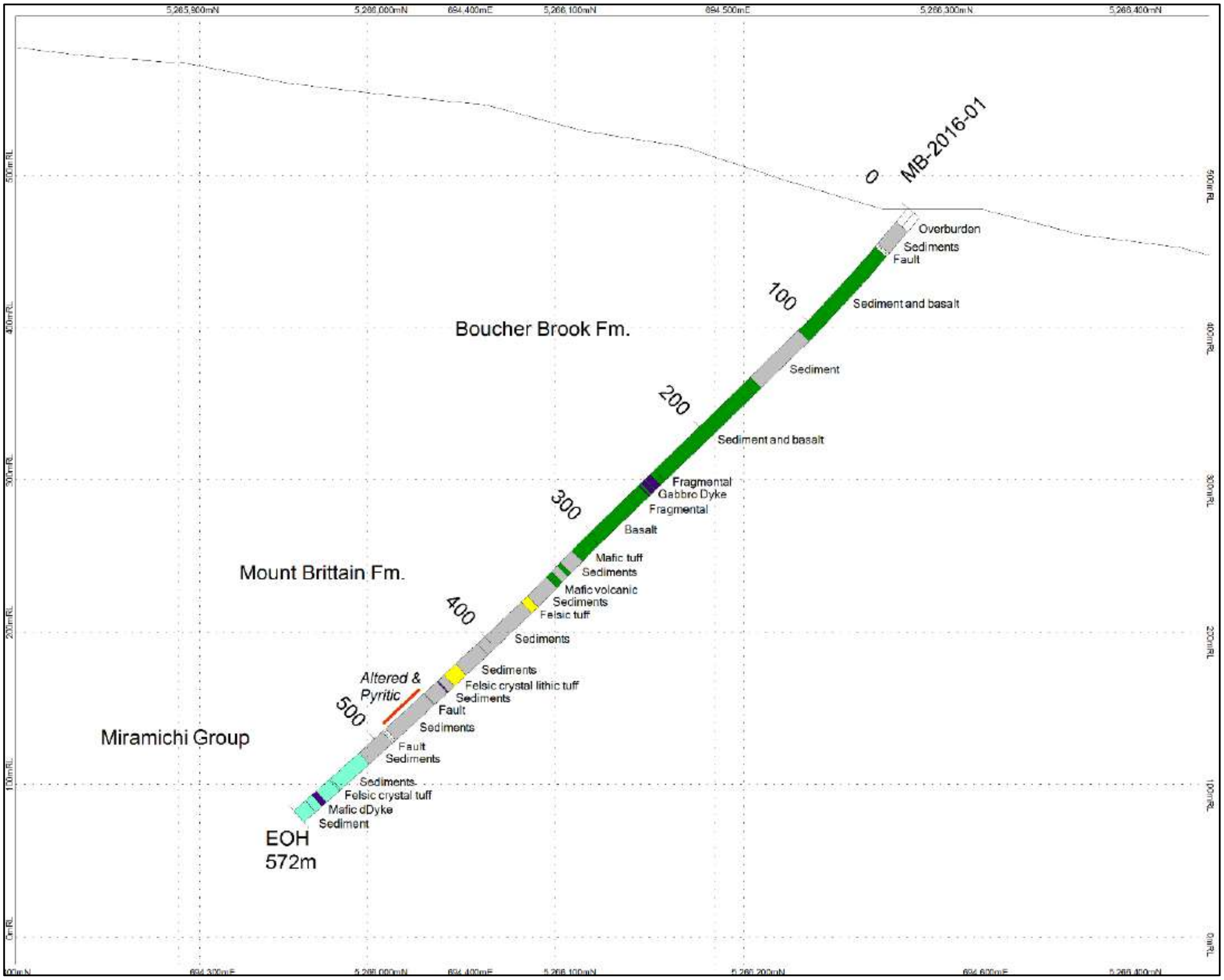
Diamond drill hole MB-2016-01 is located in an area of strong relief approximately 1,500 m east of the Murray Brook mine open pit (Figure 10-6). The drill hole is located at UTM coordinates 694,596 m E and 5,266,276 m N (NAD 83 Zone 19N) at 477.9 masl. The drill hole was completed at an azimuth 220° and dip of -50° to a final depth of 572 m. A downhole survey took readings for azimuth, dip, and magnetic susceptibility every 50 m. The drill hole cut basalt and associated sedimentary rocks of the Boucher Brook Formation from the top of the hole to a depth of 361 m. A fault zone at 30 m to 34 m downhole is probably part of the southeast-trending thrust faults mapped in this area. A vertical cross-section is presented in Figure 10-7.

Figure 10-6: Drill Hole Location for MB-2016-01



Source: VMC (2016).

Figure 10-7: Drill Hole Vertical Cross-section for MB-2016-01



Source: VMC (2016).

Fifteen drill core samples collected from the hydrothermal alteration zone in drill hole MB-2016-01 were sent for analysis. The samples were at 1 m intervals from the alteration zone and were sawed and stored in plastic sample bags. The samples were sent in two 5-gallon plastic buckets by Day and Ross courier to TSL Laboratories in Saskatoon, Saskatchewan, for analysis. A hydrothermally altered zone with pyrite and trace base metals was intersected in drill hole MB-2016-01 and is hosted by the Charlotte Brook Member sedimentary rocks of the Mount Brittain Formation. This alteration zone is the stratigraphic equivalent of the Murray Brook deposit. The 2016 drilling did not intersect significant mineralization. Assay results for selected elements are presented in Table 10-5.

Table 10-5: Assay Results for Drill Hole MB-2016-01

Sample ID	From (m)	To (m)	Ag (ppm)	As (ppm)	Ba (ppm)	Cu (ppm)	Fe (%)	Mn (ppm)	Mo (ppm)	Pb (ppm)	Zn (ppm)
577245	468.9	470	0.5	28	70	61.2	4.35	421	1.3	244.2	308
577246	470	471	<0.1	21	578	26.1	5.28	1023	2.3	18.4	96
577247	472	473	0.1	23	452	35.1	4.43	1027	1.4	93.5	95
577248	473	474	0.1	19	206	26.2	4.48	867	1.3	135.8	178
577249	479	480	0.1	39	109	20.9	4.32	546	1.5	67.5	75
577250	480	481	<0.1	14	746	28.1	4.92	938	0.9	20.0	102
577251	481	482	<0.1	15	243	14.8	3.38	460	0.9	36.1	101
577252	482	483	0.1	16	65	13.2	3.02	396	0.9	222.0	342
577253	483	484	<0.1	20	227	11.9	2.61	468	1.1	43.6	83
577254	484	485	<0.1	9	713	12.4	2.45	381	0.9	33.5	79
577255	485	486	<0.1	6	671	20.9	2.33	404	1.1	204.0	403
577256	486	487	<0.1	25	111	16.4	3.78	722	1.5	90.7	134
577257	487	488	<0.1	24	38	33.7	7.50	682	1.8	221.7	247
577258	488	489	<0.1	42	35	21.6	7.03	820	1.5	79.1	132
577259	489	490	0.3	46	54	42.3	7.01	443	1.9	245.1	339

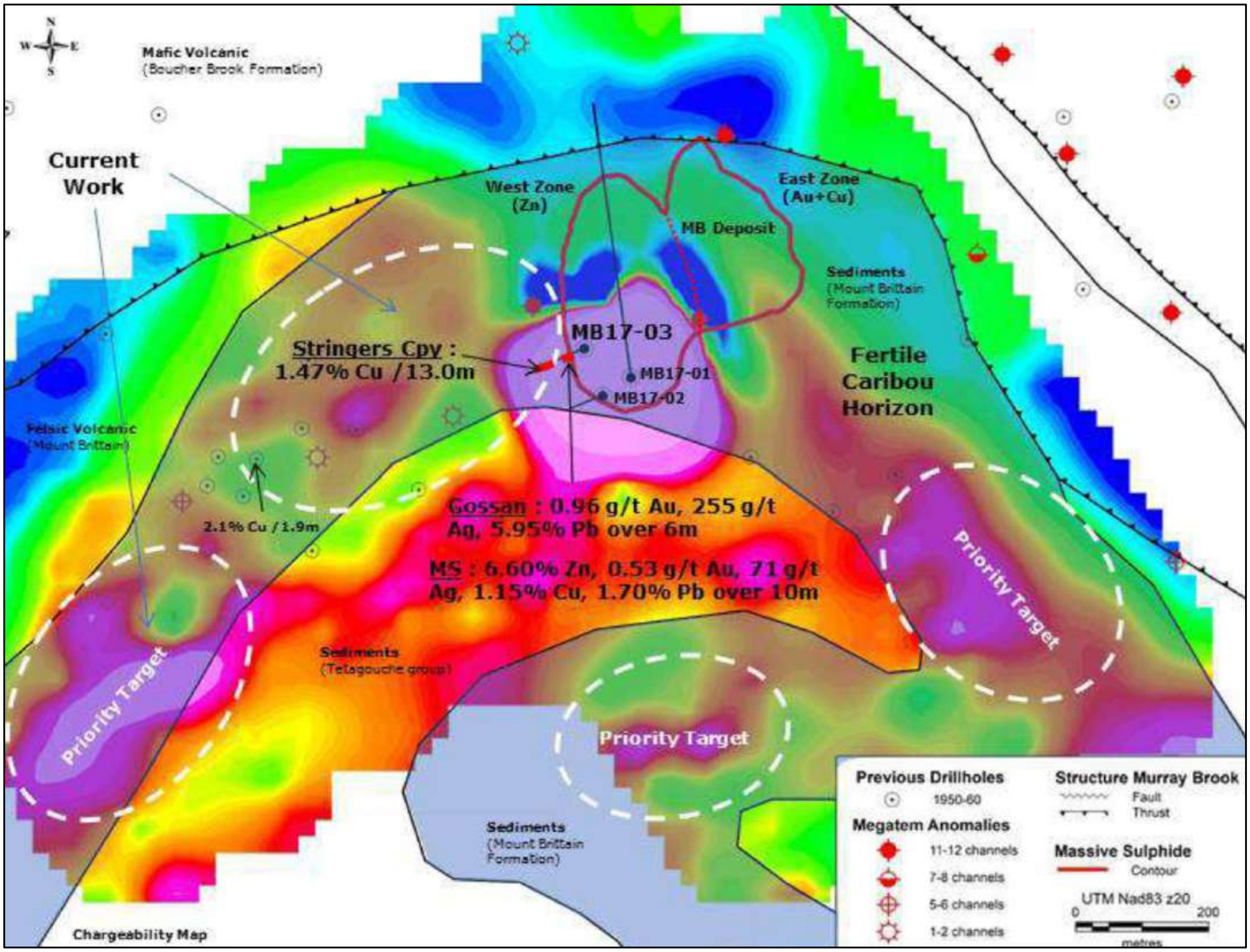
10.4 2017 Drilling – Puma

In 2017, Puma completed three diamond drill holes at Murray Brook totalling 1,025 m. Drill hole locations and results are shown in Figures 10-8 and 10-9 and Table 10-6. Selected assay results are listed in Table 10-7.

The 2017 drill holes included one deep hole (MB17-01) and two shallow drill holes (MB17-02 and MB17-03). Drill hole MB17-01 was designed to verify the extensions at depth of the Murray Brook deposit. That drill hole successfully intersected 405 m of massive sulphide mineralization grading 3.3% Zn, 1.1% Pb, 0.95 g/t Au, 42 g/t Ag, and 0.30% Cu.

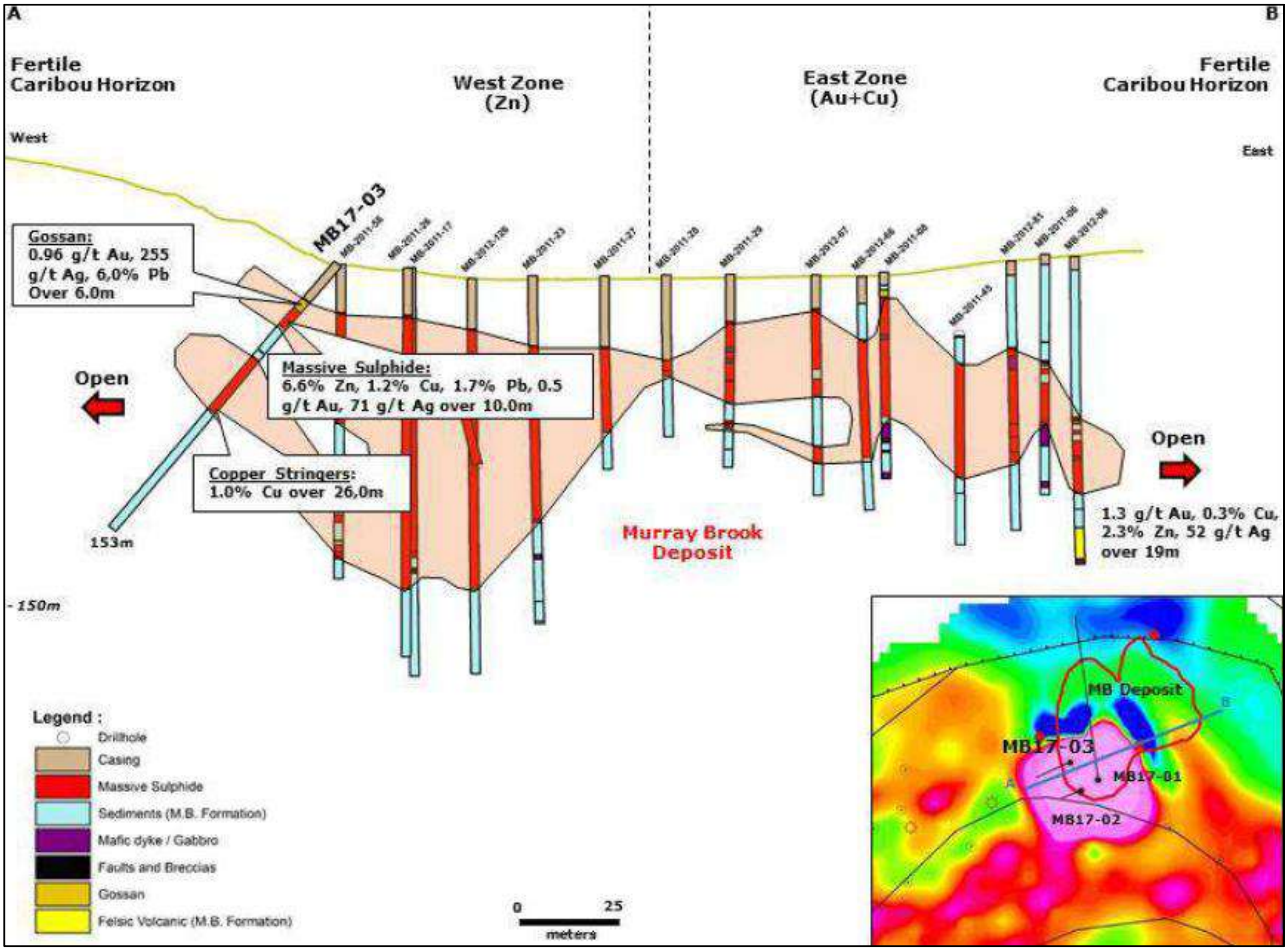
The two shallow drill holes, MB17-02 and MB17-03, followed to confirm the continuity of the fertile Caribou Horizon near the Murray Brook deposit. Drill hole MB17-02 was collared in the south part of the West Zone (zinc) and drilled at a declination of -45° for a total length of 72 m (Figure 10-8). A major, 8-meter-wide fault was intercepted and consisted mainly of quartz veins. The sedimentary rocks in this drill hole were less altered, with a few interbedded intervals of felsic volcanic rocks. The best mineralized intersection was 2.7 m grading 0.42 g/t Au, 0.96% Pb+Zn in the altered footwall sedimentary rocks. However, drill hole MB17-02 confirmed previous exploration results and directs future exploration programs to the more favourable Caribou Horizon located directly west of that drill hole.

Figure 10-8: 2017 Drilling Plan View



Source: Puma press release dated November 9, 2017.

Figure 10-9: 2017 Cross-sectional Projection of Drill Hole MB2017-03



Source: Puma press release dated November 9, 2017.

Table 10-6: 2017 Drill Hole Collar Information

Drill Hole ID	Easting (m)	Northing (m)	Elevation (masl)	Length (m)	Azimuth (deg)	Dip (deg)
MB17-01	693,193	5,266,628	475	800.00	354.0	-45.0
MB17-02	693,142	5,266,608	484	72.00	245.0	-45.0
MB17-03	693,129	5,266,682	475	153.00	245.0	-50.0
Total				**1,025.00**		

Table 10-7: Significant Intercepts in 2017 Drilling

Drill Hole ID	From (m)	To (m)	Length (m)	Au (g/t)	Ag (g/t)	Cu (%)	Pb (%)	Zn (%)	Comment
MB17-01	27	432	405	0.95	41	0.34	1.14	3.30	Continuous massive sulphide
incl.	27	39	12	0.31	28	2.15	1.02	4.54	Extends Cu+Zn zone by 50 m
incl.	65	77	12	0.20	97	0.38	3.10	5.57	Fills 73 m gap between 2 holes
incl.	134	160	26	0.34	36	0.08	1.05	5.14	Fills 70 m gap between two holes
incl.	174	180	6	0.03	50	0.02	2.09	5.12	
incl.	253	306	53	1.05	77	0.06	2.83	5.83	
incl.	341	400	59	2.62	72	0.68	1.52	5.26	Below mineral resources
MB17-02*	23.3	26	2.7	0.42	8	0.16	0.28	0.68	Disseminated
MB17-03*	21	27	6	0.96	255	-	5.95	-	Gossan
and	27	37	10	0.53	71	1.15	1.70	6.60	Massive sulphides
and	56	82	26	0.05	12	1.02	0.15	0.52	Stockwork
incl.	56	69	13	0.05	15	1.47	0.2	0.62	Stockwork

Note: *True thickness estimated to be approximately 70 to 90% of the reported intervals.

Drill hole MB17-03, completed at a declination of -50° to a total depth of 153 m, was collared at the western boundary of the West Zone (zinc) of the Murray Brook deposit (Figure 10-8). A new zone of copper stringers was intersected near surface that graded 1.02% Cu over 26 m, including a higher-grade intersection of 1.47% Cu over 13 m (Figure 10-9). This new zone opens the western side of the Murray Brook deposit along the favourable and fertile Caribou Horizon, which shows over a length of 1 km high priority targets (refer to Figure 10-8). Structural constraints can be outlined and future exploration work oriented from the from results from drill hole MB17-02.

10.5 2018 Drilling – Puma

Exploration and metallurgical drilling at the Murray Brook deposit and exploration drilling on Murray Brook east claim 4925 were carried out in 2018. In total, 16 drill holes were completed totalling 5,392 m (Table 10-8).

Table 10-8: 2018 Drill Hole Collar Information

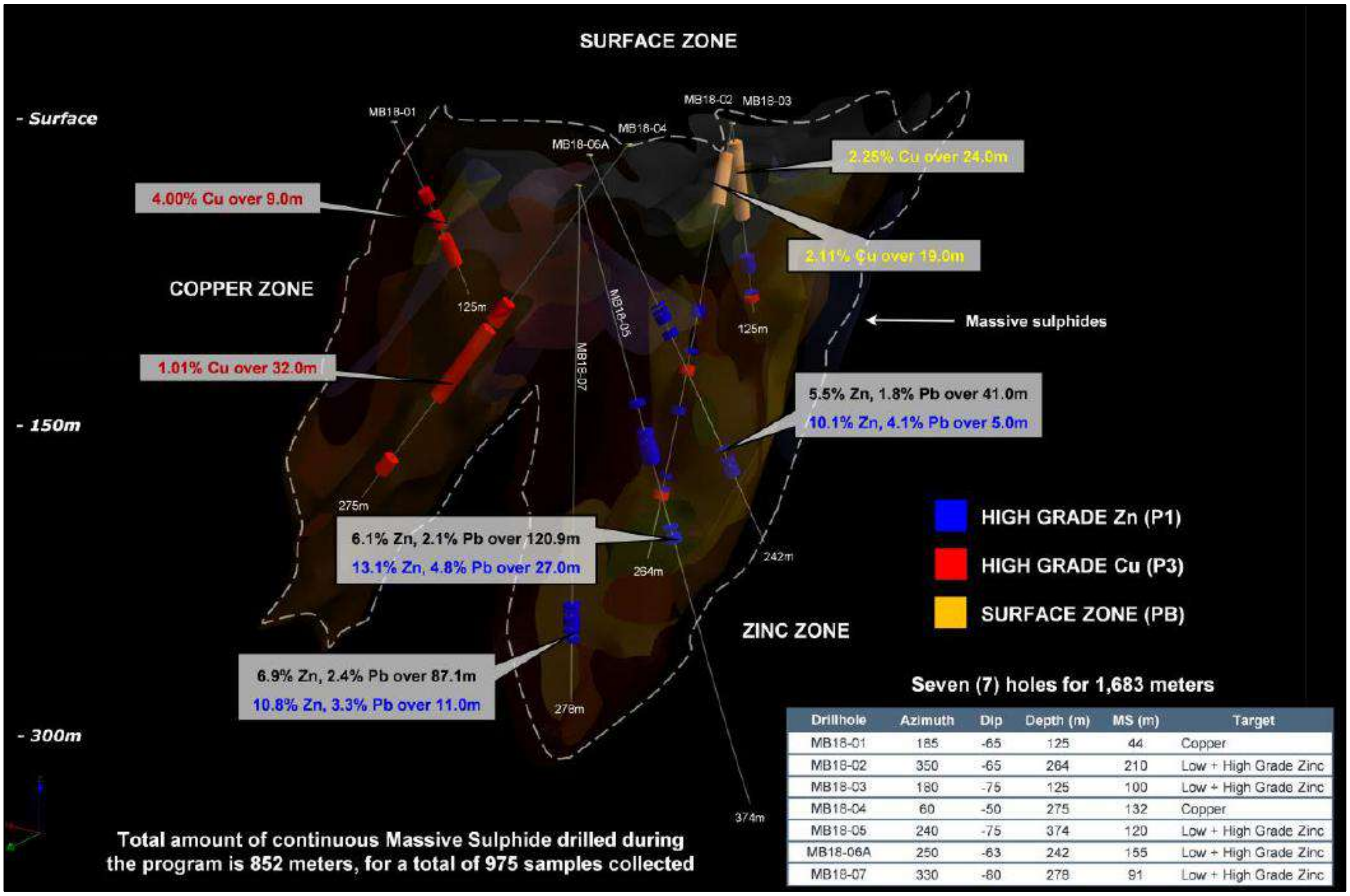
Drill Hole ID	Easting (m)	Northing (m)	Elevation (masl)	Length (m)	Azimuth (deg)	Dip (deg)
MB18-01	693,363	5,266,826	474	125.00	185.0	-65.0
MB18-02	693,192	5,266,696	473	264.00	350.0	-65.0
MB18-03	693,192	5,266,696	473	125.00	180.0	-75.0
MB18-04	693,197	5,266,798	469	275.00	60.0	-50.0
MB18-05	693,190	5,266,852	453	374.00	240.0	-75.0
MB18-06	693,197	5,266,798	469	47.00	250.0	-72.0
MB18-06A	693,220	5,266,806	463	242.00	250.0	-63.0
MB18-07	693,190	5,266,852	453	278.00	330.0	-80.0
MB18-08	693,189	5,266,761	473	200.00	350.0	-45.0
MB18-09	693,177	5,266,762	474	341.00	170.0	-45.0
MB18-10	693,165	5,266,770	474	551.00	250.0	-45.0
MB18-11	693,165	5,266,770	474	296.00	70.0	-45.0
MB18-12	693,315	5,266,723	475	831.00	354.0	-45.0
MB18-13	693,413	5,266,740	479	329.00	170.0	-45.0
MBE18-01	698,955	5,269,804	500	554.00	95.0	-60.0
MBE18-02	698,327	5,269,265	450	560.00	125.0	-48.0
Total				**5,392.00**		

10.5.1 2018 Drilling at Murray Brook

The 2018 drilling program at Murray Brook saw the completion of 14 drill holes totalling 4,278 m. Drill holes MB18-01 to MB-07 were completed as metallurgical drill holes; that is, drill holes completed to sample fresh material for metallurgical testwork. The drill hole traces and assay results are represented in Figure 10-10. Significant mineralized intercepts are listed in Table 10-9. Note that drill hole MB18-06 was abandoned for technical reasons and redrilled as MB18-06A. Drill holes MB18-05 and MB18-07 were extended to characterize the alteration halo and sulphide content of the host rock surrounding the massive sulphide deposit. The results show that within 40 to 50 m of the main deposit, the base metals content varies from 0.1% to 0.3% Pb+Zn. Beyond that distance, the base metals background is generally <100 ppm Zn.

The remaining six drill holes, MB18-08 to MB18-13, were completed to explore for mineralization along strike to the west and down-dip to the north of the Murray Brook deposit. The drill hole collars and traces with mineralized intercepts are illustrated in Figures 10-11 and 10-12. Drill hole MB18-10 was completed on an azimuth of 250° and intersected a highly silicified semi-massive to massive pyritic zone that graded 0.10 g/t Au and 0.12% Pb+Zn over 110.4 m, including 0.14 g/t Au and 0.18% Pb+Zn over 30.9 m. Drill hole MB18-11 was completed on an azimuth of 70° and intersected 0.84 g/t Au, 0.75% Cu, and 3.27% Pb+Zn over 50 m, including 1.0 g/t Au, 1.03% Cu, and 3.16% Pb+Zn over 20 m. Drill hole MB18-12, completed on an azimuth of 354°, intersected 1.17 g/t Au, 0.50% Cu, and 4.16% Pb+Zn over 153.2 m, including 1.18 g/t Au, 0.80% Cu and 2.52% Pb+Zn over 50.6 m. Drill hole MB18-12 also intersected the deepest alteration and zinc mineralization drilled below the Murray Brook deposit, which graded 1.05% Zn over 5.4 m from 556.0 m downhole.

Figure 10-10: 2018 Metallurgical Drilling 3-D Model View at Murray Brook



Drillhole	Azimuth	Dip	Depth (m)	MS (m)	Target
MB18-01	185	-65	125	44	Copper
MB18-02	350	-65	264	210	Low + High Grade Zinc
MB18-03	180	-75	125	100	Low + High Grade Zinc
MB18-04	60	-50	275	132	Copper
MB18-05	240	-75	374	120	Low + High Grade Zinc
MB18-06A	250	-63	242	155	Low + High Grade Zinc
MB18-07	330	-80	278	91	Low + High Grade Zinc

Source: Puma press release dated March 28, 2019.

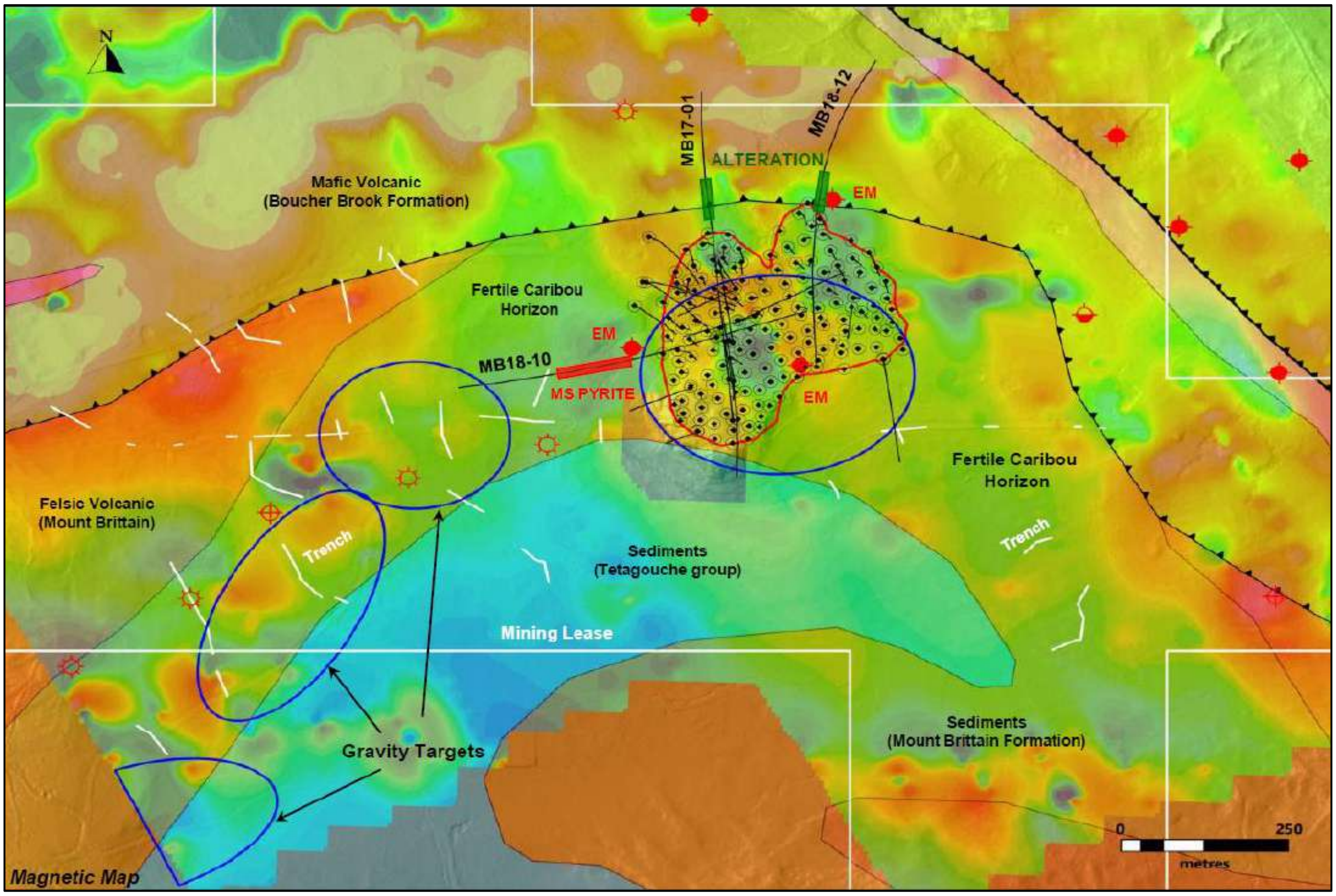
Table 10-9: Significant Intercepts in 2018 Drilling

Drill Hole ID	From (m)	To (m)	Length (m)*	Au (g/t)	Ag (g/t)	Cu (%)	Pb (%)	Zn (%)	Comment
MB-2017-01**	644.30	649.00	4.70	0.02	7	0.12	0.15	0.37	Zn mineralization below Murray Brook
and	669.90	680.40	10.50	0.06	10	0.04	0.11	0.38	
and	686.50	694.90	8.40	0.02	1	0.05	0.02	0.32	
MB-2018-01	48.00	54.00	6.00		51	2.00	0.83	1.71	metallurgical drill hole
and	64.40	101.00	36.60		41	1.35	0.63	1.38	
incl.	80.00	101.00	21.00		45	2.18	0.65	1.46	
MB-2018-02	19.00	38.00	19.00		31	2.11	0.61	1.85	metallurgical drill hole
incl.	20.00	29.00	9.00		37	3.12	0.70	1.99	
and	220.00	224.00	4.00		26	2.20	0.15	0.25	
MB-2018-03	12.00	36.00	24.00		41	2.53	0.63	3.19	metallurgical drill hole
incl.	12.00	26.00	14.00		42	3.12	0.43	2.75	
MB-2018-04	114.00	189.00	75.00		33	0.85	0.67	1.30	metallurgical drill hole
incl.	114.00	128.00	14.00		20	1.03	0.22	0.48	
and	240.00	252.30	12.30		14	1.20	0.37	0.92	
MB-2018-05	74.00	79.00	5.00	0.13	3	0.05	0.06	0.31	metallurgical drill hole
and	88.00	102.00	14.00	0.09	4	0.06	0.12	0.20	
and	109.40	230.30	120.90		70	0.11	2.05	6.11	
incl.	112.00	139.00	27.00		88	0.07	2.22	7.31	
incl.	179.00	230.30	51.30		93	0.13	3.07	8.54	
and	241.00	243.00	2.00	0.00	2	0.00	0.11	0.21	
and	271.00	276.00	5.00	0.01	1	0.04	0.05	0.14	
MB-2018-06A	57.00	215.10	158.10		38	0.18	1.20	3.75	metallurgical drill hole
incl.	79.00	97.00	18.00		89	0.10	1.76	6.08	
incl.	120.00	125.00	5.00		35	0.08	1.24	5.17	
incl.	146.00	151.00	5.00		52	0.03	1.79	5.33	
incl.	173.00	214.00	41.00		46	0.13	1.82	5.45	
MB-2018-07	132.00	150.00	18.00	0.06	4	0.04	0.09	0.21	metallurgical drill hole
and	154.90	242.00	87.10		96	0.13	2.35	6.86	
incl.	154.90	193.00	38.10		142	0.12	2.93	9.04	
incl.	223.00	241.00	18.00		97	0.23	3.16	9.66	
and	257.60	263.00	5.45	0.02	1	0.00	0.13	0.22	
MB-2018-10	253.10	363.50	110.40	0.10	2	0.02	0.04	0.08	mineralization west of Murray Brook
incl.	265.20	269.00	3.85	0.39	9	0.13	0.16	0.30	
incl.	298.00	300.00	2.00	0.13	4	0.06	0.12	0.28	
incl.	326.70	357.60	30.90	0.14	2	0.02	0.06	0.12	
incl.	339.30	347.30	7.95	0.08	2	0.01	0.06	0.20	
MB-2018-12	14.60	336.60	321.95	0.82	48	0.33	1.08	2.09	Cu mineralization below Murray Brook
i	522.00	561.40	39.40	0.07	5	0.02	0.07	0.21	
and	665.00	669.00	3.95	0.15	7	0.26	0.18	0.32	
and	767.00	771.00	4.00	0.01	7	0.15	0.17	0.55	

Notes: * True thickness is estimated to be approximately 70 to 90% of the drill core intercepts. **Drill hole MB-2017-01 extended from 435 m to 800 m long during the 2018 drilling program.

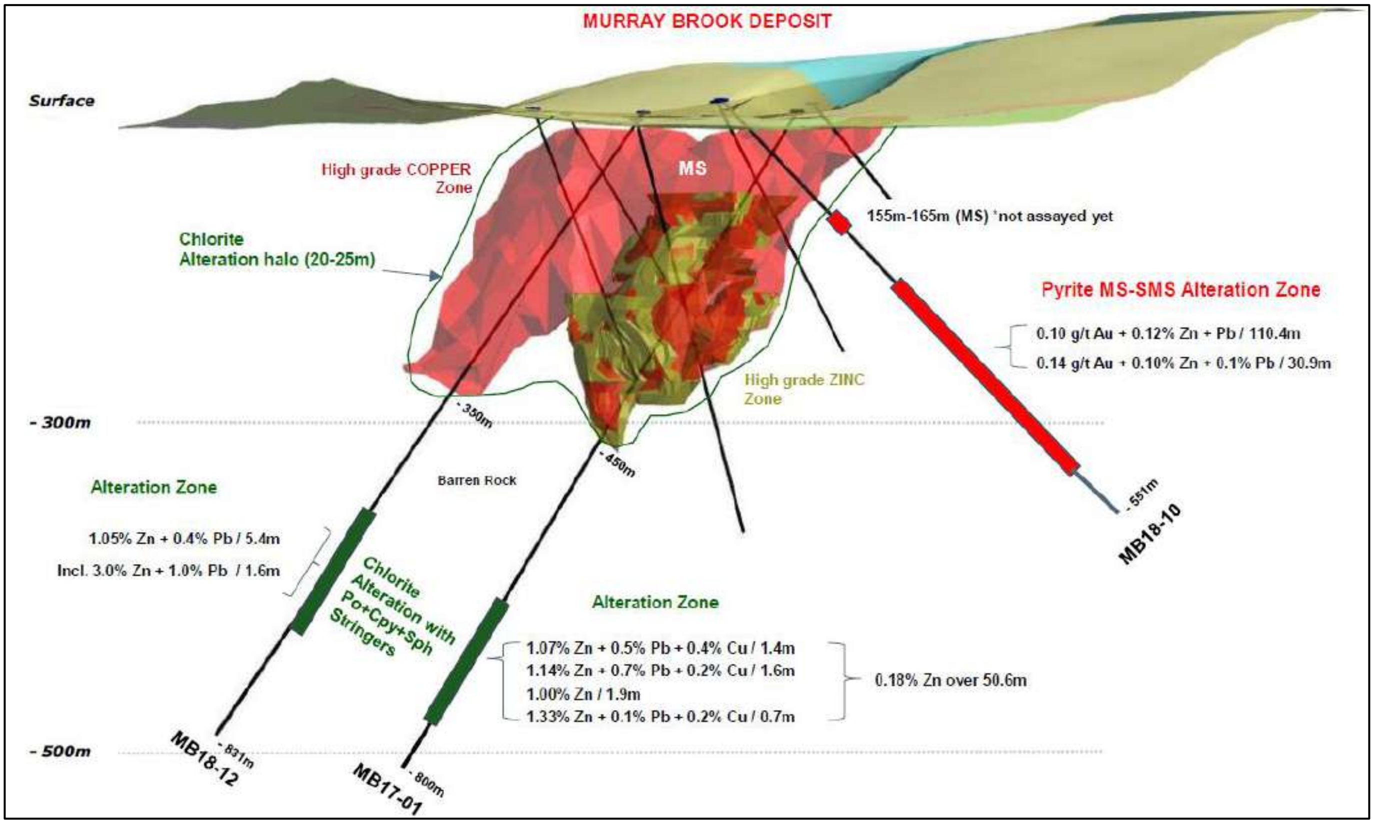
In addition, exploration drill hole MB-2017-01 from 2017 was extended to 800 m depth during the 2018 program and intersected additional mineralization (Figures 10-11 and 10-12). Mineralized intercepts in the extension of drill hole MB-2017-01 are listed in Table 10-9. From 435 to 635 m, the drill hole intersected the favourable sedimentary host rock lacking significant alteration and mineralization. From 625 to 700 m, a chloritic alteration zone was observed that overprints weak silicification and sericitization with four mineralized intervals grading above 1% Zn. The alteration halo is also present, which suggests potential for the Murray Brook deposit to continue at depth.

Figure 10-11: 2018 Exploration Drilling Plan View at Murray Brook



Source: Puma press release dated January 31, 2018.

Figure 10-12: 2018 Exploration Drilling 3-D Model View at Murray Brook



Source: Puma press release dated January 31, 2018.

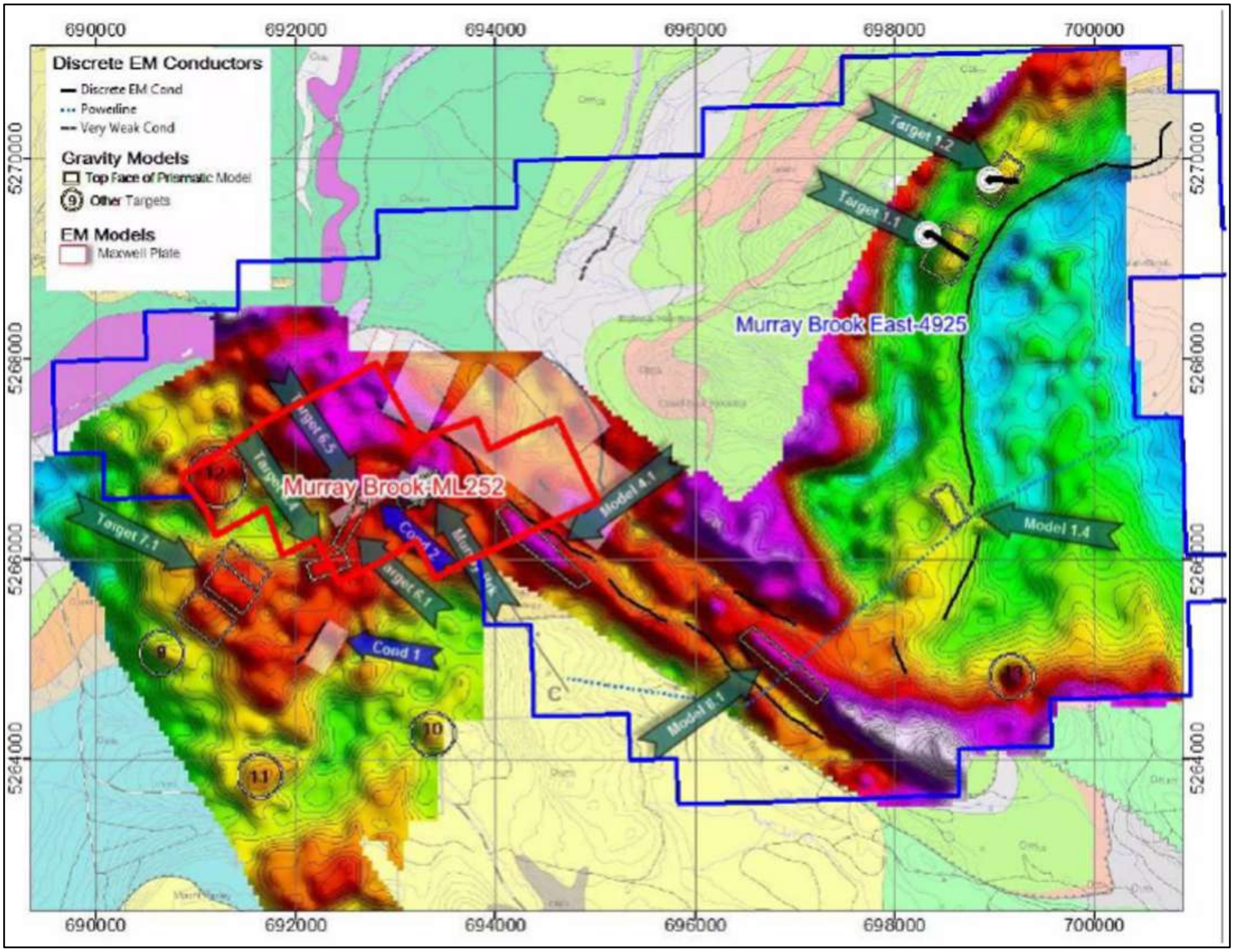
The extension of drill holes MB17-01 and MB18-12 to depths of 800 and 831 m, respectively, successfully confirmed the potential for making new discoveries at depth. The extension of the favourable horizon continues at least to a depth of 700 m and significant alteration with small higher-grade intervals were intersected in both drill holes, which are located 300 m away from each other. At Murray Brook, the alteration and mineralization halo consists a low-grade mineralization grading between 0.10 to 0.40% Zn located within 50 m of the high-grade massive sulphide deposit.

10.5.2 Drilling on Murray Brook East Claim 4925

10.5.2.1 Drilling Results

The Puma Exploration diamond drilling program completed during the summer of 2018 on Claim 4925 totalled two NQ-size drill holes and 1,114 m. Their locations are shown on Figure 10-13. The details of the two drill holes are presented above in Table 10-9. No significant assay results were reported for these two drill holes. However, each was surveyed for off-hole conductor features of interest.

Figure 10-13: Drill Holes MBE18-01 and MBE18-02 Locations on Gravity Anomaly Map



Source: Gagné and Hupé (2019).

10.5.2.2 Borehole Electromagnetic Surveys

The information in this section is taken largely from Gagné and Hupé (2019).

The selected drill holes were surveyed by Eastern Geophysics during September 2018 for off-hole conductive features of interest. Due to the blocky characteristic of the bedrock, this program required the installation of polyvinyl chloride (PVC) tubes prior to the surveying. This work was completed by the drilling company. The insertion of the PVC tubes was blocked 290 m downhole in drill hole MBE18-01 and was completed to the bottom in drill hole MBE18-02.

The borehole electro-magnetic survey in drill hole MBE18-01 revealed an in-hole anomaly below 240 m that suggested a large moderate to low conductor below to the right of the hole (Figure 10-14). This drill hole, collared to test the 1.2 gravity anomaly, intercepted the following four main lithic domains:

- Domain 1 – 2 to 188.8 m, a mafic domain (mafic sill/dyke and possibly volcanic) including numerous felsic tuff interlayers between 108.3 and 155.7 m
- Domain 2 – 188.8 to 282.4 m, a felsic volcaniclastic (crystal tuff) domain (Mount Brittain Formation)
- Domain 3 – 282.4 to 500 m, a sedimentary rock (siltstone/shale) unit with local melange units (Mount Brittain Formation)
- Domain 4 – 500 to 554 (end-of-hole) an altered felspar crystal tuff (Nepisiguit Falls Formation).

Sulphide mineralization (mainly pyrite) in drill hole MBE18-01 was discontinuously encountered in Domains 1, 3, and 4. Mafic units of Domain 1 include numerous intervals with trace to 5% fine-grained pyrite. Domain 3 showed two main intervals with mineralization consisting of disseminated pyrite ± graphite in two shear zones. Domain 4 contains mineralized pyrite in quartz stringers from 509 to 515 m.

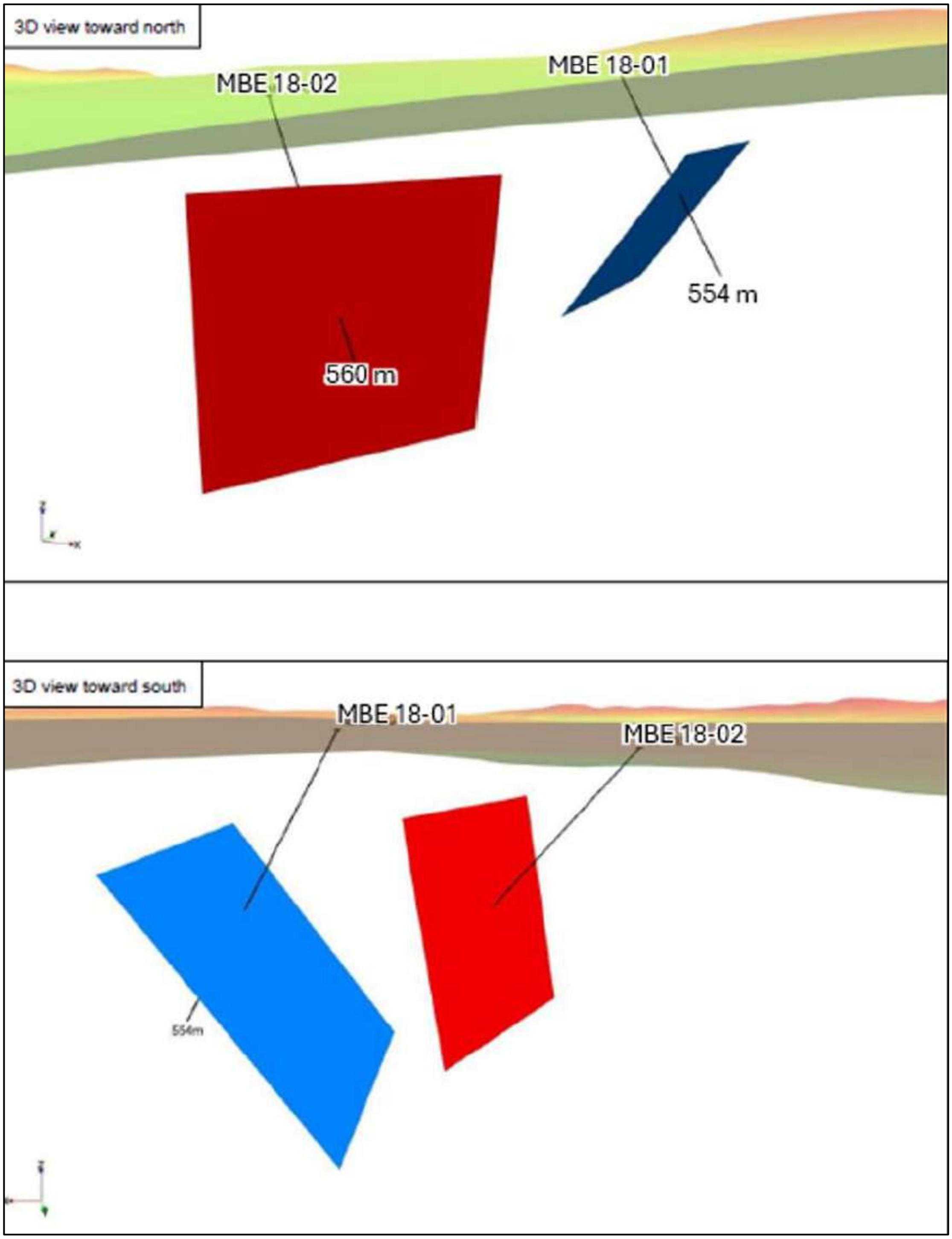
Considering that the lithic domains encountered in this drill hole contained only disseminated mineralization, the electromagnetic conductor anomaly is not explained. However, the anomaly is near the contact with the sedimentary rocks below 282.4 m. Those sedimentary rocks generally contain graphite in the foliation that could be conductive.

The borehole electro-magnetic survey of drill hole MBE18-02 indicates a large in-hole anomaly with moderate to low conductance at the bottom of the hole and centred at 500 m (Figure 10-14). This drill hole was completed to test the 1.1 gravity anomaly (see Figure 10-13). The hole intercepted the following three main lithic domains:

- Domain 1 – 6 to 257.2 m, a felsic volcanic lava flow (rhyodacite)/feldspar crystal tuff domain (Mount Brittain Formation)
- Domain 2 – 257.2 to 411 m, a mixed mafic, sedimentary rock and felsic volcaniclastic (crystal tuff) domain
- Domain 3 – 411 to 560 m, a sedimentary rock (siltstone with interbedded fine grain sandstone) unit (Mount Brittain Formation).

Sulphide mineralization in MB18-02 was encountered only in lithic Domain 2, which contained trace to 7% fine-grained pyrite. As in drill hole MB18-01, the mineralization observed does not explain the electromagnetic anomaly. However, graphite occurs in the sedimentary rocks from 411 to 560 m downhole.

Figure 10-14: Borehole Electro-Magnetic Conductor Plate Models for Drill Holes MBE18-01 & MBE18-02



Source: Gagné and Hupé (2019).

10.6 2019 Drilling – Puma

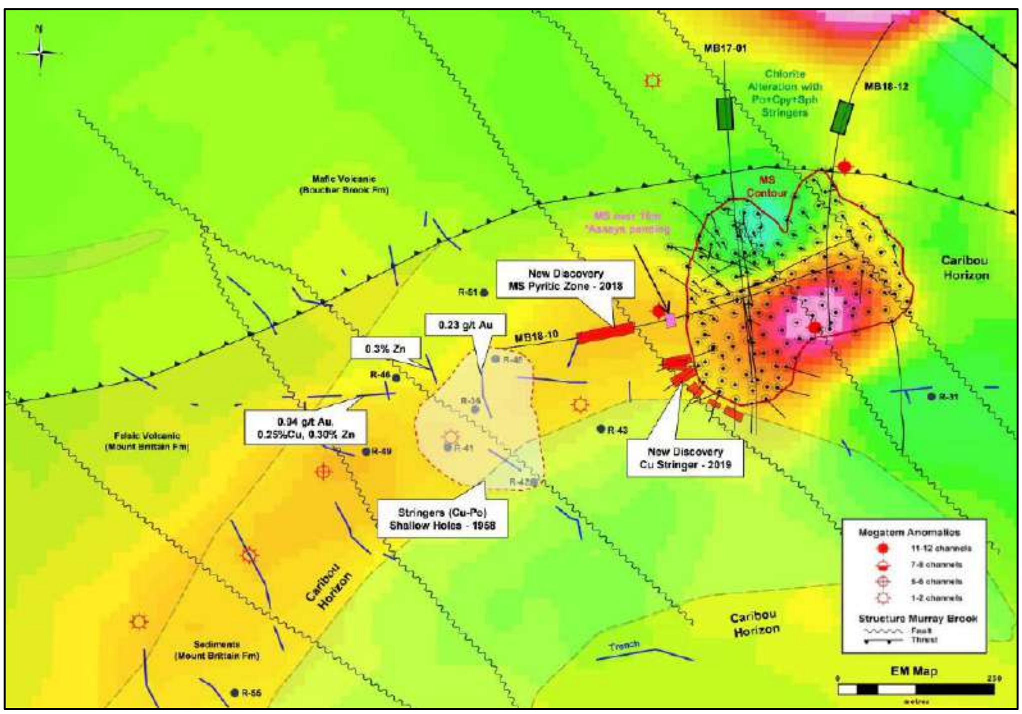
A small drilling program and precious metal study of Murray Brook drill core was completed in 2019. The results of this work are summarized below.

10.6.1 Drilling

In 2019, Puma completed nine relatively short drill holes totalling 714 m. The drill hole collar locations, trajectories, and lengths are demonstrated in Figure 10-15 and listed in Table 10-10.

The nine drill holes were collared on the southwestern margin of the Murray Brook deposit and drilled outwards with the purpose of exploring for mineralization to the west and south. The drilling resulted in the discovery of a new zone containing copper stringers extending from surface and outside of the Murray Brook mineral resources. The discovery is a new horizon adjacent to the massive sulphide mineralization that corresponds to a strong historical MegaTEM anomaly at depth in the same area where drill hole MB18-10 had intersected significant sulphide mineralization over 100 m (see Figure 10-15). This newly discovered copper stringer zone was traced over a distance of 180 m at surface by four drill holes from the January 2019 drilling program and by previous drill hole MB17-03 (Figure 10-16). These drilling results, together with the 2011 and 2012 results, suggest that the copper stringer zone extends from surface to a depth of 170 m and varies in apparent thickness from 1.6 to 26.6 m. Such copper stringer zones adjacent to massive sulphide zones are a common feature of volcanogenic massive sulphide (VMS) deposits.

Figure 10-15: 2019 Drilling Plan View

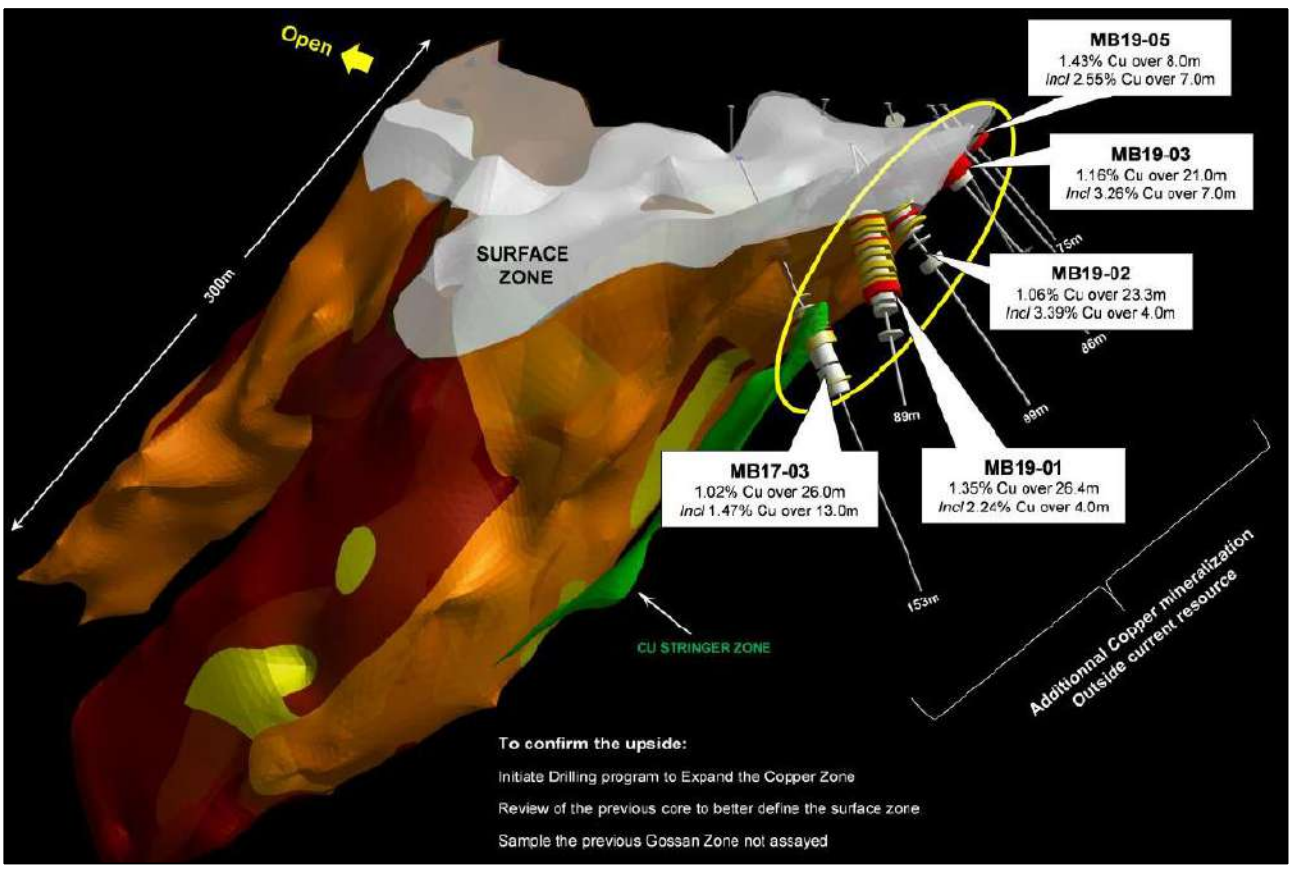


Source: Puma press release dated April 10, 2019.

Table 10-10: 2019 Drill Hole Collar Information

Drill Hole ID	Easting (m)	Northing (m)	Elevation (masl)	Length (m)	Azimuth (deg)	Dip (deg)
MB-2019-01	693,106	5,266,652	484	89.00	270.0	-45.0
MB-2019-02	693,106	5,266,652	484	99.00	230.0	-45.0
MB-2019-03	693,115	5,266,624	484	86.00	210.0	-45.0
MB-2019-04	693,142	5,266,609	484	78.00	196.0	-45.0
MB-2019-05	693,168	5,266,595	483	75.00	180.0	-45.0
MB-2019-06	693,194	5,266,589	477	71.00	180.0	-45.0
MB-2019-07	693,217	5,266,598	474	62.00	135.0	-45.0
MB-2019-08	693,234	5,266,618	473	74.00	103.0	-45.0
MB-2019-09	693,247	5,266,655	469	80.00	107.0	-45.0

Figure 10-16: 2019 Drilling in 3-D Model View



Source: Puma press release dated April 10, 2019.

Drill hole MB19-01 returned 26.4 m grading 1.35% Cu, 19 g/t Ag and 0.18 g/t Au, including 4.0 m grading 2.24% Cu and 0.25 g/t Au. Furthermore, several additional sulphide intercepts with grades >1% Cu over >20 m were recorded in the 2019 drill holes MB-2019-02, MB-2019-03, MB-2019-04, and MB-2019-05 along the western boundary of Murray Brook (Table 10-11). In addition, the 2019 drill holes intersected the Gossan Zone on the Murray Brook massive sulphide deposit with similar grades of gold, silver, and lead to those found by NovaGold in the late 1980s and early 1990s (Table 10-12).

Table 10-11: Significant Intercepts from 2019 Drilling

Drill Hole ID	From (m)	To (m)	Length (m)*	Cu (%)	Au (g/t)	Ag (g/t)	Pb (%)	Zn (%)
MB-2019-01	20.0	70.0	50.0	0.81	0.13	13	0.17	0.54
incl.	27.6	54.0	26.4	1.35	0.18	19	0.23	0.74
MB-2019-02	17.0	48.0	31.0	0.88	0.12	14	0.12	0.65
incl.	17.0	40.3	23.3	1.06	0.13	17	0.14	0.80
MB-2019-03	11.0	32.0	21.0	1.16	0.22	12	0.13	0.22
incl.	11.0	21.0	10.0	2.40	0.36	22	0.10	0.22
MB-2019-05	15.0	23.0	8.0	1.43	0.26	12	0.05	0.12
incl.	16.0	20.1	4.1	2.55	0.35	16	0.03	0.13
MB-2019-06	13.3	19.2	5.9	0.25	0.02	1	0.01	0.03

Note: * True thickness ranges from 75% to 95% of the drill core intercepts.

Table 10-12: Significant Intercepts from 2019 Drilling of the Gossan Zone

Drill Hole ID	From (m)	To (m)	Length (m)	Au (g/t)	Ag (g/t)	Cu (%)	Pb (%)	Zn (%)
MB-2019-03	6.6	11.0	4.4	1.09	80	0.05	0.38	0.02
MB-2019-04	15.0	19.5	4.5	0.23	23	0.09	0.34	0.39
MB-2019-05	13.0	15.0	2.0	1.09	10	0.18	0.06	0.03
MB-2019-07	8.4	10.9	2.5	0.19	18	0.10	0.53	0.03

10.6.2 Precious Metals Evaluation Program

In addition to the drilling program in 2019, Puma commenced a precious metal evaluation program on the Murray Brook deposit. The Murray Brook deposit contains gold and silver in its three main metallogenic sub-domains, which from surface to depth are the Gossan Zone, Oxide Zone, and main Massive Sulphide Zone. The evaluation program included studying the gold and silver zonation, precious metals association, and respective recovery within the three main metallogenic sub-domains. The different mining and processing options available were evaluated to recover the maximum value of the precious metals contained.

Results for drill holes MB-2018-15 and MB-2018-07 from the 2018 metallurgical drilling program are summarized in Table 10-13. In drill hole MB18-07, the massive sulphide contains broad gold and silver enrichment over its entire length with 0.67 g/t Au and 96 g/t Ag over 87.0 m. The higher results of 1.15 g/t Au and 87 g/t Ag over 22.0 m are for samples from the footwall contact with the altered sedimentary rocks.

Table 10-13: Significant Gold Assays for the 2019 Precious Metal Evaluation Program

Drill Hole ID	From (m)	To (m)	Length (m)	Au* (g/t)
MB-2018-05	110	230	120	0.39
incl.	180	230	50	0.68
MB-2018-07	155	242	87	0.62
incl.	155	168	13	0.61
incl.	181	193	12	0.75
incl.	220	242	22	1.15

Note: * Gold assays by ALS Laboratories. Assay results for silver, zinc, lead, and copper are given in Table 10-9.

11 SAMPLE PREPARATION, ANALYSES, AND SECURITY

This section discusses the sampling conducted by Votorantim Metals Canada Inc. (VMC) and Puma at the Murray Brook property between 2010 and 2019.

11.1 Sample Preparation and Security Measures

11.1.1 VMC (2010 to 2012)

The Murray Brook drilling and sampling program was supervised by VMC Senior Geologist Garth Graves, P. Geo. Drill core logging and sample marking was performed by VMC staff geologists Laura Coutts, B.Sc., GIT; Denise Martinez, B.Sc., GIT; and Barry MacCallum, B.Sc., GIT. Drill core cutting and sampling were performed by experienced technicians.

Drill core is placed in wooden drill core boxes beside the drill and secured using rubber bands (pieces of tire inner tubes). Drill core boxes are picked up once or twice per day from the drill site by VMC's staff, and delivered directly to VMC's secure drill core logging facility at 1095 Bridge Street, Bathurst, NB.

Drill core is aligned, measured, and checked for drill core recovery and rock quality designation (RQD). Magnetic susceptibility and conductivity are measured by scanning the drill core using an MPP2 metre from Geophysics GDD of Québec City. When a certified operator is present, the drill core may also be scanned using a Niton XL2-500 XRF instrument to gain a qualitative estimate of base metal, arsenic, and antimony distribution.

Drill core is logged geologically and the results recorded in an Excel™ spreadsheet. All massive and strongly disseminated sulphide intervals are marked and tagged for sampling, with up to three additional shoulder samples beyond the limits of the strong sulphide mineralization, depending on whether the contact is sharp or gradational. Samples are generally 1.0 m long, unless lithological contacts make for more logical breaks. Short intervals (<30 cm) of country rock may be included in sulphide samples; larger intervals are sampled separately.

Tags are placed in the drill core boxes to indicate where a certified reference material (CRM) or blank should be inserted in the sample stream. A line is drawn on the drill core to indicate to the sampler where to cut the drill core. When marked-up and assay tags are positioned, the drill core is photographed to preserve a record of the sample numbers and intervals before it is sawn.

Drill core is sawn in half using a VanCon diamond saw. One-half of the drill core is placed in a standard plastic sample bag and secured by a nylon cable tie and tagged for analysis; the other half returned to the drill core box for future reference.

A duplicate drill core sample is taken at random approximately every 20th sample by sawing the remaining drill core in half, leaving a ¼ drill core for reference. One of three CRM and one blank sample are inserted into the sample stream at the rate of one for every 20 samples.

Up to five or six bagged samples are placed in large polypropylene 'rice bags,' which are tied with a numbered plastic security tag, placed in a 20-litre plastic pail, and capped. Samples are shipped on palettes in batches of 30 samples (about six pails) or multiples thereof. These are picked up from the drill core facility by Day and Ross Inc., a bonded courier, and driven to independent lab TSL Laboratories Inc., (TSL) in Saskatoon.

11.1.2 Puma Exploration (2017 to 2019)

All sampling, drilling, testing, and analyses are conducted rigorously according to regulatory guidelines. The driller is responsible for ensuring that the drill core is placed into boxes in the correct order and marking the length tags inside the drill core boxes for each rod-length of drill core. This step is examined by the technician and supervised by the on-duty project geologist. The driller picks up drill core boxes from the drill site and takes them to the drill core logging facilities in Bathurst.

Boxes are then laid out on logging tables and checked to ensure the drill core is continuous and in the right order in each box. Geological logging of drill core is conducted, and the sample positions are marked to conform to lithological/alteration changes. Sample numbers are written inside of the drill core boxes corresponding to pre-printed sample tags. Quality control procedures include routine insertion of prepared CRMs, sourced blank material, and field duplicates. All samples are weighed by a company technician, with certain samples weighed both in air and water to measure bulk density by immersion. Chalk lines are marked on the drill core to identify the drill core axis and boxes for sampling are moved to the cutting area. Drill core is sawn in half using a using a hydraulic drill core saw. Drill core boxes are returned to their place on the logging tables in sequential order. Strong plastic rock sample bags are labelled with the sample number on the outside and the corresponding sample tags are placed inside the bag. One-half of the drill core is then placed in its respective sample bag, and the second half is carefully returned to the drill core box and stored at the Alliance warehouse for future reference. Individual sample bags are secured with a nylon cable tie or industrial adhesive tape and tagged for analysis, then placed in a numbered rice bag and tied with cable-ties. Drill core samples designated for metallurgical testing are entirely sampled, with ½ of the drill core sample bagged for geochemical assaying and the other half dried and bagged for metallurgical testing. Metallurgical samples are placed into 45-gallon barrels filled with nitrogen to evacuate the air before shipping.

Strict chain-of-custody procedures are followed at the project. Samples are picked up from the drill core facility by Armour Courier Services (ACS) of New Brunswick and transported to the ALS Global facility in Sudbury for sample preparation, before being shipped to the ALS Global laboratory in Vancouver for geochemical assaying. Company employees deliver drill core samples designated for metallurgical testing by truck to either the Trevali Caribou mine laboratory for geochemical assaying or they are shipped to the Research and Productivity Council (RPC) of New Brunswick in Fredericton for metallurgical testing. No security issues were noted during the 2017 to 2019 programs.

11.2 Sample Analyses

11.2.1 VMC (2010 to 2012)

Drill core samples at TSL are crushed to 70% passing -10 mesh (1.70 mm), from which a 1,000 g portion is riffle-split and pulverized to 95% passing -150 mesh (106 µm). All equipment is cleaned with compressed air and brushes after every sample. Both pulps and rejects are stored at TSL in Saskatoon.

Samples are assayed for copper, lead, zinc, and silver using a four-acid total digestion followed by atomic absorption spectrometry (AAS) Gold is determined by a standard lead-collection fire assay procedure using a 30 g aliquot with an AAS finish. Samples exceeding 3,000 parts per billion (ppb) are re-analysed using the fire assay procedure followed by gravimetric weighing.

TSL was established in 1981 and was an accredited laboratory certified to perform, inter alia, assay and umpire assay work for the five elements routinely assayed for the Murray Brook Project. The TSL quality system conformed to requirements of ISO/IEC Standard 17025 guidelines, and the laboratory qualified for the Certificates of Laboratory Proficiency since the program's inception in 1997.

11.2.2 Puma Exploration (2017 to 2019)

Samples at ALS Global are crushed to 70% less than 2 mm, from which a 250-g portion is riffle-split and pulverized to better than 85% passing 75 µm. Samples at the Caribou mine laboratory are crushed and pulverized to a range of 80 to 100 µm.

Samples at ALS are assayed for copper, lead, zinc, and silver using an ultra trace aqua regia followed by Inductively coupled plasma mass spectroscopy (ICP-MS). Samples returning assay grades greater than 10,000 ppm for copper, lead, and zinc or 100 ppm for silver, are further analysed by "ore" grade aqua regia digestion with Inductively coupled plasma atomic emission spectroscopy (ICP-AES) finish. Gold is determined by a standard lead collection fire assay procedure using a 30 g aliquot with an AAS finish, with overlimit samples re-analysed by fire assay with gravimetric finish. Samples at the Caribou mine laboratory are assayed for copper, lead, zinc, silver, and iron content by atomic absorption.

Quality control protocol at the laboratory includes monitoring performance via the use of CRMs and duplicate samples at a frequency of one in every 30 samples. There are typically eight CRMs run for each batch of 25. A duplicate is analysed at least once every 50 samples. No blanks are run. Pulps are currently stored in the laboratory in labelled brown sealed bags, which in turn are stored in organized labelled boxes. Coarse rejects are stored in labelled 18 L pails on pallets inside the Alliance warehouse.

ALS is independent of Puma and has developed and implemented strategically designed processes and a global quality management system at each of its locations. The global quality program includes internal and external inter-laboratory test programs and regularly scheduled internal audits that meet all requirements of ISO/IEC 17025:2017 and ISO 9001:2015. All ALS geochemical hub laboratories are accredited to ISO/IEC 17025:2017 for specific analytical procedures.

11.3 Bulk Density

As discussed in Section 14.11 of this report, the database consists of 2,961 bulk density measurements, of which 1,888 densities were tested from 2017 and 2018 drill holes, and 1,073 analyses performed on 2011 and 2012 drill core using a wet immersion method. The bulk density varied from 2.04 to 6.86 t/m^3 and averaged 4.14 t/m^3. A total of 2,050 of these bulk densities are included inside the mineralization wireframes. The bulk density block model was interpolated with the constrained data, which was capped at 6.5 t/m^3.

Independent verification sampling carried out by the qualified person during a site visit in 2023 confirmed the on-site measurements taken. A total of 14 due diligence samples were measured independently at Actlabs, returning a mean value of 4.25 t/m^3, median value of 4.37 t/m^3, minimum value of 3.26 t/m^3, and a maximum value of 4.8 t/m^3.

11.4 VMC (2010 to 2012) Quality Assurance / Quality Control Review

VMC implemented a quality assurance / quality control (QA/QC) program for all phases of drilling from 2010 to 2012.

11.4.1 Performance of Certified Reference Materials

Three CRMs—CDN-ME-13, CDN-ME-16, and CDN-ME-17—were supplied by CDN Resource Laboratories Ltd. of Vancouver, BC. CDN-ME-13 and CDN-ME-17 were prepared from massive and semi-massive sulphides from the Archean-aged Izok Lake VMS deposit, whereas CDN-ME-16 was prepared from a "mixture of ores". CRMs were inserted into the sample stream at a rate of 1 per every 20.

There were 201 CRMs inserted into the batches sent for geochemical assaying. VMC geologists monitored the results on a real-time basis as the reports were received from the lab. For monitoring purposes, two standard deviations above and below the mean were used as warning limits. VMC also produced a complete and detailed QC report at the end of the drilling phase. The author reviewed all results received from the laboratory and the Murray Brook Joint Venture QC report, and no material issues were observed.

11.4.2 Performance of Blanks

Sandblasting-grade ground glass purchased in Bathurst was employed as the blank material at the project. It was inserted into the sample stream at a rate of 1 per every 20 (5%).

There were 201 blanks inserted into the sample stream. Gold reported at or below its lower limit of detection of 5 ppb, with six outliers. The highest gold value was 15 ppb (0.015 g/t). Silver reported almost all values less than 1 g/t, with six outliers. The highest silver value was 2.0 g/t. Copper and zinc reported all values at or below detection limit with six and seven outliers respectively. Most lead values exceeded the lower detection limit, with a mean value of 220 ppm. Only seven outliers were flagged, with a high value of 800 ppm. It is likely that these assays are most simply explained by the nugget effect of high-lead glass particles, since lead is a common constituent of glass. None of the outliers was judged to have any impact on the metal value.

11.4.3 Performance of Duplicates

Drill core duplicates were produced by ¼ sawing drill core roughly every 20 samples and sending the ¼ split to the laboratory as a duplicate of the half drill core, leaving a ¼ drill core sample in the box as a witness in those cases. Two-hundred and one duplicate drill core samples were taken. Simple scatter graphs for each of the five elements were plotted to show the correlation between the ¼ and half drill core sample. Even the gold in the deposit, which would be expected to demonstrate poor precision at this level of homogeneity, in fact demonstrated excellent precision. All the other four elements demonstrated excellent precision close to 1:1.

11.4.4 Umpire Sampling Program

Sample pulps were forwarded from the principal laboratory (TSL) to a secondary laboratory (ACME labs in Vancouver) for umpire sampling to verify the performance at TSL. A total of 151 pulp samples were sent for analysis at VMC's request, with effort made to select samples representative of the distribution of grades of the massive sulphide body. The correlation coefficients between the original and umpire samples were all very close to one. Silver displayed the poorest precision, with a correlation coefficient of 0.89. Results for silver at ACME were on average 11% lower than the results at TSL. This difference is nevertheless considered acceptable by the author, considering that the samples were analysed at two different labs.

11.5 Puma (2017 To 2019) Quality Assurance / Quality Control Review

Puma implemented a QA/QC program for all phases of drilling from 2017 to 2019.

11.5.1 Performance of Certified Reference Materials

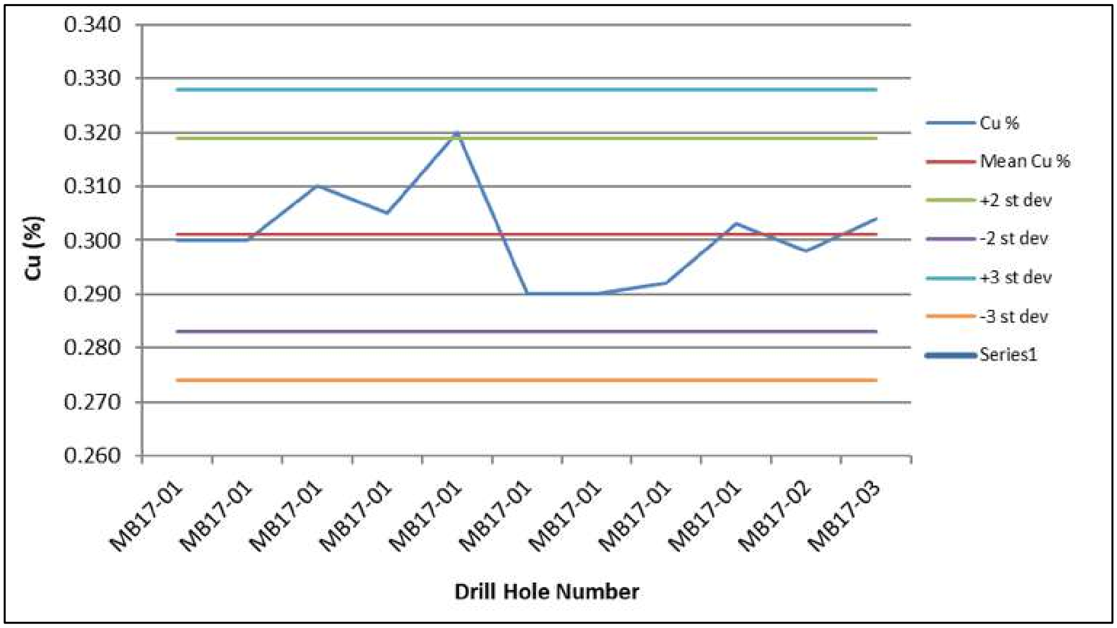
CRMs were inserted into the analysis stream at different rates each year from 2017 to 2019 at the project. The insertion rate of CRMs in 2017 was 1 per every 50; this rate increased in 2018 to 1 per every 20, and then decreased slightly to 1 per every 25 in 2019. The CRMs utilized during the drilling programs were sourced from CDN Resource Laboratories Ltd., of Langley BC, and included: CDN-ME-1402, -1204, -1410, and -1706. All four CRMs are certified for copper, lead, zinc, gold, and silver. However, no gold data were available except for a single data point for CDN-ME-1204. A fifth CRM was utilized in 2017 alone, but there are no records available as to which CRM it was, or what the certified mean value or between-laboratory standard deviation was. As a result, the author refers to this as the "2017 CRM" and has calculated the mean and standard deviation from the available 2017 data to evaluate performance.

Criteria for assessing CRM performance are based as follows: data falling within ±2 standard deviations from the accepted mean value pass; while data falling outside ±3 standard deviations from the accepted mean value fail.

There were 11 data points to review for the 2017 CRM and no failures were recorded. A total of 27 CDN-ME-1402 results were reviewed, with one low failure recorded for all four elements. Slight low biases were noted for copper, lead, and silver for this CRM. CDN-ME-1204 returned 25 results for copper, lead, zinc and silver, and a single result for gold. A single failure only for copper was noted, returning a result slightly greater than +3 standard deviations from the accepted mean value. There were seven data points to review for CDN-ME-1410 and only one failure recorded for lead, slightly below -3 standard deviations from the accepted mean value. There were eight CDN-ME-1706 results to review, and a single high failure for lead was noted, and one low and one high failure for zinc. Performance charts for all five CRMs are shown in Figures 11-1 to 11-21.

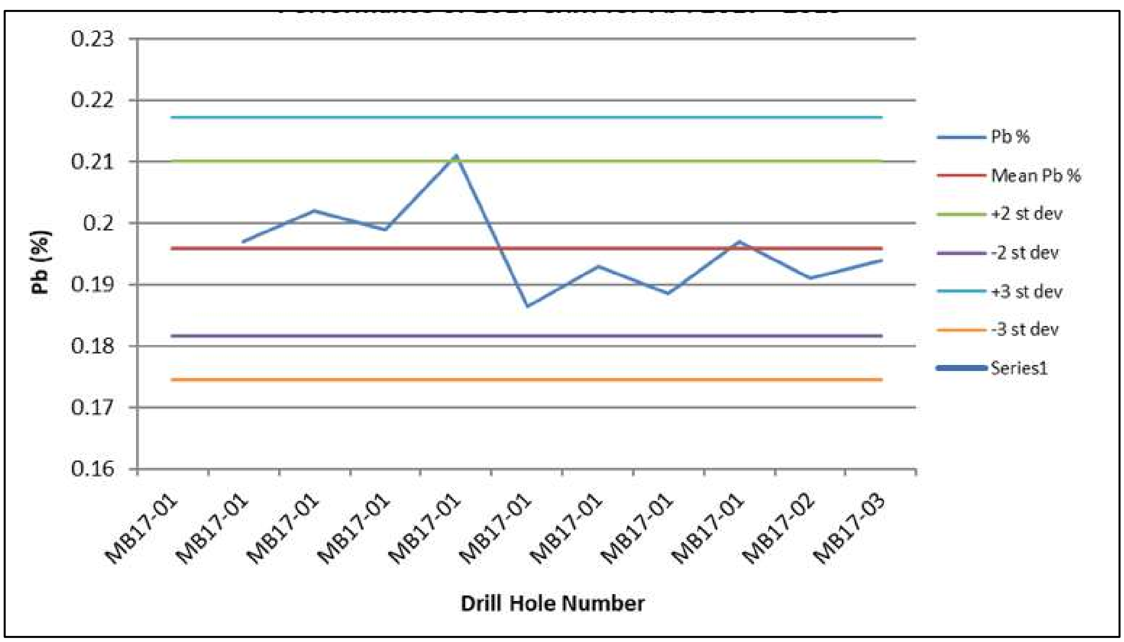
The author considers that the CRM data demonstrate acceptable accuracy in the 2017 to 2019 Murray Brook database.

Figure 11-1: Performance of 2017 CRM Cu for 2017 to 2019 Drilling



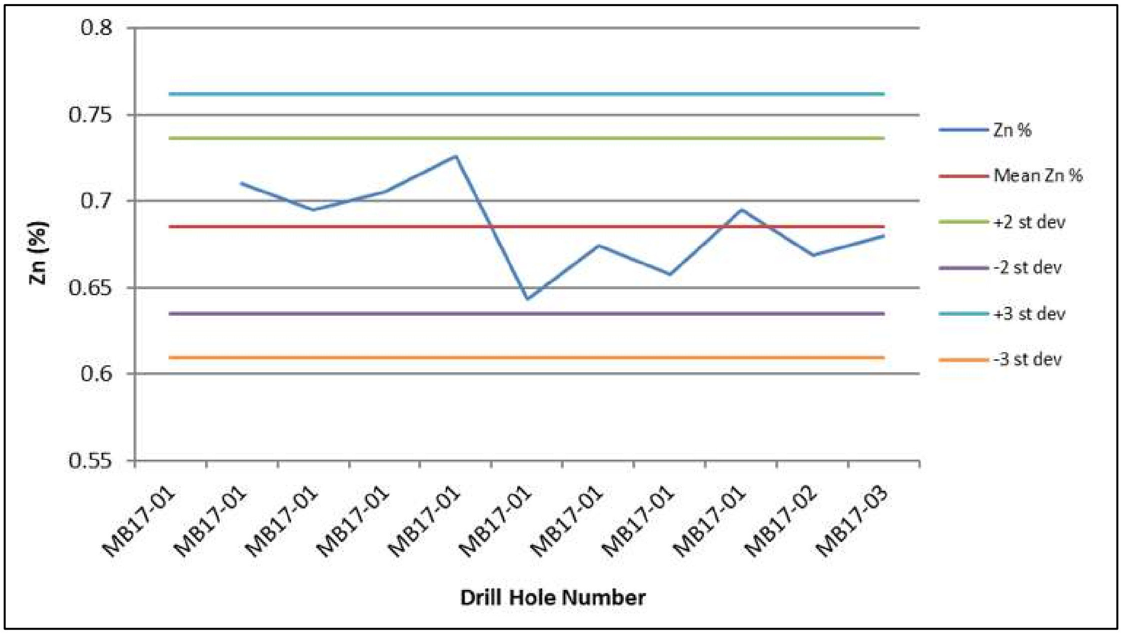
Source: P&E (2023).

Figure 11-2: Performance of 2017 CRM Pb for 2017 to 2019 Drilling



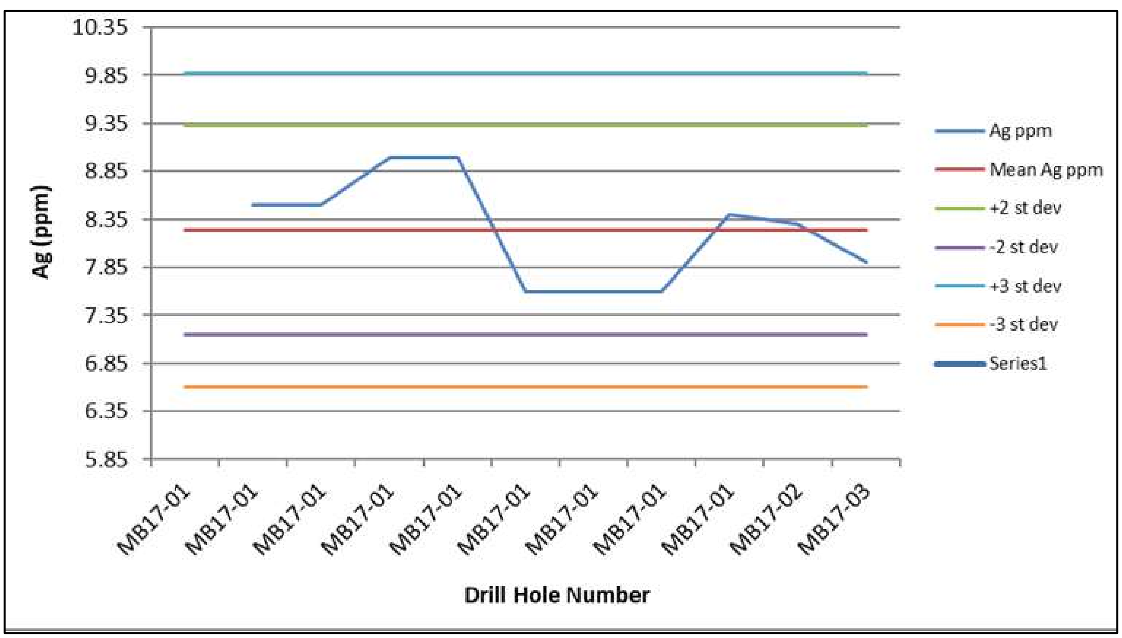
Source: P&E (2023).

Figure 11-3: Performance of 2017 CRM Zn for 2017 to 2019 Drilling



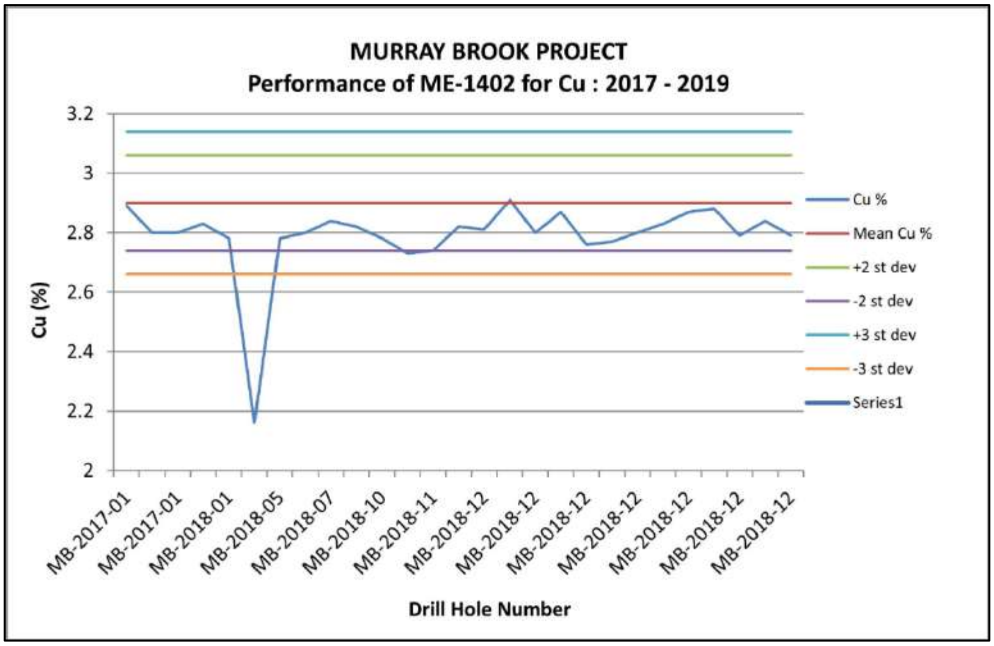
Source: P&E (2023).

Figure 11-4: Performance of 2017 CRM Ag for 2017 to 2019 Drilling



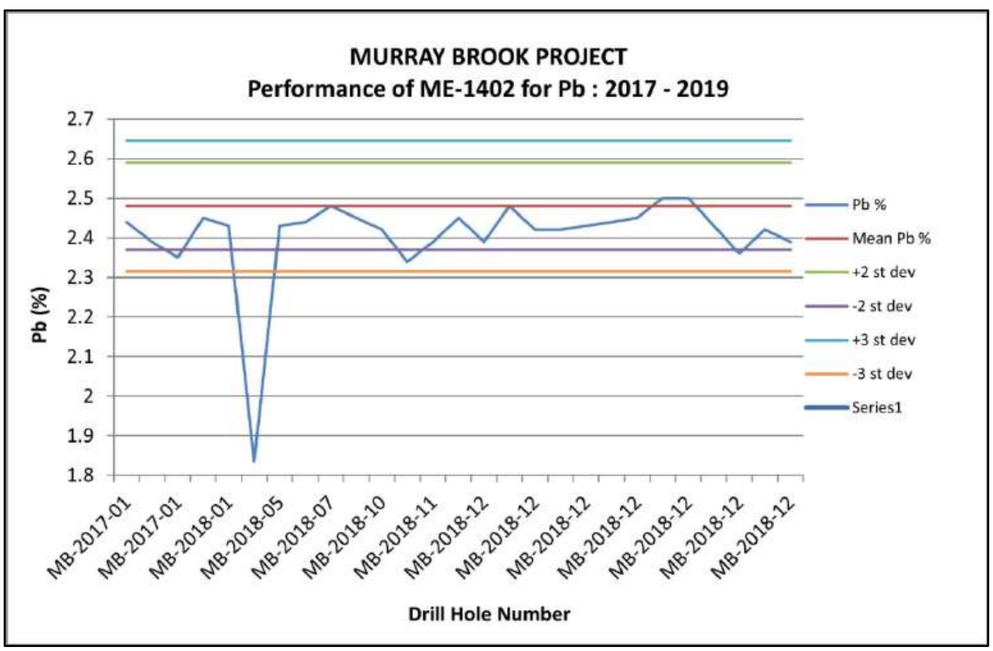
Source: P&E (2023).

Figure 11-5: Performance of CDN-ME-1402 CRM Cu for 2017 to 2019 Drilling



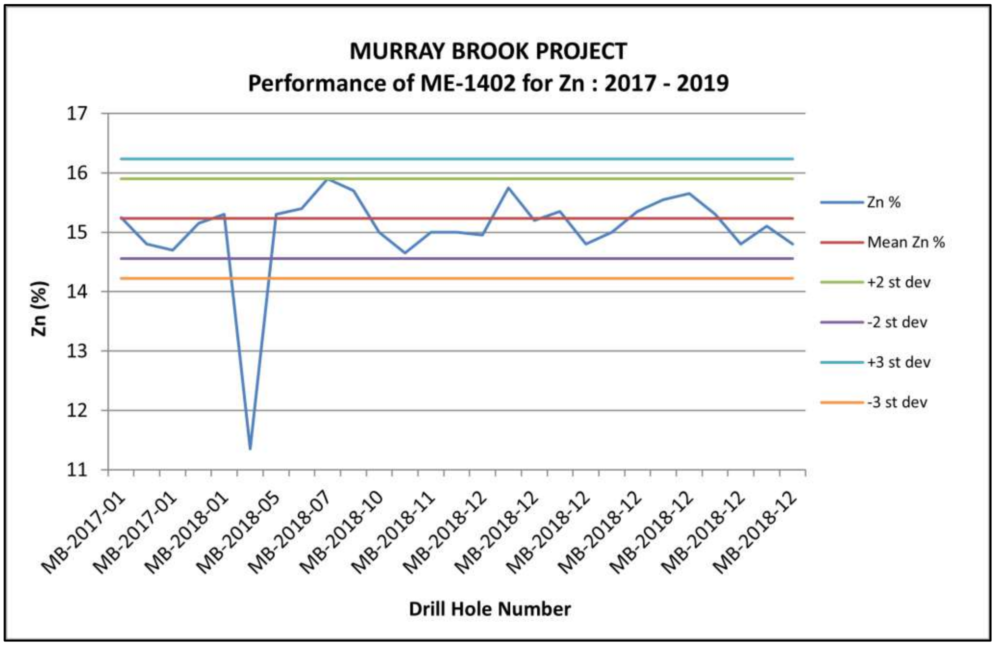
Source: P&E (2023).

Figure 11-6: Performance of CDN-ME-1402 CRM Pb for 2017 to 2019 Drilling



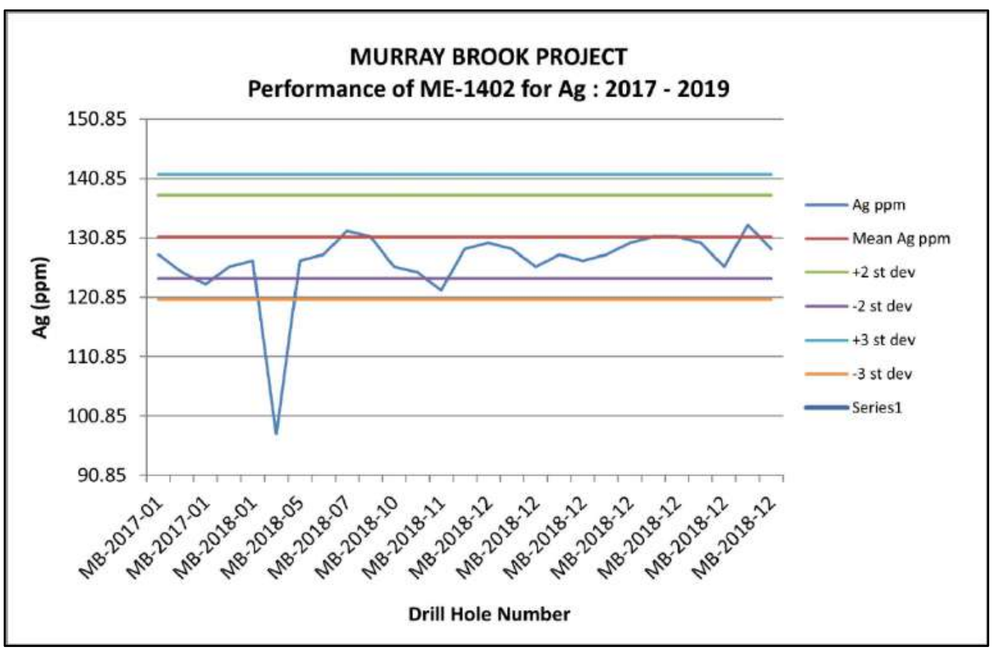
Source: P&E (2023).

Figure 11-7: Performance of CDN-ME-1402 CRM Zn for 2017 to 2019 Drilling



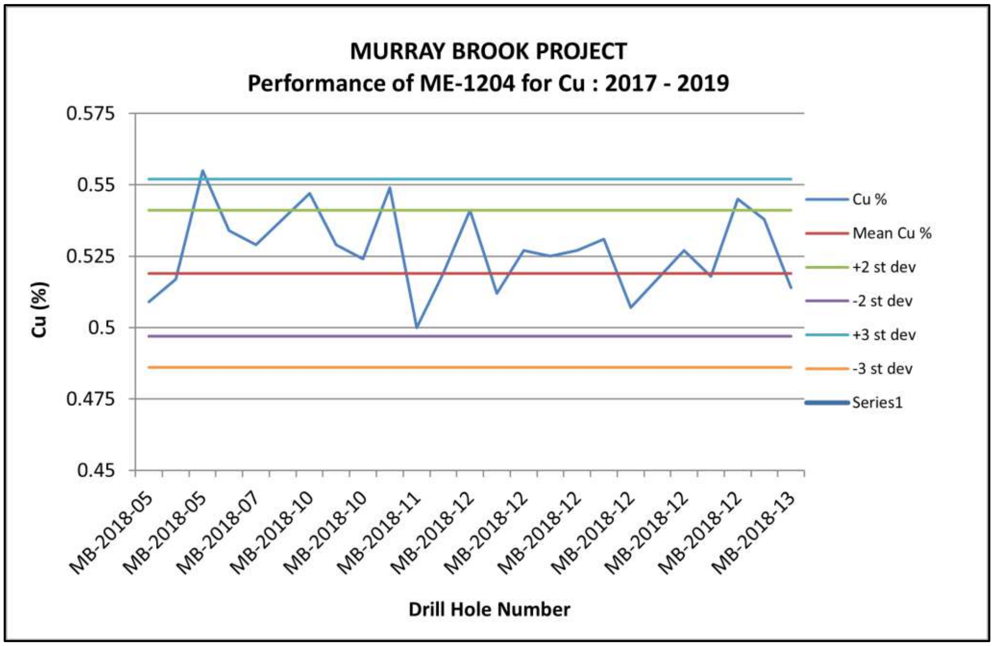
Source: P&E (2023).

Figure 11-8: Performance of CDN-ME-1402 CRM Ag for 2017 to 2019 Drilling



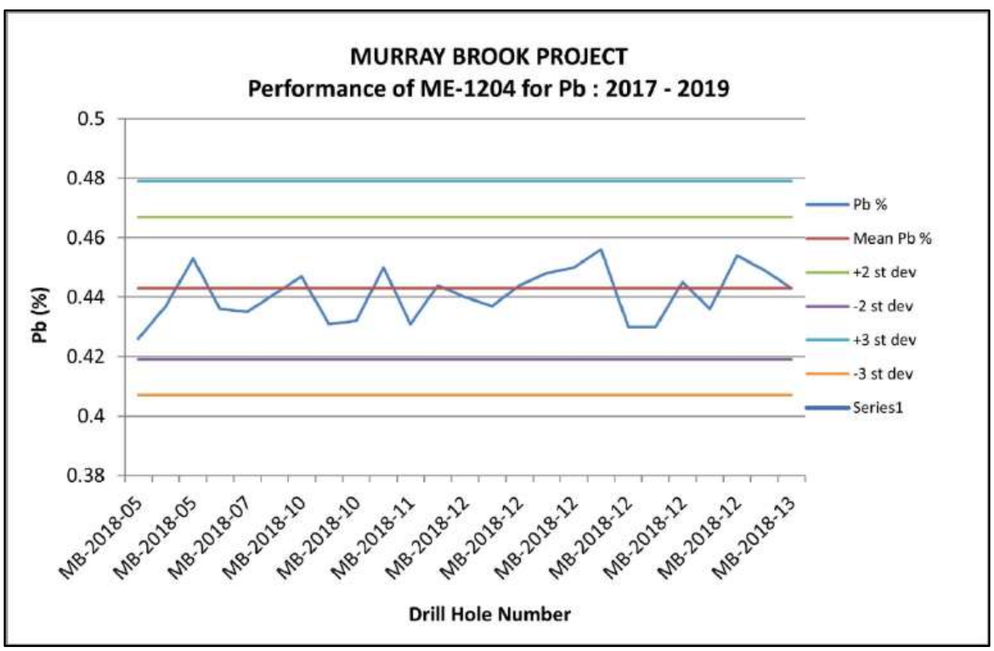
Source: P&E (2023).

Figure 11-9: Performance of CDN-ME-1204 CRM Cu for 2017 to 2019 Drilling



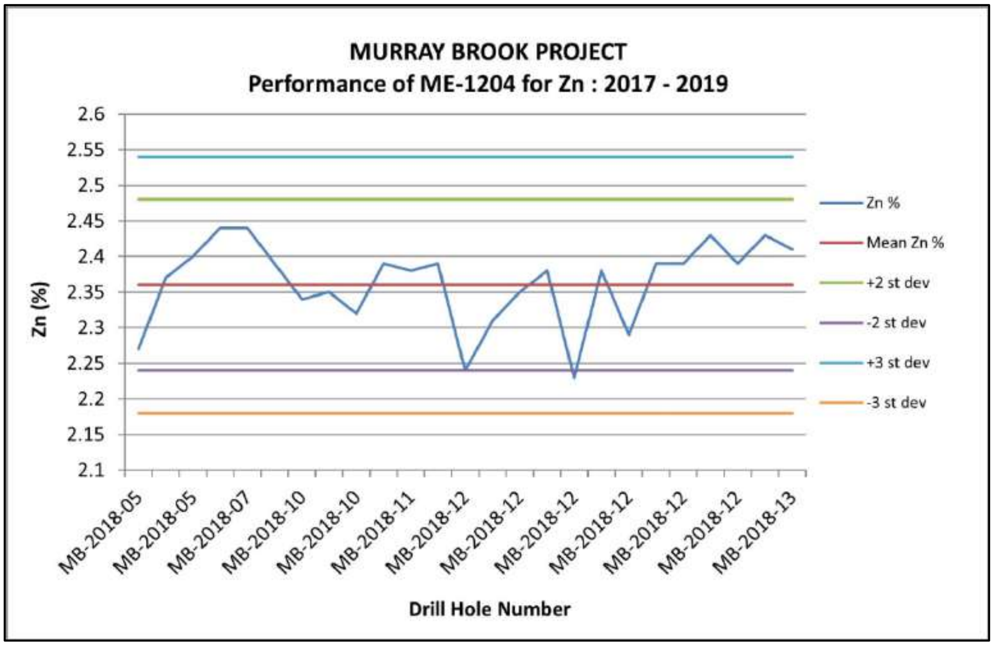
Source: P&E (2023).

Figure 11-10: Performance of CDN-ME-1204 CRM Pb for 2017 to 2019 Drilling



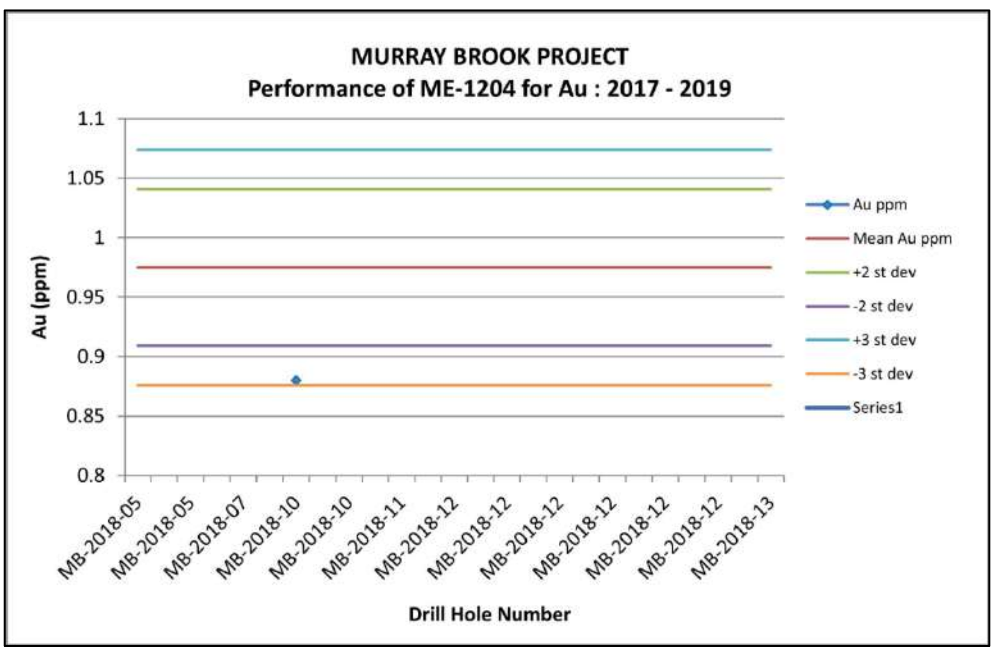
Source: P&E (2023).

Figure 11-11: Performance of CDN-ME-1204 CRM Zn for 2017 to 2019 Drilling



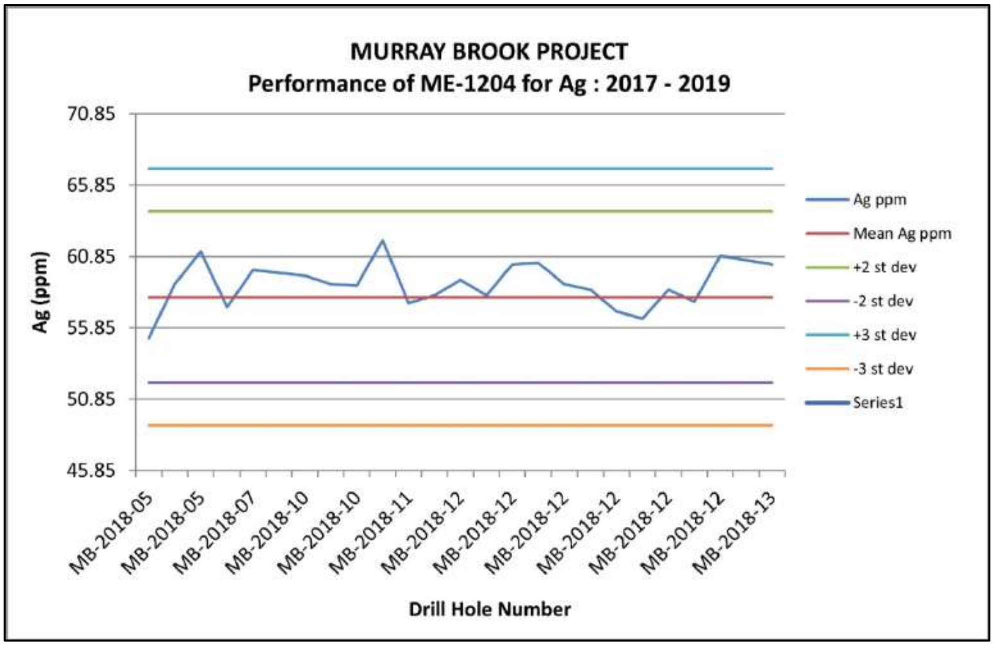
Source: P&E (2023).

Figure 11-12: Performance of CDN-ME-1204 CRM Au for 2017 to 2019 Drilling



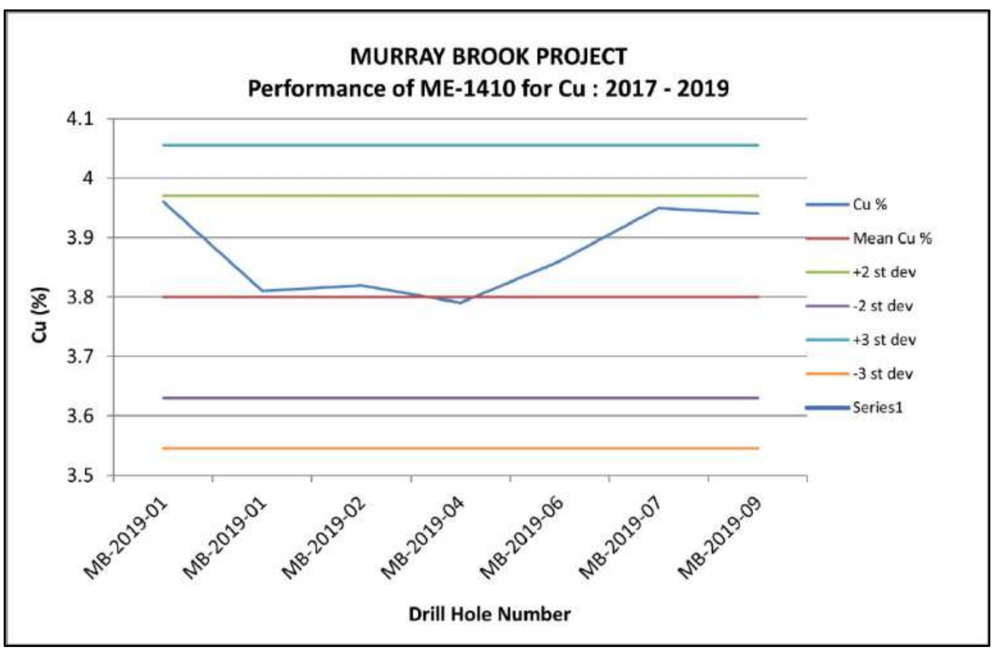
Source: P&E (2023).

Figure 11-13: Perform Performance of CDN-ME-1204 CRM Ag for 2017 to 2019 Drilling



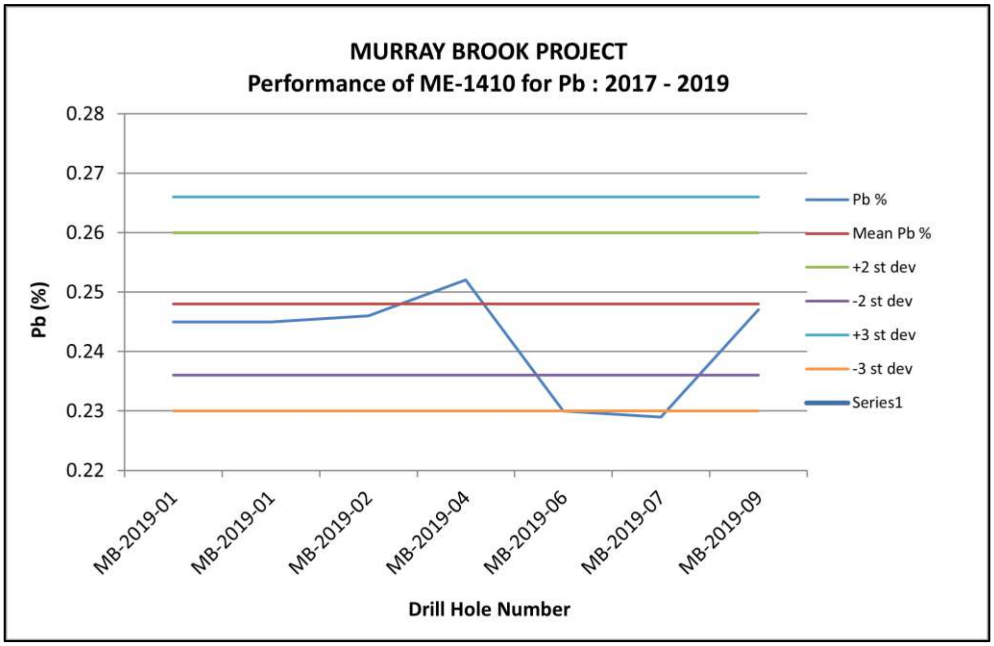
Source: P&E (2023).

Figure 11-14: Performance of CDN-ME-1410 CRM Cu for 2017 to 2019 Drilling



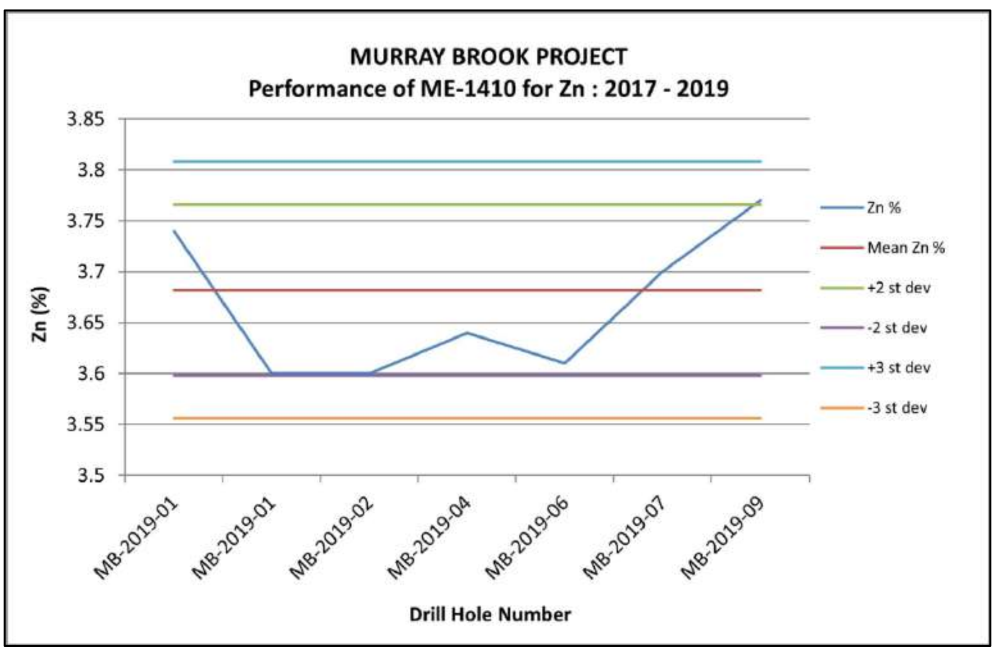
Source: P&E (2023).

Figure 11-15: Performance of CDN-ME-1410 CRM Pb for 2017 to 2019 Drilling



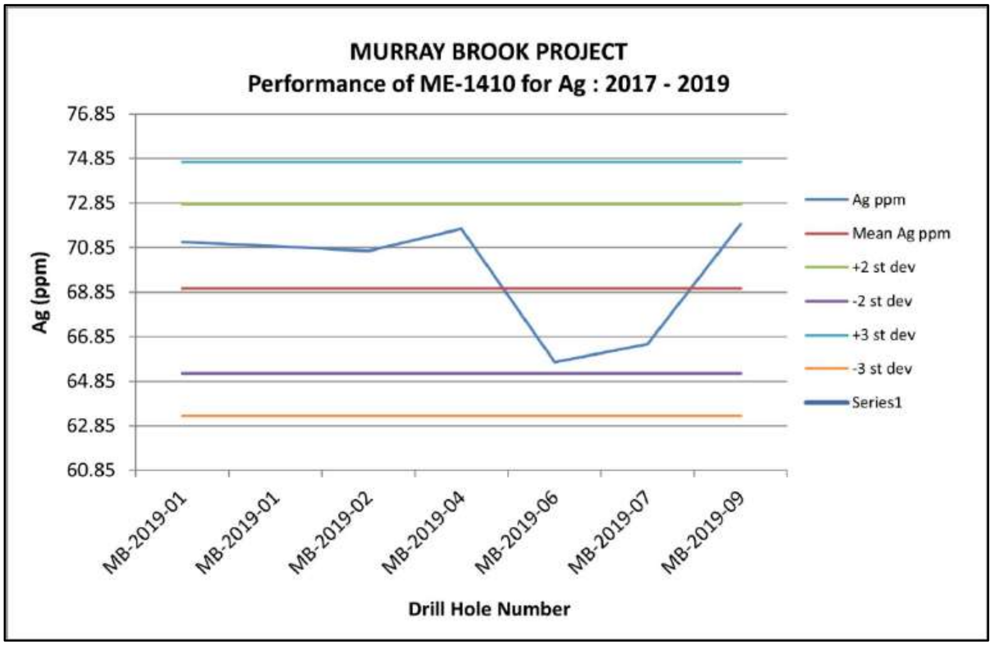
Source: P&E (2023).

Figure 11-16: Performance of CDN-ME-1410 CRM Zn for 2017 to 2019 Drilling



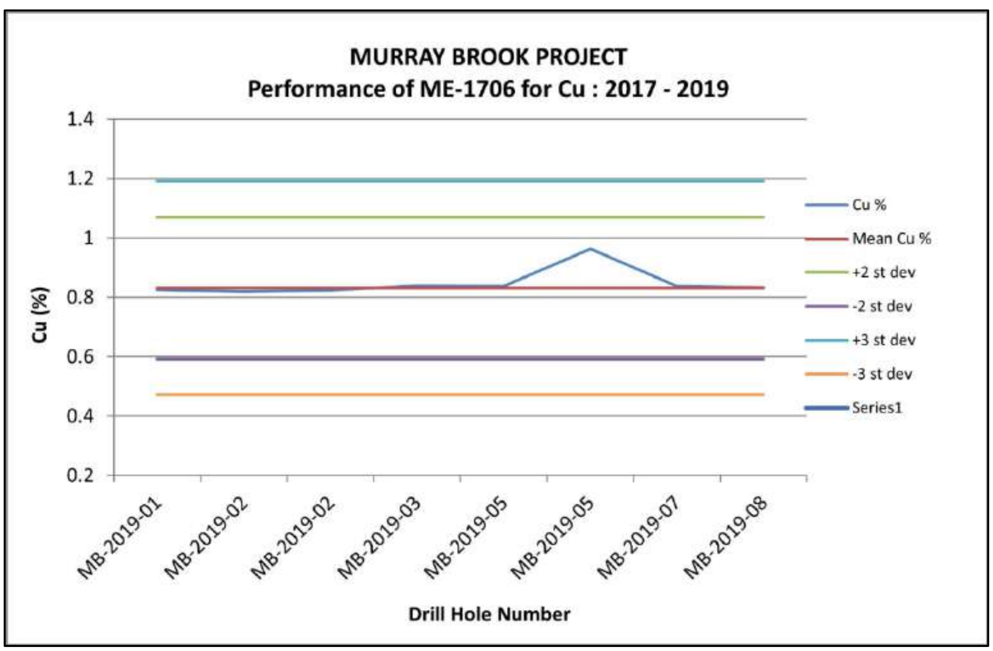
Source: P&E (2023).

Figure 11-17: Performance of CDN-ME-1410 CRM Ag for 2017 to 2019 Drilling



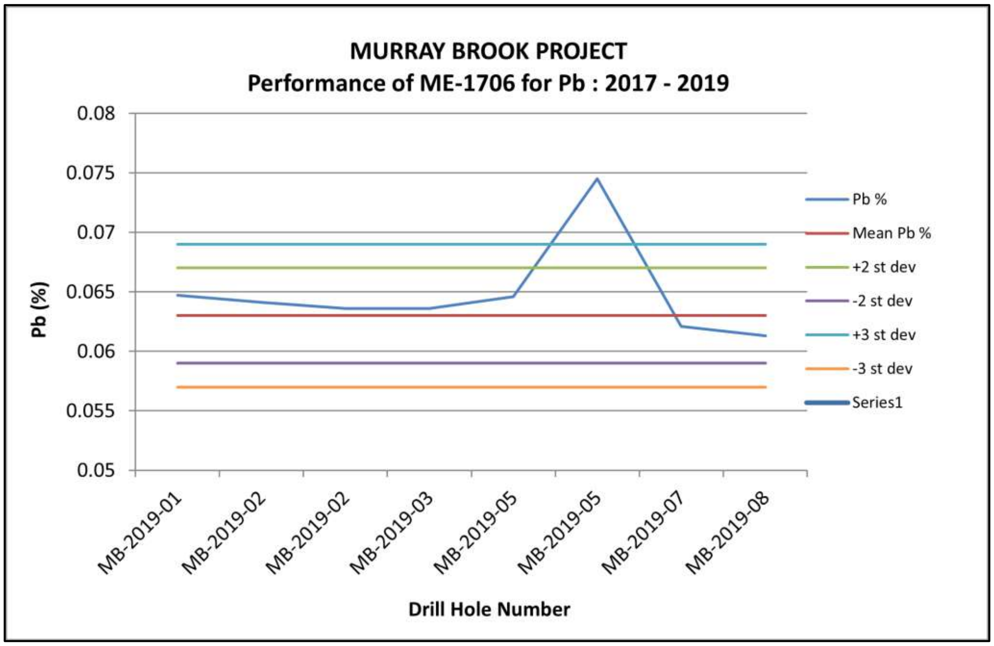
Source: P&E (2023).

Figure 11-18: Performance of CDN-ME-1706 CRM Cu for 2017 to 2019 Drilling



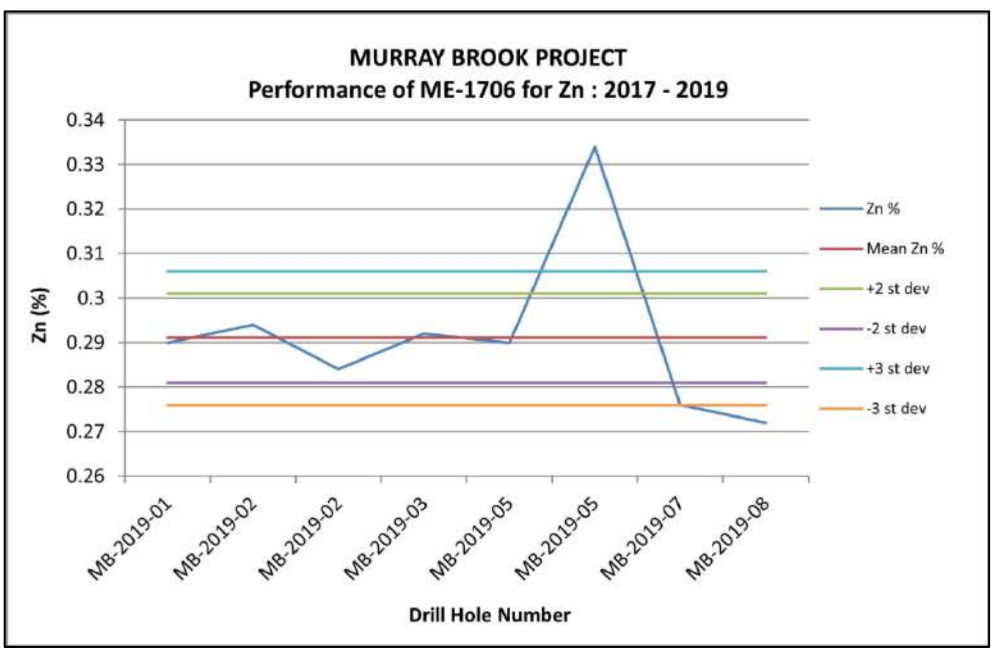
Source: P&E (2023).

Figure 11-19: Performance of CDN-ME-1706 CRM Pb for 2017 to 2019 Drilling



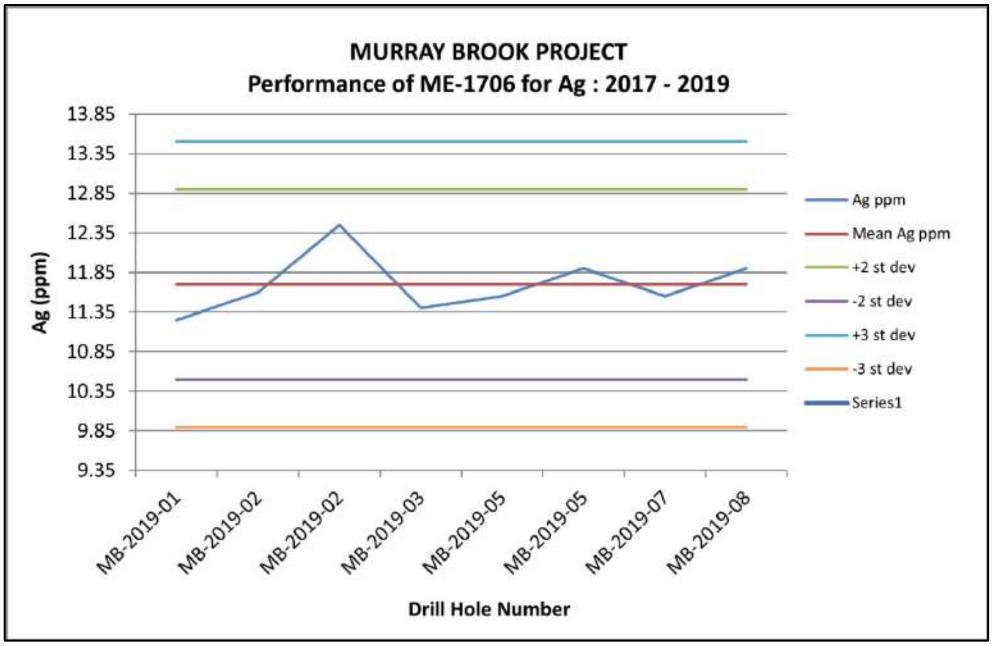
Source: P&E (2023).

Figure 11-20: Performance of CDN-ME-1706 CRM Zn for 2017 to 2019 Drilling



Source: P&E (2023).

Figure 11-21: Performance of CDN-ME-1706 CRM Ag for 2017 to 2019 Drilling



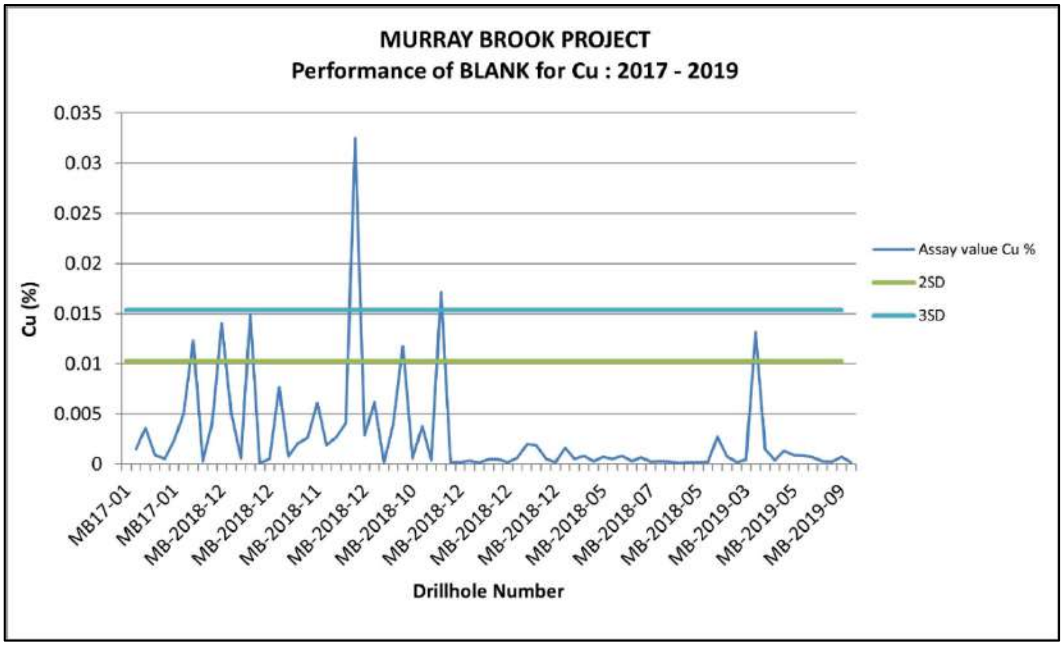
Source: P&E (2023).

11.5.2 Performance of Blanks

All blank data for copper, lead, zinc, silver, and gold were reviewed by the author. The blank material used was a decoration white stone, consisting of white marble devoid of significant levels of zinc, lead, copper, gold, or silver, and sourced from a local hardware store. There were 76 data points to examine. If the assayed value in the certificate was indicated as being less than detection limit, the value was assigned the value of one-half the detection limit for data treatment purposes. An upper tolerance limit of three times the standard deviation of all 76 data points was used.

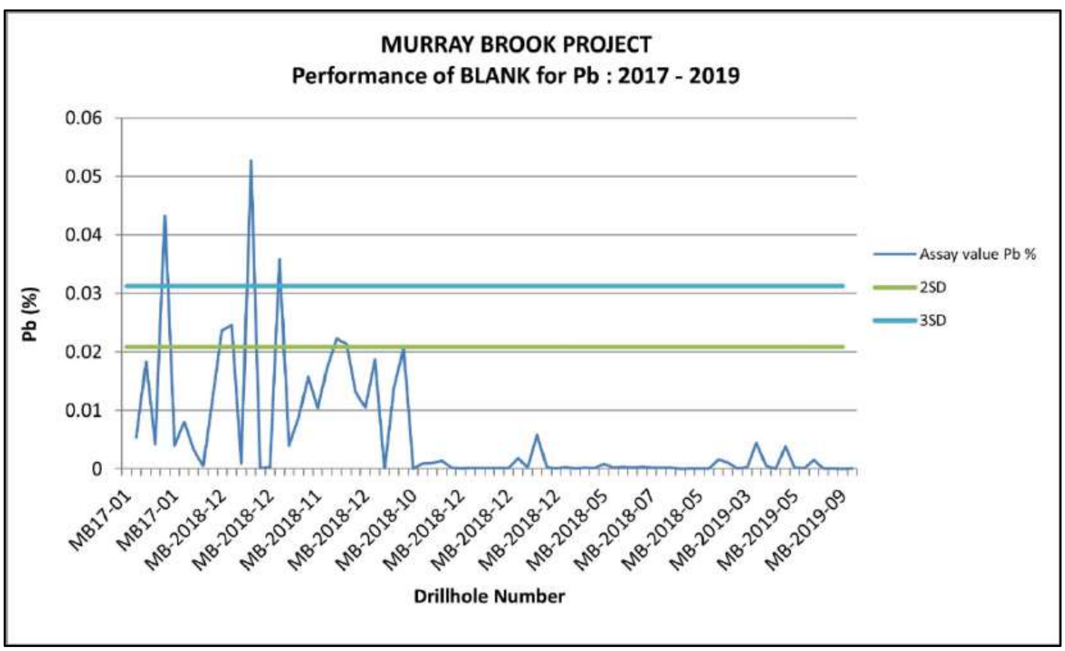
The vast majority of data plots at or below the set tolerance limits for all elements (Figures 11-22 to 11-26) and the carryover contamination observed in the data is considered acceptable. The author does not consider the very few outliers to be significant to the integrity of the data.

Figure 11-22: Performance of Blanks Cu for 2017 to 2019 Drilling



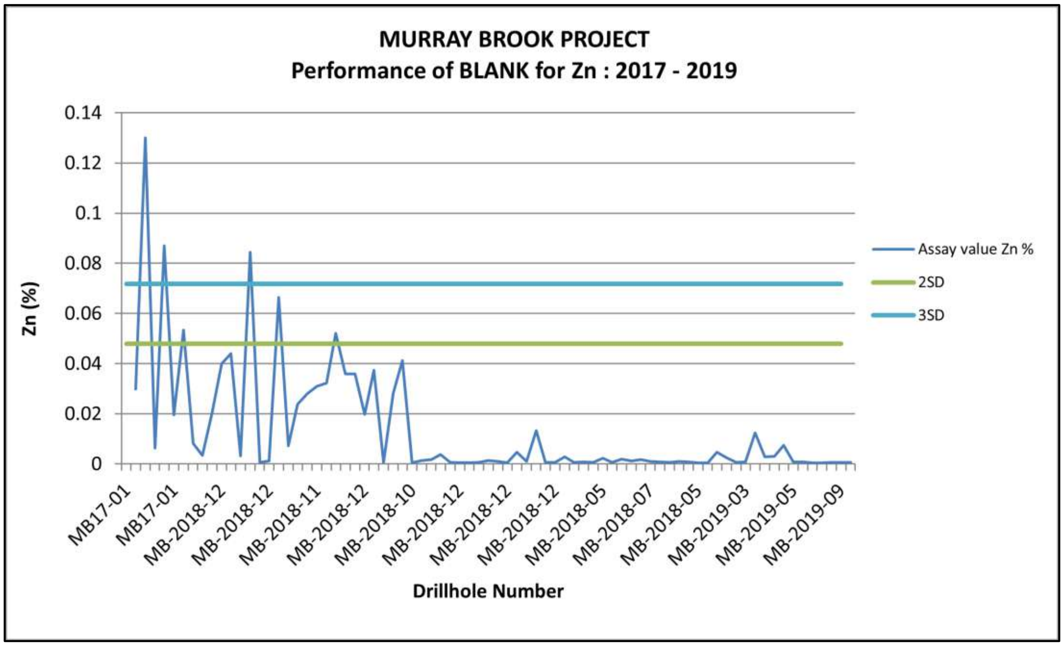
Source: P&E (2023).

Figure 11-23: Performance of Blanks Pb for 2017 to 2019 Drilling



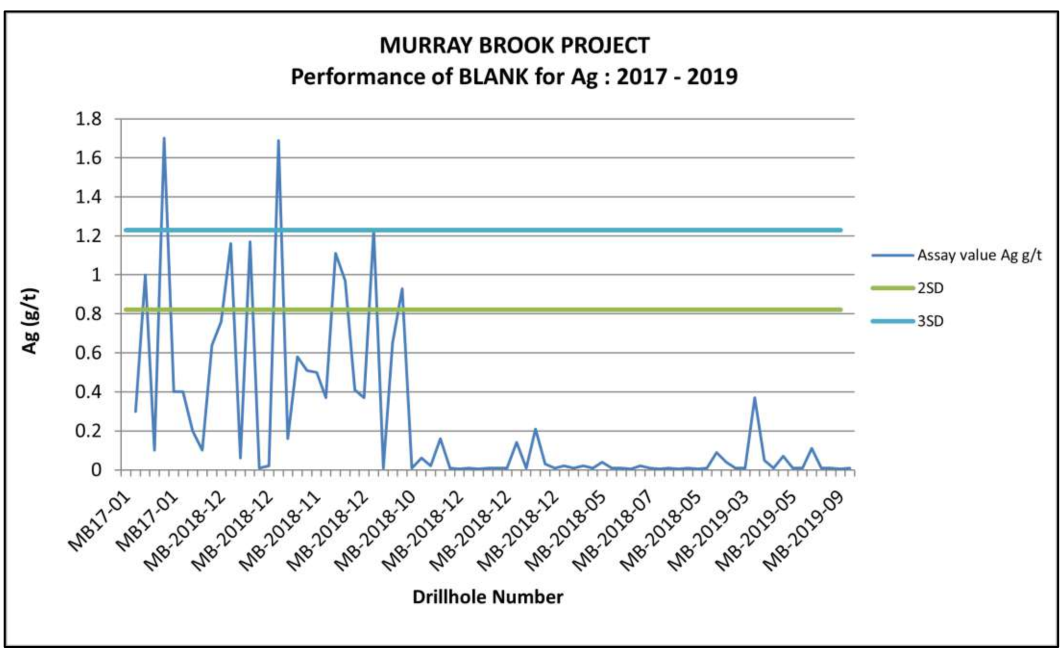
Source: P&E (2023).

Figure 11-24: Performance of Blanks Zn for 2017 to 2019 Drilling



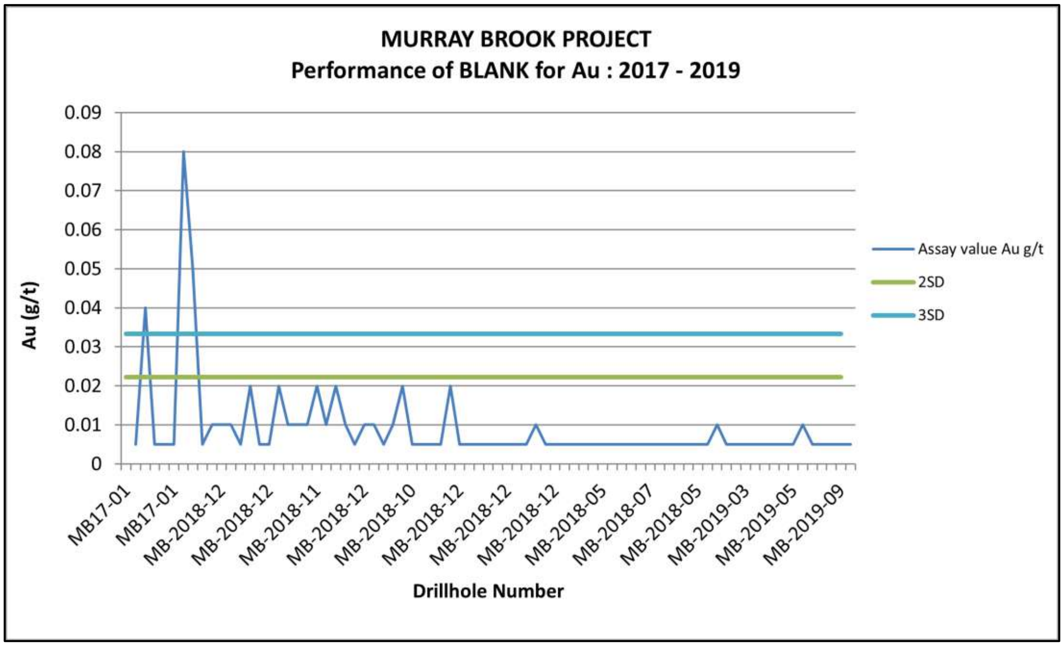
Source: P&E (2023).

Figure 11-25: Performance of Blanks Ag for 2017 to 2019 Drilling



Source: P&E (2023).

Figure 11-26: Performance of Blanks Au for 2017 to 2019 Drilling

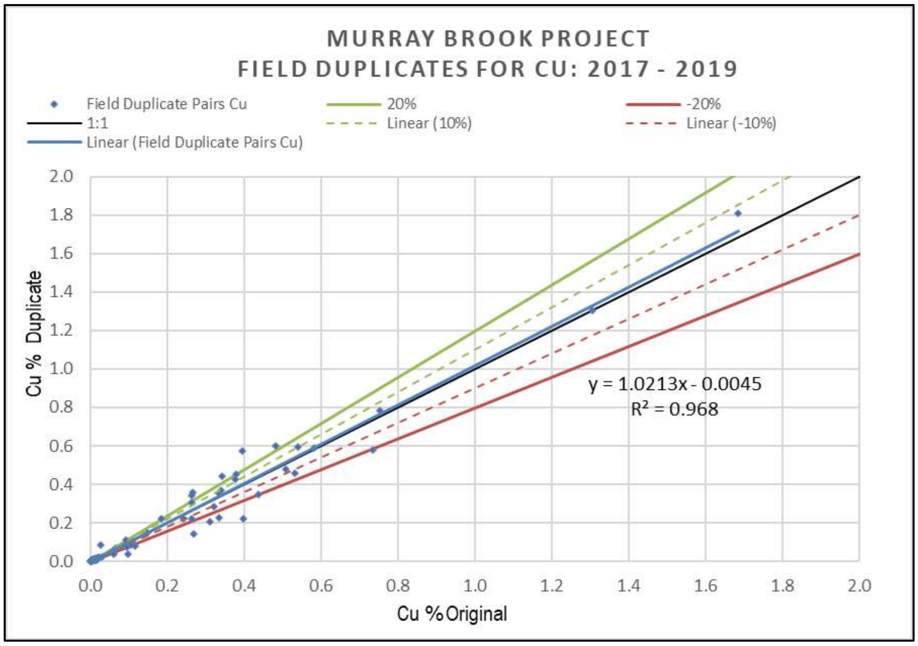


Source: P&E (2023).

11.5.3 Performance of Field Duplicates

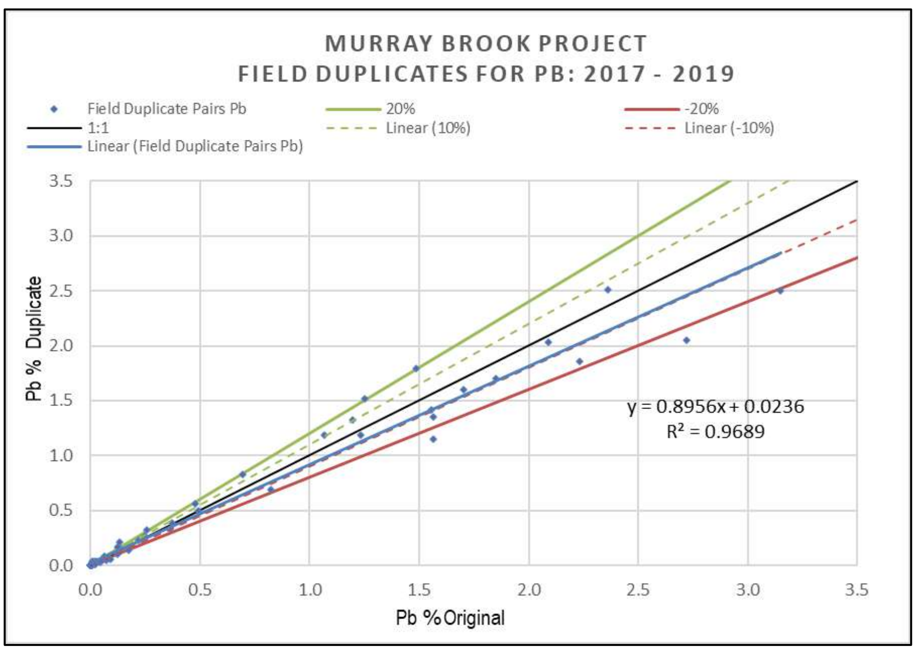
Field duplicate data for copper, lead, zinc, silver, and gold were examined for the 2017 to 2019 drill program at Murray Brook. There were 76 duplicate pairs in the dataset. Data were scatter-graphed (Figures 11-27 to 11-31) and found to have acceptable precision at the field level for all elements, with R-squared values ranging from 0.925 to 0.985 and the majority of the data plotting close to the 1:1 line.

Figure 11-27: Performance of Cu Field Duplicates for 2017 to 2019 Drilling



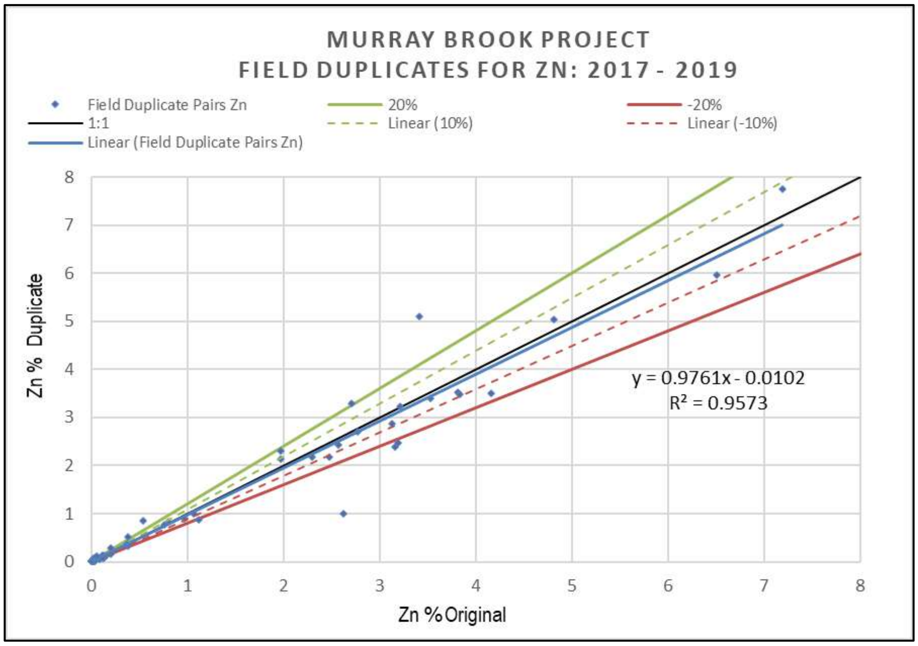
Source: P&E (2023).

Figure 11-28: Performance of Pb Field Duplicates for 2017 to 2019 Drilling



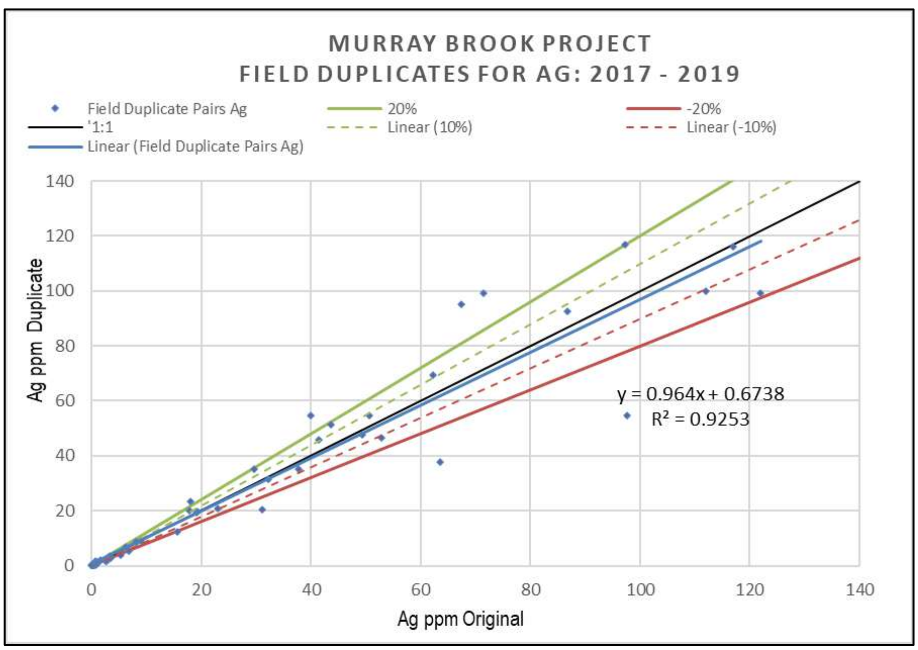
Source: P&E (2023).

Figure 11-29: Performance of Zn Field Duplicates for 2017 to 2019 Drilling



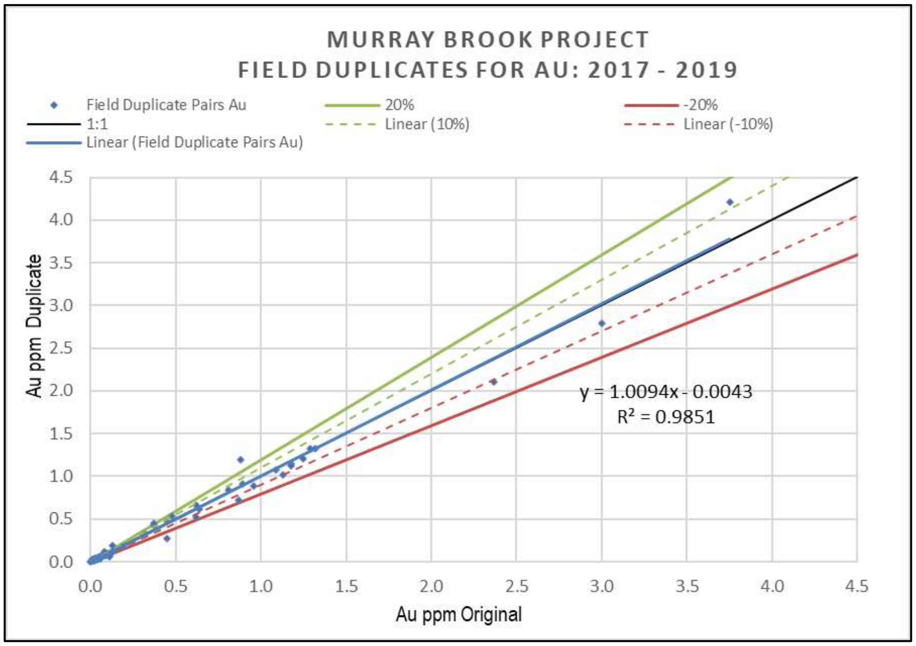
Source: P&E (2023).

Figure 11-30: Performance of Ag Field Duplicates for 2017 to 2019 Drilling



Source: P&E (2023).

Figure 11-31: Performance of Au Field Duplicates for 2017 to 2019 Drilling



Source: P&E (2023).

11.6 Conclusion

It is author's opinion that sample preparation, security, and analytical procedures for the 2010 to 2019 drill programs were adequate, and that the data are of good quality and satisfactory for use in the current mineral resource estimate. It is recommended that future drill core sampling at the project include the insertion and monitoring of suitable CRMs to monitor gold analyses and umpire assaying of 5% of all drill core samples.

12 DATA VERIFICATION

12.1 P&E Data Verification

12.1.1 April 2012 Assay Verification

The authors conducted verification of the Murray Brook Project drill hole assay database for copper, lead, zinc, silver, and gold by comparison of the database entries with assay laboratory certificates from TSL Laboratories. Verification of assay data entry was performed on 3,890 assay intervals and few very minor data entry errors were observed and corrected. The checked assays represented 100% of the data used in the 2012 Mineral Resource Estimate and approximately 64% of the entire 2012 database.

12.1.2 June 2013 Assay Verification

The authors conducted verification of the Murray Brook Project 2012 drill hole assay data for copper, lead, zinc, silver, and gold in June 2013. Approximately 96% (5,710 out of 5,980) of the assay data from 2012 were checked against the original laboratory certificates from TSL. During the verification process, it was found that some assays returning results below the laboratory detection limits were set to zero or half of detection limit in the database, which is acceptable for mineral resource estimation.

12.1.3 September 2023 Assay Verification

The authors again conducted verification of the Murray Brook Project drill hole assay data for copper, lead, zinc, silver, and gold in September 2023 by comparison of the database entries with assay certificates. Original digital assay laboratory certificates were downloaded directly from the ALS Webtrieve™ website by the authors in .xls (Microsoft Excel™ spreadsheet file) and .pdf (Portable Document Format file) format. Assay data from the 2017 to 2019 drilling undertaken at the Murray Brook Project were verified, with approximately 17% (483 out of 2,852 entries) of the overall data and approximately 21% (375 out of a total of 1,788 entries) of the constrained data verified. Very few minor discrepancies were encountered during the verification process, which the authors do not consider to be of material impact to the data for the mineral resource estimate.

12.1.4 Drill Hole Data Validation

Industry standard validation checks were carried out on the supplied databases, and minor corrections made where necessary. The authors typically validate a mineral resource database by checking for inconsistencies in naming conventions or analytical units, duplicate entries, interval, length, or distance values less than or equal to zero, blank or zero-value assay results, out-of-sequence intervals, intervals or distances greater than the reported drill hole length, inappropriate collar locations, and missing interval and coordinate fields.

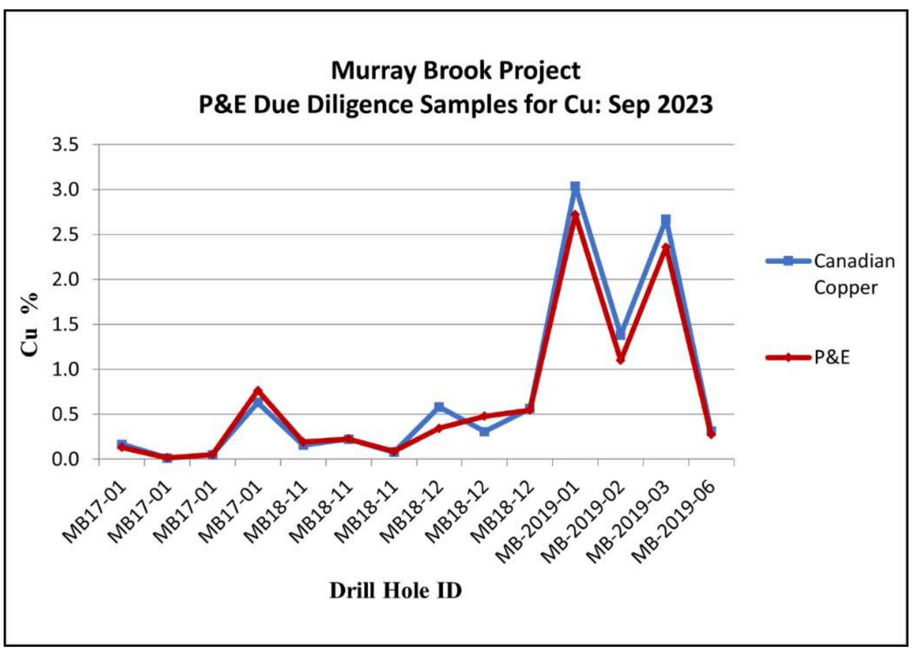
12.1.5 P&E September 2023 Site Visit and Independent Sampling

The Murray Brook Project was visited by Mr. Yungang Wu, P. Geo., of P&E, on September 9, 2023, for the purpose of completing a site visit that included drilling sites, outcrops, GPS location verifications, discussions, and due diligence drill core sampling.

Mr. Wu collected 14 drill core samples from seven diamond drill holes during the 2023 site visit. All samples were selected from holes drilled between 2017 and 2019. A range of high-, medium-, and low-grade samples were selected from the stored drill core. Samples were collected by taking a half drill core, with no drill core remaining in the drill core box. Individual samples were placed in plastic bags with a uniquely numbered tag, after which all samples were collectively placed in a larger bag and delivered by Mr. Wu to the Activation Laboratories Ltd., facility in Ancaster, Ontario, for analysis. Samples at Actlabs were analysed for copper, lead, and zinc by aqua regia digestion with ICP-OES finish and for gold and silver by fire assay with a gravimetric finish. Bulk density determinations were measured on all drill core samples by water displacement.

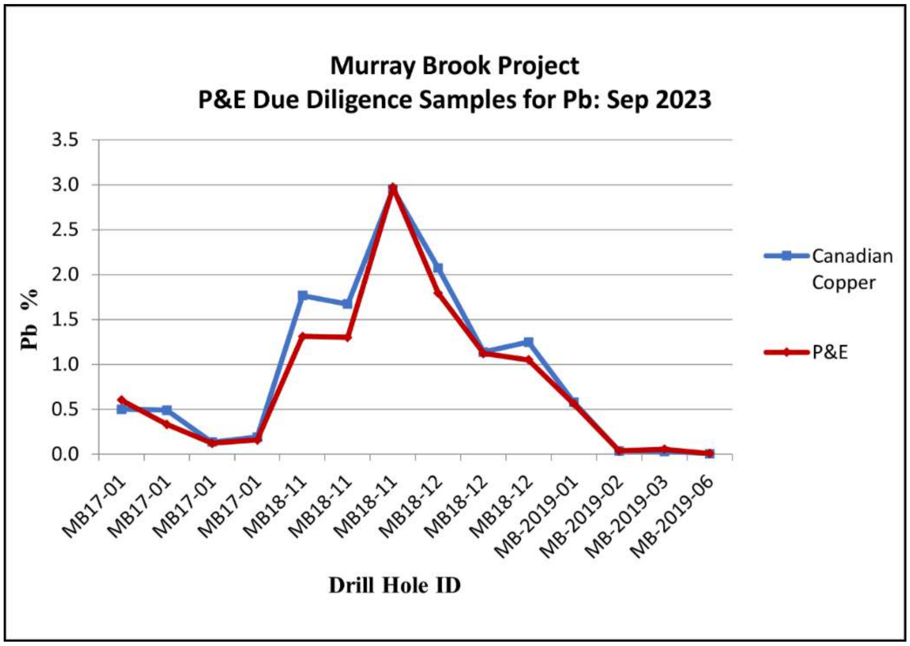
The quality system at Actlabs is accredited to international quality standards through ISO/IEC 17025:2017 and ISO 9001:2015. The accreditation program includes ongoing audits, which verify the QA system and all applicable registered test methods. Actlabs is also accredited by Health Canada. Actlabs is independent of Canadian Copper. Results of the Murray Brook 2023 site visit verification samples for copper, lead, zinc, silver, and gold are presented in Figures 12-1 through 12-5.

Figure 12-1: Results of September 2023 Cu Verification Sampling



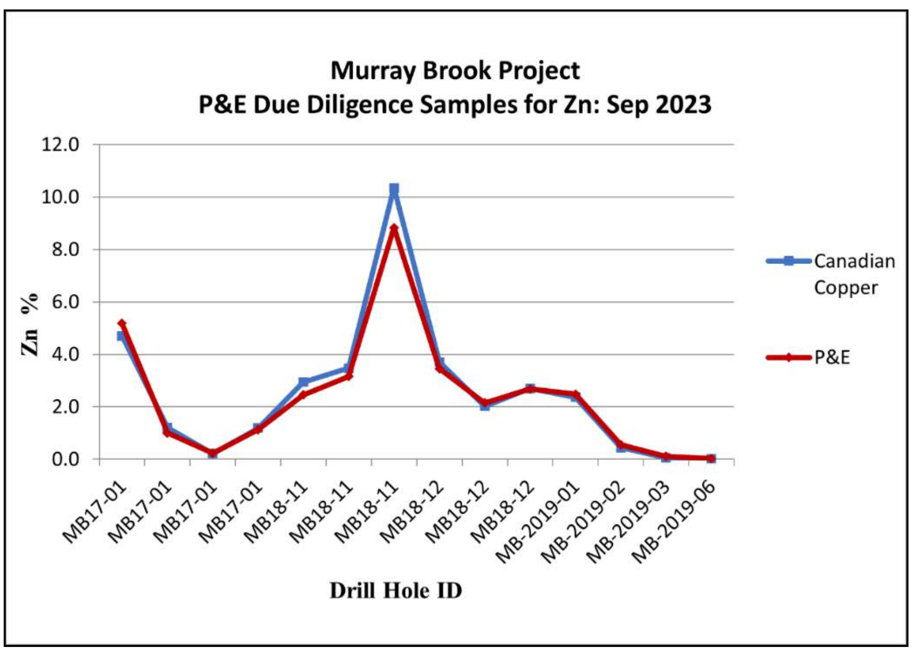
Source: P&E (2023)

Figure 12-2: Results of September 2023 Pb Verification Sampling



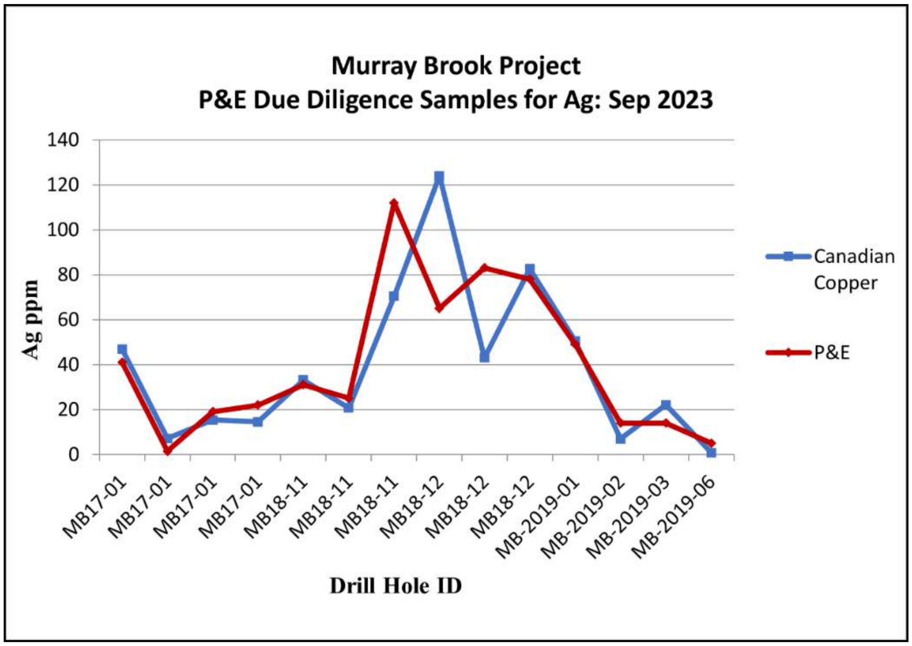
Source: P&E (2023)

Figure 12-3: Results of September 2023 Zn Verification Sampling



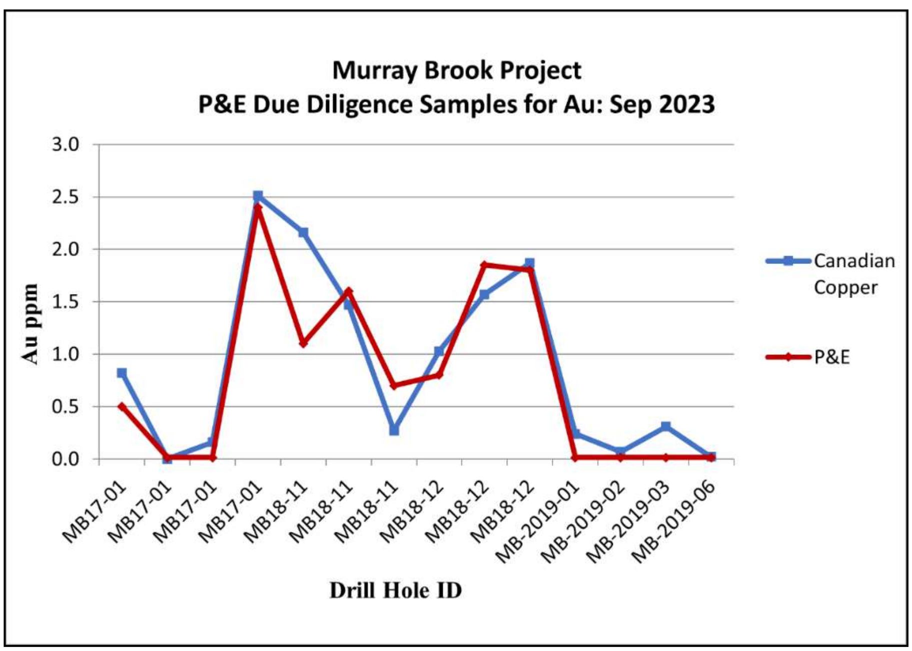
Source: P&E (2023)

Figure 12-4: Results of September 2023 Ag Verification Sampling



Source: P&E (2023)

Figure 12-5: Results of September 2023 Au Verification Sampling



Source: P&E (2023)

12.2 Adequacy of Data

Verification of the Murray Brook Project data, used for the current mineral resource estimate, has been undertaken by the authors, including multiple site visits, due diligence sampling, verification of drilling assay data from electronic assay files, and assessment of the available QA/QC data. The authors consider that there is good correlation between the copper, lead, zinc, silver, and gold assay values in Canadian Copper's database and the independent verification samples collected by the authors and analysed at Actlabs. In the authors' opinion, the data are of good quality and appropriate for use in the current mineral resource estimate.

13 MINERAL PROCESSING AND METALLURGICAL TESTING

13.1 Introduction

The Murray Brook deposit is a typical volcanogenic massive sulphide (VMS) deposit, similar to other historical deposits that have been mined in New Brunswick, such as the Heathe Steele mine, the Brunswick Mining and Smelting mine and, more recently and closer to Murray Brook, the Caribou mine. VMS deposits are characterized by iron sulphide minerals (either as pyrite, marcasite, arsenopyrite) making up the bulk (>75%) of the non-economic minerals and a mixture of copper-bearing (as chalcopyrite, covellite, tetrahedrite, tennantite), lead-bearing (as galena), and/or zinc-bearing (as sphalerite) minerals as the economic minerals. At times, there might be one, two, or all of these metals present in a quantity sufficient to justify their concurrent recoveries. Mineralogical limitations, mostly in terms of average grain size of the economic minerals and their assemblages, may further limit either the recovery or the concentrate grade achievable for one or more of these metals.

Three major metallurgical testwork campaigns were carried out on the Murray Brook property between 2012 and 2025, as follows:

- 2012, carried out by the Research and Productivity Council (RPC), Fredericton, New Brunswick; commissioned by Votorantim Metals Canada Inc. – Mineralogy; rod, ball mill work indices; open-circuit batch flotation tests; locked cycle tests.
- 2019, carried out by the Research and Productivity Council, Fredericton, New Brunswick; commissioned by Trevali Mining Corporation – Mineralogy; SMC, rod, ball mill work indices; heavy media separation test; open-circuit batch flotation tests; locked cycle tests; mineralized material aging effect on flotation response; static settling tests on zinc concentrate; cyanidation of cleaner tails; copper leaching; ABA on flotation tails.
- 2025 carried out by SGS, Lakefield, Ontario; commissioned by Canadian Copper – SMC, ball mill work indices; open-circuit batch flotation tests; liberation analysis on selected flotation products.

These campaigns focused on developing a processing flowsheet for sulphide-bearing material tailored to the production of separate base metal concentrates. In all cases, zinc was consistently present in the samples, as was either copper and/or lead. These three metals were seldom found concurrently in potentially economic amounts within the samples of the earlier testwork programs. However, the current geological block model data shows that approximately 80% of the resources are presenting these three metals concurrently and in sufficient abundance to pursue their recovery simultaneously. Furthermore, there is no plan to mine selectively lead-zinc and copper-zinc materials as plant feed.

The objective of the testwork programs was to establish a processing flowsheet using froth flotation to produce concentrates amenable to smelting by others. Testing involved establishing the primary grind size required to achieve a high recovery of the economic metals in the initial (e.g., rougher) concentrates and then establish the regrind product size targets needed to achieve proper upgrading of the rougher concentrates into saleable final concentrates while maximizing metal recoveries. Various circuit configurations, reagent regimes, and addition rates were tested during such development work. Specific plant scale-up parameters were also investigated, such as material hardness to size grinding mills and flotation times per stage, to design an industrial-scale flotation circuit. The use of mineralogical analysis, including mineral liberation analysis (MLA), was incorporated in the test programs to assess the degree of mineral liberation achieved at a different target grind and regrind sizes and to explain the deportment of minerals into the various flotation circuit products.

The early work outlined the presence of an oxidized layer of material sitting above the sulphide deposit. Limited testwork efforts were expended to resolve the issue of activated pyrite, which thwarted any efforts at upgrading rougher concentrates which contained a large weight proportion of the feed samples floated. Any samples displaying such a response were typically set aside and the flowsheet development work proceeded with non-oxidized samples. Consequently, the material within this oxidized layer is excluded from consideration in the economic analysis presented in this report. When the nearby, existing, Caribou processing plant became available and Canadian Copper secured an option to acquire it, an adapted process flowsheet was developed for the 2025 testwork campaign.

13.2 Legacy Metallurgical Testwork

13.2.1 2012 RPC Testwork Program Highlights

The 2012 RPC testwork campaign was commissioned by Votorantim Metals Canada Inc. (VMC) and was meant to test materials from the sulphide zone at a scoping level of detail. The core samples tested were taken from three contemporary diamond drill holes, and the total mineralized intercepts were either tested as a whole or divided to create sub-samples (top, middle, bottom of the boreholes) and combined in equal proportions to create a composite sample of the three holes. Table 13-1 presents the characteristics of the core samples.

Table 13-1: RPC 2012 Testwork Program – Samples Definition

Hole ID	From	To	Length	Fe (%)	Cu (%)	Pb (%)	Zn (%)	Au (ppm)	Ag (ppm)
Votorantim Average DH Assay									
MB-2012-121 – Average Top	13.5	24.0	4.6		1.03	0.76	0.95	0.88	36
MB-2012-121 – Average Bottom	24.0	150.6	126.6		0.54	0.83	1.80	0.88	42
MB-2012-124 – Average Top	25.7	45.0	19.3		1.07	1.62	5.11	0.33	80
MB-2012-124 – Average Middle	45.0	56.0	11.0		0.14	1.72	4.17	0.17	62
MB-2012-124 – Average Bottom	56.0	160.0	104.0		0.19	0.74	2.62	0.25	34
MB-2012-132 – Average	86.0	218.0	132.0		1.15	1.64	5.27	0.58	61
Metallurgical Sample Assay									
MB-2012-121				37.55	0.53	0.96	2.02	0.72	32
MB-2012-124				39.38	0.18	0.78	2.77	0.23	27
MB-2012-132				37.41	0.16	1.39	4.34	0.59	53
Composite (121,124,132)				39.44	0.33	1.14	3.42	*0.51	47

Note: *Calculated value. Source: RPC Final Report; Nov. 17, 2012; p. 8.

13.2.1.1 Mineralogy

Mineralogical evaluation through SEM-EDS indicated that pyrite comprised the bulk of all material (65% to 80%), with some quartz, chlorite, muscovite, and carbonates making up the balance of the gangue minerals. Chalcopyrite was rare and occurred as interstitial to pyrite. Hole ME-2012-124 also had covellite and minor boulangerite. Sphalerite and galena occurred as interstitial, inclusions, fine veinlets, and attachments to pyrite. Pyrite is typically the coarser-grained mineral while most of the target mineral occurrences are as <20 µm grains, except for sphalerite, interstitial with pyrite, with some 50 to 100 µm fractions. Silver is found within tetrahedrite. Arsenopyrite and some bismuth-antimony (Bi-Sb) sulphosalts (bournonite) were seen at times. No visible gold was detected.

13.2.1.2 Material Hardness

Only one composite sample, which was made up of one-third of the material from each of the three holes (referenced as the "three-hole composite"), was tested for material hardness: a rod mill work index (RWi) of 14.6 kWh/t and a ball mill work index (BWi) of 10.7 kWh/t put this material in the average and soft regions of the RWi and BWi databases, respectively.

13.2.1.3 Flotation Testwork

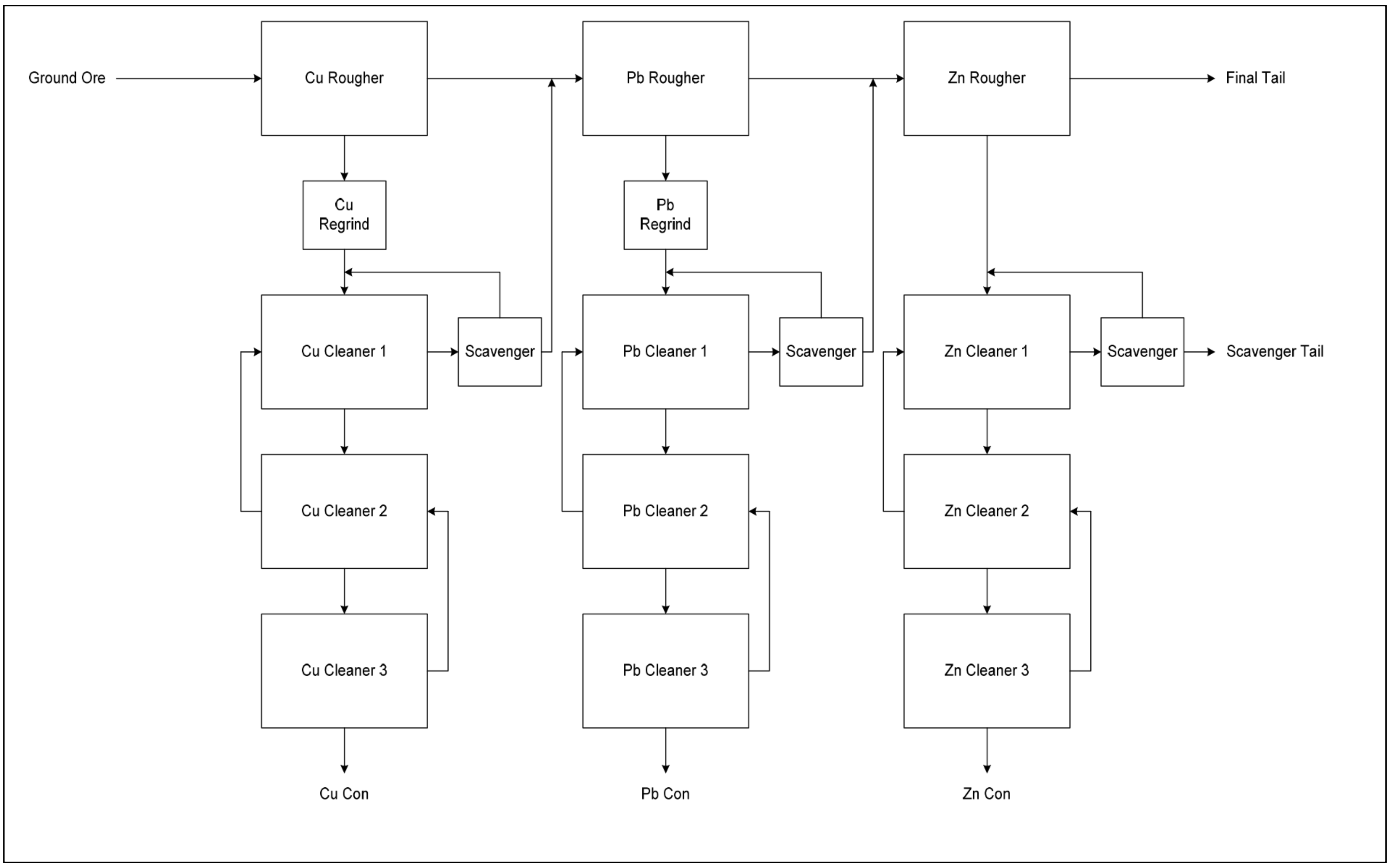
The flotation testwork program rapidly dropped material from the top section of hole MB-2012-121 and top and middle section of hole MB-2012-124, as these proved to be oxidized, based on the flotation response observed. Attempts at introducing specific reagent regimes meant to deal with this issue failed to counteract the in-situ activation of the zinc-bearing sphalerite present.

As material from only three diamond drill holes was involved in the testwork program, the representativity of the samples was limited by their sourcing and was insufficient to properly map the whole resource envelope. The feed grades involved are also not aligned to the resources' averages, comparing 0.27% Cu, 1.15% Pb and 3.42% Zn for the locked cycle test (LCT) sample against the measured and indicated resources at 0.45% Cu, 0.91% Pb, and 2.49% Zn.

This report presents the groundwork for establishing a flowsheet and reagent regime to process the polymetallic material under a sequential flotation scheme, where copper, lead, and zinc minerals are floated in sequence with the goal of producing three saleable concentrates. The circuit configuration retained for locked cycle tests is typical for such a sequential scheme, with each metal having its own rougher circuit followed by a regrinding stage (except for zinc) feeding into three counter-current stages of cleaners complemented by a cleaner-scavenger stage. Figure 13-1 offers a schematic view of this circuit configuration.

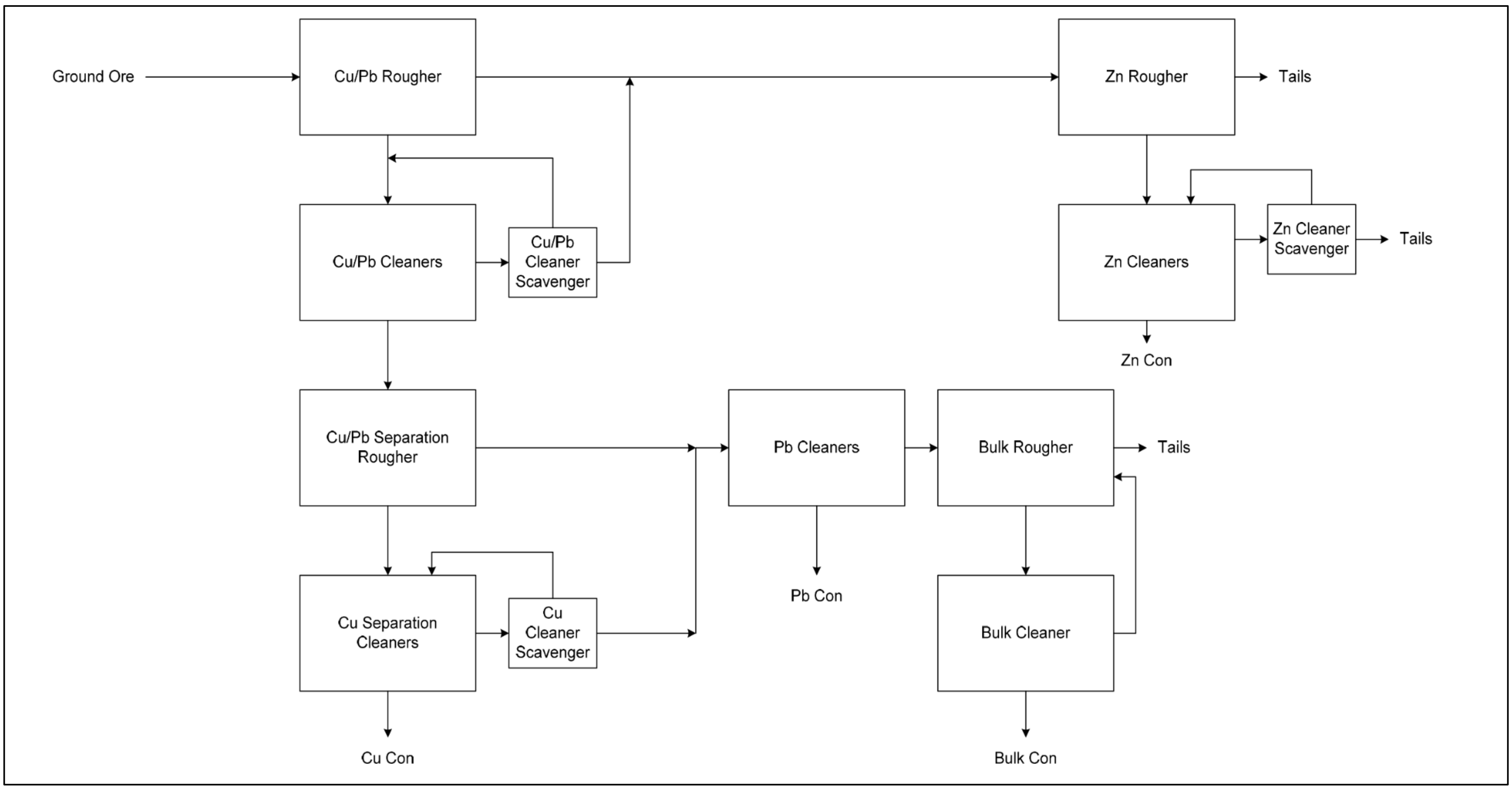
Another classical processing approach to produce three concentrates involves the flotation of a bulk copper-lead concentrate followed by zinc flotation. The bulk concentrate is divided by depressing either copper or lead (Figure 13-2). This approach was initially attempted but deemed too complex, even though the report states that it was the path followed by Heath Steele and Brunswick Mining & Smelting which mined similar VMS deposits with fine-grained sulphide minerals in New Brunswick.

Figure 13-1: RPC 2012 Testwork Program – Sequential Flotation Circuit Configuration



Source: RPC Final Report; Nov 17, 2012; p.16.

Figure 13-2: RPC 2012 Testwork Program – Bulk Cu-Pb + Zn Flotation Circuit Configuration



Source: RPC Final Report; Nov 17, 2012; p.19.

13.2.1.4 Locked Cycle Tests

The testwork program went on to optimize first the rougher results and then the results for the cleaning circuits. The resulting circuit operating parameters (flotation time, reagent types, and addition points and rates) were applied to the completion of a locked cycle test, with the three-hole composite. Because of limitations related to the available laboratory equipment sizes, small individual batches were ground and processed through rougher flotation over a few weeks to accumulate material from 27 roughing stages before going through the locked cycled cleaning stages. This approach permitted oxidation of the sulphide-rich slurries to occur during this "rest period" and may not have delivered optimal results, to no fault of the sample or approach.

Table 13-2 presents the LCT metallurgical balance, with recoveries per stage, except when indicated as overall values.

Per Table 13-2, the LCT delivered a marginal copper concentrate grade at 17.45% Cu and limited overall recovery of 51.4%, although the behaviour of the cleaning circuit, with 80.8% of the copper contained in the rougher concentrate, was good. The low feed grade of 0.27% Cu is certainly challenging. Good rejection of the other metals was achieved though: only 4.5% and 1.7% of the overall lead and zinc, respectively, were deported to the final copper concentrate. Only 17.5% Ag and 1.8% Au reported to this concentrate, leaving the resulting gold grade in the concentrate, at 1.05 g/t Au, near the typical payable limit threshold seen in smelting contracts.

The lead circuit yielded an acceptable concentrate grade of 50.3% Pb, but at a low overall recovery of 36.6% Pb. In this case, both the roughing circuit, at 58.3% recovery, and the cleaning circuit, at 62.7%, contributed to this overall low lead value. Again, zinc rejection was excellent, with only 1.4% of the overall zinc content reporting to the lead concentrate. Gold and silver recoveries were again limited, with 17.5% Au and 1.8% Ag making their way into the lead concentrate.

The zinc circuit fared much better than the copper and lead circuits. With a final concentrate at 53.78% Zn and an overall recovery of 88.8%, this circuit performance was good, providing high stage recoveries at both the roughing stage (91.9% recovery) and the cleaning stage (96.6% recovery).

The details of the overall grinding (in a rod mill, with stainless-steel media), flotation times, regrinding, and reagent addition regime, as used for the LCT, are presented in Table 13-3 to Table 13-6.

As seen in Table 13-3, the testwork adopted a primary grind target P_{80} of 30 µm while those for the copper and lead cleaners regrinding stages were set at 10 µm. Zinc also benefited from a regrinding stage, yielding about 20 µm as P_{80}. The reagent regime retained for the LCT relied on the use of soda ash as the pH modifier in the copper and lead circuits while lime is used for this purpose within the zinc circuit. Depression of pyrite is mostly achieved with cyanide and sodium metabisulphite (SMBS) ahead of the first roughing stage while elevating the pH within the cleaners, assisted by SMBS in the copper cleaners and cyanide in the lead cleaners, serves this purpose. Zinc sulphate is used early in the circuit and in each of the copper and lead cleaning circuits, to help maintain zinc depressed. Three collectors (Aerophine 3418A, Aerofloat 5100, and potassium ethyl xanthate (PEX)) were deployed at times for duties not typical for some of these products (in particular, A3418A for zinc).

Table 13-2: RPC 2012 Testwork Program – Locked Cycle Test Results

Description	Sample/Circuit	Mass (g)	Mass Distribution (%)	Assays						Distribution Ratio (%)					
				Fe (%)	Cu (%)	Pb (%)	Zn (%)	Ag (g/t)	Au (g/t)	Fe	Cu	Pb	Zn	Ag	Au
Hole 121	Bottom	15,908	33.3	36.16	0.48	0.95	1.88	42	0.906	32.7	59.7	27.4	18.4	31.0	51.5
Hole 124	Bottom	15,908	33.3	39.55	0.19	0.82	2.91	35	0.263	35.8	23.3	23.8	28.4	25.6	14.9
Hole 132	Whole	15,908	33.3	34.92	0.14	1.68	5.46	59	0.591	31.6	17.0	48.8	53.2	43.5	33.6
Average		**47,724**	**100.0**	**36.87**	**0.27**	**1.15**	**3.42**	**45**	**0.590**	**100.0**	**100.0**	**100.0**	**100.0**	**100.0**	**100.0**
Cu Rougher Recovery	Rougher Concentrate	2,740	5.7	34.15	3.43	2.43	5.10	282	0.839	5.4	63.6	12.0	9.0	37.6	9.8
Cu Cleaner Recovery	Final Cu Cleaner 3 Concentrate	416	0.9	28.77	17.45	6.16	6.04	591	1.051	12.7	80.8	37.8	19.1	33.3	20.2
	Cu Cleaner Tails	2,324	4.9	34.79	0.82	1.92	4.61	203	0.800	4.7	12.9	8.0	6.9	23.0	7.9
Overall Copper Recovery										**0.7**	**51.4**	**4.5**	**1.7**	**12.5**	**2.0**
	Cu Cleaner Tails	2,324	4.9	34.79	0.82	1.92	4.61	203	0.800	4.7	12.9	8.0	6.9	23.0	7.9
Pb Rougher Recovery	Rougher Concentrate	4,270	8.9	35.39	0.37	7.58	5.28	135	0.714	8.7	10.6	58.3	14.6	28.1	13.0
Pb Cleaner Recovery	Final Pb Cleaner 4 Concentrate	405	0.8	13.25	2.40	50.30	5.27	833	0.923	3.5	67.9	62.7	9.5	62.0	14.2
	Pb Cleaner Tails	3,864	8.1	38.65	0.12	3.14	5.26	54	0.502	96.5	32.1	37.3	90.5	38.0	85.8
Overall Lead Recovery										**0.3**	**7.2**	**36.6**	**1.4**	**17.5**	**1.8**
Zn Rougher - Cu Cleaner Tails	Zn Rougher Concentrate (Cu CT)	1,168	2.4	30.03	1.54	1.66	8.78	369	0.805	2.0	12.2	3.5	6.6	21.0	4.0
Zn Rougher - Pb Cleaner Tails	Zn Rougher Concentrate (Pb CT)	1,687	3.5	32.94	0.26	2.72	11.99	82	0.564	3.2	3.0	8.3	13.1	6.7	4.1
Zinc	Zn Rougher Concentrate (Pb RT)	8,599	18.0	34.82	0.21	0.73	13.01	40	0.659	17.2	12.4	11.2	72.2	16.9	24.2
Total Zn Rougher Recovery	Total Rougher	11,454	24.0	34.05	0.36	1.11	12.43	80	0.660	22.5	27.6	23.0	91.9	44.6	32.3
Zn Cleaner Recovery	Final Zn Cleaner 4 Con	2,913	6.1	10.53	0.48	1.08	53.78	95	0.360	7.7	57.2	35.9	96.6	56.7	16.9
	Zn Cleaner Tails	8,541	17.9	43.02	0.12	0.66	0.64	25	0.610	92.3	42.8	64.1	3.4	43.3	83.1
	Zn Rougher Tail (Cu CT)	1,156	2.4	35.05	0.17	1.97	0.73	21	0.385	2.3	1.3	4.1	0.5	1.2	1.9
	Zn Rougher Tail (Pb CT)	2,178	4.6	39.83	0.11	2.28	0.73	21	0.473	5.0	1.6	8.9	1.0	2.2	4.4
Calc. Head		47,724	100.0	36.37	0.31	1.16	3.25	43	0.491	100.0	100.0	100.0	100.0	100.0	100.0
Head Assays				36.02	0.29	1.17	3.12	43	0.595						
Overall Zinc Recovery										**1.7**	**15.8**	**8.3**	**88.8**	**25.3**	**5.5**

Source: RPC Final Report; Nov 17, 2012; p.4.

Table 13-3: RPC 2012 Testwork Program – Locked Cycle Test on Three-Hole Composite: Flotation Times and Reagent Regime

Test No.	Test Stage	Aeration Time (min)	Cond. Time (min)	pH		Reagent Dosage (g/t)								Float (min)
				Modifier	Range	PEX	3418A	5100	MIBC	$ZnSO_4$	NaCN	SMBS	$CuSO_4$	
LC1 28-29 µm	Cu	10	5+2+1	Na_2CO_3	8 to 8.5			10	15	400		800		5
	Pb		5+2+1	Na_2CO_3	9 to 9.5	8	50		10	400	40			3
	Zn1		5+2+1	$Ca(OH)_2$	10 to 10.5	8	50		10				400	3
	Zn2		2+1	$Ca(OH)_2$	10 to 10.5	4	25		5					2

Table 13-4: RPC 2012– Locked Cycle Test on Three-Hole Composite: Flotation Times and Reagent Regime – Cu Cleaners

Test No.	Concentrate Mass (g)	Regrind Time (min)	Test Stage	Conditioning Time (min)	pH		Reagent Dosage (g/t)				Float Time (min)	Rotation (rpm)	Cell Size (L)
					Modifier	Range	$ZnSO_4$	5100	MIBC	SMBS			
Cu LC 1-8	330 + Scavenger Concentrate	70 (P_{80}:10-11µm)	Cleaner 1	5+1	Na_2CO_3	8-8.5	800		1	400	30	900	2
			Scavenger	2+1	Na_2CO_3	8-8.5		5	5		10	1300	2
			Cleaner 2	1	Na_2CO_3	8-8.5			10		7	900	2
			Cleaner 3	1	Na_2CO_3	8-8.5			0		5	900	2

Source: RPC Final Report; Nov 17, 2012; p.41-44.

Table 13-5: RPC 2012– Locked Cycle Test on Three-Hole Composite: Flotation Times and Reagent Regime – Pb Cleaners

Test No.	Concentrate Mass (g)	Regrind Time	Stage	Conditioning Time (min)	pH		Reagent Dosage (g/t)				Float Time (min)	Revolution (rpm)	Cell Size (L)
					Modifier	Range	$ZnSO_4$	3418	MIBC	NaCN			
Pb LC 1-9	300 + Scavenger Concentrate	105 min (P_{80}: 9 µm)	Cleaner 1	5+1	Na_2CO_3	9.70	800*		10	80*	30	900	2
			Scavenger	2+1	Na_2CO_3	9.70		5	5		10	1300	2
			Cleaner 2	5	Na_2CO_3	9.80	110		0		10	900	1
			Cleaner 3	5	Na_2CO_3	9.90	67		0		7	900	1
			Cleaner 4	5	Na_2CO_3	10.00	33		0		5	900	1

Source: RPC Final Report; Nov 17, 2012; p.41-44.

Table 13-6: RPC 2012– Locked Cycle Test on Three-Hole Composite: Flotation Times and Reagent Regime – Zn Cleaners

Test No.	Concentrate Mass (g)	Regrind Time (min)	Stage	Conditioning Time (min)	pH		Reagent Dosage (g/t)			Float Time (min)	Revolution (rpm)	Cell Size (L)
					Modifier	Range	$CuSO_4$	3418	MIBC			
Zn LC 1-8	600 + Cleaner Concentrate	60	Cleaner 1	5+1	$Ca(OH)_2$	10.50	400		0	8	900	2
			Scavenger	2+1	$Ca(OH)_2$	11.00		5	5	5	1300	2
			Cleaner 2	1	$Ca(OH)_2$	10.80			0	5	900	2
			Cleaner 3	1	$Ca(OH)_2$	11.00			0	5	900	1
			Cleaner 4	1	$Ca(OH)_2$	10.20			0	5	900	1

Source: RPC Final Report; Nov 17, 2012; p.41-44.

Mineralogical examinations of some intermediate flotation products were carried out through the optimization process. Those performed on final products from the LCT tended to confirm that a primary grind target (P_{80}) of about 30 µm was sufficient to achieve a high degree of liberation, leaving most losses as <10 µm inclusions. As for the cleaning circuits, regrinding intensity to a P_{80} of 10 µm was needed to achieve proper upgrading in the copper and lead cleaning circuits. Zinc-bearing sphalerite is coarser-grained and thus less demanding in this respect

One microprobe analysis of a zinc concentrate obtained from one of the open-circuit tests with the three-hole composite showed that the sphalerite matrix contained 8.2% Fe, indicating a maximum sphalerite concentrate grade of about 56% Zn, per mineralogical limitations. Similar levels were measured from rejects reporting to the scavenger tails.

13.2.1.5 Bulk Cu-Pb Flotation Tests

Some of the bulk copper-lead flotation testwork results with the three-hole composite are worth mentioning, even though they consisted of open-circuit tests only, since this approach is conforming to the processing plant configuration within the optioned Caribou mill. As such, the outcomes for the bulk and zinc roughing circuits with the three-hole composite are presented in Table 13-7. The best results arguably correspond to test B-1 in terms of pyrite rejection, mostly reflected by the weight recovery to the rougher concentrates and good metal recoveries. Tests B-7 and B-11 are marginally better performing in terms of recovery of the metals but at the cost of higher mass pulls in both the bulk and zinc rougher concentrates.

Test B-1 used a reagent regime fairly similar to the one deployed for the LCT except that it did not include SMBS. B-7 in turn repeated B-1 conditions except that it dropped cyanide from the depressants and the copper-lead collector was the dithiophosphate AF241, instead of the Aerophine A3418A. Finally, test B-11 introduced another dithiophosphate, AF404 as an add-on to the AF241. The lower mass pulls for both the bulk and zinc rougher concentrates, albeit at marginal reductions in terms of copper-lead and silver recovery to the bulk concentrate. This may have been the result of the cyanide acting as a more potent pyrite depressant in B-1.

The better tests provided copper and lead recoveries of around 80% in the bulk rougher, with incremental lead recovery typically requiring an increase in the weight recovery, per B-11, as more middlings of the fine-grained galena and pyrite are then pulled into the concentrate stream. Zinc deportment to the bulk concentrate was at 25% to 30% in these tests, which is somewhat high, but likely the result of a lack of liberation awaiting the application of the regrinding step on the bulk concentrate, rather than of chemical activation. Silver recovery to this product varied in the 66% to 73% range.

Zinc rougher results were excellent considering that the stage zinc recovery (excluding prior losses to the bulk concentrate) was in the 94% to 97% range. The much lower mass pull seen in B-1 allowed an upgrading to 14.1% for the zinc rougher concentrate. This is a level similar to those seen in B-2 to B-5 whereas subsequent tests, excluding cyanide as a depressant, saw the resulting zinc rougher concentrate grades drop below 10% Zn, with B-7 at 8.2% Zn and B-11 at 6.7% Zn only. The cleaning tests of the bulk (CuPb C-1 to C-10) and zinc (Zn-C1 to C5) rougher concentrates were carried out on larger weight of each product, gathered from multiple roughing tests (B-12 to B-19) mostly carried out along the lines of test B-11.

Table 13-7: RPC 2012 Testwork Program – Bulk/Zn Tests Roughing Results – Three-Hole Composite

Test	Concentrate / Tail[1]	Mass	Mass	Chemical Assay Results (%)					Distribution Ratio (%)				
		(g)	Dist. (%)	Fe	Cu	Pb	Zn	Ag (g/t)	Fe	Cu	Pb	Zn	Ag
B-1	RC1 (Cu-Pb)	274.5	15.5	34.80	1.81	6.25	5.47	210	15.3	78.2	77.4	23.3	66.6
	RC2 (Zn)	335.0	18.9	35.54	0.19	0.50	14.06	34	19.0	10.0	7.6	73.0	13.1
	RTails	1,160.1	65.6	35.48	0.07	0.29	0.21	15	65.7	11.9	14.9	3.7	20.3
	Calculated Head	1,769.6	-	35.39	0.36	1.25	3.65	49.0	-	-	-	-	-
	Head Assays	-	-	39.44	0.33	1.14	3.42	47	-	-	-	-	-
B-2	RC1 (Cu-Pb)	296.0	16.7	33.04	1.53	5.32	5.16	192	16.1	79.4	71.3	24.9	70.4
	RC2 (Zn)	306.9	17.3	33.87	0.16	0.49	13.99	29	17.1	8.6	6.9	70.0	10.9
	RTails	1,167.2	65.9	34.66	0.06	0.41	0.27	13	66.7	12.0	21.8	5.1	18.6
	Calculated Head	1,770.1		34.25	0.32	1.25	3.47	46	-	-	-	-	-
B-3	RC1 (Cu-Pb)	193.2	10.9	25.46	1.71	5.85	4.53	270	8.0	63.9	58.9	15.0	62.6
	RC2 (Zn)	315.3	17.8	30.30	0.30	0.89	14.95	42	15.6	18.5	14.7	81.0	15.9
	RTails	1,260.3	71.3	37.22	0.07	0.40	0.18	14	76.4	17.6	26.4	3.9	21.5
	Calculated Head	1,768.8	-	34.70	0.29	1.08	3.29	47	-	-	-	-	-
B-4	RC1 (Cu-Pb)	185.4	10.5	32.88	2.48	7.94	6.73	282	9.1	70.7	63.7	20.3	63.0
	RC2 (Zn)	346.8	19.6	33.52	0.28	1.07	13.52	38	17.3	14.9	16.1	76.1	16.1
	RTails	1,233.7	69.9	40.00	0.08	0.38	0.18	14	73.6	14.4	20.3	3.7	21.0
	Calculated Head	1,765.9	-	37.98	0.37	1.31	3.49	47	-	-	-	-	-
B-5	RC1 (Cu-Pb)	227.4	12.8	32.70	2.08	7.12	6.45	248	12.2	75.6	70.3	23.3	66.4
	RC2 (Zn)	323.5	18.2	31.99	0.21	0.70	14.16	37	16.9	11.0	9.8	72.8	14.3
	RTails	1,223.7	69.0	35.36	0.07	0.37	0.20	13	70.9	13.4	19.9	3.9	19.3
	Calculated Head	1,774.6	-	34.41	0.35	1.30	3.54	48	-	-	-	-	-
B-6	RC1 (Cu-Pb)	248.0	14.0	34.95	1.87	6.59	6.81	220	12.8	75.1	72.0	26.7	64.3
	RC2 (Zn)	422.2	23.9	33.69	0.18	0.49	10.47	33	21.0	12.4	9.2	69.9	16.5
	RTails	1,097.4	62.1	40.80	0.07	0.39	0.20	15	66.2	12.6	18.8	3.4	19.2
	Calculated Head	1,767.6	-	38.28	0.35	1.28	3.58	48	-	-	-	-	-
B-7	RC1 (Cu-Pb)	317.1	17.9	37.29	1.57	5.76	6.53	183	18.4	79.5	76.3	32.3	70.3
	RC2 (Zn)	501.2	28.4	33.39	0.13	0.40	8.17	25	26.0	10.3	8.4	63.9	15.3
	RTails	949.2	53.7	37.78	0.07	0.38	0.26	12	55.7	10.2	15.3	3.8	14.4
	Calculated Head	1,767.5	-	36.45	0.35	1.35	3.63	47	-	-	-	-	-
B-8	RC1 (Cu-Pb)	354.4	20.1	36.55	1.37	4.85	5.50	177	19.9	80.2	75.3	31.2	72.2
	RC2 (Zn)	552.3	31.3	37.44	0.12	0.45	7.41	24	31.8	10.8	10.8	65.6	15.1
	RTails	857.3	48.6	36.58	0.06	0.37	0.23	13	48.3	9.0	13.9	3.2	12.7
	Calculated Head	1,764.0	-	36.84	0.34	1.30	3.54	49	-	-	-	-	-
B-9	RC1 (Cu-Pb)	365.8	20.7	37.21	1.30	4.81	5.26	172	21.0	79.2	77.0	30.5	55.3
	RC2 (Zn)	452.3	25.6	34.64	0.14	0.49	9.33	25	24.1	10.6	9.7	66.9	10.0
	RTails	951.3	53.8	37.46	0.06	0.32	0.17	42	54.9	10.2	13.3	2.6	34.7
	Calculated Head	1,769.4	-	36.68	0.34	1.29	3.56	64	-	-	-	-	-
B-10	RC1 (Cu-Pb)	544.4	30.7	38.90	0.94	3.47	4.98	115	31.3	84.3	81.4	40.6	73.6
	RC2 (Zn)	561.9	31.7	38.72	0.10	0.39	6.86	21	32.2	9.1	9.4	57.8	13.9
	RTails	664.8	37.5	37.16	0.06	0.32	0.16	16	36.5	6.6	9.2	1.6	12.5
	Calculated Head	1,771.1	100.0	38.19	0.34	1.31	3.77	48	-	-	-	-	-
B-11*	RC1 (Cu-Pb)	385.9	21.8	36.65	1.27	4.48	5.26	159	21.4	80.1	77.1	31.9	73.4
	RC2 (Zn)	626.1	35.3	38.7	0.12	0.42	6.69	21	36.7	12.3	11.7	65.8	15.7
	RTails	760.3	42.9	36.49	0.06	0.33	0.20	12	41.9	7.6	11.2	2.4	10.9
	Calculated Head	1,772.3	100.0	37.32	0.35	1.27	3.59	47	-	-	-	-	-
B-12-19*	RC1 (Cu-Pb)	396.3	22.4	39.98	1.25	4.81	5.81	153	24.0	80.7	79.5	35.0	75.1
	RC2 (Zn)	582.2	32.9	36.90	0.12	0.41	7.10	21	32.5	11.4	10.0	62.8	15.1
	RTails	790.9	44.7	36.38	0.06	0.32	0.18	10	43.5	7.9	10.6	2.2	9.8
	Calculated Head	1,769.4	100.0	37.36	0.35	1.36	3.72	46	-	-	-	-	-

Note 1. RC1 = bulk rougher concentrate (Cu/Pb); RC2 = Zn rougher concentrate; RTails = Zn rougher tails. Source: RPC Final Report; Nov 17, 2012; p.21.

Table 13-8 presents the results for the bulk concentrate cleaning tests at a regrind size below 12 µm while using a jar mill with ceramic beads. It is noteworthy that the recoveries presented in Table 13-8 are relative to the feed of these tests, which was the combined bulk concentrate of tests B-12 to B-19. These tests achieved 80.7%, 79.5%, 35.0%, and 75.1% recoveries to the bulk concentrate for copper, lead, zinc, and silver, respectively.

The initial test, CuPb C-1, featured a P_{80} of 22 µm and its direct flotation failed to remove the zinc floated previously. Gradual reduction of the P_{80} showed the improvement yielded by a finer regrind target until the 10 to 12 µm range was settled as optimum by test CuPb C-7, which is largely aligned with the indications given by the mineralogical analysis regarding grain size. It thus left only CuPb C-7 to C-10 to introduce reagent regime changes to optimize the bulk cleaning circuit results at the selected regrind size. All of these settled on a mixture of AF241/AF404 as the collector and the addition of 1000 g/t of $ZnSO_4$ and 75 g/t of cyanide as depressants.

Test CuPb C-9 introduced Na_2S as an additional depressant, as covellite was thought to be responsible for sphalerite activation. The collector addition in C-9 and C-10 was drastically reduced, from 40 g/t AF241 and 25 g/t AF404 to only 10 g/t of AF241 in the cleaner-scavenger stage. Despite having brought 79.5% of the lead available in the fresh feed into the bulk concentrate, all of the cleaning tests failed to hold onto lead, with stage recovery of 30% to 35% at best. About 75% of the copper was recovered in the better tests (CuPb C-7 and C-8), along with 20% to 25% of the zinc also reporting to this product though. As much pyrite was also floating in these tests, upgrading the combined Cu+Pb grades was limited.

All tests featured long flotation time of 25 to 30 minutes for the first cleaner, 10 minutes for the cleaner-scavenger, and 10 to 25 minutes for the second cleaner.

Aging of the B12 to B19 rougher concentrates (as it was gradually accumulated to provide the weight then used as common feed for all the cleaning tests) may have been responsible for the poor performance of the cleaning trials. Aging tests that were carried out to prove the effect of oxidation on ground feed material did show much deterioration to the subsequent flotation response, as seen within the subsequent RPC 2019 testwork program. Adopting bulk roughing conditions, including the use of cyanide as per B-1 to B-5, would have reduced the mass pull seen in tests B12 to B19, and may have provided for a more effective pyrite depression, potentially carried through the cleaning stage.

The zinc rougher concentrate cleaning tests (Zn C-1 to C-5) introduced checks onto the effect of the regrinding intensity, going from the “as-is” rougher concentrate flotation for C-1 to C-3, with a P_{80} of 26 µm, down to 21 µm in C-4 and 16 µm in C-5. No collector addition was made for C-1 to C-3, while 5 g/t of A3418A to the scavenger was used for C-4 and C-5. No activation with copper sulphate was made in C-1 and C-2, while all other tests featured an addition of 400 g/t. The pH used ranged between 10.5 to 11.3 for promoting pyrite depression. Table 13-9 presents the results achieved.

Despite the lack of regrinding or activation with $CuSO_4$ in C-1, a recovery of 91.2% Zn at a second cleaner concentrate grade of 41.2% Zn was achieved. Adding two more cleaning stages in C-2 and keeping other conditions similar to C-1 produced a better upgrading to 47.8% Zn, while still holding onto 77.3% of the zinc. Regrinding and the use of activation in C-3 to C-4 provided similar upgrading but allowed to hold the recovery to the final concentrate in the 81% to 84% range. C-5, with the finest regrind product, did achieve a better at 53.6% Zn, but brought back the recovery to 76.2%.

Table 13-8: RPC 2012 Testwork Program – Bulk Concentrate Cleaning Test from Bulk/Zn Roughing – Three-Hole Composite

Test	Concentrate/ Tail[1]	Mass (g)	Mass Distribution (%)	Analytical Assays					Distribution Ratio (%)				
				Fe (%)	Cu (%)	Pb (%)	Zn (%)	Ag (g/t)	Fe	Cu	Pb	Zn	Ag
CuPb C1	Cl2	113.6	38.9	34.17	2.83	7.46	11.66	300	36.1	89.4	69.4	88.8	84.0
	Cl2T	24.0	8.2	41.72	0.35	2.45	1.21	55	9.3	2.3	4.8	1.9	3.3
	Cleaner-Scavenger	23.9	8.2	40.84	0.47	2.76	2.13	68	9.1	3.1	5.4	3.4	4.0
	CScvgT	130.9	44.8	37.33	0.14	1.90	0.67	27	45.5	5.1	20.4	5.9	8.7
	Calculated Head	292.4	100.0	36.75	1.23	4.18	5.10	139	-	-	-	-	-
CuPb C2	Cl2	40.4	13.5	36.51	3.65	8.23	6.03	296	14.0	41.2	28.1	15.7	31.4
	Cl2T	33.7	11.3	34.62	0.91	3.63	5.08	119	11.1	8.6	10.4	11.0	10.5
	Cleaner-Scavenger	39.9	13.4	34.36	2.02	5.47	7.81	200	13.1	22.5	18.5	20.1	20.9
	CScvgT	184.3	61.8	35.20	0.54	2.76	4.48	77	61.8	27.8	43.0	53.2	37.2
	Calculated Head	298.3	100.0	35.20	1.20	3.96	5.20	128	-	-	-	-	-
CuPb C3	Cl2	38.0	12.2	34.14	4.71	5.37	3.95	283	11.8	52.0	17.4	9.8	27.4
	Cl2T	27.3	8.7	38.11	0.58	3.30	4.56	116	9.5	4.6	7.7	8.2	8.1
	Cleaner-Scavenger	42.4	13.6	34.51	1.54	6.50	7.14	245	13.4	19.0	23.4	19.8	26.5
	CScvgT	204.6	65.5	34.99	0.41	2.96	4.64	73	65.3	24.4	51.5	62.2	38.1
	Calculated Head	312.3	100.0	35.09	1.10	3.76	4.89	126	-	-	-	-	-
CuPb C4	Cl2	34.7	11.2	35.89	4.06	3.77	3.95	281	11.9	40.3	11.2	9.3	25.5
	Cl2T	39.1	12.6	36.12	0.72	3.90	4.83	116	13.5	8.1	13.1	12.9	11.9
	Cleaner-Scavenger	36.4	11.7	35.51	2.26	5.32	6.44	202	12.4	23.5	16.6	16.0	19.2
	CScvgT	34.9	11.2	32.54	1.21	5.94	7.47	183	10.9	12.1	17.8	17.8	16.7
	Calculated Head	165.2	53.2	32.33	0.34	2.92	3.91	62	51.2	16.1	41.3	44.0	26.8
		310.3	100.0	33.60	1.13	3.76	4.73	123	-	-	-	-	-
CuPb C5	Cl2	107.1	35.0	34.52	2.95	4.82	4.28	280	34.0	73.9	41.7	34.1	68.6
	Cl2T	47.8	15.6	36.62	0.82	3.99	5.47	98	16.1	9.2	15.4	19.4	10.7
	Cleaner-Scavenger	47	15.4	38.47	0.81	4.42	6.94	91	16.6	8.9	16.8	24.2	9.8
	CScvgT	104	34.0	34.66	0.33	3.11	2.88	46	33.2	8.0	26.1	22.3	10.9
	Calculated Head	305.9	100.0	35.50	1.40	4.05	4.40	143	-	-	-	-	-
CuPb C6	C6-Cl2C	86.1	28.3	36.05	2.45	5.02	6.10	265	27.6	69.7	35.1	31.0	60.9
	C6-Cl2T	55.6	18.3	39.67	0.60	3.91	6.09	86	19.6	11.0	17.7	20.0	12.8
	C6-CScvgC	39.2	12.9	32.58	0.67	4.68	8.92	110	11.4	8.7	14.9	20.6	11.5
	C6-CScvgT	123	40.5	37.75	0.26	3.23	3.91	45	41.3	10.6	32.3	28.4	14.8
	Calculated Head	303.9	100.0	36.95	1.00	4.05	5.58	123	-	-	-	-	-
CuPb C7	Cl2	74.1	23.9	38.53	3.72	5.43	5.28	350	24.2	76.7	32.8	25.6	59.6
	Cl2T	81.6	26.3	39.05	0.48	3.52	5.29	92	27.1	10.9	23.4	28.2	17.2
	Cleaner-Scavenger	40.7	13.1	36.02	0.47	4.38	7.03	102	12.5	5.3	14.5	18.7	9.6
	CScvgT	113.5	36.6	37.59	0.22	3.15	3.71	52	36.2	7.1	29.2	27.5	13.6
	Calculated Head	309.9	100.0	37.99	1.16	3.96	4.94	141	-	-	-	-	-
CuPb C8	Cl2	52.6	17.3	34.24	5.05	5.36	5.69	449	16.1	74.3	25.7	19.4	54.4
	Cl2T	102.0	33.5	37.50	0.49	3.42	5.24	96	34.1	13.9	31.8	34.7	22.7
	Cleaner-Scavenger	43.6	14.3	36.00	0.47	4.29	6.26	106	14.0	5.8	17.0	17.7	10.7
	CScvgT	106.3	34.9	37.88	0.20	2.64	4.10	50	35.9	6.0	25.5	28.2	12.3
	Calculated Head	304.5	100.0	36.85	1.18	3.61	5.06	142	-	-	-	-	-
CuPb C9	Cl2	61.3	18.8	36.85	3.98	5.25	5.44	308	17.8	69.1	29.2	23.8	45.8
	Cl2T	80.2	24.6	39.86	0.56	3.77	4.29	108	25.1	12.8	27.4	24.5	21.0
	Cleaner- Scavenger	52.3	16.0	37.79	0.82	5.01	6.45	164	15.5	12.1	23.8	24.1	20.8
	CScvgT	132.8	40.7	39.83	0.16	1.62	2.92	39	41.6	6.0	19.5	27.6	12.4
	Calculated Head	326.6	100.0	38.95	1.08	3.37	4.30	126	-	-	-	-	-
CuPb C10	Cl2	33.1	17.3	34.04	6.82	4.82	5.30	400	9.2	62.5	16.1	12.5	34.1
	Cl2T	53.4	27.9	38.14	1.03	5.09	4.71	171	16.6	15.2	27.5	17.9	23.5
	Cleaner-Scavenger	41.4	21.6	37.69	1.14	5.20	7.49	204	12.7	13.0	21.7	22.0	21.7
	CScvgT	191.5	100.0	39.45	0.17	1.79	3.50	42	61.5	9.2	34.6	47.6	20.7
	Calculated Head	319.4	166.8	38.44	1.13	3.10	4.41	121	-	-	-	-	-

Note 1. Cl2C = second cleaner concentrate; Cl2T = second cleaner tails; CScvgT = cleaner-scavenger tails; CScvgC – cleaner-scavenger concentrate. Source: RPC Final Report; Nov 17, 2012; p.23.

Table 13-9: RPC 2012 Testwork Program – Zn Cleaning Tests from Bulk/Zn Roughing – Three-Hole Composite

Test	Concentrate / Tail[1]	Mass	Mass	Assays					Distribution Ratio (%)				
		(g)	Dist (%)	Fe (%)	Cu (%)	Pb (%)	Zn (%)	Ag (g/t)	Fe	Cu	Pb	Zn	Ag
Zn-C1	Cl2C	55.8	18.8	21.52	0.30	0.34	41.24	300	11.7	42.6	15.1	91.2	42.0
	Cl2T	21.7	7.3	38.69	0.11	0.42	1.68	55	8.2	6.1	7.3	1.4	5.3
	CScvgC	30.3	10.2	38.72	0.22	0.54	4.08	68	11.4	16.4	13.2	4.9	11.7
	CScvgT	189.6	63.8	37.19	0.073	0.42	0.33	27	68.7	34.9	64.3	2.5	41.0
	Calculated Cleaner 1 Con	77.5	26.1	26.33	0.25	0.36	30.17	139	19.9	48.7	22.5	92.6	47.3
	Calculated Cleaner 1 Tls	219.9	73.9	37.40	0.09	0.44	0.84	296	80.1	51.3	77.5	7.4	52.7
	Calculated Head	297.4	100.0	34.52	0.13	0.42	8.49	119	-	-	-	-	-
Zn-C2	Cl4C	64.5	10.9	14.65	0.32	0.36	47.81	200	4.1	26.4	7.9	77.3	29.8
	Cl4T	12.7	2.1	30.81	0.30	0.50	21.05	77	1.7	4.9	2.2	6.7	4.3
	Cl3T	9.9	1.7	38.53	0.20	0.64	5.96	128	1.7	2.5	2.2	1.5	2.2
	Cl2T	32.9	5.5	40.20	0.17	0.67	4.50	283	5.8	7.3	7.5	3.7	5.7
	CScvgC	67.3	11.3	40.68	0.20	0.74	2.70	116	12.0	16.9	17.0	4.6	12.0
	CScvgT	406.5	68.5	41.97	0.081	0.46	0.61	245	74.7	41.9	63.2	6.2	45.9
	Calculated Cleaner 3 Con	77.2	13.0	17.31	0.32	0.38	43.40	73	5.8	31.3	10.0	84.0	34.1
	Calculated Cleaner 2 Con	87.1	14.7	19.72	0.30	0.41	39.15	126	7.5	33.8	12.2	85.5	36.4
	Calculated Cleaner 1 Con	120.0	20.2	25.33	0.27	0.48	29.65	281	13.3	41.1	19.7	89.2	42.1
	Calculated Cleaner 1 Tls	473.8	79.8	41.79	0.10	0.50	0.91	116	86.7	58.9	80.3	10.8	57.9
	Calculated Head	593.8	100.0	38.46	0.13	0.49	6.72	202	-	-	-	-	-
Zn-C3	Cl4C	69.1	11.6	14.83	0.32	0.32	47.51	183	4.6	24.9	7.5	81.3	31.7
	Cl4T	11.1	1.9	32.11	0.29	0.47	19.53	62	1.6	3.7	1.8	5.4	4.2
	Cl3T	8.2	1.4	39.13	0.22	0.57	7.17	123	1.4	2.0	1.6	1.5	2.2
	Cl2T	62.1	10.5	40.16	0.18	0.57	4.10	280	11.1	12.5	12.1	6.3	11.5
	CScvgC	92	15.5	40.34	0.18	0.68	1.35	98	16.6	19.2	21.3	3.1	14.1
	CScvgT	351.5	59.2	41.26	0.094	0.46	0.29	91	64.7	37.7	55.6	2.5	36.3
	Calculated Cleaner 3 Con	80.2	13.5	17.22	0.31	0.34	43.64	46	6.2	28.5	9.3	86.6	35.9
	Calculated Cleaner 2 Con	88.4	14.9	19.25	0.30	0.36	40.26	143	7.6	30.6	10.9	88.1	38.2
	Calculated Cleaner 1 Con	150.5	25.3	27.88	0.25	0.45	25.34	265	18.7	43.0	23.1	94.4	49.6
	Calculated Cleaner 1 Tls	443.5	74.7	41.07	0.11	0.51	0.51	86	81.3	57.0	76.9	5.6	50.4
	Calculated Head	594.0	100.0	37.73	0.15	0.49	6.80	110	-	-	-	-	-
Zn-C4	Cl4C	67.5	11.3	13.56	0.38	0.35	48.94	45	3.9	28.8	8.2	84.1	34.5
	Cl4T	3.1	0.5	30.13	0.40	0.60	19.11	123	0.4	1.4	0.7	1.5	1.3
	Cl3T	3.2	0.5	33.07	0.39	0.78	13.40	350	0.5	1.4	0.9	1.1	1.3
	Cl2T	25.3	4.2	36.99	0.32	0.77	8.91	92	4.0	9.2	6.8	5.7	7.7
	CScvgC	87.9	14.7	42.83	0.19	0.66	1.88	102	16.1	18.7	20.0	4.2	15.7
	CScvgT	409.4	68.6	42.96	0.087	0.45	0.32	52	75.2	40.4	63.5	3.4	39.5
	Calculated Cleaner 3 Con	70.6	11.8	14.28	0.38	0.36	47.63	141	4.3	30.2	8.8	85.6	35.9
	Calculated Cleaner 2 Con	73.8	12.4	15.10	0.38	0.38	46.15	449	4.8	31.7	9.7	86.7	37.2
	Calculated Cleaner 1 Con	99.1	16.6	20.69	0.36	0.48	36.64	96	8.8	40.8	16.5	92.4	44.8
	Calculated Cleaner 1 Tls	497.3	83.4	42.94	0.10	0.48	0.60	106	91.2	59.2	83.5	7.6	55.2
	Calculated Head	596.4	100.0	39.24	0.15	0.48	6.59	50	-	-	-	-	-
Zn-C5	Cl4C	56.0	9.3	9.94	0.35	0.33	53.60	142	2.4	22.1	6.3	76.2	22.4
	Cl4T	5.5	0.9	24.00	0.44	0.72	30.54	308	0.6	2.8	1.3	4.3	2.4
	Cl3T	5.9	1.0	29.06	0.48	1.02	23.42	108	0.7	3.3	2.0	3.5	2.7
	Cl2T	24.9	4.1	35.10	0.46	1.12	12.98	164	3.8	13.0	9.4	8.2	9.9
	CScvgC	70.4	11.7	42.46	0.23	0.78	2.23	39	12.9	18.8	18.3	4.0	13.6
	CScvgT	439.8	73.0	42.05	0.080	0.42	0.34	126	79.7	40.0	62.7	3.8	49.1
	Calculated Cleaner 3 Con	61.5	10.2	11.20	0.36	0.37	51.54	400	3.0	24.9	7.6	80.5	24.7
	Calculated Cleaner 2 Con	67.4	11.2	12.76	0.37	0.42	49.08	171	3.7	28.2	9.6	84.0	27.4
	Calculated Cleaner 1 Con	92.3	15.3	18.79	0.39	0.61	39.34	204	7.5	41.2	19.0	92.2	37.3
	Calculated Cleaner 1 Tls	510.2	84.7	42.10	0.10	0.47	0.60	42	92.5	58.8	81.0	7.8	62.7
	Calculated Head	602.5	100.0	38.53	0.15	0.49	6.54	121	-	-	-	-	-

Note 1. ClXC = cleaner "X" concentrate; ClXT = cleaner "X" tails; CScvgT = cleaner-scavenger tails; CScvgC – cleaner-scavenger concentrate. Source: RPC Final Report; Nov 17, 2012; p.25.

13.2.1.6 Sequential Flotation Tests Highlights

Although not aligned with the retained circuit configuration, the outcome of the sequential flotation tests on some portions of the three holes provided for this testwork program is indicative of the metallurgical results achieved on samples where only either copper or lead at times, along with zinc, was in sufficient quantity to justify the production of a separate concentrate ahead of zinc flotation. The results provide a glimpse of the upgrading potential that such samples offer for copper and lead.

Tables 13-10 to 13-12 show the best outcomes with different samples as a feed source. The primary grind target of 30 µm as P_{80}, along with the copper and lead regrind targets of 10 to 12 µm and 20 µm for zinc regrind (per the LCT test conditions) were upheld for all these tests. The roughing stages made use of the conditions seen in test B-1 on the three-hole composite, except for the use of Aerofloat 5100 as a copper circuit collector in lieu of A3418A.

Table 13-10 shows the results for a sample where both copper and lead were significant enough to warrant production of the two in separate concentrates. The final concentrate grade of 21.55% Cu for the copper concentrate is interesting, while that of the lead product, at almost 31% Pb, is marginal, likely reflecting the fine-grained nature of galena and its complex mineralogy with pyrite. Zinc failed to be upgraded to a saleable level, with only 25.1% Zn being achieved. Pyrite rejection was therefore deficient in this stage.

Table 13-10: RPC 2012 Testwork Program – Sequential Flotation Results Highlights: MB-121 Bottom Zone

Test No.	Product	Mass Recovery Stage %	Grades				Stage Recoveries, %			
			% Cu	% Pb	% Zn	Ag, g/t	Cu	Pb	Zn	Ag
121 SQ3	Feed	100	0.59	0.90	1.86	44	100	100	100	100
121 SQ3	Cu Rougher Concentrate	10.0	4.68	1.62	3.65	216	79.2	18.1	19.7	48.9
121 SQ3	Pb Rougher Concentrate	13.6	0.35	3.31	2.33	83	8.1	50.2	17.1	25.5
121 SQ3	Zn Rougher Concentrate	21.9	0.17	0.48	5.00	27	6.3	11.7	58.8	13.3
CuCl1	Cu 3rd Cleaner Concentrate	9.7	21.55	5.26	6.21	217	38.6	28.0	15.5	8.6
PbCl1	Pb 4th Cleaner Concentrate	4.2	4.19	30.97	10.7	789	50.2	40.3	18.5	40.7
ZnCl2	Zn 4th Cleaner Concentrate	17.5	0.45	0.41	25.11	49	41.5	16.2	87.1	38.4

Source: RPC Final Report; Nov 17, 2012; p.29-30.

Table 13-11 is an example of where the production of a separate copper concentrate should not have been attempted. With only 0.12% Cu in the feed, this level is already below the rougher tails grade seen in all of the tests performed. A bulk Cu-Pb flotation should have been carried out with only lead as the eventual economic metal worth sending through a cleaning stage.

Table 13-11: RPC 2012 Testwork Program – Sequential Flotation Results Highlights: MB-124 Bottom Zone

Test No.	Product	Mass Recovery Stage %	Grades				Stage Recoveries, %			
			% Cu	% Pb	% Zn	Ag, g/t	Cu	Pb	Zn	Ag
124 Btm SQ8	Feed	100	0.12	0.80	2.81	33	100	100	100	100
124 Btm SQ8	Cu Rougher Concentrate	6.0	1.89	1.21	4.37	168	55.7	9.2	9.4	30.9
124 Btm SQ8	Pb Rougher Concentrate	12.0	0.16	4.09	3.38	70	9.4	61.7	14.5	25.6
124 Btm SQ8	Zn Rougher Concentrate	23.0	0.16	0.39	8.83	29	17.9	11.3	72.3	20.3
SQ6-8 CuCl1	Cu 3rd Cleaner Concentrate	3.1	17.64	3.81	5.36	538	28.9	9.0	3.9	11.2

Source: RPC Final Report; Nov 17, 2012; p.33-34.

Table 13-12 represent a case where a marginal copper head grade may not warrant the production of a separate copper concentrate. Combining the copper and lead rougher would have shown a lead recovery approaching 80% and may have yielded a better upgrading in a single bulk cleaning circuit. The final copper level in the lead concentrate may have diluted the achievable lead concentrate grade and negated the possibility of receiving payment for the copper content, but it is likely a better outcome for such material, with the combined silver recovery receiving favourable smelting terms.

Table 13-12: RPC 2012 Testwork Program – Sequential Flotation Results Highlights: MB-124 Bottom + Middle Zones

Test No.	Product	Mass Recovery Stage %	Grades				Stage Recoveries, %			
			% Cu	% Pb	% Zn	Ag, g/t	Cu	Pb	Zn	Ag
124(50:50) SQ2	Feed	100	0.19	1.39	3.62	50	100	100	100	100
124(50:50) SQ2	Cu Rougher Concentrate	9.1	1.11	3.46	5.56	223	54.4	22.6	13.9	40.7
124(50:50) SQ2	Pb Rougher Concentrate	11.1	0.22	6.94	4.89	100	13.1	55.2	15.0	22.3
124(50:50) SQ2	Zn Rougher Concentrate	19.1	0.16	0.64	13.17	43	15.9	8.5	67.2	16.0
SQ1-1-3 CuCl1	Cu 3rd Cleaner Concentrate	2.0	6.48	7.29	3.73	238	13.9	5.1	1.5	3.0
SQ1-1-3 PbCl1	Pb 4th Cleaner Concentrate	4.0	2.64	34.57	7.21	789	38.6	21.6	5.9	27.3
SQ1-1-3 ZnCl1	Zn 4th Cleaner Concentrate	16.9	0.31	0.46	48.15	57	29.0	12.8	61.6	24.6

Source: RPC Final Report; Nov 17, 2012; p.34-36.

13.2.2 2019 RPC Testwork Program Highlights

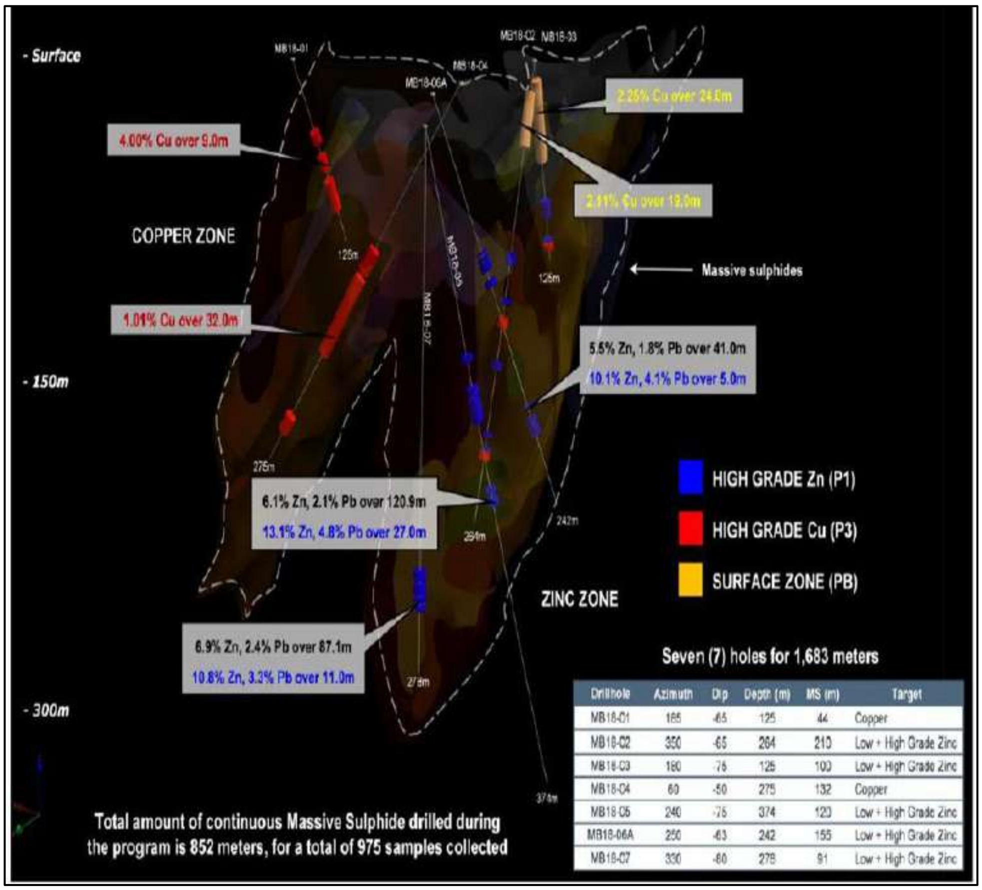
The 2019 RPC program, completed at the behest of Trevali Mining, focused on three samples: P1, PB, and P3. P1 was Pb-Zn only material, while P3 was Cu-Zn only. The PB sample represented the oxidized upper layer of the mineralized material, for which the attempts at finding an appropriate reagent regime failed. This oxidized layer is not part of the mine plan, so there is no further discussion of the results. Table 13-13 presents the characteristics of the samples, while Figure 13-3 shows sample sources.

Table 13-13: RPC 2019 Testwork Program – Samples Head Grades

Zone	Sample ID	Grade					
		Pb (%)	Zn (%)	Cu (%)	Ag (g/t)	Fe (%)	Au (g/t)
Zn	P1	1.7	5.15	0.16	68	37.0	0.365
Cu	P3	0.63	1.31	1.01	44	42.6	0.504

Source: RPC Final Report; Mar 13, 2019; p.1.

Figure 13-3: RPC 2019 Testwork Program – Samples Sourcing



Source: P&E, 2024.

A lower grade P2 sample was later prepared and various samples were shipped to RPC. As the operator of the Caribou processing plant in 2019, Trevali directed the testwork towards integrating the elements of their existing plant configuration as much as possible while using the same reagents. This likely steered the sample selection towards material with only two economic metals, because the plant featured only two roughing sections, each with its regrinding and cleaning circuits. Since the Murray Brook resources are showing that a three-concentrate approach would have to be applied over 80% of the time on a block model basis, and likely more without a selective mining approach, the RPC 2019 work has been looked at mostly in terms of the reagent regimes evaluated to float copper or lead selectively from pyrite and zinc. The zinc circuit performance as well is of interest.

13.2.2.1 Grinding Testwork

Only the P1 sample was submitted to determine its hardness. Table 13-14 indicates the values obtained for SAG mill hardness (JK parameters), rod mill work index (RWi), and ball milling (BWi). A measurement of the abrasion index (Ai), used to assess grinding media consumption and crusher and mill liners wear rate was also obtained.

Table 13-14: RPC 2019 Testwork Program – P1 Sample Hardness Determinations

Sample Name	Relative Density	JK Parameters			RWI (kWh/t)	BWI (kWh/t)	AI (g)
		A x b	t_a*	SCSE			
P1	4.37	54.7	0.32	8.4	15.7	11.4	0.538

Note: *The t_a value reported as part of the SMC procedure is an estimate. Source: RPC Final Report; Mar 13, 2019; p.10.

13.2.2.2 Flotation Testwork – Open Circuit

13.2.2.3 Mineralogy

Mineralogical examination was performed on the P1 sample only. The main gangue mineral was pyrite, which comprised 71.8% of the sample weight. Non-gangue minerals that were identified included quartz, calcite, dolomite, ankerite, chlorite, and clay minerals, such as siderite.

Weight proportions for the economic sulphide minerals were 11.2% for sphalerite, 2.1% for galena, and 0.24% for chalcopyrite. The mean particle size of the sulphides was 207 µm, mostly skewed by pyrite. Galena had the smallest average grain size, at 11 µm, followed by chalcopyrite at 12 µm and sphalerite at 20 µm.

13.2.2.3.1 Sample P1 Testing

Table 13-15 summarizes the roughing tests carried out with the Pb-Zn-bearing P1 sample, in which the copper grade was only 0.16% Cu and thus near the expected rougher tails level. Some attempts at floating copper separately were still made in some tests. Table 13-15 shows that 10% to 12% of the weight would typically report to the lead rougher concentrate along with 70% to 75% of lead. Extending the flotation time with a lead scavenger pushes the recovery near 80%, but weight recovery also jumps near 20%. Minor differences in terms of deportment of lead or zinc to their respective rougher concentrates could be seen for all the test conditions tried. The main caveat resided in the much-increased mass pull to the zinc rougher concentrate that resulted from some of these, with optimum levels of 15% to 20% passing to the mid-30% (in P1T2, T4, and T5) and even to 54.2% (P1T6).

Table 13-15: RPC 2019 Testwork Program – P1 Roughing Results

Test ID	Stream	Mass (%)	Chemical Assays (%)					Distribution Ratio (%)				
			Pb	Zn	Cu	Ag (g/t)	Fe	Pb	Zn	Cu	Ag	Fe
Previous RPC Report Reference: MIS-J1849 Results	Cu Rougher	5.7	2.43	5.10	3.43	282	34.15	12.0	9.0	63.6	37.8	5.4
	Pb Rougher	8.9	7.58	5.28	0.37	135	35.39	58.2	14.6	10.7	28.2	8.7
	Zn Rougher	18.0	0.73	13.01	0.21	40	34.82	11.3	72.3	12.2	16.8	17.2
	Tail	67.3	0.32	0.20	0.06	11	37.11	18.5	4.1	13.5	17.3	68.7
	Calculated Head	100.0	1.16	3.24	0.31	43	36.37	100.0	100.0	100.0	100.0	100.0
P1T1 - MB Optimum	Cu Rougher	2.1	8.47	6.59	3.53	797	35.37	10.7	2.8	44.3	29.1	2.3
	Pb Rougher	7.6	13.60	6.04	0.18	166	29.48	61.9	9.1	8.2	21.9	6.9
	Zn Rougher	21.4	0.89	19.68	0.21	55	27.54	11.3	83.5	27.0	20.4	18.0
	Tail	68.9	0.39	0.34	0.05	24	34.62	16.1	4.6	20.6	28.7	72.9
	Measured Head	-	1.70	5.15	0.16	68	37.01	-	-	-	-	-
	Calculated Head	100.0	1.67	5.04	0.17	58	32.73	100.0	100.0	100.0	100.0	100.0
P1T2 - Current Caribou	Pb Rougher	13.9	8.75	5.27	0.73	259	35.50	73.0	14.6	49.1	61.0	13.5
	Pb Scavenger	7.2	1.92	5.69	0.21	58	39.92	8.3	8.1	7.5	7.0	7.8
	Zn Rougher	36.6	0.46	10.20	0.15	28.61	38.92	10.1	74.5	27.0	17.7	39.0
	Tail	42.3	0.34	0.32	0.08	20	34.26	8.6	2.7	16.4	14.3	39.6
	Calculated Head	100.0	1.67	5.01	0.21	59	36.54	100.0	100.0	100.0	100.0	100.0
P1T3 - MB Optimum at 40 µm	Cu Rougher	3.8	6.30	3.86	2.14	517	36.15	12.8	2.5	41.0	31.2	3.3
	Pb Rougher	9.1	12.10	5.80	0.19	183	32.75	58.9	9.1	8.7	26.3	7.2
	Zn Rougher	24.2	0.88	20.08	0.22	48	34.29	11.4	83.5	27.2	18.2	20.2
	Tail	76.8	0.41	0.37	0.06	20	37.20	16.8	4.9	23.2	24.3	69.3
	Calculated Head	113.9	1.64	5.12	0.17	56	36.19	100.0	100.0	100.0	100.0	100.0
P1T4 - Caribou with $ZnSO_4$	Pb Rougher	12.5	11.00	5.18	0.77	251	36.39	72.4	12.7	47.0	44.1	12.6
	Pb Scavenger	7.2	2.31	5.50	0.23	81	40.05	8.8	7.8	8.0	8.2	8.1
	Zn Rougher	36.4	0.52	10.64	0.15	48	37.96	10.0	76.3	27.6	24.8	38.5
	Tail	43.9	0.38	0.36	0.08	37	33.33	8.8	3.1	17.3	22.9	40.8
	Calculated Head	100.0	1.89	5.07	0.20	71	35.88	100.0	100.0	100.0	100.0	100.0
P1T5 - T4 Incorporating T3 pH Conditions	Pb Rougher	12.9	9.64	5.09	0.80	256	36.07	74.0	12.3	47.9	46.6	12.5
	Pb Scavenger	8.6	1.63	5.23	0.21	66	39.91	8.3	8.4	8.4	8.0	9.2
	Zn Rougher	35.9	0.42	11.49	0.16	47	42.96	9.1	77.0	26.0	23.7	41.5
	Tail	42.5	0.34	0.29	0.09	36	32.22	8.6	2.3	17.7	21.6	36.8
	Calculated Head	100.0	1.68	5.36	0.22	71	37.24	100.0	100.0	100.0	100.0	100.0
P1T6 - T5 Incorporating Decreased $CuSO_4$	Pb Rougher	11.5	10.71	5.35	0.82	270	35.66	71.0	12.0	52.9	58.5	11.8
	Pb Scavenger	8.7	2.01	5.48	0.20	57	39.01	10.1	9.3	9.9	9.3	9.8
	Zn Rougher	54.2	0.39	7.24	0.09	27	38.02	12.3	76.6	28.5	27.9	59.6
	Tail	25.6	0.44	0.43	0.06	9	25.32	6.5	2.1	8.7	4.4	18.8
	Calculated Head	100.0	1.73	5.13	0.18	53	34.58	100.0	100.0	100.0	100.0	100.0
P1T7 - T6 Incorporating Decreased SIPX and Pb Cleaning Milling Curve	Pb Rougher Scavenger Concentrate	15.9	7.44	5.39	0.62	187	35.14	77.1	16.8	58.0	58.6	15.8
	Zn Rougher	18.4	0.56	21.47	0.17	39	28.72	6.7	77.5	18.8	14.2	15.0
	Tail	65.7	0.38	0.44	0.06	21	37.16	16.3	5.7	23.2	27.2	69.2
	Calculated Head	100.0	1.54	5.09	0.17	51	35.29	100.0	100.0	100.0	100.0	100.0
P1T8 - T7 Utilizing Lime Only	Pb Rougher Scavenger Concentrate	12.0	9.94	5.84	0.83	243	37.90	74.3	13.4	56.8	59.6	12.0
	Zn Rougher	14.9	0.71	27.76	0.21	49	27.76	6.6	79.3	18.1	14.9	11.0
	Tail	73.2	0.42	0.52	0.06	17	39.67	19.2	7.3	25.1	25.5	77.0
	Calculated Head	100.0	1.60	5.21	0.17	49	37.69	100.0	100.0	100.0	100.0	100.0

Source: RPC Final Report; Mar 13, 2019; p.16.

Test P1T8 is interesting, considering its use of lime as the only pH modifier throughout the circuits instead of the typical soda ash usage in copper and lead flotation and lime for zinc only (per RPC's standard). P1T8 generated results that are deemed equal to the best tests where the two pH modifiers were at play, both in terms of mass pull and metal recoveries. Lime is a cheaper reagent than soda ash. This is not the approach that was retained as a standard for the roughing stages used to generate feed material to the subsequent cleaning test series though, with P1T7 conditions instead preferred.

P1T1 was carried out using the conditions of the LCT from the RPC 2012 testwork program (MB optimum). P1T2 was then replaced the reagent regime to emulate the Caribou plant circuit, where only lead and zinc circuits are operated. These changes involved dropping SMBS and zinc sulphates in the depressant scheme, relying only on cyanide to prevent the flotation of both pyrite and zinc. Other changes included A5100, which was previously used as the collector for copper and PEX/A3418A, which was used as the collector for lead, being replaced by A3418Ain the roughers and A241 in the scavengers, whereas the zinc circuit collector went from PEX to SIPX. Over twice the amount of copper sulphate was used ahead of the zinc roughers.

Typically, when comparing the combined Cu-Pb rougher concentrates floated with the RPC conditions with the lead concentrate obtained under the Caribou conditions, lead recovery is about 70% vs. 73% at similar mass pulls of 13% to 15%. Zinc results are better with RPC conditions though: 83.5% recovery at a mass pull of 21% to 24% versus Caribou's 74% to 77% at a mass pull in the 35% to 39% range. Even reintroducing zinc sulphate as part of the Caribou depressant regime failed to approach RPC's results, until P1T7, which saw the zinc rougher mass pull reduced to 18.4% but at 77.5% Zn recovery.

Going through the lead cleaning circuit with P1T7 roughing conditions, the tests showed the need for a fine regrind target of 10 to 12 µm P_{80}. Once the amount of collector added was limited, an effective upgrading towards a final lead concentrate was achieved: 49% Pb in test P1CT3 and 46.7% Pb in P1CT4 were obtained, along with lead stage recoveries of 53.4% and 56.4%, respectively, and a zinc deportment of less than 2%.

Zinc cleaning trials saw the zinc rougher concentrate (P1T7 test conditions), and lead cleaning circuit tails (P1CT4 test conditions) being combined as zinc cleaner circuit feed. Ten cleaning tests were completed. SIPX (Caribou's flowsheet) was used as the zinc collector for the first five tests, then PEX (RPC1 recipe) was introduced for remaining ones.

Short flotation times in zinc first cleaner/cleaner-scavenger were initially seen as detrimental to recovery, while increasing copper sulphate addition was successfully attempted in the last tests. The best tests (P1ZCT7 and P1ZCT9, at a finer primary grind P_{80} of 30 µm) were still showing a 21.7% Zn and 15.5% Zn in all the cleaning stages tails combined, although those from the cleaner-scavenger tails were much lower, at 1.4% Zn and 0.9% Zn, respectively, indicating that most of those interstage losses would likely be picked up in a locked cycle situation, allowing the 70% Zn to 72% Zn recovery to third cleaner concentrate to increase towards the high 80% while maintaining a concentrate grade of about 46% Zn at a ~8.5% mass pull. The final concentrate grade was pushed above 49% Zn by adding a fourth cleaning stage, but at the cost of a 2.7% reduction in achieved recovery.

Table 13-16 presents the conditions from P1ZCT9, and Table 13-17 shows the metallurgical outcome associated with it, when A3418A is introduced as an adjunct collector to PEX.

Table 13-16: RPC 2019 Testwork Program – Optimum Zn Cleaning Conditions with Sample P1 (Test P1ZCT9)

Test P1ZCT9	ZCT8 Incorporating a Primary Grind of 80% Passing 28 µm; a Zn Regrind Size of 80% Passing 16 µm, increased $CuSO_4$ and Aero 3418A in Zn Cleaner 1 Instead of PEX												
Stage	Reagents Added, Grams per Feed Ton									Time, Minutes		pH	P_{80} (µm)
	Modifier	PEX	MIBC	Aero 5100	$ZnSO_4$	$Na_2S_2O_5$	NaCN	Aero 3418A	$CuSO_4$	Cond.	Froth		
Blue Mill Grind	-	-	-	-	-	-	-	-	-	-	-	-	28
P1ZCT6 Pb Rougher and Cleaner Conditions													
Zn Rghr Condition 1	Lime	-	-	-	-	-	-	-	1000	5	-	10.3	-
P1ZCT7 Zn Rougher (on Pb scavenger tails and Pb cleaner tails utilizing process water)													
Zn Cleaner (on combined Zn rougher concentrates utilizing process water)													
Regrind	-	-	-	-	-	-	-	-	-	-	-	-	16
Condition	Lime	-	-	-	-	-	-	-	400	5	-	10.5	-
Zn 1st Cleaner	Lime	-	9.5	-	-	-	-	5	-	1	5	10.5	-
Zn 1st Cleaner Scav	Lime	3	-	-	-	-	-	-	-	1	5	10.5	-
Zn 2nd Cleaner	Lime	-	-	-	-	-	-	-	-	-	5	10.8	-
Zn 3rd Cleaner	Lime	-	-	-	-	-	-	-	-	-	5	10.8	-
Zn 4th Cleaner	Lime	-	-	-	-	-	-	-	-	-	5	11.5	-

Source: RPC Final Report; Mar 13, 2019; p.36.

Table 13-17: RPC 2019 Testwork Program – Optimum Zn Cleaning Results with Sample P1 (Test P1ZCT9)

Test ID	Stream	Mass (%)	Grade (%)					Distribution Ratio (%)				
			Pb	Zn	Cu	Ag (g/t)	Fe	Pb	Zn	Cu	Ag	Fe
P1ZCT9 - ZCT8*	Pb 4th Cleaner Conc.	2.0	49.10	3.71	1.05	1144	16.86	62.4	1.5	9.8	35.9	0.9
	Pb Cleaner Scavenger Conc. 1	0.6	3.47	5.40	1.61	147	41.16	1.2	0.6	4.2	1.3	0.6
	Pb Cleaner Scavenger Conc. 2	0.4	2.59	5.40	1.69	115	41.87	0.6	0.4	3.1	0.7	0.4
	Pb Cleaner Scavenger Conc.	1.0	3.11	5.40	1.64	134	41.45	1.9	1.1	7.2	2.0	1.1
	Zn 4th Cleaner Conc.	7.0	1.46	49.39	0.57	89	12.24	6.4	70.0	18.3	9.6	2.3
	Zn 4th Cleaner Tail	0.6	1.85	20.50	1.05	126	30.88	0.7	2.7	3.1	1.3	0.5
	Zn 3rd Cleaner Conc.	7.6	1.49	46.96	0.61	92	13.81	7.1	72.7	21.4	10.9	2.8
	Zn 3rd Cleaner Tail	1.8	1.69	17.80	1.00	112	33.89	1.9	6.5	8.3	3.1	1.6
	Zn 2nd Cleaner Conc.	9.4	1.53	41.37	0.69	96	17.66	9.1	79.2	29.7	14.0	4.4
	Zn 2nd Cleaner Tail	3.4	1.32	7.80	0.71	90	40.75	2.8	5.4	11.2	4.8	3.7
	Zn 1st Cleaner Scavenger Conc.	2.8	1.41	11.20	0.79	103	38.63	2.5	6.4	10.2	4.5	2.9
	Zn 1st Cleaner Scavenger Tail	5.4	0.83	0.79	0.30	42	46.65	2.8	0.9	7.4	3.5	6.6
	Zn 1st Cleaner Conc.	12.9	1.47	32.39	0.69	94	23.83	11.9	84.7	41.0	18.8	8.2
	Zn 1st Cleaner Tail	8.2	1.03	4.36	0.47	63	43.90	5.3	7.2	17.6	7.9	9.5
	Combined Zn Cleaner Tails	11.3	1.18	6.79	0.58	73	41.89	8.3	15.5	30.1	12.7	12.5
	Tail	76.0	0.39	0.36	0.07	30	39.83	18.6	5.6	24.5	35.3	80.4
	Calc. Head	100.0	1.60	4.93	0.22	65	37.65	100.0	100.0	100.0	100.0	100.0

Note: Incorporating a primary grind of 80% passing 28 µm; a zinc regrind size of 80% passing 16 µm, increased $CuSO_4$ and Aero 3418A in Zn Cleaner 1 Instead of PEX. Source: RPC Final Report; Mar 13, 2019; p.39.

The testwork demonstrated that a good quality lead and zinc concentrate could be produced with the sample P1, and with good potential to improve zinc recovery under close-circuited cleaning conditions, given a low zinc deportment to the lead circuit and low losses to the zinc cleaner-scavenger tails.

13.2.2.3.2 Sample P3 Testing

The P3 sample had both copper and lead contents (1.01% Cu and 0.61% Pb) that would have justified separating both metals into distinct concentrates, but as the Caribou mill did not have a copper circuit, only a bulk Cu-Pb float was done ahead of zinc flotation. Some contamination by lead of the resulting copper concentrate is therefore expected.

The best conditions from RPC's 2012 LCT were tried first, which involved the separate production of copper and lead concentrates. The results showed that about as much lead reported to the copper rougher concentrate as to the lead rougher concentrate, at just over 27% each.

The conditions in the Caribou mill were then applied, but some reagents from the LCT were kept in order to delay the flotation of lead into a subsequent Cu-Pb scavenger product. With 75% of copper and 10.7% of lead reporting to the copper rougher concentrate, and 9.4% of copper and 45.7% of lead reporting to the Cu-Pb scavenger concentrate, test P3T6 was relatively successful, although putting 33.2% of zinc recovery in these two products. When combining the copper rougher and Cu-Pb scavenger results for the tests with both products, the total copper recovery remained at 85% to 90%, which is quite good. The conditions of test P3T6 are shown in Table 13-18.

Table 13-18: RPC 2019 Testwork Program – Optimum Roughing Conditions with Sample P3 – Test P3T6

TEST P3T6	T5 Incorporating Cu Cleaning Milling Curve												
Stage	Reagents added, grams per feed ton									Time, minutes		pH	P_{80} (µm)
	Modifier	PEX	MIBC	$Na_2S_2O_5$	$ZnSO_4$	Aero 5100	NaCN	Aero 3418A	$CuSO_4$	Cond.	Froth		
Green Mill Grind		-	-	-	-	-	-	-	-	-	-	-	40
Aeration	Soda Ash	-	-	-	-	-	-	-	-	-	7	8	-
Condition 1	Soda Ash	-	-	-	400	-	-	-	-	-	5	8.5	-
Condition 2	Soda Ash	-	-	800	-	-	-	-	-	-	2	8.5	-
Cu Rougher	Soda Ash	-	15	-	-	10	-	-	-	1	3	8.5	-
Condition 1	Soda Ash	-	-	-	400	-	40	-	-	5	-	9.3	-
Condition 2	Soda Ash	8	-	-	-	-	-	50	-	2	-	-	-
Cu Scavenger	Soda Ash	-	10	-	-	-	-	-	-	1	4	9.3	-
Cu Cleaner (on combined Cu rougher and scavenger concentrates utilizing process water)													
Milling Curve Regrind													
Zn Rougher (Cu Ro Scavenger Tail)													
Condition 1	Lime	-	-	-	-	-	-	-	400	5	-	10.3	-
Condition 2	Lime	8	-	-	-	-	-	50	-	2	-	10.3	-
Zn Rougher 1	Lime	-	10	-	-	-	-	-	-	1	3	10.3	-
Condition	Lime	4	-	-	-	-	-	25	-	2		10.3	-
Zn Rougher 2	Lime	-	5	-	-	-	-	-	-	1	3	10.3	-

Source: RPC Final Report; Mar 13, 2019; p.23.

Test P3T7 repeated the test P3T6 conditions but introduced a finer primary grind of 28 µm versus 40 µm. Table 13-19 shows the finer primary grind tends to decrease the mass pull of the roughers—if the CuPb scavenger product in P3T7 is excluded—at equal copper recoveries, and improves lead rejection from the copper rougher.

Table 13-19: RPC 2019 Testwork Program – Effect of Primary Grind on Rougher Results – Sample P3

Test ID	Stream	Mass (%)	Chemical Assays (%)					Distribution Ratio (%)				
			Pb	Zn	Cu	Ag (g/t)	Fe	Pb	Zn	Cu	Ag	Fe
P3T6 – T5 Incorporating Cu Cleaning Milling Curve	CuPb Rghr Scavenger Conc	12.9	2.16	2.29	5.21	131	38.27	56.2	27.8	78.8	60.2	13.2
	Zn Rougher	23.8	0.35	2.90	0.39	20	41.39	17.0	65.0	10.8	17.3	26.3
	Tail	63.3	0.21	0.12	0.14	10	35.87	26.8	7.2	10.4	22.5	60.6
	Measured Head	-	0.63	1.31	1.01	44	42.60	-	-	-	-	-
	Calculated Head	100.0	0.50	1.06	0.85	28	37.49	100.0	100.0	100.0	100.0	100.0
P3T7 - T5 Incorporating 80% Passing 28 µm Grind Size	Cu Rougher	8.8	1.85	1.94	9.59	151	37.20	33.3	13.3	78.4	44.2	8.6
	CuPb Scavenger	10.2	1.82	2.19	1.14	74	41.70	37.9	17.4	10.8	25.1	11.1
	Zn Rougher	16.2	0.34	4.95	0.36	25	40.57	11.5	62.7	5.4	13.4	17.3
	Tail	64.8	0.13	0.13	0.09	8	37.20	17.3	6.6	5.4	17.3	63.1
	Measured Head	-	0.63	1.31	1.01	44	42.60	-	-	-	-	-
	Calculated Head	100.0	0.49	1.28	1.08	30	38.21	100.0	100.0	100.0	100.0	100.0

Source: RPC Final Report; Mar 13, 2019; p.24.

Copper cleaning tests with sample P3 proceeded by relying on RPC's 2012 recipe: regrind target to 12 µm; sodium silicate and zinc sulphate as depressants; A5100 and A3418A as collectors in the roughers; and A5100 as a collector in the cleaners. The best test conditions (P3CT3) are listed in Table 13-20 and the results are summarized in Table 13-21.

Table 13-20: RPC 2019 Testwork Program – Optimum Cu Cleaning Conditions – Sample P3

TEST P3CT3	CT2 Incorporating Decreased $ZnSO_4$ and SMBS Dosages with Aero 5100 in the 1st Cu Cleaner, Shorter Cu Cleaning Float Times and Zn Milling Curve												
Stage	Reagents added, grams per feed ton									Time, minutes		pH	P_{80} (µm)
	Modifier	PEX	MIBC	$Na_2S_2O_5$	$ZnSO_4$	Aero 5100	NaCN	Aero 3418A	$CuSO_4$	Cond.	Froth		
Blue Mill Grind	-	-	-	-	-	-	-	-	-	-	-	-	40
Cu Rougher	-	-	-	-	-	-	-	-	-	-	-	-	-
Aeration	Soda Ash	-	-	-	-	-	-	-	-	-	7	8	-
Condition 1	Soda Ash	-	-	-	400	-	-	-	-	-	5	8.5	-
Condition 2	Soda Ash	-	-	800	-	-	-	-	-	-	2	8.5	-
Cu Rougher	Soda Ash	-	15	-	-	10	-	-	-	1	3	8.5	-
Condition 1	Soda Ash	-	-	-	400	-	40	-	-	5	-	9.3	-
Condition 2	Soda Ash	8	-	-	-	-	-	50	-	2	-	-	-
Cu Scavenger	Soda Ash	-	10	-	-	-	-	-	-	1	4	9.3	-
Cu Cleaner (on combined Cu rougher and scavenger concentrates utilizing process water)													
Regrind	-	-	-	-	-	-	-	-	-	-	-	-	12
Condition 1	Soda Ash	-	-	-	400	-	-	-	-	5	-	8.1	-
Condition 2	Soda Ash	-	-	100	-	-	-	-	-	2	-	8.1	-
Cu 1st Cleaner 1	Soda Ash	-	15	-	-	5	-	-	-	1	8.5	8.1	-
Condition	Soda Ash	-	-	-	-	5	-	-	-	2	-	8.2	-
Cu Cleaner Scav	Soda Ash	-	5	-	-	-	-	-	-	1	8.5	8.2	-
Cu 2nd Cleaner	Soda Ash	-	10	-	-	-	-	-	-	1	4	8.5	-
Cu 3rd Cleaner	Soda Ash	-	5	-	-	-	-	-	-	1	4	8.5	-
Cu 4th Cleaner	Soda Ash	-	5	-	-	-	-	-	-	1	4	8.5	-
Zn Rougher (on Cu scavenger tails and Cu cleaner scavenger tails)													
Condition 1	Lime	-	-	-	-	-	-	-	1000	5	-	10.3	
Condition 2	Lime	8	-	-	-	-	-	50	-	2	-	10.3	
Zn Rougher 1	Lime	-	10	-	-	-	-	-	-	1	3	10.3	
Condition	Lime	4	-	-	-	-	-	25	-	2		10.3	
Zn Rougher 2	Lime	-	5	-	-	-	-	-	-	1	3	10.3	

Source: RPC Final Report; Mar 13, 2019; p.30.

Table 13-21: RPC 2019 Testwork Program – Optimum Cu Cleaning Results – Sample P3

Test ID	Stream	Mass (%)	Chemical Assays (%)					Distribution Ratio (%)				
			Pb	Zn	Cu	Ag (g/t)	Fe	Pb	Zn	Cu	Ag	Fe
P3CT3 - CT2 Incorporating Decreased $ZnSO_4$ and SMBS Dosages with Aero 5100 in the 1st Cu Cleaner, Shorter Cu Cleaning Float Times and Zn Milling Curve	Cu 4th Cleaner Conc.	2.2	9.90	1.47	23.20	632	28.10	39.8	2.7	47.5	55.5	1.6
	Cu 4th Cleaner Tail	0.2	4.24	2.83	10.90	233	37.20	1.7	0.5	2.3	2.1	0.2
	Cu 3rd Cleaner Conc.	2.4	9.37	1.60	22.06	595	28.95	41.6	3.2	49.8	57.6	1.8
	Cu 3rd Cleaner Tail	0.5	3.77	2.63	10.90	215	37.60	3.4	1.1	4.9	4.2	0.5
	Cu 2nd Cleaner Conc.	2.9	8.44	1.77	20.19	531	30.39	44.9	4.2	54.7	61.8	2.3
	Cu 2nd Cleaner Tail	1.0	1.87	2.89	7.19	132	38.60	3.4	2.4	6.7	5.3	1.0
	Cu Cleaner Scavenger Conc.	1.4	1.75	2.67	6.83	112	39.20	4.6	3.1	9.1	6.4	1.4
	Cu Cleaner Scavenger Tail	6.2	0.49	2.50	1.54	21	41.00	5.6	12.9	9.0	5.3	6.7
	Cu 1st Cleaner Conc.	3.9	6.75	2.06	16.85	429	32.50	48.4	6.6	61.5	67.1	3.3
	Cu 1st Cleaner Tail	7.6	0.72	2.53	2.52	38	40.67	10.2	16.0	18.1	11.6	8.1
	Zn Rougher Conc	17.7	0.52	5.53	1.07	24	40.90	17.0	81.3	17.8	17.2	19.0
	Tail	77.1	0.21	0.14	0.16	*3*	37.70	30.0	9.0	11.6	9.4	76.3
	Meas. Head	-	0.63	1.31	1.01	-	42.60	-	-	-	-	-
	Calc. Head	100.0	0.54	1.20	1.06	25	38.09	100.0	100.0	100.0	100.0	100.0

Source: RPC Final Report; Mar 13, 2019; p.31.

Table 13-21 indicates a capability for upgrading the copper concentrate to 22% Cu but at a low copper recovery of 50% (or 20% Cu with 55% recovery) and 57.6% for silver. With 18.1% of the copper fed to the cleaning circuit lost in the first cleaner tails, there is potential for improving recovery by increasing either flotation time or collector addition in this stage.

The zinc cleaning stage incorporated a regrinding step to a target P_{80} at the 16 to 18 µm range. Three cleaning tests were completed. PEX was the only collector used, despite sodium isopropyl xanthate (SIPX) being Caribou's standard, on the basis of the results seen with sample P1.

Shorter flotation times were introduced after the first test failed to provide an upgrading of more than 25.5% Zn in the fourth cleaner concentrate. That test, P3ZCT1, as well as P3ZCT2, did repeat the 80% rougher zinc recovery but P3ZCT3 provided only 67.3%. The underlying copper cleaning stage was also a failure, upgrading to 18.7% Cu but only holding 30.4% of the copper, whereas the optimized copper cleaner was closer to 50%. This stems from changes from the standard that were tried while the open circuit tests were ongoing.

Although P3ZTC3 provided the best upgrading to a respectable grade of 44.6% Zn, its low recovery of 58.3% was largely caused by the failure of the rougher stage. Only P3ZCT2 remains as presenting the best overall outcome, with a more marginal 42.3% Zn achieved in the fourth cleaner concentrate. This value is partially the result of copper dilution, with a grade of nearly 3.8% Cu in the final zinc concentrate.

Table 13-22 presents the test conditions for P3ZCT2, while Table 13-23 summarizes the metallurgical outcome.

Table 13-22: RPC 2019 Testwork Program – Zn Cleaning Circuit Test P3ZCT2 Conditions

TEST P3ZCT2	ZCT1 Incorporating Shorter Float Times on the Cleaners and Additional $CuSO_4$ in Zn Rougher												
Stage	Reagents added, grams per feed ton									Time, minutes		pH	P_{80} (µm)
	Modifier	PEX	MIBC	Aero 5100	$ZnSO_4$	$Na_2S_2O_5$	NaCN	Aero 3418A	$CuSO_4$	Cond.	Froth		
P3ZCT1 Cu Rougher and Scavenger													
Cu Cleaner (on combined Cu rougher and scavenger concentrates utilizing process water)													
Regrind	-	-	-	-	-	-	-	-	-	-	-	-	12
Condition 1	Soda Ash	-	-	-	400	-	-	-	-	5	-	8.1	-
Condition 2	Soda Ash	-	-	-	-	100	-	-	-	2	-	8.1	-
Cu 1st Cleaner 1	Soda Ash	-	15	5	-	-	-	-	-	1	5	8.1	-
Condition	Soda Ash	-	-	5	-	-	-	-	-	2	-	8.2	-
Cu Cleaner Scavenger	Soda Ash	-	5	-	-	-	-	-	-	1	4	8.2	-
Cu 2nd Cleaner	Soda Ash	-	10	-	-	-	-	-	-	1	4	8.5	-
Cu 3rd Cleaner	Soda Ash	-	5	-	-	-	-	-	-	1	4	8.5	-
Cu 4th Cleaner	Soda Ash	-	5	-	-	-	-	-	-	1	4	8.5	-
Zn Rghr Condition 1	Lime	-	-	-	-	-	-	-	1000	5	-	10.3	-
P3ZCT1 Zn Rougher (on Cu Scavenger tails and Cu Cleaner Scavenger tails utilizing process water)													
Zn Cleaner (on combined Zn rougher concentrates utilizing process water)													
Regrind	-	-	-	-	-	-	-	-	-	-	-	-	12
Condition	Lime	-	-	-	-	-	-	-	400	5	-	10.5	-
Zn 1st Cleaner	Lime	-	9.5	-	-	-	-	5	-	1	5	10.5	-
Zn 1st Cleaner Scavenger	Lime	3	-	-	-	-	-	-	-	1	5	11.0	-
Zn 2nd Cleaner	Lime	-	-	-	-	-	-	-	-	-	5	10.8	-
Zn 3rd Cleaner	Lime	-	-	-	-	-	-	-	-	-	5	11.0	-
Zn 4th Cleaner	Lime	-	-	-	-	-	-	-	-	-	5	10.2	-

Source: RPC Final Report; Mar 13, 2019; p.41.

Table 13-23: RPC 2019 Testwork Program – Zn Cleaning Circuit Test P3ZCT2 Results

Test ID	Stream	Mass (%)	Grade (%)					Distribution Ratio (%)				
			Pb	Zn	Cu	Ag (g/t)	Fe	Pb	Zn	Cu	Ag	Fe
P3ZCT2 - ZCT1 Incorporating Shorter Float Times on the Cleaners and Additional $CuSO_4$ in Zn Rougher	Cu 4th Cleaner Conc.	2.3	10.01	1.63	22.39	704	26.04	37.9	2.8	46.9	49.6	1.7
	Cu 4th Cleaner Tail	0.2	3.39	2.95	13.69	252	32.75	1.3	0.5	2.9	1.8	0.2
	Cu 3rd Cleaner Conc.	2.5	9.41	1.75	21.60	663	26.65	39.2	3.3	49.8	51.4	1.9
	Cu 3rd Cleaner Tail	0.4	2.33	3.26	9.92	178	36.48	1.5	0.9	3.4	2.1	0.4
	Cu 2nd Cleaner Conc.	2.9	8.49	1.95	20.08	600	27.93	40.7	4.2	53.2	53.5	2.3
	Cu 2nd Cleaner Tail	1.0	1.89	3.14	7.28	134	34.60	3.2	2.4	6.8	4.2	1.0
	Cu Cleaner Scavenger Conc.	1.4	1.72	2.97	7.06	111	35.47	3.9	3.1	8.9	4.7	1.4
	Cu 1st Cleaner Conc.	3.9	6.76	2.26	16.73	478	29.68	43.9	6.6	60.0	57.7	3.4
	Combined Cu Cleaner Tails	1.6	2.20	3.14	8.78	161	34.78	5.9	3.8	13.1	8.1	1.6
	Zn 4th Cleaner Conc.	2.2	1.21	42.23	3.77	132	11.12	4.5	70.6	7.7	9.1	0.7
	Zn 4th Cleaner Tail	0.3	1.15	12.38	6.87	151	27.66	0.7	3.2	2.2	1.6	0.3
	Zn 3rd Cleaner Conc.	2.5	1.20	38.23	4.19	135	13.34	5.1	73.8	9.9	10.7	1.0
	Zn 3rd Cleaner Tail	0.6	0.87	4.79	4.80	103	33.16	0.8	2.0	2.5	1.8	0.5
	Zn 2nd Cleaner Conc.	3.1	1.14	32.17	4.30	129	16.93	6.0	75.8	12.4	12.5	1.5
	Zn 2nd Cleaner Tail	1.4	0.61	0.97	2.02	53	34.22	1.5	1.1	2.7	2.4	1.4
	Zn 1st Cleaner Scavenger Conc.	2.6	0.59	0.75	1.51	46	37.03	2.5	1.5	3.6	3.7	2.8
	Zn 1st Cleaner Scavenger Tail	8.1	0.35	0.18	0.36	16	35.72	4.7	1.1	2.7	4.0	8.4
	Zn 1st Cleaner Conc.	4.6	0.97	22.26	3.57	105	22.42	7.4	76.9	15.1	14.9	3.0
	Zn 1st Cleaner Tail	10.6	0.41	0.32	0.64	23	36.04	7.3	2.6	6.3	7.7	11.2
	Combined Zn Cleaner Tails	10.4	0.44	0.94	1.04	30	35.11	7.7	7.4	10.1	9.9	10.7
	Zn Rougher Conc	15.2	0.58	6.90	1.52	48	31.95	14.7	79.5	21.4	22.7	14.2
	Tail	79.6	0.28	0.18	0.13	6	34.78	37.4	10.9	9.6	14.9	81.0
	Calc. Head	100.0	0.60	1.32	1.08	32	34.16	100.0	100.0	100.0	100.0	100.0

Source: RPC Final Report; Mar 13, 2019; p.43.

13.2.2.3.3 Flotation Testing – P2 Sample

The P2 sample was prepared after work on P1 and P3 was well advanced. It consisted of lower-grade material, which once composited, yielded 0.21% Cu, 1.18% Pb, 3.88% Zn, 37 g/t Ag, and 37.6% Fe.

Four roughing tests were conducted, similar to those realized with sample P1. The first was based on the RPC 2012 work and the remaining were based on Caribou's approach, with the late inclusion of zinc sulphate as an added depressant. The results of the roughing tests carried out with the P2 sample, which included an attempt at producing a separate copper concentrate along the lines of the RPC 2012 testwork despite the low head grade of 0.21% Cu, are presented in Table 13-24. P2T4 provided the best results for the Cu-Pb circuit, both in terms of recovery, zinc rejection, and mass pull (indicative of pyrite rejection). The zinc circuit results were marginally better with the RPC standard (test P2T1) and better with either P2T2 or P2T3 than P2T4. P2T1 (79.9%) returned a higher zinc recovery than P2T2 or P2T3 (75.6% to 75.2%) and P2T4 (72.4%) and mass pull (16.9% vs. 18.7% to 19.0% with P2T3 and P2T4). P2T3 yielded an abnormally high mass pull of 38% to the zinc rougher concentrate, due to moving the first soda ash addition from the aeration stage and into the grinding mill.

Testwork on the lead cleaning circuit proceeded using P2T3 roughing conditions as the standard for feed preparation, along with a regrind target P_{80} of either 20 or 10 µm. The cleaner conditions were based on Caribou's standard practices with lime as the pH modifier, cyanide as the depressant, and A3418A as the collector.

Table 13-25 illustrates the test conditions that provided the best results (P2CT3), and Table 13-26 summarizes the results. Tests P2CT1 and P2CT2 had the same test conditions, except P2CT2 had a regrind of 11 µm and P2CT1 had a regrind of 20 µm. P2CT3 was had a P_{80} regrind of 11 µm and an added zinc rougher flotation of the combined lead rougher tails and all the tails from the cleaning stages.

As shown in Table 13-26, the finer regrind of 11 µm in P2CT2 helped increase the final lead concentrate grade from 41.6% Pb, at the 20 µm regrind, to 55.4% Pb, with almost equivalent recovery (Note: the reported third and second cleaner lead grades and recovery data in Table 13-26 are erroneous: the third cleaner concentrate grade should read 50.6% and its lead recovery 40.7%; while the second cleaner concentrate should read 39.6% Pb and 42% recovery). P2CT3, with a fine regrind as well, achieved 43.7% Pb as fourth cleaner concentrate grade along with a 48.9% recovery. That is compared with the 39.4% recovery achieved in P2CT2.

The added zinc rougher achieved 81% Zn recovery (or 85.7% as stage recovery). Zinc cleaning trials were not attempted with the P2 sample.

The testwork demonstrated that a good quality lead concentrate could be produced with sample P2. A decent zinc rougher recovery was also exhibited.

Table 13-24: RPC 2019 Testwork Program – P2 Roughing Results

Test ID	Stream	Mass (%)	Chemical Assays (%)					Distribution Ratio (%)				
			Pb	Zn	Cu	Ag (g/t)	Fe	Pb	Zn	Cu	Ag	Fe
P2T1 - MB Optimum	Cu Rougher	6.0	10.50	5.34	1.39	245	34.9	54.1	8.1	38.8	28.4	5.6
	Pb Rougher	5.5	2.56	5.13	0.77	147	40.33	12.0	7.1	19.6	15.5	5.8
	Zn Rougher	16.9	0.62	18.97	0.24	50	36.04	8.9	79.9	18.5	16.3	16.0
	Tail	71.6	0.41	0.28	0.07	29	38.52	25.0	5.0	23.1	39.8	72.6
	Meas. Head	-	1.18	3.88	0.21	37	37.62	-	-	-	-	-
	Calc. Head	100.0	1.17	4.01	0.22	52	37.98	100.0	100.0	100.0	100.0	100.0
P2T2 - Current Caribou	Pb Rougher	7.8	8.05	5.85	1.20	279	36.32	54.5	11.6	44.2	45.2	7.7
	Pb Scavenger	5.3	2.09	6.15	0.36	80	37.79	9.7	8.3	9.2	8.9	5.5
	Zn Rougher	38.0	0.54	7.85	0.16	30	38.92	17.9	75.6	28.2	23.5	40.3
	Tail	48.9	0.42	0.36	0.08	22	34.90	17.9	4.5	18.5	22.4	46.5
	Meas. Head	-	1.18	3.88	0.21	37	37.62	-	-	-	-	-
	Calc. Head	100.0	1.15	3.94	0.21	48	36.69	100.0	100.0	100.0	100.0	100.0
P2T3 - T2 Incorporating $ZnSO_4$, T1 pH Conditions and Decreased $CuSO_4$ & SIPX	Pb Rougher	6.5	9.41	5.67	1.45	265	34.37	51.7	9.0	51.4	36.3	6.1
	Pb Scavenger	6.3	2.06	5.85	0.36	83	38.60	10.9	8.9	12.1	11.0	6.6
	Zn Rougher	18.7	0.66	16.61	0.18	47	32.64	10.4	75.2	18.0	18.3	16.7
	Tail	68.5	0.47	0.42	0.05	24	37.69	27.0	7.0	18.5	34.4	70.5
	Meas. Head	-	1.18	3.88	0.21	37	37.62	-	-	-	-	-
	Calc. Head	100.0	1.19	4.14	0.19	48	36.58	100.0	100.0	100.0	100.0	100.0
P2T4 - T3 Incorporating Increased SIPX and Pb Cleaning Milling Curve	Pb Rghr Scavenger Conc	11.5	5.90	5.39	0.94	163	39.58	65.0	18.0	62.2	54.8	12.5
	Zn Rougher	19.0	0.56	13.11	0.16	30	36.03	10.3	72.4	17.8	16.7	18.8
	Tail	69.6	0.37	0.47	0.05	14	35.82	24.7	9.5	20.1	28.5	68.7
	Meas. Head	-	1.18	3.88	0.21	37	37.62	-	-	-	-	-
	Calc. Head	100.0	1.04	3.43	0.17	34	36.29	100.0	100.0	100.0	100.0	100.0

Source: RPC Final Report; Mar 13, 2019; p.59.

Table 13-25: RPC 2019 Testwork Program – Optimum Pb Cleaning Conditions with Sample P2 – Test P2CT3

TEST P2CT3	CT2 Incorporating Zn Regrind Milling Curve and Additional $CuSO_4$ in Zn Rougher												
Stage	Reagents added, grams per feed ton									Time, minutes		pH	P_{80} (µm)
	Modifier	SIPX	MIBC	R241	$ZnSO_4$	Aero 5100	NaCN	Aero 3418A	$CuSO_4$	Cond.	Froth		
Blue Mill Grind	-	-	-	-	112	-	45	-	-	-	-	8	40
Pb Rougher	-	-	-	-	-	-	-	-	-	-	-	-	-
Aeration	Lime	-	-	-	-	-	-	-	-	-	7	8.5	-
Pb Rougher 1	Lime	-	10	-	-	-	-	11	-	1	1	8.5	-
Pb Rougher 2	Lime	-	-	-	-	-	-	11	-	1	1	8.5	-
Pb Rougher 3	Lime	-	-	-	-	-	-	11	-	1	1	8.5	-
Pb Scavenger 1	Lime	-	2	1.5	-	-	-	-	-	2	2	9.3	-
Pb Scavenger 2	Lime	-	-	1	-	-	-	-	-	1	2	9.3	-
Pb Cleaner (on combined Pb rougher and scavenger concentrates utilizing process water)													
Regrind		-	-	-	-	-	-	-	-	-	-	-	11
Condition	Lime	-	-	-	-	-	25	-	-	2	-	8.1	-
Pb 1st Cleaner 1	Lime	-	0.5	-	-	-	-	0.5	-	1	0.5	8.1	-
Pb 1st Cleaner 2	Lime	-	0.75	-	-	-	-	0.5	-	1	1.5	8.1	-
Pb 1st Cleaner 3	Lime	-	0.75	-	-	-	-	0.5	-	1	2.5	8.1	-
Pb Cleaner Scavenger 1	Lime	2	-	-	-	-	-	-	-	2	2	8.1	-
Pb Cleaner Scavenger 2	Lime	2	-	-	-	-	-	-	-	2	2	8.1	-
Condition	Lime	-	-	-	-	-	15	-	-	2	-	8.2	-
Pb 2nd Cleaner 1	Lime	-	0.5	-	-	-	-	-	-	1	4.5	8.2	-
Pb 2nd Cleaner 2	Lime	-	0.75	-	-	-	-	-	-	1	4	8.2	-
Pb 3rd Cleaner	Lime	-		-	-	-	-	-	-		4	8.5	-
Pb 4th Cleaner	Lime	-	-	-	-	-	-	-	-		4	8.5	-
Zn Rougher (on Pb scavenger tails and Pb cleaner tails utilizing process water)													
Condition 1	Lime	-	-	-	-	-	-	-	800	5	-	10.3	-
Condition 2	Lime	71	-	-	-	-	-	-	-	2	-	10.3	-
Zn Rougher 1	Lime	-	12	-	-	-	-	-	-	1	1	10.3	-
Zn Rougher 2	Lime	16	12	-	-	-	-	-	-	1	2	10.3	-
Zn Rougher 3	Lime	-	12	-	-	-	-	-	-	1	3	10.3	-
Zn Cleaner (on combined Zn rougher concentrates utilizing process water)													
Milling Curve Regrind													

Source: RPC Final Report; Mar 13, 2019; p.62.

Table 13-26: RPC 2019 Testwork Program – Pb Cleaning Results with Sample P2

Test ID	Stream	Mass (%)	Chemical Assays (%)					Distribution Ratio (%)				
			Pb	Zn	Cu	Ag (g/t)	Fe	Pb	Zn	Cu	Ag	Fe
P2CT1 – 20 µm Pb Regrind Size	Pb 4th Cleaner Conc.	1.4	34.92	6.16	3.95	531	21.02	41.6	2.3	5.7	21.9	0.7
	Pb Cleaner Scavenger Conc.	1.0	4.08	7.54	1.37	295	38.92	3.6	2.1	1.5	9.0	0.9
	Combined Cleaner Tails	4.9	1.93	5.21	0.63	141	42.59	8.4	7.0	3.4	21.2	5.0
	Pb Rghr Scavenger Tail	92.7	0.57	3.53	0.9	17	42.14	46.4	88.7	89.4	47.9	93.4
	Measured Head	-	1.18	3.88	0.21	37	37.62	-	-	-	-	-
	Calculated Head	100.0	1.14	3.69	0.93	33	41.84	100.0	100.0	100.0	100.0	100.0
P2CT2 – 11 µm Pb Regrind Size	Pb 4th Cleaner Conc.	0.8	55.37	4.13	4.2	581	10.19	39.4	0.9	10.1	11.2	0.2
	Pb 4th Cleaner Tail	0.1	12.80	8.90	4.76	541	30.29	1.3	0.3	1.6	1.5	0.1
	Pb 3rd Cleaner Conc.[1]	0.9	50.64	4.66	4.26	576	12.42	40.7	1.2	11.7	12.7	0.3
	Pb 3rd Cleaner Tail	0.3	6.38	8.80	3.48	500	34.09	1.4	0.6	2.5	2.9	0.2
	Pb 2nd Cleaner Conc.[1]	1.2	39.58	5.70	4.07	557	17.85	42.0	1.8	14.2	15.6	0.5
	Pb 2nd Cleaner Tail	0.6	4.66	7.80	2.05	372	37.79	2.2	1.2	3.3	4.8	0.5
	Pb Cleaner Scavenger Conc.	0.9	6.99	8.17	2.03	364	34.77	5.7	2.0	5.6	8.0	0.8
	Pb Cleaner Scavenger Tail	5.5	1.41	4.73	0.51	112	39.45	6.6	6.8	8.1	14.2	5.5
	Pb 1st Cleaner Conc.	1.8	27.94	6.40	3.40	495	24.50	44.2	3.0	17.5	20.4	1.0
	Pb 1st Cleaner Tail	6.4	2.23	5.24	0.73	149	38.76	12.3	8.9	13.6	22.2	6.3
	Pb Rghr Scavenger Tail	91.8	0.55	3.65	0.26	27	39.64	43.3	88.2	68.8	57.4	92.6
	Measured Head	-	1.18	3.88	0.21	37	37.62	-	-	-	-	-
	Calculated Head	100.0	1.17	3.80	0.35	43	39.30	100.0	100.0	100.0	100.0	100.0
P2CT3 - CT2 Incorporating Zn Regrind Milling Curve and Additional $CuSO_4$ in Zn Rougher	Pb 4th Cleaner Conc.	1.3	43.66	5.85	4.32	564	16.34	48.9	2.1	25.2	15.7	0.5
	Pb Cleaner Scavenger Conc.	1.3	4.88	8.60	1.41	287	37.37	5.9	3.4	8.8	8.6	1.1
	Zn Rougher	12.7	1.19	21.94	0.52	88	32.10	13.5	81.0	30.7	24.9	9.2
	Tail	84.7	0.42	0.55	0.09	27	46.92	31.7	13.5	35.3	50.8	89.2
	Measured Head		1.18	3.88	0.21	37	37.62	-	-	-	-	-
	Calculated Head	100.0	1.12	3.45	0.22	45	44.52	100.0	100.0	100.0	100.0	100.0

Note 1. P2CT2 grades and recoveries for 1st, 2nd and 3rd cleaner concentrates corrected from erroneous values presented in RPC's report. Source: RPC Final Report; Mar 13, 2019; p.63.

13.2.2.4 Aging Effect on Flotation Response – P1 Sample

Some tests were carried out with sample P1 to test the effect of aging on the flotation response. The material was stage-crushed to a nominal 6 mesh size, ready for grinding per the standard flotation procedures, and left on the work bench for variable periods of time. All flotation tests were then carried out under the same roughing conditions of test P1T7. The outcome of the tests, summarized in Table 13-27, indicate that the recoveries of copper and lead improved somewhat, but with an increase of the mass pull from 15.9% for the base case to 25% to 33% for the aged material. As for zinc, a drop from an initial 77.5% recovery to 67% to 71% is seen. This is accompanied by an increase in mass pull to the zinc rougher concentrate, from the base case at 18.4% to nearly 30% from 3 to 7 days of aging, and then to 50% for 2 to 4 weeks of aging.

These indications therefore point to a pyrite activation issue from aging the material and not to a reduction in potential recovery by surface oxidation of the economic minerals surfaces, except for zinc to a limited extent. This pyrite activation issue is likely to lead to lower final concentrate grades if it is not countered by modifications to the reagent regime in the cleaning stages.

Table 13-27: RPC 2019 Testwork Program – Aging Effect on Rougher Flotation Response with Sample P1

Test ID	Stream	Mass (%)	Chemical Assays					Distribution Ratio				
			Pb (%)	Zn (%)	Cu (%)	Ag (g/t)	Fe (%)	Pb (%)	Zn (%)	Cu (%)	Ag (%)	Fe (%)
P1T7 - 0 Days of Aging	Pb Rghr Scavenger Conc	15.9	7.44	5.39	0.62	187	35.14	77.1	16.8	58.0	58.6	15.8
	Zn Rougher	18.4	0.56	21.47	0.17	39	28.72	6.7	77.5	18.8	14.2	15.0
	Tail	65.7	0.38	0.44	0.06	21	37.16	16.3	5.7	23.2	27.2	69.2
	Meas. Head	-	1.70	5.15	0.16	68	37.01	-	-	-	-	-
	Calc. Head	100.0	1.54	5.09	0.17	51	35.29	100.0	100.0	100.0	100.0	100.0
P1A1 - 3 Days of Aging	Pb Rghr Scavenger Conc	25.7	5.54	5.52	0.49	153	38.21	82.7	30.1	64.9	67.2	27.8
	Zn Rougher	29.4	0.40	10.73	0.13	33	35.83	6.9	67.0	19.0	16.6	29.8
	Tail	44.9	0.40	0.30	0.07	21	33.40	10.4	2.9	16.1	16.1	42.4
	Meas. Head	-	1.70	5.15	0.16	68	37.01	-	-	-	-	-
	Calc. Head	100.0	1.72	4.71	0.20	58	35.35	100.0	100.0	100.0	100.0	100.0
P1A2 - 1 Week of Aging	Pb Rghr Scavenger Conc	26.2	5.66	5.67	0.49	143	39.22	82.2	28.2	67.0	67.8	28.2
	Zn Rougher	27.0	0.46	13.33	0.13	31.05	36.73	6.9	68.5	18.3	15.2	27.2
	Tail	46.8	0.42	0.37	0.06	20	34.60	10.9	3.3	14.7	17.0	44.5
	Meas. Head	-	1.70	5.15	0.16	68	37.01	-	-	-	-	-
	Calc. Head	100.0	1.80	5.26	0.19	55	36.38	100.0	100.0	100.0	100.0	100.0
P1A3 - 2 Weeks of Aging	Pb Rghr Scavenger Conc	26.5	5.53	5.28	0.47	131	40.90	82.1	26.8	61.7	75.0	28.0
	Zn Rougher	50.1	0.41	7.45	0.12	19	43.19	11.5	71.4	29.0	20.5	55.9
	Tail	23.3	0.49	0.40	0.08	9	26.70	6.4	1.8	9.3	4.5	16.1
	Meas. Head	-	1.70	5.15	0.16	68	37.01	-	-	-	-	-
	Calc. Head	100.0	1.79	5.23	0.20	46	38.73	100.0	100.0	100.0	100.0	100.0
P1A4 - 4 Weeks of Aging	Pb Rghr Scavenger Conc	33.8	4.94	5.36	0.43	114	39.92	86.1	36.0	68.9	66.4	36.1
	Zn Rougher	50.3	0.40	6.32	0.11	33	41.20	10.3	63.2	25.1	28.5	55.6
	Tail	15.9	0.44	0.24	0.08	19	19.50	3.6	0.8	6.0	5.2	8.3
	Meas. Head	-	1.70	5.15	0.16	68	37.01	-	-	-	-	-
	Calc. Head	100.0	1.94	5.03	0.21	58	37.32	100.0	100.0	100.0	100.0	100.0

Source: RPC Final Report; Mar 13, 2019; p.70.

13.2.2.5 Flotation Testwork – Locked Cycle Tests

Both the P1 and P3 samples were tested in LCT. The configuration used for both was the same, except P1 was floating lead and zinc, while P3 targeted copper and zinc. The zinc cleaning circuit in both cases did not use the conventional counter-current arrangement, but had all the zinc cleaning stage tails recombined and sent back to the zinc first cleaner feed.

Overall, the reagent conditions were similar, except zinc sulphate and SMBS were used as depressants for P3 copper cleaning, and only cyanide was used for the P3 lead cleaning circuit (Table 13-28). The reagent additions were not modulated to reflect the head grades of the samples.

Table 13-28: RPC 2019 Testwork Program – P3 Locked Cycle Test Conditions

TEST P3LC	Locked Cycle Comprising of Six Cycles												
Stage	Reagents added, grams per feed tonne									Time, minutes		pH	P_{80} (µm)
	Modifier	PEX	MIBC	Aero 5100	$ZnSO_4$	$Na_2S_2O_5$	NaCN	Aero 3418A	$CuSO_4$	Cond.	Froth		
Blue Mill Grind	-	-	-	-	-	-	-	-	-	-	-	-	40
Cu Rougher	-	-	-	-	-	-	-	-	-	-	-	-	-
Aeration	Soda Ash	-	-	-	-	-	-	-	-	-	7	8	-
Condition 1	Soda Ash	-	-	-	400	-	-	-	-	5	-	8.5	-
Condition 2	Soda Ash	-	-	-	-	800	-	-	-	2	-	8.5	-
Cu Rougher	Soda Ash	-	15	10	-	-	-	-	-	1	5	8.5	-
Condition 1	Soda Ash	-	-	-	400	-	40	-	-	5	-	9.3	-
Condition 2	Soda Ash	8	-	-	-	-	-	50	-	2	-	9.3	-
Cu Scavenger	Soda Ash	-	10	-	-	-	-	-	-	1	5	9.3	-
Cu Cleaner (on combined Cu rougher and scavenger concentrates utilizing process water)													
Regrind	-	-	-	-	-	-	-	-	-	-	-	-	16
Condition 1	Soda Ash	-	-	-	400	-	-	-	-	5	-	8.1	-
Condition 2	Soda Ash	-	-	-	-	100	-	-	-	2	-	8.1	-
Cu 1st Cleaner 1	Soda Ash	-	15	5	-	-	-	-	-	1	8.5	8.1	-
Condition	Soda Ash	-	-	5	-	-	-	-	-	2	-	8.2	-
Cu Cleaner Scav	Soda Ash	-	5	-	-	-	-	-	-	1	8.5	8.2	-
Cu 2nd Cleaner	Soda Ash	-	10	-	-	-	-	-	-	1	5	8.5	-
Cu 3rd Cleaner	Soda Ash	-	5	-	-	-	-	-	-	1	5	8.5	-
Cu 4th Cleaner	Soda Ash	-	5	-	-	-	-	-	-	1	5	8.5	-
Zn Rougher (on Cu Scavenger tails)													
Condition 1	Lime	-	-	-	-	-	-	-	1000	5	-	10.3	-
Condition 2	Lime	8	-	-	-	-	-	50	-	2	-	10.3	-
Zn Rougher 1	Lime	-	10	-	-	-	-	-	-	1	3	10.3	-
Condition	Lime	4	-	-	-	-	-	25	-	2	-	10.3	-
Zn Rougher 2	Lime	-	5	-	-	-	-	-	-	1	3	10.3	-
Zn Cleaner (on combined Zn rougher concentrates utilizing process water)													
Regrind										-	-	-	14
Condition	Lime	-	-	-	-	-	-	-	400	5	-	10.5	-
Zn 1st Cleaner	Lime	-	9.5	-	-	-	-	5	-	1	3	10.5	-
Zn 1st Cleaner Scav	Lime	3	-	-	-	-	-	-	-	1	3	11.0	-
Zn 2nd Cleaner	Lime	-	-	-	-	-	-	-	-	-	3	10.8	-
Zn 3rd Cleaner	Lime	-	-	-	-	-	-	-	-	-	3	11.0	-
Zn 4th Cleaner	Lime	-	-	-	-	-	-	-	-	-	3	10.2	-

Source: RPC Final Report; Mar 13, 2019; p.48.

Tables 13-29 and 13-30 present the LCT results for samples P1 and P3, respectively. As shown in Table 13-29, P1 performed very well, yielding a lead concentrate with 40% Pb and 70.4% Pb recovery along with a zinc concentrate at 48.7% Zn and a recovery of 86.6%. Only 2.9% of the zinc was deported into the lead concentrate. Silver recovery was limited to 39.9% to the lead concentrate, while only 2.6% of the gold also reported there. Silver recovery in the zinc concentrate was low, at 9.9%, leaving only 72 g/t as the concentrate grade, which is likely below the smelter's deduction.

A large bias between measured zinc and silver heads and those calculated from the test mass balance can be seen in Table 13-29. Although silver assays on some of the small sample mass and for the high-grade lead concentrate may be understandable, the measured 5.15% Zn vs. calculated 3.94% Zn head grade is less acceptable.

Table 13-29: RPC 2019 Testwork Program – P1 Locked Cycle Test Results

Stream	Mass (%)	Chemical Assays						Distribution Ratio					
		Pb (%)	Zn (%)	Cu (%)	Ag (g/t)	Fe (%)	Au (g/t)	Pb (%)	Zn (%)	Cu (%)	Ag (%)	Fe (%)	Au (%)
Pb 4th Cleaner Conc.	2.7	40.01	4.18	0.89	737	22.33	0.465	70.4	2.9	13.5	39.9	1.7	2.6
Zn 4th Cleaner Conc.	7.0	1.20	48.72	0.58	72	12.29	0.260	5.4	86.6	22.6	9.9	2.4	3.7
Zn 1st Cleaner Scavenger Tail	13.7	0.73	0.55	0.36	50	43.10	0.666	6.4	1.9	27.6	13.6	16.3	18.8
Zn Rghr Tail	76.5	0.36	0.45	0.09	24	37.89	0.475	17.8	8.6	36.3	36.5	79.7	74.8
Meas. Head	-	1.70	5.15	0.16	68	37.01	0.365	-	-	-	-	-	-
Calc. Head	100.0	1.55	3.94	0.18	50	36.38	0.486	100.0	100.0	100.0	100.0	100.0	100.0

Source: RPC Final Report; Mar 13, 2019; p.50.

As shown in Table 13-30, the copper circuit provided a concentrate grade of 19.23% Cu and a recovery of 78.6%. The grade is below a probable threshold of 22% Cu to be widely accepted in the market. This threshold may be achievable, but at the cost of some recovery points.

As for the zinc circuit, a good concentrate grade of 47.6% was obtained, but with a low 66.3% Zn recovery. This outcome stems from the low rougher recovery achieved, with 20% of the zinc deporting to the zinc rougher tails. Optimizing the zinc roughing test conditions may allow for some improvements.

Again, there is a large discrepancy between the measured and calculated zinc heads, at 1.31% Zn and 0.77% Zn, respectively. This adds doubt to the reported zinc recovery data. The copper and silver data are not much better.

Table 13-30: RPC 2019 Testwork Program – P3 Locked Cycle Test Results

Stream	Mass (%)	Chemical Assays					Distribution Ratio				
		Pb (%)	Zn (%)	Cu (%)	Ag (g/t)	Fe (%)	Pb (%)	Zn (%)	Cu (%)	Ag (%)	Fe (%)
Cu 4th Cleaner Conc.	3.6	5.39	1.97	19.23	403	32.11	38.7	9.1	78.6	50.2	3.0
Zn 4th Cleaner Conc.	1.1	0.90	47.57	1.46	91	12.19	1.9	66.3	1.8	3.4	0.3
Zn 1st Cleaner Scavenger Tail	14.1	0.43	0.26	0.49	17	44.30	12.1	4.6	7.9	8.3	16.2
Zn Rghr Tail	81.3	0.29	0.19	0.13	14	37.94	47.2	20.0	11.7	38.1	80.4
Meas. Head	-	0.63	1.31	1.01	44	37.01	-	-	-	-	-
Calc. Head	100.0	0.50	0.77	0.88	29	38.35	100.0	100.0	100.0	100.0	100.0

Source: RPC Final Report; Mar 13, 2019; p.50.

13.2.2.6 Dewatering Testwork

A sample of the zinc concentrate produced from the LCT performed with sample P1 was submitted to some dewatering testwork procedures, as follows:

1. Settling testwork and flocculant screening – The tests compared an ionic (Cyfloc N-300) against a non-ionic (MagnaFloc 351) flocculant. The best results were obtained with 7 g/t of Cyfloc N-300 although MagnaFloc 351 was nearly as efficient.
2. Leaf filtration testwork – Vacuum filtration testwork only was carried out, although the fine nature of the concentrate and the need to bring the residual moisture content below the transportable moisture limit value are going to require the use of pressure filters, as installed in the Caribou plant.

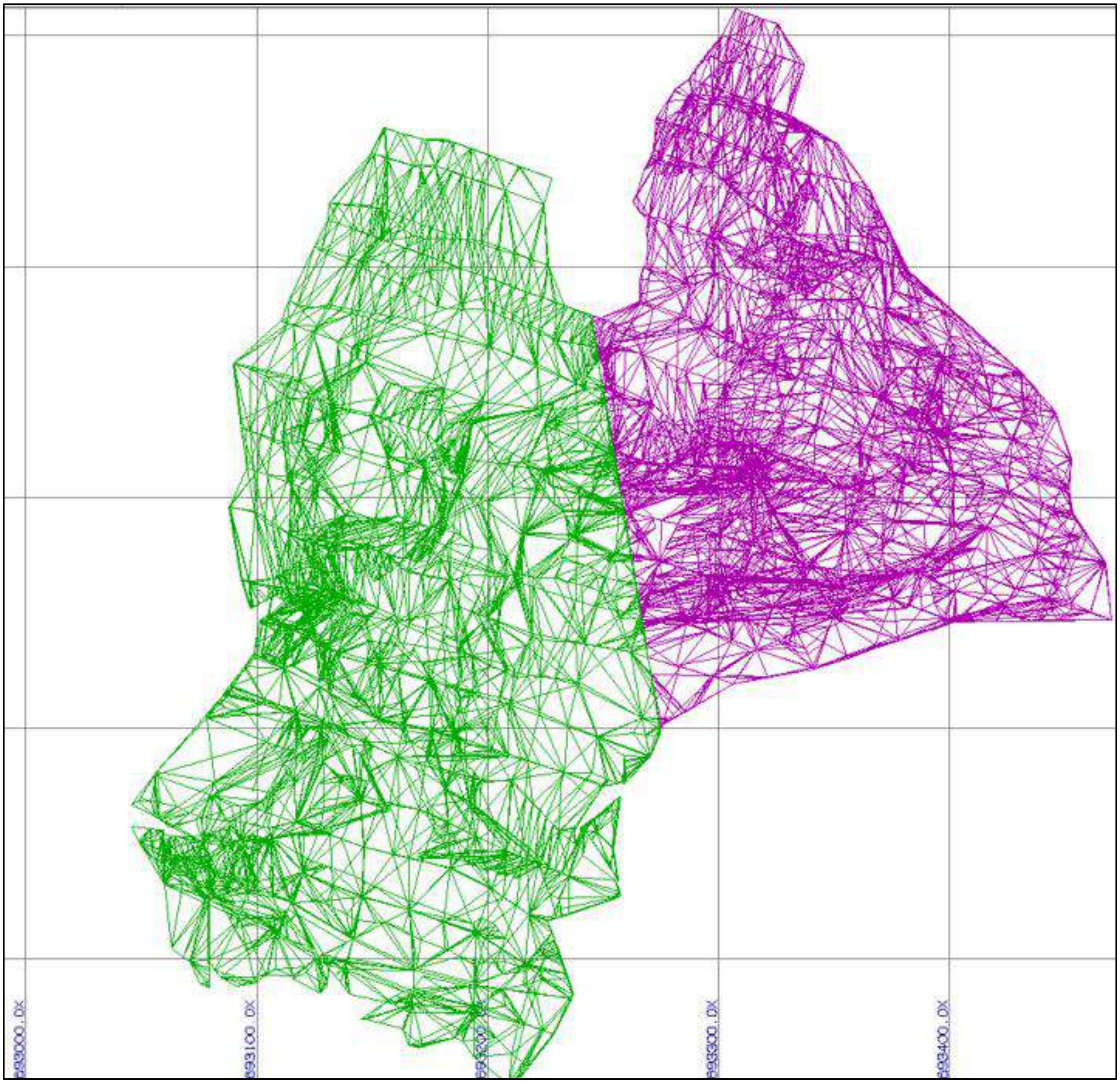
Three additional procedures were carried out on the P1 sample of LCT zinc concentrate—namely, transportable moisture limit determination; self-heating; and tailings characterization—but the results of these procedures were either not reported by RPC, or in the case of environmental tailings characterization, are not discussed.

13.3 Canadian Copper Metallurgical Testwork (2025)

Canadian Copper commissioned a testwork program with SGS laboratories, Lakefield, Ontario. The sample selection process was completed through November 2024. It implied a search of intercepts from the historical drill hole logs, with the goal of encountering samples that all involved enough copper, lead, and zinc concurrently to require the separation of relevant metals into separate concentrates. Samples with very high grades of any of these metals were excluded to maintain consistency with the expected average plant feed grades ((0.61% Cu, 0.76% Pb, and 2.2% Zn) for the first five years of operation at the time.

There is no indication from past testwork that distinct geometallurgical domains might exist within the deposit, apart from the physical definition of a West and East lobe. These are becoming broadly combined as they approach the surface layer of the deposit (Figure 13-4). The West lobe typically displays higher lead grades at depth, while the East lobe has higher copper values at depth. Higher grades of lead and copper can be found closer to the surface in the respective lobes, while zinc grades typically increase at depth (Figure 13-4).

Figure 13-4: General Morphology of the Murray Brook Deposit



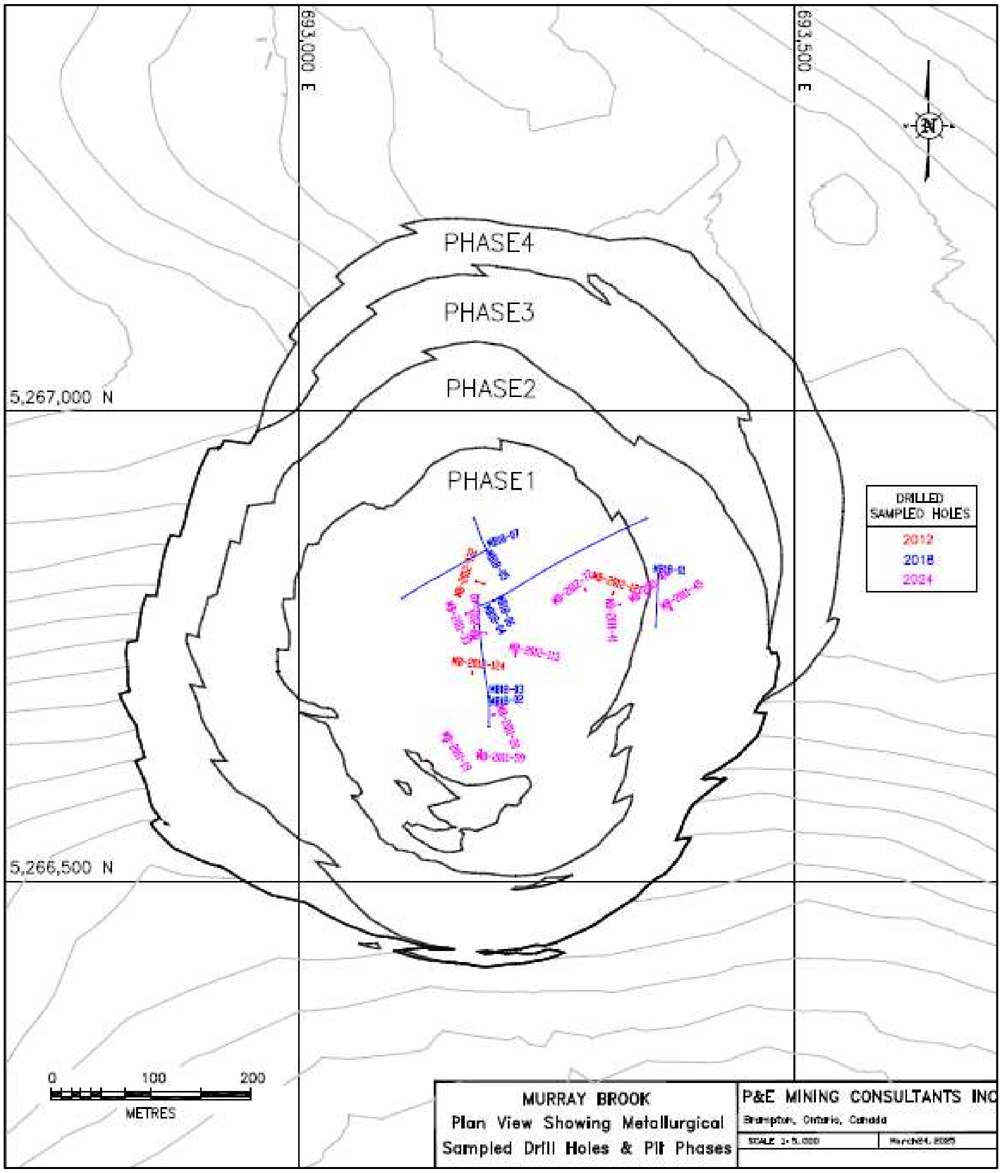
Note: East lobe shown in magenta; West lobe shown in green. Source: P&E, December 2024.

As part of the sample selection process, intercepts were sought to create samples from each lobe with a low, medium, or high ratio of copper to lead. The actual copper to lead head grades ratio was calculated as five times the copper grade vs. two times the lead grade. These proportions are reflecting expectations of the amount of concentrate that could be produced from each mineral carrying these metals, based on achieving a target 20% Cu for the copper concentrate and 50% Pb for the lead concentrate. This approach, and the availability of suitable intercepts to fulfil it from the core shack material, resulted in selecting the intercepts and composites shown in Table 13-31. Figure 13-5 shows the origin of the samples within the proposed pit envelope. The source of the samples involved in the earlier RPC testwork campaigns is also shown through colour coding.

Table 13-31: SGS Testwork Program - Sample Definition and Head Grades

Composite	Holes	From	To	m	5Cu:2Pb	% Cu	% Pb	% Zn
WLM2	2011-33	59	78	19	1.71	0.48	0.63	1.87
	2011-19	30	66	36	1.64	0.38	0.52	1.46
				55	1.67	0.41	0.56	1.60
ELM	2012-72	75	101	24	1.61	0.60	0.84	1.79
	2011-44	36	46.8	9	1.06	0.62	1.31	2.43
	2011-44	66.8	99	32.2	0.71	0.32	1.02	1.89
				65.2	1.05	0.46	0.99	1.93
WLH	2011-20	19	41	19	5.54	0.64	0.26	1.62
	2011-49	68	101	34	7.3	0.62	0.19	1.25
				53	6.56	0.63	0.22	1.38
ELL	2011-41	59	90	31	0.74	0.43	1.30	2.79
	2011-45	20.2	32	11.8	0.63	0.35	1.25	2.94
	2011-45	44.3	67	22.7	0.56	0.27	1.08	2.32
				65.5	0.67	0.36	1.21	2.65

Figure 13-5: Metallurgical Testwork Samples Sourcing vs. Proposed Mine Development Phases



Source: P&E; 2024.

The material selected for the four samples included in the SGS testwork program were received at the facilities in December 2024. Following sample preparation, initial flotation testwork and hardness determinations were carried out.

13.3.1 Grinding Testwork

Each sample was tested by SMC methodology to determine semi-autogenous (SAG) mill power demand (Axb value, drop weight index (DWi), and specific energy factor testwork). Ball mill work index (BWi) and abrasion index (Ai) testwork was also carried out. The results are presented in Table 13-32.

Table 13-32: SGS 2025 Testwork Program – Hardness Determinations

Parameter	WLM2	WLH	ELM	ELL
A	65.3	63.6	63.5	65.6
b	1.13	1	1.15	1.05
Axb	73.8	63.6	73.0	68.9
ta	0.42	0.39	0.43	0.41
Dwi, kWh/m^3	6.19	6.58	6.03	6.28
SCSE, kWh/t	7.23	7.98	7.54	7.58
Mia, kWh/t	10.8	12.4	10.9	11.5
Mic, kWh/t	4.1	4.7	4.1	4.3
SG	4.56	4.19	4.42	4.33
BWi, kWh/t	11.8	13.1	11.9	12.3
P_{80}, µm	23	33	30	28
Ai, g	0.361	0.459	0.317	0.338

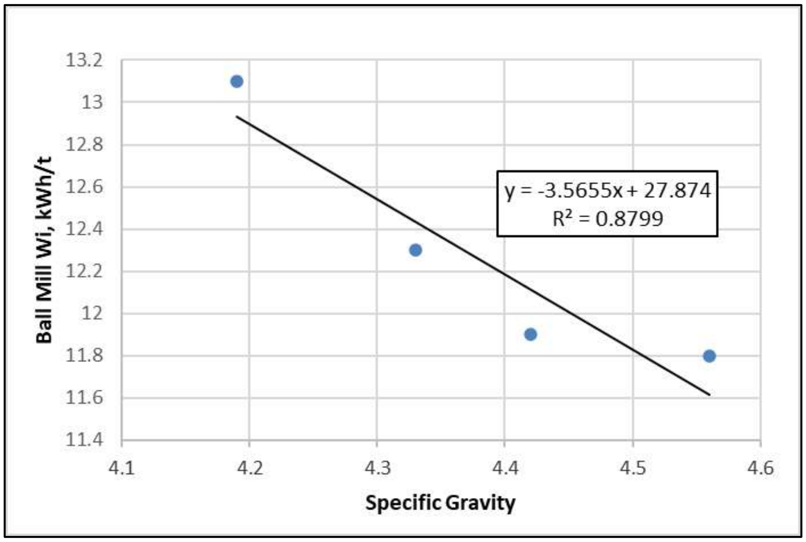
Source: JKTech Job 25007/P3, January 2025 – BWi and Ai: SGS Lakefield, January 2025.

The Axb values indicate soft materials with limited variability seen between the four samples tested. The ball mill work indices are also within a narrow band and at the softer end of the material database. The P_{80}, indicative of the final product size achieved by applying BWi determination procedures, is well aligned with the target P_{80} of 30 µm.

The BWi values indicate a strong correlation to the sample specific gravity (Figure 13-6). Such a correlation is frequent for massive sulphide deposits where the specific gravity is driven by the total sulphide mineral content, which is soft. A lower specific gravity value is indicative of harder gangue minerals replacing the soft sulphides.

The abrasion index falls into the “mildly abrasive” range.

Figure 13-6: SGS 2025 Testwork Program – BWi vs. Material SG Relationship



Source: Pierre Lacombe (2025).

13.3.2 Flotation Testwork

13.3.2.1 Roughing Tests

The flotation testwork program adopted an approach where one sample would be submitted to roughing tests under variable reagent regimes to optimize the results achieved, and then the other samples would undergo testing under these "standard" roughing circuits conditions to ensure the conditions were applicable to all samples, with minor adjustments to some of the reagent addition rates and/or flotation times to reflect the different sample feed grades.

The initial sample was then involved in the development of the bulk and zinc cleaning circuits reagent regime, which included evaluating the effect of regrinding intensity. In time, the other samples were tested under the standard cleaning circuit conditions. Sample ELM, with 0.46% Cu, 0.99% Pb, and 1.93% Zn, was selected as the initial sample to be tested, based on its higher weight availability.

The circuit configuration to be tested was aligned with the option recently obtained by Canadian Copper to purchase the Caribou plant. This dictated the selection of a bulk Cu-Pb float ahead of the zinc circuit, and subsequent Cu-Pb separation in a circuit yet to be established but in what Caribou had envisioned, and then fully engineered and partially constructed before suspending operations, as being a copper circuit meant to reclaim rejected copper from the lead cleaning circuit.

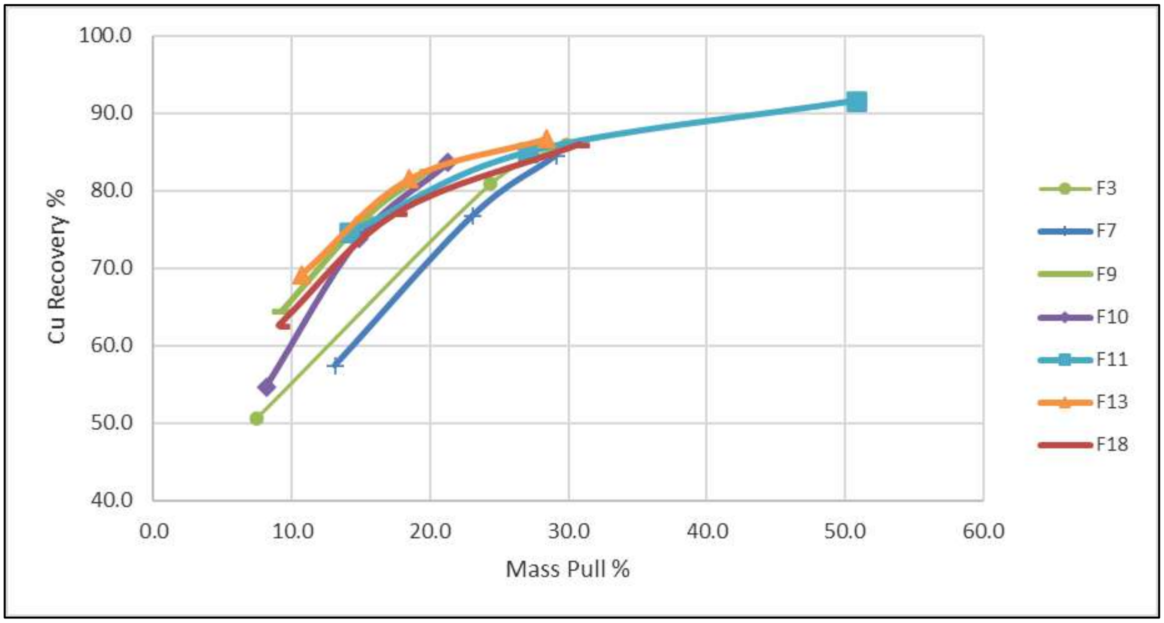
A primary grind target of 25 µm was adopted at the outset, representing a conservative setting that was to be probed once the optimal reagent regime was set. Grinding was directed to make use of mild steel media, as this is more aligned with the Caribou plant environment than the stainless-steel charge used in RPC's work. The initial reagent regimes that were adopted were broadly based on RPC's 2019 rougher scoping work. In particular, the good performance of lime as the sole pH modifier in Test P1T8 warranted a repeat early into the trials at SGS. The effect of each depressant involved earlier, ahead of the initial flotation stage (zinc sulphate, MBS, cyanide), as well as its addition point and rate, were main areas of investigation. As another change, the PEX/A3418A combination as a collector was replaced by PEX/A241. The unconventional use of A3418A as a zinc collector was also called into question, with A5100 and sodium isopropyl xanthate (SIPX) being recognized as more affordable replacements.

Table 13-33 tabulates the 18 iterations run with the ELM sample while searching for the best roughing conditions. Figure 13-7 to 13-10 show the flotation kinetics for the best-timed tests for copper, lead, and zinc to the bulk rougher concentrate and the stage recovery of zinc through its roughing stage.

Table 13-33: SGS 2025 Testwork Program – Roughing Test Conditions with ELM Sample – 25 µm grind

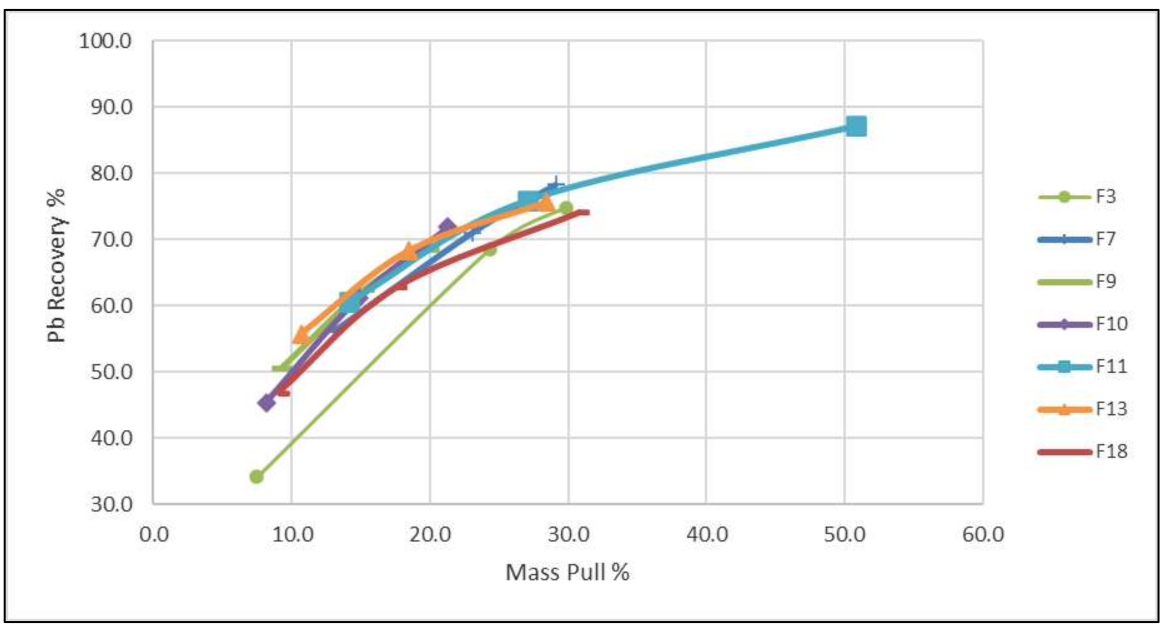
Test ID	Objective	Stage	Time	Aeration	pH	Na_2CO_3	$Ca(OH)_2$	$ZnSO_4$	NaCN	SMBS	$CuSO_4$	PEX	SIPX	241	3418	3421	5100
			min	min		g/t	g/t	g/t	g/t	g/t	g/t	g/t	g/t	g/t	g/t	g/t	g/t
F1	Baseline using B7/B11	Bulk Ro	6.0	10	9.2	1700	-	800	-	-	-	12	-	75	-	-	-
	Conditions from 2013	Zinc Ro	6.0	-	10.3	-	tbd	-	-	-	350	9	-	-	60	-	-
F2	Lower Bulk pH	Bulk Ro	6.0	10	7.5	-	-	800	-	-	-	12	-	60	-	-	-
	SIPX in Zn Cct	Zinc Ro	6.0	-	10.5	-	tbd	-	-	-	350	9	45	-	-	-	-
F3	Bulk as F1 but NaCN	Bulk Ro	6.0	10	9.3	1115	-	800	150	-	-	12	-	60	-	-	-
	Zn as F2	Zinc Ro	6.0	-	10.3	-	tbd	-	-	-	400	9	45	-	-	-	-
F4	As B1 from 2013, w/NaCN	Bulk Ro	6.0	10	9.3	700	-	600	40	-	-	8	-	-	55	-	-
	3418 in both circuits	Zinc Ro	6.0	-	10.2	-	tbd	-	-	-	350	9	-	-	60	-	-
F5	As F1 but lime in place of Na_2CO_3	Bulk Ro	6.0	10	9.6	-	2000	800	-	-	-	12	-	75	-	-	-
	5100 in Zn Cct	Zinc Ro	6.0	-	10.5	-	-	-	-	-	350	-	-	-	-	-	45
F6	As F5, but 3418A in Bulk	Bulk Ro	6.0	10	9.3	-	1500	800	-	-	-	8	-	-	55	-	-
		Zinc Ro	6.0	-	10.3	-	-	-	-	-	350	-	-	-	-	-	60
F7	As F3, but more $ZnSO_4$ and longer bulk float	Bulk Ro	8.0	10	9.3	2750	-	1500	150	-	-	12	-	60	-	-	-
	more $CuSO_4$ and no PEX in Zn Cct	Zinc Ro	6.0	-	10.2	-	900	-	-	-	500	-	45	-	-	-	-
F8	As F7, but more NaCN	Bulk Ro	8.0	10	9.3	1350	-	1500	300	-	-	12	-	60	-	-	-
	5100 in Zn Cct	Zinc Ro	6.0	-	10.5	-	900	-	-	-	500	-	-	-	-	-	60
F9	As F7, but SMBS and pH adjusted	Bulk Ro	8.0	10	10.0	6700	-	1500	150	1000	-	12	-	60	-	-	-
		Zinc Ro	6.0	-	10.1	-	950	-	-	-	500	-	45	-	-	-	-
F10	As F9, but replace soda ash with lime	Bulk Ro	8.0	10	10.4	-	2200	1500	150	1000	-	12	-	60	-	-	-
		Zinc Ro	6.0	-	10.3	-	850	-	-	-	500	-	45	-	-	-	-
F11	As F7, but replace 241 with 3418A	Bulk Ro	8.0	10	9.3	2750	-	1500	150	-	-	12	-	-	95	-	-
		Zinc Ro	6.0	-	10.2	-	1050	-	-	-	500	-	45	-	-	-	-
F12	As F11, but higher NaCN dosage	Bulk Ro	8.0	10	9.2	3000	-	1500	300	-	-	12	-	-	95	-	-
		Zinc Ro	6.0	-	10.2	-	1050	-	-	-	500	-	45	-	-	-	-
F13	As F10, with longer aeration time, split bulk conditioner	Bulk Ro	8.0	30	8.6	-	1060	1500	150	1000	-	14	-	90	-	-	-
	increased 241, lower bulk pH, increased CuSO4	Zinc Ro	6.0	-	10.2	-	1150	-	-	-	600	-	45	-	-	-	-
F14	As F13, with 3418 in place of 241	Bulk Ro	8.0	30	8.7	-	345	1500	150	1000	-	14	-	-	95	-	-
	5100 in Zn Cct	Zinc Ro	6.0	-	10.2	-	900	-	-	-	600	-	-	-	-	-	60
F15	As F14, with higher SMBS dosage	Bulk Ro	8.0	30	8.6	-	tbd	1500	150	2000	-	14	-	-	95	-	-
		Zinc Ro	6.0	-	10.2	-	850	-	-	-	600	-	-	-	-	-	60
F16	As F14, with 3421 in place of 3418 and no PEX	Bulk Ro	8.0	30	8.7	-	600	1500	150	1000	-	-	-	-	-	65	-
		Zinc Ro	6.0	-	10.2	-	900	-	-	-	600	-	-	-	-	-	60
F17	As F11, but increased aeration time,	Bulk Ro	4.0	30	9.2	1150	-	1500	150	-	-	10	-	-	70	-	-
	reduced bulk cct reagents, reduced bulk cct time	Zinc Ro	6.0	-	10.7	-	1050	-	-	-	600	-	45	-	-	-	-
F18	As F17, but 3421 in place of 3418	Bulk Ro	4.0	30	9.2	1600	-	1500	150	-	-	10	-	-	-	65	-
		Zinc Ro	6.0	-	10.6	-	1050	-	-	-	600	-	45	-	-	-	-

Figure 13-7: SGS 2025 Testwork Program – Selected Copper Roughing Test Results with ELM Sample



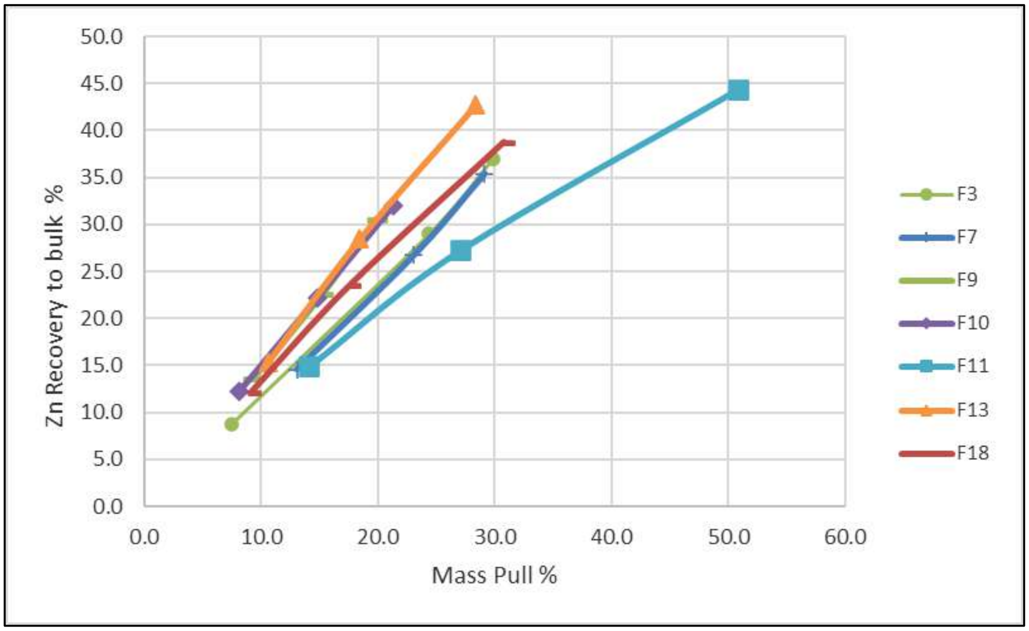
Source: Pierre Lacombe (2025).

Figure 13-8: SGS 2025 Testwork Program – Selected Lead Roughing Test Results with ELM Sample



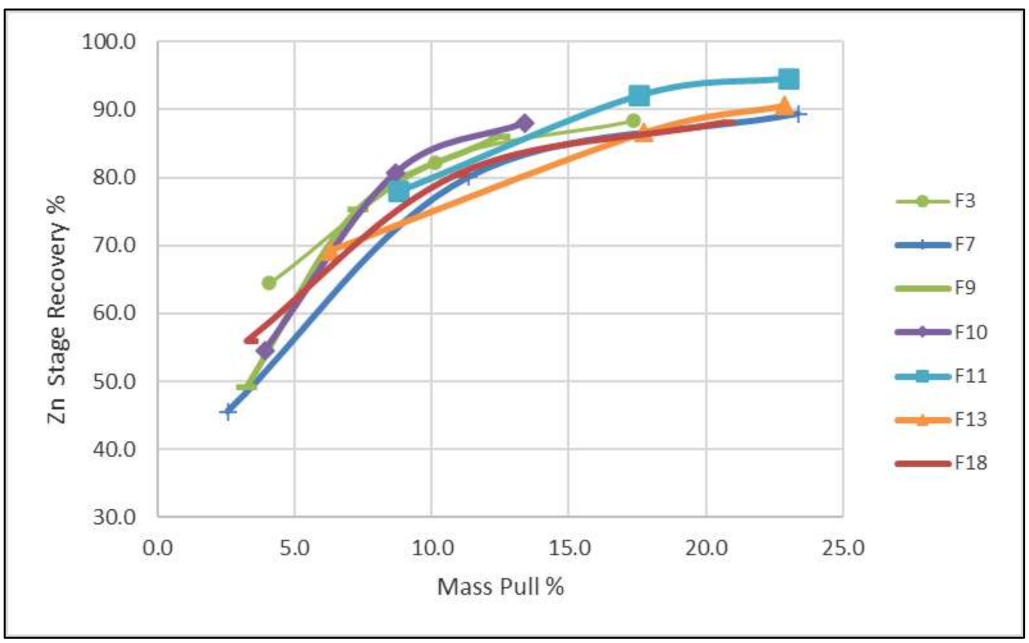
Source: Pierre Lacombe (2025).

Figure 13-9: SGS 2025 Testwork Program – Selected Zinc to Bulk Rougher Test Results with ELM Sample



Source: Pierre Lacombe (2025).

Figure 13-10: SGS 2025 Testwork Program – Selected Zinc Stage Recovery to Zinc Rougher Test Results with ELM Sample



Source: Pierre Lacombe (2025).

A number of reagent combinations were capable of delivering adequate rougher recoveries (Figure 13-7 to 13-10). The main difference was the mass pull required to reach those recovery levels and at the lowest zinc deportment into the bulk concentrate, with the lowest mass pull being indicative of an efficient pyrite depression. Some depressant regimes were found to slow down flotation kinetics, as seen from ascending recovery curves at the end of the flotation period, which indicates longer required flotation times. The shape of the kinetic curves in Figure 13-9, with zinc recovery aligned with incremental mass pull, indicates that zinc flotation into the bulk concentrate did not occur because of activation, but by a combination of entrainment or as middlings with the other sulphides being floated.

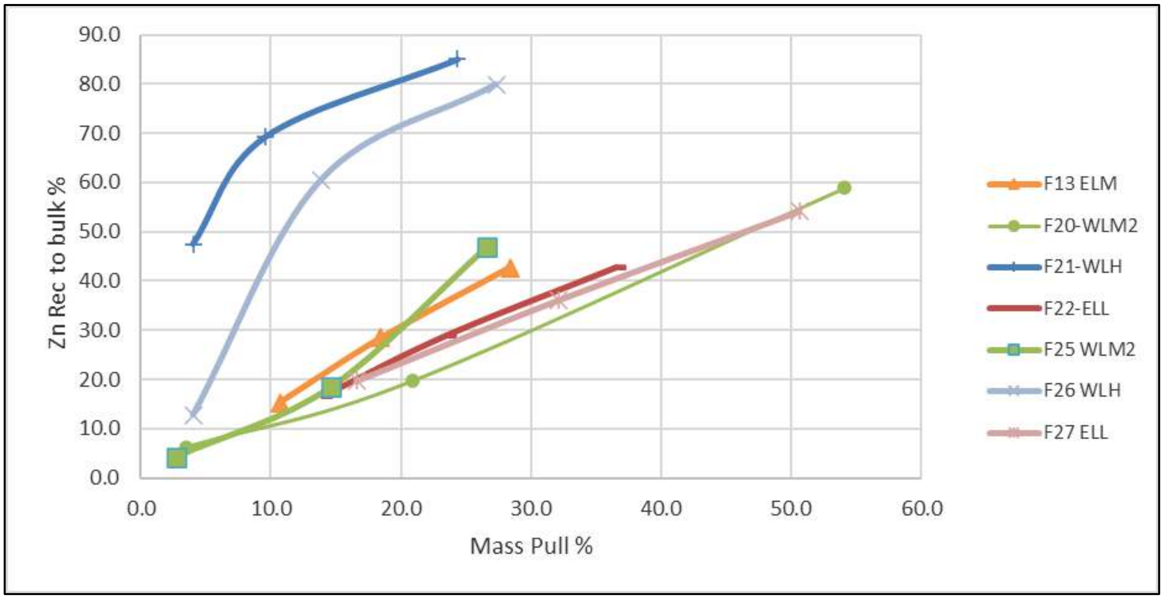
Test F11 represents, if stopping flotation sooner than the last kinetic data point, the best performance achieved with soda ash being used in the bulk circuit and with A3418A as the bulk collector. For those tests with lime throughout, F13 was flagged as the standard going forward with the ELM cleaning circuits testwork. In parallel, applying F13 conditions (Table 13-34) for roughing tests with the other samples was initiated.

Table 13-34: SGS 2025 Testwork Program – Standard Roughing Tests Conditions from Test F13

Stage	Reagents Added (g/t)									Time (min)		Actual pH	Eh mV
	$Ca(OH)_2$	SMBS	PEX	SIPX	AF-241	$ZnSO_4$ $7H_2O$	NaCN	$CuSO_4$. $5H_2O$	MIBC	Cond.	Froth		
Grind	none	1000	-	-	-	1500	150	-	-	-	-	7.8	-10
Aeration	none	-	-	-	-	-	-	-	-	30	-	7.6	120
Cu-Pb Circuit		-	-	-	-	-	-	-	-	-	-	-	-
Condition 1	700	-	-	-	-	-	-	-	-	3	-	8.6	120
Condition 2	140	-	6	-	30	-	-	-	-	2	-	8.6	120
Rougher 1	*	-	-	-	-	-	-	-	5	-	2	-	-
Rougher 2	220	-	4	-	30	-	-	-	*	1	2	8.6	100
Rougher 3	*	-	4	-	30	-	-	-	*	1	4	8.5	100
Bulk Rougher Tails to Zn Rougher	-	-	-	-	-	-	-	-	-	-	-	-	-
Zn Circuit	-	-	-	-	-	-	-	-	-	-	-	-	-
Condition 1	600	-	-	-	-	-	-	-	-	5	-	10.4	50
Condition 2	200	-	-	-	-	-	-	600	-	2	-	10.1	50
Zn Rougher 1	*	-	-	15	-	-	-	-	-	1	1	10.2	40
Zn Rougher 2	150	-	-	15	-	-	-	-	-	1	2	10.3	80
Zn Rougher 3	200	-	-	15	-	-	-	-	-	1	3	10.2	80

The relationship of zinc deportment to bulk rougher concentrate, as shown when applying F13 roughing conditions to all samples, is illustrated in Figure 13-11. The figure shows that the sample WLH is composed of oxidized material, with zinc being activated within the bulk circuit. This similar to the response seen by RPC in 2012 with some sections of the holes tested, and again in 2019, with sample PB. A few unsuccessful attempts with different reagent regimes were made to prevent this from occurring. As a result, no further work was conducted with WLH. Of note, WLH included an intercept logged from 19 to 41 m down the hole (Table 13-29), while it appears that the surface oxidation zone can extend 50 m deep in some part of the deposit.

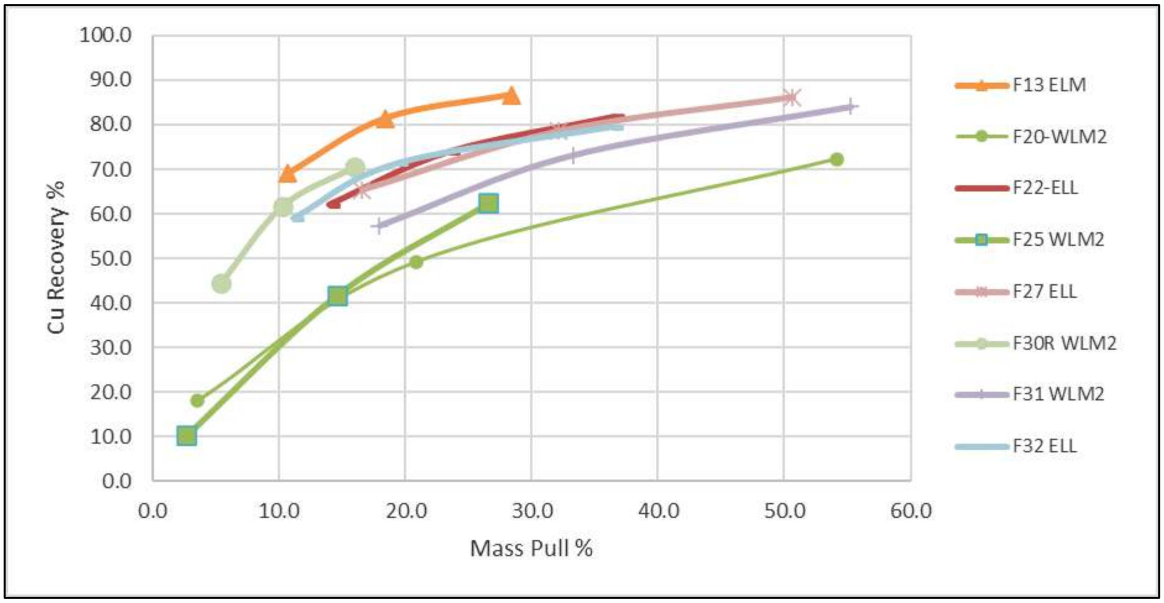
Figure 13-11: SGS 2025 Testwork Program – Comparative Zinc Deportment to Bulk Roughers



Source: Pierre Lacombe (2025).

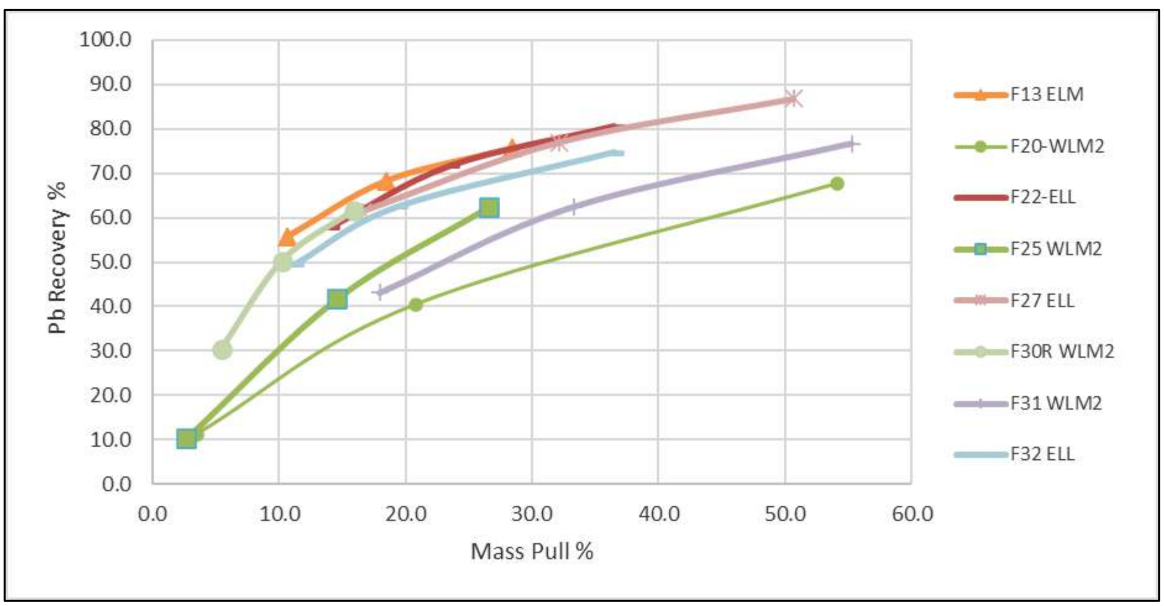
Using the remaining ELL and WLM2 samples to optimize roughing test results continued. Figures 13-12 to 13-15 show the response of these samples compared to the baseline test F13 with sample ELM. ELL behaves similarly to ELM, with an expected drop in achievable copper recovery in line with its lower copper head grade. Other metal recoveries and mass pull are aligned with F13 data.

Figure 13-12: SGS 2025 Testwork Program – Comparative Copper Recovery to Bulk Roughers



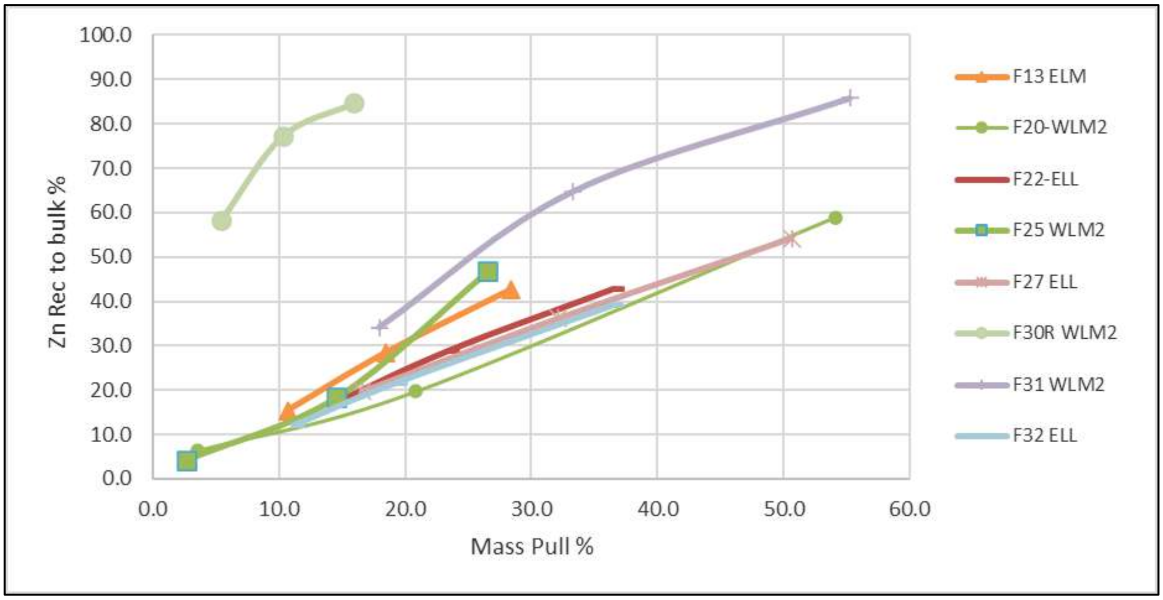
Source: Pierre Lacombe (2025).

Figure 13-13: SGS 2025 Testwork Program – Comparative Lead Recovery to Bulk Roughers



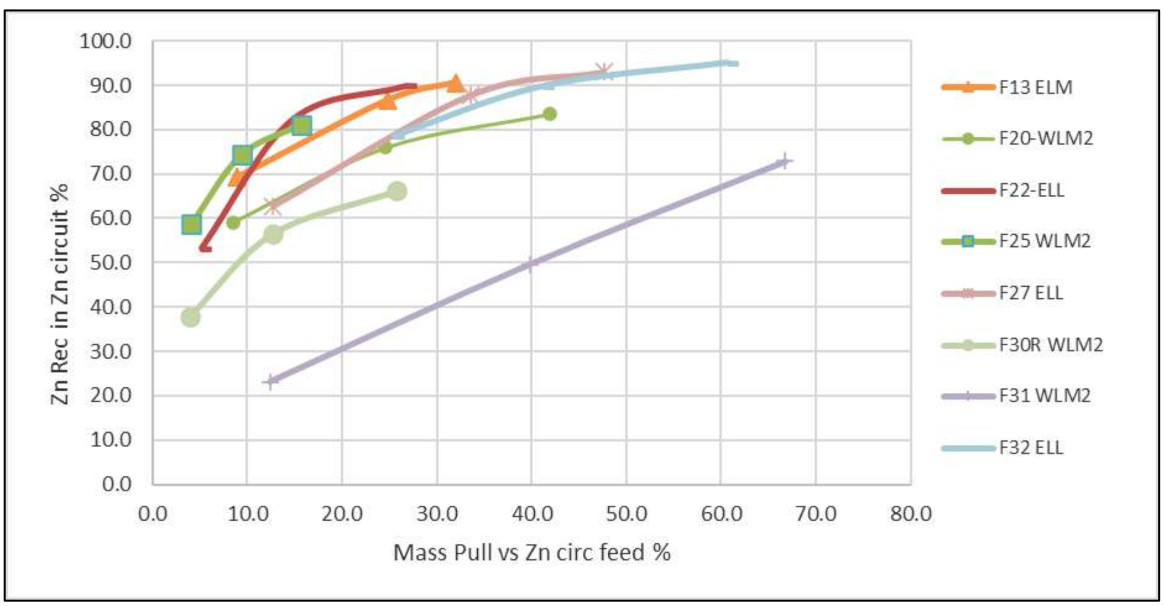
Source: Pierre Lacombe (2025).

Figure 13-14: SGS 2025 Testwork Program – Comparative Zinc Deportment to Bulk Roughers



Source: Pierre Lacombe (2025).

Figure 13-15: SGS 2025 Testwork Program – Comparative Zinc Recovery to Zn Roughers



Source: Pierre Lacombe (2025).

With ELL, tests F22 and F32 provide similar responses in the bulk circuit, but faster kinetics for the zinc rougher. ELL was therefore deemed ready to proceed to the cleaning trials. WLM2's results that show lead and zinc falling below the F13 values can be explained by lower head grades.

The poor copper behaviour cannot be ascribed to a lower head grade, because WLM2 carries 0.46% Cu, placing it between ELM at 0.57% Cu and ELL at 0.38% Cu. While trying to improve upon the results from F25, test F30R was completed. It showed a marked improvement in both copper and lead recoveries to the bulk concentrate but also activated zinc, which floated freely into the bulk concentrate. The only modification between these two tests that may be responsible for such a response is that the lime addition after the primary grind was moved from the conditioning stage just ahead of flotation in F25 to the aeration stage just after exiting the grinding mill.

13.3.2.2 Cleaning Tests

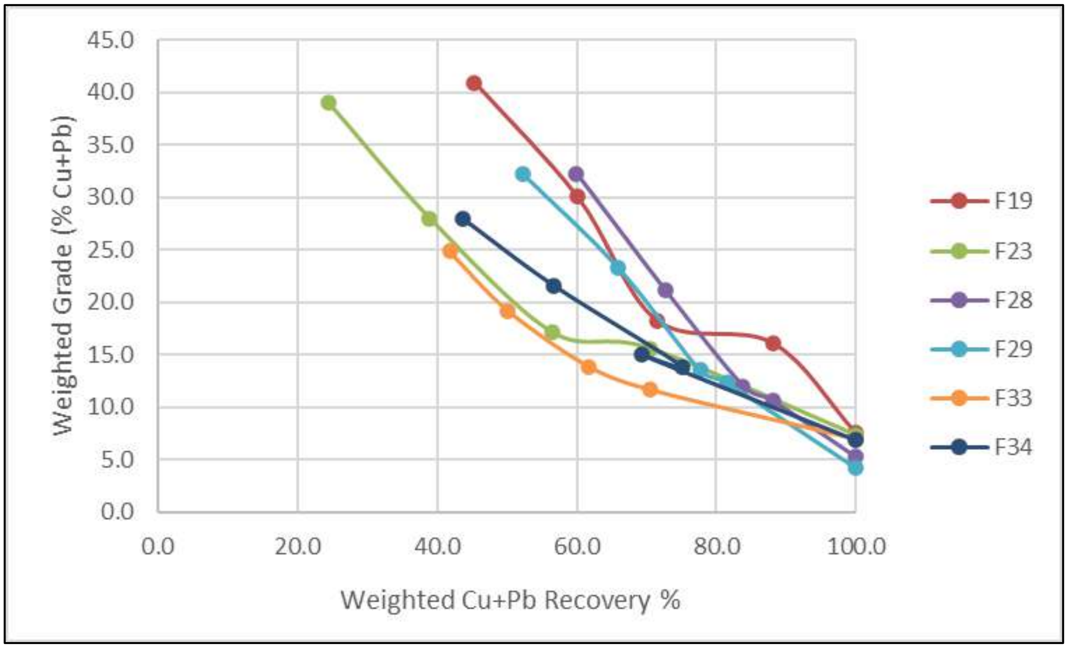
The initial cleaning tests with sample ELM did not meaningfully upgrade the bulk or zinc concentrates (Figures 13-16 and 13-17). As the bulk circuit aims to simultaneously upgrade both the copper and lead, the presentation of the cleaning circuit performance is based on an approach that is weighing the eventual contribution of for both metals to the recovery and concentrate purity. On the recovery side, a mathematical average between the stage recovery of copper and lead is calculated, taking the stage recovery versus the rougher concentrate. Hence, if R(Cu) and R(Pb) represent the copper and lead recoveries achieved in a given cleaning stage, versus what had been recovered in the rougher concentrate used as feed to the cleaning circuit, the combined recovery for this cleaning stage is (R(Cu) + R(Pb))/2.

As for the concentrate grade, a bulk cleaning efficiency factor was devised. It is derived by averaging the actual copper and lead grades achieved in every cleaning stage, versus the targeted concentrate grades of 20% Cu and 45% Pb in their eventual respective concentrate. Hence, if G(Cu) and G(Pb) represent the copper and lead grades achieved in a given cleaning stage, respectively, the upgrading efficiency for this cleaning stage is (G(Cu)/0.2 + G(Pb)/0.45)/2.

The best upgrading results (test F19) yielded a grade of 9.0% Cu and 16.6% Pb. A positive note was that the depression of zinc in the bulk cleaning circuit was quite effective, leaving 2% to 5% of the overall zinc recovery units in the final bulk concentrate.

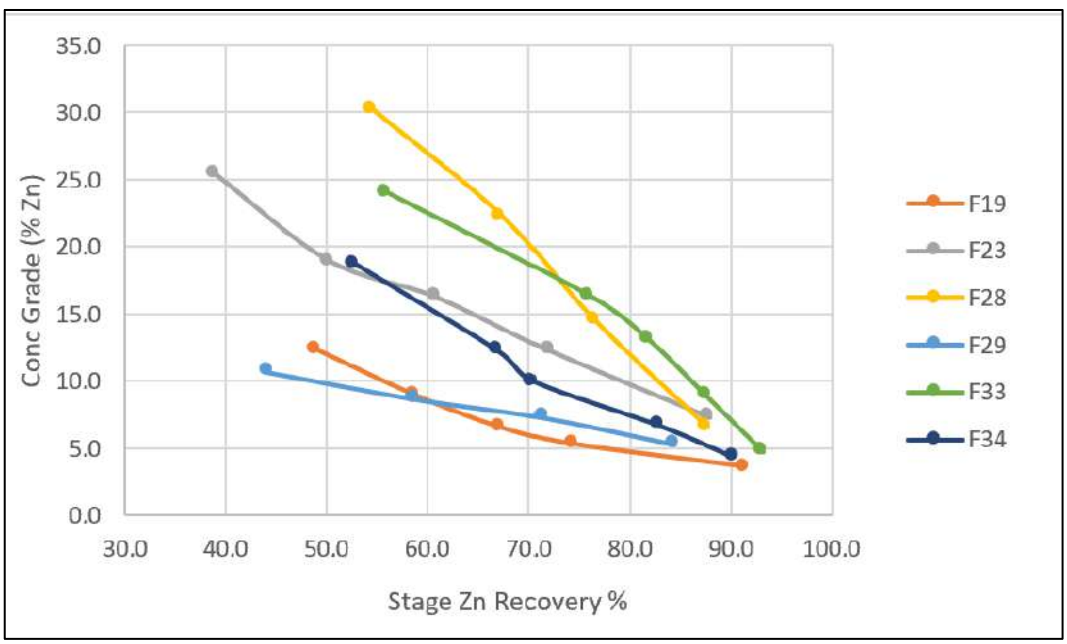
The copper cleaning circuit was floated at the fine 10 to 12 µm regrind target, as per earlier RPC work. Attempts with finer regrinding sizes for the zinc circuit (i.e., 20 µm to 12 to 15 µm), did not help, because pyrite was still reporting to this concentrate. A liberation analysis performed on some of the flotation products from these tests indicated that the metals of economic interest were mostly well liberated, as was the bulk of the pyrite being floated.

Figure 13-16: SGS 2025 Testwork Program – Bulk Cleaning Circuit Upgrading Curves – ELM Sample



Source: Pierre Lacombe (2025).

Figure 13-17: SGS 2025 Testwork Program – Zinc Cleaning Circuit Upgrading Curves – ELM Sample



Source: Pierre Lacombe (2025).

The testwork program was then suspended without achieving sufficient upgrading to any rougher concentrates produced or carrying through the Cu-Pb separation step.

13.4 Metallurgical Variability

All core intercepts used in the testwork program were extracted from either Phase 1 or Phase 2 of the proposed pit development (Figure 13-5), which provides a high level of representativity to the samples tested. The spatial coverage may have been limited under RPC's 2012 testwork program, as all the samples were extracted from three drill holes at different depths. The samples prepared for the 2019 testwork campaign were composites that included a more varied array of samples, with some components also submitted to a limited variability testwork program.

The 2019 samples were mostly dealing with either Cu-Zn or Pb-Zn samples. Even if at times copper flotation was attempted from very low copper feed grades in the earlier RPC testwork programs, only the 2024 SGS campaign recognized that the plant will regularly have to deal with feed material exhibiting simultaneously copper, lead, and zinc values warranting flotation of all these metals in separate concentrates. The outcome of this work did establish whether a different response could be expected in terms of the lobe definition used to extract the samples.

13.5 Deleterious Elements

Minor element assays for some concentrates produced during the SGS 2012 campaign are available (Table 13-35). Some elements have yielded values above the upper threshold of the ICP apparatus, which may indicate potential penalties. These include antimony in copper and lead concentrates, and mercury in zinc concentrate. Missing from this table are the potential cross-contaminants from economic metals (lead and zinc in copper concentrate; copper and zinc in lead concentrate) and the iron content in the zinc concentrate, although these are available from the test mass balance. These elements should be tracked more closely in future concentrate streams and assayed with an appropriate method.

The data from Table 13-35 and the mass balance would indicate penalties for the following:

- arsenic, mercury, antimony, and lead + zinc in the copper concentrate
- "possible" antimony in the lead concentrate
- "possible" mercury and iron in the zinc concentrate.

Table 13-36 shows the suite of elements found in the samples provided to SGS for the 2025 testwork. The table shows elevated amounts of arsenic, cadmium, antimony, mercury, and bismuth that may be concentrated into one or more of the concentrates extracted from these samples.

Table 13-35: RPC 2012 Testwork Program – Minor Elements Determination for Three-Hole Composite LCT Concentrates

Sample ID	Final Cu Con	Final Pb Con	Final Zn Con	Sample ID (ppm)	Final Cu Con	Final Pb Con	Final Zn Con
Total S (%)	36.0	24.7	33.6	Sr	3.0	5.8	1.0
Na (%)	< 0.01	< 0.01	< 0.01	Zr	19	6	2
Mg (%)	0.03	0.02	0.01	Nb	0.1	< 0.1	0.3
Al (%)	0.05	0.05	0.01	Mo	42.1	17.4	10.4
K (%)	< 0.01	0.01	< 0.01	In	77.2	19.7	> 100
Ca (%)	0.135	0.21	0.08	Sn	153	46	28
Li (ppm)	< 0.5	< 0.5	< 0.5	Sb	> 500	> 500	304
Cd (ppm)	153	112	1,037	Te	< 0.1	0.1	0.2
V (ppm)	4	2	< 1	Ba	9	1.5	3
Cr (ppm)	73.2	36.4	15.3	La	< 0.1	< 0.1	< 0.1
Mn (ppm)	191	164	211	Ce	1.8	0.7	0.3
Hf (ppm)	0.15	0.15	< 0.1	Pr	0.2	< 0.1	< 0.1
Hg (ppm)	49.5	36.2	> 100	Nd	0.75	0.3	0.1
Ni (ppm)	52.2	25.8	15.7	Sm	0.2	< 0.1	< 0.1
Er (ppm)	0.1	< 0.1	< 0.1	Gd	0.2	< 0.1	< 0.1
Be (ppm)	< 0.1	0.1	0.2	Tb	< 0.1	< 0.1	< 0.1
Cs (ppm)	0.1	0.07	< 0.05	Dy	0.2	< 0.1	< 0.1
Co (ppm)	68.5	19.3	4.6	Ge	0.95	1.3	0.3
Eu (ppm)	0.11	0.07	< 0.05	Yb	0.15	< 0.1	< 0.1
Bi (ppm)	262	751	27.4	Ta	< 0.1	< 0.1	0.2
Se (ppm)	83.2	301	28.6	W	0.2	0.15	0.2
Ga (ppm)	2.3	1.55	9.1	Re	0.03	0.018	0.011
As (ppm)	3,335	2,080	1,460	Tl	89.1	356	30.6
Rb (ppm)	0.8	0.6	< 0.2	Th	1.4	0.6	0.1
Y (ppm)	1.1	0.4	0.1	U	2.1	1.4	1.3

Source: RPC Final Report, Nov 2012; p.47.

Table 13-36: SGS 2025 Testwork Program – ICP Scans of Samples Tested

Sample ID	WLM2	ELM	WLH	ELL
Cu (%)	0.46	0.57	0.70	0.38
Zn (%)	1.56	2.00	1.43	2.58
Pb (%)	0.59	1.02	0.21	1.26
Fe (%)	42.2	38.2	38.4	37.2
As (%)	0.89	0.63	0.21	0.47
Ag (g/t)	25	43	15	54
Au (g/t)	1.20	0.91	0.20	0.77
Al (g/t)	2,420	3,360	225	3,490
Ba (g/t)	11	16	4	9
Be (g/t)	0.07	0.11	0.05	0.11
Bi (g/t)	71	97	35	117
Ca (g/t)	982	20,300	2,610	16,800
Cd (g/t)	37	63	< 30	79
Co (g/t)	230	121	52	80
Cr (g/t)	40	25	41	20
Hg (g/t)	16.7	18.4	13.2	18.0
K (g/t)	350	585	< 100	< 100
Li (g/t)	< 10	< 10	< 10	< 10
Mg (g/t)	513	8140	1230	7240
Mn (g/t)	284	1980	1240	1970
Mo (g/t)	< 10	< 10	< 10	< 10
Na (g/t)	< 100	< 100	< 100	< 100
Ni (g/t)	< 20	< 20	< 20	< 20
P (g/t)	< 200	< 200	< 200	< 200
S (%)	48.3	38.8	41.8	37.4
S (%)	48.1	38.6	41.5	37.4
Sb (g/t)	385	1,000	238	849
Se (g/t)	37	39	< 30	37
Sn (g/t)	< 20	< 20	< 20	< 20
Sr (g/t)	3	17	2	17
Ti (g/t)	98	72	< 5	173
Tl (g/t)	< 30	< 30	< 30	< 30
V (g/t)	10	10	< 4	15
Y (g/t)	< 0.6	1.4	< 0.6	2.2

These high values are not anomalous to these samples, as the ones tested by RPC in 2012 also reported similar levels of the same metals (Table 13-37).

Table 13-37: RPC 2012 Testwork Program – Minor Elements Determination for Selected Concentrate Samples

Sample ID	Hole 121 (Btm)	Hole 124 (Btm)	Hole 132 (Whole)	Ground Feed	Sample ID (ppm)	Hole 121 (Btm)	Hole 124 (Btm)	Hole 132 (Whole)	Blended Feed
Total S (%)	36.0	44.3	40.4	39.6	Sr	43.2	5.9	8.1	14.9
Na (%)	0.07	0.01	0.02	0.02	Zr	42	21	20	26
Mg (%)	0.61	0.17	0.27	0.27	Nb	5	1.2	0.9	2.5
Al (%)	0.88	0.23	0.33	0.36	Mo	9.4	14.3	13.2	67.1
K (%)	0.14	0.07	0.12	0.09	In	4.1	11.7	20.1	11.8
Ca (%)	2.36	0.59	1.18	0.91	Sn	31	30	58	18
Li (ppm)	4.2	2.7	1.7	1.7	Sb	> 500	> 500	> 500	> 500
Cd (ppm)	53.8	42.1	109	53.2	Te	< 0.1	< 0.1	< 0.1	< 0.1
V (ppm)	51	17	16	17	Ba	46	21	19	25
Cr (ppm)	12.6	22.5	18.6	333	La	4.5	1.1	2.2	2.5
Mn (ppm)	1,530	772	1,160	1,040	Ce	10.1	3.2	5.2	5.9
Hf (ppm)	0.7	<0.1	<0.1	0.5	Pr	1.2	0.4	0.6	0.7
Hg (ppm)	11.1	18.0	22.1	17.0	Nd	5.0	1.4	2.1	2.6
Ni (ppm)	9.3	10.5	11	237	Sm	1.1	0.3	0.4	0.6
Er (ppm)	0.6	0.2	0.2	0.3	Gd	1.3	0.3	0.4	0.5
Be (ppm)	0.5	0.2	0.2	0.2	Tb	0.2	< 0.1	< 0.1	< 0.1
Cs (ppm)	0.44	0.27	0.42	0.33	Dy	1.2	0.4	0.4	0.5
Co (ppm)	130	143	57.7	89.3	Ge	0.7	2.0	3.1	1.1
Eu (ppm)	0.65	0.25	0.38	0.36	Yb	0.5	0.2	0.2	0.3
Bi (ppm)	106	36.6	25.5	45.5	Ta	0.7	0.4	0.5	< 0.1
Se (ppm)	48.2	25.2	22.5	27.9	W	1.6	1.6	1.7	2.8
Ga (ppm)	7.4	4.0	4.8	4.9	Re	0.004	0.011	0.004	0.016
As (ppm)	2,850	6,500	5,920	4,420	Tl	30.2	51.2	63.3	45
Rb (ppm)	11.6	5.6	9.8	7.8	Th	1.0	0.5	1.0	0.6
Y (ppm)	6.2	2.3	2.2	3.0	U	2.4	1.5	2.8	2.0

Source: RPC Final Report, Nov 2012; p.46.

13.6 Recovery Estimates

The approach used to evaluate recoveries incorporated the results of the RPC testwork programs as much as possible, with additional confirmation from SGS testwork for the roughing recovery stage.

The overall recovery was divided into the individual circuits within the plant flowsheet, namely the roughing and cleaning sections for the bulk and zinc circuits. In addition, the Cu-Pb separation circuit had to be estimated, despite lacking any data from past testwork regarding its eventual behaviour.

The rougher recovery projections are based on achieving a fixed rougher tailings grade, as informed by the database of testwork available. This in turn indicates that the recovery will vary with metal head grades. Based on available tests, but excluding some outliers, the following is gleaned from the rougher data:

- Minimum bulk rougher tails copper grade: 0.1% Cu
- Minimum bulk rougher tails lead grade: 0.39% Pb
- Minimum zinc rougher tails zinc grade: 0.26% Zn.

As the feed grades increase, so will the indicated recovery. Since no testwork with very high feed grades is available to place an adequate cap on the maximum rougher recovery feasible, an arbitrary limit of 92% for both copper and lead was adopted. This cap would be met for feed grades of ≥1.25% Cu and ≥4.88% Pb. For zinc, this cap is raised to 95%, reached with ≥5.2% Zn.

For silver, a fixed recovery to the bulk rougher concentrate of 80% is proposed, along with a fixed recovery within the bulk cleaning circuit of 70%. From the limited test data with copper and lead floated sequentially, an expectation arises that two-thirds of the silver brought to the Cu-Pb separation circuit will report to the copper concentrate and the other on-third to the lead concentrate.

From the same sources, it may be possible to retain 87% of the copper within the bulk cleaning circuit, whereas only 80% would be kept according to it more intimate mineralogy with pyrite. Assumptions as to the eventual operation of the untested Cu-Pb separation circuit have been made on the basis that lead will be the floated product and copper will be the tailings product. As such, it is assumed that the copper remaining with the lead concentrate will be minimal, while 6% of the lead will be deported to the copper concentrate.

For the zinc cleaning circuit, sufficient test data is available to target a loss through deportment into the cleaned bulk concentrate of 5% and a zinc cleaning circuit recovery of 96%.

The mathematical expression of these assumptions is therefore:

- Copper recovery to copper concentrate (%) = min(0.92, max(Cu head – 0.1)/Cu head)) x 87
- Lead recovery to lead concentrate (%) = min(0.92, max(Pb head – 0.39)/Pb head)) x 0.94 x 80
- Zinc recovery to zinc concentrate (%) = min(0.95, max*Zn head – 0.26)/Zn head)) x 0.95 x 96

- Silver recovery to copper concentrate (%) = 80 x 0.7 x 0.66
- Silver recovery to lead concentrate (%) = 80 x 0.7 x 0.33

As for the projected concentrate grades underlying these recovery equations, the data available are deemed to justify the following expectations:

- Copper concentrate (% Cu) : 22% Cu if copper head ≤ 0.8% Cu
  : 25% Cu if copper head > 0.8% Cu
- Lead concentrate (% Pb) : 45% Pb
- Zinc concentrate (% Zn) : 48% Zn

13.7 Comments on Mineral Processing and Metallurgical Testing

Historical testwork programs have seen some of the proposed samples behaving poorly to conventional approaches to processing VMS material. Some responses have been attributed to contamination of the samples by an oxide layer that sits atop the deposit. Others have been attributed to the aging of the material involved. This demonstrates the importance of defining the boundary between the oxide layer and the fresh VMS material that is to be sent to the processing plant. In-situ markers, either related to soluble copper content, mineralogical evidence, or poor RQD indicative of the degradation of the material, need to be deployed to properly isolate this upper material that is not floatable (to date).

From the historical work, which had the benefit of dealing with fresh material (and excluding the samples from the oxide layer), processing of this polymetallic mineralized material is feasible as long as sufficient primary grinding and regrinding intensities are deployed to counteract the limitations created by the fine-grained nature of the lead, copper, and zinc, in decreasing order of difficulty. Relatively high losses of both lead and copper are to be expected if middlings with pyrite are rejected and then an upgrading of these two metals into separate concentrates is carried out to achieve saleable final grades.

Flotation of zinc has been the most straightforward, with rejection from the prior circuits dealing with copper and lead being very effective, even if an amount of the zinc is initially deported into the initial rougher concentrate of copper, lead, or bulk Cu-Pb. A more lenient regrinding target can also be adopted and would still result in a final concentrate grade that is quite acceptable, despite the high 8% Fe content carried by the sphalerite matrix per limited microprobe data.

Expected recovery levels at the roughing stages for all economic metals have been fairly well defined by the testwork to date. Floating a bulk concentrate Cu-Pb, instead of the sequential route, is also ensuring a high rougher recovery levels for both metals as it is not necessary to introduce depressant against one or another at the roughing stage yet. All rougher recovery predictions can then be aligned with an expectation of minimum remaining metal grade in the rougher tails, as incorporated in the methodology adopted for projecting the metal recoveries and typical of plant operations. The main area requiring further work remains the upgrading capability of separate copper and lead concentrates, in the bulk cleaning circuit and the separation circuit, as the limitations brought by the intimate mineralogical assemblage of copper and lead with pyrite will dictate the extent to which recovery has to be sacrificed in order to achieve the minimum level of upgrading required to produce saleable concentrates.

Silver recovery in the bulk concentrate is acceptable; however, as cleaning progresses for the copper and/or lead, the tetrahedrite that carries a portion of the silver units is rejected, along with whatever amount that may also be found in the discarded pyrite. This limits residual silver recovery. Dividing the resulting overall silver recovery in the bulk concentrate between separate copper and lead concentrates will reduce the silver grade of one of these products. The payability of the silver content, after smelter deduction, may thus decrease for the portion of silver that goes to the copper concentrate, if silver deportment favours its recovery with the lead concentrate. The typical lead concentrate smelting terms, on the other hand, have no upfront deduction and pay for a fixed percentage of the silver content, if above a minimum level.

Gold has not been tracked extensively through the testwork. It may be worth confirming its mode of occurrence and grain size in the future through a QEMSCAN/PIMA analysis. Indications are that its recovery is limited and its eventual grade into the copper concentrate, the only product from where it could be paid by the smelters, will barely surpass the typical minimum deduction from standard smelting contracts.

14 MINERAL RESOURCE ESTIMATES

14.1 Introduction

The purpose of this section is to update the mineral resource estimate with drill holes completed in 2017 to 2019 for the Murray Brook Project. The mineral resource estimate presented herein is reported in accordance with the Canadian Securities Administrators' National Instrument 43-101 (2014) and has been estimated in conformity with the generally accepted CIM "Estimation of Mineral Resource and Mineral Reserves Best Practices" guidelines (2019). Mineral resources are not mineral reserves and do not have demonstrated economic viability. There is no guarantee that all or any part of the mineral resource will be converted into a mineral reserve. Confidence in the estimate of inferred mineral resource is insufficient to allow the meaningful application of technical and economic parameters or to enable an evaluation of economic viability worthy of public disclosure. Mineral resources may be affected by further infill and exploration drilling that may result in increases or decreases in subsequent mineral resource estimates.

This mineral resource estimate was based on information and data supplied by Canadian Copper, and was undertaken by Yungang Wu, P. Geo., Antoine Yassa, P. Geo., and Eugene Puritch, P.Eng., FEC, CET, of P&E Mining Consultants Inc. (P&E) of Brampton, Ontario, all independent qualified persons as defined in NI 43-101. The effective date of this mineral resource estimate is October 3, 2023.

14.2 Previous Mineral Resource Estimate

The previous mineral resource estimate for Murray Brook dated December 21, 2016, at a net smelter return (NSR) cut-off of C$85/t for potential underground mining is presented in Table 14-1. This previous mineral resource estimate is superseded by the mineral resource estimate reported herein.

Table 14-1: Mineral Resource Estimate on December 21, 2016, at C$85/t NSR Cut-off

Zone	Classification	Tonnes (k)	Cu (%)	Cu (Mlb)	Pb (%)	Pb (Mlb)	Zn (%)	Zn (Mlb)	Au (g/t)	Au (koz)	Ag (g/t)	Ag (Moz)
Oxide	Measured (M)	434	1.13	10.8	1.44	13.8	4.51	43.2	0.31	4.3	60.5	0.8
	Indicated (I)	105	1.94	4.5	0.82	1.9	2.84	6.6	0.46	1.6	45.3	0.2
	M + I	539	1.29	15.3	1.32	15.7	4.19	49.8	0.34	5.9	57.5	1.0
	Inferred	4	3.94	0.3	0.19	0.0	0.62	0.0	0.46	0.1	26.6	0.0
Sulphide	Measured	3,681	0.36	29	1.87	151.9	5.57	451.7	0.56	65.8	70.5	8.3
	Indicated	1,603	0.7	24.8	1.63	57.4	4.48	158.4	0.88	45.1	65.3	3.4
	M + I	5,284	0.46	53.8	1.8	209.3	5.24	610.1	0.65	110.9	68.9	11.7
	Inferred	125	2.16	5.9	0.92	2.5	2.58	7.1	0.54	2.2	47.3	0.2

14.3 Database

All drilling data consisting of collar coordinates, survey, lithology, bulk density, assay, and drill core recovery were provided in the form of Excel data files by Canadian Copper. The GEOVIA GEMS™ V6.8.4 database for this mineral resource estimate, compiled by the authors, consisted of 189 drill holes totalling 36,498 m, of which 28 drill holes totalling 7,131 m were completed from 2017 to 2019, after the previous mineral resource estimate. A total of 165 drill holes intersected the mineralized wireframes used for the mineral resource estimate (see Table 14-2). All pre-2010 drill holes were not utilized for this mineral resource estimate due to their non-verifiable nature. A drill hole plan is shown in Figure 14-1 (overleaf).

Table 14-2: Murray Brook Drill Hole Database Summary

Year	Number of Drill Holes	Drill Hole Length (m)	Number of Drill Holes Intersecting Wireframes	Length* of Drill Holes Intersecting Wireframes (m)
2010 to 2012	161	29,367	146	26,544
2017 to 2019	28	7,131	19	4,395
Total	189	36,498	165	30,939

Note: *entire length of hole.

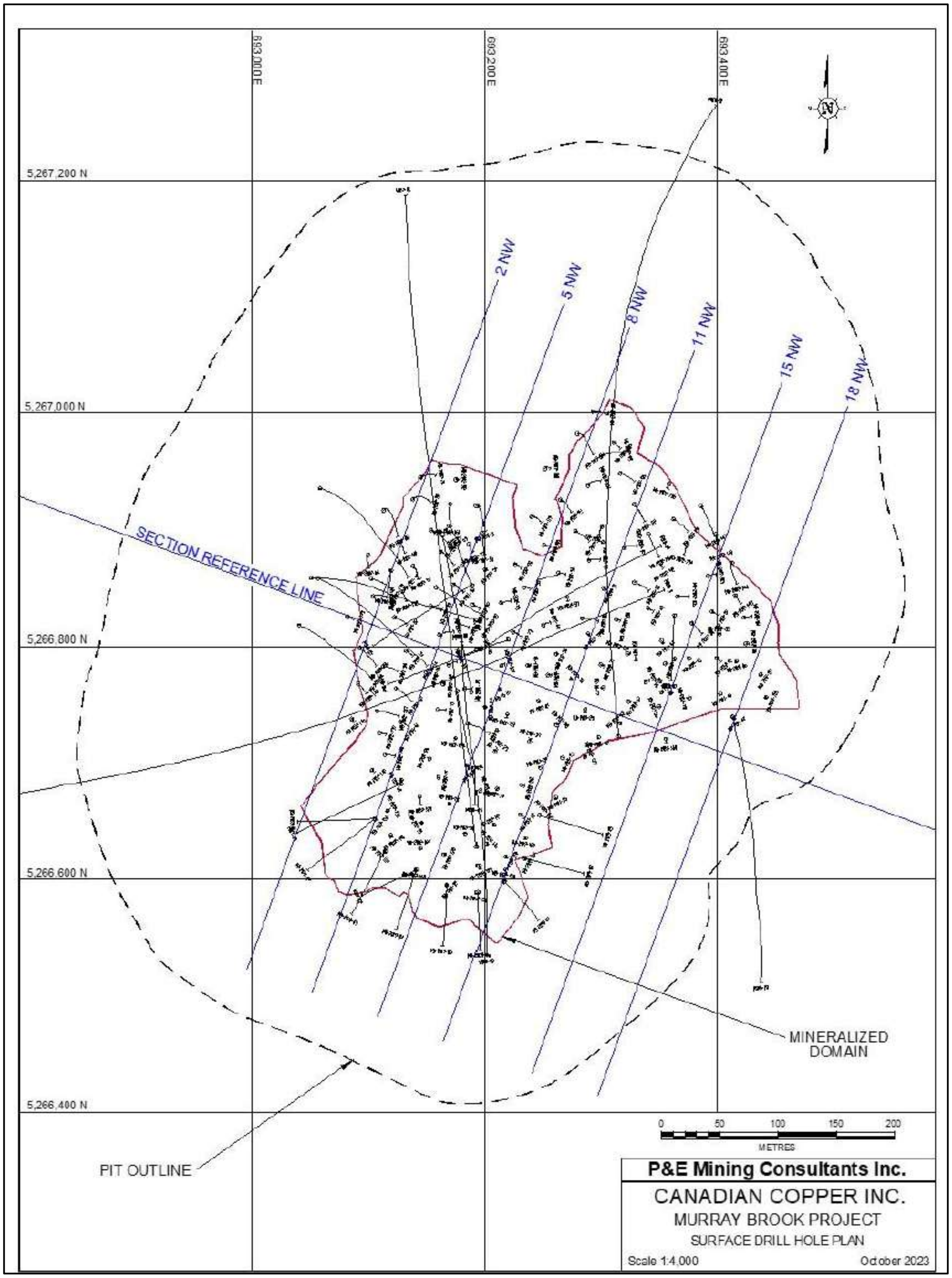
The drill hole database contained assays for copper, zinc, lead, silver, gold, and other lesser elements of non-economic importance as well as bulk density. The basic statistics of all raw assays are presented in Table 14-3. All drill hole survey and assay values are expressed in metric units, whereas grid coordinates are in NAD 83 UTM Zone 19.

Table 14-3: Murray Brook Assay Database Summary

Variable	Cu (%)	Zn (%)	Pb (%)	Ag (g/t)	Au (g/t)	Length (m)	Density (t/m³)
Number of Samples	12,896	12,896	12,896	12,896	11,921	12,896	2,961
Minimum Value	0.00	0.00	0.00	0.00	0.00	0.08	2.04
Maximum Value	11.88	30.66	20.50	729.90	5.49	21.00	6.86
Mean	0.39	2.14	0.78	32.92	0.46	1.02	4.14
Median	0.17	0.94	0.29	18.70	0.21	1.00	4.31
Variance	0.52	9.77	1.58	1,754.68	0.38	0.11	0.72
Standard Deviation	0.72	3.13	1.26	41.89	0.61	0.33	0.85
Coefficient of Variation	1.87	1.46	1.61	1.27	1.32	0.33	0.21
Skewness	5.22	2.72	4.06	3.21	2.37	25.20	-0.09
Kurtosis	41.83	12.85	32.86	23.88	10.78	1196.02	1.99

Note: Cu = copper, Zn = zinc, Pb = lead, Ag = silver, Au = gold, length = assay interval.

Figure 14-1: Drill Hole Plan



14.4 Data Verification

The authors verified the assay data against the original laboratory certificates during previous and current mineral resource estimates; and also validated the database by checking for inconsistencies in analytical units, duplicate entries, interval, length or distance values less than or equal to zero, blank or zero-value assay results, out-of-sequence intervals, intervals, or distances greater than the reported drill hole length, inappropriate collar locations, survey and missing interval and coordinate fields. No errors were identified in the database. The authors consider that the supplied database is suitable for mineral resource estimation.

14.5 Domain Interpretation

A single mineralized domain was created with computer screen digitizing on drill hole cross-sections in GEOVIA GEMS™ by the authors. The domain outline was determined from lithology, structure, and NSR value by visually inspecting the drill hole vertical cross-sections. Twenty drill cross-sections were developed on 20 m spacing looking along an azimuth of 290°. The digitized outlines were influenced by the selection of mineralized material above a cut-off NSR value of C$20/t that demonstrated zonal continuity along strike and down dip. In some cases, mineralization below C$20/t NSR were included for the purpose of maintaining zonal continuity. On each cross-section, polyline interpretations were digitized from drill hole to drill hole, but not extended nominally more than 30 m into untested territory. Minimum constrained drill core length for interpretation was approximately 2.0 m. The interpreted polylines from each cross-section were connected from each vertical cross-section to form a 3-D wireframe of each domain in GEMS™.

As presented in Table 14-4, copper is negatively correlated with zinc and lead. Higher-grade copper is predominately resided at edge, whereas higher-grade zinc and lead are at the middle of the deposit. Using Cu >0.5% and Pb+Zn <2% as boundary limits, a copper sub-domain and a Pb+Zn sub-domain have been generated within the mineralized domain.

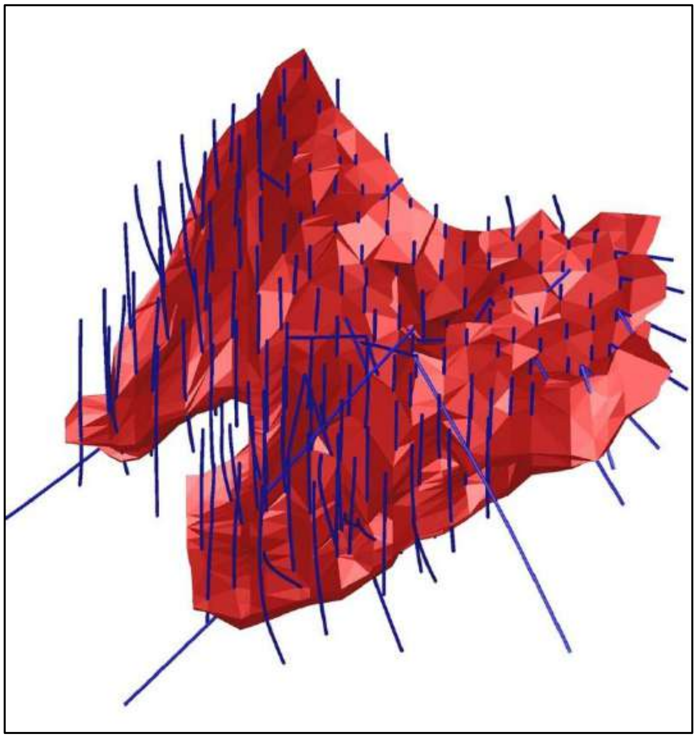
Table 14-4: Correlation Coefficient Table of Constrained Assays

Element	Cu	Zn	Pb	Pb+Zn	Ag	Au	Length	Bulk Density
Copper	1	-0.16	-0.15	-0.16	-0.05	0.02	0.04	0.04
Zinc	-0.16	1	0.87	0.99	0.80	0.23	-0.00	0.18
Lead	-0.15	0.87	1	0.93	0.86	0.28	-0.01	0.14
Lead + Zinc	-0.16	0.99	0.93	1	0.84	0.25	-0.01	0.17
Silver	-0.05	0.80	0.86	0.84	1	0.36	0.02	0.11
Gold	0.02	0.23	0.28	0.25	0.36	1	-0.02	0.28
Length	0.04	-0.00	-0.01	-0.01	0.02	-0.02	1	-0.10
Bulk Density	0.04	0.18	0.14	0.17	0.11	0.28	-0.10	1

Continuous interior waste (NSR <C$20/t) intervals were excluded from the mineralized domain.

Topographic, overburden, and oxidation boundary surfaces were generated too. The mineralized domain was truncated with the overburden surface. Oxidized and sulphide mineralization were separated with an oxidation surface. The resulting domains were utilized for statistical analysis, grade interpolation, rock coding and mineral resource reporting purposes. The wireframe of the mineralized domain is displayed in Figure 14-2.

Figure 14-2: Wireframe of the Mineralized Domain



14.6 Rock Code Determination

A unique rock code was assigned to each domain in the mineral resource model as presented in Table 14-5.

Table 14-5: Rock Codes Used for the Mineral Resource Estimate

Rock Type	Rock Code	Notes
Zinc + Lead	100	Mineralization sub-domain
Copper	200	Mineralization sub-domain
Oxide	110	Oxidized mineralization
Sulphide	120	Fresh mineralization
Air	0	-
Overburden	10	-
Waste	99	-

14.7 Wireframe Constrained Assays

Wireframe constrained assays were back coded in the assay database with model rock codes that were derived from intersections of the mineralization solids and drill holes. The basic statistics of mineralized wireframe constrained assays are presented in Table 14-6.

Table 14-6: Basic Statistics of Constrained Assays

Domain	Variable	Cu (%)	Zn (%)	Pb (%)	Ag (g/t)	Au (g/t)	Length (m)
Zinc + Lead	Number of Samples	8,880	8,880	8,880	8,880	8,097	8,880
	Minimum Value	0.001	0.001	0.001	0.01	0.001	0.08
	Maximum Value	11.88	30.66	20.50	729.90	5.49	11.00
	Mean	0.32	3.00	1.09	44.23	0.62	1.01
	Median	0.20	1.96	0.68	31.90	0.36	1.00
	Variance	0.21	11.75	1.95	2,062.50	0.45	0.05
	Standard Deviation	0.46	3.43	1.40	45.41	0.67	0.22
	Coefficient of Variation	1.44	1.14	1.28	1.03	1.09	0.22
	Skewness	6.43	2.31	3.60	2.98	1.98	17.19
	Kurtosis	85.51	10.15	27.10	21.57	8.54	619.29
Copper	Number of Samples	1,434	1,434	1,434	1,434	1,353	1,434
	Minimum Value	0.001	0.000	0.001	0.001	0.000	0.16
	Maximum Value	9.48	6.77	1.58	109.60	5.08	21.00
	Mean	1.42	0.49	0.18	17.33	0.29	1.06
	Median	0.91	0.32	0.11	13.70	0.19	1.00
	Variance	2.06	0.29	0.04	187.57	0.12	0.56
	Standard Deviation	1.43	0.54	0.20	13.70	0.35	0.75
	Coefficient of Variation	1.01	1.11	1.09	0.79	1.21	0.71
	Skewness	2.28	2.81	2.37	1.90	4.16	17.24
	Kurtosis	8.97	19.34	11.13	8.36	37.32	395.28

Note: Cu = copper, Zn = zinc, Pb = lead, Ag = silver, Au = gold, length = assay interval.

14.8 Compositing

In order to regularize the assay sampling intervals for grade interpolation, a 1.0 m compositing length was selected for the drill hole intervals that fell within the constraints of the above-mentioned mineral resource wireframe domains. The composites were calculated over 1.0 m lengths starting at the first point of intersection between assay data hole and hanging wall of the three-dimensional (3-D) zonal constraint. The compositing process was halted on exit from the footwall of the aforementioned constraint. A total of 864 intervals missing gold assays from the 2018 drill holes were treated as nulls. If the last composite interval was <0.25 m, the composite length was adjusted to make all composite intervals equal within the domain. The resulting composite length ranged from 0.9 to 1.3 m. This process would not introduce any short sample bias in the grade interpolation process. The constrained composite data were extracted to a point file for a grade capping analysis. The composite statistics are summarized in Table 14-7.

Table 14-7: Composite Summary Statistics

Variable	Cu (%)	Zn (%)	Pb (%)	Ag (g/t)	Au (g/t)	Length (m)
Number of Samples	10,411	10,411	10,411	10,411	9,538	10,411
Minimum Value	0.00	0.00	0.00	0.00	0.00	0.90
Maximum Value	9.06	30.25	19.81	501.07	5.28	1.30
Mean	0.48	2.65	0.96	40.84	0.57	1.00
Median	0.25	1.62	0.54	28.01	0.33	1.00
Variance	0.60	10.42	1.68	1782.10	0.40	0.00
Standard Deviation	0.77	3.23	1.30	42.21	0.63	0.01
Coefficient of Variation	1.60	1.22	1.35	1.03	1.11	0.01
Skewness	4.49	2.42	3.57	2.73	2.14	3.83
Kurtosis	30.29	10.82	26.09	16.05	9.54	106.27

Note: Cu = copper composite, Zn = zinc composite, Pb = lead composite, Ag = silver composite, Au = gold composite, length = composite length.

14.9 Grade Capping

Grade capping was investigated on the 1.0 m composite values in the database within the constraining domain to ensure that the possible influence of erratic high-grade values did not bias the database. Log-normal histograms and probability plots for copper, zinc, lead, silver, and gold composites were generated for each mineralized domain and the selected resulting graphs are exhibited in Figure 14-3. The grade capping values are detailed in Table 14-8. The capped composite statistics are summarized in Table 14-9. The capped composites were utilized to develop variograms and for use in block model grade interpolation.

Figure 14-3: Log-Normal Histograms and Probability Plots

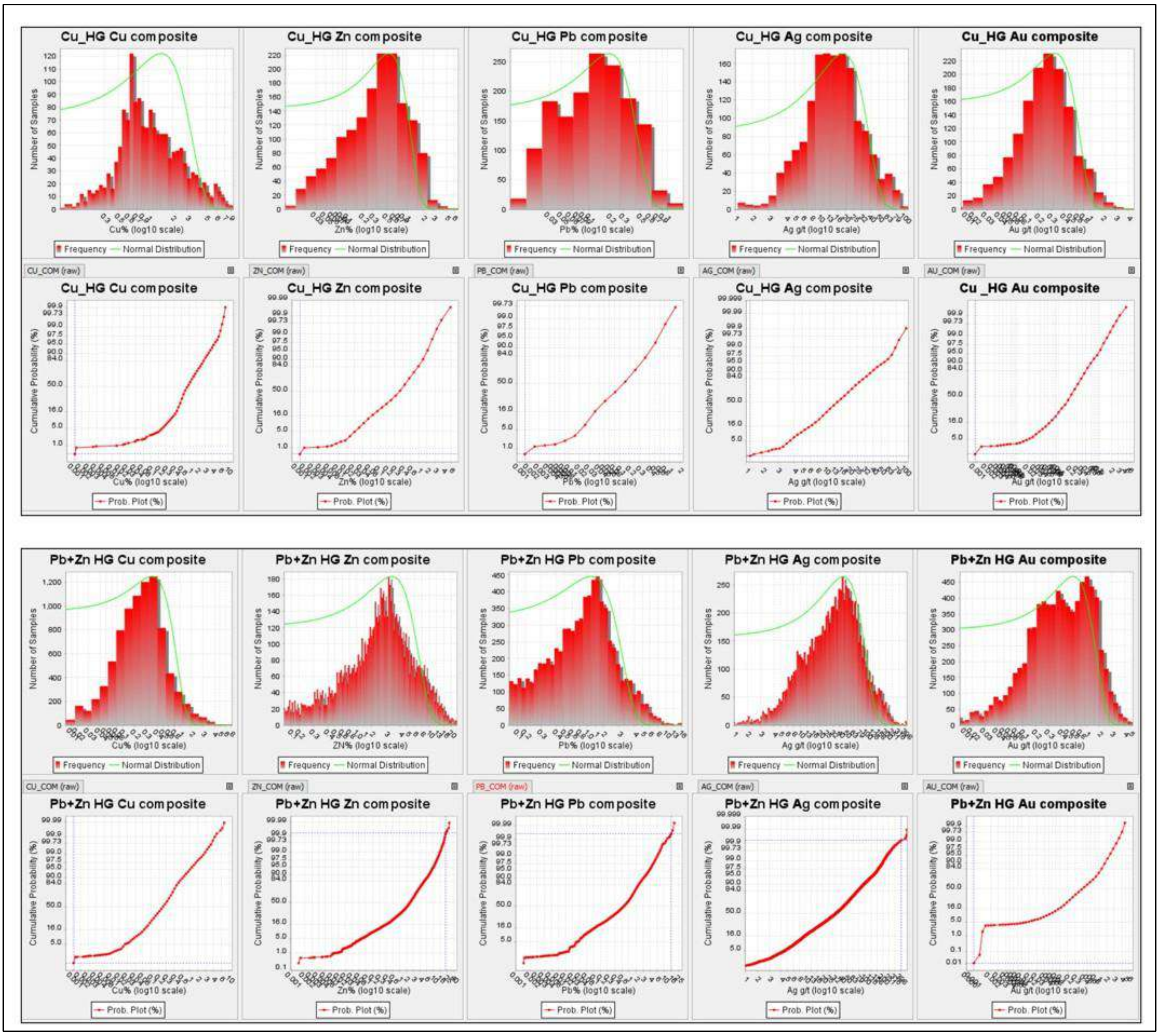


Table 14-8: Murray Brook Grade Capping Values

Domain	Element	Total No. of Composites	Capping Value*	No. of Capped Composites	Mean of Composites*	Mean of Capped Composites*	CoV of Composites	CoV of Capped Composites	Capping Percentile (%)
Lead + Zinc	Copper	8,831	No Cap	0	0.320	0.320	1.37	1.37	100.0
	Zinc	8,831	24	5	3.038	3.035	1.10	1.10	99.94
	Lead	8,831	16	3	1.104	1.103	1.23	1.22	99.97
	Silver	8,831	410	5	44.800	44.750	0.99	0.98	99.94
	Gold	8,831	No Cap	0	0.620	0.620	1.06	1.06	100.0
Copper	Copper	1,580	No Cap	0	1.400	1.400	0.98	0.98	100.0
	Zinc	1,580	No Cap	0	0.500	0.500	1.08	1.08	100.0
	Lead	1,580	No Cap	0	0.180	0.180	1.06	1.06	100.0
	Silver	1,580	No Cap	0	18.720	18.720	0.84	0.84	100.0
	Gold	1,499	No Cap	0	0.290	0.290	1.15	1.15	100.0

Notes: CoV = coefficient of variation. * Unit of measure for copper, zinc and lead is percentage (%); unit of gold and silver is grams per tonne (g/t).

Table 14-9: Capped Composite Summary Statistics

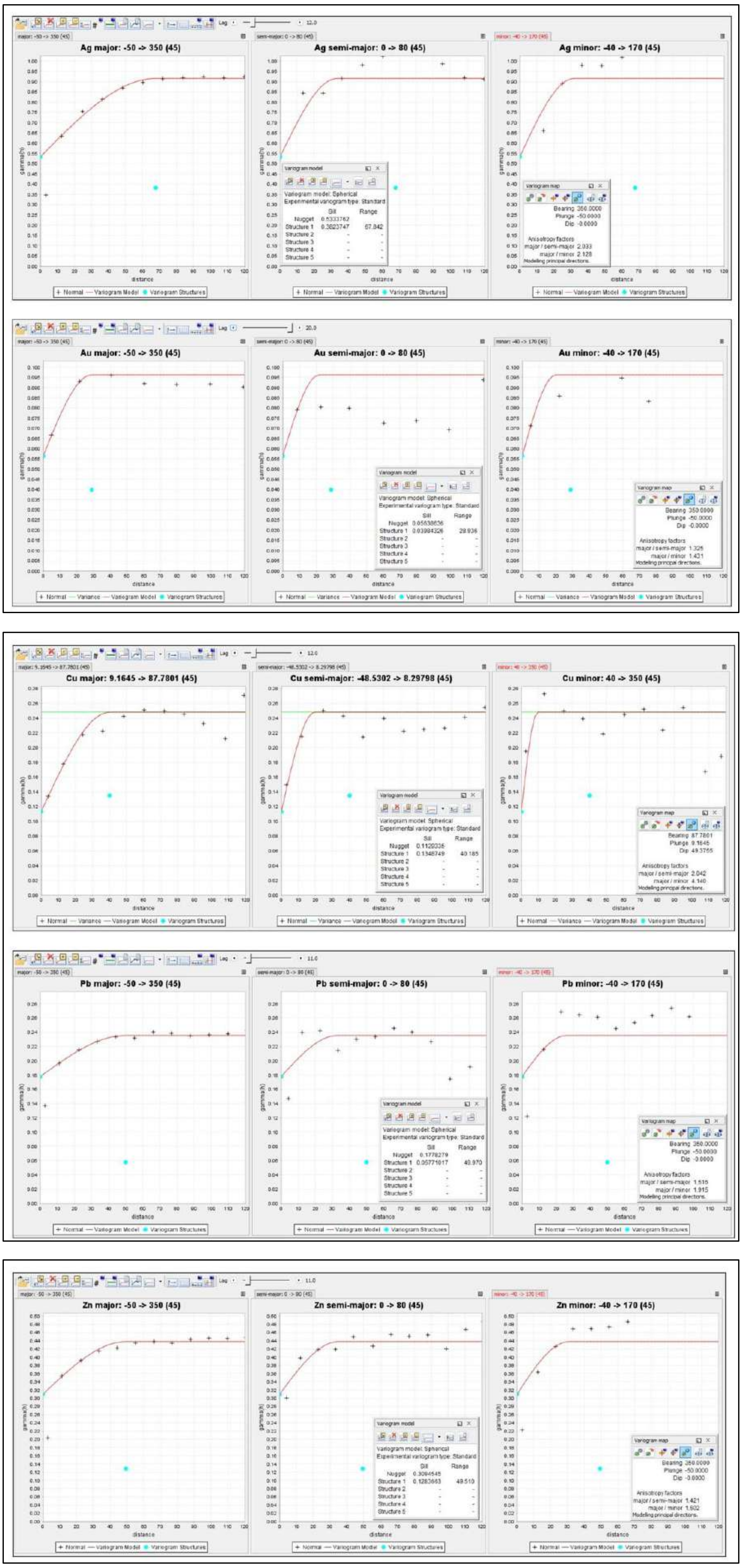
Variable	Cu (%)	Zn (%)	Pb (%)	Ag (g/t)	Au (g/t)
Number of Samples	10,411	10,411	10,411	10,411	9,538
Minimum Value	0.00	0.00	0.00	0.00	0.00
Maximum Value	9.06	24.00	16.00	410.00	5.28
Mean	0.48	2.65	0.96	40.80	0.57
Median	0.25	1.62	0.54	28.01	0.33
Variance	0.6	10.33	1.66	1,750.20	0.40
Standard Deviation	0.77	3.21	1.29	41.84	0.63
Coefficient of Variation	1.60	1.21	1.34	1.03	1.11
Skewness	4.49	2.35	3.40	2.53	2.14
Kurtosis	30.29	10.03	22.55	13.11	9.54

Notes: Cu = copper capped composite, Zn = zinc capped composite, Pb = lead capped composite, Ag = silver capped composite, Au = gold capped composite.

14.10 Variography

A variography analysis was performed as a guide to determining a grade interpolation search strategy. Directional variograms were attempted using the capped composites. Selected variograms are shown in Figure 14-4.

Figure 14-4: Variography Analysis Results



Continuity ellipses based on the observed ranges were subsequently generated and utilized as the basis for estimation search ranges, distance weighting calculations, and mineral resource classification criteria.

14.11 Bulk Density

The database consists of a total of 2,961 bulk density measurements, of which 1,888 bulk densities were determined from 2017 to 2018 drill holes, and 1,073 analyses were performed on 2011 to 2012 drill core, all using the wet immersion method. The bulk density varied from 2.04 to 6.86 t/m^3 and averaged 4.14 t/m^3. A total of 2,050 of these bulk densities were constrained inside the mineralization wireframes. The bulk density block model was interpolated with the constrained data, which was capped at 6.5 t/m^3.

14.12 Block Modelling

The Murray Brook block model was constructed using GEOVIA GEMS™ V6.8.4 modelling software. The block model origin and block size are presented in Table 14.10. The block model consists of separate model attributes for estimated grades of copper, zinc, lead, silver, gold, and NSR, rock type (mineralization domains), volume percent, bulk density, and classification.

Table 14-10: Murray Brook Block Model Definition

Direction	Origin	No. of Blocks	Block Size (m)
X	692,511.73	368	3
Y	5,266,507.171	362	3
Z	612	154	3
Rotation	-20 (clockwise)		

Note: Origin for a block model in GEMS™ represents the coordinate of the outer edge of the block with minimum X and Y, and maximum Z.

All blocks in the rock type block model were initially assigned a waste rock code of 99, corresponding to the surrounding country rocks. The mineralized domain was used to code all blocks within the rock type block model that contain 0.01% or greater volume within the domain. These blocks were assigned rock type codes as presented in Table 14-5. The oxidation surface was utilized to update all mineralization blocks above the surface to oxide and below to sulphide. The surfaces of overburden and topography were subsequently utilized to assign rock codes 10 and 0, corresponding to overburden and air respectively, to all blocks 50% or greater above the surfaces.

A volume percent block model was set up to accurately represent the volume and subsequent tonnage that was occupied by each block inside the constraining wireframe domain. As a result, the domain boundary was properly represented by the volume percent model ability to measure individual infinitely variable block inclusion percentages within that domain. The minimum percentage of the mineralized block was set to 0.01.

The copper, lead, zinc, and bulk density blocks were interpolated with inverse distance squared (ID^2), while inverse distance cubed (ID^3) was used for the gold and silver grade interpolation, with the capped composites. Nearest neighbour (NN) was utilized for validation. Multiple passes were executed for the grade interpolation to progressively capture the sample points to avoid over-smoothing and preserve local grade variability. Search ranges and directions were based on the variograms. Grade blocks were interpolated using the parameters in Table 14-11.

Table 14-11: Murray Brook Block Model Interpolation Parameters

Attribute	Pass	Major Range (m)	Semi-major Range (m)	Minor Range (m)	Max. No. of Samples per Hole	Min No. of Samples	Max. No. of Samples
Copper	I	25	15	10	3	7	12
	II	40	25	15	3	4	12
	III	80	50	30	3	2	12
Zinc, Lead, Silver, Gold & Bulk Density	I	35	25	20	3	7	12
	II	55	40	35	3	4	12
	III	110	80	70	3	2	12

The NSR value of the mineralized blocks was calculated using the formula below:

NSR $/t = (Cu % x 81) + (Pb % x 12) + (Zn % x 13) + (Ag g/t x 0.90).

Selected cross-sections and plans of the copper, zinc, and NSR grade blocks are presented in Figure 14-5 to 14-10, Figure 14-11 to Figure 14-16, and Figure 14-17 to Figure 14-22, respectively.

Figure 14-5: Copper Block Model Cross-Section – 5 NW

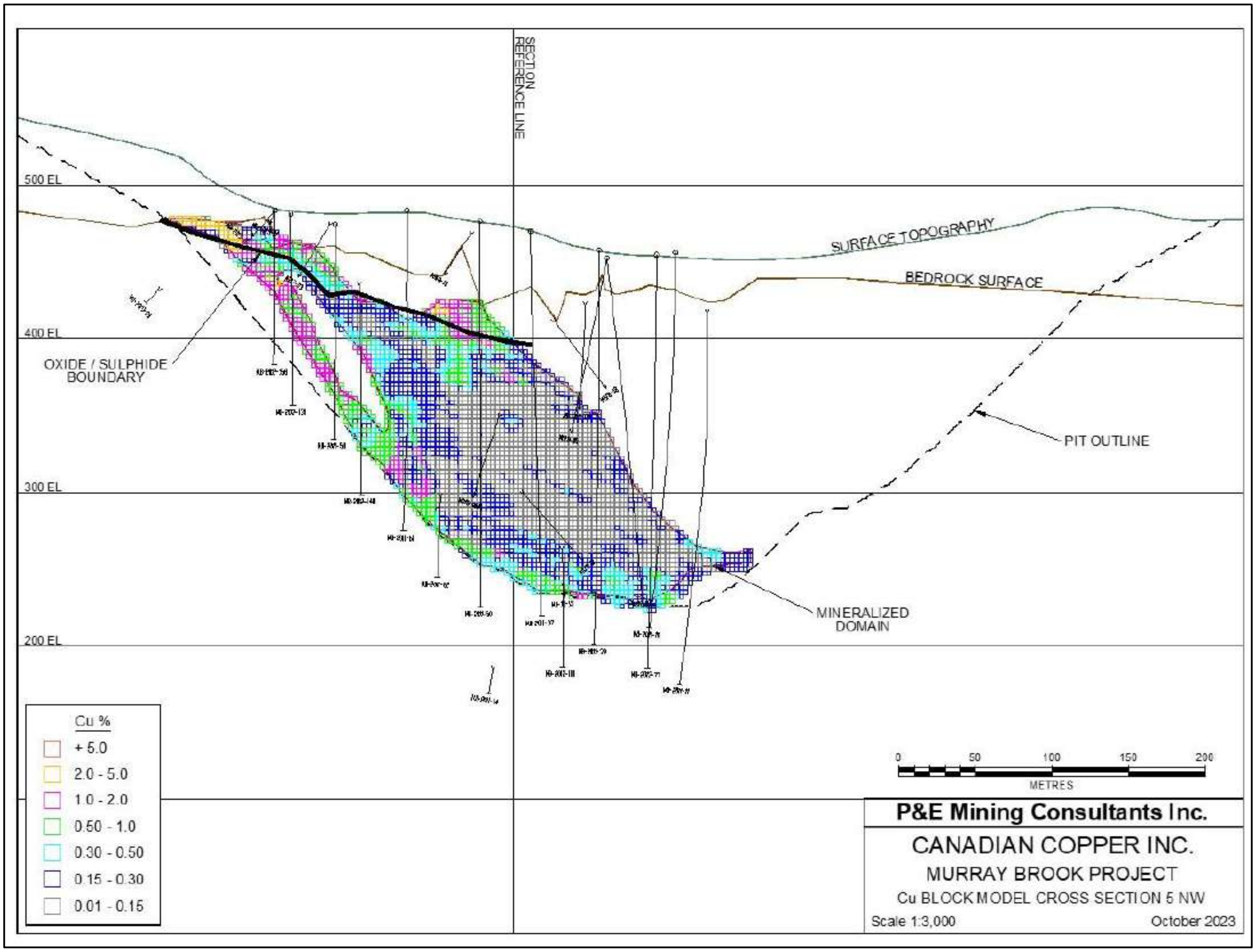


Figure 14-6: Copper Block Model Cross-Section – 8 NW

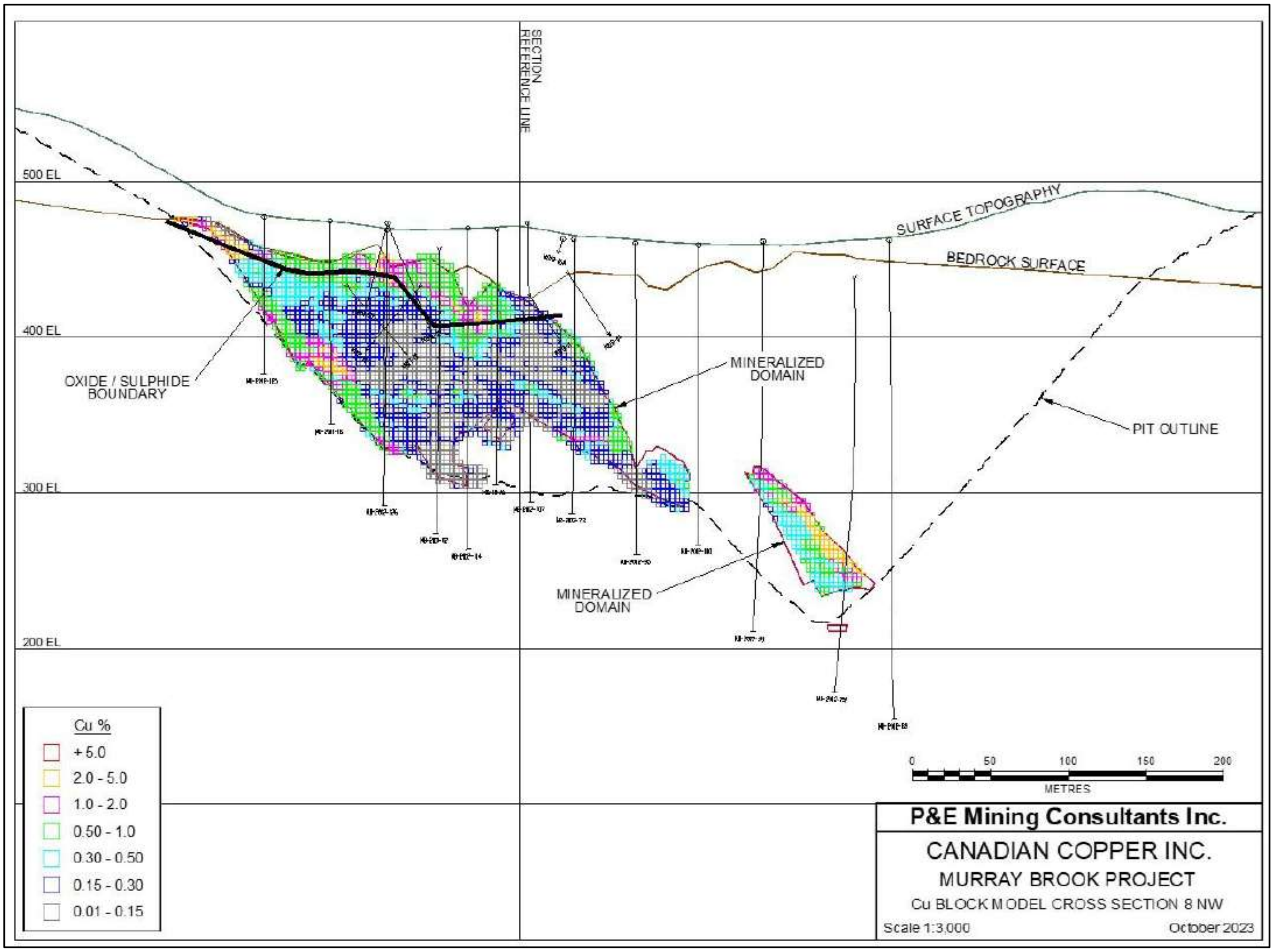


Figure 14-7: Copper Block Model Cross-Section – 11 NW

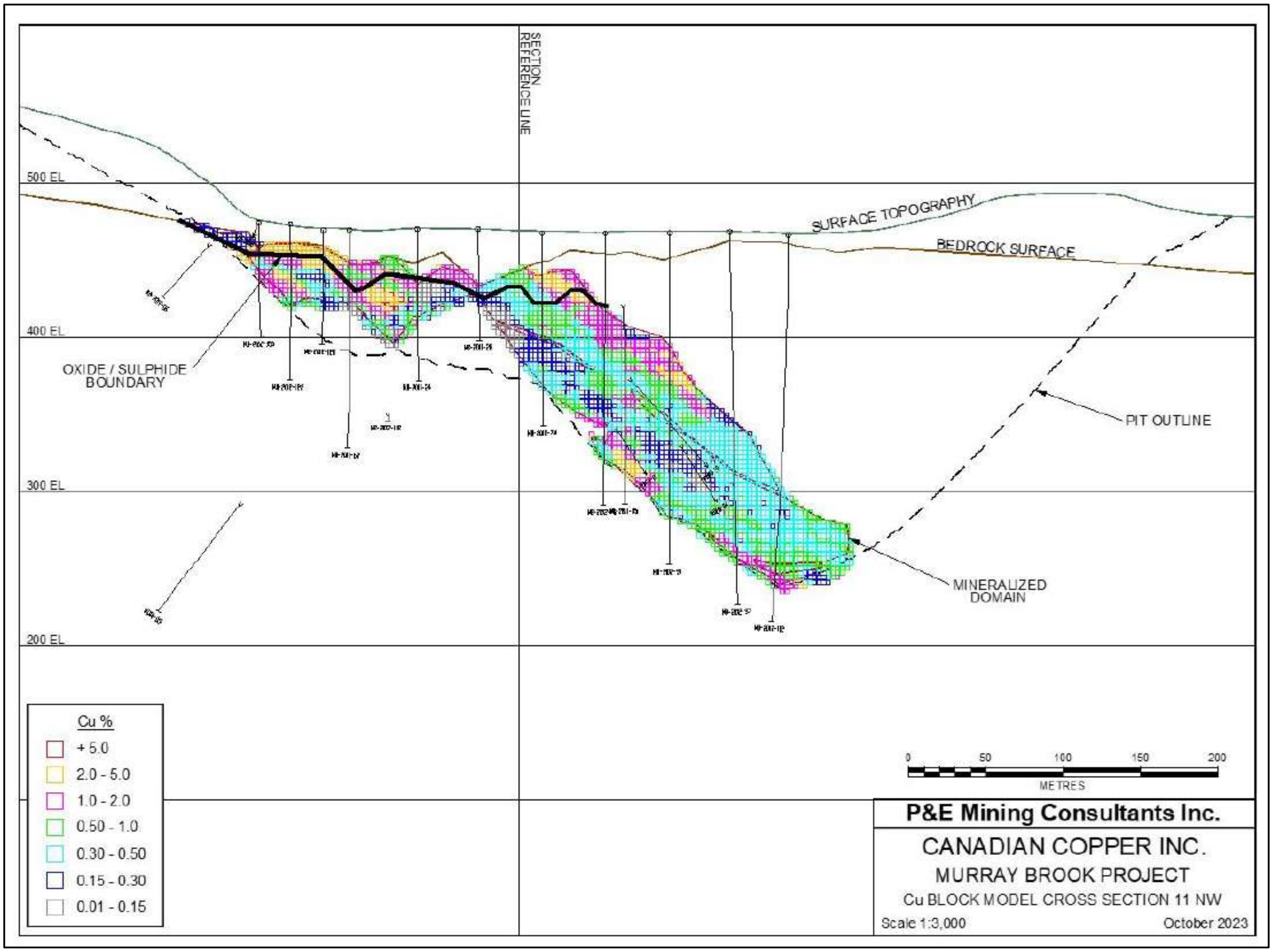


Figure 14-8: Copper Block Model Plan 400 El

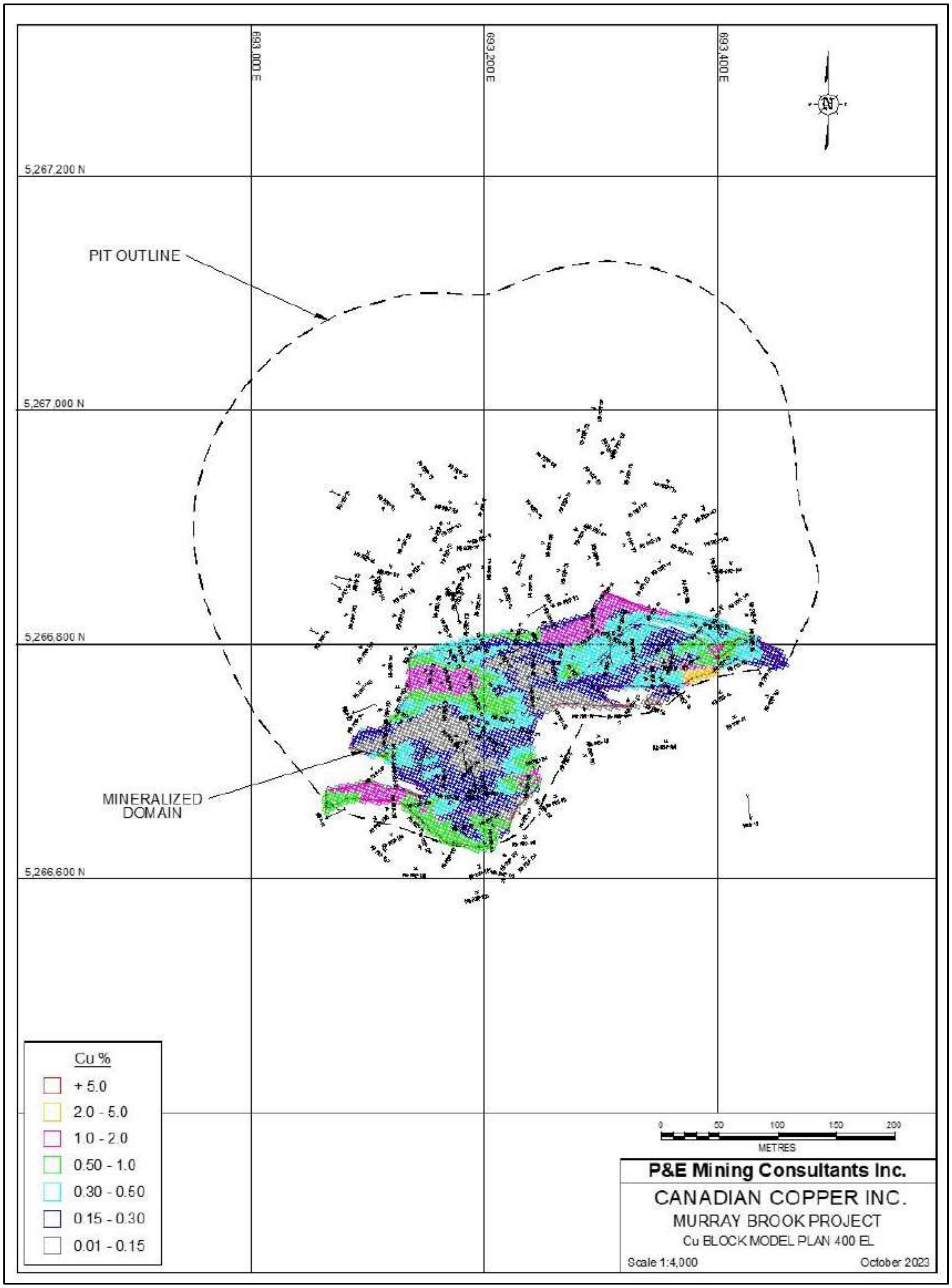


Figure 14-9: Copper Block Model Plan 350 El

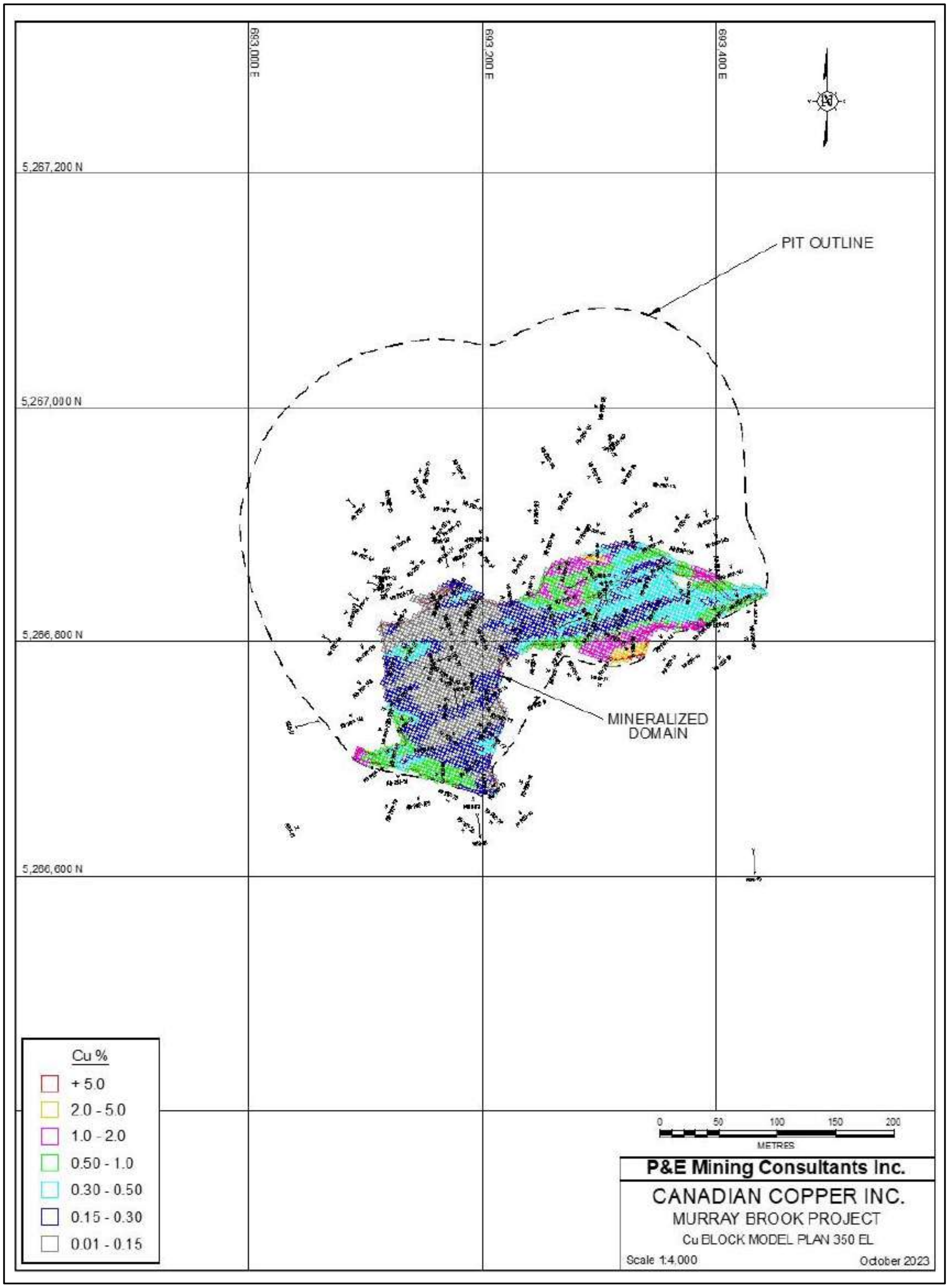


Figure 14-10: Copper Block Model Plan 300 El

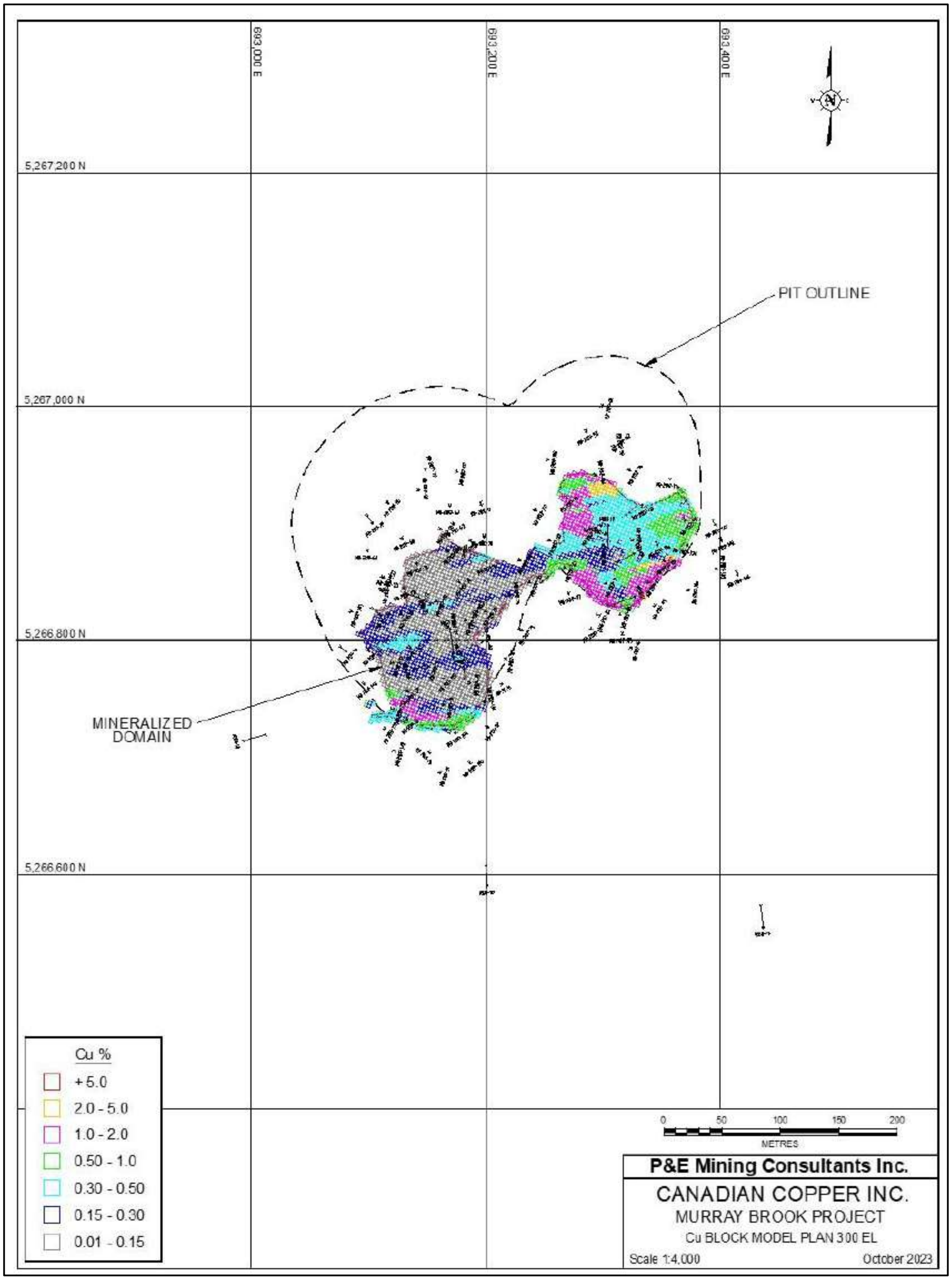


Figure 14-11: Zinc Block Model Cross-Section – 5 NW

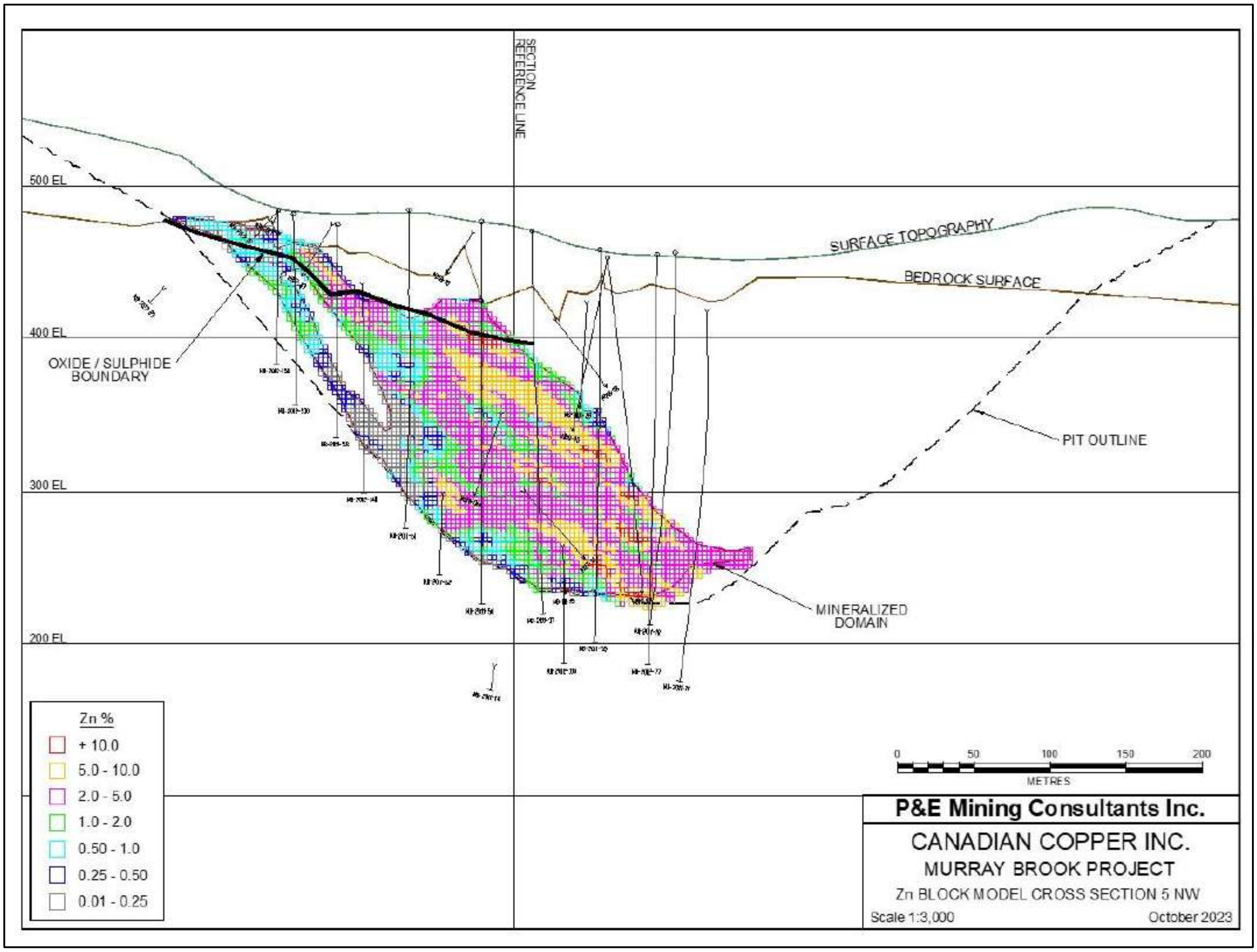


Figure 14-12: Zinc Block Model Cross-Section – 8 NW

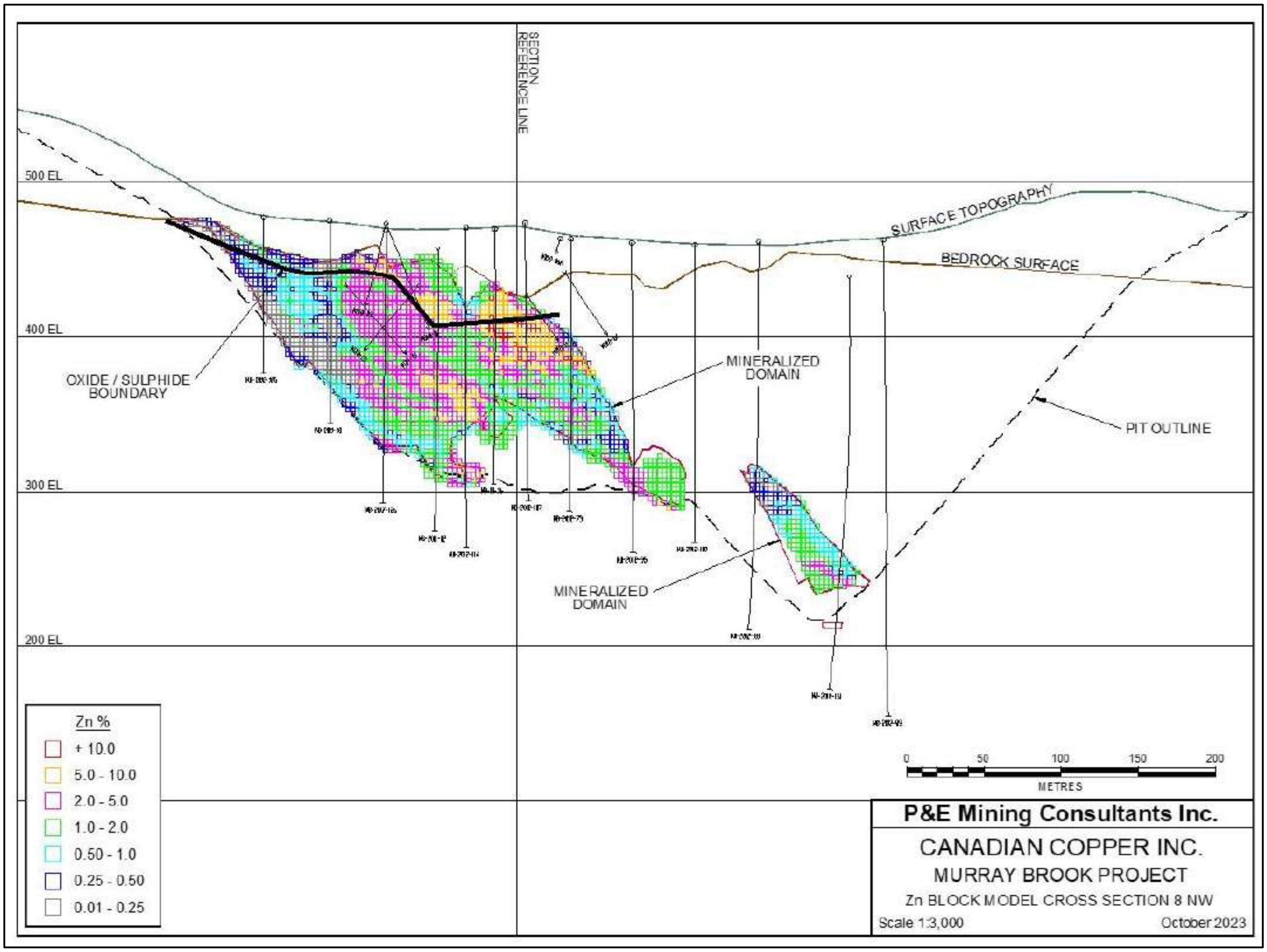


Figure 14-13: Zinc Block Model Cross-Section – 11 NW

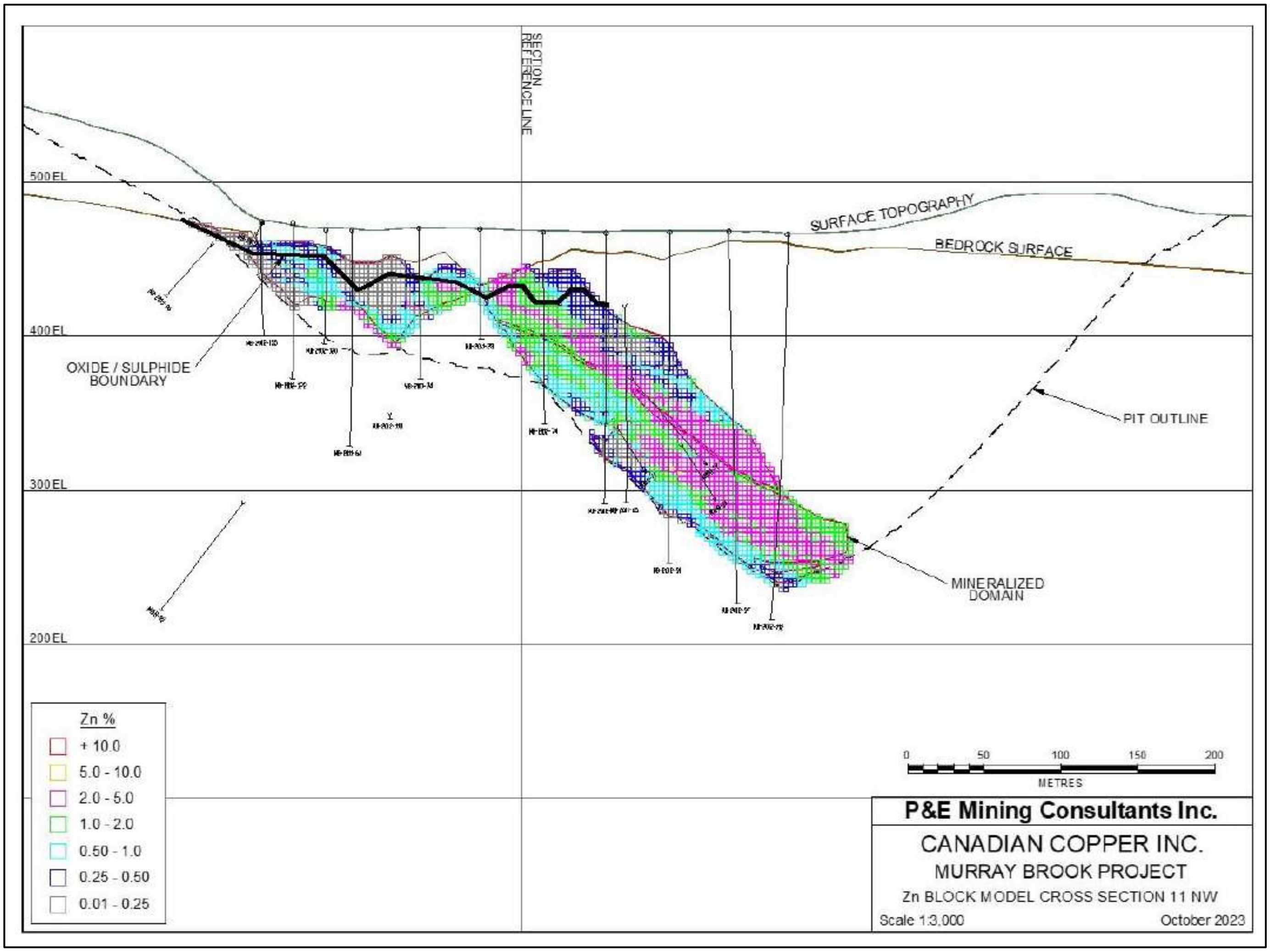


Figure 14-14: Zinc Block Model Plan 400 El

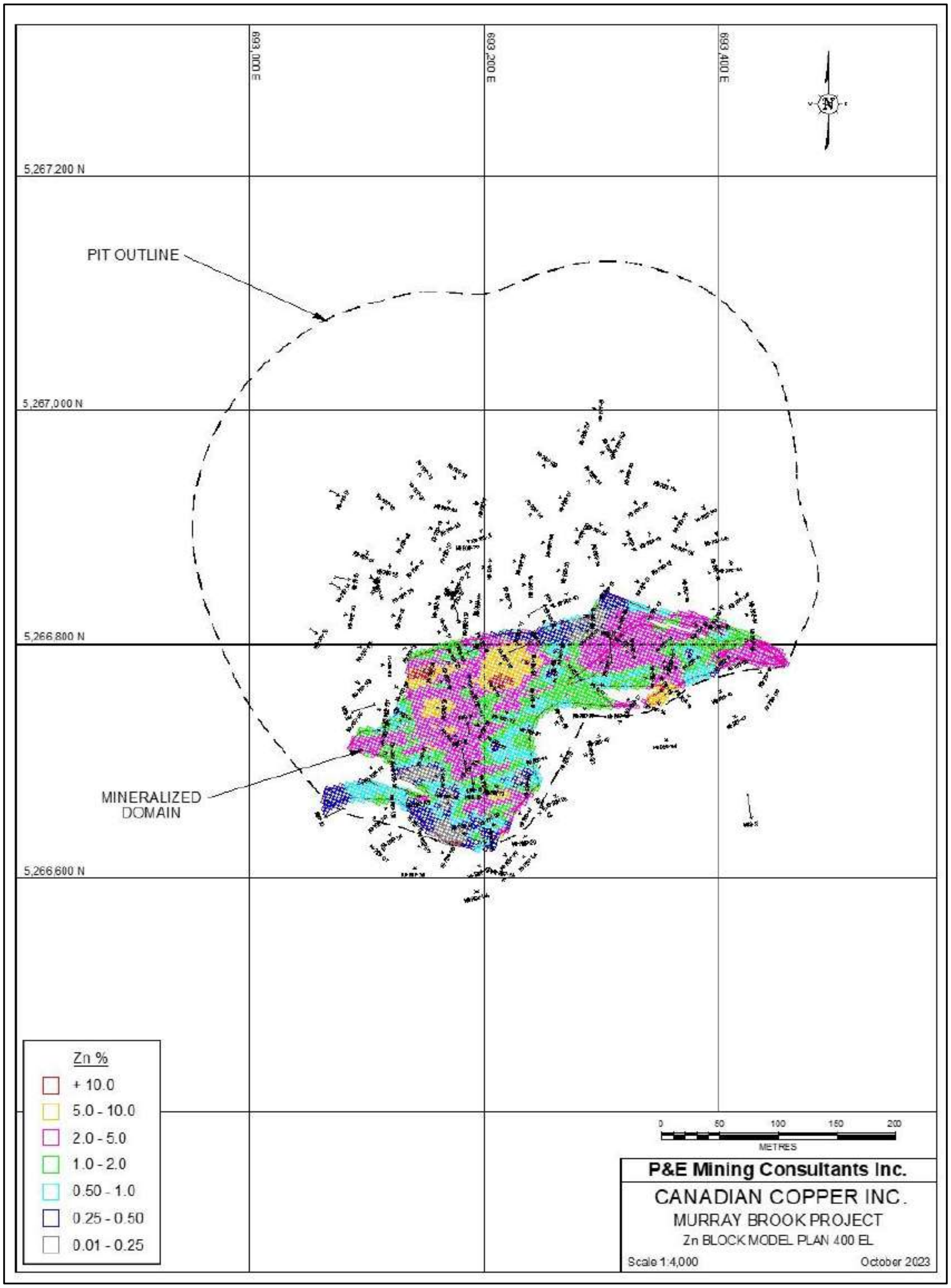


Figure 14-15: Zinc Block Model Plan 350 El

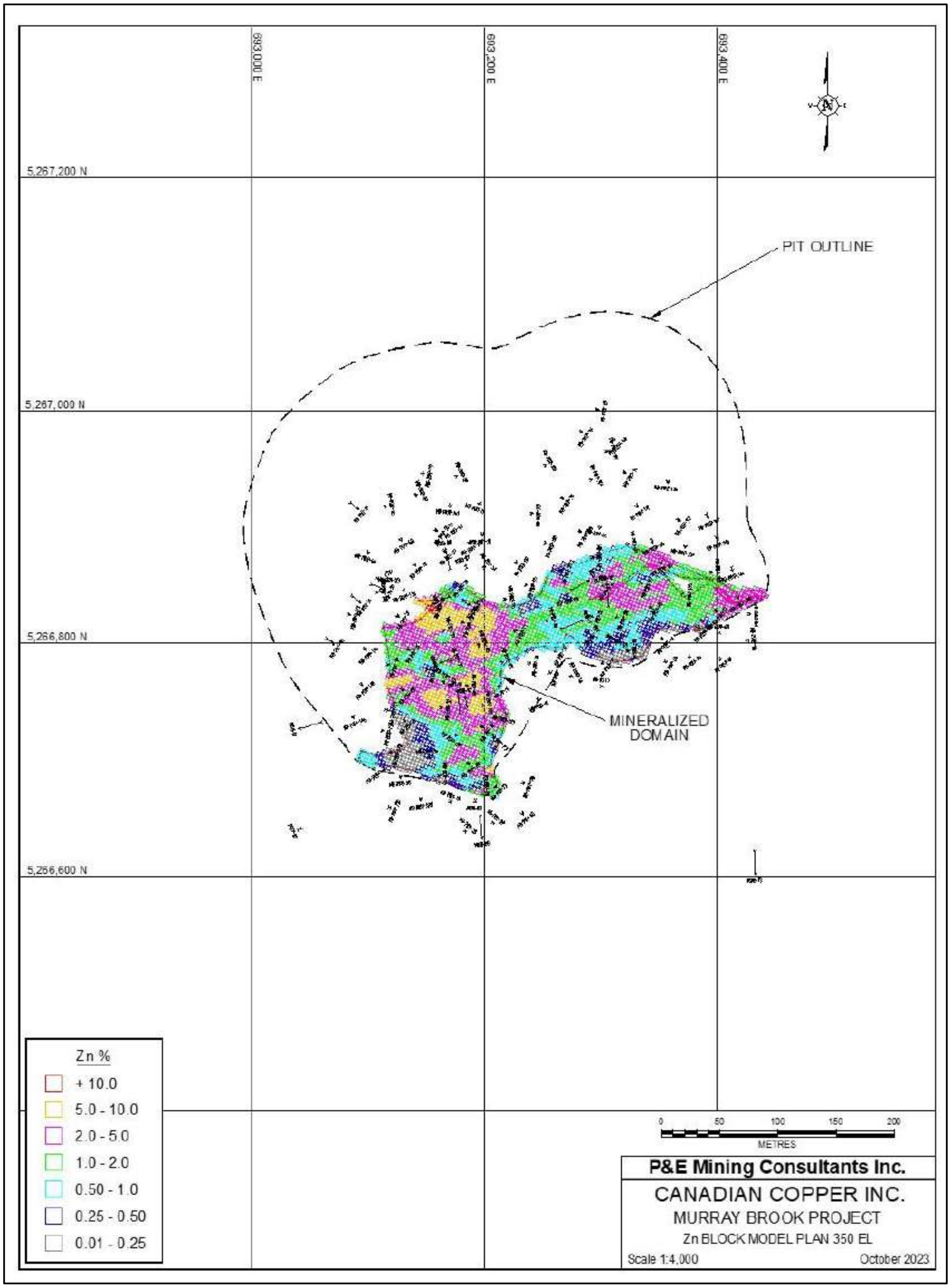


Figure 14-16: Zinc Block Model Plan 300 El

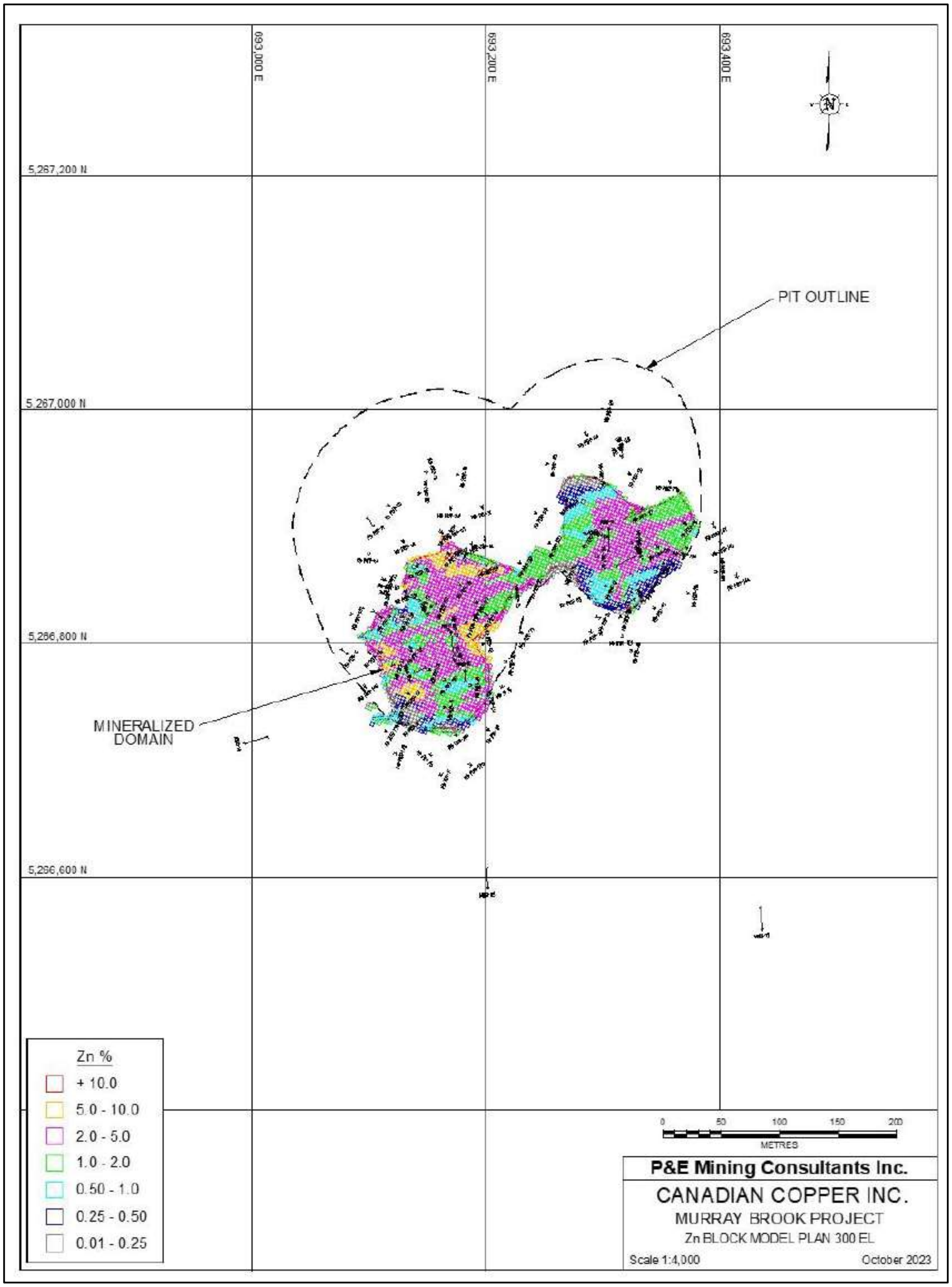


Figure 14-17: NSR Block Model Cross-Section – 5 NW

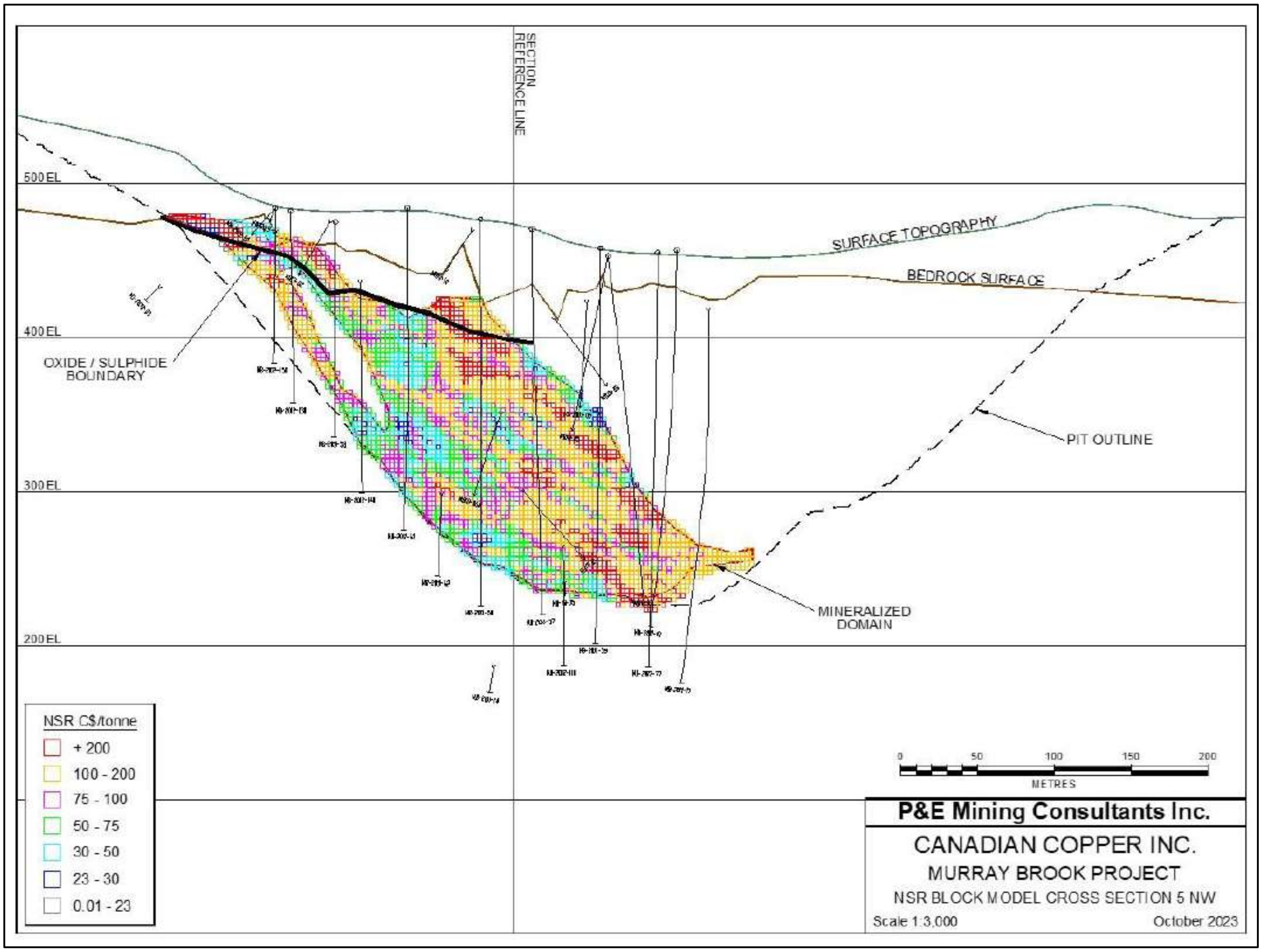


Figure 14-18: NSR Block Model Cross-Section – 8 NW

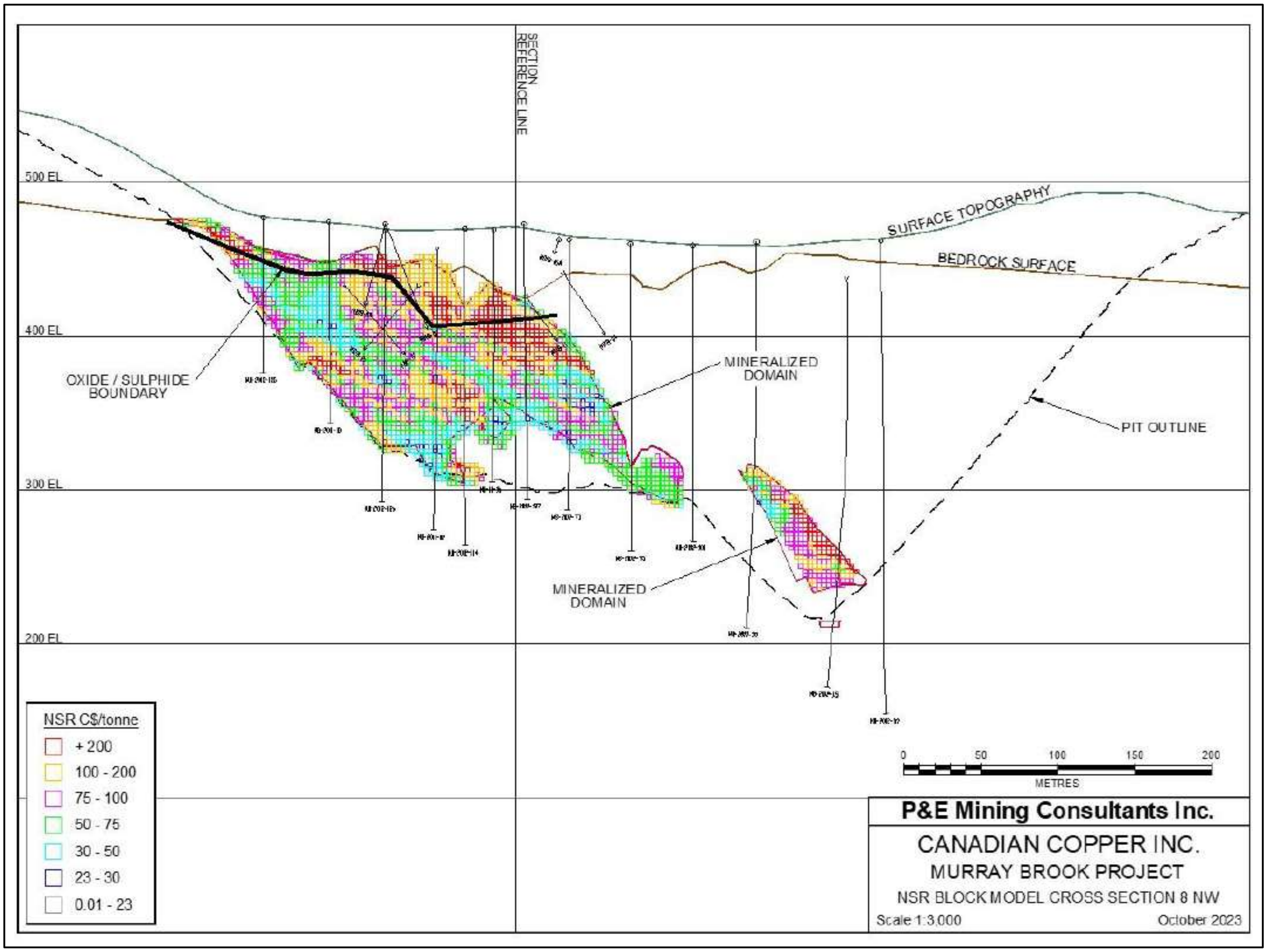


Figure 14-19: NSR Block Model Cross-Section – 11 NW

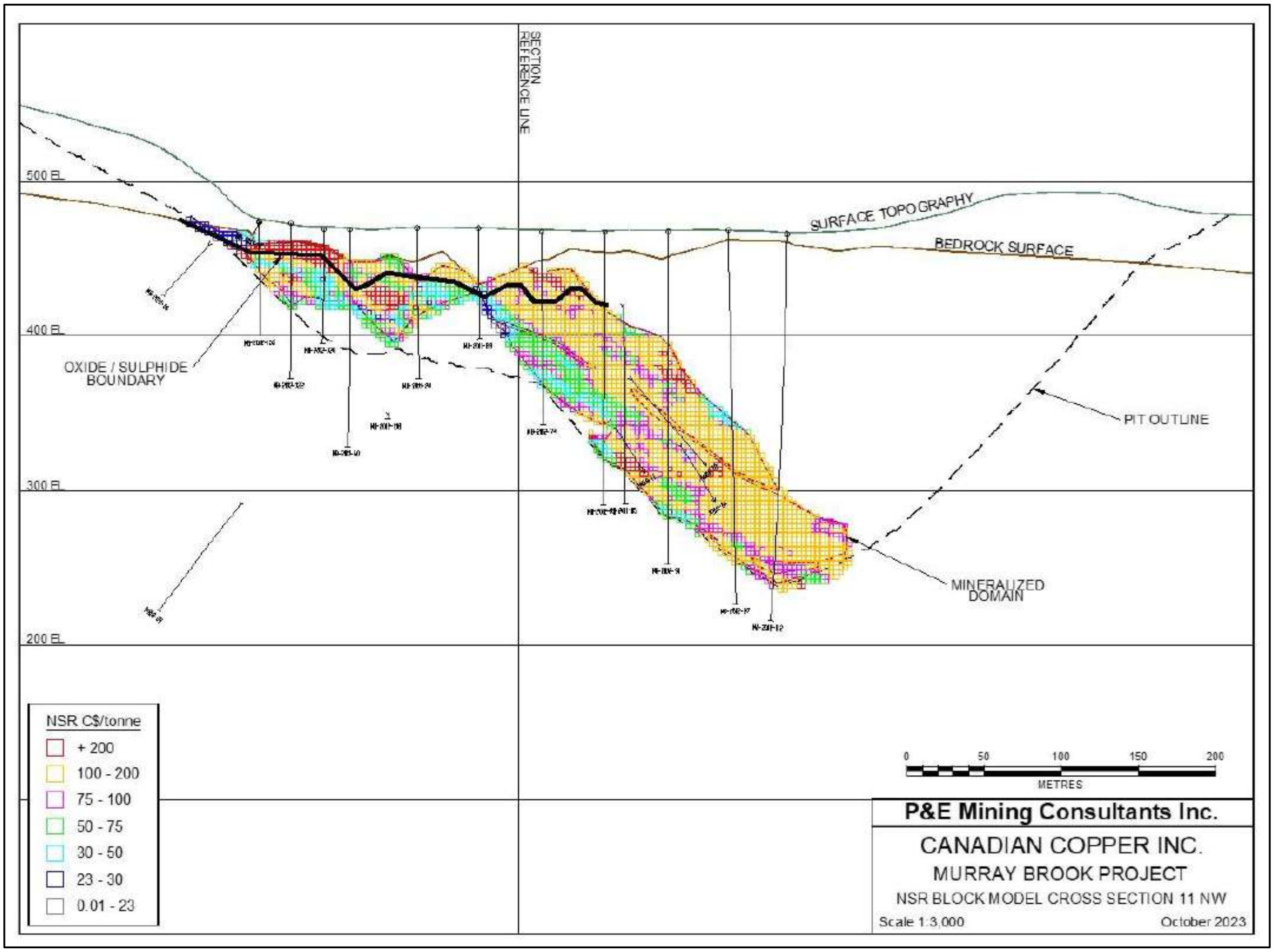


Figure 14-20: NSR Block Model Plan 400 El

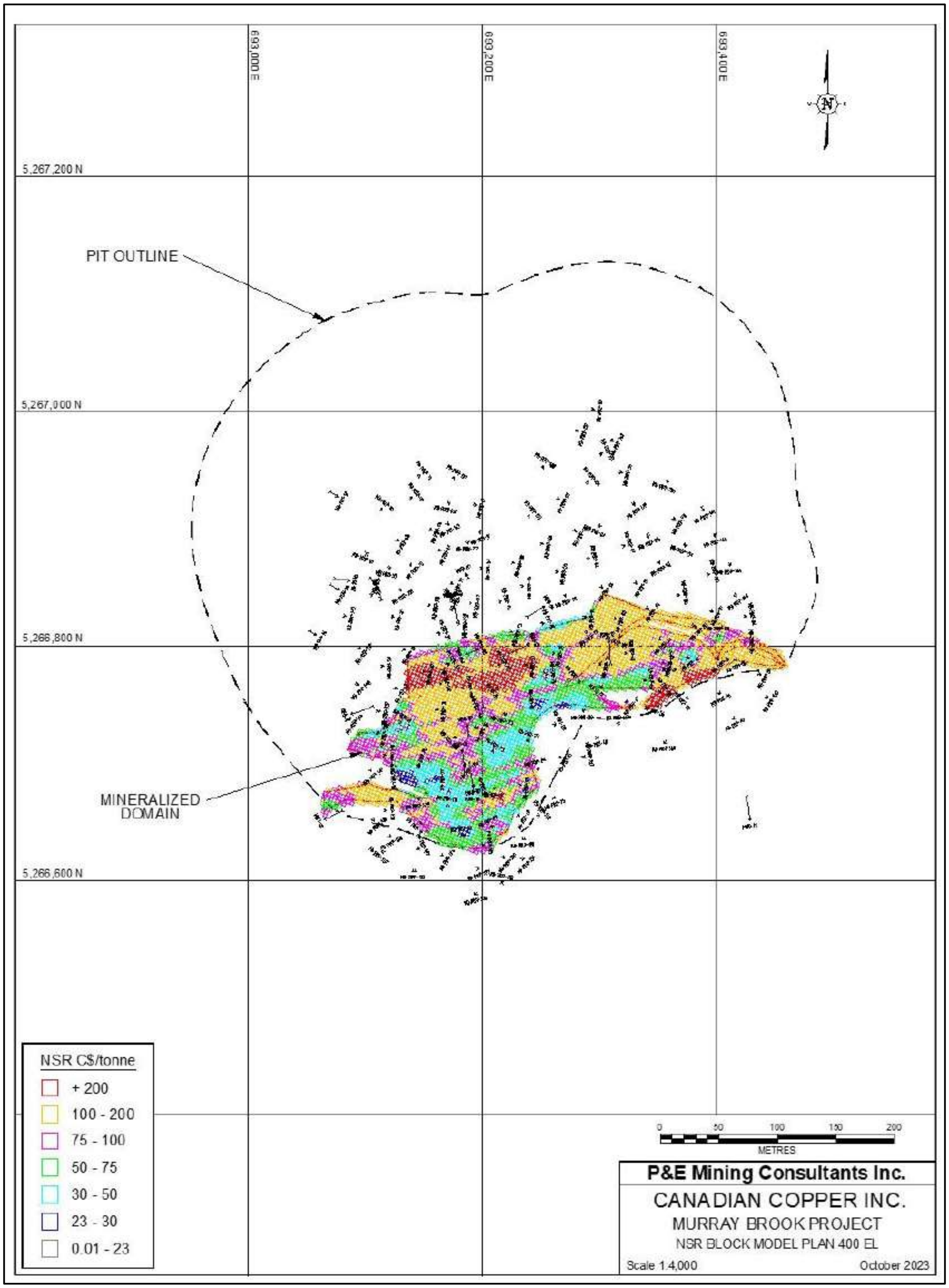


Figure 14-21: NSR Block Model Plan 350 El

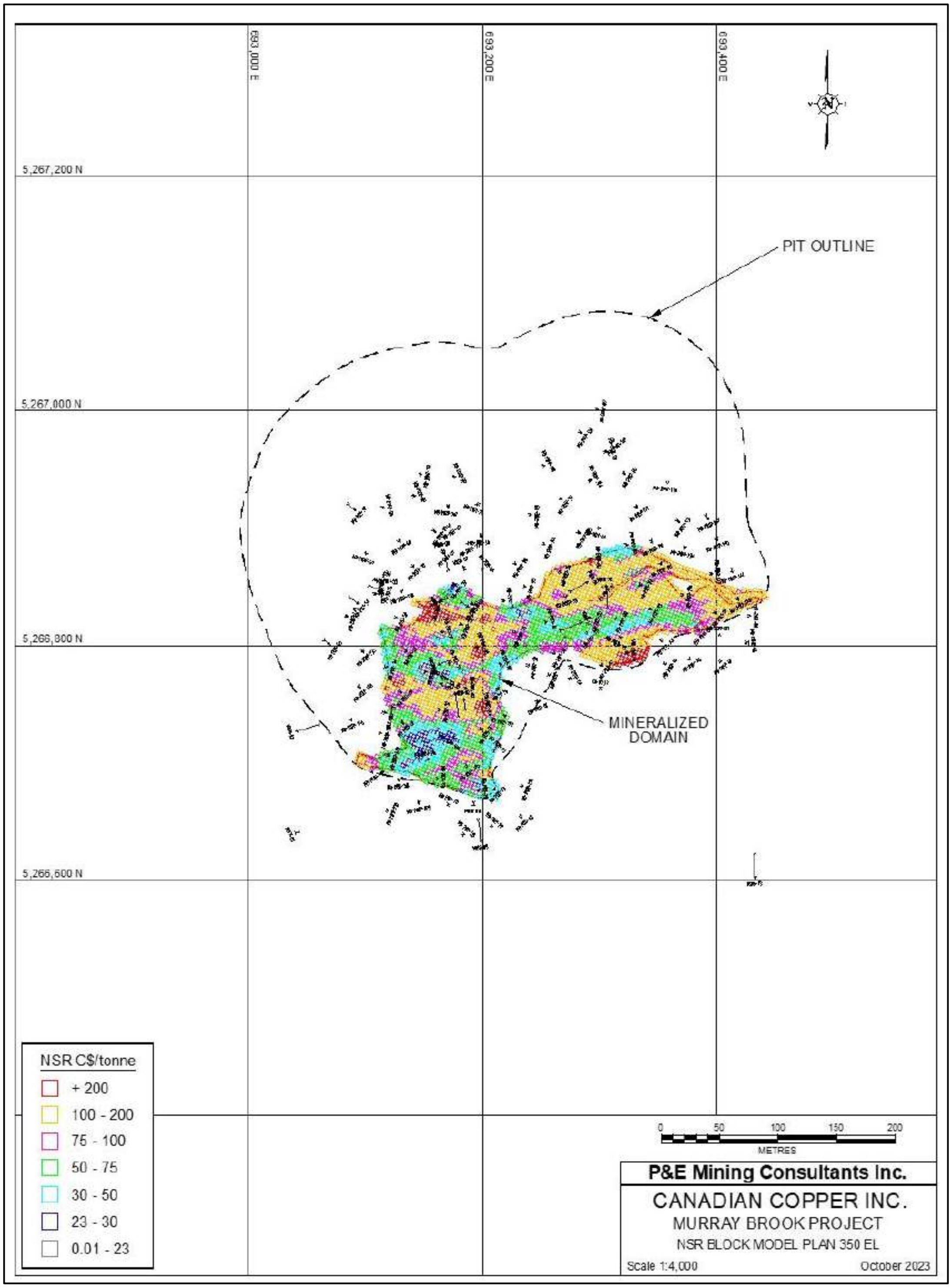
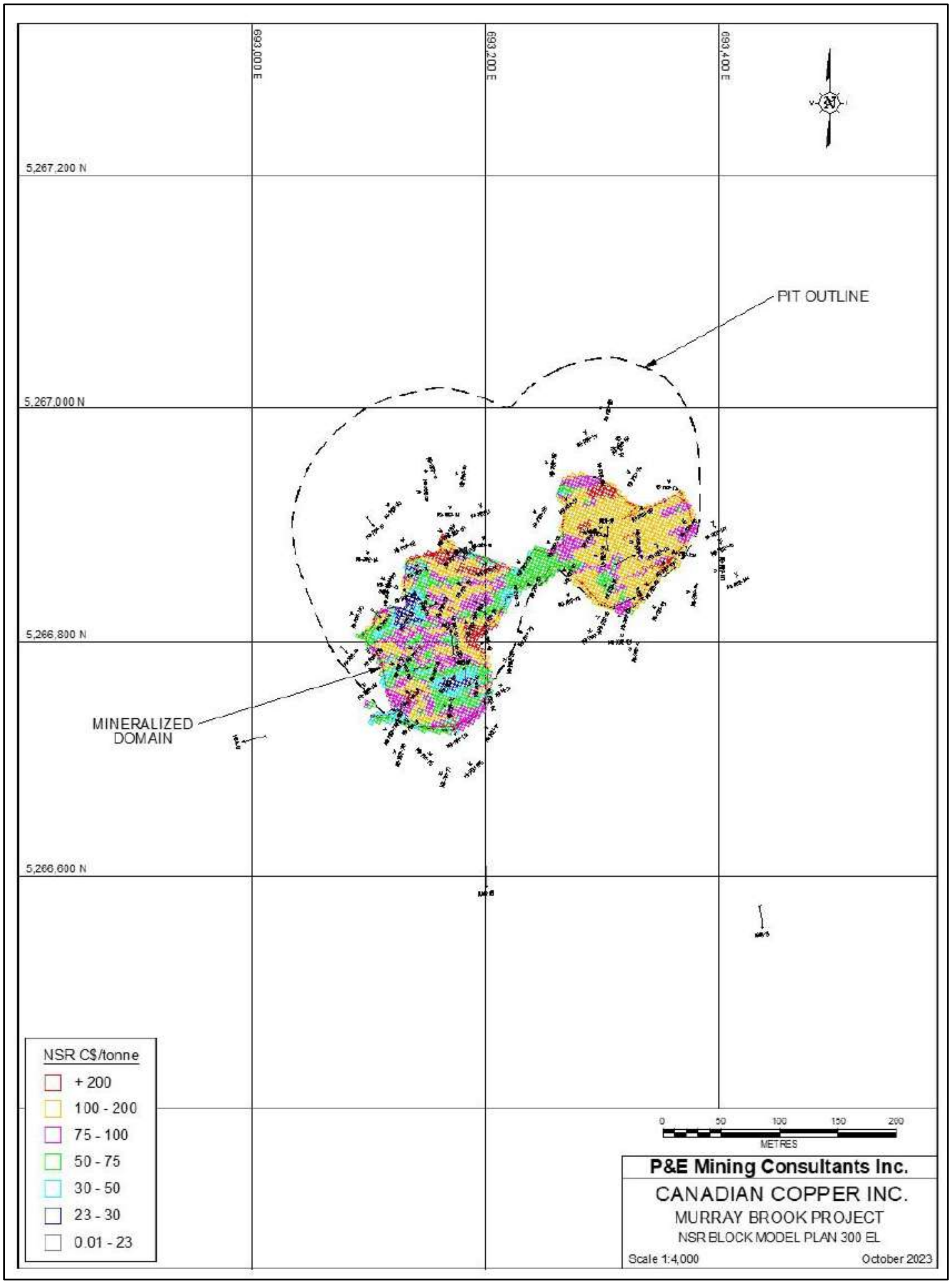


Figure 14-22: NSR Block Model Plan 300 El



14.13 Mineral Resource Classification

It is the opinion of the authors that all drilling, assaying, and exploration work on the Murray Brook Project supports this mineral resource estimate and are sufficient to indicate a reasonable potential for economic extraction, and thus qualify it as a mineral resource under the CIM definition standards. the mineral resource was classified as measured, indicated, and inferred based on the geological interpretation, variogram performance, and drill hole spacing. The measured mineral resource was qualified for the blocks interpolated using at least four drill holes with average spacing <25 m; indicated mineral resource was classified for the blocks interpolated with the Pass II, which used at least four composites from a minimum of two holes; and inferred mineral resources were categorized for all remaining grade blocks within the mineralized domain. The classifications have been adjusted on a longitudinal projection to reasonably reflect the distribution of each classification. Selected classification block cross-sections and plans are attached in Figures 14-23 to 14-28.

Figure 14-23: Classification Block Model Cross-Section – 5 NW

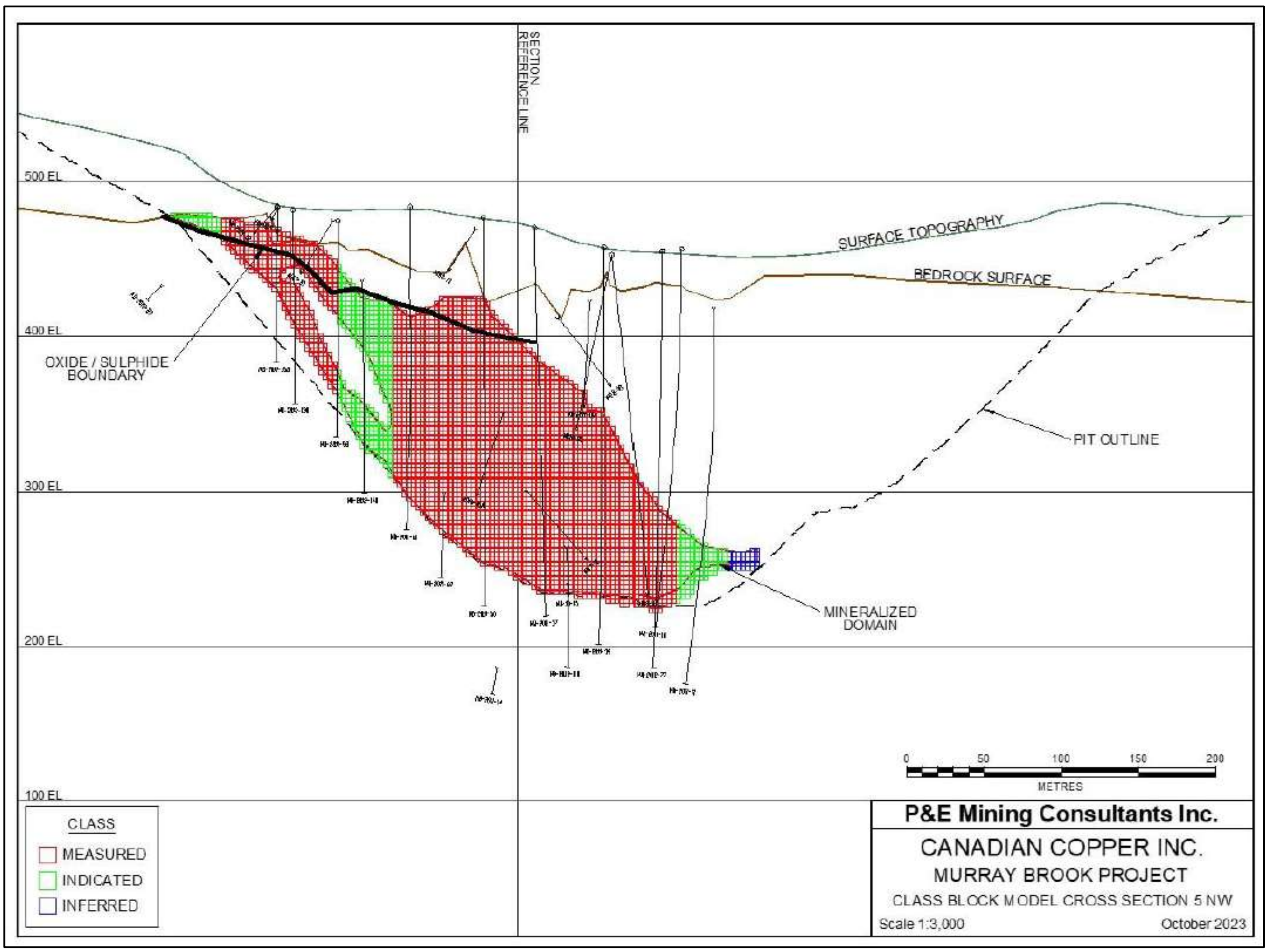


Figure 14-24: Classification Block Model Cross-Section – 8 NW

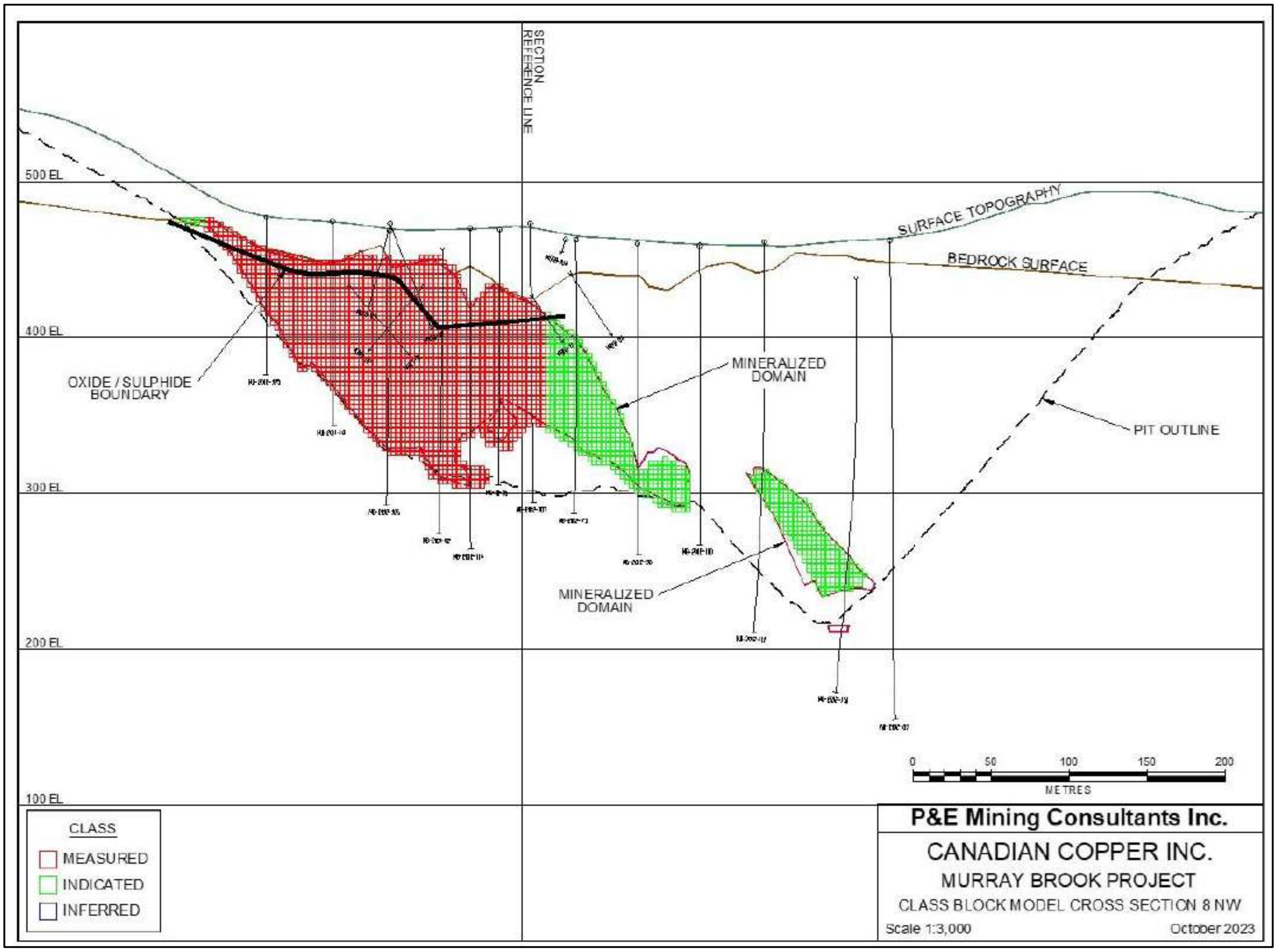


Figure 14-25: Classification Block Model Cross-Section – 11 NW

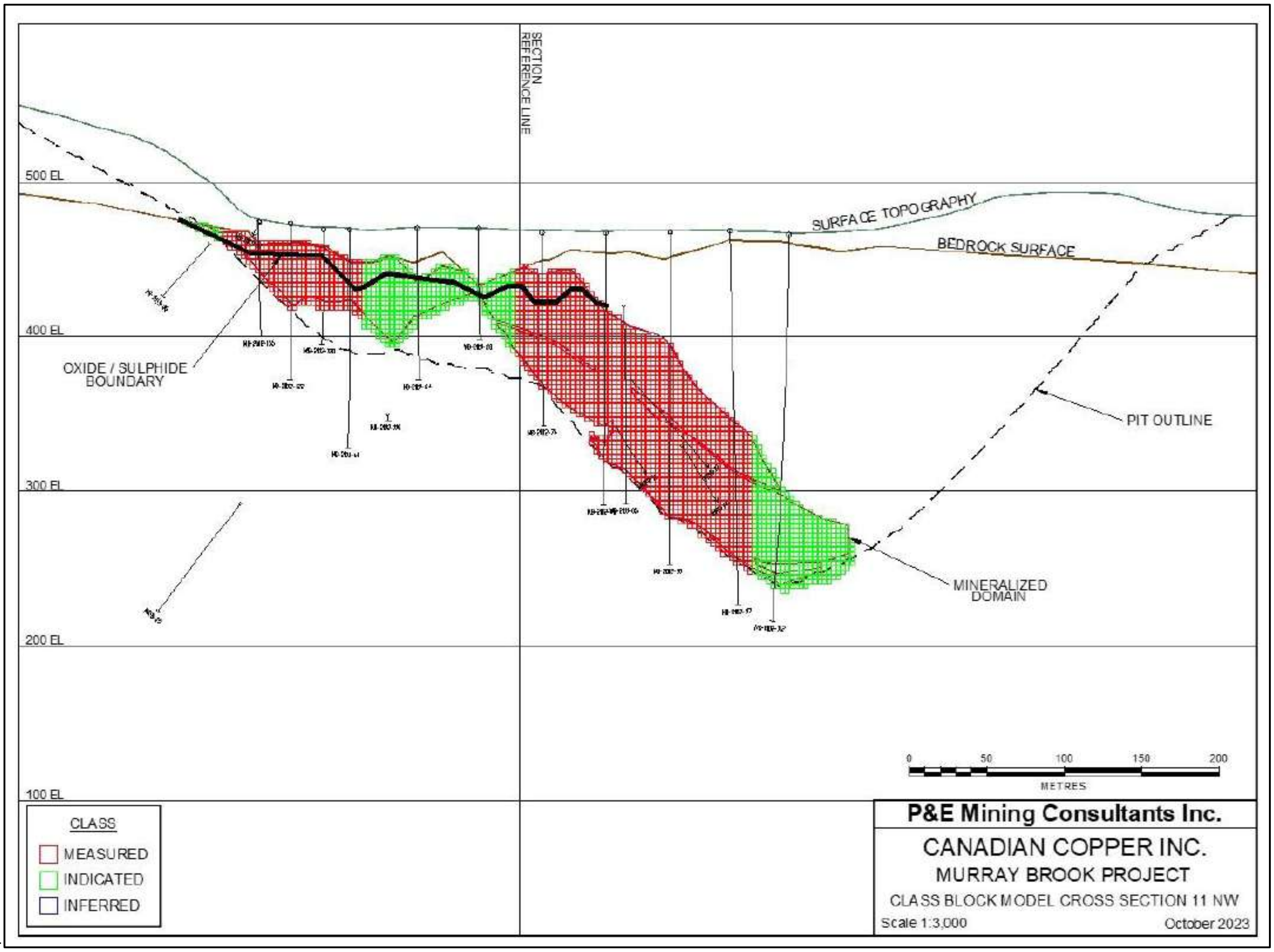


Figure 14-26: Classification Block Model Plan – 400 El

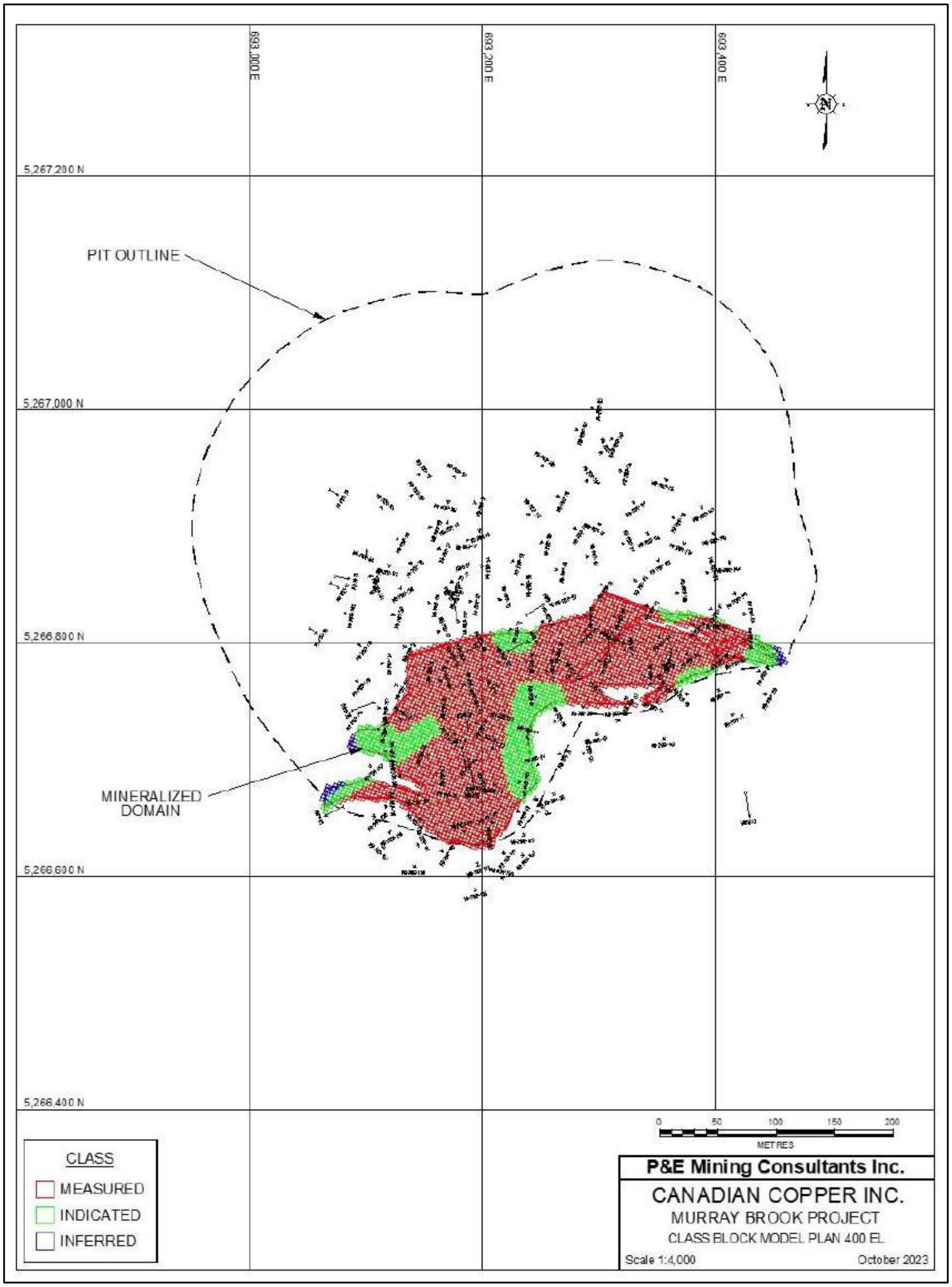


Figure 14-27: Classification Block Model Plan – 350 El

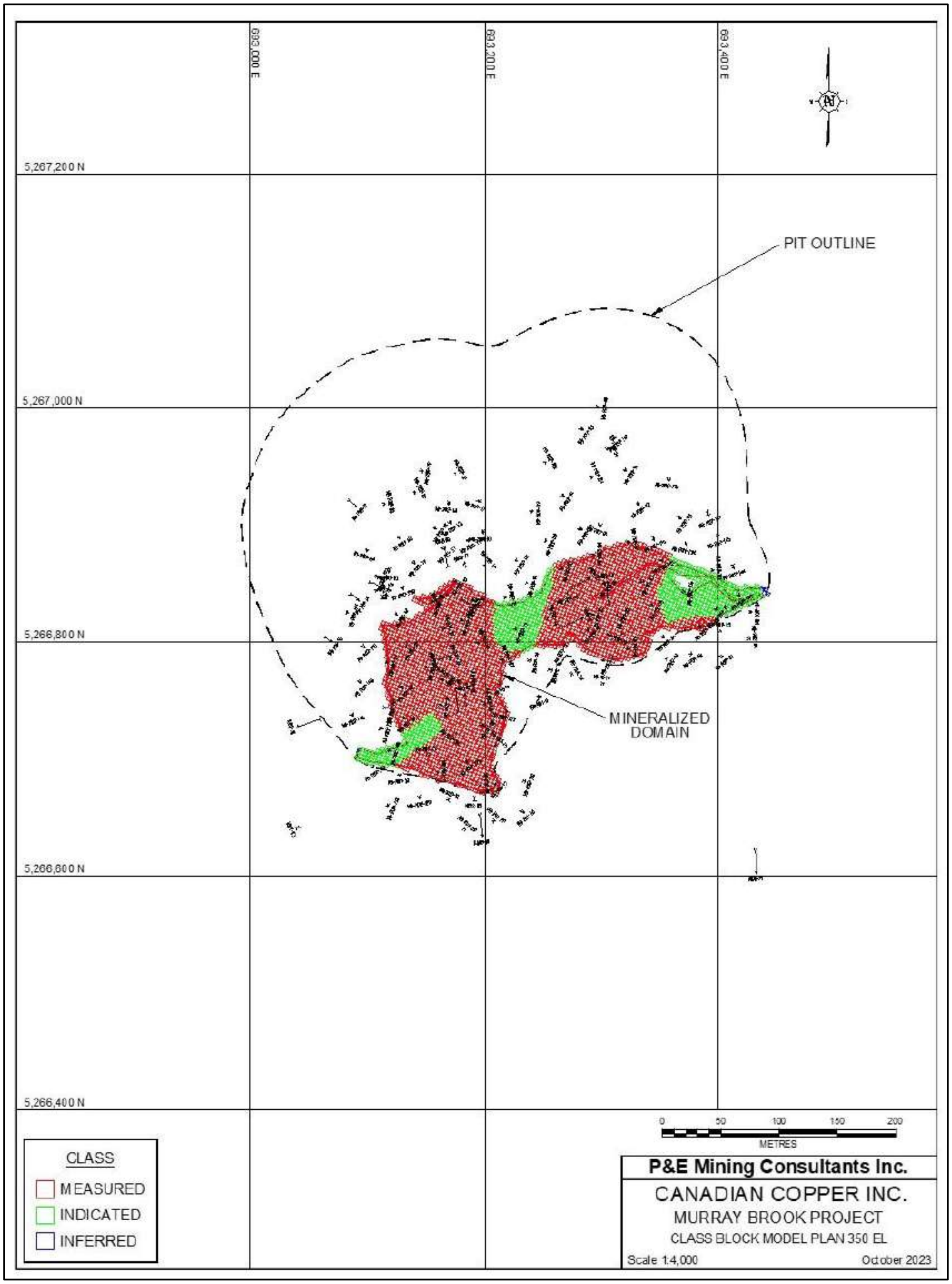
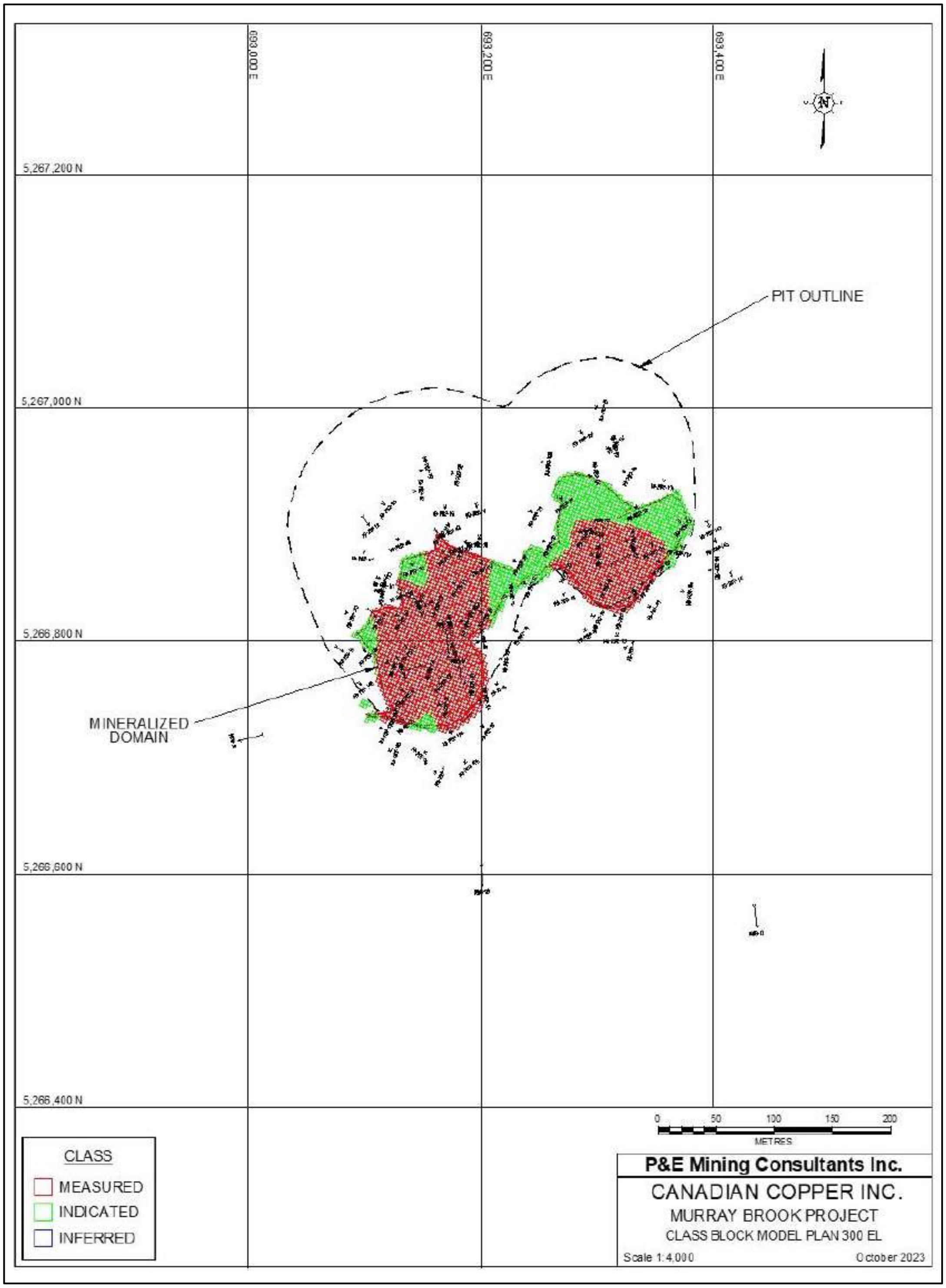


Figure 14-28: Classification Block Model Plan – 300 El



14.14 NSR Cut-off Calculation

The Murray Brook mineral resource estimate was derived from applying NSR cut-off to the block models and reporting the resulting tonnes and grades for potentially mineable areas. The following parameters were used to calculate the NSR cut-off that determines the open pit mining potentially economic portions of the constrained mineralization.

14.15 Open Pit NSR Cut-off Grade Calculation

The NSR cut-off value was based on July 2023 approximate consensus economics forecast US$ metal prices of:

Copper Price .... US$4.00/lb

Lead Price .... US$0.95/lb

Zinc Price .... US$1.25/lb

Silver Price .... US$23.00/oz

Gold Price .... US$1,850/oz

CAD/USD Exchange Rate $0.76

Copper Concentrate Recovery 80%

Zinc Concentrate Recovery 87%

Lead Concentrate Recovery 75%

Silver Concentrate Recovery 90%

Gold Concentrate Recovery 0%

Copper Smelter Payable 96.5%

Lead Smelter Payable 95%

Zinc Smelter Payable 90%

Silver Smelter Payable 90%

Gold Smelter Payable 0%

Transport/Storage/Loading .... US$45/t per WMT

Zinc Smelter Treatment Charge .... US$250/t per DMT

Copper Smelter Treatment Charge .... US$85/t per DMT

Lead Smelter Treatment Charge .... US$150/t per DMT

Humidity Factor 8.0%

Processing Cost .... C$20.00/t processed

General & Administration .... C$3.00/t processed

In the anticipated open pit operation, processing and G&A costs combine for a total of (C$20 + C$3) = C$23/t processed which becomes the NSR cut-off value.

NSR value was calculated based on the above parameters. The formula is as follows:

NSR $/t = (Cu % x 81) + (Pb % x 12) + (Zn % x 13) + (Ag g/t x 0.90).

14.16 Pit Optimization Parameters

In order for the constrained open pit mineralization in the Murray Brook mineral resource model to be considered potentially economic, a first pass Whittle 4X pit optimization was carried out to create a pit shell for resource reporting purposes (see Figure 14-29) utilizing the criteria below:

Overburden mining cost per tonne .... C$2.00

Waste mining cost per tonne .... C$2.25

Mineralized mining cost per tonne .... C$2.50

Process cost per tonne .... C$20.00

General and administration cost per tonne .... C$3.00

Process production rate (tonnes per year) 2,000,000

Pit rock slopes (overall wall angle) .... 50°

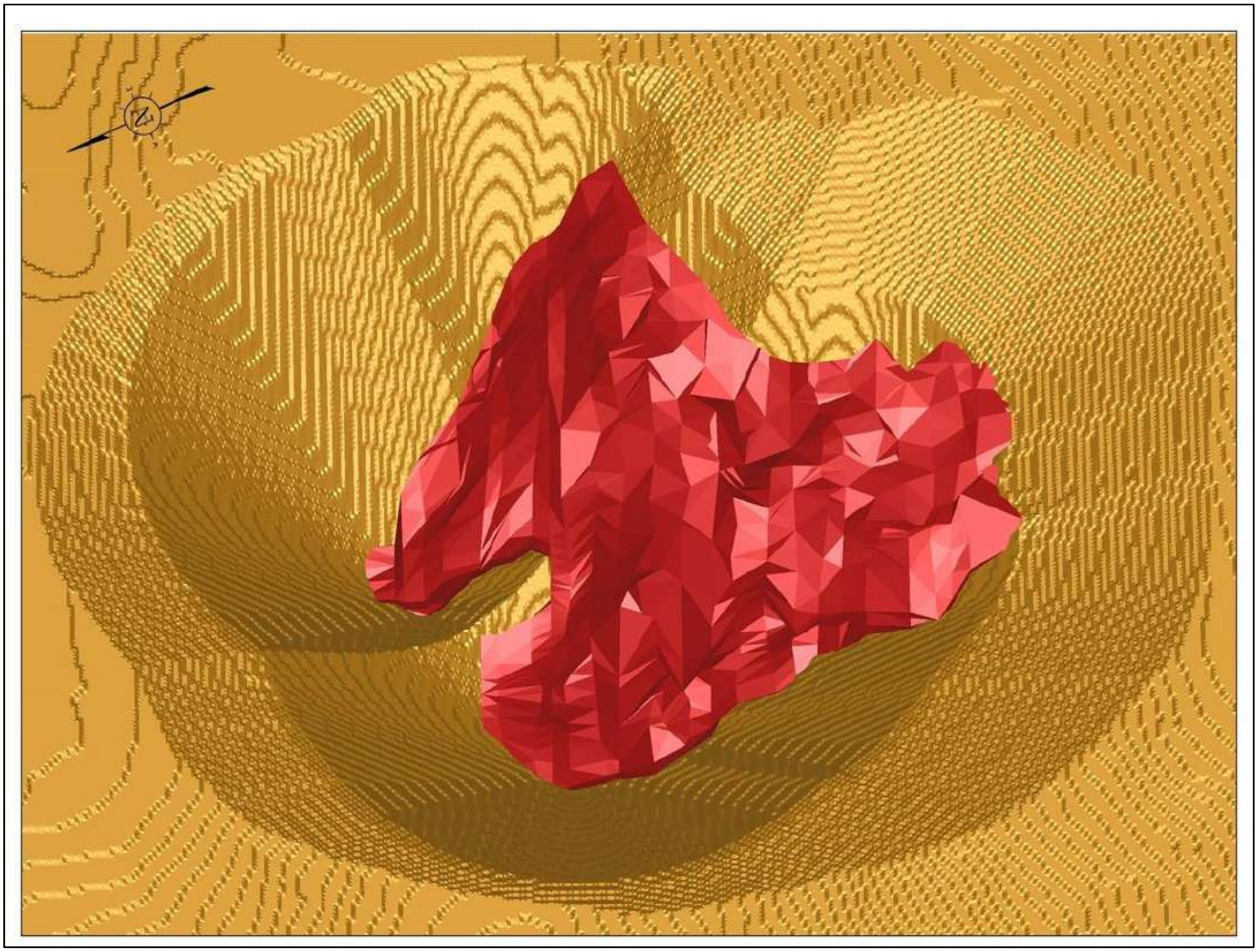
Pit overburden slopes .... 30°

Average mineralized rock bulk density .... 4.26/m^3

Waste rock bulk density .... 2.70 t/m^3

Overburden bulk density .... 1.80 t/m^3

Figure 14-29: Whittle 4X Optimization Pit Shell



14.17 Mineral Resource Estimate

The resulting mineral resource estimate as of the effective date of this technical report is tabulated in Table 14-12. The mineralization of the Murray Brook Project is considered to be potentially amenable to open pit economic extraction. The mineral resource estimates are sensitive to the selection of a reporting NSR cut-off value, as demonstrated in Table 14-13.

Table 14-12: Murray Brook In-Pit Mineral Resource Estimate on at CD$23/t NSR Cut-off (1 to 7)

Zone	Classification	Tonnes (k)	Cu (%)	Cu (Mlb)	Zn (%)	Zn (Mlb)	Pb (%)	Pb (Mlb)	Au (g/t)	Au (koz)	Ag (g/t)	Ag (Moz)	ZnEq (%)	CuEq (%)	NSR (C$/t)
Oxide	Measured (M)	1,641	1.05	37.9	2.2	79.6	0.73	26.6	0.36	19	38	2	5.94	1.85	156
	Indicated (I)	373	0.97	7.9	2.31	19	0.78	6.4	0.51	6	44.7	0.5	6.02	1.88	158
	M + I	2,014	1.03	45.9	2.22	98.6	0.74	32.9	0.39	25	39.2	2.5	5.95	1.86	157
Sulphide	Measured	15,830	0.43	150.8	2.6	908.3	0.92	322.2	0.52	264	39	19.8	4.83	1.51	115
	Indicated	5,275	0.52	60.9	2.14	248.9	0.85	98.9	0.67	114	37.3	6.3	4.58	1.43	114
	M + I	21,105	0.45	211.7	2.49	1,157.2	0.91	421.1	0.56	378	38.6	26.2	4.77	1.49	115
	Inferred	110	0.41	1	1.82	4.4	0.68	1.6	0.62	2	30.4	0.1	3.75	1.17	92
Total	Measured	17,471	0.49	188.7	2.56	987.9	0.91	348.8	0.5	283	38.9	21.8	4.93	1.54	119
	Indicated	5,648	0.55	68.9	2.15	267.8	0.85	105.3	0.66	120	37.8	6.9	4.68	1.46	117
	M + I	23,119	0.51	257.5	2.46	1,255.70	0.89	454.1	0.54	403	38.6	28.7	4.87	1.52	118
	Inferred	110	0.41	1	1.82	4.4	0.68	1.6	0.62	2	30.4	0.1	3.75	1.17	92

Notes: **1.** Mineral resources, which are not mineral reserves, do not have demonstrated economic viability. **2.** The estimate of mineral resources may be materially affected by environmental, permitting, legal, title, taxation, socio-political, marketing, or other relevant issues. **3.** The inferred mineral resource in this estimate has a lower level of confidence than that applied to an indicated mineral resource and must not be converted to a mineral reserve. It is reasonably expected that the majority of the inferred mineral resource could be upgraded to an indicated mineral resource with continued exploration. **4.** The mineral resources in this report were estimated using the Canadian Institute of Mining, Metallurgy and Petroleum (CIM), CIM Standards on Mineral Resources and Reserves, Definitions and Guidelines (2019) prepared by the CIM Standing Committee on Reserve Definitions and adopted by the CIM Council. **5.** Totals of tonnage and contained metal may differ due to rounding. **6.** NSR $/t = (Cu % x 81) + (Pb % x 12) + (Zn % x 13) + (Ag g/t x 0.90). **7.** CuEq = Cu%/0.30; and ZnEq = Zn%/0.52.

Table 14-13: In-Pit Mineral Resource Estimate Sensitivity

Zone	Classification	Cut-off NSR (C$/t)	Tonnes (k)	Cu (%)	Cu (Mlb)	Zn (%)	Zn (Mlb)	Pb (%)	Pb (Mlb)	Au (g/t)	Au (koz)	Ag (g/t)	Ag (Moz)
Oxide	Measured	100	1,115	1.31	32.3	2.81	69.1	0.94	23.1	0.38	14	46.9	1.7
		50	1,540	1.10	37.4	2.31	78.5	0.77	26.2	0.36	18	39.6	2.0
		45	1,563	1.09	37.5	2.29	78.9	0.76	26.3	0.36	18	39.2	2.0
		40	1,587	1.08	37.7	2.26	79.2	0.76	26.4	0.36	19	38.9	2.0
		35	1,608	1.07	37.8	2.24	79.4	0.75	26.5	0.36	19	38.5	2.0
		30	1,627	1.06	37.8	2.22	79.5	0.74	26.5	0.36	19	38.2	2.0
		25	1,638	1.05	37.9	2.20	79.6	0.74	26.6	0.36	19	38.0	2.0
		23	1,641	1.05	37.9	2.20	79.6	0.73	26.6	0.36	19	38.0	2.0
	Indicated	100	260	1.20	6.8	2.78	15.9	0.93	5.3	0.54	4	56.3	0.5
		50	358	1.00	7.9	2.37	18.7	0.80	6.3	0.52	6	46.1	0.5
		45	362	0.99	7.9	2.36	18.9	0.79	6.3	0.52	6	45.7	0.5
		40	366	0.98	7.9	2.35	18.9	0.79	6.4	0.52	6	45.4	0.5
		35	367	0.98	7.9	2.34	19.0	0.79	6.4	0.51	6	45.3	0.5
		30	369	0.97	7.9	2.33	19.0	0.78	6.4	0.51	6	45.1	0.5
		25	372	0.97	7.9	2.32	19.0	0.78	6.4	0.51	6	44.8	0.5
		23	373	0.97	7.9	2.31	19.0	0.78	6.4	0.51	6	44.7	0.5
Sulphide	Measured	100	7,910	0.56	97.6	3.80	662.3	1.40	243.3	0.68	172	57.2	14.5
		50	13,995	0.46	142.1	2.85	878.1	1.02	313.9	0.56	251	42.6	19.2
		45	14,524	0.45	144.9	2.78	889.3	0.99	317.0	0.55	255	41.5	19.4
		40	14,982	0.45	147.2	2.72	897.5	0.97	319.3	0.54	259	40.6	19.6
		35	15,349	0.44	148.9	2.67	902.9	0.95	320.7	0.53	261	39.9	19.7
		30	15,633	0.44	150.1	2.63	906.3	0.93	321.7	0.52	263	39.4	19.8
		25	15,795	0.43	150.7	2.61	908.0	0.93	322.1	0.52	264	39.0	19.8
		23	15,830	0.43	150.8	2.60	908.3	0.92	322.2	0.52	264	39.0	19.8
	Indicated	100	2,720	0.70	41.9	2.94	176.3	1.22	73.0	0.94	82	53.0	4.6
		50	4,707	0.56	58.3	2.30	239.1	0.92	95.9	0.73	110	40.6	6.1
		45	4,861	0.55	59.1	2.26	242.5	0.90	96.9	0.71	112	39.7	6.2
		40	5,009	0.54	59.9	2.22	245.2	0.89	97.8	0.70	113	38.8	6.3
		35	5,112	0.54	60.3	2.19	246.8	0.87	98.3	0.69	113	38.3	6.3
		30	5,202	0.53	60.7	2.16	248.0	0.86	98.7	0.68	114	37.8	6.3
		25	5,258	0.53	60.9	2.14	248.7	0.85	98.9	0.68	114	37.4	6.3
		23	5,275	0.52	60.9	2.14	248.9	0.85	98.9	0.67	114	37.3	6.3
	Inferred	100	45	0.37	0.4	2.70	2.7	1.08	1.1	0.92	1	45.1	0.1
		50	98	0.42	0.9	1.98	4.3	0.74	1.6	0.67	2	33.1	0.1
		45	101	0.42	0.9	1.94	4.3	0.73	1.6	0.65	2	32.5	0.1
		40	103	0.42	0.9	1.92	4.4	0.72	1.6	0.65	2	32.0	0.1
		35	105	0.41	1.0	1.89	4.4	0.71	1.6	0.64	2	31.6	0.1
		30	107	0.41	1.0	1.86	4.4	0.69	1.6	0.63	2	31.0	0.1
		25	110	0.41	1.0	1.82	4.4	0.68	1.6	0.62	2	30.4	0.1
		23	110	0.41	1.0	1.82	4.4	0.68	1.6	0.62	2	30.4	0.1

Note: Highlighted rows are the base case. All sensitivity NSR cut-off values are greater than the base case, and are therefore reasonable prospects for eventual economic extraction (RPEEE).

14.18 Confirmation of Estimate

The block model was validated using a number of industry standard methods, including visual and statistical methods.

Visual examination of composites and block grades on successive plans and vertical cross-sections were performed on-screen to confirm that the block models correctly reflect the distribution of composite grades. The review of estimation parameters included:

- number of composites used for estimation
- number of drill holes used for estimation
- number of passes used to estimate grade
- mean value of the composites used
- mean distance to sample used
- actual distance to closest point
- grade of true closest point.

A comparison of mean grades of composites with the block model is presented in Table 14-14.

Table 14-14: Average Grade Comparison of Composites with Block Model

Data Type	Cu (%)	Zn (%)	Pb (%)	Ag (g/t)	Au (g/t)
Composites	0.48	2.65	0.96	40.84	0.57
Capped Composites	0.48	2.65	0.96	40.80	0.57
Block Model ID	0.52	2.32	0.85	37.34	0.54
Block Model NN	0.52	2.28	0.83	36.89	0.54

Notes: ID = copper, zinc and lead block model grades were interpolated with ID^2, whereas silver and gold grades were interpolated with ID^3. NN= block model grades were interpolated using nearest neighbour.

The average grades of block models were different from that of composites used for the grade estimations. These were most likely due to smoothing by the grade interpolation process. The block model values will be more representative than the composites, due to 3-D spatial distribution characteristics of the block models.

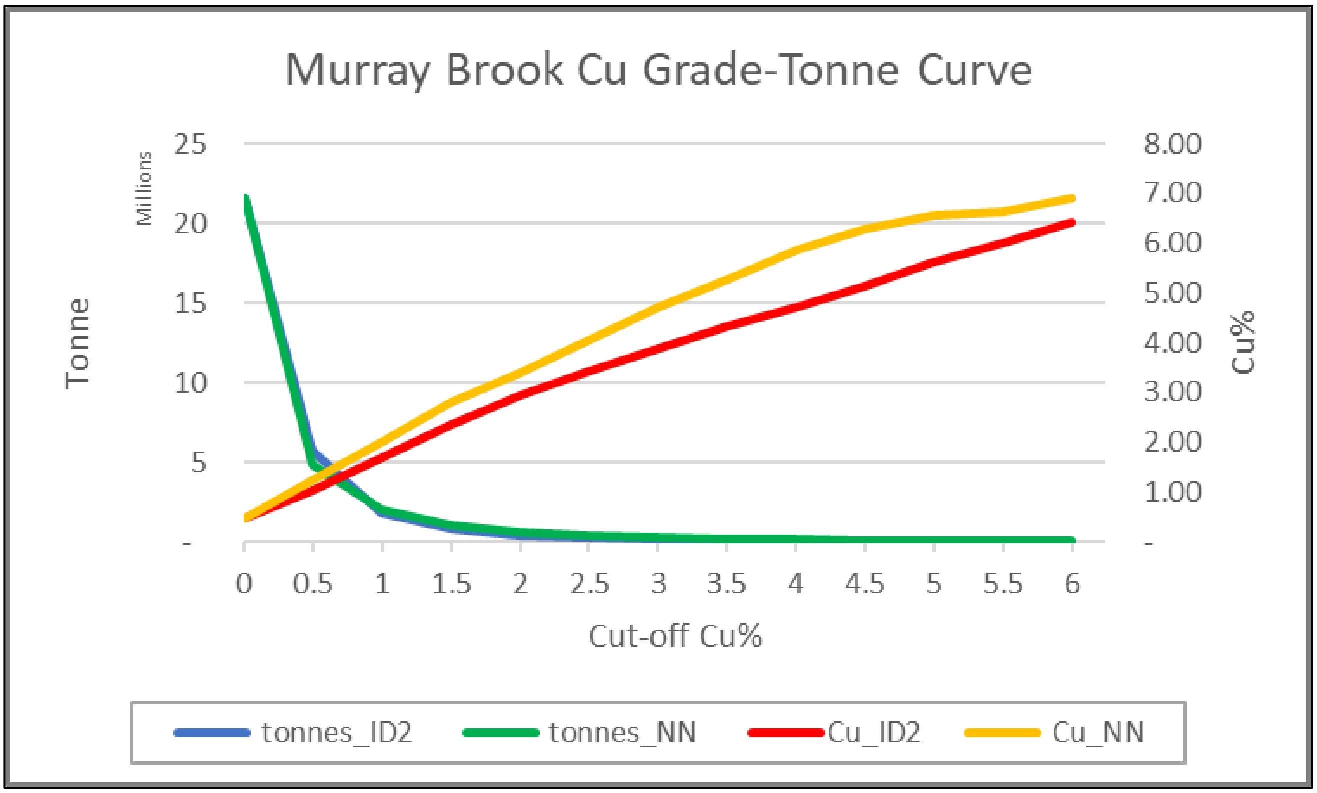
A volumetric comparison was performed with the block model volume versus the geometric calculated volume of the domain solids and the differences are shown in Table 14-15.

Table 14-15: Volume Comparison of Block Model with Geometric Solid

Description	Volume
Geometric Volume of Wireframe	5,537,668 m^3
Block Model Volume	5,537,683 m^3
Difference	0.0%

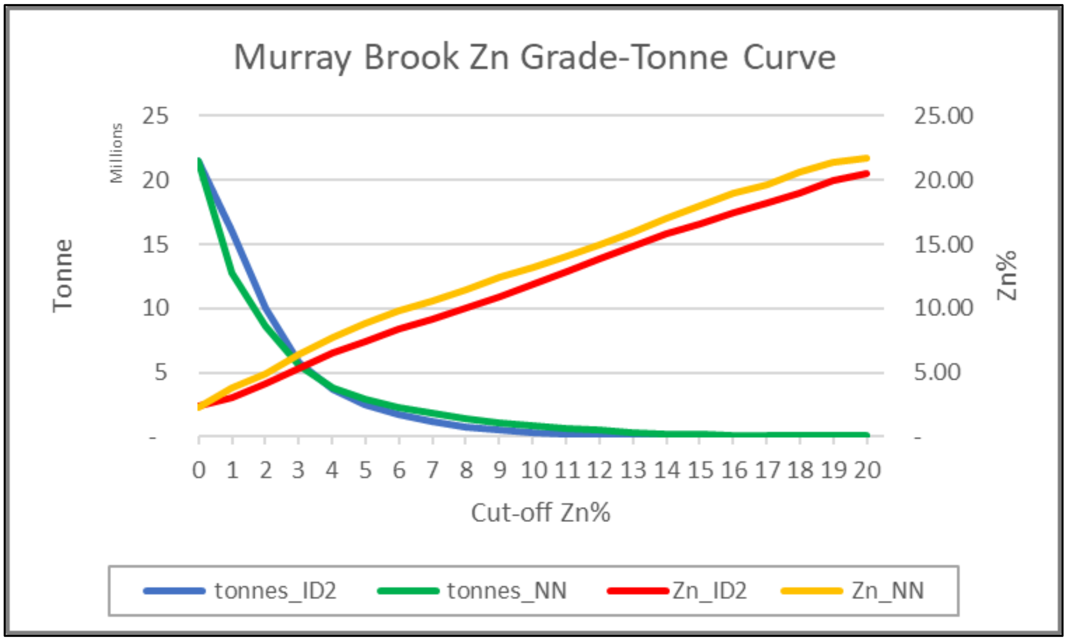
A comparison of the grade-tonnage curve of the grade model interpolated with ID^2 for copper and zinc and ID^3 for silver, and NN on a global mineral resource basis are presented in Figures 14-30 to 14-32.

Figure 14-30: Cu Grade-Tonnage Curve for ID^2 and NN Interpolation



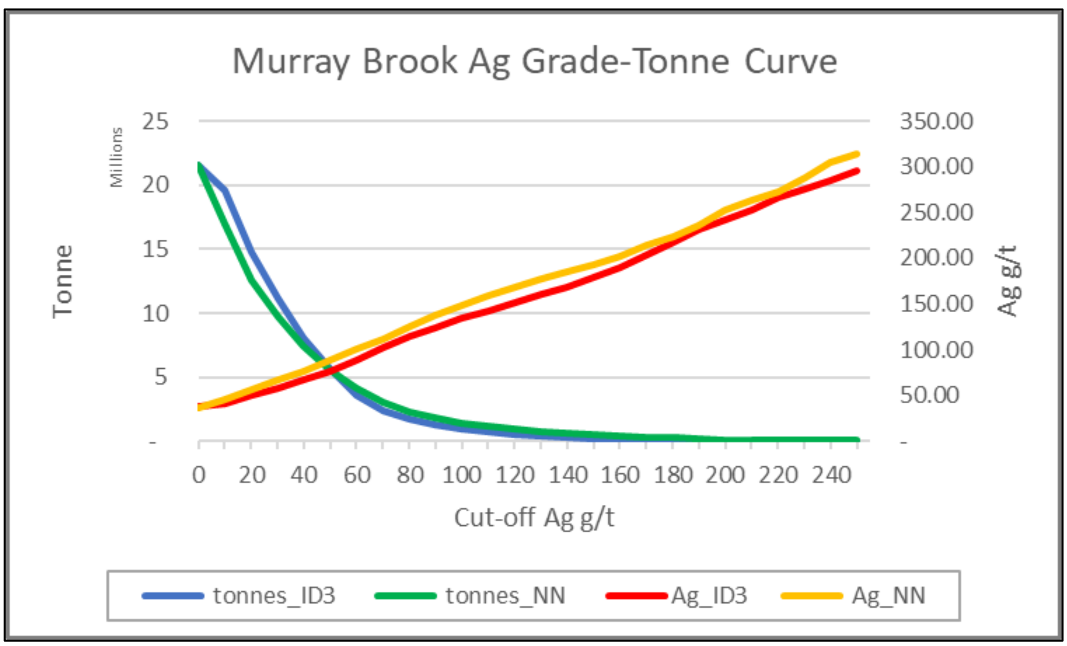
Source: P&E (2023).

Figure 14-31: Zn Grade-Tonnage Curve for ID² and NN Interpolation



Source: P&E (2023).

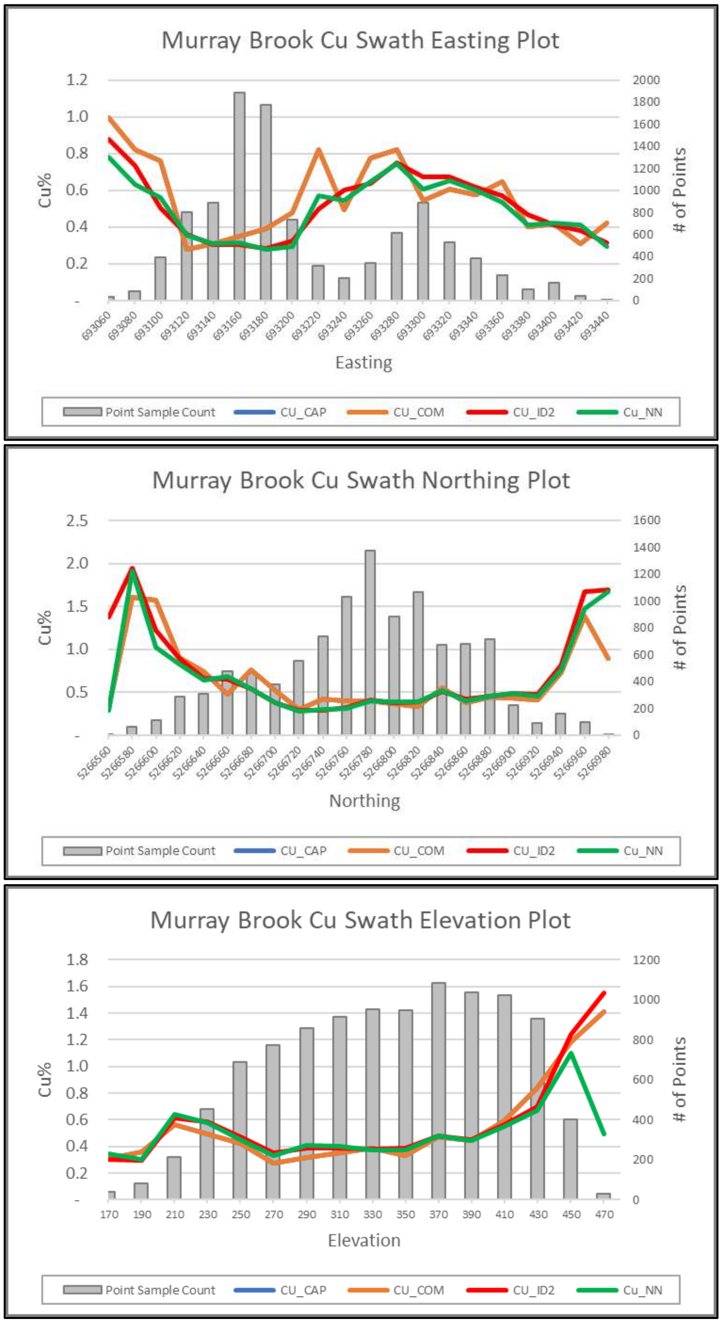
Figure 14-32: Ag Grade-Tonnage Curve for ID³ and NN Interpolation



Source: P&E (2023).

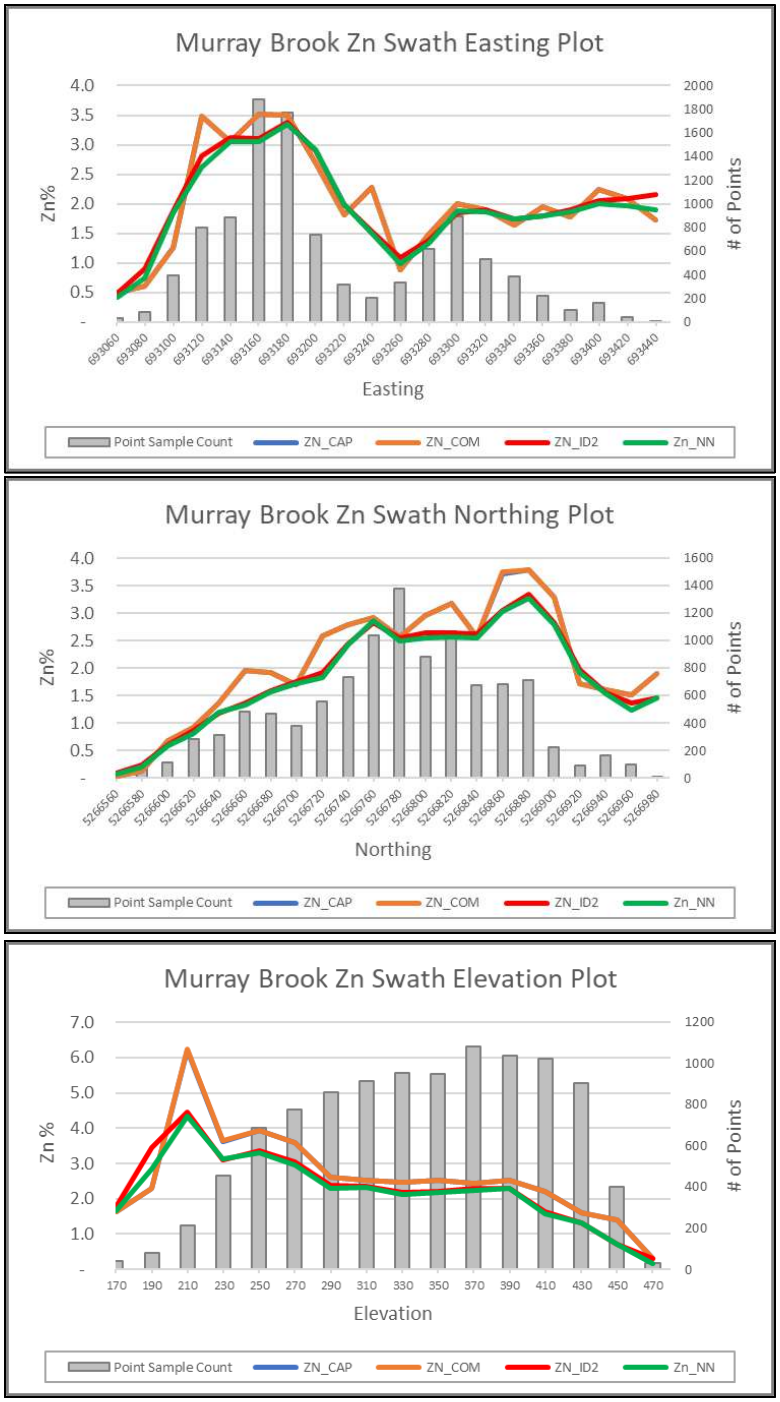
Copper, zinc, and silver local trends were evaluated by comparing the inverse distance and NN estimate against the composites. As shown in Figures 14-33 to 14-35, the grade interpolations with ID and NN agreed well.

Figure 14-33: Cu Grade Swath Plots



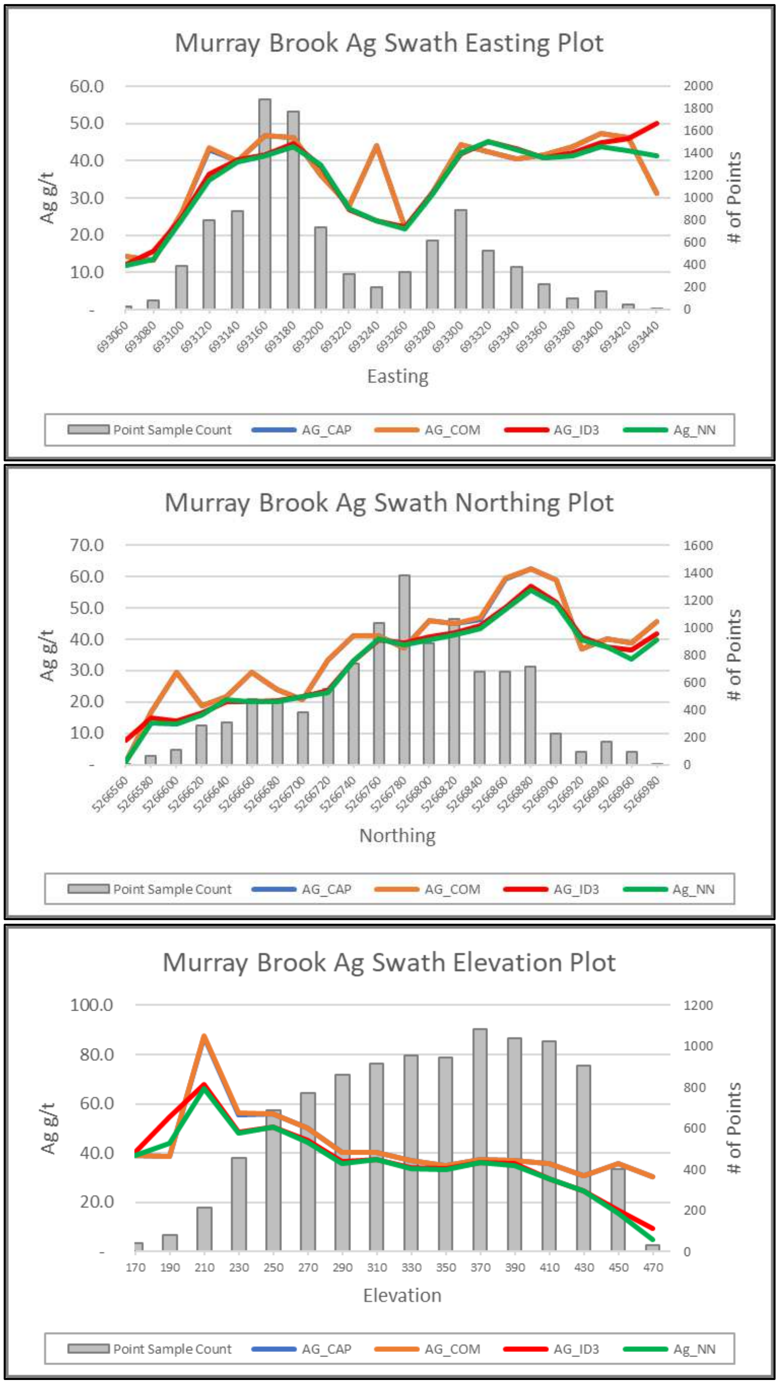
Source: P&E (2023).

Figure 14-34: Zn Grade Swath Plots



Source: P&E (2023).

Figure 14-35: Ag Grade Swath Plots



Source: P&E (2023).

15 MINERAL RESERVE ESTIMATES

This section is not applicable to a preliminary economic assessment.

16 MINING METHODS

16.1 Introduction

The Murray Brook deposit is relatively shallow in depth and lends itself to conventional open pit mining methods. A single open pit will be developed that will have a maximum depth of approximately 310 metres from the highest point (540 masl) to lowest point (230 masl).

A project mine production plan has been developed and used in the financial analysis. This production plan does not utilize inferred mineral resources.

The open pit will require the excavation of four different materials:

1. Overburden (placed into an overburden stockpile)
2. Non-acid generating (NAG) waste rock (placed into a NAG waste rock storage facility)
3. Potentially-acid generating (PAG) waste rock (placed into a separate PAG waste rock storage facility)
4. Sulphide process feed (processed through the Caribou process plant).

Developing the mine production schedule involved several steps in sequence:

1. Perform pit optimizations to select the optimal ultimate pit shell
2. Design an operational pit (with ramps and benches) based on the selected optimal ultimate shell in the previous step
3. Design intermediate pit phases (pushbacks) to enhance the annual production schedule
4. Develop a life-of-mine production schedule.

16.2 Pit Optimization

Pit optimization was completed using Geovia Whittle™ software. The pit optimization analysis produces a series of nested pit shells, each containing mineralized material that is potentially economically mineable according to a given geological block model and a set of geotechnical and economic inputs. An optimal pit shell is then selected as the basis for the operational final pit design.

Pit optimization was completed using the parameters shown in Table 16-1. The geological block model contains information regarding the grades of copper, lead, zinc, silver, and gold. The NSR formula, described in Section 14, was based on the revenues of copper, lead, zinc, and silver while no revenue was attributed to gold since gold was considered unrecoverable/unpayable in the current PEA. Metals from oxide material cannot be economically recovered in the Caribou process plant so the material is planned to be placed on the PAG waste rock storage facility.

Metal prices that have been used in the NSR formula (Table 16-1) are based on two-year trailing averages (January 31, 2025) and consensus economics forecasts.

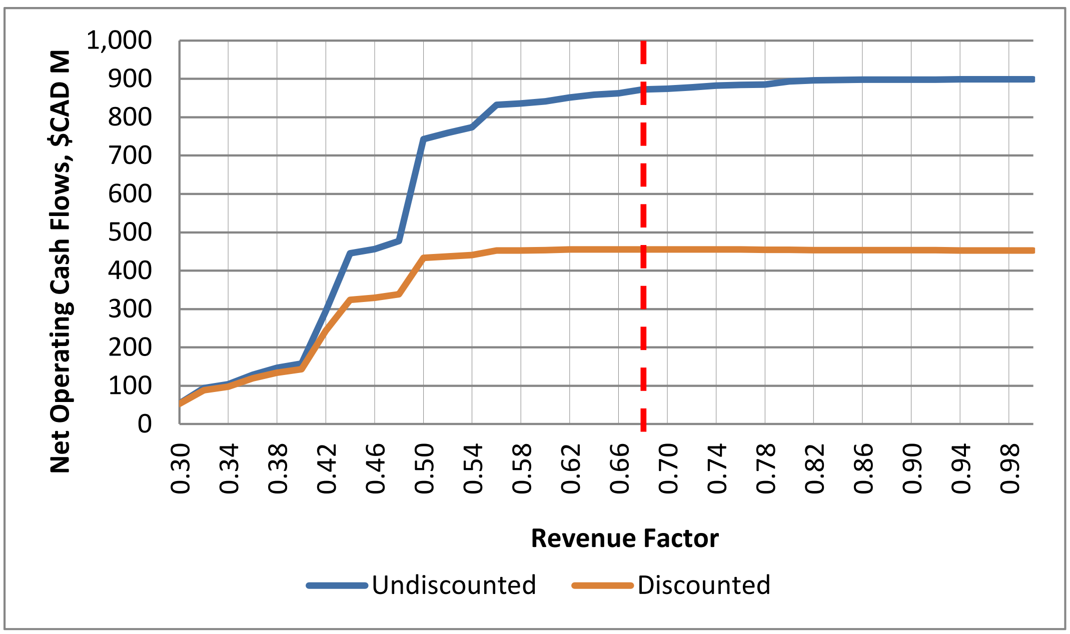
Table 16-1: Murray Brook NSR and Pit Optimization Parameters

Parameter	Unit	Value
Copper Price	US$/lb	4.25
Lead Price	US$/lb	1.00
Zinc Price	US$/lb	1.30
Silver Price	US$/oz	25.00
Exchange Rate	USD/USD	0.75
Copper Process Recovery	%	83
Lead Process Recovery	%	72
Zinc Process Recovery	%	91
Silver Process Recovery	%	80
Copper Concentrate Mass Pull	%	1.5
Lead Concentrate Mass Pull	%	1.1
Zinc Concentrate Mass Pull	%	4.5
Copper Concentrate Grade	%	22
Lead Concentrate Grade	%	45
Zinc Concentrate Grade	%	48
Copper, Lead, Zinc Concentrate Transport	US$/wmt	190
Concentrate Moisture	%	8
Copper Smelter Charges	US$/dmt	80
Lead Smelter Charges	US$/ dmt	100
Zinc Smelter Charges	US$/ dmt	165
Copper Refining Charges	US$/lb	0.08
Silver Refining Charges	US$/oz	1.00
Overburden Mining Cost	$/t mined	2.00
Waste Rock Mining Cost	$/t mined	3.00
Oxide Waste Mining Cost	$/t mined	3.00
Sulphide Mining Cost	$/t mined	3.00
Processing Cost	$/t processed	24.00
G&A Cost	$/t processed	4.50
Transport from Pit to Caribou Plant	$/t processed	5.67
NSR Cut-off Value	$/t processed	34.17
Pit Slopes for Optimization (Overburden/Rock)	degrees	30°/45°

A series of optimized pit shells were generated by changing the revenue factor (RF) between 0.3 and 1.0 at 0.02 intervals. A RF of 1.0 corresponds to the base case metal prices listed in Table 16-1. The optimization results are shown graphically in Figures 16-1 and 16-2. The net cash flow numbers in Figure 16-1 are indicative merit measures used to compare different optimized pit shells. The numbers are based on revenues and operating costs that are calculated using the pit optimization parameters. These measures do not incorporate the detailed operating cost estimations reported in Section 21, capital costs (initial or sustaining), taxes, or closure costs, and therefore are not directly comparable to the actual project economic merit measures (Section 22). These results provide an estimate for the potentially economic portion of the sulphide mineral resource process feed for each revenue factor as well as potential strip ratio.

Figure 16-1 shows that the discounted value of net operating cash flows peaks at an RF of 0.68. Therefore, the optimized pit shell corresponding to the 0.68 RF was selected as the optimal ultimate pit shell to form the basis for further detailed final pit design.

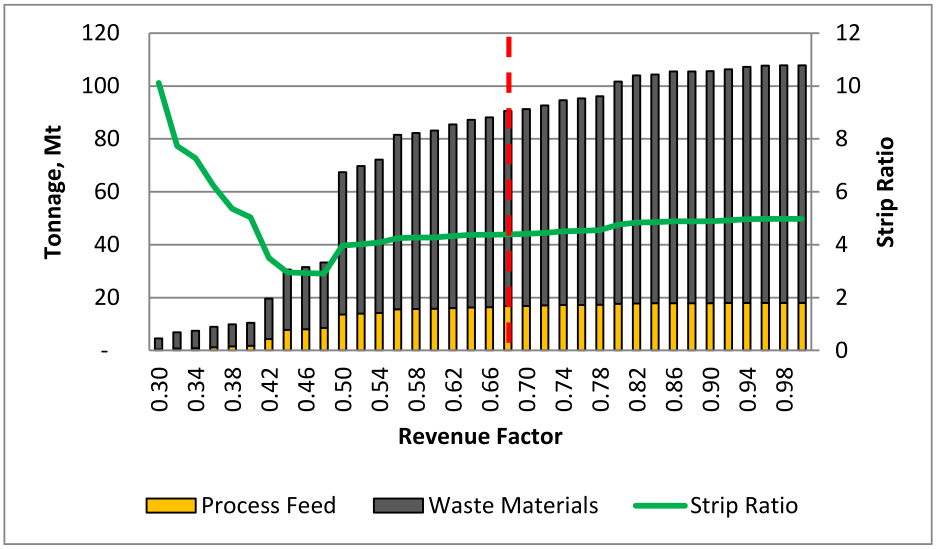
Figure 16-1: Pit Optimization Net Operating Cash Flows



Source: P&E, (2025).

The tonnages shown in Figure 16-2 represent the potentially economic portion of the mineral resource contained in the optimized pit shell; however, the tonnages that will be reported in the production schedule will be derived from the operational final pit design.

Figure 16-2: Pit Optimization Tonnages and Strip Ratio



Source: P&E, (2025).

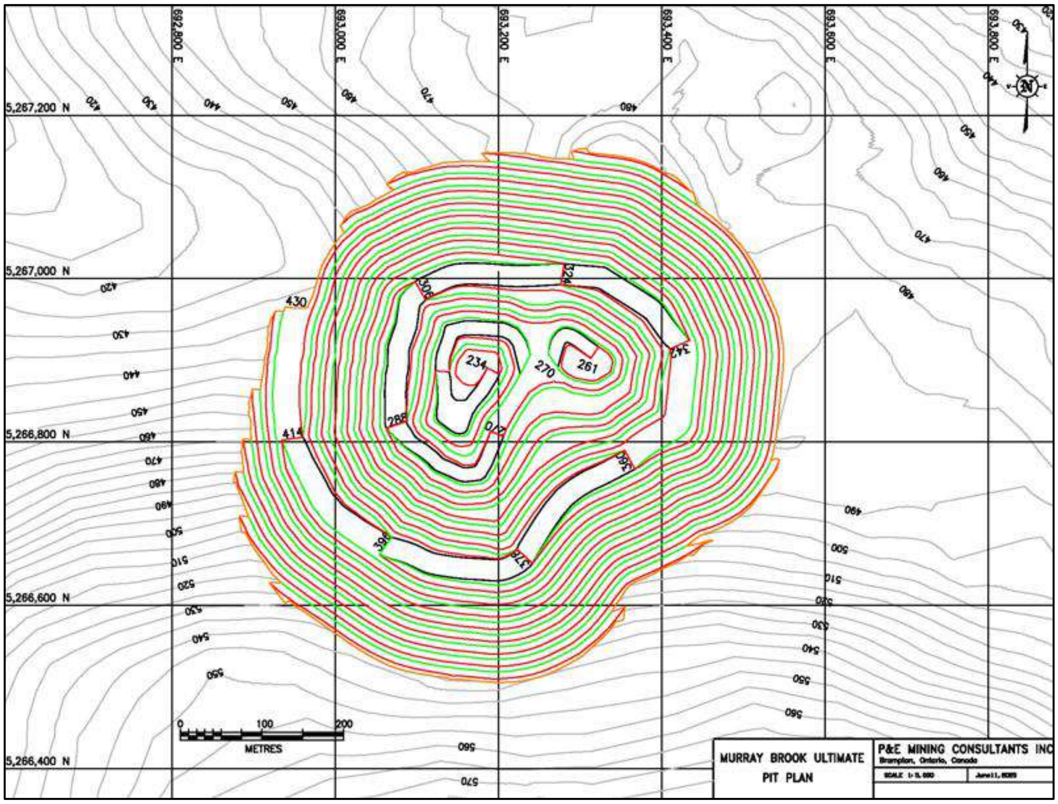
16.3 Pit Designs

An operational pit design was created using the selected optimized pit shell corresponding to RF of 0.68. Benches and haul roads were added according to the parameters in Table 16-2. Figure 16-3 presents a plan view of the final pit design.

Table 16-2: Pit Design Parameters

Item	Unit	Value
Haul Road Width (Double Lane)	m	26
Haul Road Width (Single Lane)	m	17
Haul Road Grade (Maximum)	%	10
Overburden Slope		
Bench Height	m	18
Bench Face Angle	degrees	40
Catch-Bench Width	m	9.7
Inter-Ramp Angle	degrees	30
Rock Slope		
Bench Height (triple bench)	m	18
Bench Face Angle	degrees	70
Catch-Bench Width	m	8.6
Inter-Ramp Angle	degrees	50

Figure 16-3: Final Pit Design



Source: P&E, (2025).

16.4 Geotechnical Studies

No geotechnical studies have been completed for this preliminary economic assessment; therefore, pit slope angles listed in Table 16-2 were proposed based on the QP's experience with similar rock types. Golder (2018a) prepared a technical memorandum on preliminary guidance on a reasonable range of Whittle angles for an open pit shell, and the recommendations were taken into consideration by the QP.

16.5 Hydrogeological Studies

No hydrogeological studies have been completed on the Murray Brook property to assess groundwater conditions. Golder (2018b) prepared a technical memorandum on high-level investigation of inflow rate estimates based on the Restigouche property that is 10 km away from the Murray Brook property, and the recommendations were taken into consideration by the QP.

16.6 Mining Dilution and Mining Losses

Dilution and losses will occur during mining. It is assumed that some waste rock and low-grade mineralized material surrounding the mineralized zones will be mixed with the planned mineralized material during mining, thereby causing dilution.

To address dilution, the mineral resource block model was regularized from a volume percent model to a whole block selective mining unit (SMU) model. Regularizing the model resulted in waste and mineralized portions of a block being combined into a single block value. In some cases, the resulting block value may be below the applied cut-off value, in which case that block would be considered a loss. In other cases, while the block grade may decrease due to waste dilution, the entire block remains above the cut-off value, resulting in an increase in mineralized tonnage but at a lower grade. Hence, the SMU model is considered a diluted model and was used for open pit production scheduling.

The amount of dilution incorporated varies depending on block size selectivity and vein width. Table 16-3 compares the undiluted and diluted tonnes and grade and summarizes the expected net dilution. The net dilution incorporates both the total dilution and mining losses. Based on the experience with similar mining operations and rock types, mining losses were assumed at 5%.

Table 16-3: Net Dilution

Category	Mineralization (Mt)	Copper (%)	Lead (%)	Zinc (%)	Silver (g/t)
Undiluted[1]	15.6	0.48	1.00	2.76	42.4
Diluted[2]	15.5	0.47	0.96	2.67	41.1
% Dilution	0.0	2.3	3.4	3.2	3.1

Notes: [1]Cut-off value is $34.2/t NSR. [2]Combined dilution and mining loss. Tonnage is 16.3 Mt before mining losses.

16.7 Potentially Mineable Portion of the Mineral Resource

Table 16-4 summaries the tonnages and grades of the potentially mineable portion of the mineral resource as well as the waste tonnages within the designed final pit, incorporating mining dilution. These tonnages are used as the basis for the production schedule.

The total open pit process plant feed of 15.5 Mt consists of 79.1% measured mineral resource and 20.9% indicated mineral resource. Table 16-5 summarizes the breakdown of the mineral resource classification. There is no inferred mineral resource contained within the open pit design.

Table 16-4: Potentially Mineable Portion of the Resource (Diluted)

Item	Unit	Value
Total Material in Pit	Mt	93.2
Overburden	Mt	5.9
NAG Waste Rock	Mt	60.1
PAG Waste Rock	Mt	11.8
Total Waste Material	Mt	77.7
Strip Ratio	w:o	5.0
Sulphide Feed (Diluted)	Mt	15.5
NSR	$/t	103.47
Gold	g/t	0.55
Silver	g/t	41.1
Copper	%	0.47
Lead	%	0.96
Zinc	%	2.67

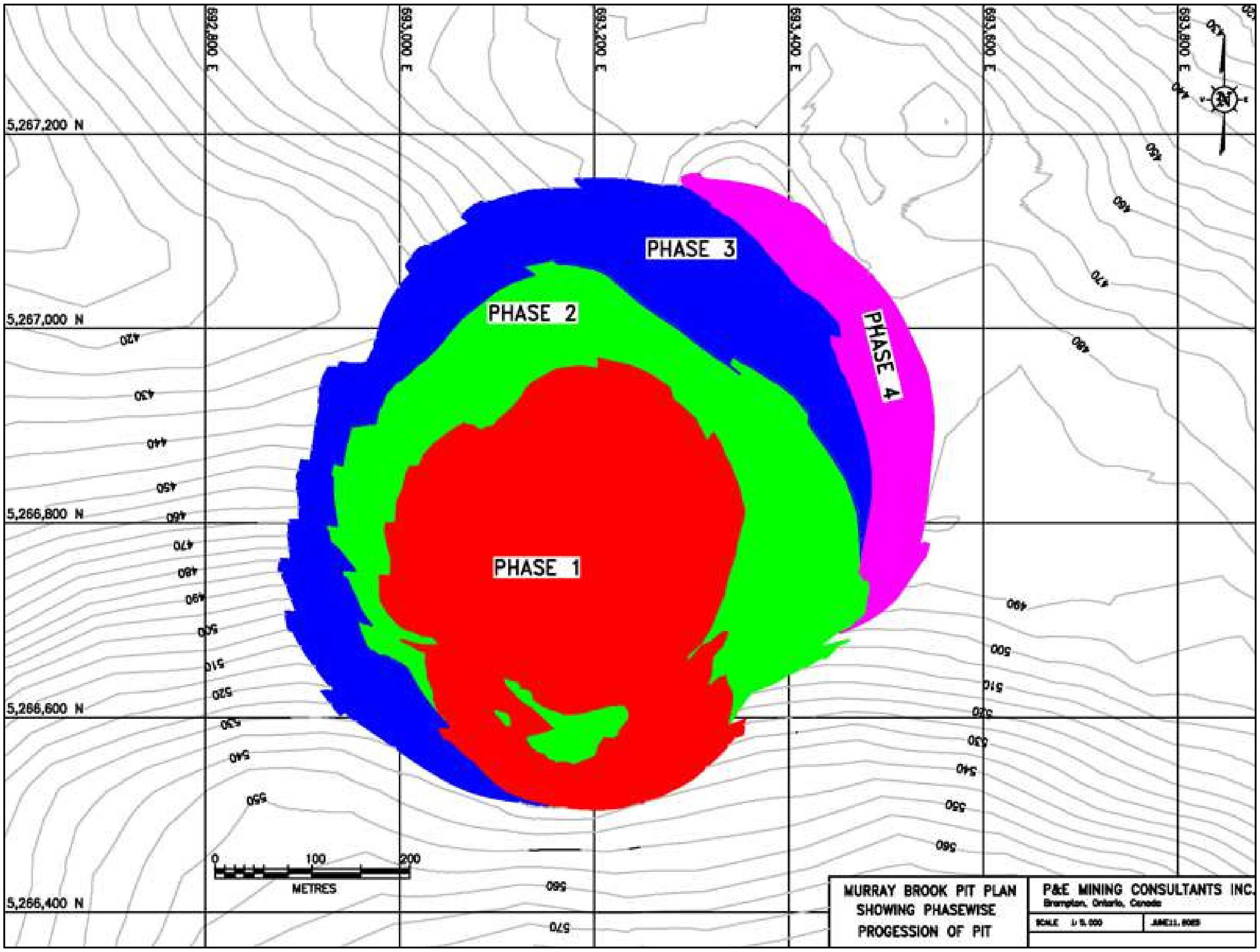
Table 16-5: Open Pit Feed Classification

Mineral Resource	Feed (Mt)	Copper (%)	Lead (%)	Zinc (%)	Silver (g/t)
Measured	12.24	0.46	1.01	2.83	42.4
Indicated	3.24	0.51	0.79	2.07	35.9
Total Measured & Indicated	**15.48**	**0.47**	**0.96**	**2.67**	**41.0**

16.8 Pit Phases

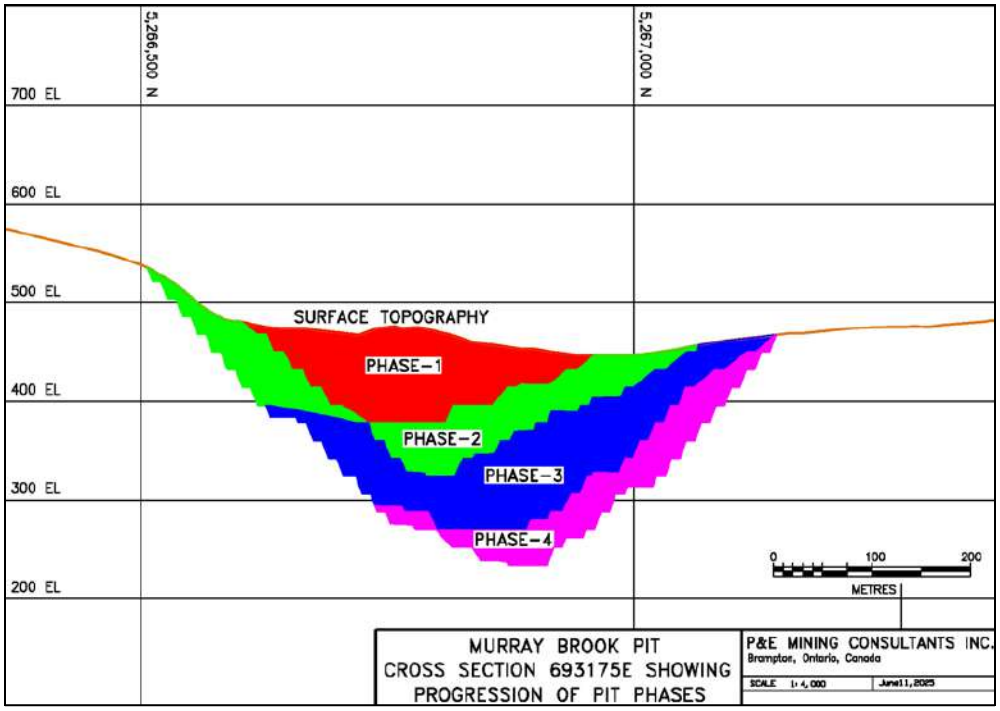
A series of intermediate pit phases was designed for life-of-mine production scheduling purposes. The phases are used to enhance project economics by distributing waste tonnages over time and allowing mining to start in an area near surface with limited waste rock to strip. In this respect, the pit was subdivided into four phases. These intermediate phases are shown in Figure 16-4 in plan view and Figure 16-5 in cross-section. The tonnages and grades contained within each phase are shown in Table 16-6.

Figure 16-4: Plan View of Pit Phases



Source: P&E, (2025).

Figure 16-5: Cross-section Through Pit Phases



Source: P&E, (2025).

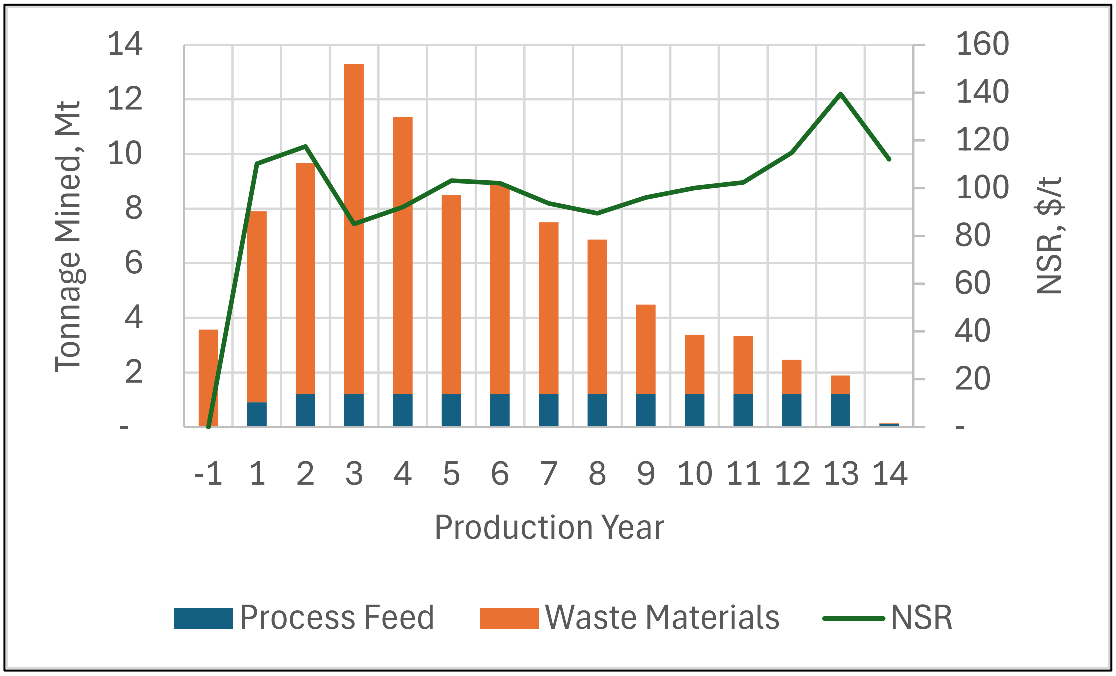
Table 16-6: Pit Phase Tonnages

Item	Unit	Phase 1	Phase 2	Phase 3	Phase 4	Total
Total Material in Pit	Mt	10.5	25.1	33.7	23.9	93.2
Overburden	Mt	3.4	1.5	0.7	0.3	5.9
NAG Waste Rock	Mt	3.1	14.1	24.1	18.8	60.1
PAG Waste Rock	Mt	2.4	4.5	3.6	1.3	11.8
Total Waste Material	Mt	8.8	20.0	28.5	20.4	77.7
Strip Ratio	waste:ore	5.2	4.0	5.4	5.8	5.0
Sulphide Feed (Diluted)	Mt	1.7	5.0	5.2	3.5	15.5
NSR	$/t	116.18	97.45	99.77	111.51	103.47
Gold	g/t	0.31	0.50	0.49	0.81	0.55
Silver	g/t	41.46	35.92	40.82	48.73	41.09
Copper	%	0.52	0.55	0.39	0.44	0.47
Lead	%	0.98	0.80	0.98	1.16	0.96
Zinc	%	3.14	2.26	2.77	2.90	2.67

16.9 Production Schedule

The life-of-mine production schedule was generated on an annual basis. As presented in Figure 16-6 and shown in Table 16-7, the mine production schedule consists of one pre-production year (pre-stripping), followed by 14 years of active mine and process production. The last production year is not a full year.

Figure 16-6: Mine Production Schedule



Source: P&E, (2025).

The target processing rate is approximately 1.2 Mt/a or 3,300 t/d. Over the life of mine, the pit will produce 15.5 Mt of process plant feed material grading 41.1 g/t Ag, 0.47% Cu, 0.96% Pb, and 2.67% Zn. Gold is not considered to be recoverable or payable in this assessment. The average life-of-mine NSR is estimated at $103.5/t. The process feed will be hauled by truck on a 17 km haul road to the Caribou plant, located east of the open pit.

The open pit will produce 5.9 Mt of overburden, 60.1 Mt of NAG waste rock, and 11.8 Mt of PAG waste rock. The total waste material mined is 77.7 Mt with a life-of-mine strip ratio of 5:1. As shown in Figure 16-7, each of the overburden, NAG waste, and PAG waste materials will be stored in a separate storage location near the pit area to the northwest, east, and north of the pit, respectively.

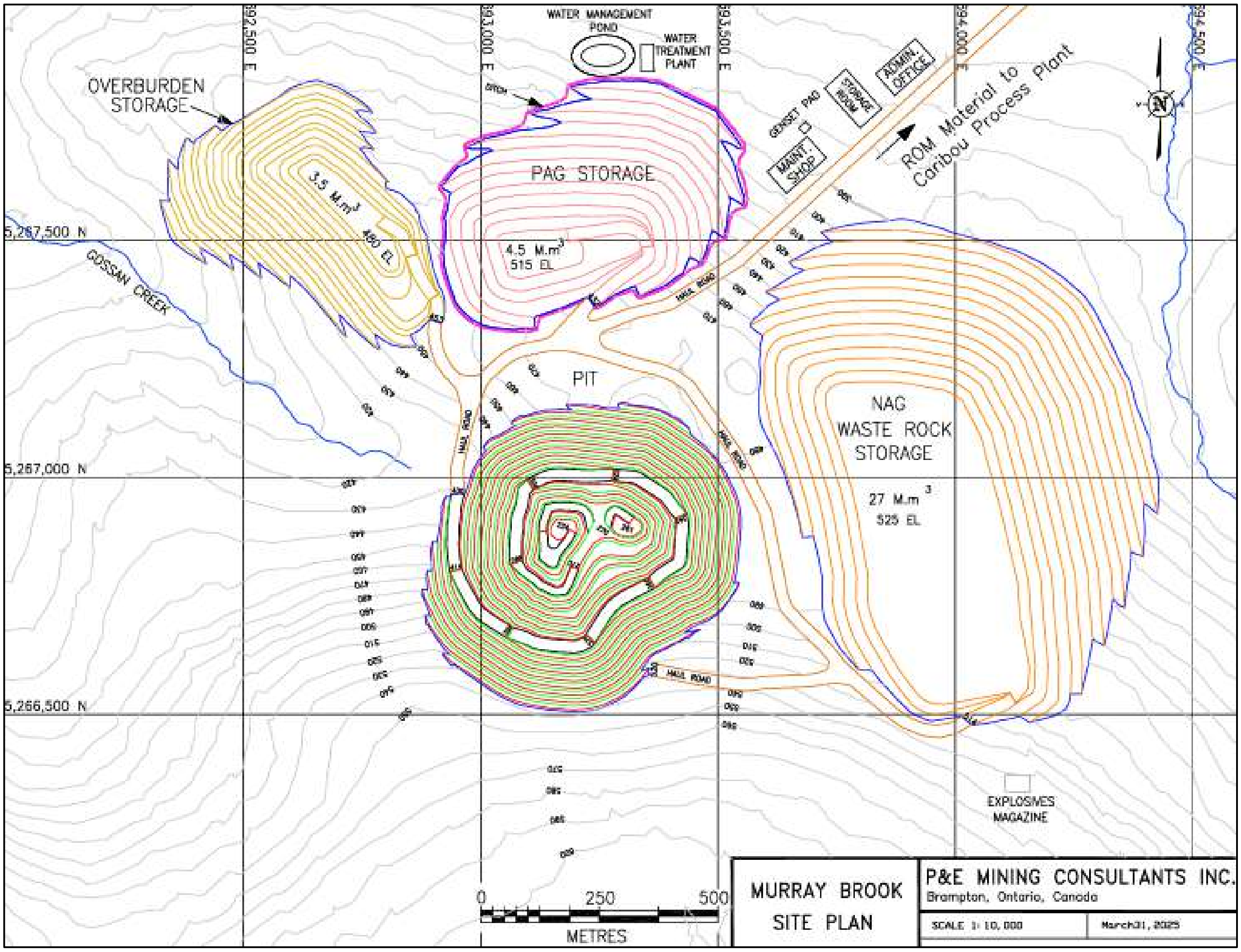
No temporary process feed stockpile will be created near the open pit. All mineralized material will be transported directly to the Caribou plant and placed into a run-of-mine stockpile near the primary crusher.

Table 16-7: Mine Production Schedule (Life of Mine)

Year	Tonnage																					PAG/NAG Split			
	Tonnage Mined (Mt)					Process Feed (Mt)					Head Grade						Total Waste Mined (Mt)					NAG (Mt)			PAG (Mt)
	Phase 1	Phase 2	Phase 3	Phase 4	Total	Phase 1	Phase 2	Phase 3	Phase 4	Total Process Feed	Ag (g/t)	Au (g/t)	Cu (%)	Pb (%)	Zn (%)	NSR ($/t)	Phase 1	Phase 2	Phase 3	Phase 4	Total	Over-burden	Waste Rock	Total	Total
-1	1.38	1.04	1.08	0.06	3.57	-	-	-	-	-	-	-	-	-	-	-	1.38	1.04	1.08	0.06	3.57	1.16	2.41	3.57	0.00
1	7.91	-	-	-	7.91	0.90	-	-	-	0.90	34.73	0.33	0.69	0.79	2.42	110.30	7.00	-	-	-	7.00	2.61	2.21	4.82	2.18
2	1.25	7.25	1.08	0.09	9.67	0.79	0.41	0.00	-	1.20	40.87	0.33	0.57	0.90	3.02	117.42	0.46	6.84	1.08	0.09	8.47	1.04	5.63	6.66	1.80
3	-	7.30	3.26	2.74	13.30	-	1.19	0.02	-	1.20	30.15	0.54	0.66	0.57	1.44	85.00	-	6.12	3.24	2.74	12.10	0.92	9.92	10.83	1.27
4	-	4.86	3.98	2.51	11.35	-	1.18	0.02	0.01	1.20	34.62	0.55	0.58	0.73	1.95	92.02	-	3.68	3.96	2.50	10.14	0.13	9.09	9.22	0.92
5	-	2.83	3.39	2.27	8.49	-	1.14	0.04	0.02	1.20	39.67	0.53	0.53	0.90	2.47	103.08	-	1.69	3.35	2.25	7.29	0.02	6.63	6.64	0.64
6	-	1.36	4.66	2.83	8.86	-	0.80	0.34	0.06	1.20	39.30	0.44	0.40	0.98	2.90	102.03	-	0.56	4.32	2.77	7.65	-	6.97	6.97	0.68
7	-	0.41	4.83	2.26	7.49	-	0.30	0.80	0.10	1.20	38.31	0.51	0.38	0.91	2.62	93.59	-	0.11	4.03	2.15	6.29	-	5.56	5.56	0.73
8	-	0.02	4.68	2.17	6.87	-	0.02	1.06	0.12	1.20	35.68	0.57	0.50	0.79	2.06	89.53	-	0.00	3.62	2.04	5.66	-	4.73	4.73	0.93
9	-	-	3.01	1.47	4.48	-	-	1.02	0.18	1.20	42.11	0.55	0.38	0.96	2.58	96.15	-	-	1.98	1.29	3.27	-	2.59	2.59	0.68
10	-	-	2.01	1.37	3.38	-	-	0.94	0.26	1.20	41.73	0.54	0.35	1.03	2.94	100.13	-	-	1.07	1.10	2.18	-	1.65	1.65	0.53
11	-	-	1.33	2.00	3.34	-	-	0.67	0.54	1.20	45.15	0.63	0.39	1.03	2.77	102.40	-	-	0.67	1.47	2.13	-	1.58	1.58	0.56
12	-	-	0.38	2.10	2.47	-	-	0.34	0.87	1.20	48.13	0.68	0.43	1.19	3.09	114.82	-	-	0.04	1.23	1.27	-	0.80	0.80	0.47
13	-	-	-	1.89	1.89	-	-	-	1.20	1.20	60.94	0.76	0.30	1.64	4.36	139.42	-	-	-	0.68	0.68	-	0.34	0.34	0.34
14	-	-	-	0.15	0.15	-	-	-	0.13	0.13	51.81	1.18	0.27	1.33	3.45	112.07	-	-	-	0.02	0.02	-	0.00	0.00	0.01
Total	**10.53**	**25.07**	**33.70**	**23.90**	**93.21**	**1.69**	**5.04**	**5.25**	**3.51**	**15.49**	**41.09**	**0.55**	**0.47**	**0.96**	**2.67**	**103.47**	**8.84**	**20.03**	**28.45**	**20.40**	**77.72**	**5.87**	**60.10**	**65.97**	**11.75**

Note: Ph. = Phase.

Figure 16-7: Murray Brook Mine Site Plan



Source: P&E, (2025).

16.10 Open Pit Mining Practices

The Murray Brook Project consists of a relatively shallow deposit that lends itself to conventional truck-and-shovel open pit mining methods. It is assumed that the Murray Brook pit will be a contractor-operated mining operation. All the drilling, blasting, loading, hauling operations, and equipment maintenance will be performed by a contractor. The Owner will supervise the contractor with a team of technical staff.

16.10.1 Drilling and Blasting

There are three different competencies of material to be mined. Similar competencies apply to both waste material or process plant feed. The three types are: (1) overburden, (2) oxide, and (3) hard rock, with overburden being the least competent rock and hard rock (sulphide process feed) being the most competent.

It is assumed that the overburden can be removed with free digging, so no drilling or blasting is required. The oxide and hard rock are more competent, so it is assumed that drilling and blasting will be required for both.

Blasting will be carried out using an ammonium nitrate and fuel oil (ANFO) mixture, which will be loaded by a bulk explosives truck directly into the drill holes. Blast initiation will be carried out using non-electric detonators and booster charges. The assumed powder factor is 0.77 kg/m^3 or 0.18 to 0.26 kg/t, depending on material density.

16.10.2 Loading and Hauling

Diesel-powered hydraulic excavators with a 6.5 m^3 heavy rock bucket will be used to free dig the overburden and excavate and load the blasted rock. The excavators will load the 65-tonne off-highway haul trucks in a 3 to 7 passes depending on the loose density of material being handled.

Loading operations will be supported by a wheel loader with a 6.5 m^3 rock bucket. Approximately, 15% to 20% of the mine haul truck loading will be performed by the wheel loader.

16.10.3 Pit Dewatering

The open pit will likely see some groundwater seepage in addition to regular precipitation events and snowmelt. Golder (2018b) estimated the expected water inflow into the pit to average 1,660 L/min.

Skid- or trailer-mounted centrifugal pumps will be staged up the side of the pit by the mining contractor to remove water from the pit sump locations during pit development.

16.10.4 Auxiliary Pit Services and Support Equipment

Primary mining operations will be supported by a fleet of support equipment consisting of Caterpillar D8 size class bulldozers with ripper attachments, Caterpillar 14 M class graders, as well as a Caterpillar 814 class wheel dozer, water truck, maintenance vehicles, and service vehicles.

16.10.5 Waste Material Storage

The open pit will require the development of several waste disposal locations. These will be of varying size and are listed in Table 16-8.

Table 16-8: Waste Disposal Facilities

Description	Quantity (Mt)	Density Placed (t/m³)	Volume (Mm³)
Overburden Stockpile	5.9	1.7	3.5
NAG Waste Rock	60.1	2.2	27.0
PAG Waste Rock	11.8	2.6	4.5

16.10.6 Support Facilities

The Murray Brook mine will require mine offices, maintenance facilities, warehousing, and cold storage areas. A mine office will be provided for mine management, engineering, geological, and survey services. These are part of the project infrastructure described in Section 18.

A maintenance shop will provide pit support services. The mine maintenance facility will consist of a truck shop with a wash facility, welding equipment, and a dedicated preventive maintenance bay. The facility will have adjoining indoor parts storage and tool crib. A fuel and lube station will be conveniently located near the maintenance facility and main haul road for equipment access. A mobile, truck-mounted fuel and lubrication system will be available to service mobile equipment in the field.

16.10.7 Mining Workforce

Mining operators and maintenance crews will be provided by the mining contractor. The Murray Brook mining operation will require a small Owner's workforce of technical, engineering, geological and surveying staff consisting approximately of 10 personnel, as summarized in Table 16-9.

Table 16-9: Owner's Staff

Year	Requirements
Mine Superintendent	1
Mine Clerk	1
Chief Engineer	1
Senior Mine Engineer	1
Geologist	2
Surveyor	1
Survey Technician	1
Mine Technician	1
Grade Control Technician	1
Total	**10**

17 RECOVERY METHODS

17.1 Background

The proposed processing strategy involves processing production from the Murray Brook deposit at the existing Caribou processing complex, which is approximately 10 km to the east. The Caribou complex is a 3,000 t/d design capacity lead and zinc concentrator, which ceased operations in 2022 due to process plant material feed delivery challenges from the underground Caribou mine. The facility is currently permitted to operate.

The Caribou complex was constructed in 1997, with operations suspended the following year due to recovery challenges. Operations were restarted in 2007, with fine grinding IsaMills installed to improve recovery. Operations were further suspended in 2008 due to a fall in zinc prices, however forecasted recoveries were achieved. Operations were subsequently restarted in 2015, with investment in a new SAG mill and copper flotation equipment. Forecasted recoveries were achieved, but the concentrator was not at design capacity due to material delivery challenges from the underground mine. Production ceased in Q2 of 2022, and the complex has been on care and maintenance.

17.1.1 Site Inspection Findings

A site inspection on the existing Caribou facilities was conducted by Marcello Locatelli of Inteloc. The conditions of the existing equipment are discussed in Section 18.

17.2 Process Plant Overview

The proposed project process flowsheet is based on the current Caribou concentrator process flowsheet, with minimal modifications to enable a 10% increase in mill feed capacity (3,300 t/d) and the recovery of a copper concentrate in addition to the zinc and lead concentrates recovered in the historical operation. The process is comprised of the following major circuits:

- jaw crushing of run-of-mine material (by service provider)
- SAG mill with trommel and classification screens, followed by a ball mill with cyclone classification
- lead and copper bulk flotation with regrinding prior to cleaner flotation
- lead and copper separation flotation
- zinc flotation with regrinding prior to cleaner flotation
- thickening, filtration and loading of copper, lead, and zinc concentrates
- tailings pumping and disposal.

The equipment sizing considers the re-use of existing equipment to reduce capital requirements and optimize project economics. The unit operations are standard technologies widely used in polymetallic concentrators. Table 17-1 shows a summary of the major process equipment required versus the equipment available at the Caribou concentrator. Note: primary crushing would be managed by a service provider, so the primary crusher capacities and capabilities have been excluded from this summary.

Table 17-1: Major Process Equipment Summary

Area	Equipment	Unit	Size	No. Required	No. Available	Comment
Grinding	SAG Mill	kW	1491	1	1	Split duty parallel ball mills; 746 kW ball mill re-purposed from zinc regrind to debottleneck primary grinding.
	Ball Mills	kW	1864 & 746	2	2	
Bulk Flotation	Rougher Flotation Cells	m^3	14.2	9	10	No modification of process flow or plant layout. Flotation cells to be bypassed when not required.
	Regrind Mills	kW	500	2	2	
	1st and 2nd Cleaner Flotation Cells	m^3	8.5	11	18	
	2nd and 4th Cleaner Flotation Cells	m^3	2.8	5	20	
Lead / Copper Cleaners	Lead Rougher Flotation Cells	m^3	2.8	1	6	Installed but not used in the previous Caribou process. Intended for a copper circuit by the previous operator
	Lead Cleaner Flotation Cells	m^3	0.8	2	6	Installed but not used in the previous Caribou process. Intended for a copper circuit by the previous operator
Zinc Flotation	Rougher Flotation Cells	m^3	8.5	10	18	No modification of process flow or plant layout. Flotation cells to be bypassed when not required.
	Regrind Mill	kW	500	1	1	Repurposed from lead cleaner regrind duty in previous process flow.
	Cleaner Flotation Cells	m^3	8.5	17	27	No modification of process flow or plant layout. Flotation cells to be bypassed when not required.

17.3 Process Design

The Caribou concentrator is intended to process 1.2 Mt/a or 3,300 t/d of production from the Murray Brook deposit, recovering saleable lead, copper, and zinc concentrates. The process design criteria established after a review of the available metallurgical testwork, historical operating data and comparable industry benchmarks are summarized on Table 17-2.

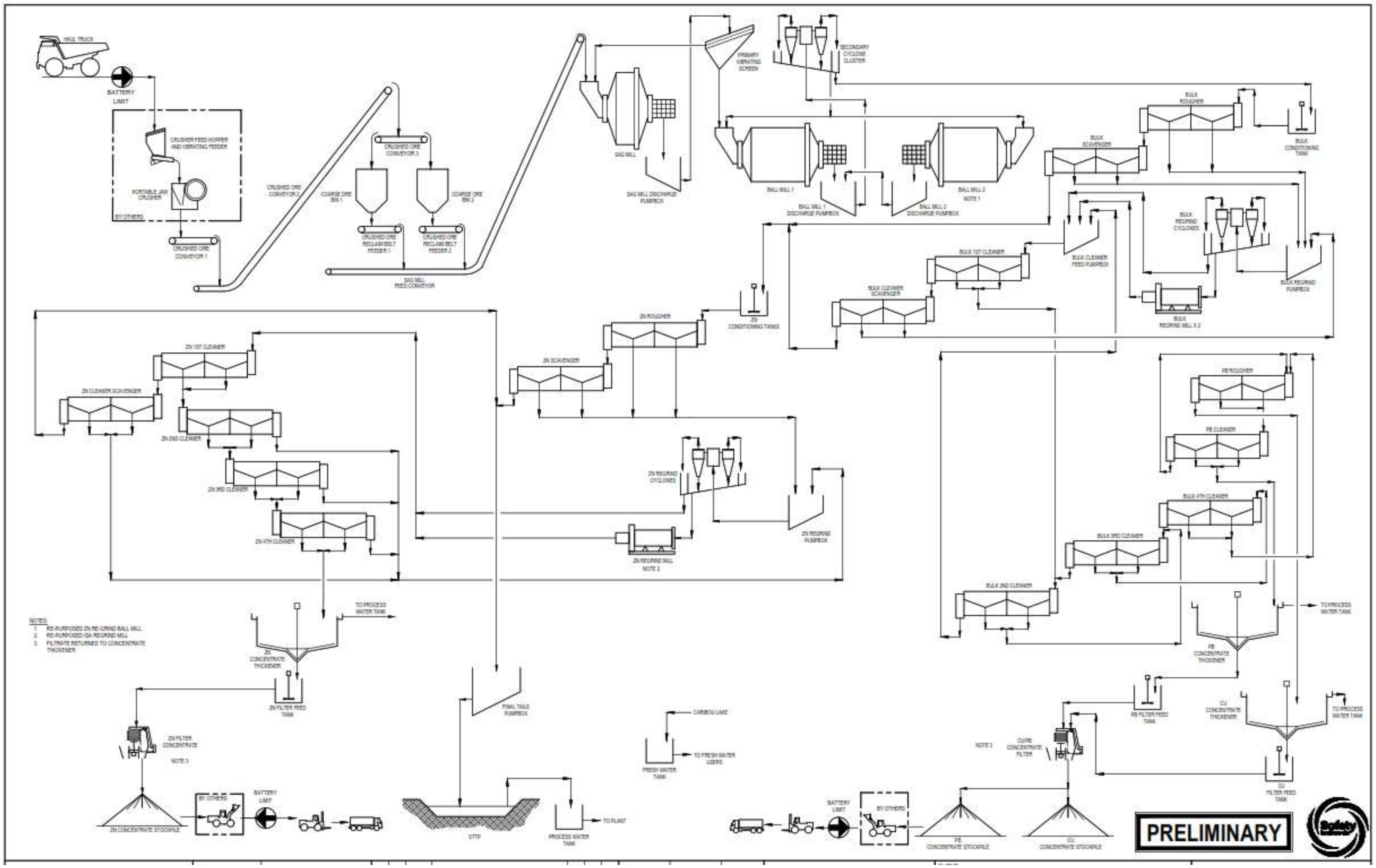
17.4 Process Flowsheet

The simplified overall process flow diagram (PFD), showing the major unit processes is shown on Figure 17-1.

Table 17-2: Key Process Design Criteria

Design Parameter	Units	Value
Throughput	Mt/a	1.2
	t/d	3,300
Run-of-Mine head Grade – Design		
– Copper	%	0.57
– Lead	%	0.90
– Zinc	%	3.02
Operating Availabilities		
– Grinding and Flotation	%	92
– Filtration	%	84
Semi-autogenous Grind Mill Feed Storage Bins Residence Time	h	16
Run-of-Mine Production Specific Gravity	-	4.42
JK SMC Axb	-	63.6
Bond Rod Mill Work Index, Design	kWh/t	15.7
Bond Ball Mill Work Index, Design	kWh/t	12.3
Bond Abrasion Index, Design	g	0.40
Crushing Circuit Product Size, k_{80}	mm	102
Primary Grind Size, k_{80}	µm	30
Design Copper and Lead Flotation Residence Times		
– Bulk Rougher and Scavenger	min	25
– Bulk Cleaner 1	min	32
– Bulk Cleaner 1 Scavenger	min	14
– Bulk Cleaner 2	min	18
– Bulk Cleaner 3	min	14
– Bulk Cleaner 4	min	18
– Lead Rougher	min	21
– Lead Cleaner	min	14
Copper and Lead Bulk Flotation Regrind Product Size, k_{80}	µm	12
Bulk Flotation Regrind Mill Specific Energy	kWh/t	24
Design Zinc Flotation Residence Times, Design		
– Zinc Rougher and Scavenger	min	18
– Zinc Cleaner 1	min	18
– Zinc Cleaner 1 Scavenger	min	18
– Zinc Cleaner 2	min	12
– Zinc Cleaner 3	min	18
– Zinc Cleaner 4	min	18
Zinc Flotation Regrind Product Size, k_{80}	µm	20
Bulk Flotation Regrind Mill Specific Energy	kWh/t	9
Metal Recoveries to Concentrate		
– Zinc to Zinc Concentrate	%	83
– Copper to Copper Concentrate	%	72
– Lead to Lead Concentrate	%	43
Target Concentrate Grades		
– Zinc Concentrate	%	48
– Copper Concentrate	%	22
– Lead Concentrate	%	45
Target Concentrate Filter Cake Moistures		
– Zinc Concentrate	% w/w	8.5
– Copper Concentrate	% w/w	8.0
– Lead Concentrate	% w/w	9.2

Figure 17-1: Simplified Overall Process Flowsheet



Source: Ausenco, 2025.

17.5 Plant Design

The individual processing areas are described in the following subsections.

17.5.1 Primary Crushing

The primary crushing of run-of mine production is outsourced to a service provider using a portable jaw crusher to reduce the mineralized material product size to 80% passing (P_{80}) 102 mm.

Jaw crusher discharge is transported by the crusher product conveyors to two SAG mill feed storage bins. Each bin has a storage capacity of 2,000 tonnes for a total live storage equivalent to 29 hours of mill feed at nominal rates. Belt feeders reclaim material from the storage bins onto the SAG mill feed conveyor. Each reclaim feeder can supply the design mill feed rate of 149 t/h.

The major equipment in this circuit includes the following:

- crusher feed hopper and vibrating feeder (to be provided with contract mining)
- portable jaw crusher (to be provided with contracted mining)
- crushed feed conveyor, existing (914 mm belt width, 171 m length)
- two existing SAG mill feed storage bins (1200-tonne capacity each)
- two reclaim belt feeders, existing (1069 mm width)
- SAG mill feed conveyor, existing (914 mm belt width, 31 m length).

17.5.2 Primary Grinding

The primary grinding circuit consists of a SAG mill in closed circuit with a vibrating screen and is followed by a ball mill in closed circuit with a hydrocyclone cluster. The circuit is designed to reduce SAG mill feed from a P_{80} of 106 mm to 40 µm.

The SAG mill feed conveyor transfers crushed material into the SAG mill. Process water is added to the SAG mill feed to maintain the required slurry pulp density. The SAG mill product is discharged through a trommel screen, and the oversize “scats” discharged into a bunker. The trommel screen underflow flows into a pumpbox where the material is pumped onto a vibrating screen. Oversized material (+12 mm) is returned to the SAG mill, and undersize material is discharged into ball mill no. 1. The ball mill product is similarly discharged through a trommel screen into the discharge pumpbox, which feeds the primary grinding cyclone cluster.

To enable the processing of harder Murray Brook deposit production and 10% increase in plant capacity over original design, the existing Caribou concentrator zinc regrind mill is repurposed to primary grinding duty as ball mill no. 2. This ball mill is fed by a portion of the cyclone underflow (approximately 30%), with product discharging through a trommel screen, into a pumpbox for pumping to the ball mill no. 1 discharge pumpbox feeding the cyclone cluster.

Hydrocyclone overflow has a target product size P_{80} of 40 µm and slurry pulp density of 41% percentage by mass (w/w). The cyclone underflow returns to the ball mills, with a design recirculating load of 350%. Process water is added to the various pumpboxes to maintain required slurry densities.

The major equipment in this circuit is as follows:

- SAG mill (6.7 m diameter x 2.1 m length, 1,491 kW)
- ball mill No. 1 (4.27 m diameter x 6.7 m length, 1,864 kW)
- ball mill No. 2 (3.0 m diameter x 6.9 m length, 746 kW)
- SAG mill discharge vibrating screen
- mill discharge pumps and pumpboxes
- primary cyclone classification cluster.

17.5.3 Bulk Flotation

Primary cyclone cluster overflow reports to the bulk flotation conditioning tank where hydrated lime, zinc sulphate, cyanide, collector, and frother are added. The conditioned slurry then flows by gravity to the bulk rougher flotation cells. Rougher concentrate is collected from each rougher cell and transported via a launder into the bulk regrind cyclone feed pumpbox, while the tailings are advanced to the bulk scavenger flotation cells. Bulk scavenger concentrate also reports to the bulk regrind cyclone feed pumpbox, and the tailings are sent to the zinc flotation condition tanks.

The bulk concentrate regrind circuit includes cyclones in open-circuit configuration ahead of two parallel, horizontal, stirred regrind mills. The purpose of the bulk regrind circuit is to reduce the concentrate to a P_{80} of 12 µm for cleaner flotation. The hydrocyclone overflow, which achieves the target product size, bypasses the bulk regrind mills to report to the bulk cleaner feed pumpbox. Hydrocyclone underflow reports to the two regrind mills, which also discharge the reground product to the bulk cleaner feed pumpbox. The regrind mills have internal product size control ensuring the mill product is at the required size distribution. Lime and zinc sulphate are also added to the regrind circuit ahead of cleaner flotation.

The reground material and cyclone overflow report to bulk cleaner flotation, which consists of four sequential cleaning stages. Concentrates flow from the first stage through to the fourth stage, whereafter the concentrate is advanced to lead/copper cleaning flotation. Flotation tailings flow counter-currently to the concentrate, and the first cleaner tailings are sent to a cleaner scavenger. Concentrate from the cleaner scavenger is recycled to the bulk regrind pumpboxes for further liberation, and the tailings are combined with the bulk rougher scavenger tailings to zinc rougher flotation. The cleaning stages are dosed with cyanide, collector, and frother as required.

The major equipment in the bulk flotation circuit is as follows:

- 10 rougher flotation cells (14.2 m^3 each)
- 2 horizontal stirred regrind (IsaMill) mills (500 kW each)
- 18 first and second cleaner flotation cells (14.2 m^3 each)
- 20 third and fourth cleaner flotation cells (2.8 m^3 each)
- regrind classification cyclone cluster.

17.5.4 Lead/copper Cleaner Flotation

The fourth bulk flotation cleaner concentrate is fed to the lead rougher flotation cells to separate lead and copper concentrates. The lead rougher tailings, which is the copper concentrate, are sent to the copper concentrate thickener. The lead rougher concentrate is sent to a cleaning stage, and the concentrate is sent to the lead concentrate thickener. The cleaner stage tailings are recycled to the lead rougher stage. The lead rougher is dosed with a depressant and frother to facilitate the separation of the copper and lead concentrates.

The major equipment in this circuit is as follows:

- 6 lead rougher flotation cells (2.8 m^3 each)
- 6 lead cleaner flotation cells (0.8 m^3 each).

17.5.5 Zinc Flotation

The bulk flotation rougher and first cleaner scavenger tailings report to the zinc flotation conditioning tank, where copper sulphate, lime, collector, and frother are added. The conditioned slurry overflows and gravitates to the zinc rougher flotation cells. Rougher concentrate is collected from each rougher cell and transported via a launder into the zinc regrind cyclone feed pumpbox, while the tailings are advanced to the zinc rougher scavenger flotation cells. Zinc rougher scavenger concentrate also reports to the zinc regrind cyclone feed pumpbox, and the tailings are sent to the final tailings pumpbox.

The zinc regrind circuit includes cyclones operating in open circuit configuration and a single, horizontal, stirred regrind mill. The target regrind product size distribution is a P_{80} of 20 µm. The hydrocyclone overflow reports to the zinc cleaner feed pumpbox. The cyclone underflow reports to the regrind mill, which also discharges the mill product to the same zinc cleaner feed pumpbox. The regrind mill has internal product size control ensuring the mill product has the required size distribution. Lime and copper sulphate are also added to the regrind circuit ahead of cleaner flotation.

The zinc cleaner circuit consists of four sequential cleaning stages. Concentrates flow from the first stage through to the fourth stage, whereafter the concentrate reports to the zinc concentrate thickener. The first cleaner cell tailings flow to a cleaner scavenger, whose concentrate is combined with the cleaner two through four tailings and recycled to the bulk regrind pumpboxes. The cleaner scavenger tailings are combined with the bulk rougher scavenger tailings to the final tails pump box. The cleaning stages are dosed with lime, collector, and frother as required.

The major equipment in the zinc flotation circuit is as follows:

- 45 flotation cells (8.5 m^3 each)
- 1 horizontal stirred regrind mill (IsaMill) (500 kW)
- regrind classification cyclone cluster.

17.5.6 Concentrates Dewatering and Handling

The concentrates are thickened in their respective concentrate thickeners, and the thickener overflow reports to the process water tank. The underflows are pumped to individual filter feed tanks for further dewatering through two filter presses. One filter alternates between filtering copper and lead concentrates, while the other is dedicated to zinc concentrate. The resulting filter cakes are deposited onto individual concentrate stockpiles where they are reclaimed by front end loaders and loaded into trucks to be shipped off site. Filtrates are recycled to their respective concentrate thickener. The target moisture content for the copper, lead, and zinc concentrates is 8.0%, 9.2% and 8.5%, respectively.

The major equipment in the concentrate dewatering and handling circuits is as follows:

- copper concentrate thickener (5 m diameter)
- lead concentrate thickener (5 m diameter)
- zinc concentrate thickener (12 m diameter)
- lead/copper filter press (33 plates)
- zinc filter press (41 plates).

17.5.7 Tailings Handling

The tailings from zinc scavenger flotation are pumped to the tailings management facility for impoundment. The solids consolidate and decant water is recovered and returned to the process water tank for re-use in the concentrator.

17.5.8 Reagents Handling & Storage

In general, each reagent mixing and storage system is located within containment areas to prevent spillage and contamination of the environment as well as unintended mixing with other reagents. Storage tanks are equipped with level indicators, instrumentation, and alarms to ensure spillage does not occur during normal operation. Additional controls such as ventilation, fire and safety protection, and eyewash stations will be considered as appropriate. Safety data sheets (SDSs) will be available for all reagents.

Reagent dosages will be controlled via dosing pumps and/or control valves as required. The reagents used are summarized in table 17-3.

Table 17-3: Reagent Handling & Storage Summary

Reagent	Delivery	Preparation Method	Use
Quicklime	Truck	Slaked to hydrated lime and mixed to 20% w/w strength	pH modifier
Sodium Cyanide	Bulk Bag	Mixed to 20% w/w strength	Zinc depressant
Zinc Sulphate Monohydrate	Bulk Bag	Mixed to 20% w/w strength	Zinc depressant
Potassium Ethyl Xanthate (PEX)	Bulk Bag	Mixed to 20% w/w strength	Collector
Copper Sulphate (Pentahydrate)	Bulk Bag	Mixed to 15% w/w strength	Collector
Methyl Isobutyl Carbinol (MIBC)	IBC	Used neat from intermediate bulk container (IBC)	Frother
Sodium metabisulphite (SMBS)	Bulk Bag	Mixed to 20% w/w strength	Depressant
Flocculant	Bulk Bag	Hydrated to 0.25% w/v and dosed at 0.025% w/v strength	Settling aid

17.6 Energy, Water, and Process Materials Requirements

17.6.1.1 Process Materials

Reagent and consumable usage rates were estimated from historical operating data, preliminary testwork data, and comparable benchmarks. Table 17-4 shows the expected average annual reagent and consumables usage rates.

Table 17-4: Reagent and Consumables Usage Rates

Item	Unit	Usage Rate
Reagents		
Quicklime	t/a	2,770
Sodium Cyanide	t/a	542
Zinc Sulphate Monohydrate	t/a	1,127
Potassium Ethyl Xanthate (PEX)	t/a	33
Copper Sulphate (Pentahydrate)	t/a	337
Methyl Isobutyl Carbinol (MIBC)	t/a	36
Sodium Metabisulphite (SMBS)	t/a	1,205
Flocculant	t/a	33
Operating consumables		
SAG Mill Media	t/a	373
Ball Mill Media	t/a	866
Regrind Mill Media	t/a	8.3
Maintenance Consumables		
SAG Mill Liners	set/a	1
Ball Mill Liners, per Mill	set/a	2
Regrind Mill Liners, per Mill	set/a	3
Filter Cloths, per Filter	set/a	14

17.6.2 Process and Instrument Air

Three plant air compressors will supply air to various plant services at a pressure of up to 862 kPa. An air dryer and oil separator will remove moisture for instrumentation air. A dedicated air compressor and four receivers will be provided for the concentrate filter. Five blowers will supply air to the flotation cells.

17.6.3 Water Requirements

The process will require both fresh and reclaim water to maintain the plant water balance. Fresh water will be sourced from Caribou Lake and used for gland seal, reagent mixing, and potable water supply to the administration building. Excluding potable water users, the freshwater requirement is estimated at 21 m^3/h or 0.17 Mm^3/a.

Process water supply consists of reclaimed water from the concentrate thickener overflows and tailings facility decant supernatant, and make-up water to account for the volumes entrained in the tailings. The make-up water is supplied from a firewater retention pond called the "fire pond." The fire pond, also the source of water for the fire protection systems, has a pumphouse with a standby generator in the event of power supply interruption.

The process water demand is estimated at 257 m^3/h or 2.01 Mm^3/a. The make-up water demand will be approximately 35 m^3/h or 0.26 Mm^3/a.

17.6.4 Power Requirements

The overall facility has 11.6 MW of installed power, with an estimated nominal demand of 8.4 MW. The estimated annual power consumption is 68 GWh/a. The power usage by process area is summarised in Table 17-5.

Table 17-5: Projected Power Consumption

Area	Installed Power (kW)	Nominal Demand (kW)	Consumption (MWh/a)
Crushing	171	111	896
Primary Grinding	3,710	3,104	25,018
Bulk Flotation and Regrind	2,208	1,435	11,564
Zinc Flotation and Regrind	1,820	1,392	11,217
Lead/Copper Cleaners	7	5	39
Concentrate Dewatering	331	215	1,734
Reagents	83	54	434
Services	3,272	2,127	17,141
Total	**11,602**	**8,443**	**68,044**

18 PROJECT INFRASTRUCTURE

18.1 Introduction

Infrastructure to support the Murray Brook Project consists of existing infrastructure related to the Caribou process plant as well as new infrastructure to support the Murray Brook mine site.

The existing project infrastructure at the Caribou site includes:

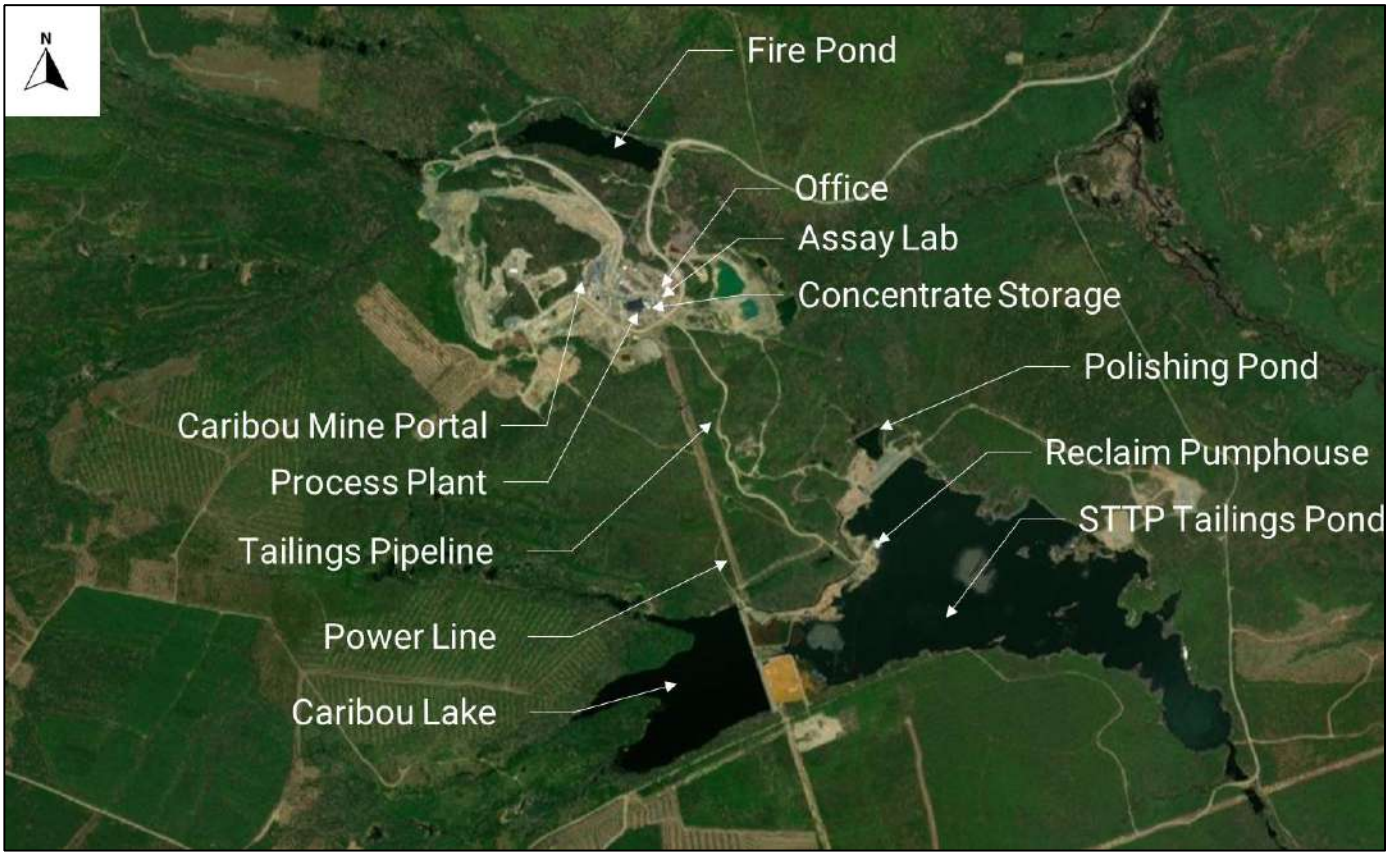
- a site access road, and other facility roads
- process facilities include the process plant, crushed feed material bins and conveyors, process plant workshop, assay laboratory, and tailings management facility
- an administration office, warehouse, maintenance shop
- water treatment systems and building
- high-voltage powerline and on-site electrical substation
- diversions, ditches, ponds, and effluent treatment for management of contact and redirection of non-contact water
- potable water, fire water, compressed air, power, diesel, communication, and sanitary systems.

The Caribou Process plant site is shown in Figure 18-1.

The Murray Brook mine site project infrastructure will include:

- site civil work
- diversions, ditches, ponds, and effluent treatment to manage contact water
- mine facilities include offices, maintenance shop, and storage facilities
- electrical power provided by on-site generation.

Figure 18-1: Caribou Process Plant Layout



Source: Ausenco 2025.

18.2 Site Access

The Caribou process plant and Murray Brook mine site are located approximately 50 km west of the city of Bathurst. The Caribou process plant can both be accessed using a gravel road from New Brunswick Route 180, a two-lane highway. Route 180 is maintained in all seasons by the New Brunswick Department of Transportation. Access to the Caribou site is from Route 180 is along a 3.8 km gravel road.

Bathurst is 220 km north of Moncton by road and has a regional airport with regularly scheduled flights to and from Montreal.

The Port of Belledune is located 75 km northeast of the Caribou site by road and is the closest deep-water port. The port was previously used by the Caribou facility to ship concentrates. Terminal 1 is capable of exporting mineral concentrates and the berth is 155 m in length and 11 m in depth. There are four terminals on site with the capability of receiving ships up to a Panamax type.

18.3 Buildings

18.3.1 Caribou Site Buildings

The existing Caribou site buildings are summarized in Table 18-1.

No new buildings will be required; however, concentrate storage will be expanded with a fabric building with dimensions 12.2 m length x 6 m width x 8 m height.

Table 18-1: List of Process Buildings

Building Description	L (m)	W (m)
Administration Office	40	30
Core Shack	40	10
Mill Building	90	40
Concentrate Storage (Existing)	45	30
Concentrate Storage (Addition)	12	6
Assay Laboratory	20	15
Warehouse	50	20
Maintenance Shop	18	24
Water Treatment Building	20	13

Source: Ausenco, 2025.

18.3.2 Caribou Site Building Condition

The Caribou mine site is currently idle, with the process plant on care and maintenance. During a site visit in October, 2024, the administration office, mill building, concentrate storage, assay laboratory and water treatment building, warehouse and maintenance shop were visited, and the core shack viewed from the exterior. A small crew was on site performing security, operations, and maintenance duties associated with site water treatment. The facilities appeared to be in good repair, generally neat and tidy and remained connected to electrical power. No significant building maintenance requirements were apparent. The concentrate storage extension and work partially completed for the flotation circuit upgrades in the mill building are two areas that will require attention for restart.

18.3.3 Caribou Process Plant Existing Condition Restart Review

Marcello Locatelli of Inteloc reviewed the existing mill infrastructure including the mill mechanical equipment and water treatment facilities. The existing mill infrastructure reviewed by Inteloc includes the mill mechanical equipment and water treatment facilities. During the review the mechanical equipment was visually inspected. From the inspection, Inteloc provided recommendations on potential repairs and replacements. These were considered in the restart estimate. The following infrastructure areas were not reviewed by Inteloc: mining, tailings deposition, electrical.

The capabilities of the existing electrical infrastructure and water circuit for Ausenco's additional scope were not considered by Inteloc.

The existing water treatment facilities were visually inspected by Inteloc during their visit. The mechanical equipment was in working condition and any repairs or replacements suggested by Inteloc are included in the restart estimate.

18.3.4 Murray Brook Site Mining Buildings

The contractor will be responsible for supplying its office facilities with staff. It is expected that trailers will be utilized. The contractor will also supply maintenance facilities for the mining equipment, and storage facilities for parts. The Owner will supply office facilities for its staff.

18.3.5 Accommodation

Due to the project's proximity to the city of Bathurst, accommodations for the employees at the mine will be based in Bathurst and the surrounding area and as a result, on-site accommodations will not be needed.

18.4 Murray Brook to Caribou Process Plant Hauling

A 17 km haul road will be constructed/upgraded from the Murray Brook open pit to the Caribou process plant. The mining contractor will be responsible for construction during the pre-production period.

18.5 Caribou Process Plant Feed Receiving and Storage

Mineralized material will be delivered to the Caribou site and dumped approximately 200 m northeast of the process plant building on an existing run-of-mine pad. A front-end loader will move the dumped material into the primary crusher (operated by the mining contractor). The crushed material will move to a hopper that feeds a conveyor system connected to two 2,000-tonne feed material storage bins. Belt feeders feed each bin individually.

18.6 Existing Caribou Mine Water Treatment Plant

Underground mine water from Caribou is treated in a building 200 m southeast of the milling complex. The water is treated in tanks housed by a metal-clad building using a lime mixing tank which is dosed into a second tank with the mine water prior to pumping to the tailings facility. Lime is stored in a 40-tonne silo adjacent to the building.

18.7 Tailings Management Facilities

The Murray Brook mine is expected to have a life of mine of 13 years and the run-of-mine material will be extracted at a rate of 3,300 t/d for a total of approximately 15.5 Mt. The run-of-mine material will be trucked to the nearby Caribou mine milling complex for processing. Tailings generated during processing will be piped as slurry and deposited subaqueously into the designated tailings management facilities described below.

For the first three years of production, tailings will be subaqueously deposited within the existing South Tributary tailings pond. It is understood that prior to the start of mining activities, the South Tributary tailings pond will be raised by others to allow for additional tailings storage (KCB, 2025). Based on the tailings properties outlined in Table 18-2, the raised tailings pond will have the capacity to store approximately 4 Mt (or 1.8 Mm^3) of tailings.

Once the capacity of the South Tributary tailings pond is exhausted, a new tailings management facility will be required. The proposed tailings management facility dam will be constructed to the northeast of the Caribou mine site, downstream of the existing South Tributary tailings pond and fire pond, as shown on Figure 18-2. The facility will be constructed in a valley along the North Tributary to Forty Mile Brook.

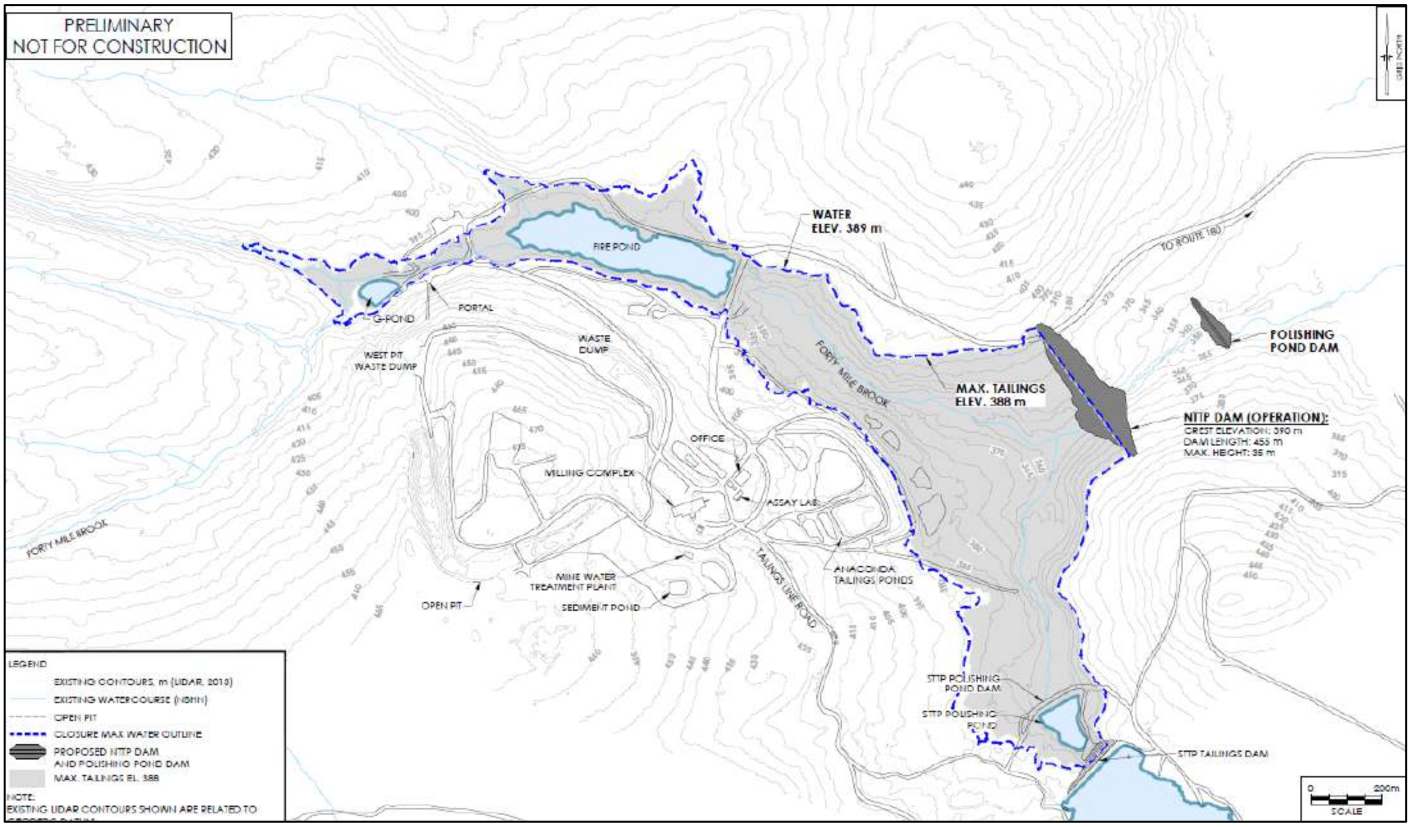
The North Tributary tailings pond dam will consist of a geomembrane-faced rockfill dam constructed in stages using a downstream raise method. Both operational and emergency spillways will be required, along with diversion structures to divert non-contact water around the tailings management facility. A new polishing pond dam will be constructed downstream of the main tailings dam. This structure will capture seepage and support effluent treatment and settling before discharge. Details of the design are provided in Table 18-2, while Table 18-3 summarizes the tailings management plan over the life of mine.

Table 18-2: North Tributary Tailings Pond Dam Details

Item	Detail
Crest Width	13 m
Maximum Crest Elevation	390 amsl
Maximum Dam Height	40 m above ground level (agl)
Dam Length	355 m
Upstream Dam Slope (Horizontal to Vertical)	2.5:1 to 3:1
Downstream Dam Slope (Horizontal to Vertical)	2.0:1
Gravity Material	Rockfill
Low Permeability Material	glacial till / geomembrane
Material Specific Gravity	4.4
Void Ratio of Deposited tailings	1.0
Deposited Dry Density	2.18 t/m^3

The dams will be constructed using locally sourced materials. Due to uncertainty regarding the availability of low-permeability materials, geomembranes will be incorporated as part of the low-permeability barriers. The conceptual dam design includes layers of rockfill, filter materials, native borrow, and geomembrane, with upstream slope protection.

Figure 18-2: Layout of North Tributary Tailings Pond Storage Facility



Source: Stantec, 2025.

Table 18-3: Tailings Management over the Life of Mine

Tailings Facility	Storage (Mt)	Storage Volume (Mm³)	Operating Cycle (Years)
South Tributary Tailings Pond	4.00	1.83	0 - 3
North Tributary Tailings Pond	14.07	6.45	8 - 15
Total	**18.07**	**8.29**	**0 - 15**

18.7.1 Operation of the Tailings Management Facility

The preliminary operational plan for the tailings management facility involves subaqueous deposition of slurry tailings via floating pipes. Initial deposition will focus on areas near the dam to help reduce stability and seepage risks. A detailed tailings deposition plan will be developed to optimize storage within the facility.

Given the potential for the tailings to be acid-generating, the preliminary design assumes a 1-metre water cover will be maintained above the deposited tailings. Additionally, a 1-metre freeboard allowance has been included above the operational water level for containment and safety purposes.

18.7.2 Tailings Management Facility – Additional Considerations

The proposed location of the new tailings management facility was selected based on its proximity to the existing infrastructure and its alignment with the existing closure strategy for the Caribou mine. Additional details of the closure concept are provided in Section 20.

At this preliminary stage, no additional environmental constraints were considered in selecting the location of the tailings management facility.

18.8 Caribou Site Water Treatment

Water treatment is expected to be required during operations and closure at the Caribou mine. Treatment needs will depend on several factors, including water quality from the milling process, final water quality in the tailings pond, volume of non-contact water diverted around the system, and other site-specific variables.

At this stage, a partial treatment stream has been assumed, reflecting the anticipated high volume of effluent requiring treatment. The preliminary treatment concept includes an initial concentration step to reduce chemical consumption, followed by treatment using lime, polyaluminum chloride (PAC), polymer, and clarifiers.

The preliminary treatment requirements are based on existing water quality data from the South Tributary tailings pond at the Caribou site and are designed to meet the Metal and Diamond Mining Effluent Regulations (MDMER) discharge limits.

High-level treatment concepts and associated costs were developed using vendor-supplied information. The water treatment plant will be required once the North Tributary tailings pond becomes operational.

18.9 Waste Rock Storage Facilities

The open pit will require several waste disposal locations to be developed for overburden material, and PAG and NAG waste rock. The PAG facility will be lined to ensure potential acid drainage is collected and treated. All waste rock storage areas will have a perimeter drainage collection ditch and a water collection pond. Water in the ponds will be treated as required, and water in the PAG collection pond will be pumped to a new water treatment facility to be installed at Murray Brook, before being discharged.

18.10 Explosives Storage

The mining contractor will be responsible for constructing an explosives storage and magazine facility. The facility is planned to be located approximately 750 m southeast of the open pit.

18.11 Power and Electrical

The estimated total connected load at the Caribou site is 11 MW with a nominal load of approximately 8.4 MW.

18.11.1 Caribou Process Plant Facility Power Supply

Primary power to the Caribou site is provided through an existing transformer with equipment tag T1A. The T1A transformer is a 138 / 4.16 kV transformer. The 138 kV high-voltage transmission line to the transformer is supplied by an overhead transmission connected to the site from the south of the facility and is owned by New Brunswick Power. After travelling south for 3 km, the 138 kV transmission line travels east to a substation in Bathurst.

Backup power is provided by an 800 kW, 1000 kVA diesel generator.

18.12 Fuel

The diesel fuel storage facility at the Caribou process plant will be used. A mobile tanker will transport diesel to the open pit mining equipment.

18.13 Water Supply and Management

Process water is composed of reclaim water and make-up water as required. Reclaim water is pumped back from the South Tributary tailings pond to feed the mill, and make-up water to the process is supplied from the fire water retention pond located approximately 600 m north of the process plant.

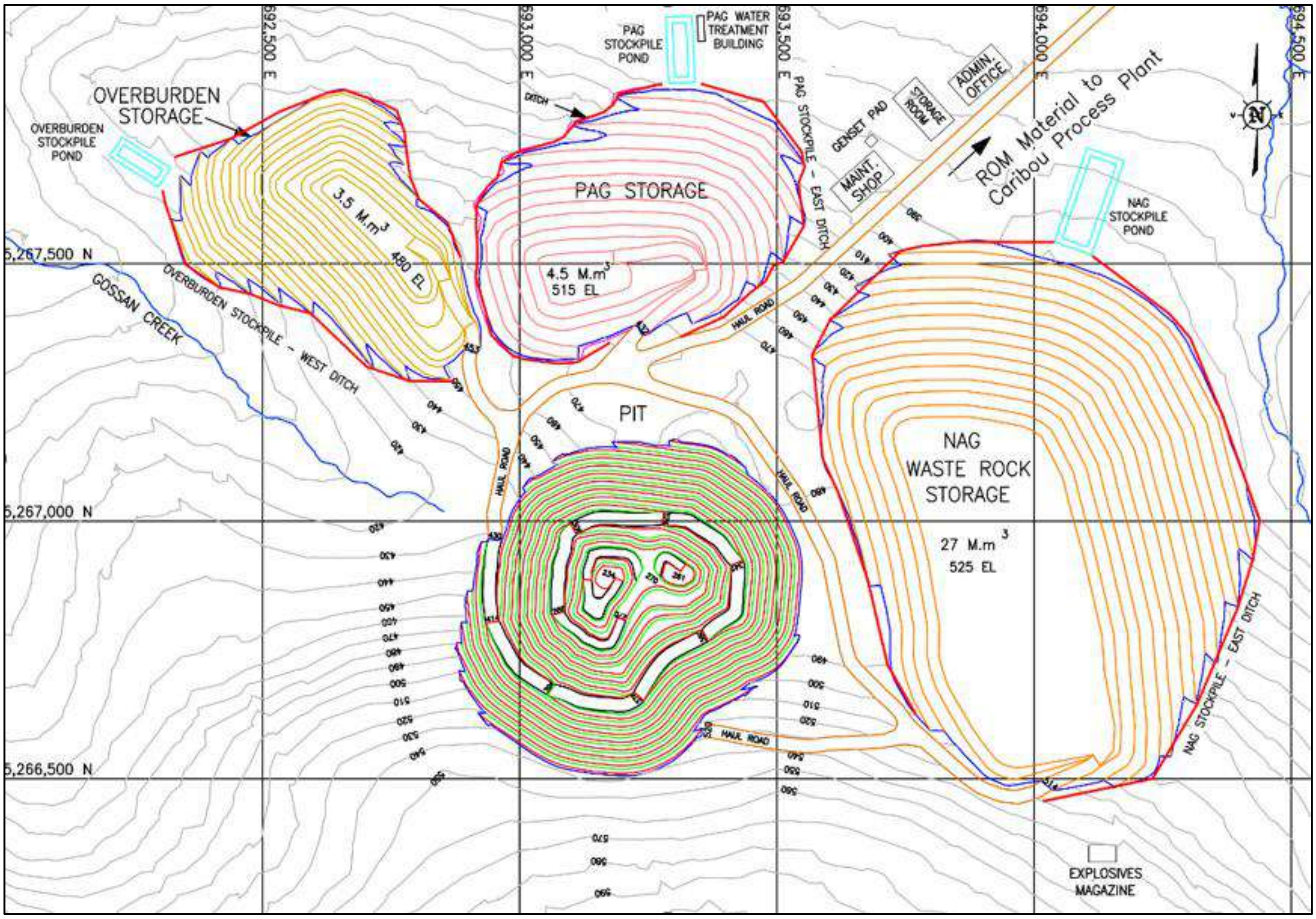
18.14 Murray Brook Site Water Management

This section discusses the site-wide management of water, the design of water management structures, hydrology, and the water balance. A site-wide water management strategy has been developed based on the proposed mine arrangement and will be implemented during mine development and operations.

The investigation of site-wide water management considered two types of water: contact water, which may require treatment, and contact water that would be treated for suspended solids but would otherwise be of sufficient quality to be released to the environment. Water requiring treatment would be collected from the PAG stockpile and from pit dewatering. Runoff from the NAG and overburden stockpiles would be collected in their own ponds for sediment removal and passive release to the receiving environment.

Figure 18-3 shows the project's water management structures, including collection channels and collection ponds.

Figure 18-3: Water Management Structures



Source: P&E (2025).

18.14.1 Design Basis Criteria

The criteria used to design the water management facilities are summarized in Table 18-4.

Table 18-4: Water Management Design Criteria Summary

Item	Design Criteria
Meteorological Data	Environment Canada Climate Normal, 1991-2020 (Bathurst Station) The IDF_CC tool (ungauged stations; project location)
Collection Ditches	Design flow return period for hydraulic design (i.e., ditch size, erosion protection): – 1/10-year, 24-hour storm event (NAG and overburden stockpile) – 1/100-year, 24-hour storm event (PAG Stockpile)
Collection Ponds	Sediment storage: 0.5 m above the basin floor for storage of accumulated sediment
	Pond storage volume: – runoff resulting from the 1-in-10-year, 24-hour storm event (NAG and overburden stockpile) – runoff resulting from the 1-in-100-year, 24-hour storm event (PAG stockpile)
	Freeboard (where water contained by berms): minimum 0.5 m between the pond berm crest and the 200-year water level
Culvert Road Crossings	Design flow return period for hydraulic design: 1/10-year, 24-hour storm event
	Minimum barrel slope: 0.5%

18.14.2 Diversion and Collection Ditches

Collection ditches collect contact runoff from the facilities and direct it to various collection locations for treatment, if required, or release it to the environment. The collection ditches for the NAG and overburden stockpiles are sized to convey the peak runoff produced from 1/10 year, 24-hour storm event, whereas the ditches for the PAG stockpile are sized for a 1/100 year, 24-hour storm event.

The collection ditches are planned to have a trapezoidal cross-section, with side slopes of 2H:1V, bottom widths of 1 to 2 meters, and a minimum required depth varying between approximately 0.5 to 1.3 meters. Ditches will be formed by either excavating into current overburden or reshaping surface materials through grading to achieve the required ditch geometry.

18.14.3 Collection Ponds

Collection ponds will be constructed to collect and temporarily store or attenuate runoff from disturbed areas prior to treatment or release to the environment:

- PAG stockpile pond will collect runoff from the PAG stockpile for treatment. Precipitation on the open pit will also be pumped to this pond for treatment. It has been assumed that the pond will be lined with a geomembrane. This pond has been sized to store runoff from a 1/100 year, 24-hour storm event.
- Overburden stockpile pond and NAG stockpile pond will collect runoff from their respective stockpiles, where sediment will be removed and flow will be passively released to the receiving environment. It has been assumed

that these ponds will be used for sediment removal, but treatment will otherwise not be required. Both ponds have been sized to store and/or attenuate runoff from a 1/10 year, 24-hour storm event.

The pond designs include excavation and containment by constructed berms where appropriate based on local topographical considerations. The ponds were sized considering 3H:1V side slope. Ponds include a spillway with erosion protection (riprap) to convey flows that may result from a storm event greater than their design events.

18.14.4 Site-Wide Water Balance

A preliminary site-wide water balance analysis was performed using average monthly precipitation and snowmelt rates. Pit inflows were estimated by Golder (2018) to be approximately 1,660 L/min, based on pit geometry, as well as hydrogeological data from a nearby site with presumably similar geology and water table elevation. Table 18-5 presents the water balance quantities for the site.

Table 18-5: Site-Wide Water Balance

Mine Facilities	Jan	Feb	Mar	Apr	May	Jun	Jul	Aug	Sep	Oct	Nov	Dec
Average Monthly Inflow (m^3 per Month)												
Overburden Stockpile	11,303	6,402	8,028	13,849	57,205	21,774	21,863	19,265	19,441	26,332	21,400	19,240
NAG Stockpile	37,389	21,178	26,554	45,810	189,229	72,027	72,319	63,725	64,308	87,103	70,789	63,645
PAG Stockpile	14,234	8,063	10,109	17,440	72,041	27,421	27,532	24,261	24,482	33,161	26,950	24,230
Precipitation on Pit	27,154	23,934	28,880	25,926	35,917	32,830	32,963	29,046	29,312	39,702	32,266	33,129
Pit Seepage	74,102	66,931	74,102	71,712	74,102	71,712	74,102	74,102	71,712	74,102	71,712	74,102
Inflow to Water Treatment (m^3 per Month)												
PAG Stockpile + Pits	115,491	98,928	113,092	115,078	182,061	131,964	134,598	127,409	125,506	146,965	130,928	131,462

19 MARKET STUDIES AND CONTRACTS

No market studies were completed for commodity pricing as part of the preliminary economic assessment. Market price assumptions were based on a review of public information, industry consensus, standard practices, and specific information from comparable operations in the region. Smelter terms and transportation costs were based on estimated terms provided by established concentrate marketing specialists.

20 ENVIRONMENTAL STUDIES, PERMITTING & SOCIAL OR COMMUNITY IMPACT

20.1 Introduction

This chapter provides an overview of the setting of the Murray Brook Project. It outlines existing biological and physical baseline conditions, proposed new baseline studies to support future permitting applications, existing permits, and future regulatory and permitting requirements including required management plans for water, site environmental monitoring, and waste disposal. In addition, this chapter also discusses socioeconomic baseline conditions, the status of community consultation and engagement, and conceptual mine closure and reclamation planning for the project. Recommendations are also provided in the event that the project progresses through the feasibility, environmental assessment and permitting phases.

The project will consist of the Murray Brook mine site and the Caribou milling complex connected to each other via a 10 km haul road (Figure 20-1). The Murray Brook Project consists of two sites, Caribou and Murray Brook. The process plant, tailings management facility, and other infrastructure are located at the Caribou site. The Caribou complex is permitted and reported to be available to process Murray Brook feed under current agreements. The deposit and mine are located at the Murray Brook site which has not yet been permitted. Note that at this time, the alignment of the road (shown on Figure 20-1) is approximate and subject to future studies, potential constraints, and re-alignment. The Murray Brook site and the Caribou site are previously disturbed. The Caribou complex has been operating intermittently since the 1970s, with the concentrator having been commissioned in 1990. The New Brunswick government has managed the care and maintenance of the Caribou complex since January 2023. The Murray Brook site operated as small open pit which discontinued in 1992, when the open pit was reclaimed.

The project is approximately 60 km west of the City of Bathurst in the Parish of Balmoral, Restigouche County, New Brunswick, Canada, and consists of combined lease and claim area of 5,082 ha. Located in the Miramichi Highlands, the sites are characterized by rounded and glacially scoured hills. Land use in the area is mainly for tourism, forestry, and mining. The property is accessible for exploration work and project development year-round and water is plentiful in nearby streams and creeks.

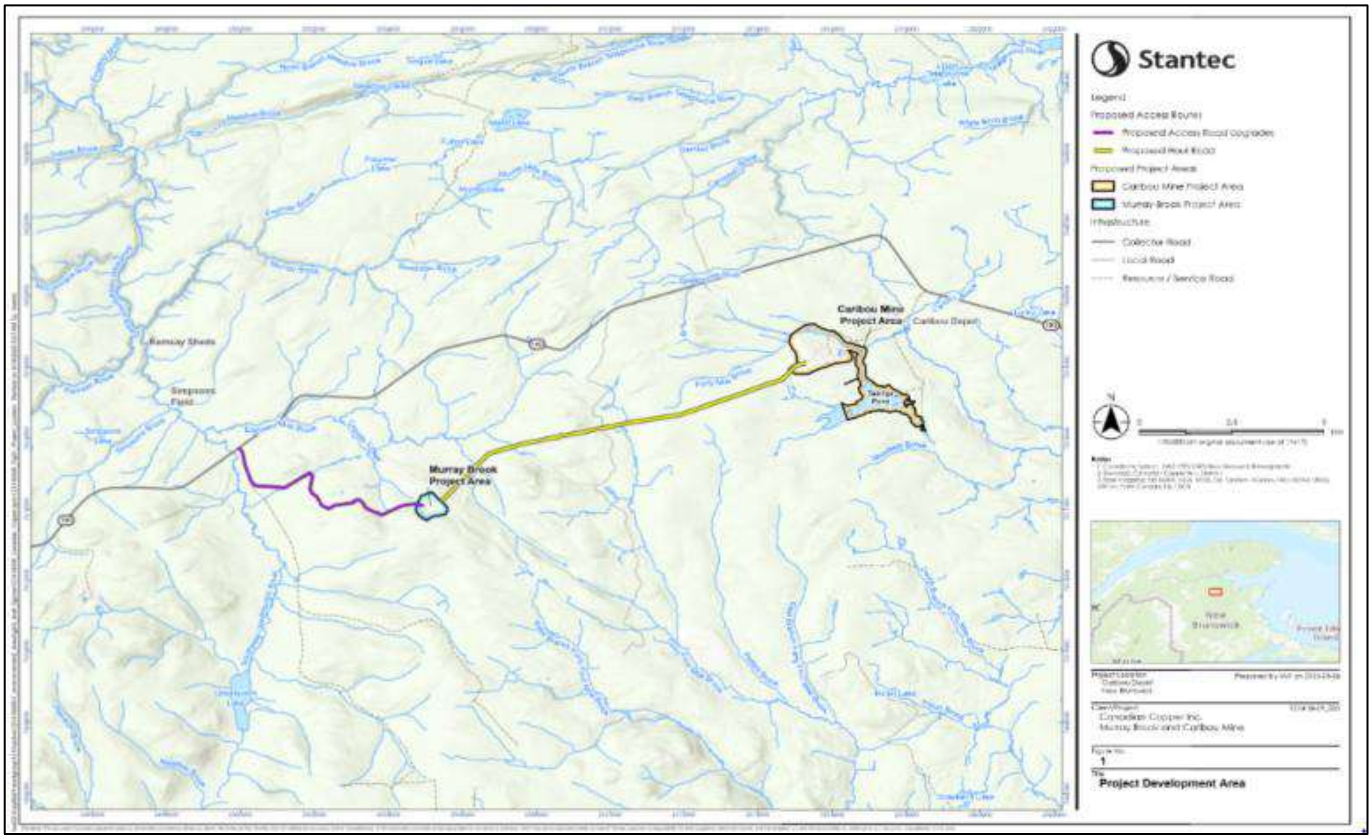
20.2 Baseline and Supporting Studies

A socio-environmental desktop study was completed by Stantec (2025) that focused on a study area that consisted of a 5 km buffer area around the proposed project. Field studies at the Murray Brook site have been initiated in 2025 including bird, bat and terrestrial wildlife surveys and are ongoing. The desktop sources of information included publicly available information obtained primarily from provincial government sources. The subject areas investigated included:

- provincially mapped wetlands, lakes, and watercourses, as well as provincially significant wetlands (PSW)
- protected federal and provincial areas
- areas of ecological significance

- heritage resources and elevated archaeological potential
- Species at Risk and Species of Conservation Concern (from the Atlantic Canada Conservation Data Centre)
- other environmental features of significance
- socioeconomic features.

Figure 20-1: Proposed Project Development Area



Source: Stantec (2025).

Other sources of information for this section included:

- meteorological data obtained by Ausenco from nearby ECCC weather station at Bathurst, NB
- groundwater seepage estimates from Golder (2018) based on limited hydrological information obtained from geotechnical boreholes from a nearby mining property (Restigouche)
- preliminary static ABA testing and geochemistry estimates of potentially acid generating (PAG) material were calculated by means of block model and assumption that any mineralized waste rock is considered PAG.

20.2.1 General Ecological and Physiographical Considerations

The study area is situated at the intersection of two ecoregions: the Northern Uplands Ecoregion, and the Highlands Ecoregion, each of these ecoregions consisting of a number of distinct ecodistricts. The Tetagouche Ecodistrict is transitional between the higher elevations and harsher climates of the Ganong Ecodistrict, located within the Highlands Ecoregion and the milder Upsalquitch Ecodistrict, located within the Northern Uplands Ecoregion. The above ecodistricts are characterized by several common characteristics including cool winters, higher elevations, and wetlands concentrated in low valleys, particularly in the floodplains of watercourses. The project is situated in the northern portion of the Ganong Ecodistrict, which is characterized by abundant balsam fir dominated forests and nutrient-poor soils.

20.2.2 Weather, Watercourses and Wetlands

The study area has a typically continental climate with cold winters with a mean temperature of -13°C in January. Extreme low temperatures of -30°C to -35°C are commonly reported. The Murray Brook area generally receives between 300 and 400 cm of snow annually for approximately 33% of its annual total of precipitation. In summer, average daytime highs vary between 20°C and 28°C. Spring and early summer are generally dry, but there is ample water during the growing season. The area records approximately 1,200 mm of rainfall a year, with the heaviest amounts falling during the summer months. Winds generally blow predominantly from the west and northwest in the winter months and from the south and southwest during warmer conditions with average wind speeds in the 12 to 20 km range.

The provincial database (GeoNB) indicates numerous watercourses, waterbodies, and wetlands throughout the study area. The two main watersheds include the Nepisiguit River Basin (Caribou Mine site) and the Restigouche River Basin (Murray Brook site). The study area does not contain any protected watersheds. The study area contains numerous mapped and unmapped watercourses some of which are close (less than 30 m) from the proposed project footprint.

The access road from Route 180 to the Murray Brook as well as the proposed haul road between the Murray Brook and Caribou sites crosses one and three mapped watercourses, respectively, the named watercourses being Forty Mile Creek and tributaries to Eighteen Mile Brook. Several watercourses are located near the Caribou mine site including Forty Mile Brook, the Nepisiguit River, and Caribou Lake.

Mapped wetlands throughout the study area are distributed in areas of relatively low elevation and topographical relief, primarily floodplains of watercourses. There are seven mapped wetlands within 30 m of the proposed project footprint. Wetlands in the study area classified mainly as forested and shrub swamps.

Until such time as site-based fisheries and wetland investigations are conducted, the identified water courses should be considered as fish habitat and frequented by fish.

20.2.3 Groundwater

Golder (2018) completed a seepage estimate for the proposed Murray Brook open pit mine for the then owners, Trevali Mining Corporation. The report presented preliminary groundwater seepage and direct seepage estimates for potential inflows into the proposed open pit. Using a steady-state analytical solution proposed by Marinelli & Niccoli (2000), Golder calculated an estimated average flow of 1,660 L/min into the fully developed pit. The study utilized available data including topographical geometry, preliminary final pit shell geometry, and four geotechnical drill holes (packer test data) from the nearby Restigouche mine.

Stantec (2025) reports that there are nine groundwater wells within the study area, none of which are located within the footprint of proposed development. Two of these wells are industrial non-drinking water wells located in the west of the study area and drilled in 2008. The remaining wells are domestic, for purposes of drinking water. Five of these domestic wells were drilled in 2014, and two were drilled in 2018.

Groundwater monitoring wells are located throughout the Murray Brook Deposit and surrounding area and are utilized to monitor groundwater quality as part of sampling and reporting requirements for Approval to Operate I-12407. There is quarterly reporting for the water quality of the groundwater monitoring wells. The results from these reports were not available for this report.

Groundwater monitoring wells were installed and monitored at the Caribou mine complex. The results of this monitoring program were not available for this report, nor is it known whether a hydrogeological model was ever developed for the site.

20.2.4 Geochemistry

Preliminary static ABA testing was reported to have been completed for the Murray Brook site utilizing exploration drill core. Utilizing this information and the geology block model, a preliminary estimate of the quantity of potentially acid generating (PAG) waste rock and soil material. This method is based on the presence of sulphide mineralization below mineralized material grade cut-off. All mineralized waste rock material was assumed to be PAG. The following assumptions were made for this purpose:

- Overburden is assumed to be non-acid-generating (NAG).
- Waste rock is assumed to be NAG.
- All materials within the Murray Brook low-grade PAG halo are assumed to be PAG.
- All mineralized blocks in the model are assumed to be PAG.
- The oxidized zone is PAG.

Based on the above assumptions the current estimate of PAG material for the purpose of infrastructure design at the Murray Brook site is 11.8 Mt of PAG (15%) waste rock out of 77.7 Mt of waste material mined. Once a detailed review of historical geochemistry test results and geological model is completed, additional geochemistry work may be completed. The development of geochemical source terms may be required for the purpose of effluent quality prediction and effluent treatment design.

20.2.5 Species at Risk

Stantec (2025) submitted an Atlantic Canada Conservation Data Centre (AC CDC) data request for rare or uncommon flora and fauna previously identified with the study area (5 km buffer). The AC CDC report identified 40 observations of 11 Species at Risk (SAR)[1]. These data are summarized in Table 20-1. It should be noted that the information provided regarding SAR is based on a desktop survey of what species may be present in the study area and future field fauna and flora habitat surveys are required to confirm current presence or absence. These field studies are reported to be underway.

Table 20-1: AC CDC Records of SAR Observed Within the Study Area

Scientific Name	Common Name	COSEWIC	SARA	NB SARA	S-Rank
Mammals					
Lynx canadensis	Canada Lynx	NAR	-	EN	S4
Birds					
Catharus bicknelli	Bicknell's Thrush	TH	TH	TH	S2B
Chaetura pelagica	Chimney Swift	TH	TH	TH	S2S3B, S2M
Riparia riparia	Bank Swallow	TH	TH	EN	S2B
Hirundo rustica	Barn Swallow	SC	TH	TH	S2B
Cardellina canadensis	Canada Warbler	SC	TH	TH	S3S4B
Contopus cooperi	Olive-sided Flycatcher	SC	SC	TH	S3B
Chordeiles minor	Common Nighthawk	SC	SC	SC	S3B, S4M
Euphagus carolinus	Rusty Blackbird	SC	SC	SC	S2S3B, S3M
Coccothraustes vespertinus	Evening Grosbeak	SC	SC	-	S3B, S3S4N, SUM
Vascular Plants					
Fraxinus nigra	Black Ash	TH	-	-	S3S4

Notes: Species classified under SARA, NS ESA, or COSEWIC have the following rankings: Endangered = EN Threatened = TH Vulnerable = VU Special Concern = SC Not at Risk = NAR; Species ranked as Critically Imperilled (S1), Imperilled (S2), or Vulnerable (S3) by the Atlantic Canada Conservation Data Centre (AC CDC 2023b) and recorded within 5 km of the project by desktop data source, where: S1: Critically Imperilled – Critically imperilled in the province because of extreme rarity (often 5 or fewer occurrences). May be especially vulnerable to extirpation. S2: Imperilled – Imperilled in the province because of rarity due to very restricted range, very few populations (6 to 20 occurrences or few remaining individuals). May be vulnerable to extirpation due to rarity or other factors. S3: Vulnerable – Vulnerable in the province due to a restricted range, relatively few populations (often 80 or fewer). S4: Apparently Secure – Uncommon but not rare; some cause for long-term concern due to declines or other factors (80+ occurrences). S5: Secure – Common, widespread, and abundant in the province. S#S#: A numeric range rank (e.g., S2S3) is used to indicate any range of uncertainty about the status of the species or community. B: Breeding – Conservation status refers to the breeding population of the species in the province. N: Nonbreeding – Conservation status refers to the non-breeding population of the species in the province. M: Migrant – Migrant species occurring regularly on migration at particular staging areas or concentration spots where the species might warrant conservation attention. Conservation status refers to the aggregating transient population of the species in the province. Source: Stantec (2025).

[1] Species at risk (SAR) include those listed under the federal Species at Risk Act (SARA), the New Brunswick Species at Risk Act (NB SARA), and the Committee on the Status of Endangered Wildlife in Canada (COSEWIC).

20.2.6 Critical Habitat

A search of the Government of Canada's Critical Habitat Species at Risk Database by Stantec (2025) indicates that the study area, including a portion of the project footprint, overlaps with an area of critical habitat[2] for Bicknell's Thrush (*Catharus bicknelli*).

The study area also intersects with provincially recognized habitat for Bicknell's Thrush, including both survival and recovery habitat. The project footprint doesn't overlap with this habitat directly except for a portion of the proposed road between Murray Brook and Caribou mine which is immediately adjacent to an area of recovery habitat. A finalized version of the draft Bicknell's Thrush recovery strategy and a more detailed management plan are under development by the province and are not currently publicly available.

The above information is based on desktop information only. Early engagement with federal and provincial authorities will help to determine a scope for field study that would provide a more detailed understanding of the presence of Bicknell's Thrush and implications for development within the project area. These field studies are reported to be currently underway.

20.2.7 Species of Conservation Concern

Stantec (2025) also requested that the AC CDC report review for Species of Conservation Concern (SOCC)[3] and the report identified 19 species within 5 km of the project footprint. The 19 species include 17 birds, one fish, and one vascular plant.

20.2.8 Managed Areas

There are several types of managed areas in New Brunswick. These include areas managed for various conservation goals, areas managed by provincial entities for recreational purposes, and areas owned and managed by non-governmental groups, typically also for conservation purposes.

Stantec (2025) requested a search of the Canadian Protected and Conserved Areas Database (CPCAD) and GeoNB data. The results revealed no protected areas or parks within the study area, no Important Bird Areas (IBAs), and no Key Biodiversity Areas (KBAs) within the study area. Stantec (2025) noted that two areas of ecological importance were identified within the study area:

- AC CDC records indicated the presence of the Southeast Upsalquitch Mixed Softwood Ecologically Significant Area (ESA) in the northeast of the study area.
- Deer Wintering Areas occur primarily along the east side of the study area.[4]

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[2] SARA defines critical habitat as "… habitat that is necessary for the survival or recovery of a listed species." (ECCC 2020).

[3] Species of Concern defined as those species that do not meet the definition of SAR (Section 20.2.3) but have an S- Rank of S1 (critically imperilled), S2 (imperilled), or S3 (Vulnerable; AC CDC 2025a).

[4] Deer Wintering Areas are Crown lands where forestry activities are managed to provide winter habitat for herds of White-tailed Deer (Odocoileus virginianus).

20.2.9 Socioeconomic, Cultural Resources, Community/Indigeous Engagement

20.2.9.1 Socioeconomic

Stantec (2025) conducted a desktop study that identified socioeconomic features within the study area. A summary of the results are as follows:

- The study area includes the relatively uninhabited communities of Simpsons Field and Caribou Depot, and Highway 180.
- Review of available aerial imagery shows the study area is situated in an area of industrial forestry use, with a patchwork of forested and regenerating stands and associated logging road networks.
- Caribou Wind Farm turbines are situated on the crests and upper slopes of Mount Jack and Mount Perley with the nearest wind turbine situated approximately 1.6 km from the project footprint.
- The project footprint is previously developed and includes pre-existing infrastructure associated with Caribou mines.
- Small-scale fishing, hunting, and recreational ATV and snowmobile use take place within the study area; there is a network of ATV trails and shelters in the area.
- There are apparently no widely used hiking trails or canoe routes in the study area.

Commercial infrastructure within the study area includes:

- A restaurant and accommodation facility situated on Highway 180.
- Mapped Maple Syrup Sugaries located approximately 4 km from Caribou deposit.
- The Island Lake Club (located just outside the study area) but accessed near Caribou mine with a number of cottages. Cottagers may use the study area for hunting and fishing.

As the project advances through feasibility-level design, environmental assessment, and permitting, continued meaningful engagement will need to be conducted with local rightsholders and stakeholders, including local landowners, First Nations, communities, businesses, and recreational land users. The purpose of this engagement is to understand and address local concerns and to communicate both the potential impacts and benefits to interested parties. The project is at an early stage in the engagement process.

20.2.9.2 Cultural Resources – Archaeology

The Archaeological and Heritage Branch (AHB), Province of New Brunswick identifies areas of archaeological potential within the province using their archaeological predictive model.

The model typically considers the following areas to have an elevated potential for heritage resources:

- Riparian areas within 80 m of watercourse shorelines.
- Confluences of two or more watercourses within 100 m of the watercourse confluence.
- Other areas of land based on known or registered heritage resources or such areas as paleo-shorelines.

Based on the above, it should be assumed that the archaeological predictive model would indicate these archaeological buffers (i.e., 80 and 100 m) apply to watercourses within the project footprint.

Once further details on project description and development become available, the model should be requested and an archaeological walkover survey be conducted by a professional archaeologist operating under a Provincial Archaeological Field Permit to help confirm the presence or absence of heritage resources.

20.2.9.3 First Nations Engagement

New Brunswick is home to two main Indigenous nations: the Mi'kmaq and the Maliseet (also known as Wolastoqiyik). These nations have a long history in the region, with evidence of their presence dating back over 10,000 years. New Brunswick is home to 15 First Nations, with 31 reserves, primarily belonging to the Mi'kmaq and Maliseet nations. The Mi’kmaq nation is traditionally located in the eastern and northern parts of the province and include nine (9) separate communities. These communities are represented in part by the organization Mi'gmawe'l Tplu'taqnn. The Mi’kmaq nation closest to the project is known as the Pabineau First Nation (also known as Oinpegitjoig L’Noeigati). Pabineau is located approximately 10 km south of Bathurst, NB, and approximately 55 km west of the Murray Brook project.

In October 2024, Canadian Copper signed a non-binding Memorandum of Understanding (MOU) with Pabineau First Nation[5]. The company’s press release states:

> *“Canadian Copper and the Pabineau First Nation (PFN) wish to cooperate, communicate openly and directly, and explore opportunities to work collaboratively regarding mineral development in New Brunswick. PFN has an extensive network of business relationships within both First Nation and non-indigenous communities including signatory agreements with the deepwater Port of Belledune and the former Caribou Mine.”*

The MOU is a significant step for Canadian Copper in First Nation engagement that allows for the development of open lines of communication and mutual collaboration so that PFN can benefit economically through job creation, contracting opportunities including potential business and investment partnerships.

Historically, Trevali, as the previous owner of the Caribou mine, entered into cooperation agreements with a number of Mi’kmaq First Nations (including Pabineau First Nation) in May 2011 and then again in February 2017. However, since the closure of operations in August 2022, the subsequent bankruptcy of Trevali, and the custody and operation

[5] see October 8th, 2024 press release from Canadian Copper (https://canadiancopper.com/)

of the Caribou facilities by the Province of New Brunswick in 2023, it is assumed that these cooperation agreements and associated impact benefit agreements are no longer in effect, and may require renegotiation.

20.3 Permitting and Regulatory Framework

This section summarizes the current permits in place for the Murray Brook project and the federal and provincial legislation and associated permits, licenses and approvals that will apply or potentially will apply to the construction and operations of the project, as currently proposed.

Depending on its development strategy, the project could be subject to mostly provincial and local regulatory requirements. The principal provincial legislation related to the environmental impact assessment (EIA) and approval of mining projects are the *Mining Act* and General Regulation 86-98, and the *Clean Environment Act* and EIA Regulation 87-83, and Water Quality Regulation 82-126. Relevant key federal legislation includes the *Canadian Environmental Assessment Act*, *Fisheries Act* and a subsection of the *Fisheries Act*—Metal and Diamond Mining Effluent Regulations.

20.3.1 Existing Permits

Based on a review of the available information, the Murray Brook site is operating under Approval to Operate I-12407, issued to Canadian Copper, which is valid until March 31, 2029, and the Caribou mine site is operating under Approval to Operate I-12715 issued to NB DNRED, which is valid until March 29, 2028.

The current Caribou mine TMF dam raise will be completed by the Province of New Brunswick under pre-existing permits/agreements with the New Brunswick Department of Environment and Local Government (NBDELG) and will provide storage capacity for tailings generated by mineralized material milled from the Murray Brook site for three years after completion of the dam raise. The provincial government, after taking over the property in 2023, have been operating the site and keeping it in compliance. The permitting, environmental, and social aspects regarding the use of the Caribou facilities are not discussed in this report in consideration that those aspects and assigned responsibilities would be determined following detailed investigations and final agreements with the Province of New Brunswick.

20.3.2 Future Permitting Considerations – Federal Regulations and Anticipated Approvals

The key federal regulations that regulate mining projects in New Brunswick include:

- *Impact Assessment Act*
- *Fisheries Act* and Amendment to Schedule 2 of the Metal and Diamond Mining Effluent Regulations
- *Species at Risk Act* (as related to critical habitat)
- *Migratory Birds Convention Act.*

20.3.2.1 Impact Assessment Act (IAA)

The requirement and applicability of the federal IAA process for a mining project is based on the Physical Activities Regulations under the IAA (known as the project list). Based on a review of these requirements and based on project scope as currently understood, the project does not meet the criteria for a designated project and would not be required to undergo the federal impact assessment process, as the proposed. Potential future triggers for an IAA would include the following:

- A production capacity increase to 5,000 t/d, or
- an expansion of footprint required for additional waste rock storage or tailings management facility be beyond 50% of the existing site, an Impact Assessment under the IAA would be required.
- However, it is noted that in some cases, the federal Minister of Environment and Climate Change Canada (ECCC) has the authority to require a project that is not included in the Physical Activities Regulations be designated to undergo an Impact Assessment under the IAA. The justification for such a requirement would typically be the result of Indigenous or climate change concerns.

20.3.2.2 Fisheries Act and Amendment to Schedule 2 of the Metal and Diamond Mining Effluent Regulations

In effect since June 2019, Canada's modernized *Fisheries Act*, RSC 1985, c. F-14 provides protection for all fish and fish habitats. Where works may cause the harmful alteration, disruption, or destruction (HADD) of fish habitat, authorization from Fisheries and Oceans Canada (DFO) under Section 35 of the *Fisheries Act* may be required. If the proposed mine infrastructure associated with a proposed project, will impact fish-bearing water, then a Fisheries Authorization and Fish Habitat Compensation Plan may be required. Project related activities which may result in HADD include in-water work related to project infrastructure (e.g., watercourse crossings) as well as activities which will result in in-direct effects such as reduction in flow due to pit dewatering or redirection of surface water runoff.

In New Brunswick, the review of projects involving in-water works (such as watercourse crossings) is coordinated by NBDELG through a provincial watercourse and wetland alteration permitting process. DFO representatives review the WAWA permit application to determine the likelihood for HADD and issue a Letter of Advice to NBDELG and/or the proponent indicating whether authorization is required.

For mines, the requirements of Section 36 of the *Fisheries Act* are further defined and regulated by the Metal and Diamond Mining Effluent Regulations (MDMER). Mining projects in New Brunswick may also require authorizations from ECCC (Government of Canada 2022) under Section 36 of the *Fisheries Act*, which prohibits the deposition of deleterious substances into water frequented by fish. If the deposition of a deleterious substance into waters frequented by fish is due to the proposed establishment of a tailings impoundment area or other mine waste storage facility, an authorization from ECCC, in the form of a Schedule 2 (SOR/2002-222) amendment to the MDMER under Section 36 of the *Fisheries Act*, may be required. The depositing of deleterious substances produced by mines (e.g., tailings, waste rock) is addressed by the requirement for new tailings impoundment in waters frequented by fish to be added to Schedule 2 of the MDMER by regulatory amendment.

As a prerequisite for obtaining a Schedule 2 MDMER amendment, an alternatives assessment must be conducted in accordance with the "Guidelines for the Assessment of Alternatives for Mine Waste Disposal" issued by ECCC (2016). The specific requirements will require confirmation with regulatory agencies. The timeline for the Schedule 2 MDMDER amendment process is typically two years, however, there is the potential to reduce this timeline by early and comprehensive fisheries field programs and early engagement with ECCC and DFO.

Operation of the TMF to date at the Caribou mine site does not require a Schedule 2 MDMER amendment. This is in consideration of the existing tailings facility and the raising of the dam by the Province of New Brunswick to accommodate an additional three years of tailings deposition. However, based on the proposed mining schedule and run-of-mine tonnage, it is anticipated that the project would have a life of mine of up to 13.2 years. Should Canadian Copper require additional or alternative tailings storage to extend the life of the project beyond three years, an amendment to Schedule 2 of the MDMER will be required if the facility overprints on waters frequented by fish.

Additionally, should development (i.e., waste rock storage area and/or TMF) at either the Murray Brook or Caribou mine sites require overprint on waters frequented by fish then a Schedule 2 amendment would also be required.

20.3.2.3 Species at Risk and Critical Habitat

The federal *Species at Risk Act* (SARA) is administered by Environment and Climate Change Canada (ECCC) with the intent to protect species from extirpation or extinction as a result of human activity. The purpose of SARA is to prevent SAR from becoming threatened or endangered and to allow for recovery of species that are considered threatened, endangered, or extirpated. The more than 300 wild plant and animal species listed in Schedule 1 of SARA are afforded special measures to protect them and assist in their recovery.

As described in Section 20.2.6, the Murray Brook study area and portions of the project footprint directly overlaps with the mapped critical habitat of the Bicknell's Thrush. It is anticipated that a SARA permit will be required should project activities destroy this habitat; destruction of this habitat is considered to occur when it is degraded either permanently or temporarily to the point that it can no longer be utilized when the species requires it (e.g., clearing, grubbing, fragmentation, and noise). In such cases, as the presence of a SAR within or proximal to the project footprint, early engagement with regulators is recommended to reduce delays to the project schedule. In addition, site-based field work and data collection is required so as to further characterize the critical habitat and to consider whether project footprint adjustments may reduce potential adverse impacts. Field studies are reported to be currently underway.

20.3.2.4 Migratory Birds Convention Act (MBCA)

In New Brunswick, the Migratory Birds Regulations, 2022, under the *Migratory Birds Convention Act*, 1994, protect migratory birds, their nests, and eggs on federal, provincial, and private lands. The 2022 regulations include prohibitions on depositing harmful substances in areas frequented by migratory birds and damaging or disturbing nests containing a live bird or viable egg. Specific protections are provided for 18 bird species (listed in Schedule 1) that reuse nests with mechanisms to deem nests abandoned based on specific criteria. To comply with the provisions of the MBCA, construction activities that require the removal of trees and ground vegetation are normally conducted outside of migratory bird breeding season (typically April 15 to August 30 of each year).

Although the project footprint is largely developed, given the apparent habitats available in the study area for migratory bird species, field surveys are needed to determine potential for interaction of migratory birds with the project, and to inform the identification of appropriate mitigation measures that are potentially required. Field studies are reported to be currently underway.

20.3.3 Future Permitting Considerations – Provincial Regulations and Anticipated Approvals

The provincial acts and regulations that regulate mining projects in New Brunswick include:

- Environmental Impact Assessment Regulation – *Clean Environment Act*
- Water Quality Regulation – *Clean Environment Act*
- *New Brunswick Species at Risk Act*
- *New Brunswick Clean Water Act*
- *New Brunswick Heritage Conservation Act*
- *New Brunswick Mining Act.*

20.3.3.1 Environmental Impact Assessment (EIA) Regulation – Clean Environment Act

The provincial Environmental Impact Assessment Regulation – *Clean Environment Act* sets out the requirements for an environmental impact assessment (EIA) in New Brunswick. Schedule A Section (a) of the regulation states that all commercial extraction or processing of a mineral as defined in the *Mining Act* is subject to an EIA registration.

In addition, Subsection 3(2) of the Environmental Impact Assessment (EIA) Regulation – *Clean Environment Act*, requires that the modification, extension, abandonment, demolition or rehabilitation of undertakings specified in Schedule A are also considered as undertakings and must be registered for review including the following activities:

- a permanent increase of 25% percent or more in the physical footprint (area occupied on the ground) of an undertaking
- a permanent increase in existing rates of emissions of any contaminant as defined by Section 1 of the *Clean Environment Act*
- a permanent increase in existing total quantities or concentrations of emissions or discharges of any contaminant as defined by Section 1 of the *Clean Environment Act*
- a permanent change in the horizontal or vertical location at which a contaminant will enter the environment
- a change in the effluent treatment method or technology
- a permanent increase in throughput that has the potential to result in environmental impacts that were not previously considered: the product type, the actual production or processing volume; the theoretical production or processing capacity; the method of production or processing; or the method of waste treatment or disposal
- a change to the purpose of a facility.

At minimum, it is likely that an EIA registration will be required for the Murray Brook project. An EIA registration will require adequate environmental baseline field studies and predictive modelling, followed by several months to prepare and submit an EIA Registration document and then iterative discussions with the Provincial government and likely requests for additional information. Following EIA Registration submission, the EIA Branch coordinates a review of the registration document by a Technical Review Committee (TRC), currently scheduled for July 9th, 2025. Based on this review, the Minister of Environment and Climate Change determines whether a "Comprehensive Review" is required, or, alternatively, if the project can proceed subject to conditions of approval. Simultaneous to this effort, there should be a concerted effort regarding local community and Indigenous engagement and documentation of engagement record, concerns, and how those concerns are addressed.

Possible outcomes of the Determination Review process are a Certificate of Determination (an approval for the undertaking to proceed with conditions); Project Denial; or a decision that further information and study is required (via a Comprehensive Review). It is likely that the Murray Brook project would receive a Certificate of Determination. as the site is an existing historical mining site, though this cannot be guaranteed.

The current Caribou mine TMF dam raise is being completed by NBDELG under pre-existing permits/agreements and will provide storage capacity of three years after completion. The proposed mine schedule anticipates a life of mine of up to 13.2 years. Therefore, although there will be enough capacity in the TMF for the first three years, additional tailings capacity will be required in the future that will require additional permitting at a later date. This approach will need to be discussed with regulators.

20.3.3.2 Water Quality Regulations

Following the fulfilment of the EA Regulation requirements, pursuant to the Water Quality Regulation (Regulation 82-126) under the *Clean Environment Act*, the construction and operation of the TMF requires an Approval to Construct/Operate to be obtained and/or the current Approval to be amended to include the proposed TMF and Murray Brook site. The Approval to Operate is issued pursuant to paragraph 8(1) of the Water Quality Regulation, and defines terms and conditions (such as discharge limits, testing and monitoring requirements, reporting requirements, and the like) that a facility must comply with as part of its operation in order to remain in compliance with the regulations.

Currently, the Caribou mine site is operating under Approval to Operate I-12715 issued to NB DNRED, which is valid until March 29, 2028. The Murray Brook site is operating under Approval to Operate I- 12407, issued to Canadian Copper, which is valid until March 31, 2029. It is unclear if the project could be approved as a revision to these existing Approvals to Operate or if a separate consolidated Approval to Construct/Operate would be required. Early discussions with NBDELG will help to clarify this point.

20.3.3.3 New Brunswick Species at Risk Act and Recovery and Survival Habitat

Species at risk in New Brunswick are protected under the New Brunswick *Species at Risk Act* (NB SARA). NB SARA is administered by the NB DNRED and has two regulations: the List of Species at Risk Regulation 2013-38 and the Prohibition Regulation – *Species at Risk Act* (2013-39). Species listed within Schedule A (as contained in the Prohibition Regulation – *Species at Risk Act* (2013-39)) are protected. Of the species listed on Schedule A of the Prohibition Regulation, only Canada Lynx was identified in the Stantec (2025) desktop review of Species at Risk (refer to Section 20.2.5).

Portions of the study area intersect with provincially recognized habitat for Bicknell's Thrush, including both recovery and survival habitat. Under specific circumstances, the Minister of the NB DNRED may issue a permit for activities that would normally not be allowed in areas under a habitat designation. To obtain this permit, the proponent must assess the options for reasonable alternatives. If no reasonable alternatives are identified, then it would need to be demonstrated that the proposed activity will not jeopardize the survival of the species and will have only an incidental impact. In addition, reasonable measures would need to be taken to minimize any impact. In addition, the permit may also contain conditions requiring the proponent to rehabilitate damaged or destroyed habitat, enhance habitat in another area, or provide financial compensation.

Site-based field work and early engagement with regulators is necessary to reduce potential delays related to habitat disturbance. This field work and engagement would be conducted in conjunction with the work associated with the federal critical habitat described previously.

20.3.3.4 New Brunswick Clean Water Act

Watercourses and wetlands are protected in New Brunswick under the *Clean Water Act*. Activities that could alter watercourses or wetlands are regulated under the Watercourse and Wetland Alteration (WAWA) Regulation of the *Clean Water Act*, administered by the New Brunswick Department of Environment and Local Government (NBDELG). Under the WAWA Regulation, permits are required for activities including vegetation clearing, soil excavation, construction, watercourse crossings, water withdrawal, or landscaping, within 30 m of a watercourse or wetland, or within the bed and banks of a watercourse or boundary of a wetland. If work results in permanent changes to a wetland, off-setting compensation measures are likely to be required.

In New Brunswick, NBDELG coordinates the review of WAWA permit applications involving in-water work with DFO. DFO representatives review the WAWA permit application to determine the likelihood for HADD and issue a Letter of Advice to NBDELG and/or the proponent indicating whether authorization is required.

20.3.3.5 New Brunswick Heritage Conservation Act

Heritage resources in New Brunswick are regulated under the *Heritage Conservation Act* (2010). The regulatory management of provincial heritage resources falls under the New Brunswick Department of Tourism, Heritage, and Culture, and is administered by its Archaeology and Heritage Branch. The Province of New Brunswick provides guidance for conducting heritage assessments, such as the Guidelines and Procedures for Conducting Professional Archaeological Assessments in New Brunswick (AHB 2012). All work related to archaeological impact assessments must be completed under a permit (archaeological field research permit) issued by the province to the individual conducting the assessment. The study area may contain areas of archaeological potential that will require further investigation by an archaeologist.

20.3.4 Environmental Management and Monitoring Plans

Water management controls and facilities are described in Section 18 and consists of water treatment and collection ponds, diversion and collection ditches, water supply, waste rock storage facilities for PAG and NAG waste rock (for Murray Brook site), and TMF facilities design and operation (for Caribou mine site).

An existing Environmental Management System (EMS) was established previously for the Caribou mine site and is currently implemented. Under the EMS, there is a comprehensive list of environmental and social management operating procedures (ESMOPs) used by mine personnel under the supervision of the government of New Brunswick to support for the ongoing site activities which involve environmental monitoring, compliance, incident reporting, waste management, water treatment, and regulatory reporting requirements, among others. The results of ongoing monitoring for the Caribou mine site were not available for review. It is recommended that a request is made for available monitoring data and a comprehensive review be performed to assess environmental and social performance for past and more recent site activities at the Caribou mine site. As part of the EIA development, the EMS and ESMOPs will be reviewed for applicability for the Murray Brook project and revised/reviewed in accordance with current best practices.

In general, the list of anticipated environmental management and monitoring plans will be required for EIS and permitting phases of the project.

- Surface Water Management and Monitoring Plan
- Groundwater Contingency and Monitoring Plan
- Fish and Aquatic Effects Monitoring Plan
- Air Quality Management and Monitoring Plan
- Surface Erosion and Sediment Control Plan
- Hazardous Materials Management Plan
- Heritage Management Plan
- Access Management Plan
- Emergency Response Plan
- Spill Contingency Plan
- Mine Waste, Tailings, and ML/ARD Management Plan
- Soil Handling Management Plan
- Ecosystems Management Plan
- Vegetation Management and Monitoring Plan
- Wetland Monitoring Plan
- Invasive Plant Management and Monitoring Plan
- Wildlife Management and Monitoring Plan
- Waste Management Plan
- Water Treatment Plan
- Stakeholder and First Nations Communication Plan
- Occupational Health and Safety Plan
- Reclamation and Closure Plan.

20.4 Closure and Reclamation Planning

In New Brunswick, the exploration and development of Crown-owned minerals falls under the *New Brunswick Mining Act*. The mining lease approval process is led by NB DNRED. The review of a mining lease approval application can be conducted in tandem with an EIA Determination however a mine lease approval cannot be granted until EIA approval is received.

In accordance with the New Brunswick Regulation 86-98-General Regulation of the *Mining Act*, mining developments require an approved Program for the Protection, Reclamation, and Rehabilitation of the Environment (also known as a

Reclamation Plan). The Reclamation plan will specify the approach with respect to the progressive rehabilitation, decommissioning, removal, and disposal of site equipment and structures, and for site rehabilitation, closure, and care and maintenance of remaining facilities. It will also contain measures to achieve targeted environmental goals and will include a contingency to allow for shutdown during the anticipated project life, if required. A preliminary or conceptual Reclamation Plan is typically provided for review along with the EIA Registration document.

20.4.1 Closure and Reclamation – Caribou Mine Site

20.4.1.1 Caribou Mill and Associated Infrastructure

Reclamation for the Canadian Copper assets at the Caribou mine site include demolition of the existing water treatment plant, milling complex, laboratory and administration building as well grading and revegetation of these areas. It is understood that other areas and liabilities are the responsibility of others as part of Canadian Coppers limited liability agreement discussions with the Province of New Brunswick.

20.4.1.2 TMF Reclamation, Water Treatment and Monitoring

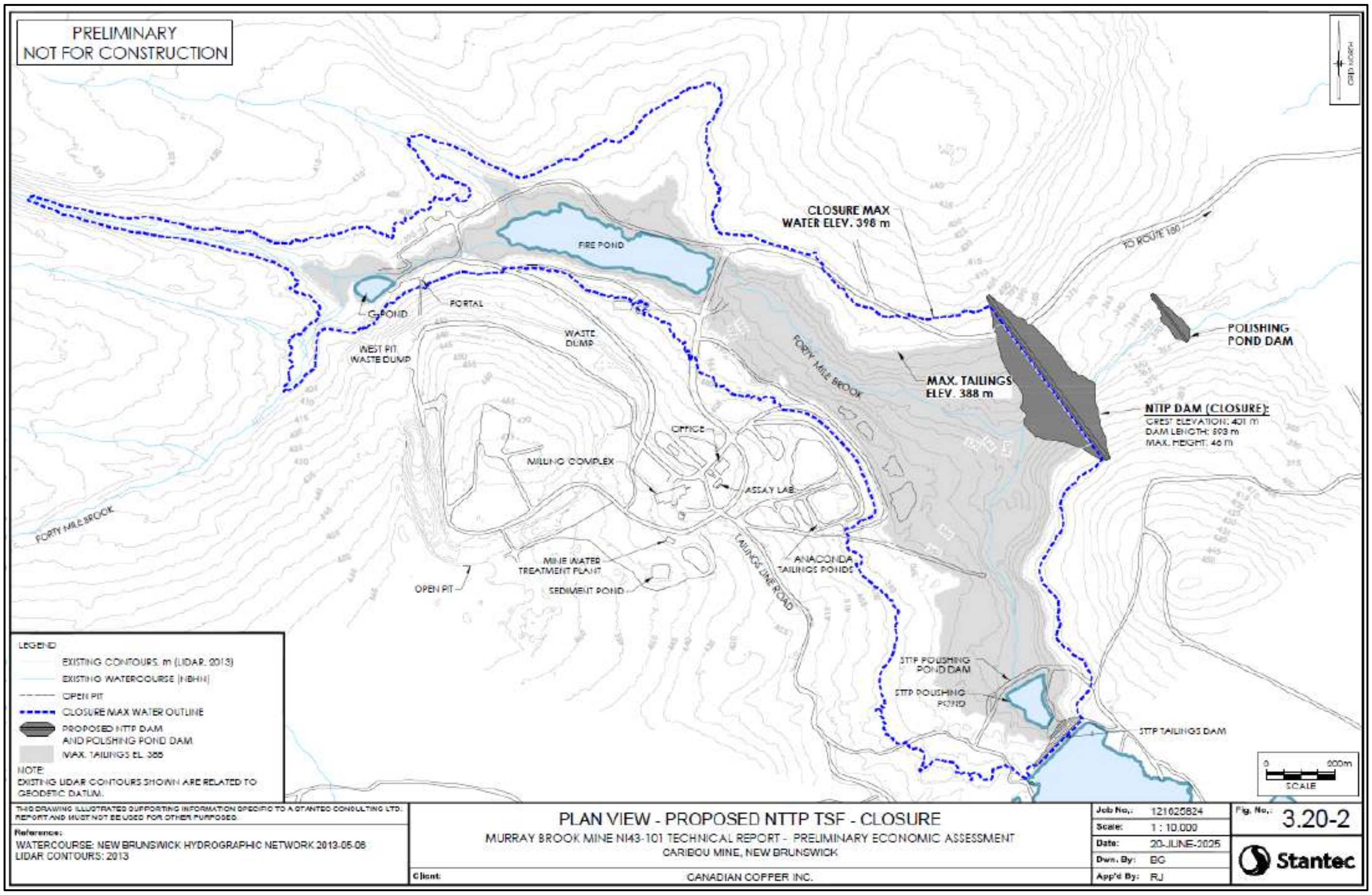
Due to the acid-generating nature of the tailings, and in alignment with the existing closure strategy for the Caribou mine, the proposed closure concept for the TMF areas includes raising the North Tributary tailings pond dam to a final crest elevation of 401 m, with a closure water level set at 398 m. This approach is intended to provide an enhanced water cover over the tailings and to submerge historical liabilities such as waste rock, legacy tailings, and the underground mine portal to reduce impacts from these sources.

The dam will be constructed as a downstream raise with similar crest widths and slopes as the operational stages of the North Tributary tailings pond. The tailings management facilities will provide water cover over the tailings long-term. A layout of the proposed closure configuration of the tailings pond is included in Figure 20-2.

Reclamation activities related to the tailings management facilities at this stage includes various upgrades to dams and sludge cells at the site which are related to water treatment or are connected to the North Tributary tailings pond. Long-term care and maintenance, and monitoring including Dam Safety Inspections, Dam Safety Reviews and periodic upgrades to the structure are also included.

Perpetual water treatment has been assumed as part of the closure plan, unless future monitoring and site conditions demonstrate that treatment is no longer necessary.

Figure 20-2: Layout of North Tributary Tailings Pond – Closure



Source: Stantec, 2025.

20.4.1.3 Caribou Mine Mill and TMF Closure Costs

Closure costs for the Caribou mine site includes demolition of select buildings, general site grading, and revegetation of disturbed areas by Canadian Copper. Closure costing for the Caribou mine tailings management facility includes various dam upgrades, maintenance, inspections, monitoring and water treatment in perpetuity which is assumed to be 100 years.

The total estimated closure costs related to these items at the Caribou mine site including these long-term costs are estimated at $219.6 million. Of the total, $171.9 million of the total cost is related to water treatment costs in perpetuity (100 years). With the exception of water treatment costs, the estimate includes a 10% for engineering and a 20% contingency.

20.4.2 Closure and Reclamation – Murray Brook Site

The Murray Brook mineral lease is the site of past production from open-pit mining of the oxidized surficial portion of the Murray Brook deposit. A site assessment by Stantec (2012) for VMC reports that the liability associated with the Murray Brook property consists of short-term monitoring liability and long-term restoration liability. The current liability is for monitoring rather than restoration of fish habitat in nearby Gossan Creek and Copper Creek. First VMC and now Canadian Copper have provided a C$2 million irrevocable letter of credit to the New Brunswick government as security for this liability.

The estimated closure cost for the proposed Murray Brook site will need to be updated in future to include the following preliminary list of activities:

- Flooding of open pit (calculation of time for pit to fill with water to be based on inputs of groundwater and surface water).
- Water treatment to continue until pit water quality including all mine contact wate meets applicable water quality criteria at which time water diversions and containment ponds will be decommissioned, and the pit water overflow will be allowed to discharge naturally (via a closure spillway).
- At closure, PAG rock will be managed by rehandling into the pit to keep it permanently submerged in the pit lake or capping it with low permeability cover to reduce seepage and oxygen infiltration.
- NAG waste rock stored on the surface will be capped with soil and revegetated.
- Buildings on the infrastructure pad will be dismantled, removed, and the pad re-contoured and revegetated.
- Mine infrastructure such as infrastructure pads and waste rock piles will be revegetated.

21 CAPITAL AND OPERATING COSTS

21.1 Introduction

The capital and operating cost estimates presented in this PEA provide substantiated costs that can be used to assess the preliminary economics of the Murray Brook Project. The estimates are based on the development of an open pit mine and related mining infrastructure at the Murray Brook site, and the recommissioning of an existing process plant, tailings management facility, and related infrastructure, as well as Owner's costs and contingency.

All capital and operating cost estimates are reported in Canadian dollars (currency abbreviation: CAD; symbol: C$), with no allowance for escalation or exchange rate fluctuations. An exchange rate of 0.75 (CAD to USD) and 0.65 (CAD to Euro) has been applied as necessary.

The capital cost estimate conforms to Class 5 guidelines for a preliminary economic assessment level estimate with a ±50% accuracy according to the Association for the Advancement of Cost Engineering International (AACE International). The capital cost estimate was developed in Q2 2025 Canadian dollars based on Ausenco's in-house database of projects and studies as well as experience from similar operations.

21.2 Capital Costs

21.2.1 Basis of Estimate

The total initial capital for the Murray Brook Project is $63.7 million and the life-of-mine sustaining cost is $48.9 million. The $63.7 million includes $57.7 million as summarized by WBS in Table 21-1, plus an additional $6 million to be used to exercise an option to purchase the Caribou process plant and related infrastructure from the New Brunswick government with a closing date of July 11, 2026.

The capital cost estimate was developed in Q2 2025 Canadian dollars based on Ausenco's in-house database of projects and studies as well as experience from similar operations. Based on the methodology used to develop the capital estimate and the conceptual level of engineering definition, the estimate has an accuracy of ±50%, which is in accordance with the Association for the Advancement of Cost Engineering International (AACE International) guidelines for a preliminary economic assessment.

Data input for the estimates has been obtained from numerous sources, including the following:

- mining schedule
- repair and replacement costs identified during a site visit and plant restart review by Inteloc
- contractor quotes received for mining, hauling, drill/blast and crushing
- mechanical equipment costs determined from first principles and Ausenco's database of historical projects

- costs for concrete, steel, electrical, instrumentation, in-plant piping, and platework factored by benchmarking against similar projects with equivalent technologies and unit operations
- topographical information
- engineering design at a preliminary economic assessment level by Ausenco, P&E, and Stantec.

Table 21-1: Summary of Capital Cost Estimate

WBS	WBS Description	Capital Cost ($M)	Sustaining Cost ($M)	Total Cost ($M)
1000	Mine	29.1	5.4	34.5
2000	Water Treatment	0.2	0.0	0.2
3000	Process Plant	7.2	2.8	10.0
4000	On-Site Infrastructure	0.9	0.0	0.9
5000	Off-Site Infrastructure	0	29.9	29.9
Total Direct Costs		**37.4**	**38.1**	**75.5**
6000	Indirects	5.8	0.1	5.9
7000	Project Delivery	4.2	4.5	8.7
8000	Owner's Costs	2.1	0.0	2.1
Total Indirect Costs		**12.1**	**4.6**	**16.7**
Total Direct + Indirect Costs		**49.5**	**42.6**	**92.2**
9000	Contingency	8.2	6.3	14.5
Total Capital Cost Excluding Mill Purchase		**57.7**	**48.9**	**106.6**
Mill and Mill Infrastructure Purchase Cost		**6.0**	**0.0**	**6.0**
Total Capital Cost Including Mill Purchase		**63.7**	**48.9**	**112.6**

21.2.2 Mine Capital Costs (WBS 1000)

The mining capital costs are summarized in Table 21-2 and discussed in the following sections.

Table 21-2: Mine Capital Costs

WBS	WBS Description	Capital Cost ($M)	Sustaining Cost ($M)	Total Cost ($M)
1000	Mine	-	-	-
1100	Haul Road	4.4	4.4	8.8
1300	Mine Site Water Management and Treatment	5.6	0.0	5.6
1500	Mining	19.2	1.0	20.2
	Total Mining Costs	**29.1**	**5.4**	**34.5**

21.2.2.1 Haul Road Capital Costs (WBS 1100)

Construction of the 17 km haul road from the Murray Brook open pit to the Caribou process plant is estimated at $8.8 million and will be constructed by the mining contractor. Half of the cost has been included as initial capital in Year -1. The other half of the cost has been included in sustaining capital over a five-year period in Years 2 to 6 and is assumed to be payments to a critical minerals infrastructure fund (CMIF) application that will cover half of the haul road cost.

21.2.2.2 Critical Minerals Infrastructure Fund

The project assumes the utilization of the critical minerals infrastructure fund (CMIF). The CMIF is intended to support infrastructure projects that enable the development and expansion of critical minerals in Canada. No contracts or agreements have been obtained regarding the use of the fund, but a review of the CMIF requirements indicates that a haul road between the Murray Brook site and the Caribou site would allow for a 50% contribution of the infrastructure cost. Therefore, the initial capital of $4.4 million for the haul road is based on a cost of $8.8 million with $4.4 million of contributions from the government. The contribution was assumed to be a repayable contribution based on CMIF documentation, to be repaid once the project has positive cash flow.

21.2.2.3 Mine Site Water Management and Treatment Capital Costs (WBS 1200)

Mine site water management and treatment capital costs are based on quotes of similar sized water treatment equipment from 2025.

21.2.2.4 Mining Capital Costs (WBS 1500)

It is assumed the Murray Brook mining operations will be performed by a mining contractor. All drilling, blasting, loading, hauling operations, and equipment maintenance will be performed by the mining contractor. The Owner will supervise the contractor with a team of technical staff. The mining capital cost has been subdivided into three areas; (1) mining equipment including pick-up trucks for mining staff; (2) office supplies, software, and surveying equipment; and (3) capitalized pre-stripping costs. As shown in Table 21-3, the initial mine capital cost estimate is $19.1 million and the sustaining capital cost estimate is $1.0 million. The total life-of-mine mining initial and sustaining capital cost is estimated at $20.1 million.

Table 21-3: Mining Capital Cost Summary (WBS 1500)

Description	Initial ($M)	Sustaining ($M)	Total ($M)
Mining Equipment	0.3	1.0	1.3
Office Supplies, Software and Surveying Equipment	0.4	-	0.4
Capitalized Pre-stripping Costs	18.4	-	18.4
Total	**19.1**	**1.0**	**20.1**

21.2.3 Water Treatment and Process Plant Capital Costs (WBS 2000 and 3000)

Water treatment and process plant costs are summarized in Table 21-4 and described in the following subsections. Direct costs include all contractors' direct and indirect labour, permanent equipment, materials, freight, and mobile equipment associated with the physical construction of the areas.

Table 21-4: Process Capital Costs

WBS	WBS Description	Capital Cost ($M)	Sustaining Cost ($M)	Total Cost ($M)
2000	**Water Treatment**	**0.2**	**0.0**	**0.2**
2200	Caribou Water Treatment Plant (Area 22)	0.2	0.0	0.2
3000	**Process Plant**	**7.2**	**2.8**	**10.0**
3100	Grinding (Area 31)	0.3	0.0	0.3
3200	Lead Flotation (Area 32)	0.1	0.0	0.1
3300	Zinc Flotation (Area 33)	0.7	0.0	0.7
3400	Cu Flotation (Area 34)	3.6	0.0	3.6
3500	Filter Press and Concentrate Handling (Area 35)	0.4	0.0	0.4
3600	Feed and Feed Storage Bins (Area 30)	0.5	0.0	0.5
3800	Reagents (Area 38)	1.0	0.0	1.0
3900	Process Plant Services and Common (Area 39)	0.6	2.8	3.4
	Total Processing Costs	**7.4**	**2.8**	**10.2**

21.2.3.1 Caribou Water Treatment (WBS 2000)

Capital costs under WBS 2000 include the replacement of two reactor tanks and the repair of one transfer pump.

21.2.3.2 Process Plant Capital Costs (WBS 3000)

Process plant costs are summarized in Table 21-4 and described in the following sections. Direct costs include all contractors' direct and indirect labour, permanent equipment, materials, freight, and mobile equipment associated with the physical construction of the areas.

21.2.3.2.1 Process Plant – Excluding WBS 3400

Capital costs under WBS 3000, excluding WBS 3400, include the repair and clean-out of existing mechanical equipment.

21.2.3.2.2 Process Plant - WBS 3400

Capital costs under WBS 3400 include completing the installation of a copper circuit that was previously partially installed at the Caribou process plant. The completed circuit will include:

- flotation cells
- associated pumping and piping
- instrumentation
- electrical equipment
- additional concentrate storage.

The flotation cells have been purchased and physically placed into the installation location in the building; however, the installation of associated piping, pumping, instrumentation, and electrical equipment has not been completed.

21.2.4 On-Site and Off-Site Infrastructure Capital Costs (WBS 4000 and 5000)

Infrastructure costs are summarized in Table 21-5 and include costs associated with installing an additional transformer and motor control centre (MCC) to support the completion of the copper circuit installation.

Table 21-5: On-Site and Off-Site Infrastructure Costs

WBS	WBS Description	Capital Cost ($M)	Sustaining Cost ($M)	Total Cost ($M)
4000	**On-Site Infrastructure**	**0.9**	**0.0**	**0.9**
4200	HV Power Switchyard and Power Distribution	0.9	0.0	0.9
5000	**Off-Site Infrastructure**	**0.0**	**29.9**	**29.9**
5100	**Tailings Management Facility**	**0.0**	**14.9**	**14.9**
5300	**Tailings Water Treatment**	**0.0**	**15.0**	**15.0**
	Total Infrastructure Costs	**0.9**	**29.9**	**30.8**

21.2.5 Indirect Capital Costs

Indirect costs are summarized in Table 21-6 and described in the following subsections.

Table 21-6: Indirect Capital Costs

WBS	WBS Description	Capital Cost ($M)	Sustaining Cost ($M)	Total Cost ($M)
6000	**Project Indirects**	**5.8**	**0.1**	**5.9**
6100	Temporary Construction Facilities and Services	0.3	0.0	0.3
6300	Commissioning Reps & Assistance (New Equipment)	0.03	0.0	0.03
6400	Spares	3.6	0.0	3.6
6500	First Fills & Initial Charges	1.3	0.0	1.3
6600	Off-Site Accommodation Costs	0.3	0.0	0.3
6700	Freight and Logistics	0.3	0.1	0.3
7000	**Project Delivery**	**4.2**	**4.5**	**8.7**
7100	Field Indirects	0.5	0.0	0.5
7200	Engineering Services	2.6	4.5	7.1
7300	Commissioning Services (Plant Restart)	1.1	0.0	1.1
8000	**Owner's Costs**	**2.1**	**0.0**	**2.1**
8100	Owner's Project Management	2.1	0.0	2.1
	Total Indirect Costs	**12.1**	**4.6**	**16.7**

21.2.5.1 Project Indirects (WBS 6000)

Project indirects are required during the project delivery period to enable and support construction activities and are related to the process plant. These costs include the following:

- temporary construction facilities and services
- commissioning representatives and assistance
- spares (commissioning, initial, and insurance)
- first fills and initial charges
- off-site accommodation for project personnel
- on-site materials transportation and storage.

21.2.5.1.1 Project Indirects – Excluding WBS 6300, 6500

The project indirect and project delivery costs for WBS 6000 (except WBS 6300 and 6500) are related to plant restart costs and have been based on Inteloc estimates using the methodology in Table 21-7.

Table 21-7: Project Indirects Factors (Excluding WBS 6300 and 6500)

WBS	WBS Description	Basis of Factor	Factor Percentage (%)
6100	Temporary Construction Facilities & Services	Total direct costs	10
6400	Spares	Two of each agitator, one of each pump, 20% idlers for all conveyors, belting for each conveyor excluding liners and other operating costs, allowance for electrical spares	1
6600	Off-Site Accommodation Costs	Approximately $185/d for engineering CM team and contractors	(direct calculation, not factored)
6700	Freight and Logistics	10% of mechanical equipment supply cost and 5% of local bulk supply cost) on new equipment/bulk materials only (including mill bearings, pumps, etc.)	10 / 5

21.2.5.1.2 Project Indirects – WBS 6300, 6500

The project indirect and project delivery costs for WBS 6300 and 6500 have been based on Ausenco estimates using the methodology in Table 21-8.

Table 21-8: Project Indirects and Project Delivery Factors

WBS	WBS Description	Basis of Factor	Factor Percentage (%)
6300	Commissioning Representatives and Assistance (New Equipment)	Total mechanical supply (new WBS 3400 scope only)	10
6500	First Fills & Initial Charges	Based on equipment sizing, process requirements, and historical pricing	(direct calculation, not factored)

21.2.5.2 Project Delivery (WBS 7000)

Project delivery costs are composed of the following:

- field Indirects, intended to reflect project delivery related indirect costs
- EPCM services Including costs related to
  - process plant restart
  - new equipment

- process plant commissioning services, including costs related to:
  - process plant restart
  - new equipment.

21.2.5.2.1 Field Indirects

Field Indirect costs were estimated based on 5% of the total direct costs for new equipment in the project (WBS 1300, WBS 3400, WBS 4200).

21.2.5.2.2 EPCM Services – Process Plant Restart

Process plant restart EPCM services were estimated by Inteloc as $550,000 based on engineering required for three new compressors, two baghouses, and two dust extraction units.

21.2.5.2.3 EPCM Services – New Process Equipment

EPCM services for new equipment was estimated by Ausenco as 20% of total direct costs related to WBS 1300, WBS 3400, and WBS 4200.

21.2.5.2.4 Process Plant Commissioning Services – Process Plant Restart

Commissioining services related to the process plant restart have been estimated by Inteloc as $991,600 based on 11 persons for 6 weeks, as well as an allowance for quality assurance and quality control personnel for 25 days.

21.2.5.2.5 Process Plant Commissioning Services – New Equipment

Commissioining services related to new process plant equipment have been estimated by Ausenco as $93,000 based on a factor of 3% for commissioning personnel and vendor representatives. This factor is based on benchmarking of historical projects.

21.2.5.3 Owner's Costs (WBS 8000)

Owner's costs were factored from total non-mining direct and indirect costs as 15% of those costs (or $2.1 million), and include the following:

- project staffing and miscellaneous expenses
- pre-production labour
- home office project management
- home office finance, legal, and insurance.

21.2.6 Contingency (WBS 9000)

Contingency accounts for the difference in costs from the estimated and actual costs of materials and equipment. The level of contingency varies depending on the nature of the contract and the client's requirements. Due to uncertainties at the time the capital cost estimate was developed, it is essential that the estimate includes a provision to cover the risk from these uncertainties.

The estimate contingency does not allow for the following:

- abnormal weather conditions
- changes to market conditions affecting the cost of labour or materials
- changes of scope within the general production and operating parameters
- effects of industrial disputations
- financial modelling
- technical engineering refinement
- estimate inaccuracy.

The total estimated contingency for the Murray Brook Project is C$8.2 million. The estimated contingency percentages are listed in the following subsections.

21.2.6.1 Mining Contingency

Contingency for mining was estimated at 15% of mining capital costs, listed under WBS 1500.

21.2.6.2 New Process Plant Equipment

Contingency was 25% of total direct and indirect costs of new equipment (WBS 1300, WBS 3400, WBS 4200), based on the engineering definition and pricing estimate methodology.

21.2.6.3 Process Plant Restart Contingency

Contingency was 18% of total direct and indirect costs related to the process plant restart.

21.2.7 Sustaining Capital

Sustaining capital is comprised of costs related to mining, process plant infrastructure, tailings management facilities, water treatment, and haul road repayment. The sustaining costs are summarized in Table 21-1.

21.2.7.1 Mining (WBS 1000)

Mining sustaining capital is based on the purchase of pickup trucks at Years 4, 8, and 12 to support the mining operations.

21.2.7.2 Process Plant (WBS 3000)

Capital costs under WBS 3000 include the replacement of the following:

- liner handler
- compressor no. 1 and associated filters
- compressor no. 2 and associated filters
- compressor no. 3 and associated filters.

Freight for WBS 3000 has been calculated as a percentage of equipment and bulk material supply for all new items defined within the scope. Bulk materials utilise a 5% freight cost for inland freight which is typically local supply, while equipment utilizes a 10% freight factor for general North American supply.

21.2.7.3 Tailings Management (WBS 4000)

Costs for the tailings management facility include the construction of the North Tributary tailings pond dam, encompassing grubbing and clearing of the dam footprint, hauling and placement of materials sourced on site or imported locally, the supply and installation of geomembranes and water diversion structures, and water management during construction. Estimated volumes were developed based on quantities from the preliminary design. Unit rates were developed using contractor quotes, cost estimating tools, and pricing from past projects.

All costs related to the tailings management facilities have been developed based on information available at this time and the details outlined in Section 18.6.

The total construction cost for the North Tributary tailings pond dam is estimated at $38.9 million, with construction assumed to occur in two phases: Years 2 to 3 and Years 9 to 10 of the project timeline. A 10% allowance for engineering and project management and a 20% contingency have been included. These costs were provided in 2025 dollars to Ausenco for inclusion in the project's financial model.

21.2.7.4 Water Treatment (WBS 5000)

Water treatment plant costs for the Caribou Mine were developed using vendor quotes for similar water treatment concepts at other projects. A 10% allowance of contingency has been included.

21.2.7.5 Contingency

Sustaining capital contingency is composed of contingency related to the process plant and tailings and costs related to water management. The components of the total contingency are described in the following subsections.

21.2.7.5.1 Process Plant Sustaining Capital Contingency

The contingency for process plant sustaining capital was included at 10% of all costs.

21.2.7.5.2 Tailings Management Facility Contingency

A 20% contingency was applied to the total estimated costs for construction of the new tailings management facility.

21.2.7.5.3 Water Treatment Contingency

A 10% contingency was applied to the total estimated costs for the water treatment plant at the Caribou Mine.

21.3 Operating Costs

21.3.1 Overview

The operating cost estimate has been developed in Canadian dollars with a base date of Q2 2025. No allowance has been included for escalation. The estimate includes mining, processing, and general and administrative (G&A) costs.

The overall life-of-mine operating cost is C$1,228 million over 13.2 years, or an average of C$79.32/t processed. Table 21-9 provides a summary of the project operating costs.

Table 21-9: Operating Cost Summary (Life-of-Mine Average)

Cost Area	Total (C$M/a)	C$/t Processed	% of Total
Mining	48.9	41.7	52.5
Processing	36.6	31.2	39.3
G&A	7.6	6.5	8.2
Total	**93.1**	**79.3**	**100.0**

21.3.2 Basis of Estimate

The operating cost estimate was developed with the principles of a Class 5 estimate according to the Association for the Advancement of Cost Engineering International (AACE International).

Key information used to support the cost estimates include the following:

- production rate of 1.2 Mt/a
- for unit costs quoted in US dollars, an exchange rate of 1.34 USD/CAD
- reagent consumption based on the project mass balance, supported by metallurgical testwork
- grinding media consumption rates estimated based on the feed material characteristics
- unit cost estimates for reagents and consumables made by vendor quotation or in-house data from similar projects in the area
- annual maintenance cost estimated as a factor of the total capital cost estimate
- power consumption based on the project mechanical equipment list, with the power unit cost obtained from the local utility provider (NB Power)
- the staffing plan, including positions, provided by Canadian Copper to determine the labour cost
- salaries estimated by Ausenco based on in-house data from reference projects in the area
- mobile equipment cost provides for fuel and maintenance, not for purchase or vehicle lease
- crushing cost estimated on a per tonne processed basis, provided by a local rental equipment provider
- water treatment costs provided by Stantec for the Caribou Mine based on supplier quotes from similar projects
- mining costs from a quote provided by an established local mining contractor
- G&A costs that were benchmarked against previous similar projects.

The following items are excluded from the operating cost estimate:

- off-site charges, including transport, treatment, penalties, and refining charges
- royalties
- closure costs.

These exclusions are accounted for elsewhere in the financial model.

21.3.3 Mine Operating Costs

Mining operations will be performed by the mining contractor. Hauling of mineralized material from the open pit area to the Caribou process plant (approximately 17 km) will be undertaken by the mining contractor. The mine operating cost is summarized in Table 21-10 and averages $7.20 per tonne mined ($41.66/t processed) over the life of the project.

Table 21-10: Mine Operating Cost per Tonne of Material Mined (Life-of-Mine Average)

Operating Cost Category	Operating Cost Per Tonne Material Mined ($/t Material Mined)
Drilling and Blasting (Contractor)	1.67
Loading (Contractor)	0.80
Hauling (Contractor, Excluding Hauling to Caribou)	1.84
Services/Roads/Dumps (Contractor)	1.28
Hauling to Caribou Plant (Contractor, $7.86/t Processed)	1.36
General, Supervision & Technical (Owner)	0.25
Total Operating Cost	**7.20**
Operating Cost Per Tonne Processed ($/t)	**41.66**

21.3.4 Process Operating Costs

The process plant operating cost estimate includes costs related to process plant power consumption, labour, reagent and consumables consumption, laboratory costs, mobile equipment, and maintenance expenditures. Crushing equipment rental costs and water treatment costs are also included in the process operating costs. G&A costs were also developed as part of the estimate, but are presented separately from the process operating costs. Averaging over the life of mine, the process operating cost is estimated at C$36.6 million per year, or C$31.18 per tonne processed. A breakdown of the operating costs is presented in Table 21-11 and described in additional detail in subsequent sections.

Table 21-11: Process Operating Cost Breakdown (Life-of-Mine average)

Cost Area	Annual Cost (C$M/a)	Unit Cost (C$/t Processed)
Reagents and Consumables	12.7	10.9
Maintenance	1.8	1.6
Power	6.9	5.9
Laboratory	0.8	0.7
Labour	7.8	6.6
Mobile Equipment	1.2	1.0
Crushing Equipment Rental	4.1	3.5
Water Treatment	1.3	1.1
Total	**36.6**	**31.2**

21.3.4.1 Reagents and Consumables

Total reagent consumption and cost is summarized in Table 21-12.

Table 21-12: Reagent Consumption and Cost Summary

Reagent	Annual Consumption (t/a)	Unit Cost (C$/kg)	Annual Cost (C$M/a)
Quicklime	2,700	0.58	1.6
Sodium Cyanide	542	4.29	2.3
Zinc Sulphate	1,127	1.84	2.0
PEX Collector	33	4.39	0.1
Copper Sulphate	337	4.19	1.4
MIBC Frother	36	4.92	0.2
SMBS Depressant	1,205	0.86	1.0
Flocculant	33	5.20	0.2
Total	**-**	**-**	**8.7**

The process operating cost consumables includes operating consumables, namely mill media and filtration cloths, as well as mill liners which are considered maintenance consumables. The annual consumption and cost are presented in Table 21-13.

Table 21-13: Operating and Maintenance Consumables Usage and Cost Summary

Cost Item	Annual Consumption	Unit	Unit Cost (C$/unit)	Annual Cost (C$M/a)
SAG Mill Media	373,395	kg	1.80	0.67
Ball Mill Media	865,519	kg	1.86	1.61
Bulk Re-Grind Mill Media	5,510	kg	5.75	0.03
Zinc Re-Grind Mill Media	2,755	kg	5.75	0.01
Filtration Cloths	13.7	ea.	4,500	0.06
SAG Mill Liners	1.0	ea.	759,300	0.7
Ball Mill Liners, per Mill	0.8	ea.	502,200	0.8
Bulk Re-Grind Mill Liners, per Mill	1.0	ea.	72,700	0.1
Zinc Re-Grind Mill Liners	1.0	ea.	27,100	0.03
Total	**-**	**-**	**-**	**4.0**

21.3.4.2 Maintenance

The maintenance costs are estimated at C$1.82 million per year or C$1.55 per tonne processed. Maintenance costs were estimated as a percentage of the total major capital equipment cost for the project. This allowance includes expected costs for the maintenance of all mechanical equipment, pipes, chutes, motors, electrical components, and valves. Note that mill liner costs are not included in this allowance; they are included under the reagents and consumables section.

21.3.4.3 Power

The local utility provider (NB Power) charges users for electricity under two categories—demand charge and energy charge. The demand charge is billed as C$19.17/kW of the maximum demand per month, and the energy change is billed as C$0.075/kWh.

Based on the project mechanical equipment list, the nominal power demand is estimated as 8.44 MW or 68,044 MWh per year, giving a total annual estimated power cost of C$6.9 million.

21.3.4.4 Laboratory

Laboratory costs are the costs related to sample preparation and assays for process plant samples. The laboratory costs are estimated at C$0.86 million per year or C$ 0.72 per tonne processed.

21.3.4.5 Labour

The total labour force estimate for plant operations and plant maintenance is 81 people. The staffing plan, including positions, was provided to Ausenco by Canadian Copper. Salaries were estimated by Ausenco based on in-house data from reference projects in the area. The annual operating cost estimate for labour is C$7.77 million per year, or C$6.62 per tonne processed. The annual labour cost summary is presented in Table 21-14. The staffing plan is presented in Table 21-15 for operations staff, and Table 21-16 for maintenance staff.

Table 21-14: Labour Cost Summary

Area	Annual Cost (C$M/a)
Process Plant Operations	4.44
Process Plant Maintenance	3.33
Total	**7.77**

Table 21-15: Process Plant Operations Staffing Plan

Position	Number of Staff
Plant Manager	1
Senior Metallurgist	1
Metallurgist	1
Metallurgist-in-Training	1
Metallurgy Technician	2
Assay Laboratory Supervisor	1
Assay Laboratory Technician	1
Assay Laboratory Sampler	1
Mill Superintendent	1
Mill Supervisor	4
Flotation Operator	4
Grinding Operator	4
Swing Operator	4
Water Treatment Operator	4
Reagent Mix Operator	2
Dewatering Operator	4
Grinding Helper	4
Mill Labourer	6
Load-out Operator	2
Mill Operations Project Supervisor	1
Total	**49**

Table 21-16: Process Plant Maintenance Staffing Plan

Position	Number of Staff
Mill Maintenance Superintendent	1
Maintenance Planner	1
Mill Maintenance Supervisor	1
Industrial Mechanic	9
Maintenance Apprentice (Third Year)	1
Pipefitter	3
Welder	3
Surface and Electrical Manager	1
Electrical and Instrumentation Planner	1
PLC and Instrumentation Supervisor	1
Instrumentation	2
Electrical Senior Supervisor	1
Electrician	4
Surface Labourer	3
Total	**32**

21.3.4.6 Mobile Equipment

Vehicle costs are the expenditures related to spares, consumables, and fuel costs for the vehicles operated at the processing plant. The vehicle costs will be approximately C$1.18 million per year or C$1.0 per tonne processed.

21.3.4.7 Crushing Equipment Rental

The crushing equipment will be rental equipment. The cost has been provided to Ausenco by a local equipment vendor and has been quoted on a price per tonne basis. The total crushing costs will be C$4.11 million per year or C$3.50 per tonne processed.

21.3.4.8 Caribou Mine Water Treatment

The operating cost estimate for water treatment at Caribou mine is C$1.5 million per year, starting in Year 4 of production and continuing throughout the life of mine. The average cost is C$1.07 per tonne processed. This estimate was provided to Ausenco by Stantec, and is based on similar vendor quotations for similar water treatment systems. The estimate includes reagents and electricity required for water treatment. Labour required is included in the operations labour estimate.

21.3.5 General and Administrative Operating Costs

G&A costs are expenses not directly related to the operation of the process plant but are required to support the effective operation of the facility and to sometimes satisfy legislative requirements. The G&A estimate was developed using Ausenco's in-house data on existing operations, and includes costs such as:

- human resources, including training, recruiting and community relations
- site administration, maintenance, and security
- health and safety, including personal protective equipment, hospitals service costs and first aid
- environmental, such as water sampling
- IT and telecommunications, including hardware and support services
- contract services, including insurance, consulting, sanitation and cleaning feeds, and legal fees.

The annual costs are estimated to be C$7.61 million per year or C$6.49 per tonne processed.

22 ECONOMIC ANALYSIS

22.1 Forward-Looking Information Cautionary Statements

The results of the economic analyses discussed in this section represent forward-looking information as defined under Canadian securities law. The results depend on inputs that are subject to known and unknown risks, uncertainties, and other factors that may cause actual results to differ materially from those presented here. Information that is forward-looking includes the following:

- mineral resource estimates
- assumed commodity prices and exchange rates
- proposed mine production plan
- projected mining and process recovery rates
- assumptions as to mining dilution and estimated future production
- sustaining costs and proposed operating costs
- assumptions as to closure costs and closure requirements
- assumptions as to environmental, permitting, and social risks.

Additional risks to the forward-looking information include:

- changes to costs of production from what is assumed
- unrecognized environmental risks
- unanticipated reclamation expenses
- unexpected variations in quantity of mineralized material, grade, or recovery rates
- accidents, labour disputes, and other risks of the mining industry
- geotechnical or hydrogeological considerations during mining being different from what was assumed
- failure of mining methods to operate as anticipated
- failure of plant, equipment, or processes to operate as anticipated
- changes to assumptions as to the availability of electrical power, and the power rates used in the operating cost estimates and financial analysis
- changes to site access, use of water for mining purposes and to time to obtain environment and other regulatory permits
- ability to maintain the social licence to operate
- changes to interest rates
- change4s to tax rates.

Readers are cautioned that the 2025 PEA is preliminary in nature and there is no certainty that the 2025 PEA will be realized.

Mineral resources that are not mineral reserves do not have demonstrated economic viability.

22.2 Financial Model Update Since May 22, 2025 Press Release

The economic analysis included in this technical report have been updated from the press release issued on May 22, 2025 by Canadian Copper. The updates included:

- the addition of a 0.67% royalty, payable to MQM (total royalties increased from 0.58% to 1.25%)
- the refinement of the corporate tax rate from 30% to 29% to reflect New Brunswick corporate tax rates.

These refinements result in changes to the pre-tax and post-tax NPV and IRR.

22.3 Methodologies Used

The project was evaluated using a discounted cash flow (DCF) analysis based on a 7% discount rate. Cash inflows consisted of annual revenue projections. Cash outflows consisted of capital expenditures, including pre-production costs; operating costs; taxes; and royalties. These were subtracted from the inflows to arrive at the annual cash flow projections. Cash flows were taken to occur at the mid-point of each period. It must be noted that tax calculations involve complex variables that can only be accurately determined during operations and, as such, the actual post-tax results may differ from those estimated. A sensitivity analysis was performed to assess the impact of variations in metal prices, discount rate, operating costs, initial capital costs, metal recovery, and head grade.

The capital and operating cost estimates are presented in Section 21 of this report in Q2 2025 Canadian dollars. The economic analysis was run on a constant dollar basis with no inflation with a discount rate of 7%.

22.4 Financial Model Parameters

The economic analysis was performed assuming a copper price of US$4.25/lb, a silver price of US$27.00/oz, a lead price of US$1.10/lb; and a zinc price of US$1.30/lb and exchange rate of C$0.746 to US$1.00. These metal prices were based on consensus analyst estimates and recently published economic studies. The forecasts used are meant to reflect the average metals price expectation over the life of the project. No price inflation or escalation factors were considered. Commodity prices can be volatile, and there is the potential for deviation from the forecast.

The economic analysis also used the following assumptions:

- construction period of 18 months
- mine life of 14 years (last year is a partial year)
- results based on 100% ownership
- capital cost funded with 100% equity (no financing cost assumed)
- all cash flows discounted to start of construction period using mid-period discounting convention

- all metal products are sold in the same year they are produced
- project revenue is derived from the sale copper concentrate, zinc concentrate, and lead concentrate
- no contractual arrangements for concentrate offtake currently exist.

22.4.1 Taxes

The project has been evaluated on an after-tax basis to provide an approximate value of the potential economics.

The calculations are based on the tax regime in place as of the date of the preliminary economic analysis. At the effective date of the cashflow, the project was assumed to be subject to a corporate income tax rate of 29%.

At the base case price assumption, total tax payments are estimated to be $162.2 million over the life of mine.

22.4.2 Working Capital

An estimate of working capital has been incorporated into the economic analysis based on the assumptions listed in Table 22-1.

Table 22-1: Estimate of Working Capital

Item	Time
Accounts Receivable	0 days
Inventories	30 days
Accounts Payable	30 days

22.4.3 Closure Costs

Closure costs are applied at the end of the life of mine and were estimated to be $52.6 million.

22.4.4 Royalties

A net smelter return (NSR) royalty on the Murray Brook asset at 1.0% and a final instalment of C$1 million will be paid by the Company to MQM and a private NSR royalty on the Murray Brook asset at 0.25% and a final instalment of C$2 million will be paid by the Company to VMC.

Buyback options are included.

Based on the agreements in place with MQM and VMC, the total royalty costs were estimated to be $27.2 million over the life of mine.

22.4.5 Selling Costs

Selling costs are listed in Table 22-2.

Table 22-2: Selling Costs

Payability	Unit of Measure	Value
Copper in Concentrate	%	96.50
Lead in Concentrate	%	95.00
Zinc in Concentrate	%	85.00
Silver in Copper Concentrate	%	90.00
Silver in Lead Concentrate	%	95.00
Offsite Charges		
Transport		
Copper Concentrate	US$/wmt	69.82
Lead Concentrate	US$/wmt	69.82
Zinc Concentrate	US$/wmt	69.82
Treatment		
Copper Concentrate	US$/dmt	65
Lead Concentrate	US$/dmt	60
Zinc Concentrate	US$/dmt	175
Refining		
Copper in Concentrate	US$/lb	0.0650
Lead in Concentrate	US$/lb	0.000
Zinc in Concentrate	US$/lb	0.000
Silver in Copper Concentrate	US$/oz	0.376
Silver in Lead Concentrate	US$/oz	1.000

Selling costs, payabilities and transport costs were determined using input from Ocean Partners.

22.5 Economic Analysis

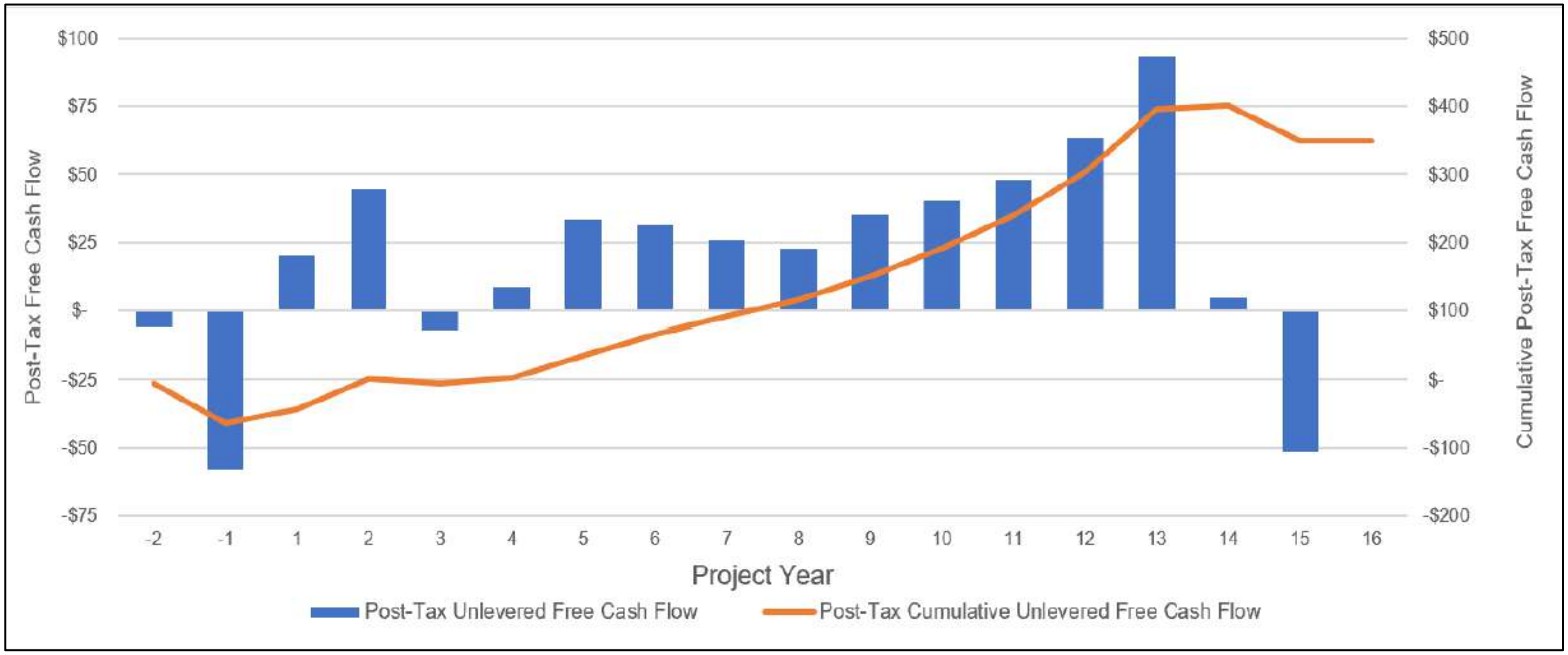
The pre-tax NPV discounted at 7% is $256.1 million; the IRR is 47.6%; and the payback period is 1.6 years. On a post-tax basis, the NPV discounted at 7% is $169.0 million; the IRR is 35.4%; and payback period is 2.0 years. A summary of project economics is shown in Table 22-3 and illustrated in Figure 22-1. The analysis was done on an annual cashflow basis; the cashflow output is shown Table 22-4.

Table 22-3: Economic Analysis Summary

Area	Description	Unit of Measure	Total/Average
General	Copper Price	US$/lb	4.25
	Silver Price	US$/oz	27.00
	Lead Price	US$/lb	1.10
	Zinc Price	US$/lb	1.30
	FX rate	CAD/USD	0.746
Production	Mine Life	years	13.2
	Total Resource Mined	**kt**	**15,486**
	Life-of-Mine Copper Equivalent Grade	%	1.91
	Total Waste Mined	kt	77,725
	Strip Ratio	Waste: Mineral	5.02:1
	Average Annual Mined Resource	kt/a	1,173
	Total Recovery Copper	%	68.3
	Total Recovery Silver	%	55.4
	Total Recovery Lead	%	44.7
	Total Recovery Zinc	%	82.3
	Total Payable Copper	**Mlbs**	**103.8**
	Total Payable Copper Equivalent	**Mlbs**	**396.0**
	Average Annual Payable Copper	Mlbs/a	8
	Average Annual Payable Copper Equivalent	Mlbs/a	30
	Total Payable Zinc	**Mlbs**	**626**
	Total Payable Zinc Equivalent	**Mlbs**	**1,296**
	Average Annual Payable Zinc	Mlbs/a	47
	Average Annual Payable Zinc Equivalent	Mlbs/a	98
Revenue/Costs	Total Revenue	$M	2,258.1
	Average Annual Revenue	$M	171.1
	Copper Revenue (as % of Gross Revenue)	%	26
	Zinc Revenue (as % of Gross Revenue)	%	48
	EBITDA	$M	676.8
	Average Annual EBITDA	$M	51.3
	Total On-Site Operating Costs (Mining, Process, G&A)	**$M**	**1,228**
	Average Annual On-Site Operating Cost	$M	93.05
	Mining Unit Cost	$/t mined	7.20
	Mining Unit Cost	$/t milled	41.66
	Process Unit Cost	$/t milled	31.18
	G&A Unit Cost	$/t milled	6.49
	Total Off Site Operating Costs (Transport, Treatment & Refining)	**$M**	**325.8**
Cash Costs	Total Cash Cost*	US$/lb CuEq	3.0
	All-in Sustaining Cost**	US$/lb CuEq	3.2
	Total Cash Cost*	US$/oz ZnEq	0.9
	All-In Sustaining Cost**	US$/oz ZnEq	1.0
Capital Costs	Initial Capital Cost (Total)	$M	64.00
	Initial Capital Cost – Mining	$M	29.12
	Initial Capital Cost – Other	$M	34.58
	Sustaining Capital Cost	$M	48.9
	Sustaining Capital Cost – Mining	$M	1.0
	Sustaining Capital Cost – Other	$M	47.9
	Closure Cost	$M	52.6
Pre-Tax Economics	NPV (7%)	$M	256.1
	IRR	%	47.6
	Payback	years	1.6
	NPV/Initial Capital	-	4.0
Post-Tax Economics	NPV (7%)	$M	169.0
	IRR	%	35.4
	Payback	years	2.0
	NPV/Initial Capital	-	2.7

Notes: [1]Total cash costs consist of mining costs, processing costs, mine-level G&A, royalties, and off-site charges. [2]AISC includes cash costs plus sustaining capital and closure costs

Figure 22-1: Projected Life- of-Mine Post-Tax Unlevered Free Cash Flow



Source: Ausenco, 2025

Table 22-4: Cashflow Statement on an Annualized Basis

Dollar figures in Real 2025 C$M unless otherwise noted	Units	Total / Avg.	-2	-1	1	2	3	4	5	6	7	8	9	10	11	12	13	14	15
Copper Price	US$/lb	4.3	4.3	4.3	4.3	4.3	4.3	4.3	4.3	4.3	4.3	4.3	4.3	4.3	4.3	4.3	4.3	4.3	4.3
Silver Price	US$/oz	27.0	27.0	27.0	27.0	27.0	27.0	27.0	27.0	27.0	27.0	27.0	27.0	27.0	27.0	27.0	27.0	27.0	27.0
Lead Price	US$/lb	1.1	1.1	1.1	1.1	1.1	1.1	1.1	1.1	1.1	1.1	1.1	1.1	1.1	1.1	1.1	1.1	1.1	1.1
Zinc Price	US$/lb	1.3	1.3	1.3	1.3	1.3	1.3	1.3	1.3	1.3	1.3	1.3	1.3	1.3	1.3	1.3	1.3	1.3	1.3
Exchange Rate	CAD:USD	0.75	0.75	0.75	0.75	0.75	0.75	0.75	0.75	0.75	0.75	0.75	0.75	0.75	0.75	0.75	0.75	0.75	0.75
Free Cash Flow Valuation																			
Revenue	**C$M**	**2,258**	**--**	**--**	**139**	**200**	**138**	**153**	**174**	**174**	**159**	**150**	**163**	**172**	**174**	**197**	**246**	**21**	**--**
Operating Cost	C$M	**(1,228)**	--	--	(86)	(100)	(111)	(109)	(104)	(104)	(99)	(99)	(85)	(81)	(84)	(80)	(77)	(10)	--
Offsite Charges	C$M	**(326)**	--	--	(19)	(29)	(16)	(20)	(24)	(26)	(24)	(20)	(24)	(26)	(26)	(29)	(39)	(3)	--
Royalties	C$M	**($27)**	--	--	**($5)**	($2)	($2)	**($2)**	($2)	($2)	**($2)**	($2)	($2)	**($2)**	($2)	($2)	**($3)**	($0)	--
EBITDA	**C$M**	**$677**	**--**	**--**	**$29**	**$69**	**$9**	**$22**	**$45**	**$42**	**$34**	**$29**	**$52**	**$63**	**$63**	**$85**	**$127**	**$7**	**--**
Initial Capex	C$M	(64)	(6)	(58)	--	--	--	--	--	--	--	--	--	--	--	--	--	--	--
Sustaining Capex	C$M	(49)	--	--	(3)	(6)	(15)	(9)	(1)	(1)	(1)	(0)	(4)	(7)	--	(0)	--	--	--
Closure Capex	C$M	(53)	--	--	--	--	--	--	--	--	--	--	--	--	--	--	--	(1)	(51)
Change in Working Capital	C$M	--	--	--	--	--	--	--	--	--	--	--	--	--	--	--	--	--	--
Pre-Tax Unlevered Free Cash Flow	**C$M**	**($0)**	**($6)**	**($58)**	**$26**	**$63**	**($6)**	**$13**	**$44**	**$41**	**$33**	**$29**	**$48**	**$55**	**$63**	**$85**	**$127**	**$6**	**($51)**
Pre-Tax Cumulative Unlevered Free Cash Flow	C$M		($6)	($64)	($38)	$25	$19	$31	$75	$117	$150	$179	$227	$282	$345	$430	$557	$563	$512
Tax Payable	**C$M**	$162	--	--	$6	$18	$1	$4	$11	$10	$8	$6	$13	$15	$15	$22	$34	$1	--
Post-Tax Unlevered Free Cash Flow	**C$M**	**($0)**	**($6)**	**($58)**	**$20**	**$44**	**($7)**	**$9**	**$33**	**$31**	**$26**	**$23**	**$35**	**$40**	**$48**	**$63**	**$93**	**$5**	**($51)**
Post-Tax Cumulative Unlevered Free Cash Flow	C$M		($6)	($64)	($44)	$1	($6)	$2	$36	$67	$93	$116	$151	$191	$239	$303	$396	$401	$349
Production																			
Production Summary																			
Total Resource Mined	kt	15,486	--	--	903	1,204	1,204	1,204	1,204	1,204	1,204	1,204	1,204	1,204	1,204	1,204	1,204	129	--
Total Waste Mined	kt	77,725	--	3,572	7,003	8,466	12,099	10,141	7,286	7,652	6,289	5,664	3,275	2,178	2,133	1,268	684	17	--
Total Material Mined	kt	93,210	--	3,572	7,906	9,670	13,303	11,346	8,490	8,857	7,493	6,868	4,479	3,382	3,337	2,473	1,889	146	--
Mill Feed	kt	15,486	--	--	903	1,204	1,204	1,204	1,204	1,204	1,204	1,204	1,204	1,204	1,204	1,204	1,204	129	--
Mill Head Grade – Silver	g/t	41.09	--	--	34.73	40.87	30.15	34.62	39.67	39.30	38.31	35.68	42.11	41.73	45.15	48.13	60.94	51.81	--
Mill Head Grade – Copper	%	0.47	0.00	0.00	0.69	0.57	0.66	0.58	0.53	0.40	0.38	0.50	0.38	0.35	0.39	0.43	0.30	0.27	0.00
Mill Head Grade – Lead	%	0.96	0.00	0.00	0.79	0.90	0.57	0.73	0.90	0.98	0.91	0.79	0.96	1.03	1.03	1.19	1.64	1.33	0.00
Mill Head Grade – Zinc	%	2.67	0.00	0.00	2.42	3.02	1.44	1.95	2.47	2.90	2.62	2.06	2.58	2.94	2.77	3.09	4.36	3.45	0.00
Metal Recovered – Silver in Concentrate	koz	11,340.65	--	--	559.16	877.36	647.29	743.21	851.59	843.74	822.47	766.01	904.05	895.79	969.36	1,033.20	1,308.39	119.03	--
Metal Recovered – Copper in Concentrate	kt	108.78	--	--	10.16	10.94	12.87	10.98	10.04	6.82	6.36	9.26	6.55	5.67	6.65	7.55	4.52	0.41	--
Metal Recovered – Lead in Concentrate	kt	146.81	--	--	6.06	10.27	3.52	6.75	10.27	11.85	10.36	8.00	11.45	12.69	12.71	15.97	24.89	2.00	--
Metal Recovered – Zinc in Concentrate	kt	751.31	--	--	39.22	66.82	28.61	40.85	53.46	63.94	57.09	43.48	56.12	64.81	60.83	68.63	99.20	8.26	--
Total Payable Metal – Silver	koz	10,340	--	--	512	802	593	681	778	768	751	702	825	816	884	938	1,181	108	--
Total Payable Metal – Copper	kt	104	--	--	10	10	12	10	10	7	6	9	6	5	6	7	4	0	--
Total Payable Metal – Lead	kt	137	--	--	6	10	3	6	10	11	10	7	11	12	12	15	23	2	--
Total Payable Metal – Zinc	kt	626	--	--	33	56	24	34	45	53	48	36	47	54	51	57	83	7	--
Revenue	**C$M**	**2,258**	**--**	**--**	**139**	**200**	**138**	**153**	**174**	**174**	**159**	**150**	**163**	**172**	**174**	**197**	**246**	**21**	**--**
Operating Costs																			
Total Operating Costs	**C$M**	**1,228**	**--**	**--**	**86**	**100**	**111**	**109**	**104**	**104**	**99**	**99**	**85**	**81**	**84**	**80**	**77**	**10**	**--**
Mine Operating Costs	C$M	645	--	--	49	56	68	64	58	59	54	53	40	36	38	35	32	2	--
Processing Costs	C$M	483	--	--	29	36	36	38	38	38	38	38	38	38	38	38	38	6	--
G&A Costs	C$M	100	--	--	8	8	8	8	8	8	8	8	8	8	8	8	8	2	--
Offsite Charges and Royalties																			
Total Offsite Costs	**C$M**	**326**	**--**	**--**	**19**	**29**	**16**	**20**	**24**	**26**	**24**	**20**	**24**	**26**	**26**	**29**	**39**	**3**	**--**
Transport Costs	C$M	111	--	--	7	10	6	7	8	9	8	7	8	9	9	10	13	1	--
Treatment Costs	C$M	198	--	--	11	18	9	12	14	16	15	12	15	16	16	18	25	2	--
Refining Costs	C$M	17	--	--	1	2	2	1	1	1	1	1	1	1	1	1	1	0	--
Royalties	**C$M**	**27**	**--**	**--**	**5**	**2**	**2**	**2**	**2**	**2**	**2**	**2**	**2**	**2**	**2**	**2**	**3**	**0**	**--**
Cash Costs																			
Cash Cost*	US$/lb CuEq	3	--	--	3	3	4	4	3	3	3	3	3	3	3	2	2	3	--
All-in Sustaining Cost (AISC)**	US$/lb CuEq	3	--	--	3	3	4	4	3	3	3	3	3	3	3	2	2	3	--
Cash Cost*	US$/lb ZnEq	1	--	--	1	1	1	1	1	1	1	1	1	1	1	1	1	1	--
All-in Sustaining Cost (AISC)**	US$/lb ZnEq	1	--	--	1	1	1	1	1	1	1	1	1	1	1	1	1	1	--
Capital Expenditures																			
Initial Capital	**C$M**	**64**	**6**	**58**	**--**	**--**	**--**	**--**	**--**	**--**	**--**	**--**	**--**	**--**	**--**	**--**	**--**	**--**	**--**
Mining	C$M	29	--	29	--	--	--	--	--	--	--	--	--	--	--	--	--	--	--
Water Treatment	C$M	0	--	0	--	--	--	--	--	--	--	--	--	--	--	--	--	--	--
Process Plant	C$M	7	--	7	--	--	--	--	--	--	--	--	--	--	--	--	--	--	--
On-Site Infrastructure	C$M	1	--	1	--	--	--	--	--	--	--	--	--	--	--	--	--	--	--
Off-Site Infrastructure	C$M	--	--	--	--	--	--	--	--	--	--	--	--	--	--	--	--	--	--
Indirects	C$M	6	--	6	--	--	--	--	--	--	--	--	--	--	--	--	--	--	--
Project Delivery	C$M	4	--	4	--	--	--	--	--	--	--	--	--	--	--	--	--	--	--
Owner's Costs	C$M	2	--	2	--	--	--	--	--	--	--	--	--	--	--	--	--	--	--
Contingency	C$M	8	--	8	--	--	--	--	--	--	--	--	--	--	--	--	--	--	--
Mill Purchase	C$M	6	6	--	--	--	--	--	--	--	--	--	--	--	--	--	--	--	--
Total Sustaining Capital	**C$M**	**49**	**--**	**--**	**3**	**6**	**15**	**9**	**1**	**1**	**1**	**0**	**4**	**7**	**--**	**0**	**--**	**--**	**--**
Air Compressors and Liner Handler	C$M	3	--	--	3	--	--	--	--	--	--	--	--	--	--	--	--	--	--
Mining	C$M	1	--	--	--	--	--	0	--	--	--	0	--	--	--	0	--	--	--
North Tributary Tailings Pond Construction	C$M	39	--	--	--	12	12	--	--	--	--	--	7	7	--	--	--	--	--
North Tributary Tailings Pond NB Government Contribution	C$M	(15)	--	--	--	(6)	(6)	--	--	--	--	--	(3)	--	--	--	--	--	--
Water Treatment Plant	C$M	17	--	--	--	--	8	8	--	--	--	--	--	--	--	--	--	--	--
Haul Road Repayment	C$M	4	--	--	--	--	1	1	1	1	1	--	--	--	--	--	--	--	--
Closure Cost	**C$M**	**53**	**--**	**--**	**--**	**--**	**--**	**--**	**--**	**--**	**--**	**--**	**--**	**--**	**--**	**--**	**--**	**1**	**51**

Notes: * Cash costs consist of mining costs, processing costs, mine-level G&A, royalties, and off-site charges. **AISC includes cash costs plus sustaining capital, and closure costs.

22.6 Sensitivity Analysis

A sensitivity analysis was conducted on the pre-tax and post-tax NPV and IRR of the project using the following variables: discount rate, operating costs, initial capital cost, metal recovery, head grade, and exchange rate.

The base case used the metal prices and exchange rate sourced on May 14, 2025, as listed in the base case summary table (Table 22-5).

Table 22-5: 2025 PEA Base CaseEconomic Highlights

Base Case Economics	Units	Spot Price
Net Present Value (After-Tax NPV 7%)	$M	169.0
Internal Rate of Return	%	35.9
Payback	Years	2.0
Initial Capital Cost	$M	63.7
Sustaining Capital Cost	$M	48.9
Economic Assumptions	**Units**	**Spot Price**
Copper	US$/lb	4.25
Silver	US$/oz	27.00
Lead	US$/lb	1.10
Zinc	US$/lb	1.30
Exchange Rate	USD/CAD	0.746

Table 22-6 shows the pre-tax sensitivity analysis results; post-tax sensitivity results are shown in Table 22-7.

As presented in Figures 22-2 and 22-3, the sensitivity analysis showed that the project is most sensitive to commodity prices, head grade, and recovery.

Table 22-6: Pre-Tax Sensitivity Analysis ($M CAD)

Sensitivity to Metal Price

Pre-Tax NPV Sensitivity To Discount Rate

Discount Rate	Commodity Price (20.0%)	(10.0%)	–	10.0%	20.0%
3.0%	$6	$181	$356	$531	$706
5.0%	$4	$154	$305	$455	$606
7.0%	($5)	$126	$256	$387	$517
8.0%	($10)	$112	$234	$356	$478
10.0%	($18)	$89	$196	$303	$410

Pre-Tax IRR Sensitivity To Discount Rate

Discount Rate	Commodity Price (20.0%)	(10.0%)	–	10.0%	20.0%
3.0%	3.8%	25.7%	47.5%	70.0%	92.2%
5.0%	5.6%	26.0%	47.6%	70.0%	92.2%
7.0%	6.2%	26.1%	47.6%	70.0%	92.2%
8.0%	6.4%	26.1%	47.6%	70.0%	92.2%
10.0%	6.7%	26.2%	47.6%	70.0%	92.2%

Pre-Tax NPV Sensitivity To Opex

Opex	Commodity Price (20.0%)	(10.0%)	–	10.0%	20.0%
(20.0%)	$145	$275	$406	$537	$667
(10.0%)	$70	$200	$331	$462	$592
–	($5)	$126	$256	$387	$517
10.0%	($80)	$51	$181	$312	$442
20.0%	($155)	($24)	$106	$237	$367

Pre-Tax IRR Sensitivity To Opex

Opex	Commodity Price (20.0%)	(10.0%)	–	10.0%	20.0%
(20.0%)	30.8%	52.9%	75.3%	97.2%	118.4%
(10.0%)	18.0%	39.2%	61.5%	83.8%	105.5%
–	6.2%	26.1%	47.6%	70.0%	92.2%
10.0%	0.0%	14.3%	34.1%	56.1%	78.6%
20.0%	0.0%	3.6%	21.9%	42.3%	64.7%

Pre-Tax NPV Sensitivity To Initial Capex

Initial Capex	Commodity Price (20.0%)	(10.0%)	–	10.0%	20.0%
(20.0%)	$7	$137	$268	$399	$529
(10.0%)	$1	$132	$262	$393	$523
–	($5)	$126	$256	$387	$517
10.0%	($11)	$120	$250	$381	$511
20.0%	($17)	$114	$244	$375	$505

Pre-Tax IRR Sensitivity To Initial Capex

Initial Capex	Commodity Price (20.0%)	(10.0%)	–	10.0%	20.0%
(20.0%)	8.2%	31.6%	58.7%	86.8%	113.9%
(10.0%)	7.2%	28.6%	52.5%	77.5%	101.9%
–	6.2%	26.1%	47.6%	70.0%	92.2%
10.0%	5.4%	24.0%	43.6%	63.9%	84.2%
20.0%	4.6%	22.2%	40.2%	58.9%	77.5%

Pre-Tax NPV Sensitivity To Recovery Mill

Recovery Mill	Commodity Price (20.0%)	(10.0%)	–	10.0%	20.0%
(20.0%)	($177)	($72)	$32	$137	$241
(10.0%)	($91)	$27	$144	$262	$379
–	($5)	$126	$256	$387	$517
10.0%	$81	$224	$368	$512	$655
20.0%	$167	$323	$480	$637	$793

Pre-Tax IRR Sensitivity To Recovery Mill

Recovery Mill	Commodity Price (20.0%)	(10.0%)	–	10.0%	20.0%
(20.0%)	0.0%	0.0%	11.8%	27.7%	44.8%
(10.0%)	0.0%	11.0%	29.0%	48.4%	68.6%
–	6.2%	26.1%	47.6%	70.0%	92.2%
10.0%	19.3%	42.3%	67.0%	91.4%	115.0%
20.0%	32.8%	59.4%	86.2%	112.1%	137.0%

Pre-Tax NPV Sensitivity To Head Grade

Head Grade	Commodity Price (20.0%)	(10.0%)	–	10.0%	20.0%
(20.0%)	($209)	($109)	($9)	$90	$190
(10.0%)	($107)	$8	$123	$238	$354
–	($5)	$126	$256	$387	$517
10.0%	$97	$243	$389	$535	$681
20.0%	$199	$361	$522	$684	$845

Pre-Tax IRR Sensitivity To Head Grade

Head Grade	Commodity Price (20.0%)	(10.0%)	–	10.0%	20.0%
(20.0%)	0.0%	0.0%	5.6%	20.5%	36.2%
(10.0%)	0.0%	8.3%	25.7%	44.5%	64.3%
–	6.2%	26.1%	47.6%	70.0%	92.2%
10.0%	21.7%	45.4%	70.5%	95.1%	119.0%
20.0%	38.4%	66.1%	93.6%	120.0%	145.4%

Pre-Tax NPV Sensitivity To Exchange Rate

Exchange Rate	Commodity Price (20.0%)	(10.0%)	–	10.0%	20.0%
(20.0%)	$210	$373	$536	$699	$862
(10.0%)	$90	$235	$381	$526	$671
–	($5)	$126	$256	$387	$517
10.0%	($83)	$36	$154	$273	$392
20.0%	($148)	($39)	$69	$178	$287

Pre-Tax IRR Sensitivity To Exchange Rate

Exchange Rate	Commodity Price (20.0%)	(10.0%)	–	10.0%	20.0%
(20.0%)	40.0%	68.0%	95.6%	122.2%	147.8%
(10.0%)	20.7%	44.2%	69.1%	93.7%	117.5%
–	6.2%	26.1%	47.6%	70.0%	92.2%
10.0%	0.0%	12.4%	30.6%	50.4%	70.8%
20.0%	0.0%	0.8%	17.4%	34.4%	52.7%

Table 22-7:: Post-Tax Sensitivity Analysis ($M CAD)

Sensitivity to Metal Price

Post-Tax NPV Sensitivity To Discount Rate

	Commodity Price				
Discount Rate	(20.0%)	(10.0%)	–	10.0%	20.0%
3.0%	($18)	$109	$233	$358	$482
5.0%	($15)	$95	$202	$309	$416
7.0%	($20)	$76	$169	$262	$354
8.0%	($23)	$67	$154	$240	$327
10.0%	($30)	$50	$127	$203	$278

Post-Tax IRR Sensitivity To Discount Rate

	Commodity Price				
Discount Rate	(20.0%)	(10.0%)	–	10.0%	20.0%
3.0%	0.0%	19.1%	35.2%	51.2%	67.2%
5.0%	2.6%	19.6%	35.4%	51.2%	67.2%
7.0%	3.6%	19.8%	35.4%	51.3%	67.2%
8.0%	3.8%	19.9%	35.4%	51.3%	67.2%
10.0%	4.2%	20.0%	35.5%	51.3%	67.2%

Post-Tax NPV Sensitivity To Opex

	Commodity Price				
Opex	(20.0%)	(10.0%)	–	10.0%	20.0%
(20.0%)	$90	$183	$275	$368	$461
(10.0%)	$36	$130	$222	$315	$408
–	($20)	$76	$169	$262	$354
10.0%	($80)	$21	$116	$209	$301
20.0%	($155)	($36)	$62	$155	$248

Post-Tax IRR Sensitivity To Opex

	Commodity Price				
Opex	(20.0%)	(10.0%)	–	10.0%	20.0%
(20.0%)	23.2%	39.1%	55.1%	71.0%	86.7%
(10.0%)	13.5%	29.4%	45.2%	61.2%	77.0%
–	3.6%	19.8%	35.4%	51.3%	67.2%
10.0%	0.0%	10.4%	25.8%	41.4%	57.3%
20.0%	0.0%	1.6%	16.6%	31.8%	47.4%

Post-Tax NPV Sensitivity To Initial Capex

	Commodity Price				
Initial Capex	(20.0%)	(10.0%)	–	10.0%	20.0%
(20.0%)	($10)	$86	$179	$271	$364
(10.0%)	($15)	$81	$174	$267	$359
–	($20)	$76	$169	$262	$354
10.0%	($25)	$71	$164	$257	$350
20.0%	($30)	$66	$159	$252	$345

Post-Tax IRR Sensitivity To Initial Capex

	Commodity Price				
Initial Capex	(20.0%)	(10.0%)	–	10.0%	20.0%
(20.0%)	5.1%	23.8%	43.0%	62.9%	82.7%
(10.0%)	4.3%	21.6%	38.8%	56.4%	74.1%
–	3.6%	19.8%	35.4%	51.3%	67.2%
10.0%	2.9%	18.2%	32.6%	47.0%	61.6%
20.0%	2.3%	16.9%	30.2%	43.5%	56.8%

Post-Tax NPV Sensitivity To Recovery Mill

	Commodity Price				
Recovery Mill	(20.0%)	(10.0%)	–	10.0%	20.0%
(20.0%)	($177)	($72)	$8	$84	$158
(10.0%)	($91)	$4	$89	$173	$256
–	($20)	$76	$169	$262	$354
10.0%	$44	$147	$249	$350	$452
20.0%	$105	$217	$328	$439	$551

Post-Tax IRR Sensitivity To Recovery Mill

	Commodity Price				
Recovery Mill	(20.0%)	(10.0%)	–	10.0%	20.0%
(20.0%)	0.0%	0.0%	8.3%	21.0%	33.5%
(10.0%)	0.0%	7.7%	22.0%	36.0%	50.3%
–	3.6%	19.8%	35.4%	51.3%	67.2%
10.0%	14.5%	31.7%	49.1%	66.6%	84.0%
20.0%	24.8%	43.7%	62.9%	81.9%	100.5%

Post-Tax NPV Sensitivity To Head Grade

	Commodity Price				
Head Grade	(20.0%)	(10.0%)	–	10.0%	20.0%
(20.0%)	($209)	($109)	($24)	$51	$122
(10.0%)	($107)	($10)	$74	$156	$238
–	($20)	$76	$169	$262	$354
10.0%	$55	$160	$263	$367	$471
20.0%	$129	$243	$358	$473	$587

Post-Tax IRR Sensitivity To Head Grade

	Commodity Price				
Head Grade	(20.0%)	(10.0%)	–	10.0%	20.0%
(20.0%)	0.0%	0.0%	3.0%	15.4%	27.3%
(10.0%)	0.0%	5.3%	19.5%	33.2%	47.1%
–	3.6%	19.8%	35.4%	51.3%	67.2%
10.0%	16.4%	33.9%	51.6%	69.4%	87.0%
20.0%	28.9%	48.4%	68.2%	87.7%	106.7%

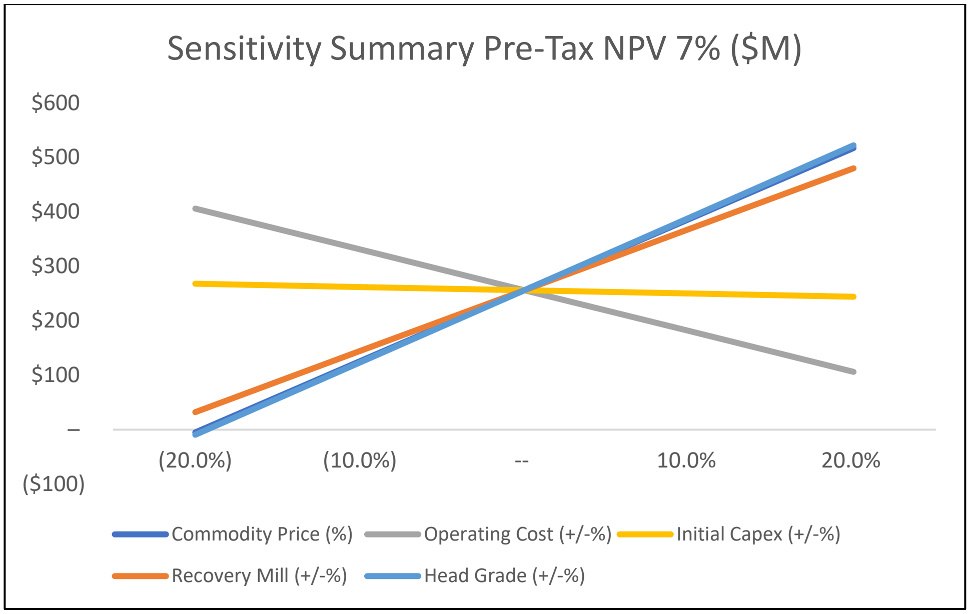
Post-Tax NPV Sensitivity To Exchange Rate

	Commodity Price				
Exchange Rate	(20.0%)	(10.0%)	–	10.0%	20.0%
(20.0%)	$136	$252	$368	$484	$600
(10.0%)	$51	$154	$257	$360	$463
–	($20)	$76	$169	$262	$354
10.0%	($83)	$11	$97	$181	$265
20.0%	($148)	($47)	$36	$114	$191

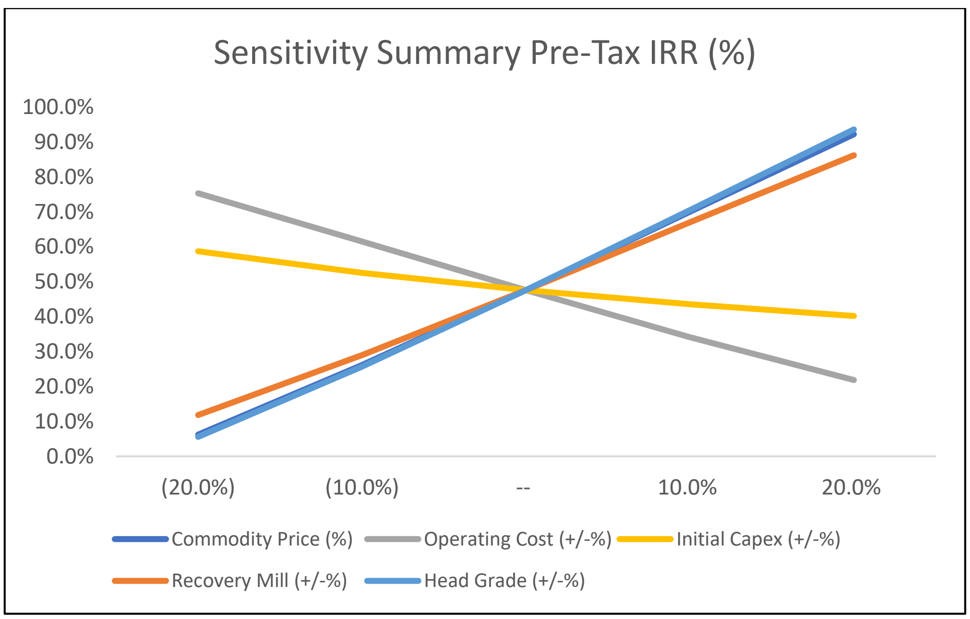
Post-Tax IRR Sensitivity To Exchange Rate

	Commodity Price				
Exchange Rate	(20.0%)	(10.0%)	–	10.0%	20.0%
(20.0%)	30.0%	49.8%	69.7%	89.4%	108.5%
(10.0%)	15.6%	33.0%	50.6%	68.3%	85.9%
–	3.6%	19.8%	35.4%	51.3%	67.2%
10.0%	0.0%	8.8%	23.2%	37.4%	51.8%
20.0%	0.0%	0.0%	13.0%	26.0%	39.0%

Figure 22-2: Pre-Tax NPV, IRR Sensitivity Results

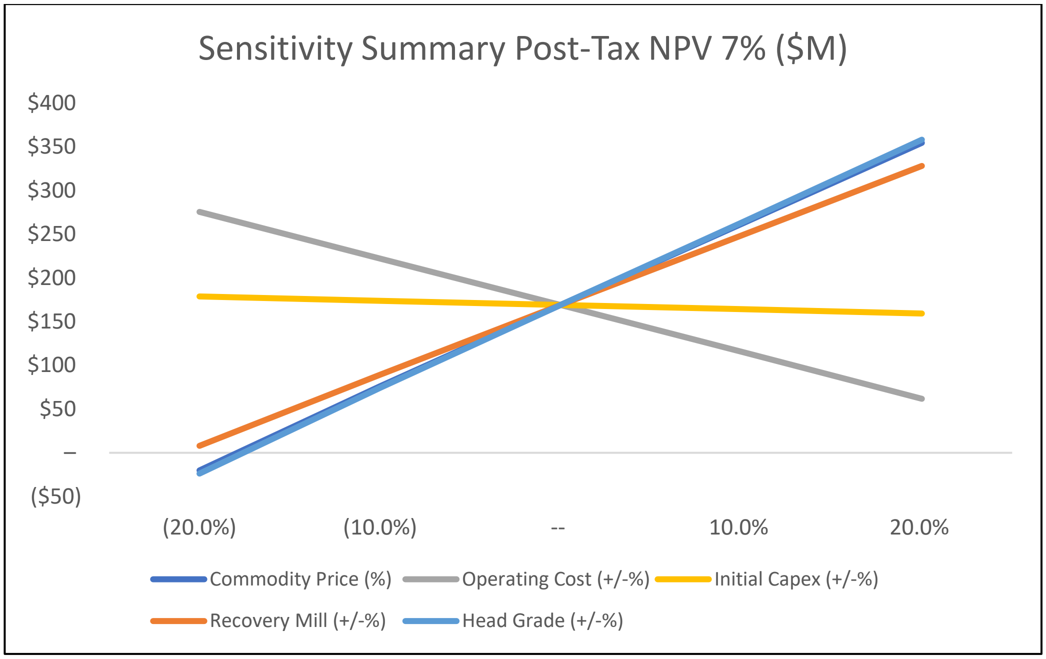


Note: Series lines for commodity price and head grade overlap in the above figures.
Source: Ausenco, 2025.

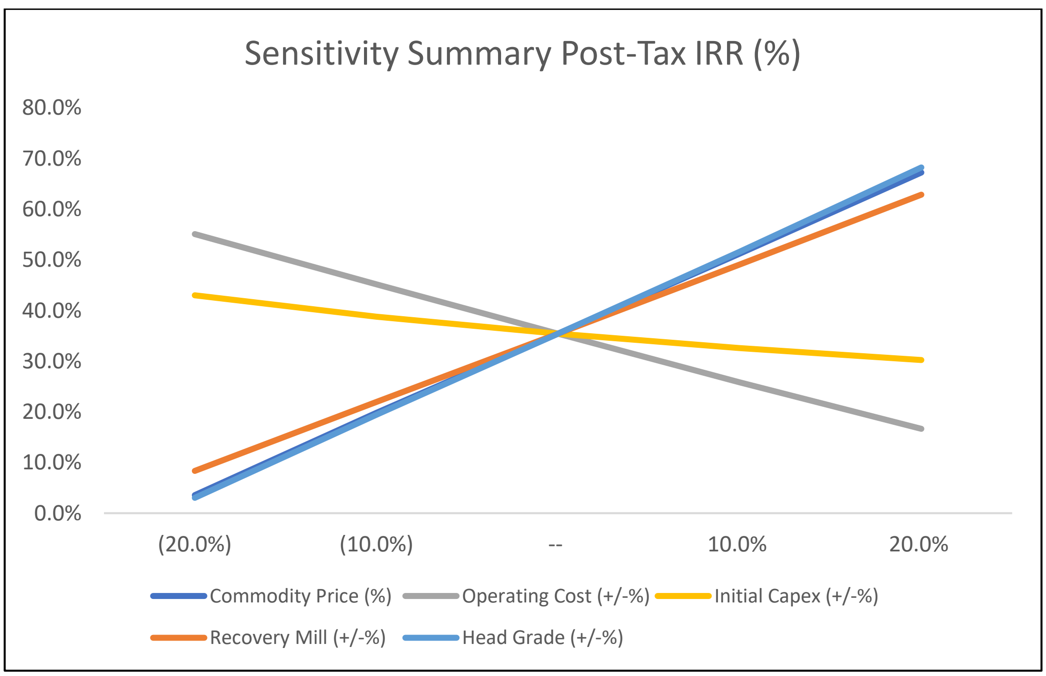


Note: Series lines for commodity price and head grade overlap in the above figures
Source: Ausenco, 2025.

Figure 22-3: Post- Tax NPV, IRR Sensitivity Results



Note: Series lines for commodity price and head grade overlap in the above figures. Source: Ausenco, 2025.



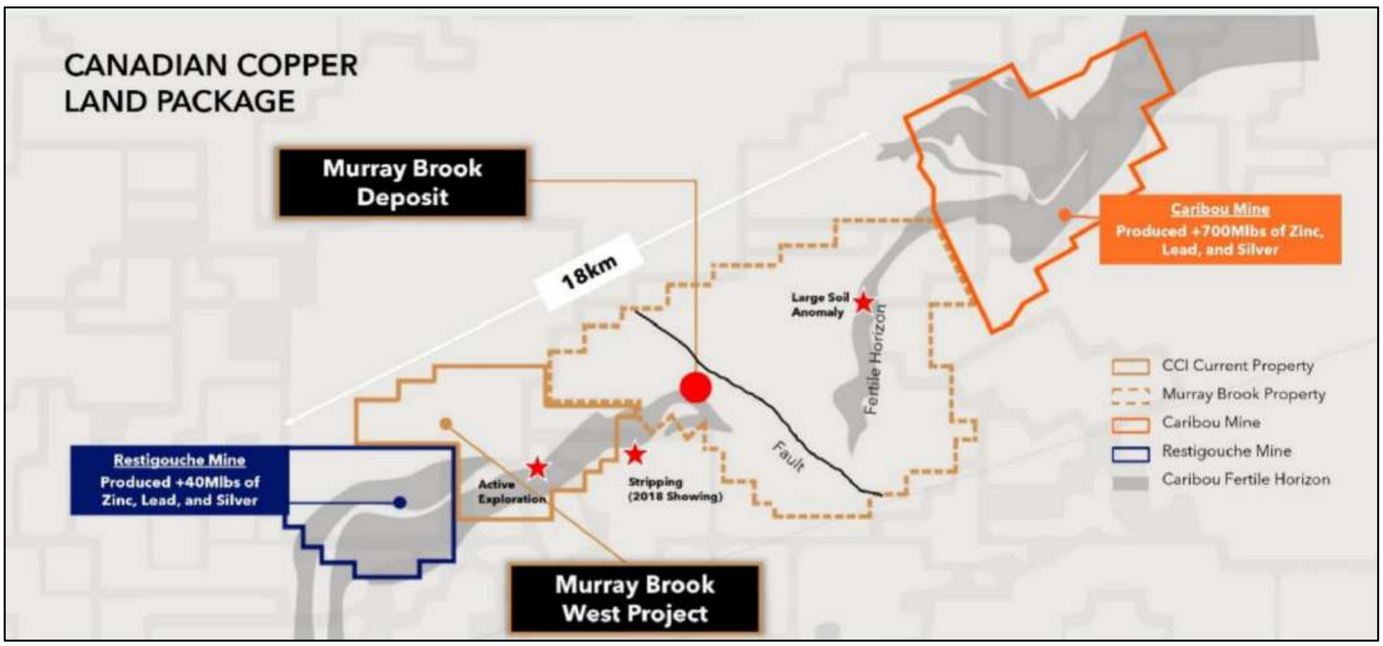
Note: Series lines for commodity price and head grade overlap in the above figures. Source: Ausenco, 2025.

23 ADJACENT PROPERTIES

The Murray Brook deposit is hosted by the Ordovician California Lake Group, an important host to mineralization in the Bathurst mining camp. Murray Brook is in the northwest part of the camp and is located approximately 42 km west-northwest of the decommissioned Bathurst No. 12 mine, formerly the largest producer in the camp. In the vicinity of Murray Brook, the California Lake Group has a regional strike direction of approximately 70°.

Excluding the Caribou Mine property (see Section 4 of this report), the most important property adjacent to Murray Brook is the Restigouche mine, which is 10 km to the west of the Murray Brook property (Figure 23-1). The geology and mineralization at the Restigouche mine are broadly similar to the Murray Brook deposit.

Figure 23-1: Properties Adjacent to Murray Brook



Source: Canadian Copper website (July 7, 2025).

The Restigouche massive sulphide deposit consists of at least two separate lenses of massive sulphide underlain by a chlorite-pyrite stringer zone. Historically, the Restigouche mine produced 756,000 tonnes grading 6.45% Zn, 4.87% Pb, and 107 g/t Ag in the late 1990s and in 2008. Trevali Mining Corporation (Trevali) acquired the Restigouche mine in July 2017. Trevali released an updated mineral resource statement for the underground Restigouche mine dated December 31, 2021, in their press release on March 31, 2022. The measured and indicated mineral resources consisted of 1.08 Mt grading 5.00% Zn, 3.30% Pb, 0.22% Cu, 46.30 g/t Ag and 0.52 g/t Au and the inferred mineral resources consisted of 0.58 Mt grading 6.10% Zn, 4.30% Pb, 0.28% Cu, 67.83 g/t Ag and 0.81 g/t Au (Table 23-1).

Table 23-1: Restigouche Project Mineral Resources as of December 31, 2021

Classification	Tonnes (M)	Zn (%)	Pb (%)	Cu (%)	Ag (g/t)	Au (g/t)	Zn (Mlb)	Pb (Mlb)	Cu (%)	Ag (koz)	Au (koz)
Measured	0.29	4.63	3.08	0.21	38.80	0.45	30	20	1.4	364	4
Indicated	0.79	5.19	3.36	0.22	49.07	0.55	91	59	3.8	1,249	14
Measured & Indicated	1.08	5.00	3.30	0.22	46.30	0.52	119	79	5.3	1,613	18
Inferred	0.58	6.10	4.30	0.28	67.83	0.81	77	55	3.6	1,256	15

Notes: **1.** All mineral resources have been estimated in accordance with the CIM Definition Standards. Mineral resources are inclusive of mineral reserves. Mineral resources that are not mineral reserves do not have demonstrated economic viability. Numbers may not add due to rounding. **2.** The Restigouche underground mine mineral resource estimate is reported based on zinc equivalent cut-off grade of 3% ZnEq. The Restigouche underground mine mineral resource estimate has been prepared by Trevali's exploration geology department and non-independent technical consultants with an effective date of December 31, 2021, under the supervision of and approval by Yan Bourassa (P. Geo.), a qualified person as defined in NI 43-101. Mr. Bourassa was Vice President Technical Services & Exploration of Trevali and accordingly, is not independent. Source: Trevali press release dated March 31, 2022.

The author cautions that a qualified person has not done the work necessary to verify the validity or reliability of the mineral resource estimate. The key assumptions and parameters used to prepare the estimate have not been verified and, as such, the estimate should not be relied upon.

Magna Terra Minerals Inc. (Magna Terra) announced in their press release dated February 27, 2025 that they had acquired an option on the Restigouche mine property. Magma Terra commenced exploration work on the property in May 2025.

The author has not done sufficient work to verify the information about the mineral properties discussed in this report section and the information in this section of the report is not necessarily indicative of the mineralization on the Murray Brook property.

24 OTHER RELEVANT DATA AND INFORMATION

This section is not relevant to this report.

25 INTERPRETATION AND CONCLUSIONS

25.1 Property Description and Location

The Murray Brook property is located approximately 60 km west of Bathurst in the Parish of Balmoral, Restigouche County, New Brunswick, Canada. The property consists of surveyed Mineral Lease No. 252, which covers 505 ha, and the larger Murray Brook east claim 4925, which covers 5,082 ha. A 5 km gravel access road extends southward from Highway 180 to the property. The City of Bathurst to the east provides access to rail and ocean shipping facilities.

25.2 Mineral Tenure, Surface Rights, Water Rights, Royalties and Agreements

Mineral Lease No. 252 was recorded on October 17, 1989 by Murray Brook Resources Inc. The initial term of the Lease was for 20 years with three automatic 20-year renewals. The current expiry date is October 16, 2029 and the rental fees are current. Murray Brook East claim 4925 is currently in good standing with an expiry date of September 7, 2025. Canadian Copper acquired the Murray Brook Joint Venture property through separate agreements with VMC and MQM in 2023 for 100% ownership of the property.

25.3 Accessibility, Climate, Local Resources, Infrastructure and Physiography.

On October 28, 2024, Canadian Copper announced the signing of a term sheet and exclusivity agreement providing the company the exclusive right to acquire the Caribou processing plant complex, 10 km to the east of Murray Brook. The Caribou complex would be utilized to produce copper, lead, and zinc concentrates with recoverable silver from Murray Brook mineralized feed material. The closing date for the transaction is October 1, 2025.

The property is located in the Miramichi Highlands, which is characterized by rounded and glacially scoured hills. Land use in the area is mainly for tourism, forestry, and mining. The property is accessible for work year-round and water is plentiful in streams and creeks. Although the site has been reclaimed from the 1990s silver-gold mining and the equipment sold off, part of the historical open pit mine, leach pads and tailings management area are still evident. The massive sulphide deposit has never been mined. A 10 kV powerline links Murray Brook to the power station at the Caribou mine, 10 km to the east. However, service was discontinued in 1996.

25.4 Geology and Mineralization

The Murray Brook area is located in the Bathurst mining camp in northern New Brunswick. The Bathurst mining camp is an Ordovician back-arc complex of poly-deformed sedimentary, felsic volcanic, and mafic volcanic rocks formed in separate sub-basins within the back-arc basin, and which have been juxtaposed by five periods of folding and thrusting, and collectively referred to as the Bathurst Supergroup. The sedimentary and volcanic rocks have been intruded by gabbro, diabase, and quartz porphyritic rocks of Ordovician age.

The Murray Brook deposit is hosted in sedimentary rocks of the Charlotte Brook Member, in the lower part of the Mount Brittain Formation. The upper felsic volcanic member of the Mount Brittain Formation hosts to Restigouche deposit, 10 km to the west of Murray Brook. The Mount Brittain Formation is considered to be equivalent of the Spruce Lake Formation, which hosts the Caribou mine deposit, 10 km to the east.

25.5 Deposit Types

Although Murray Brook is a single body of massive sulphide, infill drilling results indicate that the deposit consists of two connected thick lenses or lobes. The western lens is deeper and richer in zinc and lead, and the eastern lens is closer to surface and richer in copper.

25.6 Exploration and Drilling

Mineral prospecting, geological mapping, trenching, soil geochemistry, ground geophysical surveys, and airborne geophysical exploration surveys were completed by VMC and Puma between 2010 and 2020. The ground and airborne geophysical surveys included magnetics, gravity and electromagnetics, which, in combination with ongoing geological and geochemistry work, led to the development of targets for drill testing. Since 2010, 190 drill holes for 37,070 m have been completed on the Murray Brook property: 162 drill holes for 29,938.7 m were completed by VMC between 2010 and 2016; and 28 drill holes totalling 7,131 m were completed by Puma between 2017 and 2019.

25.7 Sampling, Analyses and Data Verification

Robust quality assurance and quality control (QA/QC) programs have been used since the start of exploration activities on the property in 2010. In the author’s opinion that the sample preparation, analytical and security procedures, and QA/QC program meet industry standards, and that the data are of good quality and satisfactory for use in the mineral resource estimate reported in this technical report.

Mr. Yungang Wu, P.Geo., of P&E, an independent qualified person as defined under NI 43-101, conducted a site visit to the Murray Brook property on September 7 and 8, 2023. The site visit included an inspection of the property, drill sites, drill collars, and drill core storage facilities. A data verification sampling program was completed as part of the on-site review. Drill core samples were collected to independently confirm the presence and grades of base and precious metal mineralization. Previously, Mr. Eugene Puritch, P.Eng., of P&E, an independent qualified person under the terms of NI 43-101, conducted a site visit to the property on March 18, 2013. A data verification sampling program was conducted as part of the on-site review.

The authors consider that there is good correlation between gold and silver assay values in the Project database and the independent verification samples. It is the authors’ opinion that the data are of good quality and appropriate for use in the current mineral resource estimate.

25.8 Metallurgical Testwork

The metallurgical testwork programs completed to date indicated the need for excluding an oxidized layer sitting above the deposit. How the boundaries of such layer should be defined has yet to be determined since visual evidence of oxidation is not sufficient.

The earlier testwork phase (RPC 2012) was the only one that benefited from the availability of fresh samples whereas the subsequent work (2019, 2024-25) made use of cores dating back from 2013 or earlier. Despite efforts made to exclude visually degraded material from the intercepts used in the sample composition, some degree of oxidation of the cores is likely to have occurred over that period of time. It is unclear therefore if some of the shortcomings seen in these later testwork campaigns, and especially the most recent one performed for Canadian Copper, are the results of sample aging or the incorporation of some material from the oxidized zone.

Additional confirmatory work is recommended and should involve fresh material from contemporary drill cores and from intercepts taken at sufficient depth to ensure the exclusion of oxidized material from the top layer of the deposit.

The zinc-lead zone yielded recoveries of 86.6% Zn and 70.4% Pb with 39.9% of silver reporting to the lead concentrate. These results are higher than the published results by Trevali for the Caribou (underground) mine and are also significantly higher than those for the initial recovery tests conducted by VMC in 2013.

For the first time, recovery tests were conducted on the newly defined copper-zinc zone at Murray Brook. The copper-zinc zone also reported positive results with initial copper recovery of 79.5 % Cu with 54.9% of silver reporting to the copper concentrate and zinc recovery of 65.6%. Further optimization testwork was recommended to evaluate the potential to further improve recoveries and reduce reagent consumption.

25.9 Mineral Resource Estimate

A mineral resource estimate was prepared in accordance with NI 43-101 for the sulphide and oxide mineralization at a C$23/t NSR cut-off. The mineral resource estimate for the sulphide mineralization consists of 15.8 Mt grading 2.60% Zn, 0.43% Cu, 0.92% Pb, 0.52 g/t Au, and 39.0 g/t Ag (4.83% ZnEq or 1.51% CuEq) in measured mineral resources; 5.3 Mt grading of 2.14% Zn, 0.52% Cu, 0.85% Pb, 0.67 g/t Au, and 37.3 g/t Ag (4.58% ZnEq or 1.43% CuEq) in indicated mineral resources; and 0.1 Mt grading 1.82% Zn, 0.41% Cu, 0.68% Pb, 0.62 g/t Au, and 30.4 g/t Ag (3.75% ZnEq or 1.17% CuEq) in inferred mineral resources. The oxide mineralization consists of 1.6 Mt grading 2.2% Zn, 1.05% Cu, 0.73% Pb, 0.36 g/t Au, and 38.0 g/t Ag (5.94% ZnEq or 1.85% CuEq) in measured mineral resources and 0.4 Mt grading 2.31% Zn, 0.97% Cu, 0.78% Pb, 0.51 g/t Au, and 44.7 g/t Ag (6.02% ZnEq or 1.88% CuEq) in indicated mineral resources.

The mineral resources in this report were estimated using the Canadian Institute of Mining, Metallurgy and Petroleum (CIM), CIM Standards on Mineral Resources and Reserves, Definitions and Guidelines prepared by the CIM Standing Committee on Reserve Definitions and adopted by the CIM Council (2014) and Best Practices Guidelines (2019). Mineral resources, which are not mineral reserves, do not have demonstrated economic viability. The estimate of mineral resources may be materially affected by environmental, permitting, legal, title, taxation, socio-political, marketing, or other relevant issues. The inferred mineral resource in this estimate has a lower level of confidence that that applied to an indicated mineral resource and must not be converted to a mineral reserve. It is reasonably expected that the majority of the inferred mineral resource could be upgraded to an indicated mineral resource with continued exploration.

25.10 Mining Methods

The Murray Brook deposit is near the surface and will be mined by conventional open pit mining methods. A contractor will be engaged to mine the open pit. The contractor will undertake all drilling and blasting, loading, hauling, and mine site maintenance activities. The Owner will provide overall mine management and technical services.

Mining will be accomplished on 6 m high benches using conventional equipment such as hydraulic excavators, front-end loaders, and 65 t haulage trucks. Dilution in mineralized material is estimated at 3% and mining losses are estimated at 5%. The overall strip ratio for open pit mining is 5:1.

Over the life of mine, a total of 15.5 Mt of mineralized material will be mined from open pits at average grades of 41.1 g/t Ag, 0.47% Cu, 0.96% Pb, and 2.67% Zn. The average life-of-mine NSR is estimated at $103.5/t. Waste rock will be transported to nearby storage facilities, and mineralized material will be hauled to a temporary stockpile at the Caribou process plant.

25.11 Recovery Methods

The proposed project processing strategy involves processing mineralized material from the Murray Brook deposit at the existing Caribou processing complex 10 km east of the Murray Brook deposit. The process flowsheet design is based on the current Caribou concentrator process flow design, with minimal modification to enable a 10% increase in mill feed design capacity (3,000 to 3,300 t/d) and the recovery of a copper concentrate (in addition to the zinc and lead concentrates recovered in the historical operation). Equipment sizing considers the re-use of existing equipment to optimize project economics.

The recovery methods chosen for processing the mineralized materials are supported by preliminary historical testwork and operating data, industrial standard practices, and financial evaluations to optimize concentrate recovery while minimizing capital expenditure and life-of-mine operating costs. The unit operations are standard technologies widely used in polymetallic concentrators.

25.12 Project Infrastructure

25.12.1 Caribou Site General Infrastructure

The existing main site infrastructure at Caribou includes access roads connecting to route 180, a tailings management facility, a high-voltage electrical substation, backup diesel generator, fire water pond, process facilities, and a workshop, assay laboratory and administration building.

Infrastructure changes to the Caribou site include adding a transformer to support increased electrical demand from the copper separation circuit and expanding the concentrate storage building.

25.12.2 Murray Brook Site Water Management

Murray Brook site water management will include diversion and collection ditches around the NAG, PAG, and overburden stockpiles to direct water to collection ponds. The NAG and overburden stockpile ponds have been sized for 1-in-10-year, 24-hour storm events. The PAG stockpile pond is sized for a 1-in-100-year, 24-hour storm event.

25.13 Environmental, Permitting and Social Considerations

The key considerations related to environmental, permitting, and social aspects of the proposed project are summarized below:

- Field studies at the Murray Brook site have been initiated in 2025 including bird, bat and terrestrial wildlife surveys. The desktop studies completed assumed a study area that consisted of a 5 km buffer area around the proposed project.
- There are numerous streams and waterbodies in the project area; some of which will be impacted by the project footprint. Currently, the characteristics regarding fish and fish habitat as well as habitat water quality and hydrology for the Murray Brook study area are not known.
- The potential changes in the groundwater regime at the Murray Brook site are not understood at this time, especially in regard to the potential drawdown of groundwater in the area during mining operations and the potential impact to nearby water bodies, wetlands, and existing groundwater users. Groundwater monitoring has been conducted at the Murray Brook site since 1991 and this data should be reviewed and considered in the development of future groundwater monitoring plans.
- According to the desktop Species at Risk (SAR) study, 11 species at risk were identified, consisting of one mammal, nine birds, and one vascular plant. The study area includes a portion of the project footprint that intersects with an area of critical habitat for Bicknell’s thrush which is a bird listed as threatened and imperilled. It is currently uncertain as to how the project will interact with the critical habitat from a permitting perspective.
- It is anticipated that a SARA permit will be required should Project activities interact negatively with critical Bicknell’s thrush habitat.
- It may be possible to adjust the project footprint to avoid fish habitat and to minimize interactions with SAR.
- The permitting, environmental, and social aspects regarding the use of the Caribou mine facilities and its potential liabilities are not fully understood in consideration that those aspects and assigned responsibilities would be determined following detailed investigations and agreements with the Province of New Brunswick. In particular, the potential environmental liabilities of the site have not been fully investigated, nor the responsibilities assigned.
- The Murray Brook Project, based on the current development strategy, would be subject to mostly provincial regulatory requirements. The project will not likely trigger a federal impact assessment under the *Impact Assessment Act*. It is possible that some of the proposed project related activities will trigger a HADD (harmful alteration, disruption, or destruction of fish habitat) and fish habitat compensation plan, as well as a wetland offset plan.

- At minimum, it is likely that an EIA registration will be required for the Murray Brook project. An EIA registration will require environmental field studies followed by preparation and submission of an EIA Registration document. The likely outcome of the EIA is a Certificate of Determination (approval of the project with conditions) It is likely that the Murray Brook project would receive a certificate of determination, in consideration that the site is an existing historical mining site, though this cannot be guaranteed.
- It is unlikely that the project as envisioned in the short term (first 3 years) will trigger a Schedule 2 Amendment under the Metal and Diamond Mining Effluent Regulations, however, in the longer term, a Schedule 2 Amendment may be triggered due to the requirement later in the project life for a new tailings management facility that would accommodate the volumes of feed to be processed over the life-of-mine.
- The nearest First Nation community to the project area is the Pabineau First Nation (PFN) community, located near Bathurst. In October 2024, Canadian Copper signed a non-binding memorandum of understanding with the PFN regarding their participation in the Murray Brook project. Historically, the previous owner of the Caribou mine had agreements with multiple First Nation Mi'kmaq communities.
- Reclamation for the Canadian Copper assets at the Caribou mine site include demolition of the existing water treatment plant, milling complex, laboratory, and administration building as well grading and revegetation of these areas. It is understood that other areas and liabilities are the responsibility of others as part of Canadian Copper's limited liability agreement discussions with the Province of New Brunswick.
- Due to the acid-generating nature of the tailings, and in alignment with the existing closure strategy for the Caribou mine, the proposed closure concept for the TMFs includes raising the dam to provide an enhanced water cover over the tailings and to submerge historical liabilities such as waste rock, legacy tailings, and the underground mine portal to reduce impacts from these sources.
- Reclamation activities related to the TMFs at this stage include various upgrades to dams and sludge cells at the site which are related to water treatment as well as long-term monitoring. Perpetual water treatment has been assumed as part of the closure plan, unless future monitoring and site conditions demonstrate that treatment is no longer necessary.
- The Murray Brook mineral lease is the site of past production from open-pit mining of the oxidized surficial portion of the Murray Brook deposit. The current liability associated with the Murray Brook property consists of short-term monitoring liability and long-term restoration liability. Additional liability will be incurred once the Murray Brook site is developed including flooding of the open pit, water treatment, pit water overflow via a closure spillway, management of PAG waste rock, vegetative covers on disturbed land, and dismantling and removal of site buildings, and re-contoured and revegetating.

25.14 Capital Cost Estimates

The capital cost estimate conforms to Class 5 guidelines for a PEA-level estimate with a ±50% accuracy according to the Association for the Advancement of Cost Engineering International (AACE International). The capital cost estimate was developed in Q2 2025 Canadian dollars based on Ausenco's in-house database of projects and studies as well as experience from similar operations.

The estimates are based on the following:

- open pit mining operation with stockpiles and associated water management
- restart of an existing process plant
- completion of the installation of copper separation circuit
- construction of the North Tributary tailings pond
- additional on-site and off-site infrastructure
- Owner's costs and provisions.

The total initial capital cost for the Canadian Copper project is $63.7 million, and life-of-mine sustaining costs are $48.9 million. Closure costs are estimated at $52.6 million.

25.15 Operating Cost Estimates

The overall life-of-mine operating cost is C$1,228 million over the 13-year mine life, or C$79.32/t of mineralized material milled. The operating cost estimates for the life of mine are provided in Table 25-1.

Table 25-1: Operating Cost Summary (Life-of-Mine Average)

Cost Area	Total (C$M/a)	C$/t Processed	% of Total
Mining	48.9	41.7	52.5
Processing	36.6	31.2	39.3
G&A	7.6	6.5	8.2
Total	**93.1**	**79.3**	**100.0**

Source: Ausenco, 2025.

25.16 Economic Analysis

An engineering economic model was developed to estimate the project's annual pre-tax and post-tax flows and sensitivities based on an 7% discount rate.

The analysis used the following key inputs:

- mine life of 13.2 years
- exchange rate of 0.746 (CAD:USD)
- base case metal prices of US$27.00/oz Ag, US$4.25/lb Cu, US$1.10/lb Pb, and US$1.30/lb Zn

- cost estimates in constant Q2 2025 Canadian dollars with no inflation or escalation factors considered
- results based on 100% ownership with a combined 1.25% net smelter return royalties and $3 million of completion payments due after production begins
- capital costs funded with 100% equity (no financing costs assumed).

The pre-tax NPV discounted at 7% is $264 million, the IRR is 47.6%, and the payback period is 1.6 years. On a post-tax basis, the NPV discounted at 7% is $169 million, the IRR is 35.4%, and the payback period is 2.0 years.

A sensitivity analysis was conducted on the base case post-tax NPV and IRR of the project using metal prices, discount rate, foreign exchange rates, operating cost, and initial capital cost as the variables. The sensitivity analysis revealed that the project is most sensitive to changes in metals price and foreign exchange, and less sensitive to operating cost and initial capital cost.

The preliminary economic assessment is preliminary in nature, that it includes 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 preliminary economic assessment will be realized.

25.17 Adjacent Properties

The most important adjacent property to Murray Brook is the Restigouche mine property 10 km to the west. The Restigouche mine deposit consists of at least two separate lenses of massive sulphides underlain by a chlorite-pyrite stringer zone. The geology and mineralization at the Restigouche mine are broadly similar to the Murray Brook deposit (and the Caribou deposit 12 km farther east), which altogether define the fertile Caribou Horizon that has been traced on surface for a distance of 18 km in the northwestern part of the Bathurst mining camp.

Historically, the Restigouche mine produced a total of 756,000 tonnes grading 6.45% Zn, 4.87% Pb, and 107 g/t Ag in the late 1990s and in 2008. Trevali acquired the Restigouche mine in July 2017 and released an updated mineral resource statement (December 31, 2021) for the underground Restigouche mine, as announced in their March 31, 2022 press release. The measured and indicated mineral resources consisted of 1.08 Mt grading 5.00% Zn, 3.30% Pb, 0.22% Cu, 46.30 g/t Ag, and 0.52 g/t Au and the inferred mineral resource consisted of 0.58 Mt grading 6.10% Zn, 4.30% Pb, 0.28% Cu, 67.83 g/t Ag, and 0.81 g/t Au.

Magna Terra Minerals Inc. announced in their press release dated February 27, 2025 that they had acquired an option on the Restigouche mine property. Magma Terra commenced exploration work on the property in May 2025.

The QP has been unable to verify the information and the information is not necessarily indicative of the mineralization on the property that is the subject of this technical report.

25.18 Risks and Opportunities

25.18.1 Risks

25.18.1.1 Mineral Resource Estimate

Future metal prices could cause a revision of the mineral resource estimate.

25.18.1.2 Metallurgical Testing

The metallurgical testwork does not as yet hold a sufficient database of hardness to indicate the variability that may exist within the deposit. This may warrant for an eventual modulation of the expected plant's annual throughput capability, or reduced recovery from a coarser primary grind if throughput targets are maintained.

The testwork performed to date, with the proposed configuration of the flotation circuit, has yet to demonstrate in full its capability at delivering the final concentrate grades projected or the basis for recovery projections used for economic calculations.

Lacking reliable final concentrate samples from recent testwork, and not having available samples of such from earlier programs, means there may be minor elements that will require the payment of penalties by the eventual buyers of the concentrates. Arsenic is of particular concern, because its grade to the copper concentrate may approach a level where blending may be required, thus forcing the use of an intermediary to sell the product.

25.18.1.3 Mining Methods

There have been no geotechnical or hydrogeological studies completed for the Murray Brook deposit. Open pit slopes and water inflow quantities may differ than those estimated by the QP.

25.18.1.4 Recovery Methods

Metal recovery and concentration in the concentrates have not yet been successfully demonstrated at the levels proposed in the process design and recovery models. Optimal reagent selection and consumption have also not been confirmed by testwork. Additional testing is required to confirm metallurgical performance and the amenability of the mineralized material to the proposed flowsheet.

The grinding circuit is designed for a feed size F_{80} of 106 mm. Deviations from this will affect the grinding circuit capacity and capabilities. The ability of the primary crushing equipment to meet the required size specifications at the stated throughout should be confirmed with the crushing service provider.

The Caribou concentrator consistently has not processed mineralized material at the design rates due to insufficient material delivery from the previous underground Caribou mine. There are potentially unforeseen challenges with the material handling systems, which may require capital expenditure to resolve.

25.18.1.5 Project Infrastructure

Trevali stopped operations in 2022 at the Caribou process plant; however, since then personnel have been on site maintaining the facility. While an assessment of the facility has been conducted, until operations restart and commissioning is carried out, there is a risk of unknown or unforeseen infrastructure issues that may result in schedule delays or increased capital costs beyond the estimates in this report.

The Port of Belledune formerly shipped the concentrate produced by the Caribou process plant; however, the infrastructure at the port was not inspected for this PEA, and there is the potential risk that the existing facilities may require repair before being recommissioned. Additionally, if not being used, the existing infrastructure may be decommissioned prior to the start of production and future studies should include further review of the port facilities and any contracts that may be required prior to production.

A new site access road is required before the North Tributary tailings dam closure configuration is completed and modifications to the existing access road or new access road may be required during operation. This will need to be addressed during the next phase of the project.

Tailings properties for Murray Brook based on limited test data and need to be confirmed via additional testing.

The Caribou mine is a complex site with existing infrastructure, the new tailings facility design, and closure concepts are based on flooding existing liabilities including the portal of the underground mine and require treatment, maintenance and monitoring in perpetuity. These concepts require additional studies and design to be proven in subsequent pre-feasibility and feasibility level studies including water quality assessment and hydrogeological modelling.

Water treatment capital and operating costs are based on partial effluent treatment to meet minimum MDMER guidelines. Water quality in the tailings pond is unknown at this time and conceptual water treatment to be advanced with additional studies.

25.18.1.6 Mining Costs

The mining costs reported in this PEA have been based on a single quote from a local contractor. Multiple quotes should be obtained to provide higher cost accuracy.

25.18.1.7 Environmental, Permitting and Social Considerations

Canada and New Brunswick have rigorous regulatory processes for permitting that involves many stakeholders and rightsholders that could impact the timing and approval of the project.

In the absence of comprehensive field-based environmental baseline studies, it is not possible to definitively identify and qualify the level of risk related to groundwater, mine rock geochemistry, fish and fish habitat, wetlands, and Species at Risk (SAR).

Unanticipated impacts to fish and fish habitat that cannot be readily compensated for, resulting in difficulties in receiving Fisheries Authorization and/or MDMER Schedule 2 amendment.

Regarding the Caribou mine site, due to the limited availability of historical environmental monitoring reports prior to acquisition of the mill, data and other information such as reportable spills, non-compliance events, and other incidents, it was not possible to fully assess environmental risk and liabilities associated with the operational history of the site. Although it is not likely that the project will trigger a federal impact assessment under IAAC (2019), there remains a minor risk in the event of significant Indigenous or climate change related concerns.

25.18.2 Opportunities

25.18.2.1 Metallurgical Testing

Additional testwork may provide a way to overcome the issue of zinc activation seen with the oxide layer, allowing this material to be included in future mine plans.

25.18.2.2 Mining Methods

The process plant feed within the open pit design consists of 15.5 Mt at 79.1% measured mineral resource, 20.9% indicated mineral resource and no inferred mineral resource, which provides a high level of confidence in the in-pit mineral resources.

25.18.2.3 Recovery Methods

There may be an opportunity to optimize reagent usage while maintaining target recoveries. Metallurgical testing is required to confirm this.

There is an opportunity to optimize process water recovery and reduce tailings volumes by installing a tailings thickener. This addition would require a review of the tailings dam design and deposition strategy, as well as rheological profiling to quantify the impact and benefits of this change.

25.18.2.4 Project Infrastructure

The existing Caribou underground mine is equipped with pumps, motors, and electrical equipment that may be used as spares or to complete the existing copper circuit. A more detailed assessment of the existing equipment should be carried out to identify its potential uses.

The existing water treatment facility uses expensive hydrated lime. An opportunity exists to convert the facility to use quicklime. Using quicklime would require the installation of a lime slaker; however, overall operating costs would be reduced.

25.18.2.5 Project Economics

The economic analysis was conducted based on three NSR royalties—0.33% with MQM, 0.67% with MQM, and 0.25% with VMC—for a combined royalty payment of 1.25%. The 0.67% royalty with MQM was included in the study based on limited available documentation; additional documentation is required to confirm.

25.18.2.6 Environmental and Permitting

The early initiation of field-based studies and associated analyses/modelling will help better inform potential risks and options for mitigation, and likely reducing permitting timelines.

Similarly, the acceleration of engagement activities with government, community, and Indigenous nations will likewise reduce permitting and timeline risks.

Cost savings could be achieved by coordinating field and lab work such as the drilling, sampling and testing between the exploration, geotechnical, geochemical, hydrogeological teams, as well as coordination of sampling and analytical test work on process and metallurgy/tailings with the geochemistry team.

26 RECOMMENDATIONS

26.1 Introduction

The work carried out to date has justified the continued exploration and development of the project. Work is recommended to advance the project with additional drilling and a pre-feasibility study, to allow for additional metallurgical testwork and to improve the mineral resource estimate.

The estimated cost of the recommended program is $5.2 million (Table 26-1).

Table 26-1: Recommended Program and Budget for Murray Brook

Activity	Purpose	Units	Quantity	Estimated Cost (C$k)
Drilling	Further deposit delineation	Meters	3,000	750
Metallurgical Testwork	PFS-level testwork	Testwork program	1	954
Exploration	Review and drill nearby targets	Meters	2,000	500
Geotechnical and Hydrogeological Studies	Further geotechnical and hydrogeological understanding	-	-	500
Environmental Baseline Studies	Environmental baselining for Murray Brook site	-	-	800
PFS Study Budget	Pre-feasibility study	-	-	750
Support		-	-	300
Subtotal		-	-	4,554
Contingency (15%)		-	-	683
Total		-	-	**5,237**

26.2 Exploration and Geology

The Murray Brook Project contains a significant zinc-lead-copper-silver mineral resource and the authors recommend that Canadian Copper proceed with a pre-feasibility study. It is also recommended that Canadian Copper proceed with the following:

- additional drilling to better delineate the western and at depth margins of the deposit
- further metallurgical testwork to potentially recover gold
- property environmental studies and project stakeholder engagement work for future permitting
- regional exploration to generate new targets for drill testing.

Specifically, it is recommended that Canadian Copper take the following actions:

- A 12-hole, 3,000 m drill program should be completed to determine the extent of a recently recognized copper and gold zone defined previously by 12 of the 2017 to 2019 drill holes, which are an approximate 50-metre step-out to prior drilling.
- Future drill core sampling should include the insertion and monitoring of suitable CRMs to monitor gold analyses and umpire assaying of 5% of all drill core samples.
- Process options should be reviewed to potentially recover the 400,000 oz gold that exists within the current oxide and sulphide mineral resources. Presently, no recovery is attributed for gold for economic purposes.
- The proposed project should be reviewed with regulatory authorities and local communities (this may include but not be limited to carrying out environmental and social impact assessments and public consultation activities). It may be beneficial to proactively carry out studies that are likely to be required or that may require an extended time, whilst also recalling that environmental assessment supporting studies requirements are established as part of the environmental impact assessment process.
- Regional exploration along the 18 km long Caribou horizon corridor should be reinitiated.

26.3 Metallurgical Testing

It is recommended to carry out additional metallurgical testwork on a suite of fresh samples. The testwork program should include the development of a complete reagent regime related to the bulk flotation and copper-lead separation circuit on a limited number of composite samples, culminating with locked cycle tests.

The mineralogy of variability samples should be assessed, including precious metal point-count analysis to explain deportment of silver and gold through the flotation circuit. The same samples should be tested for material hardness to confirm the robustness of the standard flotation regime via locked cycle tests. A full suite of minor elements should be assayed from various representative samples of each of the final concentrates.

The work outlined above would carry an estimated cost of $954,000, as detailed in Table 26-2.

A bulk composite should be floated in larger batches to accumulate concentrates, confirm regrinding power demand, and confirm filtration and thickening equipment sizing as well as tailings for environmental testing. Since the proposed overall flotation circuit is complex, it will involve multiple circulating loads. As well, pilot plant testing with either material from the full core of large diameter boreholes or from a starter pit is recommended. These assessments may be delayed until the project reaches the feasibility study stage though.

Table 26-2: Recommended Metallurgical Testwork Program and Budget for Pre-Feasibility Phase

Activity	Purpose	Units	Quantity	Estimated Cost (C$k)
Sample Preparation, Storage, Head Assays	Sizing for subsequent test phases, metal contents	Samples	15	42
Hardness Determination	For SMC, rod and ball mill work indices; signature plot for regrinding	Samples	15	81
Open Circuit Flotation Testwork	Optimization of reagent regime, selection of regrind size, retention times	Tests	45	276
Locked Cycle Tests	Confirmation of open circuit indication, effect of circulating loads on circuit performance	Tests	15	180
Mineralogy / Liberation Analysis	Confirm minerals, grain size, precious metals deportment basis	Samples	15 heads 30 tails	100
Test Management / Supervision / Reporting		%	22	150
Contingency		%	15	124
Total		-	-	**954**

26.4 Mining Methods

The authors recommend that Canadian Copper continue with project development activities on the Murray Brook property and proceed with a pre-feasibility study. To advance the project and initiate a pre-feasibility study, hydrogeological studies are recommended, in addition to additional drilling to develop a better understanding of geotechnical (open pit wall slopes and waste storage facility foundation locations) considerations. Costs are estimated at $300,000 for drilling and $200,000 for geotechnical and hydrogeological studies.

26.5 Recovery Methods

Additional metallurgical testwork to advance the project should include the following:

- Comminution testing on variability samples that provide spatial coverage of the deposit and sufficiently represent the materials to be encountered. This testing should be completed on ½ HQ drill core so that SMC tests can be conducted. Bond ball mill Wi tests should also be conducted on all samples.
- Bench-scale flotation testing on master composites and variability samples should be carried out to provide sufficient spatial coverage of the deposit. The testing should include open and locked cycle testing.
- Primary and regrind size, as well as reagent usage, should be optimised to reduce operating costs.
- Regrind energy tests should be carried out on the concentrates at optimum grind sizes to confirm the capabilities of the regrind mills.

26.6 Project Infrastructure

26.6.1 Copper Circuit Installation Completion

The completion of the partially installed copper circuit was conducted to a PEA-level estimate on the equipment and work required to finish the installation.

- Further engineering should be completed on the partially installed equipment and new equipment to determine equipment, piping, instrumentation, electrical equipment, concrete, and steel requirements.
- Existing geotechnical information related to the concentrate building expansion should be reviewed. If it is unavailable, appropriate geotechnical investigation should be conducted.

26.6.2 Caribou Site Water Treatment Facility

The existing water treatment facility uses hydrated lime to control the pH of the treated water. The following is recommended:

- An economic comparison should be conducted on the use of hydrated lime with the existing equipment versus using quicklime with new slaking equipment. The result of the comparison may indicate savings would be realized with an alternative flowsheet.
- A geotechnical investigation should be carried out for the Murray Brook water treatment facility.

26.6.3 Existing Equipment in Caribou Underground Mine

The existing Caribou underground mine contains equipment (i.e., pumps and electrical) that are not required to process the Murray Brook mineralized material. It is recommended that the existing electrical equipment and pumps be reviewed to determine if they can be repurposed or used as spares.

26.7 Environmental, Permitting, and Community

The following recommendations are made regarding future studies and activities related to areas of environment, permitting and community engagement. These studies and activities will be necessary to support the project to the pre-feasibility stage and provide a strong basis for future environmental assessment preparation and permitting. The recommended studies, presented below, are sufficient to take the project through the next project stage, but not necessarily through the environmental assessment preparation and permitting. The recommended studies would provide a start to this work, but further scoping studies would be required.

- Data Review – A comprehensive desktop review should be conducted of all pertinent data available for historical operations at Caribou mine and Murray Brook site to identify useful data that can be used to support EA, permitting applications, and future feasibility design. This would include identifying past non-compliances/causes, and horizontal and vertical extent of legacy contamination issues based on the desktop review. This information could then be used as a basis for planning and implementing an existing conditions environmental site assessment in the field for the Caribou and Murray Brook sites.

- Water Resources
  - For the Murray Brook and Caribou sites, review current surface and groundwater monitoring programs under existing approvals to operate, for adequacy, focusing on areas that will be potentially affected by mine infrastructure based on proposed infrastructure plans (refer to Section 18) and the requirements of an EA submission. Additional surface and groundwater monitoring may be recommended for both sites.
  - Hydrogeological testing of monitoring wells should be carried out to support groundwater inflow estimates for the Murray Brook open pit.
  - Consideration should be given to installing a professional grade meteorological station at the project site to monitor local conditions.
  - Consideration should be given to the development of a conceptual groundwater model for both the Murray Brook and Caribou Mine sites. This should be supplemented by a well survey in the area adjacent to the project which will identify current groundwater users who may be impacted by the project.
- Geochemical Characterization Program
  - Existing geochemical data collected for the Murray Brook and Caribou Mine sites should be reviewed and a gap assessment conducted. Based on the results of the gap assessment, additional geochemical testing and modelling may be required to support EA and provide guidance on the requirements of PAG and NAG waste rock storage. This program should be conducted in accordance to the Canadian Industry standard guidance for geochemical characterization described by MEND (2009).
  - Types of analyses to be considered for additional test work would include: acid base accounting and available metals; shake flask extraction (short-term leach); NAG pH; mineralogy; humidity cell testing (minimum 40 weeks).
- Aquatics, Fish and Fish Habitat and Wetlands
  - Site-based fish and fish habitat sampling and assessments are recommended for the areas of proposed project disturbance. Crews will need to initially conduct further studies to identify fish-bearing waterbodies and watercourses and where there is potential for a direct loss of habitat or harmful alteration, disruption, or destruction. Also, to survey areas where mine wastes are to be stored to assess whether there is potential to trigger a Schedule 2 Amendment under MDMER.
- Terrestrial and Wildlife Monitoring
  - Surveys will need to be completed related to the areas of vegetation/ecosystem and wildlife/wildlife habitat for the mine infrastructure presented in Section 18. In particular, a critical area of focus should be the portion of the study area that intersects with an area of critical habitat for Bicknell’s thrush. Early engagement with regulators on this issue receiving their input on the proposed monitoring plan is recommended. This work is reported to be currently underway.

- Socio-Economic, Cultural Baseline Studies and Community/Government Engagement
  - Continued engagement with First Nations from the perspective of cooperation, information sharing with the Mi'kmaq nations in New Brunswick with an emphasis on those Indigenous communities that are nearby or have historically and currently been active within or near the study area.
  - An archaeological walkover should be completed on locations of proposed project infrastructure and in areas that are identified as high
  - Commence engagement with local communities, nearby businesses and landowners, recreational users, and NGOs who are interested or who have concerns about the project.
  - Initiate a socio-economic study that will focus on impacts (both positive and negative) of the operation on the local and regional study areas.
  - Engage with the provincial and federal governments regularly and in an effort to understand where concerns may arise.
- Other Studies
  - Dustfall monitoring stations and other monitoring methods should be conducted to establish baseline airborne dust and particulate levels. Review the historical database to see if monitoring data are available from previous operations.
  - Greenhouse gas studies should be initiated to identify ways the project can reduce energy use and its carbon footprint.

27 REFERENCES

Baily, D.C. and Hudson, D.I. (1991). The Neves-Corvo Project – The success and challenges. Institution of Mining and Metallurgy, Transactions, v. 100, A1-A10.

Bulatovic, S. M. (2007). Handbook of flotation reagents: Chemistry, Theory and practice. Volume 1: Flotation of Sulphide Ores. Elsevier Science.

Desrosier, C. (2007). Technical Report on the Camel Back Property, Restigouche County, Province of New Brunswick, Canada. Prepared for Murray Brook Minerals Inc. by Christian Derosier Geologue-Conseil Inc. Dated October 24, 2007. 81 pages.

Desrosier, C. (2008). Technical Report on the Murray Brook Property, Restigouche County, Province of New Brunswick, Canada. Prepared for Murray Brook Minerals Inc. by Christian Derosier Geologue-Conseil Inc. Dated October 22, 2008. 135 pages.

Dimroth, E., Imreh, L., Goulet, N. and Rocheleau, M. (1983). Evolution of The South-Central Segment of the Archean Abitibi Belt, Québec; Part II, Tectonic Evolution and Geomechanical Model. Canadian Journal of Earth Sciences 20, 1355-1373.

DNR. (2013). http://dnr-mrn.gnb.ca/Lexicon/Lexicon/Lexicon_Search.aspx?lang=e.

El Nino Ventures Inc., News Release April 17, (2012). El Nino Ventures Inc. Announces SEDAR Filing of NI 43-101 Mineral Resource Estimate for the Murray Brook Zn-Cu-Pb-Ag-Au Deposit, Bathurst Mining Camp.

El Nino Ventures Inc., News Release April 24, (2012). Exploration Update on the Bathurst Mining Camp Base Metal Project, Tri-Party Agreement, New Brunswick

El Nino Ventures Inc., News Release. August 14, (2012). El Nino Ventures Inc. Announces a 4.3% Zinc, 1.4% Lead and 54 g/t Silver over 81 Metres at the Murray Brook Polymetallic Deposit, Bathurst Mining Camp, New Brunswick.

El Nino Ventures Inc., News Release. August 28, (2012). El Nino Ventures Inc. Announces a 122-metre-thick mineralized interval of massive sulphide mineralization assaying 6.15% Zn, 0.10% Cu, 1.92% Pb, 0.64 gpt Au and 70.6 gpt Ag at the Murray Brook Deposit, New Brunswick.

El Nino Ventures Inc., News Release. August 30, (2011). Significant drill results continue to be intersected in the Murray Brook deposit, the fifth largest massive sulphide deposit in the Bathurst Mining Camp.

El Nino Ventures Inc., News Release. February 28, (2012). El Nino Ventures Inc. Announces an Initial NI 43-101 Mineral Resource Estimate for the Murray Brook Zn-Cu-Pb-Ag-Au Deposit, Bathurst Mining Camp, NB.

El Nino Ventures Inc., News Release. January 16, (2012). El Nino Ventures Inc. Continues to Intersect High Grade Zinc, Lead and Silver Over Broad Intercepts at the Murray Brook Deposit in the Bathurst Mining Camp.

El Nino Ventures Inc., News Release. January 16, (2013). El Nino Releases Phase 1 Metallurgical Results for the Murray Brook Base Metal Deposit, Bathurst Mining Camp, New Brunswick.

El Nino Ventures Inc., News Release. January 23, (2012). El Nino Ventures Announces Additional High-Grade Zinc, Lead and Silver Over Broad Intervals at the Murray Brook Deposit, Bathurst Mining Camp.

El Nino Ventures Inc., News Release. July 17, (2012). El Nino Ventures Inc. Drills 9.23% Zinc, 108.7 gpt Silver, 0.18% Copper, 0.336 gpt gold and 2.58% lead over 31 metres at the Murray Brook Polymetallic Deposit, Bathurst Mining Camp, New Brunswick.

El Nino Ventures Inc., News Release. July 30, (2012). El Nino Ventures Inc. Announces a 10.34% Zinc, 126.0 gpt Silver, 0.29% Copper, 1.396 gpt Gold and 4.11% Lead drill intersection over 13 Metres at Murray Brook Polymetallic Deposit, Bathurst Mining Camp, New Brunswick.

El Nino Ventures Inc., News Release. June 27, (2012). El Nino Ventures Inc. continues to intersect High Grade Zinc, Gold and Silver over significant intervals at Murray Brook Polymetallic Deposit, Bathurst Mining Camp, New Brunswick.

El Nino Ventures Inc., News Release. June 5, (2012). El Nino Ventures Inc. Announces High Grade Zinc, Gold and Silver intersections over significant intervals from the Phase II 18,000 metre Drill Program at Murray Brook Polymetallic Deposit, Bathurst Mining Camp, New Brunswick.

El Nino Ventures Inc., News Release. March 27, (2012). 2012 Exploration Program on the Murray Brook Polymetallic Base Metals Project, New Brunswick.

El Nino Ventures Inc., News Release. May 11, (2011). Up to 27 Metres (88.6 ft) of 9.56% Zinc; 122.1 g/t (3.56 opt) Silver; 0.1 g/t Gold and 0.47% Copper in Three-Hole Due Diligence Drill Program at Murray Brook Massive Sulphide Deposit, New Brunswick.

El Nino Ventures Inc., News Release. May 11, (2012). El Nino Announces Initial 50% Earn-in for Murray Brook Project, Bathurst, New Brunswick; Gives Notice to Earn Additional 20%.

El Nino Ventures Inc., News Release. May 17, (2012). El Nino Ventures Inc. Announces High Grade Zinc, Lead and Silver on Phase II 18,000 metre Drill Program at Murray Brook Polymetallic Deposit, in the Bathurst Mining Camp, New Brunswick.

El Nino Ventures Inc., News Release. November 28, (2011). El Nino Ventures Reports High Grade Zinc, Lead and Silver over Broad Intercepts at Murray Brook Deposit, in the Bathurst Mining Camp.

El Nino Ventures Inc., News Release. October 2, (2012). El Nino Ventures Inc. announces a 45.4 metre drill intersection assaying 8.49% Zinc, 0.18% Copper, 4.58% Leas, 0.59 gpt Gold and 152.2 gpt Silver at the Murray Brook Deposit, New Brunswick.

El Nino Ventures Inc., News Release. September 6, (2012). El Nino Ventures Inc. Announces completion of its 35% earn-in; Receives Notice of option to earn additional 15% (for a total of 50%) in Murray Brook Polymetallic Project, New Brunswick, Canada.

Exceptional Mineral Potential of Québec, A Reality Yet to be Discovered; Information Seminar (1993), Conference Abstracts. DV – Direction Generale de l'Exploration Geologique et Minerale. 49-50.

Franklin, J.M. (1993). Volcanic-Associated Massive Sulphide Deposits. In Kirkham, R.V., Sinnclair, W.D., Thorpe, R.I. and Duke, M.J. eds, Mineral Deposit Modelling. Geological Association of Canada, Special Paper 40, 315-334.

Franklin, J.M. (1996). Volcanic-Associated Massive Sulphide Base Metals; Geology of Canadian Mineral Deposit Types, (ed.) O.R. Eckstrand, W. D. Sinclair and R. I. Thorpe; Geological Survey of Canada, No. 8, 158-183.

Franklin, J.M. (2007). Volcanogenic Massive Sulphide Deposits Part 1: Classification and Model. VMS Short Course "Exploration for Volcanic Massive Sulphide Deposits", Manitoba Mining and Minerals Convention, November 2007.

Franklin, J.M., Lydon, J.W. and Sangster, D.F. (1981). Volcanic-Associated Massive Sulphide Deposits. Economic Geology 75th Anniversary Volume, 485-627.

Gagné, D and Hupé, A. (2017). Assessment Report 478240 Murray Brook East Claim Block 4925. 103 pages.

Gagné, D and Hupé, A. (2018). Assessment Report 478485 Murray Brook East Claim Block 4925. 91 pages.

Gagné, D and Hupé, A. (2019). Assessment Report 478732 Murray Brook East Claim Block 4925. 129 pages.

Galley, A.G., Bailes, A.H. and Kitzler, G. (1993. Geological Setting and Hydrothermal Evolution of the Chisel Lake and North Chisel Zn-Pb-Ag-Au Massive Sulphide Deposit, Snow Lake, Manitoba: Exploration and Mining Geology 2, 271–295.

Galley, A.G., Hannington, M. and Jonasson, I. (2007). Volcanogenic Massive Sulphide Deposits. In Goodfellow, W.D., ed., Mineral Deposits of Canada: A synthesis of Major Deposit Types, District Metallogeny, The Evolution of Geological Provinces, And Exploration Methods. Geological Association of Canada, Mineral Deposits Division, Special Publication 5, 141-161.

Galley, A.G., Syme, E.C., and Bailes, A.H., (2007a), Metallogeny of the Paleoproterozoic Flin Flon Belt, Manitoba and Saskatchewan, in Goodfellow, W.D., ed., Mineral Deposits of Canada: A Synthesis of Major Deposit Types, District Metallogeny, the Evolution of Geological Provinces, and Exploration Methods: Geological Association of Canada, Mineral Deposits Division, Special Publication No. 5, p. 509-531.

Gibson, H.L. and Watkinson, D.H. (1990). Volcanogenic Massive Sulphide Deposits of the Noranda Cauldron and Shield Volcano, Québec, in Rive, M., Verpaelst, P., Gagnon, Y., Lulin, J.M., Riverin, G., and Simard, A., eds., The Northwestern Québec Polymetallic Belt. Canadian Institute for Mining and Metallurgy Special Volume 43, 119-132.

Golder (2018a). Preliminary Guidance on a Reasonable Range of Whittle Angles for a Murray Brook Open Pit Shell. Project No. 18101667 by Golder Associates Ltd to Trevali Mining Corporation. 14 pages.

Golder (2018b). Combined Groundwater Seepage and Direct Seepage Estimate for the Murray Brook Open Pit Mine, Steady State Condition and Assumed Average Parameters. Project No. 18101667 by Golder Associates Ltd to Trevali Mining Corporation. 12 pages.

Gower, S.J. and van Staal, C.R. (1997). Geology of the Murray Brook area (NTS 21 O/09d). Restigouche and Northumberland Counties, New Brunswick. New Brunswick Department of Natural Resources and Energy, Minerals and Energy Division, Plate 97-18.

Graves, Garth, P. Geo. July to June (2015) Assessment Report, Votorantim Metals Canada Inc.

Graves, Garth, P. Geo. October to July 2015) - August 2016 Assessment Report, Votorantim Metals Canada.

Harris, Robert and Peter Pheeney. (2001). Encapsulation of Mercury in Gossan Tailings, Murray Brook Mine, New Brunswick, Canada. Technical Paper for 33rd Canadian Mineral Processors Operators Conference. January 2001.

Harris, Robert and Peter Pheeney. (2000) Undated. Murray Brook Mining Project…from cradle to grave. MBRI 2000 Presentation.

Harron, G.A. (2012). Technical Report on the Murray Brook Property, Restigouche County, New Brunswick, Canada for El Nino Ventures Inc. Prepared by G.A. Harron & Associates Inc. Signing date April 13, 2012. 95 pages.

Helmstaedt, H. (1973). Structural Geology of the Bathurst-Newcastle District: New England Intercollegiate Geol. Conf., Geology of New Brunswick, University of New Brunswick Field Guide to Excursion Trip 5-A, 34-46.

Jacques Whitford and Associates Limited. (1999). Preparation for Closure, Murray Brook Resources Inc., Murray Brook Mine, Bathurst, NB. Project No. 11452. Report dated Oct 1999.

Jacques Whitford and Associates Limited. (2007). Murray Brook Mine Data Review. Presentation by Nicolette Stanley, M.Sc., MIT and dated April 3, 2007.

Klohn Crippen Berger. (2025). Caribou South Tributary Tailings Pond Remaining Tailings Storage Capacity. Submitted to New Brunswick Department of Transportation and Infrastructure. Dated May 6, 2025.

Large, R.R. (1992). Australian Volcanic-Hosted Massive Sulphide Deposits: Features, Styles, and Genetic Models. Economic Geology 87, 471–510.

Lydon, J.W. (1988). Volcanogenic Massive Sulphide Deposits, Part 1: A Descriptive Model; in Ore Deposit Models, Geological Association of Canada, edited by Roberts, R.G. and Sheahan, P.A., 145-154.

McCutcheon, S. R., and Walker, J. (2009). Stratigraphic Setting of Base Metal Deposits in the Bathurst Mining Camp (BMC), New Brunswick, Field Trip Guidebook, 67 pages.

McCutcheon, S.R. (1992). Base Metal Deposits of the Bathurst-Newcastle District: Characteristics and Depositional Models. Exploration and Mining Geology 1, 105-119.

McCutcheon, S.R., Walker, J., Bernard, P., Lentz, D., Downey, W. and McClenaghan, S. (2005). Stratigraphic Setting of Base Metal Deposits in the Bathurst Mining Camp, New Brunswick, Field Trip B4 Guidebook, GAC/MAC Ann Mtg. Halifax, Nova Scotia, 105 pages.

Morton, J.L. and Franklin, J.M. (1987). Two-Fold Classification of Archean Volcanic-Associated Massive Sulphide Deposits. Economic Geology 82, 1057-1063.

P&E. (2013). Technical Report and Preliminary Economic Assessment (PEA) of the Murray Brook Project, New Brunswick, Canada. Prepared for Votorantim Metals Canada Inc. and El Nino Ventures Inc. by P&E mining Consultants Inc. Signing date July 19, 2013. 180 pages.

P&E. (2017). Amended and Restated Technical Report and Updated Mineral Resource Estimate on the Murray Brook Project, New Brunswick, Canada. Prepared for Puma Exploration Inc. by P&E Mining Consultants Inc. Signing date February 20, 2017. 135 pages.

P&E. (2023). Technical Report and Updated Mineral Resource Estimate of the Murray Brook Zn-Pb-Cu-Ag Project, New Brunswick, Canada. Prepared for Canadian Copper Inc. by P&E Mining Consultants Inc. Signing date November 16, 2023. 226 pages.

Peltier, C. and Beausoleil, C. (2006). NI43-101 Technical Report on the Caribou and Restigouche Properties, Blue Note Metals Inc. 133 pages.

Perusse, J. (1957). Final Report on Murray Brook Project (1956, Unpub. Kennco Explorations (Canada) Limited, Internal Report, NovaGold Resources Inc.

Poulsen, H. and Hannington, M. (1995). Auriferous Volcanogenic Sulphide Deposits, in Eckstrand, O.R., Sinclair, W.D. and Thorpe, R.I., eds.

Powell, W.G., Carmichael, D.M. and Hodgeson, C.J. (1993). Relative Timing of Metamorphism and Tectonism During the Evolution of the Southern Abitibi Greenstone Belt. in Caty, J.L., ed.

Rennick, M. M. and Burton, D.M. (1992). The Murray Brook Deposit, Bathurst Camp, New Brunswick: Geologic setting and recent developments. Exploration and Mining Geology 1, 137-142

Roscoe Postle Associates, Inc. December 31, (2017). Technical Report on the Caribou Mine, Bathurst, New Brunswick, Canada.

RPC. (2012). Final Report Re: Murray Brook VMS Zn-Cu-Pb-Ag-Au Deposit, Scoping Metallurgical Test Program, Report Reference No: MIS-J1859, dated November 27, 2012. Prepared for Votorantim Minerals Canada Inc. by Gilders, R. and Cheung, L., P.Eng. 245 pages.

RPC. (2019). Murray Brook Metallurgical Testing Program; Trevali Mining (New Brunswick) Limited. Reference No. MIS-J2079, dated February 28, 2019. Prepared by Botha, N., P.Eng. and Cheung, L., P.Eng. 101 pages.

Shanks III, P., Cynthia Dusel-Bacon, W.C., Kosky, R., Morgan, L.A., Mosier, D., Piatak, N.M., Ridley, I., Seal II, R.R., Schulz, K.J., Slack, J.F. and Thurston, R. (2009). A New Occurrence Model for National Assessment of Undiscovered Volcanogenic Massive Sulphide Deposits, USGS Open File Report 2009-1235, 27 pages.

SRK. (2013). Independent Technical Report for the Caribou Massive Sulphide Project, Bathurst, New Brunswick, Canada. Prepared for Trevali Mining Corporation by Dr. Gilles Arsenault of SRK Consulting (Canada) Inc., dated November 21, 2013. 98 pages.

Stantec Consulting Ltd. (2012). Environmental Monitoring in Support of Condition 14 of Approval to Operate (I-6920) Issued to Murray Brook Resources Inc., Stantec file 121910870 Task 200, February 6, 2012.

Stantec Consulting Ltd. (2015). Environmental Impact Assessment (EIA) Registration: Caribou Mine Copper Circuit. Prepared for Trevali Mining Corporation by Stantec Consulting Ltd. Project No. 121616495, dated February 13, 2015

Thomas, M.D., Walker, J.A., Keating, P., Shives, R., Kiss, F. and Goodfellow, W.D. (2000). Geophysical Atlas of Massive Sulphide Signatures, Bathurst Mining Camp, New Brunswick; Geological Survey of Canada Open File D3887, New Brunswick Department of Natural Resources and Energy, Minerals and Energy Division, Open File 2000-9.

Urabe, T., Marumo, K. and Nakamura, K. (1990). Mineralization and Related Hydrothermal Alteration in Izena Cauldron (Jase site), Okinawa Trough, Japan [abstract]. Geological Society of America, Abstracts with Programs 22, p. A9.

Van Staal, C.R., Wilson, R.A., Rogers, N., Fyffe, L.R., Langton, J.P., McCutcheon, S.R., McNicoll, V. and Ravenhurst, C.E. (2003). Geology and Tectonic History of the Bathurst Supergroup, Bathurst Mining Camp, and Its Relationships to Coeval Rocks in Southwestern New Brunswick and Adjacent Main – A Synthesis. Economic Geology Monograph 11, 2003, 37-66.

Van Stall, C.R., Fyffe, L.R., Langton, J.P. and McCutcheon, S.R. (1992). The Ordovician Tetagouche Group, Bathurst Camp, Northern New Brunswick, Canada: History, Stratigraphy, and Tectonic Setting of Massive Sulphide Deposits. Exploration and Mining Geology 1, 93-104.

Von Stakelburg, U. and Brett, R. (1990). Extended Hydrothermal Activity in the Lau Basin (southwest Pacific): First results of R.V. Sonne cruise, 1990 [abstract]. EOS 71, 1680.