

Opémiska Project

NI 43-101 Technical Report and Preliminary Economic Analysis

Quebec, Canada

Effective Date: October 17, 2025

Report Date: November 27, 2025

Prepared for:

XXIX Metal Corp.

Suite 1102, 141 Adelaide Street W.

Toronto, ON, Canada, M5H 3L5

Prepared by:

Ausenco Engineering Canada ULC

15th Floor, 11 King Street West

Toronto, Ontario, M5H

List of Qualified Persons:

Renée Barrette, ing., Ausenco Engineering Canada ULC

Alexandre Burelle, ing., Evomine Consulting Inc.

Stephen Coates, ing., Evomine Consulting Inc.

Maude Lévesque Michaud, ing., Geodoz Conseil Inc.

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

Jean- François St-Laurent, ing., SRK Consulting (Canada) Inc.

Charles Veilleux, ing., SRK Consulting (Canada) Inc.



CERTIFICATE OF QUALIFIED PERSON
Renée Barrette, ing.

I, Renée Barrette, ing., certify that:

1. I am employed as a Principal Metallurgist with Ausenco Engineering Canada ULC, (Ausenco), with an office address of Suite 1550 – 11 King St West, Toronto, ON, M5H 4C7.
2. This certificate applies to the technical report titled “NI 43-101 Technical Report and Preliminary Economic Analysis of the Opémiska Project, Québec, Canada” that has an effective date of October 17, 2025 and a report date of November 27, 2025 (the “Technical Report”).
3. I graduated from Laurentian University with a Bachelor of Applied Science degree in Extractive Metallurgical Engineering in 2001.
4. I am a professional engineer registered with the Ordre des ingénieurs du Québec (No. 6019759).
5. I have practiced my profession continuously for 24 years with experience in development, design, operation, and commissioning of mineral processing plants, focusing on Gold, Base Metals, and other PGM projects, both domestic and internationally. To name a few specific examples, I have completed multiple due diligence and design reviews on Gold processing facilities in Val d’Or, including the O3 Mining Marban Project, Radisson O’Brien Project and Probe Novador Project. I have also completed design reviews to commissioning on a 17,000 mt/d Base Metals Project in Sudbury, Ontario.
6. 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.
7. I have visited the project site on October 27, 2025.
8. I am responsible for Sections 1.1, 1.2, 1.12, 1.15, 1.21, 2.1, 2.2, 2.3, 2.4.1, 2.5, 2.8, 12.4, 13, 17, 18.1, 18.2, 18.3, 18.7, 21.2.4, 21.2.6, 21.3.4, 21.3.6, 25.1, 25.6, 25.9, 25.10.1, 25.16.1.1, 25.16.1.3.1, 25.16.2.1, 25.16.2.4, 26.1, 26.3, and 27 of this Technical Report.
9. I am independent of XXIX Metal Corp as independence is defined in Section 1.5 of NI 43-101.
10. I have not been previously involved with the Opémiska Project.
11. 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: November 27, 2025

/Signed/

Renee Barrette, ing.

CERTIFICATE OF QUALIFIED PERSON

Alexandre Burelle, P.Eng.

I, Alexandre Burelle, ing., certify that:

1. I am employed as a Senior Mining Engineer with Evomine Consulting Inc., (Evomine) with an office address of 419 des Hirondelles, Beloeil, Quebec, Canada.
2. This certificate applies to the technical report titled “NI 43-101 Technical Report and Preliminary Economic Analysis of the Opémiska Project, Québec, Canada” that has an effective date of October 17, 2025 and a report date of November 27, 2025 (the “Technical Report”).
3. I graduated from McGill University with a Bachelor of Science in Mining Engineering in 2012 and from the Imperial College London with a Master of Science in Metals and Energy Finance in 2013.
4. I am a professional engineer registered with the Ordre des ingénieurs du Québec (No. 5019855).
5. I have practiced my profession continuously for 11 years with experience in mining operations, technical study delivery, due diligence, mine financing, business development, and strategic development. My work experience includes participating in the operation of the Bracemac-McLeod mine in Quebec, Canada and of the Renard diamond mine in Quebec, Canada, as well as participating in project development for the Back Forty project in Michigan, USA, the Gaspé Copper project in Quebec, Canada, the Iron Hills project in Quebec, Canada, and participating in authoring several NI 43-101 Technical Reports.
6. 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.
7. I have not visited the project site.
8. I am responsible for Sections 1.19, 1.20, 2.4.2, 21.1, 21.2.1, 21.2.2, 21.2.3, 21.2.7, 21.3.1, 21.3.2, 21.3.3, 21.3.5, 22, 25.13, 25.14, and 25.15 of this Technical Report.
9. I am independent of XXIX Metal Corp as independence is defined in Section 1.5 of NI 43-101.
10. I have not been previously involved with the Opémiska Project.
11. 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: November 27, 2025

/Signed/

Alexandre Burelle, P.Eng

CERTIFICATE OF QUALIFIED PERSON
Stephen Coates, ing.

I, Stephen Coates, ing., certify that:

1. I am employed as a Senior Mining Engineer with Evomine Consulting Inc., (Evomine), with an office address of 419 rue des Hirondelles, Beloeil, Quebec, Canada, J3G 6G8.
2. This certificate applies to the technical report titled “NI 43-101 Technical Report and Preliminary Economic Analysis of the Opémiska Project, Québec, Canada” that has an effective date of October 17, 2025 and a report date of November 27, 2025 (the “Technical Report”).
3. I graduated from McGill University, Montreal, Quebec with a Bachelor of Engineering in Mining Engineering in 2013.
4. I am a professional engineer registered with the Ordre de ingénieurs du Québec (No. 5047905).
5. I have practiced my profession continuously for 10 years with experience in mining operations, technical study delivery, due diligence, mine financing, business development, and strategic development.
6. 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.
7. I have visited the project site on October 21st, 2025.
8. I am responsible for Sections 1.14, 1.17, 2.4.3, 14.17, 15, 16, 18.8, 19, 25.8, 25.11, 25.16.1.2, 25.16.2.3, and 26.4 of this Technical Report.
9. I am independent of XXIX Metal Corp as independence is defined in Section 1.5 of NI 43-101.
10. I have not been previously involved with the Opémiska Project.
11. 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: November 27, 2025

/Signed/

Stephen Coates, P.Eng

CERTIFICATE OF QUALIFIED PERSON

Maude Lévesque Michaud, ing.

I, Maude Lévesque Michaud, ing., certify that:

1. I am employed as an engineer with Geodoz conseil Inc. (Geodoz), with an office address of 51 Quidoz, Sainte-Thérèse, Quebec, Canada, J7E 4L3.
2. This certificate applies to the technical report titled “NI 43-101 Technical Report and Preliminary Economic Analysis of the Opémiska Project, Québec, Canada” that has an effective date of October 17, 2025 and a report date of November 27, 2025 (the “Technical Report”).
3. I graduated from Laval University, Quebec City, QC with a Bachelor of Engineering in Geological Engineering in 2010 and from University of Quebec in Abitibi-Témiscamingue, Rouyn-Noranda, QC, with a Master of Applied Science in Mineral Engineering in 2016.
4. I am a professional engineer registered with the Ordre des Ingénieurs du Québec (No. 5015957).
5. I have practiced my profession continuously for 15 years with experience in exploration fieldworks from grassroots to advanced projects, as well as geochemical and environmental studies. My work experience includes participation in the development of various mining projects in Quebec : Troilus (gold-copper), Novador (gold), Casa Berardi extension (gold), Perron (gold), Iron Hills (iron), Rose (lithium-tantalum), Whabouchi (lithium), Fayolle (gold), Uatnan (graphite) and BlackRock (iron-vanadium).
6. 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.
7. I have visited the project site on October 21, 2025.
8. I am responsible for Sections 1.18, 2.4.4, 20, 25.12, 25.16.1.4, 25.16.2.5, and 26.6 of this Technical Report.
9. I am independent of XXIX Metal Corp. as independence is defined in Section 1.5 of NI 43-101.
10. I have not been previously involved with the Opémiska Project.
11. 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: November 27, 2025

/Signed/

Maude Lévesque Michaud, ing.

CERTIFICATE OF QUALIFIED PERSON

Pierre Luc Richard, P.Geo.

I, Pierre Luc Richard, P.Geo., certify that:

1. I am employed as a Geologist and Consultant with PLR Resources Inc., (PLR), with an office address of 2000 McGill College Avenue, Suite 600, Montreal, Quebec, Canada H3A 3H3.
2. This certificate applies to the technical report titled “NI 43-101 Technical Report and Preliminary Economic Analysis of the Opémiska Project, Québec, Canada” that has an effective date of October 17, 2025 and a report date of November 27, 2025 (the “Technical Report”).
3. I graduated from Université du Québec à Montréal with a Bachelor of Science in Resource Geology in 2004 and from Université du Québec à Chicoutimi with a Master of Science in Earth Sciences in 2012.
4. I am a professional geologist registered with the Ordre des Géologues du Québec (No. 1119), and the Professional Geoscientists Ontario (No. 1714), and the Northwest Territories Association of Professional Engineers and Geoscientists (No. L2465).
5. I have practiced my profession continuously for more than 20 years with experience in exploration and mining. I managed and QP’d numerous technical reports, mineral resource estimates, and audits as a consultant with different firms and for PLR Resources since 2022.
6. 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.
7. I visited the project site on May 1, 2025.
8. I am responsible for Sections 1.3 to 1.11, 1.13, 2.4.5, 2.6, 2.7, 3 to 11, 12.1, 12.2, 12.3, 12.5, 14 (except 14.17), 23, 24, 25.2, 25.3, 25.4, 25.5, 25.7, and 26.2 and 27 of this Technical Report.
9. I am independent of XXIX Metal Corp as independence is defined in Section 1.5 of NI 43-101.
10. I have not been previously involved with the Opémiska Project.
11. 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: November 27, 2025

/Signed/

Pierre Luc Richard, P.Geo.

CERTIFICATE OF QUALIFIED PERSON
Jean-François St-Laurent, ing., M.Sc.

I, Jean-François St-Laurent, ing., M.Sc., certify that:

1. I am employed as a Principal Consultant with SRK Consulting (Canada) Inc. (SRK), with an office address of 2600-320 Granville Street Vancouver, BC V6C 1S9 Canada.
2. This certificate applies to the technical report titled “NI 43-101 Technical Report and Preliminary Economic Analysis of the Opémiska Project, Québec, Canada” that has an effective date of October 17, 2025 and a report date of November 27, 2025 (the “Technical Report”).
3. I graduated from University Laval, Québec, with a Bachelors of Science in Geological Engineer in 2005 and a Master of Science in Civil Engineering in 2007.
4. I am a professional engineer registered with the Ordre des Ingénieurs du Québec (No. 140 657), and Professional Engineers Ontario (No. 100541518)
5. I have practiced my profession continuously for 18 years with experience in soils geotechnics, mine waste management and site reclamation. Currently Engineer of Record of two closed sites in Québec. My experience includes modelling embankment behaviour under various loading conditions, performing risk assessments, statutory inspections and safety reviews of tailings storage facilities. He’s been involved in the preparation of detailed engineering designs with drawings and technical specifications for numerous embankment and tailings storage facilities. Since March 2025, I have been an active member of an independent technical review board supporting a mining company operating a filtered tailings facility. In November 2025, I completed a Feasibility Study (FS) design for a co-disposal tailings facility.
6. 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.
7. I have not visited the project site.
8. I am responsible for Sections 1.16.2, 2.4.6, 18.4, 18.5, 21.2.5, 21.2.8, 25.10.2, 25.16.1.4.2, 25.16.2.2, and 26.5 of this Technical Report.
9. I am independent of XXIX Metal Corp as independence is defined in Section 1.5 of NI 43-101.
10. I have not been previously involved with the Opémiska Project.
11. 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: November 27, 2025

/Signed/

Jean-François St-Laurent, ing., P.Eng. (ON), M.Sc.

CERTIFICATE OF QUALIFIED PERSON
Charles Veilleux, ing.

I, Charles Veilleux, ing., P.Eng. (BC), certify that:

1. I am employed as a Senior Consultant with SRK Consulting (Canada) Inc., with an office address of 2600-320 Granville Street Vancouver, BC V6C 1S9 Canada.
2. This certificate applies to the technical report titled “NI 43-101 Technical Report and Preliminary Economic Analysis of the Opémiska Project, Québec, Canada” that has an effective date of October 17, 2025 and a report date of November 27, 2025 (the “Technical Report”).
3. I graduated from University Laval (Québec, QC, Canada) with a Bachelor of Science in water engineering in 2012.
4. I am a professional engineer registered with the Ordre des Ingénieurs du Québec (No. 5038360),
5. I am a member in good standing of the Ordre des Ingénieurs du Québec (#5038360), and Engineers and Geoscientists British Columbia (No. 63975)
6. I have practiced my profession for 13 years with experience in hydrology and hydrotechnical design. I have been directly involved in mine water management in projects in Nort America.
7. 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.
8. I have not visited the Opémiska project site.
9. I am responsible for Sections 1.16.1, 2.4.7, and 18.6 of this Technical Report.
10. I am independent of XXIX Metal Corp. as independence is defined in Section 1.5 of NI 43-101.
11. I have had no previous involvement with the Opémiska Project.
12. 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: November 27, 2025

/Signed/

Charles Veilleux, ing., P.Eng. (BC).

Important Notice

This report was prepared as National Instrument 43-101 Technical Report for XXIX Metal Corp. (XXIX) by Ausenco Engineering Canada ULC (Ausenco), Evomine Consulting Inc. (Evomine), , Geodoz Conseil Inc. (Geodoz), PLR Resources Inc. (PLR) and SRK Consulting (Canada) Inc. (SRK), 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 XXIX 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.

Table of Contents

1	Summary.....	1
1.1	Introduction	1
1.2	Terms of Reference.....	1
1.3	Property Description and Location	2
1.4	Royalties and Encumbrances	2
1.5	Accessibility, Climate, Local Resources, Infrastructure and Physiography	2
1.6	History	3
1.7	Geology and Mineralization.....	3
1.8	Deposit Types.....	4
1.9	Exploration.....	5
1.10	Drilling.....	5
1.11	Sampling Preparation and Security.....	5
1.12	Mineral Processing and Metallurgical Testwork.....	5
1.13	Mineral Resource Estimate	6
1.14	Mining Methods.....	8
1.15	Recovery Methods	8
1.16	Project Infrastructure.....	10
1.16.1	Water Management Plan	10
1.16.2	Tailings and Waste Rock Co-Disposal Storage Facility (CDSF)	11
1.17	Market Studies and Contracts.....	11
1.18	Environmental, Permitting and Social or Community Considerations	11
1.19	Capital and Operating Cost	12
1.19.1	Capital and Operating Cost Overview	12
1.19.2	Capital Cost Estimate.....	12
1.19.3	Operating Cost Estimate.....	13
1.20	Economic Analysis.....	13
1.20.1	Economic Summary.....	13
1.20.2	Sensitivity Analysis	15
1.21	Conclusions & Recommendations	18
2	Introduction.....	19
2.1	Introduction	19
2.2	Qualified Persons	20
2.3	Terms of Reference.....	21
2.4	Site Visits and Scope of Personal Inspection	21

2.4.1	Renée Barrette, Site Visit	21
2.4.2	Alexandre Burelle, Site Visit	21
2.4.3	Stephen Coates, Site Visit	22
2.4.4	Maude Lévesque Michaud, Site Visit	22
2.4.5	Pierre Luc Richard, Site Visit	22
2.4.6	Jean-Francois St-Laurent, Site Visit	22
2.4.7	Charles Veilleux, Site Visit	22
2.5	Effective Date	22
2.6	Information Sources and References	23
2.7	Previous Reports	23
2.8	Currency, Units, Abbreviations and Definitions	23
3	Reliance on Other Experts	27
4	Property Description and Location	28
4.1	Introduction	28
4.2	Mineral Tenure	29
4.3	Royalties and Encumbrances	38
4.4	Environmental Considerations	38
5	Accessibility, Climate, Local Resources, Infrastructure and Physiography	40
5.1	Accessibility	40
5.2	Climate	41
5.3	Local Resources and Infrastructure	41
5.3.1	Historical Mine Infrastructure	41
5.3.2	Local Workforce	41
5.3.3	Additional Services	41
5.4	Physiography	41
6	History	43
6.1	Summary of Property History	43
6.2	Discovery and Early Exploration (1929 to 1953)	43
6.3	Mine Production (1953-1991)	44
6.4	Property Exploration History (1993-2016)	45
6.4.1	Surface Work	46
6.4.2	Drilling	47
7	Geological Setting and Mineralization	48
7.1	Regional Geology	48
7.2	Project Geology	48
7.2.1	Structure	52
7.2.2	Mineralization	52

8	Deposit Types	56
9	Exploration	58
9.1	Re-Interpretation of Geological Model.....	58
9.2	Geophysics	59
9.3	Drill Hole Televiewer.....	60
9.4	Structural Geology Study	60
10	Drilling	61
10.1	Introduction	61
10.2	Drill Hole Programs	61
10.2.1	2019 Program	61
10.2.2	Winter/Spring 2021 Program	62
10.2.3	Late 2021, 2022, and 2023 Programs.....	62
10.2.4	2024-2025 Program.....	63
10.3	Drill Program Parameters	64
10.3.1	Hole Selection.....	64
10.3.2	Drill Hole Location and Set-up.....	64
10.3.3	Drill Hole Orientation at Start-up	64
10.3.4	Drill Hole Orientation During Operation	64
10.3.5	Drill Hole Coring.....	64
10.3.6	Core Handling at the Drill Rig	65
10.3.7	Receiving Core at the Core Logging Facility.....	65
10.3.8	Geological Logging Procedure	65
10.3.9	Assay Sample Selection	66
10.3.10	Core Sampling.....	66
10.3.11	Sample Shipment Preparation	66
10.3.12	Core Storage	67
11	Sample Preparation, Analyses, and Security	68
11.1	Core Handling, Sampling, and Security	68
11.2	Laboratories Accreditation and Certification.....	68
11.3	Laboratory Preparation and Assays	68
11.3.1	Sample Analysis Procedure (Laboratoire Expert)	68
11.3.2	Sample Analysis Procedure (ALS)	68
11.3.3	Sample Analysis Procedure (AGAT)	69
11.4	Quality Assurance and Quality Control.....	69
11.4.1	Standards.....	70
11.4.2	Duplicates	70
11.4.3	Blanks	70

11.5	Sample Preparation Conclusions	70
12	Data Verification	71
12.1	Drilling Database Summary	71
12.2	Historical Drill Hole Database	71
12.3	Recent Database	71
12.3.1	Site Visit	71
12.3.2	Drilling and Sampling Procedures	73
12.3.3	Assay Validation	74
12.3.4	QA/QC Validation	74
12.4	Metallurgical Testwork Data Validation	74
12.5	Conclusion	75
13	Mineral Processing and Metallurgical Testing	76
13.1	Introduction	76
13.2	Metallurgical Testwork	76
13.2.1	Legacy Testwork	76
13.2.2	XXIX Metal Corporation Testwork	77
13.3	Historical Operation Data	85
13.4	Metallurgical Variability	87
13.5	Deleterious Elements	88
13.6	Recovery Estimates	90
14	Mineral Resource Estimates	91
14.1	Introduction	91
14.2	Methodology	92
14.3	Resource Database	92
14.4	Geological Model	92
14.5	Voids Model	93
14.6	Historical Database Validation	93
14.7	Compositing	97
14.8	Capping	98
14.9	Density	102
14.10	Variogram Analysis and Search Ellipsoids	104
14.11	Block Model	106
14.12	Search Ellipsoid Strategy	107
14.13	Interpolation Method	108
14.14	Interpolation Parameters	108
14.15	Block Model Validation	109
14.15.1	Visual validation	109

14.16	Mineral Resource Classification	109
14.16.1	Mineral Resource Definition.....	110
14.16.2	Opémiska Mineral Resource Classification	110
14.17	Pit Optimization and DSO Parameters and Cut-off Grades	111
14.18	Opémiska Mineral Resource Estimate	112
15	Mineral Reserve Estimates	117
16	Mining Methods	118
16.1	Mining Overview	118
16.2	Geotechnical Considerations	118
16.3	Hydrogeological Considerations	118
16.4	Pit Optimization	119
16.4.1	Key Assumptions/Basis of Estimate	119
16.4.2	Dilution	120
16.4.3	Ultimate Pit Shell	120
16.4.4	Interim Pit Shells and Strategic Optimization.....	122
16.5	Pit Design	123
16.6	Production Schedule	127
16.7	Mining Operation	130
16.7.1	Drilling and Blasting.....	130
16.7.2	Loading	131
16.7.3	Hauling.....	131
16.7.4	Support Equipment	131
16.7.5	Dewatering	131
16.7.6	Fleet Requirements	132
16.7.7	Maintenance	133
16.7.8	Mine Management and Technical Services.....	134
16.8	Surface Mining Infrastructure.....	134
16.8.1	Haul Roads.....	134
16.8.2	Stockpiles.....	134
16.9	Mining Workforce	135
17	Recovery Methods.....	137
17.1	Process Design Criteria	137
17.2	Process Plant Description.....	137
17.2.1	Process Flowsheet	137
17.2.2	Crushing & Coarse Material Stockpiling	140
17.2.3	Grinding Circuit.....	140
17.2.4	Rougher Flotation and Regrind	141

17.2.5	Cleaner Flotation	141
17.2.6	Concentrate Dewatering & Handling	142
17.2.7	Tailings Dewatering	142
17.3	Reagent Handling and Storage.....	143
17.4	Energy, Water, and Process Materials Requirement.....	143
17.4.1	Process Materials	143
17.4.2	Water Requirements	144
17.4.3	Power Requirements	145
18	Project Infrastructure	146
18.1	Introduction	146
18.2	Site Access.....	148
18.3	Built Infrastructure.....	148
18.3.1	Plant Earthworks	148
18.3.2	Buildings	149
18.3.3	Accommodation	149
18.4	Stockpiles	149
18.5	Tailings and Waste Rock Co-Disposal Storage Facility (CDSF)	151
18.6	Surface Water Management.....	155
18.6.1	Design Criteria	155
18.6.2	Ditches and Runoff Diversion Berm	156
18.6.3	Culverts.....	158
18.6.4	Collection Ponds and Embankments.....	159
18.6.5	Pumping Stations and Pipelines	161
18.6.6	Water Treatment.....	161
18.6.7	Rainfall and Snowmelt.....	161
18.6.8	Groundwater	161
18.7	Power and Electrical.....	161
18.7.1	Facility Power Supply.....	161
18.7.2	Site Power Reticulation	162
18.8	Impact on town of Chapais	162
19	Market Studies and Contracts	163
19.1	Market Studies	163
19.2	Concentrate Marketing Assumptions	163
19.3	Commodity Price Projections.....	164
19.4	Contracts	166
20	Environmental Studies, Permitting, and Social or Community Impact	167
20.1	Environmental Studies	167

20.2	Waste Rock, Tailings, Overburden, Mineralized Material and Water Management.....	168
20.2.1	Geochemical Characterization of Waste Rock, Tailings and Mineralized Material.....	168
20.2.2	Waste Rock, Tailings and Overburden Management.....	169
20.2.3	Mineralized Material Management	170
20.2.4	Water Management	170
20.3	Permitting Requirements.....	170
20.3.1	Federal Regulations.....	170
20.3.2	Provincial Regulations	171
20.3.3	Municipal Regulations	172
20.4	Social or Community Requirements	173
20.5	Mine Closure and Reclamation.....	173
21	Capital and Operating Costs	175
21.1	Introduction	175
21.2	Capital Costs.....	175
21.2.1	Overview.....	175
21.2.2	Basis of Estimate.....	176
21.2.3	Mine Capital Costs.....	176
21.2.4	Process Capital Costs.....	177
21.2.5	Waste and Water Management Capital Costs	177
21.2.6	Infrastructure Capital Costs.....	178
21.2.7	Indirect Capital Costs.....	178
21.2.8	Sustaining Capital	179
21.3	Operating Costs.....	179
21.3.1	Overview.....	179
21.3.2	Basis of Estimate.....	179
21.3.3	Mine Operating Costs.....	180
21.3.4	Processing Operating Costs	182
21.3.5	Water Management Operating Costs	185
21.3.6	General and Administrative Operating Costs.....	185
22	Economic Analysis	187
22.1	Forward-Looking Information.....	187
22.2	Methodologies Used	188
22.3	Financial Model Parameters	188
22.3.1	Assumptions	188
22.3.2	Taxes.....	189
22.4	Economic Analysis.....	190
22.5	Sensitivity Analysis	193

23	Adjacent Properties	199
24	Other Relevant Data and Information	200
25	Interpretation and Conclusions	201
25.1	Introduction	201
25.2	Mineral Tenure, Surface Rights, Water Rights, Royalties and Agreements.....	201
25.3	Geology and Mineralization	201
25.4	Exploration and Drilling.....	201
25.5	Analytical Data Collection in Support of Mineral Resource Estimation.....	201
25.6	Metallurgical Testwork	201
25.7	Mineral Resource Estimate	202
25.8	Mining Methods.....	202
25.9	Recovery Methods	203
25.10	Infrastructure	203
25.10.1	Infrastructure	203
25.10.2	Co-Disposal Facility and Water Management Infrastructure.....	203
25.11	Markets and Contracts.....	203
25.12	Environmental, Permitting and Social Considerations	203
25.13	Capital Cost Estimate	204
25.14	Operating Cost Estimate	204
25.15	Economic Analysis.....	204
25.16	Risks and Opportunities	205
25.16.1	Risks.....	205
25.16.2	Opportunities	207
26	Recommendations.....	209
26.1	Introduction	209
26.2	Geology and Mineral Resources	209
26.3	Metallurgical Testwork	210
26.4	Mining Methods.....	210
26.5	Co-Disposal Facility and Water Management Infrastructure	210
26.6	Environmental, Permitting, Social or Community Impact	211
27	References	213

List of Tables

Table 1-1:	Opémiska Mineral Resource Estimate	7
Table 1-2:	Process Design Criteria	9
Table 1-3:	Capital Cost Summary.....	12
Table 1-4:	Operating Cost Summary	13
Table 1-5:	Economic Analysis Summary	14
Table 1-6:	Recommended Work Plan	18
Table 2-1:	Report Contributors.....	21
Table 2-2:	Abbreviations and Acronyms.....	23
Table 2-3:	Units of Measurement.....	25
Table 4-1:	Details of Mining Titles (as of July 13, 2025)	30
Table 6-1:	Historical Reports for the 1929-1953 Period.....	43
Table 6-2:	Historical Reports for the 1953-1991 period.....	45
Table 10-1:	Drill Hole Programs.....	61
Table 13-1:	Metallurgical Testwork Summary.....	76
Table 13-2:	2001 McGill University Program Flotation Testwork Summary	77
Table 13-3:	2001 McGill University Program Gravity Testwork Summary	77
Table 13-4:	Head Grade Characterization Summary	78
Table 13-5:	Modal Mineral Distribution	79
Table 13-6:	Composite 1 QEMSCAN Calculated and Direct Assay Reconciliation.....	80
Table 13-7:	Liberation and Association of Composite 1 Key Minerals	80
Table 13-8:	Gold Distribution in HLS and SP Products	81
Table 13-9:	Composite 1 Gold Minerals Liberation and Association	81
Table 13-10:	Composite 1 Comminution Test Results	82
Table 13-11:	Composite 1 Rougher Kinetics Tests Summary	82
Table 13-12:	Composite 1 Batch Cleaner Test Summary	84
Table 13-13:	Springer and Perry Mines Historical Production Data Summary	85
Table 13-14:	Composite 1 Domain, Hole IDs and Constituent Mass.....	88
Table 13-15:	Opémiska Domain Multi Element Chemical Analysis.....	89
Table 14-1:	Basic Statistics on Composites and High-Grade Capping Values for Copper	99
Table 14-2:	Basic Statistics on Composites and high-Grade Capping Values for Gold	100
Table 14-3:	Basic Statistics on Composites and High-Grade Capping Values for Silver	101
Table 14-4:	Density Basic Statistics	103
Table 14-5:	Variogram Model Parameters	104
Table 14-6:	Block Model Parameters	106
Table 14-7:	Block Model Coding.....	107
Table 14-8:	Search Ellipsoids Range and Orientation by Interpolation Passes	107
Table 14-9:	Interpolation Methods	108
Table 14-10:	Interpolation Parameters	109

Table 14-11: Optimization Parameters	111
Table 14-12: Revenue Calculation	112
Table 14-13: Opémiska Mineral Resource Estimate	113
Table 14-14: Pit-Constrained Indicated Resources at Various Cut-off Grades.....	114
Table 14-15: Pit-Constrained Inferred Resources at Various Cut-off Grades.....	114
Table 16-1: Open Pit Optimization Parameters.....	119
Table 16-2: Open Pit Optimization NSR Parameters	120
Table 16-3: Open Pit Optimization Results.....	122
Table 16-4: Grade Binning	123
Table 16-5: Pit Slope Configuration	125
Table 16-6: Pit Physical Quantities	127
Table 16-7: Annual Mining Schedule	129
Table 16-8: Drilling and Blasting Parameters	130
Table 16-9: Equipment Usage Assumptions.....	132
Table 16-10: Equipment Requirements.....	133
Table 16-11: Stockpile Slope Configuration	134
Table 16-12: Mining Workforce Requirements	136
Table 17-1: Process Design Criteria	138
Table 17-2: Reagents	143
Table 17-3: Annual Reagent and Consumables Requirements	143
Table 17-4: Power Requirements by Process Plant Area	145
Table 18-1: Buildings	149
Table 18-2: Collection Ditches/Channels.....	158
Table 18-3: Culvert Summary	158
Table 18-4: Embankment Summary	159
Table 18-5: Main Watershed Areas.....	159
Table 19-1: Copper Concentrate Assumptions.....	163
Table 19-2: Metal Price Assumptions	164
Table 21-1: Capital Cost Estimate.....	175
Table 21-2: Capital Cost Estimate Responsibilities.....	176
Table 21-3: Mining Initial Capital Costs	176
Table 21-4: Processing Initial Capital Costs	177
Table 21-5: Waste and Water Initial Capital Costs	177
Table 21-6: Infrastructure Initial Capital Costs.....	178
Table 21-7: Indirect Initial Capital Costs.....	178
Table 21-8: Sustaining Capital Costs.....	179
Table 21-9: Operating Cost Summary	179
Table 21-10: Mining Operating Costs Summary.....	180
Table 21-11: Annual Mining Operating Costs.....	181
Table 21-12: Processing Operating Costs	182

Table 21-13: Process Plant Staffing Plan and Labour Cost Summary	182
Table 21-14: Process Plant Power Cost Summary	183
Table 21-15: Reagent and Consumables Costs.....	184
Table 21-16: Process Plant Maintenance Cost Summary	185
Table 21-17: Water Management Operating Costs	185
Table 21-18: General and Administrative Operating Costs	186
Table 22-1: Economic Analysis Summary	191
Table 22-2: Physical Quantities Summary per Year	192
Table 22-3: Cashflow Summary per Year	192
Table 22-4: Exchange Rate Sensitivity Analysis	193
Table 22-5: Commodity Pricing Sensitivity Analysis	193
Table 22-6: Head Grade Sensitivity Analysis	194
Table 22-7: Metallurgical Recovery Sensitivity Analysis.....	194
Table 22-8: Operating Costs Sensitivity Analysis.....	194
Table 22-9: Capital Costs Sensitivity Analysis	195
Table 26-1: Recommended Work Program	209
Table 26-2: Metallurgical Testwork Program	210

List of Figures

Figure 1-1: Overall Process Flowsheet.....	10
Figure 1-2: Pre-Tax Net Present Value (8%) Sensitivity	15
Figure 1-3: Pre-Tax Internal Rate of Return Sensitivity	15
Figure 1-4: Pre-Tax Net Payback Period Sensitivity	16
Figure 1-5: Post-Tax Net Present Value (8%) Sensitivity	16
Figure 1-6: Post-Tax Internal Rate of Return Sensitivity.....	17
Figure 1-7: Post-Tax Payback Period Sensitivity	17
Figure 2-1: Location of Property	20
Figure 4-1: Project Location	28
Figure 4-2: Mining Titles	29
Figure 5-1: Access to the Opémiska Project	40
Figure 7-1: Regional Geology Plan	49
Figure 7-2: Project Geology Plan	50
Figure 7-3: Deposit-Scale Geology.....	51
Figure 7-4: Illustration of a Disseminated Halo (Stockwork) surrounding the Higher-Grade Zones.....	53
Figure 8-1: Opémiska Deposit Model Looking West	57
Figure 9-1: Geophysical Survey Location Plan	60
Figure 12-1: Photos taken by the QP during the Site Visit (Mineralization on Surface and Historical Tailings)	72

Figure 12-2: Core Review in the Core Logging Facility and Exterior Core Storage Facility	72
Figure 12-3: Drill Collar Validation during the Site Visit	73
Figure 12-4: Photographs of Core Logging and Storage Facilities	74
Figure 13-1: Composite 1 Rougher Kinetic Testing, Copper Recovery vs. Mass Pull & Flotation Time.....	83
Figure 13-2: Composite 1 Batch Cleaner Test Grade and Recovery Curves	84
Figure 13-3: Production Data Comparison to Composite 1 Testwork	86
Figure 13-4: Drill Hole Locations	87
Figure 14-1: Overall 3D View Looking Down showing the High-Grade Zones (Multiple Colours) and the Drill Holes (in Purple)	91
Figure 14-2: 3D Geological Model of the Opémiska Deposit (looking North)	93
Figure 14-3: 3D Grade-Distance Paired Plots for Gold in the High-Grade Zones	94
Figure 14-4: 3D Grade-Distance Paired Plots for Gold in the Stockwork Zones	95
Figure 14-5: 3D Grade-Distance Paired Plots for Silver in the High-Grade Zones (Left) and Stockwork Zones (Right)	96
Figure 14-6: 3D Grade-Distance Paired Plots for Copper in the High-Grade Zones (Left) and Stockwork Zones (Right)	97
Figure 14-7: Graphs Supporting Copper Capping on Composites in the High-Grade Zone Springer-V00.....	101
Figure 14-8: Graphs Supporting Gold Capping on Composites in the High-Grade Zone Springer-V00.....	101
Figure 14-9: Graphs Supporting Silver Capping on Composites in the Springer High-Grade Zones	102
Figure 14-10: Variography Study for Copper within the Perry-VB High-Grade Zone	105
Figure 14-11: Variography Study for Gold within the Perry-VB High-Grade Zone	105
Figure 14-12: Variography Study for Silver within the Perry-VB High-Grade Zone	106
Figure 14-13: 3D View of the Mineralized Zones and The Pit Shell (Down Plunge looking North)	115
Figure 14-14: Cross-Section View of the CuEq Grade within the Pit Shell	116
Figure 16-1: Open Pit Optimization Results.....	121
Figure 16-2: Double-Lane Ramp Configuration	124
Figure 16-3: Single-Lane Ramp Configuration	124
Figure 16-4: Slope Configuration	125
Figure 16-5: Ultimate Pit Design	126
Figure 16-6: Mining Schedule by Material Type	128
Figure 16-7: Mill Feed Schedule.....	128
Figure 16-8: Stockpile Design.....	135
Figure 17-1: Process Flowsheet	139
Figure 18-1: Infrastructure Layout Plan	147
Figure 18-2: Phase 1 of the CDSF and Temporary Overburden Stockpile	150
Figure 18-3: CDSF Final and Phases Configuration	153
Figure 18-4: CDSF Storage Capacity vs. Life of Mine – Tonnage to be Managed at the CDSF	154
Figure 18-5: CDSF Storage Capacity vs. Life of Mine – Tonnage to be Managed at the CDSF	154
Figure 18-6: Typical Cross-Section of the CDSF	155
Figure 18-7: Schematic Process Flow Diagram	156

Figure 18-8: Proposed Collection Ditches.....	157
Figure 18-9: Main Watersheds for the Project Site	160
Figure 19-1: Copper Historical Price (2020-2025).....	164
Figure 19-2: Gold Historical Price (2020-2025).....	165
Figure 19-3: Silver Historical Price (2020-2025)	165
Figure 20-1: Old Tailings Facilities Built with Waste Rock from the Former Opémiska Mine	169
Figure 22-1: Life-of-Mine Cashflow	190
Figure 22-2: Pre-Tax Net Present Value 8% Sensitivity	195
Figure 22-3: Pre-Tax Internal Rate of Return Sensitivity	196
Figure 22-4: Pre-Tax Net Payback Period Sensitivity	196
Figure 22-5: Post-Tax Net Present Value 8% Sensitivity	197
Figure 22-6: Post-Tax Internal Rate of Return Sensitivity.....	197
Figure 22-7: Post-Tax Payback Period Sensitivity	198

1 SUMMARY

1.1 Introduction

XXIX Metal Corp (XXIX) commissioned Ausenco Engineering Canada ULC (Ausenco), along with other consultants, to compile a preliminary economic assessment (PEA) for the Opémiska Project. The project is located next to the community of Chapais in the province of Quebec, approximately 480 km north of Montreal. 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/geologist consultants and firms who are providing qualified persons are as follows:

- Ausenco managed and coordinated the work related to the report. Ausenco developed the PEA-level design for the process plant, general site infrastructure and compiled the cost estimates for these.
- PLR Resources Inc. (PLR) completed the work related to property description, accessibility, local resources, geological setting, deposit type, exploration work, drilling, exploration works, sample preparation and analysis, data verification, and mineral resource estimate (MRE).
- Evomine Consulting Inc. (Evomine) designed the open pit, haul roads, mineralized material stockpiles, mine production schedules, the mine capital and operating costs, and compiled the overall project cost estimates and financial models.
- SRK Consulting (Canada) Inc. (SRK) designed the tailings and waste rock co-disposal facility and water management structures & facilities for the project.
- Geodoz Conseil Inc. (Geodoz) completed the work related to the environmental studies and permitting.

XXIX is a Canadian publicly traded company listed on the TSX Venture Exchange (TSXV) under the trading symbol “XXIX,” and has a head office in Toronto, Ontario.

1.2 Terms of Reference

This report supports the disclosure in XXIX’s news release titled “XXIX’s Opémiska PEA Confirms Positive Development Potential” dated October 21, 2025.

All measurement units used in this report are metric. All costs are in Canadian dollars unless otherwise stated.

As of the effective date of this report, the authors of this report are not aware of any known litigation potentially affecting the project. The qualified persons (QPs), as defined in NI 43-101, did not verify the legality or terms of any underlying agreement(s) that may exist concerning the project ownership, permits, off-take agreements, license agreements, royalties or other agreement(s) between XXIX and any third parties.

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

1.3 Property Description and Location

The project is located within the northeast portion of the Abitibi Metavolcanic Belt, 480 km north of Montreal. It lies adjacent to and partly within the community of Chapais.

XXIX owns a group of 241 mining titles covering a total of 12,431 ha. This information is current as of July 13, 2025. The mining titles are recorded under XXIX and are in good standing as of the effective date of this report. The MRE presented in Section 14 is found on mining titles P013681 and P014151 (registered under XXIX).

XXIX is in the process of acquiring 175 additional mining titles that cover 9,068 ha from 2736-1170 Quebec Inc. (85%), Ovalbay Geological Services Inc. (10%), and Melissa Darveau (5%), together referred to as the “Cooke/Robitaille Option” agreement. The company is still in the earn-in process with approximately \$900,000 of work obligations to complete by July 13, 2026.

1.4 Royalties and Encumbrances

XXIX fulfilled all its obligations under the terms of the option agreement with Ex-In on June 16, 2023 and executed the purchase agreement of 11 mining titles. As a result, these claims have now been transferred to and are 100% owned by XXIX, subject to a 2% NSR royalty, 50% of which can be purchased by XXIX for \$4.5 million.

XXIX is in an earn-in process to acquire 175 additional mining titles subject to a 2% NSR royalty, 50% of which can be purchased by XXIX before the commencement of commercial production for \$1.5 million.

1.5 Accessibility, Climate, Local Resources, Infrastructure and Physiography

The project is adjacent to the town of Chapais, Quebec, and about 40 km west of Chibougamau. There is year-round overland access via Highway 113, forestry roads, and former mine roads. The region is also serviced by the Chibougamau–Chapais Airport, with regular flights to Montréal.

The area has a humid sub-arctic continental climate, with cold winters (average lows of -26°C) and mild summers (average highs of 22°C). The physiography of the area consists of rolling hills, lakes, rivers, and variable forest cover. Overburden thickness ranges from 1 m to over 80 m, with limited bedrock exposure. Elevation is generally around 400 meters above sea level (masl), with drainage flowing westward toward James Bay.

Local infrastructure includes existing historical mines, transmission lines, and roads. The Chapais–Chibougamau region has a population of approximately 10,000 residents and has a long mining history, providing access to a skilled workforce and necessary support services.

1.6 History

The Opémiska property has undergone three major phases of historical exploration and mining activity:

- Early Exploration Phase (from 1929 to 1953) – Began with the discovery of a significant chalcopyrite mineralization by Leo Springer, followed by early trenching, drilling, and underground development that established the foundation for future mining.
- Mine Production Phase (from 1953 to 1991) – The Springer and Perry mines operated continuously, producing 22 Mt of mineralized material grading 2.40% Cu, 0.29 g/t Au, and 0.21 g/t Ag from seven easterly-trending mineralized zones, supported by extensive underground and surface drilling.
- Recent Exploration Phase (from 1993 to 2016) – Ex-In acquired the Springer and Perry mines in 1995 and launched a renewed exploration program including drilling, trenching, geophysical surveys, sampling programs, and metallurgical testing aimed at assessing remaining near-surface mineralization and re-evaluating the potential of previously mined areas.

Across all phases, the property has consistently demonstrated strong copper-gold prospectivity and a long history of systematic technical work.

1.7 Geology and Mineralization

The project area is located within the Superior Structural Province of the Canadian Shield, which is present in eastern Canada and the northeastern USA. The Precambrian rock units are generally covered by glacial overburden.

The Chapais-Chibougamau mining district is located in the northeast part of the Abitibi Subprovince. The Abitibi Subprovince is one of the world's largest contiguous areas of Archean metavolcanic and metasedimentary rocks and hosts many significant mineral deposits (Leclerc et al., 2010, 2012). The general lithological distribution is characterized by oval-shaped granitoid batholiths surrounded by east-to-west trending "greenstone belts" that appear to wrap around and enclose the batholiths. Regional and local folding is common, and the dips of the rock units are generally sub-vertical. The region under study is located within the Northern Volcanic Zone of the Abitibi Subprovince (Guha et al., 1988; Dube and Guha, 1992).

The metavolcanic stratigraphy in the Chapais-Chibougamau area is representative of deep-water deposition to submarine environments. The metavolcanic-sedimentary package is cut by mafic to ultramafic intrusions (Lac Dore Complex being the best-known example), mafic sills (Cummings Sills and gabbro), and younger plutonic intrusions that range from tonalite to carbonatite in composition.

This section is largely inspired by previous reports from XXIX, mainly Yassa et Puritch (2024) that summarizes internal XXIX reports.

Recent work by Leclerc et al. (2010, 2012) has refined the understanding of the complex geology and stratigraphy of the project area. The earlier stratigraphic interpretation has been modified, in order to take into consideration recent field observations.

The geology of the Opémiska property is characterized by a fold affecting the Cummings Complex introduced at the lower contact of felsic volcanics of the Blondeau Formation. The Cummings Complex are comprised of three separate differentiated sills: the Roberge Sill at the base; the Ventures Sill; and the Bourbeau Sill higher-up in the Blondeau stratigraphy.

The mineralization at Opémiska consists largely of chalcopyrite-bearing quartz veins that occupy fracture systems in the folded and faulted gabbroic portions of two conformable, regionally extensive, layered Archean ultramafic-mafic sills. The veins are generally restricted to the fracture system and in lower grade halos around the main fractures/veins. The width and frequency of the veins tend to increase toward the dilated nose of the main structure at the Springer mine (Watkins and Riverin, 1982).

The mineralization at the Springer mine is associated to a series of east-trending (090°), steeply (65°) north-dipping, sets of axial plane faults and fractures with right-handed (dextral) displacement that developed in areas of maximum inflexion of folds (Watkins and Riverin, 1982). Plan and cross-section views of Springer show at least three different orientations for the mineralized veins which could indicate a conjugate fault system or separate fracture systems. A disseminated halo (stockwork) surrounds most of the higher-grade zones.

Generally, mineralization of economic interest appears within more fractured/sheared sections of the host gabbro. These sections are generally strongly chloritized and variably silicified.

Although most of the mineralization historically mined at Springer and Perry was hosted in the upper part of the Ventures Sill, the regional and local structures are also important controls on mineralization at Opémiska. At Springer, the fold nose corresponding to the overturned anticline in the mafic-ultramafic sills controls significant amount of mineralization. A 6.0 m wide zone containing disseminated pyrrhotite and chalcopyrite occurs locally at the top of the Ventures Sill, where it is dilated at the nose of the fold (Watkins and Riverin, 1982).

1.8 Deposit Types

Mineralization at the project occurs in structurally controlled copper-gold veins hosted within folded mafic-ultramafic sills (Ventures and Bourbeau). These veins form along east-west axial planar faults and northwest-trending radial structures created during drag folding associated with sinistral movement (D2b) along the Gwillim Fault. Later shifts in regional compression (from north-south to northwest-southeast) reopened these fractures, enabling hydrothermal fluid flow and mineralization.

In the Chibougamau Mining Camp, similar structurally controlled copper-gold mineralization occurs in west-northwest-trending dextral shear zones (formed during second deformation event) and in later northeast-trending dextral shears. The Opémiska vein systems show parallels to Chibougamau, including higher gold grades in structures proximal to major faults (e.g., Gwillim Fault at Opémiska).

1.9 Exploration

Exploration work by XXIX (previously QC Copper and Gold) has been focused on diamond drilling and geological compilation and re-interpretation work. In addition, a geophysical 3D IP survey, a drill hole televiewer survey and a structural geology study were completed.

1.10 Drilling

In total, 21,918 surface and underground drill holes for 1,525,073 m are recorded as having been completed at the Opémiska property.

XXIX completed drill hole programs in 2019 (as predecessor company, PowerOre), winter-spring 2021 (as predecessor company QC Copper and Gold), autumn-winter 2021-2022 (as predecessor company QC Copper and Gold), summer 2022 (as predecessor company QC Copper and Gold), winter 2023 (as predecessor company QC Copper and Gold), and more recently, in 2024-2025 (as XXIX).

1.11 Sampling Preparation and Security

Core has been sampled to create a representative and homogenous database. Sampling honours lithological contacts, alteration boundaries and mineralization boundaries.

The sample length for the intervals collected varies from 0.50 m to 1.5 m. The core was sawn in half with a diamond saw along its length. One half was put into a plastic sample bag, and the other half was retained and kept in the core box for later reference. A sample assay tag was placed in the plastic sample bag, and the bag was tied off.

The quality assurance/quality control (QA/QC) data indicates that the overall assay results of the XXIX's drill program are valid and can be relied upon for the purpose of this report.

It is the QP's opinion that the sample preparation, security and analytical procedures are adequate and follow best practices.

The sample preparation, analytical procedures, and security of the samples during these procedures followed industry best practices but could be improved, mainly by inserting more blanks, more CRMs, and adding a field duplicate program. Sufficient efforts were made to identify items that were out of specification.

1.12 Mineral Processing and Metallurgical Testwork

Metallurgical testing was completed by QC Copper and Gold (now XXIX Metal Corporation) at SGS (Quebec City) in 2023. The objective of the testwork program was to provide sufficient metallurgical data to support the design of a flotation process to recover a copper concentrate with by-products of gold and silver, amenable to smelting by others. The testwork was conducted on a single composite sample, referred to as composite 1, made up of ¼ core selected from intervals weighted proportionally to the Opémiska deposit mineralized domains, and intersecting all lithologies. The composite had an assayed copper grade of 0.81%, consistent with the expected grades in the PEA mine plan payback period. The scope of work included head grade characterization, mineralogical analysis, gold deportment,

comminution (SMC), flotation and environmental testing. The results of this testwork program were used to inform the PEA design.

The mineralogical analysis showed the minerals of economic interest to be sulphides, in order of decreasing abundance, chalcopyrite, pyrite and arsenopyrite, all of which may host gold. Pyrrhotite was not observed, but some minor content has been reported from drilling programs. The major gangue minerals are feldspars, pyroxenes, iron oxides, quartz, calcite and chlorite. No multi-element assays, beyond quantifying valuable metal content, were conducted on concentrates. However, the concentrate is not expected to contain significant quantities of deleterious elements based on typical concentration ratios.

The Opémiska deposit encompasses the historical Springer and Perry mines, operated by various entities from 1954 to 1991. The historical operations demonstrated amenability to recovery via conventional flotation processes, with reported copper recoveries consistently above 95%. The testwork results aligned to available historical operation data, suggesting that the production data provides a reasonable basis for the interpretation of the expected metallurgical performance.

Recovery equations were developed based on the testwork and historical operation data. The expected life-of-mine recoveries based on these equations applied to PEA mine plan are 92.0%, 79.9%, and 80.3% for copper, gold, and silver respectively.

1.13 Mineral Resource Estimate

The MRE presented in this technical report covers the Opémiska deposit only. Other occurrences on the project were considered exploration targets at the time this report was being prepared. Additional exploration work is needed before they can reach the stage of mineral reserve. Mineral resources that are not mineral reserves do not have demonstrated economic viability.

Leapfrog Geo™ and Edge™ v.2024.1.3 (Leapfrog) was used to update the geological and mineralized zones and to generate the drill hole intercepts for each solid. Leapfrog was used for compositing, 3D block modelling, and interpolation. Statistical studies were conducted using Excel and Snowden Supervisor.

The methodology for the mineral resource estimation involved the following steps:

- database verification
- 3D modelling of the geological zones
- 3D modelling update of the mineralized zones
- 3D modelling of a stockwork zones
- drill hole intercept and composite generation
- basic statistics
- capping
- geostatistical analysis including variography

- block modelling and grade interpolation
- block model validation
- mineral resource classification
- cut-off grade calculation
- pit shell optimization
- DSO optimization
- preparation of the mineral resource statement.

The 2025 Opémiska MRE is constrained within a pit shell developed from pit optimization and DSO shapes using appropriate cut-off grades. Table 1-1 presents the results of the MRE.

Table 1-1: Opémiska Mineral Resource Estimate

Pit Constrained	Tonnes (k)	CuEq (%)	Cu (%)	Ag (g/t)	Au (g/t)	Cu (M lbs)	Ag (koz)	Au (koz)
0.15% CuEq Cut-Off								
Indicated	62,706	1.04	0.76	1.71	0.31	1,047	3,450	634
Inferred	78,485	0.41	0.26	0.61	0.17	457	1,530	419
Out of Pit	Tonnes (k)	CuEq (%)	Cu (%)	Ag (g/t)	Au (g/t)	Cu (M lbs)	Ag (koz)	Au (koz)
1.00% CuEq Cut-Off								
Indicated	6,947	1.85	1.59	2.76	0.28	243	617	64
Inferred	2,130	0.88	0.69	1.20	0.21	33	82	15
Total	Tonnes (k)	CuEq (%)	Cu (%)	Ag (g/t)	Au (g/t)	Cu (M lbs)	Ag (koz)	Au (koz)
0.15% & 1.00% CuEq Cut-Off								
Indicated	69,653	1.12	0.84	1.82	0.31	1,290	4,067	697
Inferred	80,615	0.42	0.28	0.62	0.17	490	1,613	433

Notes: **1.** The independent qualified persons for the MRE, as defined by National Instrument (NI) 43-101 guidelines, is Pierre Luc Richard, P.Geol., of PLR Resources Inc. with contributions from Stephen Coates, P.Eng., of Evomine for value cut-off, open pit and optimization solids, and Christian Laroche, P.Eng., from Synectiq, for metallurgical parameters. The effective date of the MRE is May 30, 2025. **2.** These mineral resources are not mineral reserves as they do not have demonstrated economic viability. The quantity and grade of reported inferred resources in this MRE are uncertain in nature, and there has been insufficient exploration to define these inferred resources as indicated or measured. However, it is reasonably expected that the majority of inferred mineral resources could be upgraded to the indicated category with continued exploration. **3.** The MRE wireframe was prepared using Leapfrog Edge v.2024.1.3 and is based on 21,918 drill holes for 1,525,073 meters and 479,242 samples. The drill hole database includes recent drilling (2002 to 2025) of 73,227 meters in 382 drill holes (Ex-In, PowerOre, QC Copper and Gold, XXIX) and also incorporates historical drill holes (1930 to 1990) for 1,451,846 meters in 21,536 drill holes (Opémiska Copper Mines, Falconbridge, Minnova). The cut-off date for the drill hole database was May 16, 2025. **4.** Resources are presented as undiluted and in situ for the open-pit scenario and include internal dilution for the underground scenario and are considered to have reasonable prospects for economic extraction. The constraining pit shell was developed using overall pit slopes of 55 degrees in bedrock and 30 degrees in overburden. The pit optimization to develop the resource-constraining pit shells was done using Deswik Pseudoflow 2024.2. **5.** Composites of 1.5 meters were created inside the high-grade zones and 3.0 meters inside the stockwork zones. High-grade capping was done on the composited assay data; composites were capped at variable grades ranging from 1.00 to 25.00% for Cu, 0.50 to 35.00 g/t for Au, and 10.00 to 120.00 g/t for Ag. **6.** Mineral resources are reported at a cut-off grade of 0.15% CuEq for open-pit resources and 1.00% CuEq for underground resources. All material within the underground stopes is being reported, including internal dilution. The cut-off grades will be re-evaluated in light of future prevailing market conditions and costs. **7.** Specific gravity values were estimated using data available in the drill hole database. Values assigned per zone and per host rock. Surrounding barren lithologies were assigned the average specific gravity value from all measured samples available. **8.** Grade model resource estimation was calculated from drill hole data using an ordinary kriging (OK) interpolation method in a sub-blocked model using blocks measuring

5 x 5 x 5 m in size and sub-blocks down to 0.625 x 0.625 x 0.625 m. Both OK and inverse square distance (ID2) interpolation methods were tested, resulting in no material difference in the mineral resource estimates. **9.** The indicated and inferred mineral resource categories are constrained to areas where drill spacing is less than 50 m and 120 m, respectively, and show reasonable geological and grade continuity. **10.** Calculations used metric units (meters, tonnes). Metal contents are presented in percent or pounds. Metric tonnages were rounded, and any discrepancies in total amounts are due to rounding errors. **11.** Canadian Institute of Mining, Metallurgy and Petroleum (CIM) definitions and guidelines for mineral resource estimates have been followed. **12.** The QPs are not aware of any known environmental, permitting, legal, title-related, taxation, socio-political or marketing issues or any other relevant issues that could materially affect this MRE.

1.14 Mining Methods

The project is planned as a conventional truck-and-shovel open pit mining operation. The nominal processing rate is set at 12,500 t/d over a 17-year mine life, with an average strip ratio of 3.7 to 1. Two ultimate pits—the Springer pit and Perry pit—will be mined over the life of mine, with interim pits designed within these ultimate pits to optimize the mineralized material grade and strip ratio extraction profile. Mined physical quantities represent 77.2 Mt of mineralized material, 270.7 Mt of waste and 15.0 Mt of overburden segregated by a block model reblocked to 5 x 5 x 5 m dimensions to adequately consider selectivity and associated mining dilution for the envisioned mining equipment.

Four mining phases are planned and detailed as follows: starter pits in both Springer and Perry (Phase 1), an intermediate pushback in Springer (Phase 2), the depletion of Perry (Phase 3), and the depletion of Springer (Phase 4).

The 17-year life of mine incorporates 13 years of direct mill feed from open pit operations and four years of stockpile rehandling. The mining rate is expected to peak at 116,000 t/d and average 76,000 t/d over the 13 years of mining. The open pit operation has also been optimized to push any impact to the neighbouring town of Chapais to the end of Phase 3 and beginning of Phase 4. Also considered are areas dedicated to overburden, waste rock, and mineralized material stockpiling.

1.15 Recovery Methods

The project flowsheet was selected based on preliminary testwork, historical operating data and subsequent economic modelling. The proposed unit operations are standard technologies in copper concentrator plants, including the following:

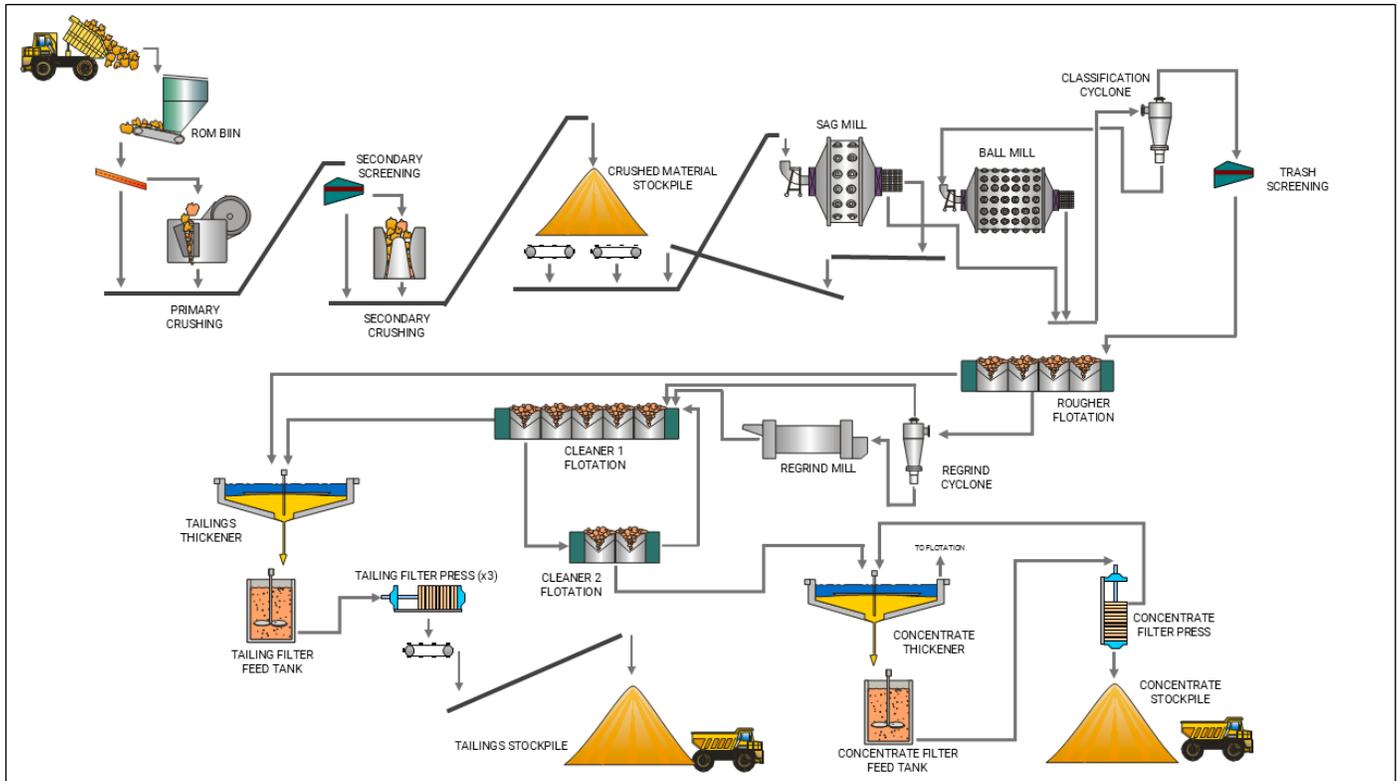
- comminution circuit consisting of two-stage crushing, a SAG mill, and a ball mill with cyclone classification in a closed-circuit configuration
- rougher flotation with regrind
- two stages of cleaner flotation
- concentrate handling
- tailings filtration and stockpiling for disposal

The key process design criteria for the plant are listed in Table 1-2, and the process flowsheet is shown in Figure 1-1.

Table 1-2: Process Design Criteria

Parameter	Units	Value
Annual Process Plant Throughput	Mt/a	4.6
Daily Process Plant Throughput	t/d	12,500
Copper Head Grade, Design	%	0.92
Life of Mine	y	18
Operating Availability, Crushing	%	65
Operating Availability, Grinding	%	92
Operating Availability, Filtration	%	84
Copper Recovery, Design	%	92
JK SMC Axb, Design	-	26.8
Bond Crushing Work Index (CWi), Design	kWh/t	26.1
Bond Rod Mill Work Index (RWi), Design	kWh/t	23.0
Bond Ball Mill Work Index (BWi), Design	kWh/t	20.0
Bond Abrasion Index, Design	g	0.403
Specific Gravity	-	2.92
Crushing Feed Size, F ₈₀	mm	423
Crushing Product Size, P ₈₀	mm	36
SAG Mill Pebble Recycle Rate, Design	% fresh feed	14
Ball Mill Circulating Load, Design	%	350
Grinding Product Size, P ₈₀	µm	105
Primary Cyclone Overflow Density	% w/w solids	35
Rougher Flotation Stage Recovery, Design	%	11
Regrind Product Size, P ₈₀	µm	30
Regrind Specific Energy	kWh/t	13
Cleaner 1 Flotation Stage Recovery, Design	%	48
Cleaner 2 Flotation Stage Recovery, Design	%	70
Concentrate Thickening Rate, Design	t/m ² /h	0.25
Concentrate Specific Filtration Rate	kg/m ² /h	500
Concentrate Filter Cake Moisture, Target	% w/w	9.5
Tailings Thickening Rate, Design	t/m ² /h	0.75
Tailings Specific Filtration Rate	kg/m ² /h	149
Tailings Filter Cake Moisture, Target	% w/w	15

Figure 1-1: Overall Process Flowsheet



Source: Ausenco, 2025.

1.16 Project Infrastructure

1.16.1 Water Management Plan

The mine water management plan addresses the surface runoff to be collected from the industrial areas, including the open pit, the rockfill/overburden/topsoil stockpiles and co-disposal storage facility (CDSF), and along the mining/access roads of the Opémiska mine site. The surface water management infrastructure (i.e., channels, ditches, ponds and pumping requirements) are sized based on the required volume of surface runoff to manage, which varies based on the catchment area of the various infrastructure of the mine site. Hence, the water management plan will extend as the drainage area increases with the mine development.

Water to be used in the mineral processing will be taken directly from the surrounding ponds as much as possible to facilitate recirculation. The remaining water will be kept in the CDSF pond before being released to the environment following monitoring of flow and water quality, in full compliance with applicable laws, regulations, and standards. At this stage of design, there is not enough information to conclude whether a water treatment plant will be required.

1.16.2 Tailings and Waste Rock Co-Disposal Storage Facility (CDSF)

Filtered tailings will be dewatered to a targeted moisture content of 15% to 18%, enabling efficient transport and compaction alongside waste rock. Both filtered tailings and waste rock will be placed and compacted together within a geomembrane-lined facility, engineered to promote basal drainage and prevent slip surfaces along the lining. The facility is designed with 10 m benches, each with minimum 10 m wide setbacks and outer slopes of 3H:1V, resulting in an overall average slope of 4.3H:1V. The CDSF will ultimately reach a maximum elevation of 525 m and accommodate approximately 162 Mm³ (326 Mt) of material.

Co-disposal is intended to minimize the overall footprint, enhance physical stability, and allow for safe, flexible operations. Waste rock inclusions are used strategically to reinforce structural zones. The base of the CDSF includes a blanket drain to help maintain a low phreatic surface and facilitate water evacuation. The facility will be constructed in six phases over 13 years, in alignment with mine planning, with ongoing surface water management developing in parallel.

Facility design and management comply with applicable provincial and industry standards, including Directive 019, MERN guidance, and the Mining Association of Canada's tailings management protocols. While detailed stability analyses are outstanding for this project phase, the design adheres to current best practices and regulatory requirements, with ongoing development of the comprehensive deposition and compaction plans.

The overburden and organic material underlying the CDSF area exhibits variable thickness, with an average of 30 cm adopted for design. Organic and overburden materials excavated from beneath the CDSF footprint will be stored within a temporary overburden stockpile south of the Phase 1 CDSF area. This stockpile will be constructed according to regulatory requirements, reaching a maximum elevation of approximately 395 m with slopes at 3H:1V, and an estimated total capacity of 190,800 m³ (equivalent to roughly 300,000 t at a dry density of 1.6 t/m³). Materials from this stockpile will be reused for reclamation of the CDSF slopes as mine development progresses.

1.17 Market Studies and Contracts

The project is expected to produce a copper concentrate. Neither XXIX nor its consultants have undertaken a formal market study regarding the sale of this concentrate. Accordingly, the marketing assumptions in this study are based on prevailing market conditions, discussions with XXIX, and terms reported in comparable recent studies and projects.

1.18 Environmental, Permitting and Social or Community Considerations

At the federal level, the *Impact Assessment Act* (IAA) indicates that projects designated by the physical activities regulations are subjected to the environmental assessment procedure, which is the case for the Opémiska Project. This procedure could take several years, so it needs to start early in the development of the project.

At the provincial level, provisions under Title II of the EQA are applicable to the James Bay and Northern Québec region. All work required for the operation of a new mine are subject to the impact assessment procedure. The Opémiska Project will be subject to the procedure for social and environmental impacts assessment, with an evaluation by the COMEV and a review by the COMEX.

At both levels, the environmental permitting process requires an understanding of the physical, biological and social environments. It includes an evaluation of the potential impacts of the mining project and proposes mitigation measures. Environmental baseline studies will start in the next phase of the project.

The Opémiska Project is located in the municipality of Chapais, part of the administrative territory of Eeyou Istchee James Bay. Information and consultation meetings have been initiated by XXIX with the town of Chapais, and will be undertaken with the First Nation authorities, other stakeholders and land users.

1.19 Capital and Operating Cost

1.19.1 Capital and Operating Cost Overview

Capital and operating cost estimates were prepared to support project development both on and off site. The capital and operating cost estimates are based on an open pit mining operation, processing of mineralized material on site at a rate of 4.6 Mt/a, and shipment of copper concentrate to customers off site. All cost figures are reported in Canadian dollars (CAD, C\$) currency, unless specified otherwise.

1.19.2 Capital Cost Estimate

The capital cost estimate developed in this PEA was prepared to a Class 5 estimate with an accuracy of $\pm 50\%$ as defined by the Association for the Advancement of Cost Engineering International (AACE International). Generally, engineering performed to date is between 1% to 5% of full project definition.

The total capital cost estimate of the project includes initial capital costs and sustaining capital costs and is estimated at \$1,048 million. Table 1-3 summarizes the total capital costs for the project.

Table 1-3: Capital Cost Summary

Cost Area	Initial Capital Cost (\$M)	Sustaining Cost (\$M)	Total Cost (\$M)
Mining	45.6	230.3	276.0
Processing	271.0	-	271.0
Waste and Water Management	21.4	60.6	82.0
On-Site Infrastructure	16.2	-	16.2
Off-Site Infrastructure	27.0	-	27.0
Town of Chapais Infrastructure Costs	-	100.0	100.0
Indirect Costs	114.6	-	114.6
Contingency	121.4	-	121.4
Closure	-	40.0	40.0
Total	617.3	430.9	1,048.2

1.19.3 Operating Cost Estimate

Operating costs are summarized in Table 1-4. These include mining, processing, waste and water management, and general and administration (G&A) costs. Operating costs were estimated at \$2,665 million over the life of mine, which represents \$34.52 per tonne processed.

Table 1-4: Operating Cost Summary

Cost Area	Total (\$M)	\$/t Processed	% of Total
Mining	1,594.7	20.66	59.8
Process	819.9	10.62	30.8
Water Management	6.5	0.08	0.2
G&A	244.0	3.16	9.2
Total	2,665.0	34.52	100

1.20 Economic Analysis

1.20.1 Economic Summary

The PEA is preliminary in nature. 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 PEA would be realized.

The economic and financial evaluation presented in this technical report utilizes a discounted cashflow method, both on a pre-tax and after-tax basis. The metal pricing used in the evaluation was determined in Section 19. The financial model provides results in terms of net present value (NPV), payback period, and IRR for the project. The economic analysis is conducted in real terms, without considering inflation factors, using Q4 2025 Canadian dollars. The analysis does not take into account project financing. The analysis considers a capital lease on mobile equipment, but does not take into account project financing.

The economic analysis was performed assuming an 8% discount rate. The pre-tax NPV (8%) is \$793.0 million; the internal rate of return IRR is 32.1%, and payback period is 2.3 years. On a post-tax basis, the NPV (8%) is \$505.2 million; the IRR is 27.2%, and the payback period is 2.3 years. A summary of project economics is shown in Table 1-5.

Table 1-5: Economic Analysis Summary

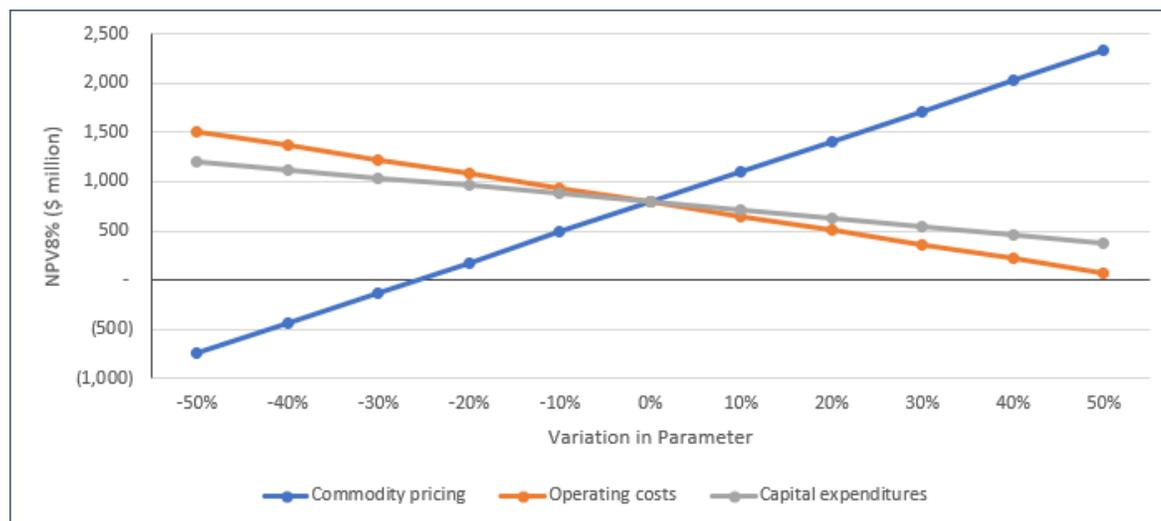
Description	Units	Value
General		
Copper Price	US\$/lb	4.35
Gold Price	US\$/oz	3,000
Silver Price	US\$/oz	30.00
Exchange Rate	USD/CAD	1.35
Mine Life	years	17.1
Production		
Mill Feed Tonnage	kt	77,201
Mill Feed Average Grade – Cu	%	0.481
Mill Feed Average Grade – Au	g/t	0.234
Mill Feed Average Grade – Ag	g/t	1.119
Average Metallurgical Recovery – Cu	%	92.0
Average Metallurgical Recovery – Au	%	79.9
Average Metallurgical Recovery – Ag	%	80.3
Total Metal Recovered – Cu	Mlbs	752.7
Total Metal Recovered – Au	koz	464
Total Metal Recovered – Ag	koz	2,231
Average Annual Production – Cu	Mlbs	44.1
Average Annual Production – Au	koz	27
Average Annual Production – Ag	koz	131
Gold Payable	koz	409
Silver Payable	koz	2,008
Capital Costs		
Initial Capital Costs	\$ million	617.3
Sustaining Capital Costs	\$ million	390.9
Closure Capital Costs	\$ million	40.0
Operating Costs		
Mining Cost	\$/t mined	4.39
Mining Cost	\$/t milled	20.66
Processing Cost	\$/t milled	10.62
Waste and Water Management Cost	\$/t milled	0.08
General and Administrative Cost	\$/t milled	3.16
Total Operating Costs	\$/t milled	34.52
C1 Cash Cost (Net of By-products)	US\$/lb Cu	1.40
C3 Cash Cost (Net of By-products)	US\$/lb Cu	2.50
Pre-Tax Valuation Indicators		
Undiscounted Cashflow	\$ million	1,747.8
NPV (8%)	\$ million	793.0
Payback Period (from Start of Operations)	years	2.3
IRR	%	32.1
After-Tax Valuation Indicators		
Undiscounted Cashflow	\$ million	1,156.8
NPV (8%)	\$ million	505.2
Payback Period (from Start of Operations)	years	2.3
IRR	%	27.2

1.20.2 Sensitivity Analysis

The project financial performance is most sensitive to commodity prices, exchange rate, head grade, and metallurgical recovery. It is significantly less sensitive to capital and operating costs.

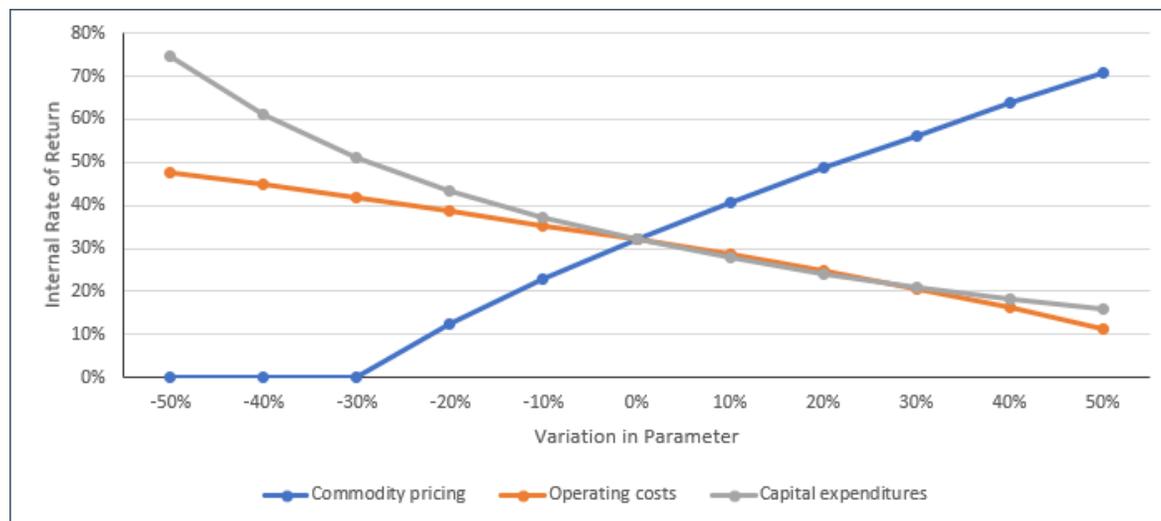
Figures 1-2 to 1-4 illustrate the sensitivities on the pre-tax NPV (8%), IRR, and payback period, respectively. Figures 1-5, 1-6, and 1-7 illustrate the sensitivities on the post-tax NPV (8%), IRR, and payback period, respectively.

Figure 1-2: Pre-Tax Net Present Value (8%) Sensitivity



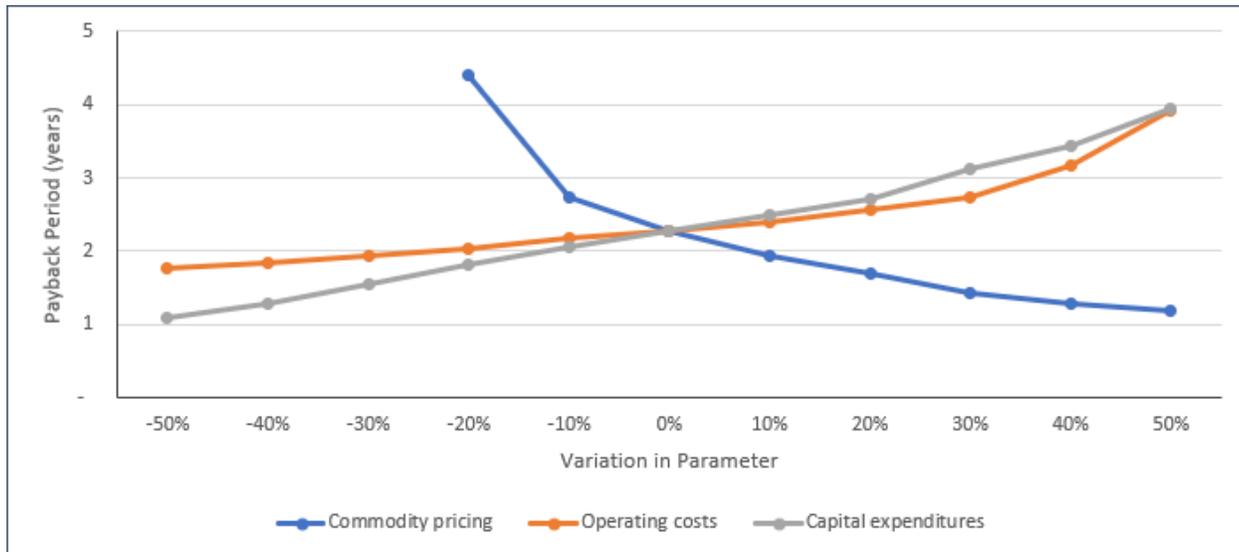
Source: Evomine, 2025.

Figure 1-3: Pre-Tax Internal Rate of Return Sensitivity



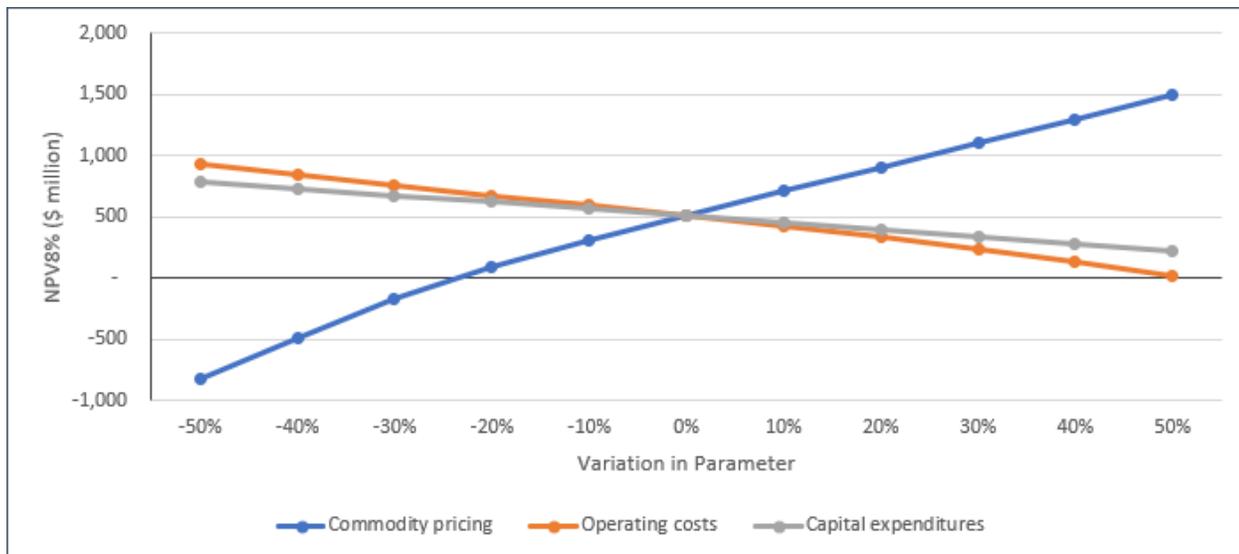
Source: Evomine, 2025.

Figure 1-4: Pre-Tax Net Payback Period Sensitivity



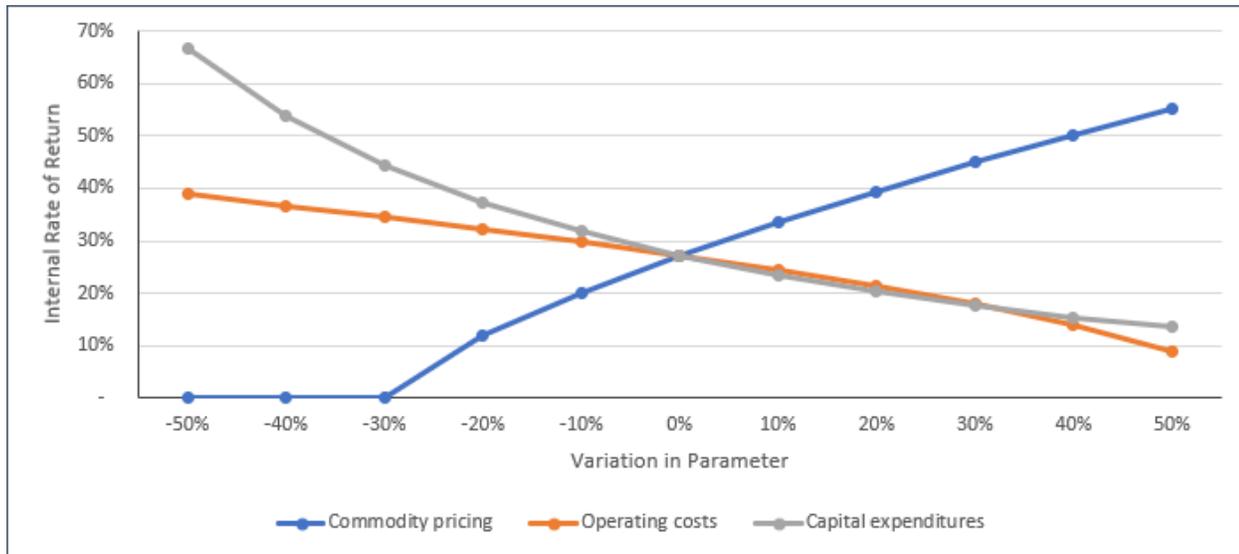
Source: Evomine, 2025.

Figure 1-5: Post-Tax Net Present Value (8%) Sensitivity



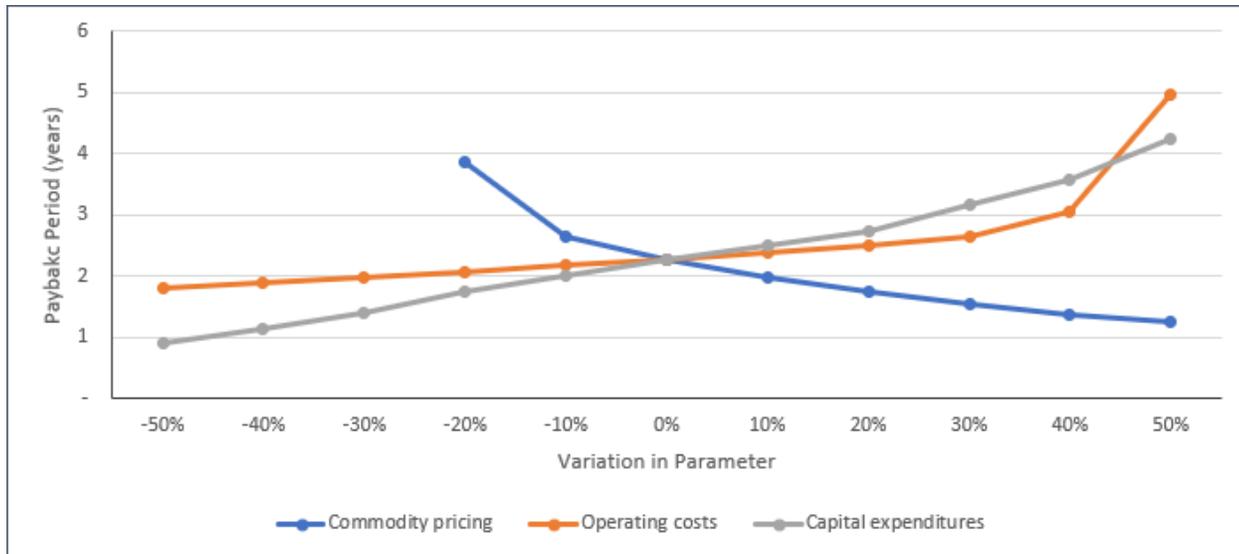
Source: Evomine, 2025.

Figure 1-6: Post-Tax Internal Rate of Return Sensitivity



Source: Evomine, 2025.

Figure 1-7: Post-Tax Payback Period Sensitivity



Source: Evomine, 2025.

1.21 Conclusions & Recommendations

This technical report is prepared in accordance with the guidelines of the Canadian Securities Administrators’ National Instrument 43-101 (NI 43-101) and Form 43-101F1. The objective of this report and the PEA is the evaluation of the potential technical feasibility and potential economic viability of the project, notably the development of an open pit mine, including processing facilities and supporting infrastructure.

This report confirms the potential technical feasibility and potential economic viability of the project based on an open pit mining operation that generates, on an after-tax basis, an NPV (8%) of \$505 million, with a 2.3-year payback period and an IRR of 27.2%.

Following the results of the financial analysis of this PEA, which demonstrates positive project economics, the authors recommend that additional work be undertaken to support a prefeasibility study or feasibility study for the project. Table 1-6 summarizes the estimated costs of the recommended future work on the Opémiska Project.

Table 1-6: Recommended Work Plan

Discipline	Program Component	Estimated Cost (\$M)
Geology and Mineral Resource	Update modelling of historical underground workings, additional drilling on the Saddle zone between the two pits, infill drilling to upgrade inferred resources (as required after update of underground workings model) and drilling program (5,000 m) on Cooke deposit	3.5
Mineral Processing and Metallurgical Testing	JK Axb, abrasion, bond work indices, concentrate regarding signature plot, gravity amenability testing, open circuit flotation, locked-cycle testing, dynamic thickening and pressure filtration, concentrate characterization, material flow properties, slurry rheology	1.0
Mining Methods	Geotechnical data collection and study, hydrogeological testing and analysis, prefeasibility study and associated trade-off studies	2.3
Co-Disposal Storage Facility and Water Management Infrastructure	PFS engineering, including CDSF foundation material stratigraphy, new tailings properties, geotechnical investigations and data collection, environmental geochemistry characteristics, develop a conceptual hydrogeological model, develop a detailed overall site water balance	0.7
Environmental, Permitting, Social or Community Impact	Stakeholder consultation, baseline studies, geochemical characterization, federal and provincial environmental assessments	2.0
Total		9.5

2 INTRODUCTION

2.1 Introduction

XXIX Metal Corp (XXIX) commissioned Ausenco Engineering Canada ULC (Ausenco), along with other consultants, to compile a preliminary economic assessment (PEA) for the Opémiska Project. The project is located next to the community of Chapais in the province of Quebec, approximately 480 km north of Montreal. 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:

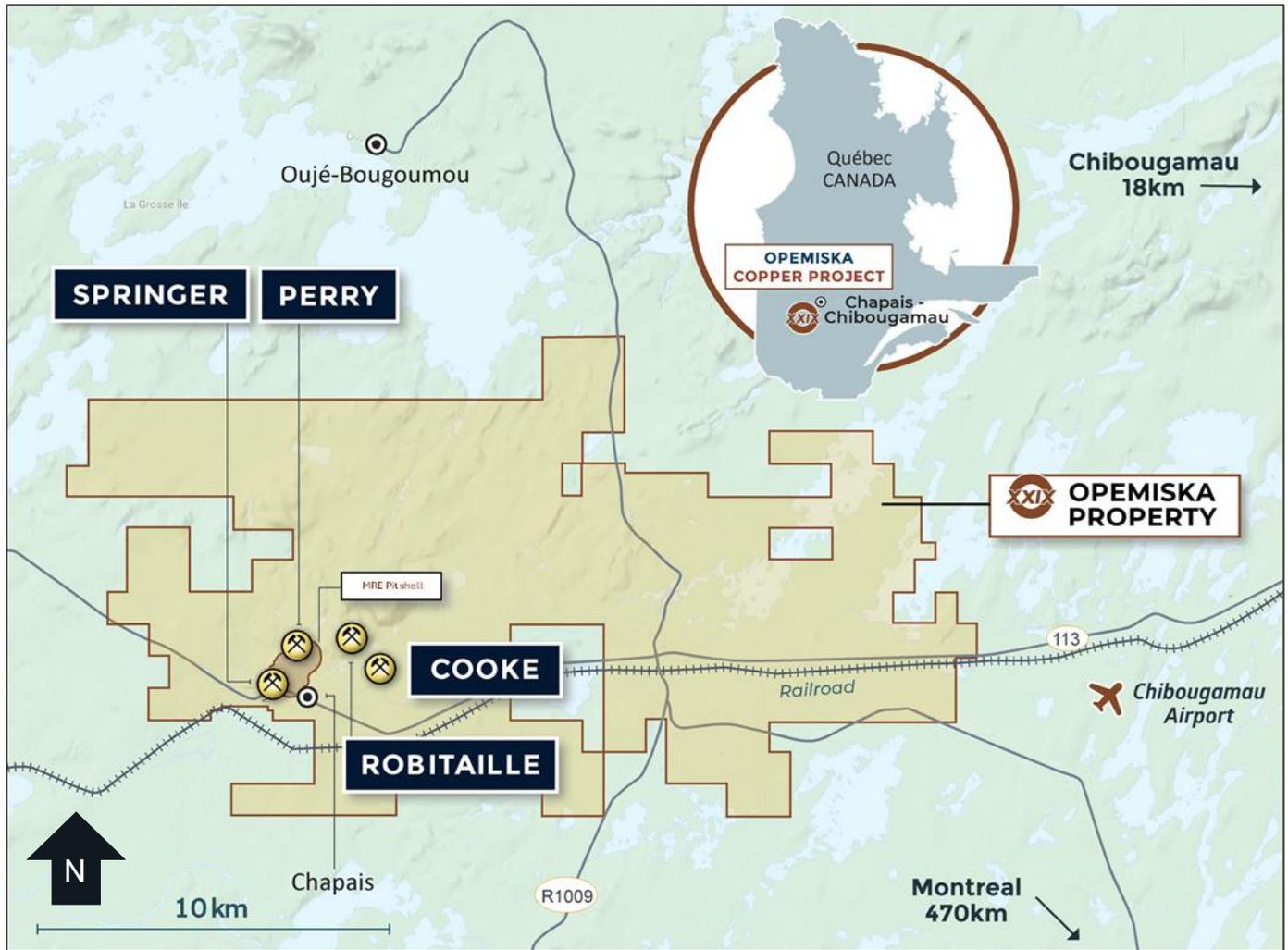
- Ausenco managed and coordinated the work related to the report. Ausenco developed the PEA-level design for the process plant, general site infrastructure and compiled the cost estimates for these.
- PLR Resources Inc. (PLR) completed the work related to property description, accessibility, local resources, geological setting, deposit type, exploration work, drilling, exploration works, sample preparation and analysis, data verification, and mineral resource estimate (MRE).
- Evomine Consulting Inc. (Evomine) designed the open pit, haul roads, mineralized material stockpiles, mine production schedules, the mine capital and operating costs, and compiled the overall project cost estimates and financial models.
- SRK Consulting (Canada) Inc. (SRK) designed the tailings and waste rock co-disposal facility and water management structures & facilities for the project.
- Geodoz Conseil Inc. (Geodoz) completed the work related to the environmental studies and permitting.

XXIX is a Canadian publicly traded company listed on the TSX Venture Exchange (TSXV) under the trading symbol “XXIX,” and has a head office in Toronto, Ontario. XXIX was previously known by the names “Power Ore” and “QC Copper and Gold”.

The project is located within the northeast portion of the Abitibi Metavolcanic Belt, 480 km north of Montreal (Figure 2-1). It lies adjacent to and partly within the community of Chapais.

The property lies on NTS map sheets 32G15.

Figure 2-1: Location of Property



Source: XXIX, 2025.

2.2 Qualified Persons

By virtue of their education, experience and professional association membership, the individuals listed in Table 2-1 are considered qualified persons, as defined by NI 43-10.

Table 2-1: Report Contributors

Qualified Person	Professional Designation	Position	Employer	Independent of Client
Renee Barrette	ing.	Principal Metallurgist	Ausenco Engineering Canada ULC	Yes
Alexandre Burelle	P.Eng.	Senior Mining Engineer	Evomine Consulting Inc.	Yes
Stephen Coates	P.Eng.	Senior Mining Engineer	Evomine Consulting Inc.	Yes
Maude Lévesque Michaud	ing.	Engineer	Geodoz Conseil Inc.	Yes
Pierre Luc Richard	P.Geo.	Principal Geologist	PLR Resources Inc.	Yes
Jean-Francois St-Laurent	ing., P.Eng., MSc.	Principal Consultant	SRK Consulting (Canada) Inc.	Yes
Charles Veilleux	ing.	Senior Consultant	SRK Consulting (Canada) Inc.	Yes

2.3 Terms of Reference

The purpose of this report is to present the results of the PEA and to support the disclosures by XXIX in a news release dated October 21, 2025 and titled “XXIX’s Opémiska PEA Confirms Positive Development Potential.”

All measurement units used in this technical report are metric unless otherwise noted. Currency is expressed in Canadian dollars (CAD, C\$). This technical report uses English.

Mineral resources are estimated in accordance with the 2019 edition of the Canadian Institute of Mining, Metallurgy and Exploration (CIM) Estimation of Mineral Resources & Mineral Reserves Best Practice Guidelines (2019 CIM Best Practice Guidelines) and are reported using the 2014 CIM Definition Standards for Mineral Resources and Mineral Reserves (2014 CIM Definition Standards).

Readers are cautioned that the PEA is preliminary in nature. 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 PEA will be realized.

2.4 Site Visits and Scope of Personal Inspection

2.4.1 Renée Barrette, Site Visit

Renée Barrette, ing., visited the site on October 27, 2025. During the visit, she reviewed the core shack, observed sample collection and labelling procedures, discussed the QA/QC procedures with the geology team and reviewed locations of future infrastructure.

2.4.2 Alexandre Burelle, Site Visit

Alexandre Burelle, ing., has not visited the project site.

2.4.3 Stephen Coates, Site Visit

Stephen Coates, P.Eng., visited the site on October 21, 2025 for one day. The visit included touring the core shack and reviewing the location of the proposed pits.

2.4.4 Maude Lévesque Michaud, Site Visit

Maude Lévesque Michaud visited the Opémiska project site on October 21, 2025. During the visit, she had an overview of the former Opémiska mine, the old tailings facilities, the site selected for the co-disposal facility, the core storage facility, and the town of Chapais.

2.4.5 Pierre Luc Richard, Site Visit

Pierre-Luc Richard visited the project site on May 1, 2025. The site visit included a visual inspection of core, as well as a field tour and discussions of the geological interpretations with geologists and geotechnicians employed by XXIX. The site visit also included a review of sampling and assaying procedures, the quality assurance / quality control (QA/QC) program, downhole survey methodologies, and the descriptions (logging) of lithologies, alteration and structures. The QP reviewed several sections of mineralized core while visiting the site. All core boxes were labelled and properly stored inside the core shack. The QP could also access the outdoor historical core storage facility during the site visit. In the reviewed core boxes, sample tags were present, and it was possible to validate sample numbers and confirm the presence of mineralization in witness half-core samples from the mineralized zones.

2.4.6 Jean-Francois St-Laurent, Site Visit

Jean-Francois St-Laurent, ing., has not visited the project site.

2.4.7 Charles Veilleux, Site Visit

Charles Veilleux, ing., has not visited the project site.

2.5 Effective Date

There are several significant dates related to this report as follows:

- Mineral resource estimate effective date: May 30, 2025
- Financial model effective date: October 17, 2025
- Report date: November 27, 2025.

The effective date of the report is the date of the financial model: October 17, 2025.

2.6 Information Sources and References

The issuer supplied information on mining titles, options agreements, royalty agreements, environmental liabilities and permits. For the latest ownership and mining title status, the QP consulted the Government of Quebec’s online claim management system at <https://gestim.mines.gouv.qc.ca>. Although the QP has reviewed the option agreements and claim status, the QP is not qualified to express any legal opinion concerning the property titles, current ownership, or possible litigations. A description of such agreements and the property and ownership thereof is provided for general information only. In this regard, the QP has relied on information supplied by the issuer and the work of experts they understand to be appropriately qualified.

This information supports Chapter 4, Property Description and Location.

2.7 Previous Reports

XXIX has filed the following previous report on the project:

- Richard, Pierre-Luc; Coates, Stephen; and Laroche, Christian (2025). “NI 43-101 Technical Report on the Opémiska Project with an Updated Mineral Resource Estimate for the Opémiska Deposit, Quebec, Canada.” July 18, 2025.

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 abbreviation: CAD) unless otherwise stated. Contained gold 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 2-2, and units of measurement are listed in Table 2-3.

Table 2-2: Abbreviations and Acronyms

Abbreviation	Description
AA	atomic absorption spectroscopy
Au	gold
AMD or ARD	acid mine drainage or acid rock drainage
Az	azimuth
BIF	banded iron formation
BWi	bond ball mill work index
CAD:USD	Canadian-American exchange rate
CDSF	Co-disposal storage facility
CDPNQ	Centre de données sur le patrimoine naturel du Québec
CIM	Canadian Institute of Mining, Metallurgy and Petroleum
CIM Best Practice Guidelines	CIM Estimation of Mineral Resources & Mineral Reserves Best Practice Guidelines
CIM Definition Standards	CIM Definition Standards for Mineral Resources and Mineral Reserves 2014
CIP	carbon in pulp
CND	contaminated neutral drainage
COMEV	Comité d’évaluation des répercussions sur l’environnement et le milieu social
COMEX	Comité d’examen des répercussions sur l’environnement et le milieu social
CoG	cut-off grade
CPTAQ	Commission de protection du territoire agricole du Québec

Abbreviation	Description
CRM	certified reference material
CWi	Bond crusher work index
DCIP	direct current resistivity and induced polarization
DDH	diamond drill hole
E-GRG	extended gravity recoverable gold
EM	electromagnetic
EQA	Environment Quality Act
FA	fire assay
Falconbridge Copper Limited	Falconbridge
FET	federal excise tax
FS	feasibility study
G&A	general and administration
GPR	gross production royalty
GQCV	greenstone-hosted quartz-carbonate vein deposits
GRAV	gravimetric finish method
ICP	inductively coupled plasma
ICP-OES	inductively coupled plasma - optical emission spectrometry
ID2	inverse distance squared
ID3	inverse distance cubed
IOCG	iron oxide copper gold
IP	induced polarization
IRGS	intrusion-related gold system
ISO	International Organization for Standardization
JBNQA	James Bay and Northern Quebec Agreement
LIDAR	light detection and ranging
LUP	land use permit
MCF	mechanized cut and fill
MELCC	Ministère de l'Environnement et de la Lutte contre les changements climatiques
MELCCFP	Ministère de l'Environnement, de la Lutte contre les changements climatiques, de la Faune et des Parcs
Minnova	Minnova Inc.
ML-ARD	metal leaching and acid rock drainage
MRE	mineral resource estimate
MRNF	Ministère des Ressources naturelles et des Forêts
NAD 83	North American Datum of 1983
NAG	non-acid generating
NI 43-101	National Instrument 43-101 (Regulation 43-101 in Quebec)
NN	nearest neighbour
NPV	net present value
NSR	net smelter return
NTS	national topographic system
OK	ordinary kriging
PAG	potentially acid generating
PEA	preliminary economic assessment
PFS	prefeasibility study
PGE	platinum group elements
QA/QC	quality assurance/quality control
QP	qualified person (as defined in National Instrument 43-101)
RCM	regional county municipality
ROM	run of mine
RQD	rock quality designation

Abbreviation	Description
SAG	semi-autogenous grinding
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
TMF	tailings management facility
UG	underground
UTM	Universal Transverse Mercator coordinate system
UV	ultraviolet
VLF-EM	very low frequency electromagnetic
VMS	volcanogenic massive sulphide
WR	Waste Rock

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 (micrometer)
cm	centimeter
cm ³	cubic centimeter
ft	foot (12 inches)
g	gram
g/cm ³	gram per cubic centimeter
g/L	gram per liter
g/t	gram per metric ton (tonne)
h	hour (60 minutes)
ha	hectare
kg	kilogram
kg/t	kilogram per tonne
km	kilometer
km ²	square kilometer
kW	kilowatt
kWh/t	kilowatt-hour per tonne
L	liter
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	millimeter
Moz	million (troy) ounces

Abbreviation	Description
Mt	million tonnes
Mt/a	million tonner per year
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
t/a	tonnes per year
USD	US dollars (currency)
US\$	US dollar (as symbol)

3 RELIANCE ON OTHER EXPERTS

The QPs did not rely on other experts for the purposes of this technical report.

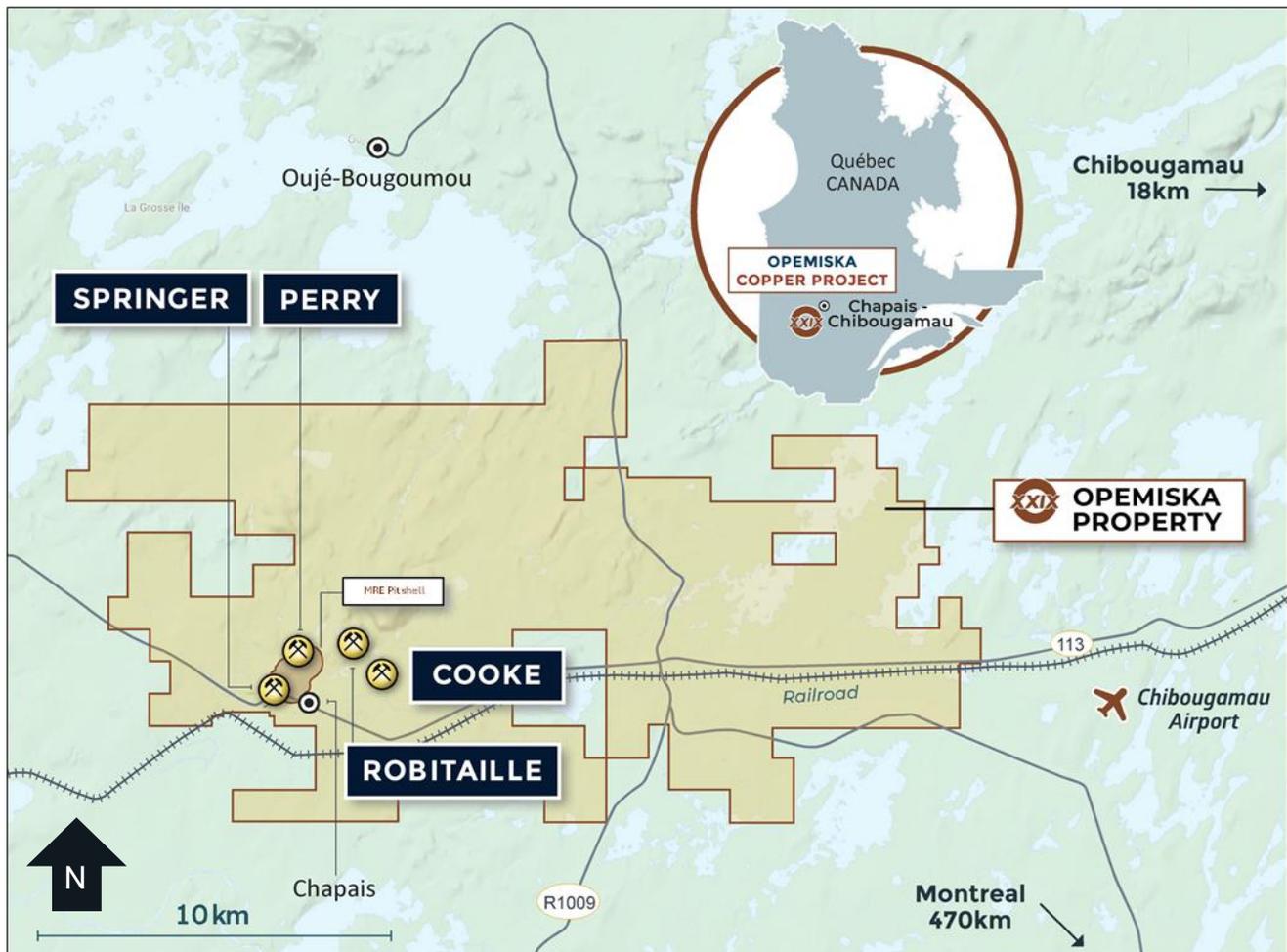
4 PROPERTY DESCRIPTION AND LOCATION

4.1 Introduction

The project is located within the northeast portion of the Abitibi Metavolcanic Belt, 480 km north of Montreal (Figure 4-1). It lies adjacent to, and partly within, the community of Chapais. The centre of the project site is approximately located at UTM coordinates N 5 516 000, E 510 000 (Zone 18N).

The property lies on NTS map sheets 32G15.

Figure 4-1: Project Location



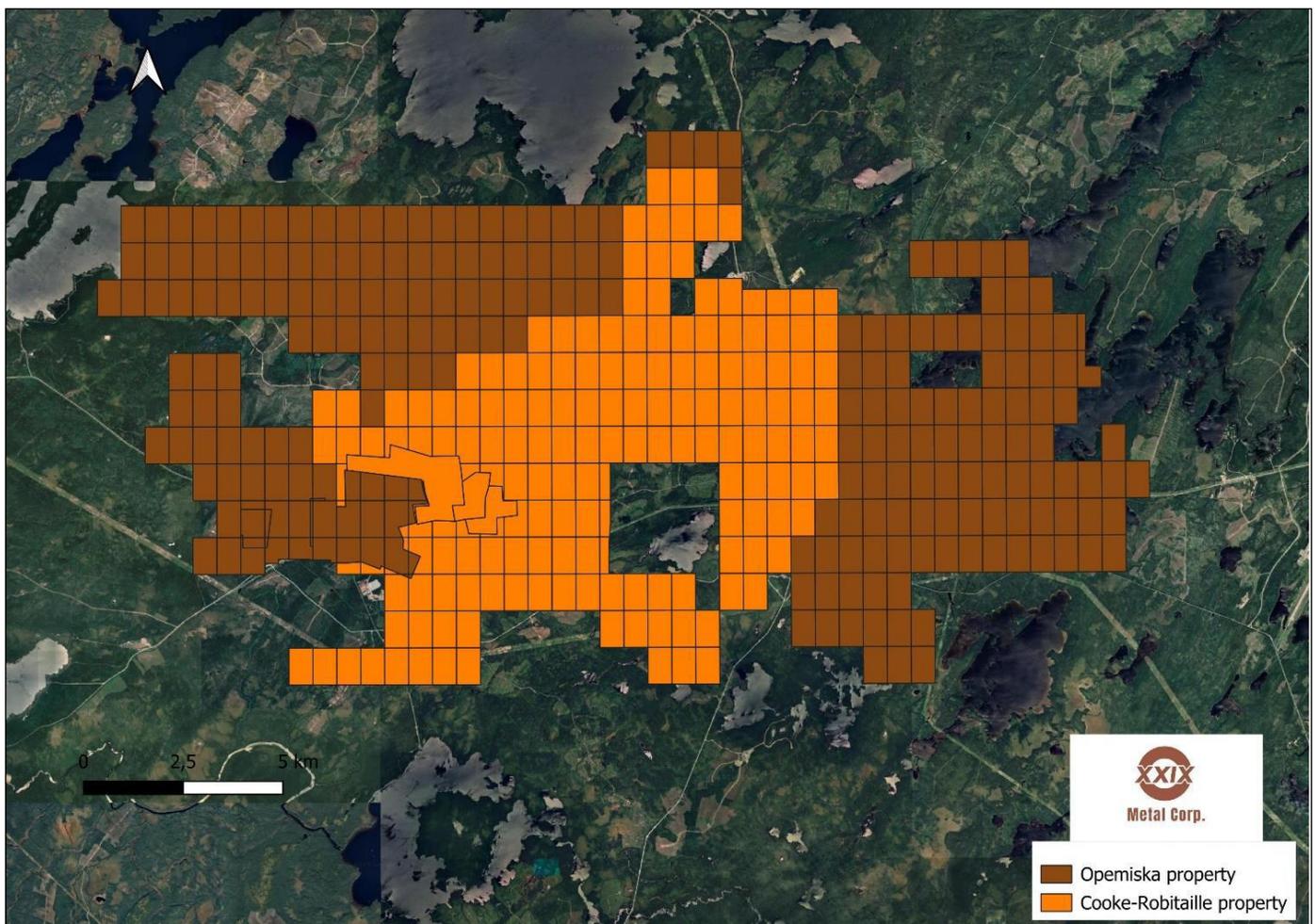
Source: XXIX, 2025.

4.2 Mineral Tenure

XXIX owns a group of 241 mining titles covering a total of 12,431 ha (Figure 4-2). This information is current as of July 13, 2025. Detailed lists of the mining titles are shown in Table 4-1. The mining titles are recorded under XXIX and are in good standing as of the effective date of this report. The MRE presented in Section 14 is found on mining titles 2837715, 2837712, 2837716, 2837721, 2837706 and 2173578 (registered under XXIX).

XXIX is currently in the process of acquiring 175 additional mining titles covering a total of 9,068 ha (Figure 4-2) from 2736-1170 Quebec Inc. (85%), Ovalbay Geological Services Inc. (10%), and Melissa Darveau (5%), together referred to as the “Cooke/Robitaille Option” agreement. XXIX is still in the earn-in process with approximately \$900,000 of work obligations to complete by July 13, 2026.

Figure 4-2: Mining Titles



Source: PLR, 2025.

Table 4-1: Details of Mining Titles (as of July 13, 2025)

Title	Ownership	Area (Ha)	Expiry Date	Renewal Work Required	Registration	Royalties
2020223	XXIX	55.56	2026-07-06	\$2,500	XXIX Metal Corp.	2% NSR (50% Buyback option)
2020224	XXIX	55.56	2026-07-06	\$2,500	XXIX Metal Corp.	2% NSR (50% Buyback option)
2020225	XXIX	55.56	2026-07-06	\$2,500	XXIX Metal Corp.	2% NSR (50% Buyback option)
2020226	XXIX	55.56	2026-07-06	\$2,500	XXIX Metal Corp.	2% NSR (50% Buyback option)
2020227	XXIX	55.56	2026-07-06	\$2,500	XXIX Metal Corp.	2% NSR (50% Buyback option)
2020228	XXIX	55.55	2026-07-06	\$2,500	XXIX Metal Corp.	2% NSR (50% Buyback option)
2020229	XXIX	55.55	2026-07-06	\$2,500	XXIX Metal Corp.	2% NSR (50% Buyback option)
2020233	XXIX	55.54	2026-07-06	\$2,500	XXIX Metal Corp.	2% NSR (50% Buyback option)
2020234	XXIX	55.54	2026-07-06	\$2,500	XXIX Metal Corp.	2% NSR (50% Buyback option)
2386468	XXIX	55.57	2027-01-14	\$2,500	XXIX Metal Corp.	2% NSR (50% Buyback option)
2386469	XXIX	55.57	2027-01-14	\$2,500	XXIX Metal Corp.	2% NSR (50% Buyback option)
2386471	XXIX	53.86	2027-01-14	\$2,500	XXIX Metal Corp.	2% NSR (50% Buyback option)
2386472	XXIX	55.58	2027-01-14	\$2,500	XXIX Metal Corp.	2% NSR (50% Buyback option)
2386473	XXIX	55.59	2027-01-14	\$2,500	XXIX Metal Corp.	2% NSR (50% Buyback option)
2386474	XXIX	55.57	2027-01-14	\$2,500	XXIX Metal Corp.	2% NSR (50% Buyback option)
2386475	XXIX	55.57	2027-01-14	\$2,500	XXIX Metal Corp.	2% NSR (50% Buyback option)
2386476	XXIX	31.72	2027-01-14	\$2,500	XXIX Metal Corp.	2% NSR (50% Buyback option)
2386477	XXIX	46.40	2027-01-14	\$2,500	XXIX Metal Corp.	2% NSR (50% Buyback option)
2386478	XXIX	55.26	2027-01-14	\$2,500	XXIX Metal Corp.	2% NSR (50% Buyback option)
2386480	XXIX	34.70	2027-01-14	\$2,500	XXIX Metal Corp.	2% NSR (50% Buyback option)
2386481	XXIX	0.26	2027-01-14	\$1,000	XXIX Metal Corp.	2% NSR (50% Buyback option)
2386482	XXIX	50.30	2027-01-14	\$2,500	XXIX Metal Corp.	2% NSR (50% Buyback option)
2386483	XXIX	41.44	2027-01-14	\$2,500	XXIX Metal Corp.	2% NSR (50% Buyback option)
2386484	XXIX	13.07	2027-01-14	\$1,000	XXIX Metal Corp.	2% NSR (50% Buyback option)
2386485	XXIX	55.59	2027-01-14	\$2,500	XXIX Metal Corp.	2% NSR (50% Buyback option)
2387110	XXIX	1.20	2027-06-21	\$1,000	XXIX Metal Corp.	2% NSR (50% Buyback option)
2387112	XXIX	5.27	2027-06-21	\$1,000	XXIX Metal Corp.	2% NSR (50% Buyback option)
2387114	XXIX	0.31	2027-06-21	\$1,000	XXIX Metal Corp.	2% NSR (50% Buyback option)
2387115	XXIX	1.71	2027-06-21	\$1,000	XXIX Metal Corp.	2% NSR (50% Buyback option)
2390804	XXIX	55.55	2027-09-16	\$1,800	XXIX Metal Corp.	2% NSR (50% Buyback option)
2390805	XXIX	55.54	2027-09-16	\$1,800	XXIX Metal Corp.	2% NSR (50% Buyback option)
2390806	XXIX	55.54	2027-09-16	\$1,800	XXIX Metal Corp.	2% NSR (50% Buyback option)
2423608	XXIX	55.56	2027-02-22	\$1,800	XXIX Metal Corp.	2% NSR (50% Buyback option)
2423989	XXIX	55.54	2027-02-26	\$1,800	XXIX Metal Corp.	2% NSR (50% Buyback option)
2426554	XXIX	55.56	2027-04-15	\$1,800	XXIX Metal Corp.	2% NSR (50% Buyback option)
2426559	XXIX	55.55	2027-04-15	\$1,800	XXIX Metal Corp.	2% NSR (50% Buyback option)
2450419	XXIX	55.53	2026-06-20	\$1,800	XXIX Metal Corp.	2% NSR (50% Buyback option)
2459098	XXIX	14.15	2026-08-24	\$750	XXIX Metal Corp.	2% NSR (50% Buyback option)
2459106	XXIX	42.25	2026-08-24	\$1,800	XXIX Metal Corp.	2% NSR (50% Buyback option)
2459109	XXIX	1.79	2026-08-24	\$750	XXIX Metal Corp.	2% NSR (50% Buyback option)
2459113	XXIX	9.18	2026-08-24	\$750	XXIX Metal Corp.	2% NSR (50% Buyback option)
2471572	XXIX	19.54	2027-01-03	\$750	XXIX Metal Corp.	2% NSR (50% Buyback option)
2520350	XXIX	55.56	2026-07-04	\$1,200	XXIX Metal Corp.	2% NSR (50% Buyback option)
2520351	XXIX	55.55	2026-07-04	\$1,200	XXIX Metal Corp.	2% NSR (50% Buyback option)
2520352	XXIX	55.55	2026-07-04	\$1,200	XXIX Metal Corp.	2% NSR (50% Buyback option)
2520353	XXIX	55.54	2026-07-04	\$1,200	XXIX Metal Corp.	2% NSR (50% Buyback option)
2520354	XXIX	55.54	2026-07-04	\$1,200	XXIX Metal Corp.	2% NSR (50% Buyback option)

Title	Ownership	Area (Ha)	Expiry Date	Renewal Work Required	Registration	Royalties
2520355	XXIX	55.54	2026-07-04	\$1,200	XXIX Metal Corp.	2% NSR (50% Buyback option)
2520356	XXIX	55.54	2026-07-04	\$1,200	XXIX Metal Corp.	2% NSR (50% Buyback option)
2520357	XXIX	40.03	2026-07-04	\$1,200	XXIX Metal Corp.	2% NSR (50% Buyback option)
2520358	XXIX	55.53	2026-07-04	\$1,200	XXIX Metal Corp.	2% NSR (50% Buyback option)
2520359	XXIX	55.53	2026-07-04	\$1,200	XXIX Metal Corp.	2% NSR (50% Buyback option)
2520360	XXIX	55.53	2026-07-04	\$1,200	XXIX Metal Corp.	2% NSR (50% Buyback option)
2520361	XXIX	55.53	2026-07-04	\$1,200	XXIX Metal Corp.	2% NSR (50% Buyback option)
2520362	XXIX	55.53	2026-07-04	\$1,200	XXIX Metal Corp.	2% NSR (50% Buyback option)
2520363	XXIX	55.53	2026-07-04	\$1,200	XXIX Metal Corp.	2% NSR (50% Buyback option)
2520364	XXIX	55.52	2026-07-04	\$1,200	XXIX Metal Corp.	2% NSR (50% Buyback option)
2520365	XXIX	55.52	2026-07-04	\$1,200	XXIX Metal Corp.	2% NSR (50% Buyback option)
2520366	XXIX	55.52	2026-07-04	\$1,200	XXIX Metal Corp.	2% NSR (50% Buyback option)
2520367	XXIX	55.51	2026-07-04	\$1,200	XXIX Metal Corp.	2% NSR (50% Buyback option)
2520368	XXIX	55.51	2026-07-04	\$1,200	XXIX Metal Corp.	2% NSR (50% Buyback option)
2520369	XXIX	55.51	2026-07-04	\$1,200	XXIX Metal Corp.	2% NSR (50% Buyback option)
2520370	XXIX	55.51	2026-07-04	\$1,200	XXIX Metal Corp.	2% NSR (50% Buyback option)
2520371	XXIX	55.51	2026-07-04	\$1,200	XXIX Metal Corp.	2% NSR (50% Buyback option)
2520388	XXIX	55.59	2026-07-09	\$1,200	XXIX Metal Corp.	2% NSR (50% Buyback option)
2520389	XXIX	55.59	2026-07-09	\$1,200	XXIX Metal Corp.	2% NSR (50% Buyback option)
2520390	XXIX	55.59	2026-07-09	\$1,200	XXIX Metal Corp.	2% NSR (50% Buyback option)
2520391	XXIX	55.59	2026-07-09	\$1,200	XXIX Metal Corp.	2% NSR (50% Buyback option)
2520392	XXIX	55.59	2026-07-09	\$1,200	XXIX Metal Corp.	2% NSR (50% Buyback option)
2520393	XXIX	55.59	2026-07-09	\$1,200	XXIX Metal Corp.	2% NSR (50% Buyback option)
2520394	XXIX	55.59	2026-07-09	\$1,200	XXIX Metal Corp.	2% NSR (50% Buyback option)
2520395	XXIX	55.59	2026-07-09	\$1,200	XXIX Metal Corp.	2% NSR (50% Buyback option)
2520396	XXIX	55.58	2026-07-09	\$1,200	XXIX Metal Corp.	2% NSR (50% Buyback option)
2520397	XXIX	55.58	2026-07-09	\$1,200	XXIX Metal Corp.	2% NSR (50% Buyback option)
2520398	XXIX	55.58	2026-07-09	\$1,200	XXIX Metal Corp.	2% NSR (50% Buyback option)
2520399	XXIX	55.58	2026-07-09	\$1,200	XXIX Metal Corp.	2% NSR (50% Buyback option)
2520400	XXIX	55.57	2026-07-09	\$1,200	XXIX Metal Corp.	2% NSR (50% Buyback option)
2520401	XXIX	55.57	2026-07-09	\$1,200	XXIX Metal Corp.	2% NSR (50% Buyback option)
2520402	XXIX	55.57	2026-07-09	\$1,200	XXIX Metal Corp.	2% NSR (50% Buyback option)
2520403	XXIX	55.57	2026-07-09	\$1,200	XXIX Metal Corp.	2% NSR (50% Buyback option)
2520507	XXIX	55.58	2026-07-09	\$1,200	XXIX Metal Corp.	2% NSR (50% Buyback option)
2520508	XXIX	55.58	2026-07-09	\$1,200	XXIX Metal Corp.	2% NSR (50% Buyback option)
2520509	XXIX	55.58	2026-07-09	\$1,200	XXIX Metal Corp.	2% NSR (50% Buyback option)
2520510	XXIX	55.57	2026-07-09	\$1,200	XXIX Metal Corp.	2% NSR (50% Buyback option)
2520511	XXIX	55.57	2026-07-09	\$1,200	XXIX Metal Corp.	2% NSR (50% Buyback option)
2520512	XXIX	55.57	2026-07-09	\$1,200	XXIX Metal Corp.	2% NSR (50% Buyback option)
2531831	XXIX	55.57	2027-02-21	\$1,200	XXIX Metal Corp.	2% NSR (50% Buyback option)
2531832	XXIX	55.57	2027-02-21	\$1,200	XXIX Metal Corp.	2% NSR (50% Buyback option)
2531833	XXIX	55.56	2027-02-21	\$1,200	XXIX Metal Corp.	2% NSR (50% Buyback option)
2531834	XXIX	55.56	2027-02-21	\$1,200	XXIX Metal Corp.	2% NSR (50% Buyback option)
2531835	XXIX	55.56	2027-02-21	\$1,200	XXIX Metal Corp.	2% NSR (50% Buyback option)
2531836	XXIX	55.55	2027-02-21	\$1,200	XXIX Metal Corp.	2% NSR (50% Buyback option)
2531837	XXIX	55.55	2027-02-21	\$1,200	XXIX Metal Corp.	2% NSR (50% Buyback option)
2531838	XXIX	55.55	2027-02-21	\$1,200	XXIX Metal Corp.	2% NSR (50% Buyback option)
2531839	XXIX	55.55	2027-02-21	\$1,200	XXIX Metal Corp.	2% NSR (50% Buyback option)
2563631	XXIX	55.62	2026-05-04	\$1,200	XXIX Metal Corp.	2% NSR (50% Buyback option)

Title	Ownership	Area (Ha)	Expiry Date	Renewal Work Required	Registration	Royalties
2563632	XXIX	55.62	2026-05-04	\$1,200	XXIX Metal Corp.	2% NSR (50% Buyback option)
2563633	XXIX	55.62	2026-05-04	\$1,200	XXIX Metal Corp.	2% NSR (50% Buyback option)
2563634	XXIX	55.61	2026-05-04	\$1,200	XXIX Metal Corp.	2% NSR (50% Buyback option)
2563635	XXIX	55.61	2026-05-04	\$1,200	XXIX Metal Corp.	2% NSR (50% Buyback option)
2563636	XXIX	55.61	2026-05-04	\$1,200	XXIX Metal Corp.	2% NSR (50% Buyback option)
2563637	XXIX	55.61	2026-05-04	\$1,200	XXIX Metal Corp.	2% NSR (50% Buyback option)
2563638	XXIX	55.61	2026-05-04	\$1,200	XXIX Metal Corp.	2% NSR (50% Buyback option)
2563639	XXIX	55.61	2026-05-04	\$1,200	XXIX Metal Corp.	2% NSR (50% Buyback option)
2563640	XXIX	55.60	2026-05-04	\$1,200	XXIX Metal Corp.	2% NSR (50% Buyback option)
2563641	XXIX	55.60	2026-05-04	\$1,200	XXIX Metal Corp.	2% NSR (50% Buyback option)
2563642	XXIX	55.60	2026-05-04	\$1,200	XXIX Metal Corp.	2% NSR (50% Buyback option)
2563643	XXIX	55.60	2026-05-04	\$1,200	XXIX Metal Corp.	2% NSR (50% Buyback option)
2563644	XXIX	55.60	2026-05-04	\$1,200	XXIX Metal Corp.	2% NSR (50% Buyback option)
2563645	XXIX	55.59	2026-05-04	\$1,200	XXIX Metal Corp.	2% NSR (50% Buyback option)
2563646	XXIX	55.59	2026-05-04	\$1,200	XXIX Metal Corp.	2% NSR (50% Buyback option)
2563647	XXIX	55.59	2026-05-04	\$1,200	XXIX Metal Corp.	2% NSR (50% Buyback option)
2563648	XXIX	55.59	2026-05-04	\$1,200	XXIX Metal Corp.	2% NSR (50% Buyback option)
2563649	XXIX	55.59	2026-05-04	\$1,200	XXIX Metal Corp.	2% NSR (50% Buyback option)
2563650	XXIX	55.59	2026-05-04	\$1,200	XXIX Metal Corp.	2% NSR (50% Buyback option)
2563651	XXIX	55.58	2026-05-04	\$1,200	XXIX Metal Corp.	2% NSR (50% Buyback option)
2563652	XXIX	55.57	2026-05-04	\$1,200	XXIX Metal Corp.	2% NSR (50% Buyback option)
2564714	XXIX	55.57	2026-05-14	\$1,200	XXIX Metal Corp.	2% NSR (50% Buyback option)
2564715	XXIX	55.56	2026-05-14	\$1,200	XXIX Metal Corp.	2% NSR (50% Buyback option)
2564716	XXIX	55.56	2026-05-14	\$1,200	XXIX Metal Corp.	2% NSR (50% Buyback option)
2564717	XXIX	55.56	2026-05-14	\$1,200	XXIX Metal Corp.	2% NSR (50% Buyback option)
2564718	XXIX	55.56	2026-05-14	\$1,200	XXIX Metal Corp.	2% NSR (50% Buyback option)
2564719	XXIX	55.56	2026-05-14	\$1,200	XXIX Metal Corp.	2% NSR (50% Buyback option)
2564720	XXIX	55.55	2026-05-14	\$1,200	XXIX Metal Corp.	2% NSR (50% Buyback option)
2564721	XXIX	55.55	2026-05-14	\$1,200	XXIX Metal Corp.	2% NSR (50% Buyback option)
2564722	XXIX	55.55	2026-05-14	\$1,200	XXIX Metal Corp.	2% NSR (50% Buyback option)
2564723	XXIX	55.55	2026-05-14	\$1,200	XXIX Metal Corp.	2% NSR (50% Buyback option)
2564724	XXIX	55.54	2026-05-14	\$1,200	XXIX Metal Corp.	2% NSR (50% Buyback option)
2564725	XXIX	55.54	2026-05-14	\$1,200	XXIX Metal Corp.	2% NSR (50% Buyback option)
2564726	XXIX	55.54	2026-05-14	\$1,200	XXIX Metal Corp.	2% NSR (50% Buyback option)
2564727	XXIX	55.53	2026-05-14	\$1,200	XXIX Metal Corp.	2% NSR (50% Buyback option)
2564728	XXIX	55.53	2026-05-14	\$1,200	XXIX Metal Corp.	2% NSR (50% Buyback option)
2564729	XXIX	55.53	2026-05-14	\$1,200	XXIX Metal Corp.	2% NSR (50% Buyback option)
2591941	XXIX	55.58	2026-12-20	\$1,200	XXIX Metal Corp.	2% NSR (50% Buyback option)
2591942	XXIX	55.58	2026-12-20	\$1,200	XXIX Metal Corp.	2% NSR (50% Buyback option)
2591943	XXIX	55.58	2026-12-20	\$1,200	XXIX Metal Corp.	2% NSR (50% Buyback option)
2591944	XXIX	55.58	2026-12-20	\$1,200	XXIX Metal Corp.	2% NSR (50% Buyback option)
2591945	XXIX	55.58	2026-12-20	\$1,200	XXIX Metal Corp.	2% NSR (50% Buyback option)
2591946	XXIX	55.57	2026-12-20	\$1,200	XXIX Metal Corp.	2% NSR (50% Buyback option)
2591947	XXIX	55.57	2026-12-20	\$1,200	XXIX Metal Corp.	2% NSR (50% Buyback option)
2591948	XXIX	55.56	2026-12-20	\$1,200	XXIX Metal Corp.	2% NSR (50% Buyback option)
2591949	XXIX	55.49	2026-12-20	\$1,200	XXIX Metal Corp.	2% NSR (50% Buyback option)
2591950	XXIX	55.48	2026-12-20	\$1,200	XXIX Metal Corp.	2% NSR (50% Buyback option)
2591951	XXIX	55.48	2026-12-20	\$1,200	XXIX Metal Corp.	2% NSR (50% Buyback option)
2591952	XXIX	55.48	2026-12-20	\$1,200	XXIX Metal Corp.	2% NSR (50% Buyback option)

Title	Ownership	Area (Ha)	Expiry Date	Renewal Work Required	Registration	Royalties
2591953	XXIX	55.48	2026-12-20	\$1,200	XXIX Metal Corp.	2% NSR (50% Buyback option)
2636669	XXIX	55.54	2027-02-20	\$1,200	XXIX Metal Corp.	2% NSR (50% Buyback option)
2636670	XXIX	55.54	2027-02-20	\$1,200	XXIX Metal Corp.	2% NSR (50% Buyback option)
2636671	XXIX	55.53	2027-02-20	\$1,200	XXIX Metal Corp.	2% NSR (50% Buyback option)
2636672	XXIX	55.53	2027-02-20	\$1,200	XXIX Metal Corp.	2% NSR (50% Buyback option)
2636673	XXIX	55.53	2027-02-20	\$1,200	XXIX Metal Corp.	2% NSR (50% Buyback option)
2636674	XXIX	55.53	2027-02-20	\$1,200	XXIX Metal Corp.	2% NSR (50% Buyback option)
2636675	XXIX	55.53	2027-02-20	\$1,200	XXIX Metal Corp.	2% NSR (50% Buyback option)
2636676	XXIX	55.53	2027-02-20	\$1,200	XXIX Metal Corp.	2% NSR (50% Buyback option)
2636677	XXIX	55.53	2027-02-20	\$1,200	XXIX Metal Corp.	2% NSR (50% Buyback option)
2636678	XXIX	55.53	2027-02-20	\$1,200	XXIX Metal Corp.	2% NSR (50% Buyback option)
2636679	XXIX	55.53	2027-02-20	\$1,200	XXIX Metal Corp.	2% NSR (50% Buyback option)
2636680	XXIX	55.53	2027-02-20	\$1,200	XXIX Metal Corp.	2% NSR (50% Buyback option)
2636688	XXIX	55.52	2027-02-20	\$1,200	XXIX Metal Corp.	2% NSR (50% Buyback option)
2636689	XXIX	55.52	2027-02-20	\$1,200	XXIX Metal Corp.	2% NSR (50% Buyback option)
2636690	XXIX	55.52	2027-02-20	\$1,200	XXIX Metal Corp.	2% NSR (50% Buyback option)
2636691	XXIX	55.52	2027-02-20	\$1,200	XXIX Metal Corp.	2% NSR (50% Buyback option)
2636692	XXIX	55.52	2027-02-20	\$1,200	XXIX Metal Corp.	2% NSR (50% Buyback option)
2636693	XXIX	55.52	2027-02-20	\$1,200	XXIX Metal Corp.	2% NSR (50% Buyback option)
2636694	XXIX	55.52	2027-02-20	\$1,200	XXIX Metal Corp.	2% NSR (50% Buyback option)
2636695	XXIX	55.52	2027-02-20	\$1,200	XXIX Metal Corp.	2% NSR (50% Buyback option)
2636696	XXIX	55.52	2027-02-20	\$1,200	XXIX Metal Corp.	2% NSR (50% Buyback option)
2636697	XXIX	55.52	2027-02-20	\$1,200	XXIX Metal Corp.	2% NSR (50% Buyback option)
2636698	XXIX	55.52	2027-02-20	\$1,200	XXIX Metal Corp.	2% NSR (50% Buyback option)
2837706	XXIX	55.58	2027-04-30	\$2,500	XXIX Metal Corp.	2% NSR (50% Buyback option)
2837707	XXIX	0.83	2027-04-30	\$1,000	XXIX Metal Corp.	2% NSR (50% Buyback option)
2837708	XXIX	11.85	2027-04-30	\$1,000	XXIX Metal Corp.	2% NSR (50% Buyback option)
2837709	XXIX	34.94	2027-04-30	\$2,500	XXIX Metal Corp.	2% NSR (50% Buyback option)
2837710	XXIX	36.16	2027-04-30	\$2,500	XXIX Metal Corp.	2% NSR (50% Buyback option)
2837711	XXIX	49.74	2027-04-30	\$2,500	XXIX Metal Corp.	2% NSR (50% Buyback option)
2837712	XXIX	55.58	2027-04-30	\$2,500	XXIX Metal Corp.	2% NSR (50% Buyback option)
2837713	XXIX	42.54	2027-04-30	\$2,500	XXIX Metal Corp.	2% NSR (50% Buyback option)
2837714	XXIX	0.80	2027-04-30	\$1,000	XXIX Metal Corp.	2% NSR (50% Buyback option)
2837715	XXIX	40.73	2027-04-30	\$2,500	XXIX Metal Corp.	2% NSR (50% Buyback option)
2837716	XXIX	37.97	2027-04-30	\$2,500	XXIX Metal Corp.	2% NSR (50% Buyback option)
2837717	XXIX	28.36	2027-04-30	\$2,500	XXIX Metal Corp.	2% NSR (50% Buyback option)
2837718	XXIX	20.77	2027-04-30	\$1,000	XXIX Metal Corp.	2% NSR (50% Buyback option)
2837719	XXIX	10.91	2027-04-30	\$1,000	XXIX Metal Corp.	2% NSR (50% Buyback option)
2837720	XXIX	47.86	2027-04-30	\$2,500	XXIX Metal Corp.	2% NSR (50% Buyback option)
2837721	XXIX	52.07	2027-04-30	\$2,500	XXIX Metal Corp.	2% NSR (50% Buyback option)
2845560	XXIX	55.52	2028-03-22	\$1,200	XXIX Metal Corp.	2% NSR (50% Buyback option)
2845561	XXIX	55.52	2028-03-22	\$1,200	XXIX Metal Corp.	2% NSR (50% Buyback option)
2845562	XXIX	55.52	2028-03-22	\$1,200	XXIX Metal Corp.	2% NSR (50% Buyback option)
2845563	XXIX	55.52	2028-03-22	\$1,200	XXIX Metal Corp.	2% NSR (50% Buyback option)
2845564	XXIX	55.52	2028-03-22	\$1,200	XXIX Metal Corp.	2% NSR (50% Buyback option)
2845565	XXIX	55.51	2028-03-22	\$1,200	XXIX Metal Corp.	2% NSR (50% Buyback option)
2845566	XXIX	55.51	2028-03-22	\$1,200	XXIX Metal Corp.	2% NSR (50% Buyback option)
2845567	XXIX	55.51	2028-03-22	\$1,200	XXIX Metal Corp.	2% NSR (50% Buyback option)
2845568	XXIX	55.51	2028-03-22	\$1,200	XXIX Metal Corp.	2% NSR (50% Buyback option)

Title	Ownership	Area (Ha)	Expiry Date	Renewal Work Required	Registration	Royalties
2845569	XXIX	55.51	2028-03-22	\$1,200	XXIX Metal Corp.	2% NSR (50% Buyback option)
2845570	XXIX	55.51	2028-03-22	\$1,200	XXIX Metal Corp.	2% NSR (50% Buyback option)
2845571	XXIX	55.51	2028-03-22	\$1,200	XXIX Metal Corp.	2% NSR (50% Buyback option)
2845572	XXIX	55.51	2028-03-22	\$1,200	XXIX Metal Corp.	2% NSR (50% Buyback option)
2845573	XXIX	55.51	2028-03-22	\$1,200	XXIX Metal Corp.	2% NSR (50% Buyback option)
2845574	XXIX	55.51	2028-03-22	\$1,200	XXIX Metal Corp.	2% NSR (50% Buyback option)
2845575	XXIX	55.51	2028-03-22	\$1,200	XXIX Metal Corp.	2% NSR (50% Buyback option)
2845576	XXIX	55.51	2028-03-22	\$1,200	XXIX Metal Corp.	2% NSR (50% Buyback option)
2845577	XXIX	55.51	2028-03-22	\$1,200	XXIX Metal Corp.	2% NSR (50% Buyback option)
2845578	XXIX	55.51	2028-03-22	\$1,200	XXIX Metal Corp.	2% NSR (50% Buyback option)
2845579	XXIX	55.51	2028-03-22	\$1,200	XXIX Metal Corp.	2% NSR (50% Buyback option)
2845580	XXIX	55.51	2028-03-22	\$1,200	XXIX Metal Corp.	2% NSR (50% Buyback option)
2845581	XXIX	55.51	2028-03-22	\$1,200	XXIX Metal Corp.	2% NSR (50% Buyback option)
2845582	XXIX	55.50	2028-03-22	\$1,200	XXIX Metal Corp.	2% NSR (50% Buyback option)
2845583	XXIX	55.50	2028-03-22	\$1,200	XXIX Metal Corp.	2% NSR (50% Buyback option)
2845584	XXIX	55.50	2028-03-22	\$1,200	XXIX Metal Corp.	2% NSR (50% Buyback option)
2845585	XXIX	55.50	2028-03-22	\$1,200	XXIX Metal Corp.	2% NSR (50% Buyback option)
2845586	XXIX	55.50	2028-03-22	\$1,200	XXIX Metal Corp.	2% NSR (50% Buyback option)
2845587	XXIX	55.50	2028-03-22	\$1,200	XXIX Metal Corp.	2% NSR (50% Buyback option)
2845588	XXIX	55.50	2028-03-22	\$1,200	XXIX Metal Corp.	2% NSR (50% Buyback option)
2845589	XXIX	55.50	2028-03-22	\$1,200	XXIX Metal Corp.	2% NSR (50% Buyback option)
2845590	XXIX	55.50	2028-03-22	\$1,200	XXIX Metal Corp.	2% NSR (50% Buyback option)
2845591	XXIX	55.50	2028-03-22	\$1,200	XXIX Metal Corp.	2% NSR (50% Buyback option)
2845592	XXIX	55.50	2028-03-22	\$1,200	XXIX Metal Corp.	2% NSR (50% Buyback option)
2845593	XXIX	55.50	2028-03-22	\$1,200	XXIX Metal Corp.	2% NSR (50% Buyback option)
2845594	XXIX	55.50	2028-03-22	\$1,200	XXIX Metal Corp.	2% NSR (50% Buyback option)
2845595	XXIX	55.50	2028-03-22	\$1,200	XXIX Metal Corp.	2% NSR (50% Buyback option)
2845596	XXIX	55.50	2028-03-22	\$1,200	XXIX Metal Corp.	2% NSR (50% Buyback option)
2845597	XXIX	55.50	2028-03-22	\$1,200	XXIX Metal Corp.	2% NSR (50% Buyback option)
2845598	XXIX	55.50	2028-03-22	\$1,200	XXIX Metal Corp.	2% NSR (50% Buyback option)
2845599	XXIX	55.50	2028-03-22	\$1,200	XXIX Metal Corp.	2% NSR (50% Buyback option)
2845603	XXIX	55.52	2028-03-23	\$1,200	XXIX Metal Corp.	2% NSR (50% Buyback option)
2845604	XXIX	55.52	2028-03-23	\$1,200	XXIX Metal Corp.	2% NSR (50% Buyback option)
2845605	XXIX	55.52	2028-03-23	\$1,200	XXIX Metal Corp.	2% NSR (50% Buyback option)
2845606	XXIX	55.52	2028-03-23	\$1,200	XXIX Metal Corp.	2% NSR (50% Buyback option)
2845607	XXIX	55.52	2028-03-23	\$1,200	XXIX Metal Corp.	2% NSR (50% Buyback option)
2845608	XXIX	55.52	2028-03-23	\$1,200	XXIX Metal Corp.	2% NSR (50% Buyback option)
2845609	XXIX	55.51	2028-03-23	\$1,200	XXIX Metal Corp.	2% NSR (50% Buyback option)
2845610	XXIX	55.51	2028-03-23	\$1,200	XXIX Metal Corp.	2% NSR (50% Buyback option)
2845611	XXIX	55.51	2028-03-23	\$1,200	XXIX Metal Corp.	2% NSR (50% Buyback option)
2845612	XXIX	55.51	2028-03-23	\$1,200	XXIX Metal Corp.	2% NSR (50% Buyback option)
2845613	XXIX	55.50	2028-03-23	\$1,200	XXIX Metal Corp.	2% NSR (50% Buyback option)
2845614	XXIX	55.50	2028-03-23	\$1,200	XXIX Metal Corp.	2% NSR (50% Buyback option)
2845615	XXIX	55.50	2028-03-23	\$1,200	XXIX Metal Corp.	2% NSR (50% Buyback option)
2851925	XXIX	55.54	2028-06-29	\$1,200	XXIX Metal Corp.	2% NSR (50% Buyback option)
2851926	XXIX	55.54	2028-06-29	\$1,200	XXIX Metal Corp.	2% NSR (50% Buyback option)
2851927	XXIX	55.54	2028-06-29	\$1,200	XXIX Metal Corp.	2% NSR (50% Buyback option)
1042118	Cooke-Robitaille	74.40	20251210	\$2,500	2736-1179 Québec inc.	2% NSR (50% Buyback option)
1042119	Cooke-Robitaille	259.85	20251210	\$3,600	2736-1179 Québec inc.	2% NSR (50% Buyback option)

Title	Ownership	Area (Ha)	Expiry Date	Renewal Work Required	Registration	Royalties
1054	Cooke-Robitaille	55.60	20270724	\$2,500	2736-1179 Québec inc.	2% NSR (50% Buyback option)
1055	Cooke-Robitaille	55.60	20270724	\$2,500	2736-1179 Québec inc.	2% NSR (50% Buyback option)
1057	Cooke-Robitaille	55.60	20270724	\$2,500	2736-1179 Québec inc.	2% NSR (50% Buyback option)
1058	Cooke-Robitaille	55.59	20270724	\$2,500	2736-1179 Québec inc.	2% NSR (50% Buyback option)
1058017	Cooke-Robitaille	77.11	20260226	\$2,500	2736-1179 Québec inc.	2% NSR (50% Buyback option)
1059	Cooke-Robitaille	55.59	20270724	\$2,500	2736-1179 Québec inc.	2% NSR (50% Buyback option)
1061	Cooke-Robitaille	55.59	20270724	\$2,500	2736-1179 Québec inc.	2% NSR (50% Buyback option)
1101292	Cooke-Robitaille	42.44	2026-09-08	\$2,500	2736-1179 Québec inc.	2% NSR (50% Buyback option)
1101293	Cooke-Robitaille	55.58	2026-09-08	\$2,500	2736-1179 Québec inc.	2% NSR (50% Buyback option)
1101294	Cooke-Robitaille	27.45	2026-09-08	\$2,500	2736-1179 Québec inc.	2% NSR (50% Buyback option)
1101295	Cooke-Robitaille	55.57	2026-09-08	\$2,500	2736-1179 Québec inc.	2% NSR (50% Buyback option)
1101296	Cooke-Robitaille	55.57	2026-09-08	\$2,500	2736-1179 Québec inc.	2% NSR (50% Buyback option)
1101297	Cooke-Robitaille	48.32	2026-09-08	\$2,500	2736-1179 Québec inc.	2% NSR (50% Buyback option)
1101298	Cooke-Robitaille	44.90	2026-09-08	\$2,500	2736-1179 Québec inc.	2% NSR (50% Buyback option)
1101299	Cooke-Robitaille	34.64	2026-09-08	\$2,500	2736-1179 Québec inc.	2% NSR (50% Buyback option)
1101300	Cooke-Robitaille	40.15	2026-09-08	\$2,500	2736-1179 Québec inc.	2% NSR (50% Buyback option)
1101301	Cooke-Robitaille	44.42	2026-09-08	\$2,500	2736-1179 Québec inc.	2% NSR (50% Buyback option)
1101302	Cooke-Robitaille	53.95	2026-09-08	\$2,500	2736-1179 Québec inc.	2% NSR (50% Buyback option)
1101303	Cooke-Robitaille	55.56	2026-09-08	\$2,500	2736-1179 Québec inc.	2% NSR (50% Buyback option)
1101304	Cooke-Robitaille	55.56	2026-09-08	\$2,500	2736-1179 Québec inc.	2% NSR (50% Buyback option)
1101305	Cooke-Robitaille	55.56	2026-09-08	\$2,500	2736-1179 Québec inc.	2% NSR (50% Buyback option)
1104455	Cooke-Robitaille	44.67	2026-11-04	\$2,500	2736-1179 Québec inc.	2% NSR (50% Buyback option)
1104456	Cooke-Robitaille	55.59	2026-11-04	\$2,500	2736-1179 Québec inc.	2% NSR (50% Buyback option)
1104457	Cooke-Robitaille	24.68	2026-11-04	\$1,000	2736-1179 Québec inc.	2% NSR (50% Buyback option)
1104458	Cooke-Robitaille	20.85	2026-11-04	\$1,000	2736-1179 Québec inc.	2% NSR (50% Buyback option)
1105196	Cooke-Robitaille	55.56	2026-11-14	\$2,500	2736-1179 Québec inc.	2% NSR (50% Buyback option)
1105197	Cooke-Robitaille	55.56	2026-11-14	\$2,500	2736-1179 Québec inc.	2% NSR (50% Buyback option)
1105198	Cooke-Robitaille	55.56	2026-11-14	\$2,500	2736-1179 Québec inc.	2% NSR (50% Buyback option)
2025762	Cooke-Robitaille	55.55	2026-09-21	\$2,500	2736-1179 Québec inc.	2% NSR (50% Buyback option)
2025763	Cooke-Robitaille	55.55	2026-09-21	\$2,500	2736-1179 Québec inc.	2% NSR (50% Buyback option)
2025764	Cooke-Robitaille	55.55	2026-09-21	\$2,500	2736-1179 Québec inc.	2% NSR (50% Buyback option)
2025765	Cooke-Robitaille	55.55	2026-09-21	\$2,500	2736-1179 Québec inc.	2% NSR (50% Buyback option)
2025766	Cooke-Robitaille	55.54	2026-09-21	\$2,500	2736-1179 Québec inc.	2% NSR (50% Buyback option)
2025767	Cooke-Robitaille	55.54	2026-09-21	\$2,500	2736-1179 Québec inc.	2% NSR (50% Buyback option)
2025768	Cooke-Robitaille	55.54	2026-09-21	\$2,500	2736-1179 Québec inc.	2% NSR (50% Buyback option)
2025769	Cooke-Robitaille	55.53	2026-09-21	\$2,500	2736-1179 Québec inc.	2% NSR (50% Buyback option)
2025770	Cooke-Robitaille	55.53	2026-09-21	\$2,500	2736-1179 Québec inc.	2% NSR (50% Buyback option)
2025771	Cooke-Robitaille	55.53	2026-09-21	\$2,500	2736-1179 Québec inc.	2% NSR (50% Buyback option)
2025772	Cooke-Robitaille	55.53	2026-09-21	\$2,500	2736-1179 Québec inc.	2% NSR (50% Buyback option)
2025773	Cooke-Robitaille	55.54	2026-09-21	\$2,500	2736-1179 Québec inc.	2% NSR (50% Buyback option)
2025774	Cooke-Robitaille	55.54	2026-09-21	\$2,500	2736-1179 Québec inc.	2% NSR (50% Buyback option)
2025775	Cooke-Robitaille	55.54	2026-09-21	\$2,500	2736-1179 Québec inc.	2% NSR (50% Buyback option)
2030875	Cooke-Robitaille	55.56	2026-10-26	\$2,500	2736-1179 Québec inc.	2% NSR (50% Buyback option)
2030876	Cooke-Robitaille	55.55	2026-10-26	\$2,500	2736-1179 Québec inc.	2% NSR (50% Buyback option)
2030877	Cooke-Robitaille	55.55	2026-10-26	\$2,500	2736-1179 Québec inc.	2% NSR (50% Buyback option)
2113248	Cooke-Robitaille	55.61	2027-07-30	\$2,500	2736-1179 Québec inc.	2% NSR (50% Buyback option)
2113249	Cooke-Robitaille	55.61	2027-07-30	\$2,500	2736-1179 Québec inc.	2% NSR (50% Buyback option)
2113250	Cooke-Robitaille	55.61	2027-07-30	\$2,500	2736-1179 Québec inc.	2% NSR (50% Buyback option)
2113251	Cooke-Robitaille	55.61	2027-07-30	\$2,500	2736-1179 Québec inc.	2% NSR (50% Buyback option)

Title	Ownership	Area (Ha)	Expiry Date	Renewal Work Required	Registration	Royalties
2117008	Cooke-Robitaille	13.05	2027-08-12	\$1,000	2736-1179 Québec inc.	2% NSR (50% Buyback option)
2117009	Cooke-Robitaille	3.48	2027-08-12	\$1,000	2736-1179 Québec inc.	2% NSR (50% Buyback option)
2117861	Cooke-Robitaille	17.62	2027-08-15	\$1,000	2736-1179 Québec inc.	2% NSR (50% Buyback option)
2172401	Cooke-Robitaille	55.52	2026-10-02	\$2,500	2736-1179 Québec inc.	2% NSR (50% Buyback option)
2172402	Cooke-Robitaille	55.52	2026-10-02	\$2,500	2736-1179 Québec inc.	2% NSR (50% Buyback option)
2172407	Cooke-Robitaille	55.51	2026-10-02	\$2,500	2736-1179 Québec inc.	2% NSR (50% Buyback option)
2172408	Cooke-Robitaille	55.51	2026-10-02	\$2,500	2736-1179 Québec inc.	2% NSR (50% Buyback option)
2173577	Cooke-Robitaille	4.25	2026-11-04	\$1,000	2736-1179 Québec inc.	2% NSR (50% Buyback option)
2173578	Cooke-Robitaille	3.50	2026-11-04	\$1,000	2736-1179 Québec inc.	2% NSR (50% Buyback option)
2173579	Cooke-Robitaille	5.83	2026-11-04	\$1,000	2736-1179 Québec inc.	2% NSR (50% Buyback option)
2173580	Cooke-Robitaille	25.27	2026-11-04	\$2,500	2736-1179 Québec inc.	2% NSR (50% Buyback option)
2173581	Cooke-Robitaille	14.62	2027-07-24	\$1,000	2736-1179 Québec inc.	2% NSR (50% Buyback option)
2173582	Cooke-Robitaille	12.05	2027-07-24	\$1,000	2736-1179 Québec inc.	2% NSR (50% Buyback option)
2173583	Cooke-Robitaille	0.03	2026-09-08	\$1,000	2736-1179 Québec inc.	2% NSR (50% Buyback option)
2173584	Cooke-Robitaille	0.37	2026-09-08	\$1,000	2736-1179 Québec inc.	2% NSR (50% Buyback option)
2173585	Cooke-Robitaille	15.11	2026-09-08	\$1,000	2736-1179 Québec inc.	2% NSR (50% Buyback option)
2247822	Cooke-Robitaille	55.54	2026-08-26	\$2,500	2736-1179 Québec inc.	2% NSR (50% Buyback option)
2247823	Cooke-Robitaille	55.53	2026-08-26	\$2,500	2736-1179 Québec inc.	2% NSR (50% Buyback option)
2247824	Cooke-Robitaille	55.53	2026-08-26	\$2,500	2736-1179 Québec inc.	2% NSR (50% Buyback option)
2247825	Cooke-Robitaille	55.53	2026-08-26	\$2,500	2736-1179 Québec inc.	2% NSR (50% Buyback option)
2247826	Cooke-Robitaille	55.53	2026-08-26	\$2,500	2736-1179 Québec inc.	2% NSR (50% Buyback option)
2247827	Cooke-Robitaille	38.99	2026-08-26	\$2,500	2736-1179 Québec inc.	2% NSR (50% Buyback option)
2247828	Cooke-Robitaille	38.98	2026-08-26	\$2,500	2736-1179 Québec inc.	2% NSR (50% Buyback option)
2247829	Cooke-Robitaille	38.96	2026-08-26	\$2,500	2736-1179 Québec inc.	2% NSR (50% Buyback option)
2247830	Cooke-Robitaille	38.94	2026-08-26	\$2,500	2736-1179 Québec inc.	2% NSR (50% Buyback option)
2367323	Cooke-Robitaille	55.59	2026-10-15	\$2,500	2736-1179 Québec inc.	2% NSR (50% Buyback option)
2367324	Cooke-Robitaille	55.59	2026-10-15	\$2,500	2736-1179 Québec inc.	2% NSR (50% Buyback option)
2367325	Cooke-Robitaille	55.58	2026-10-15	\$2,500	2736-1179 Québec inc.	2% NSR (50% Buyback option)
2367326	Cooke-Robitaille	55.58	2026-10-15	\$2,500	2736-1179 Québec inc.	2% NSR (50% Buyback option)
2367327	Cooke-Robitaille	55.57	2026-10-15	\$2,500	2736-1179 Québec inc.	2% NSR (50% Buyback option)
2367328	Cooke-Robitaille	55.57	2026-10-15	\$2,500	2736-1179 Québec inc.	2% NSR (50% Buyback option)
2367329	Cooke-Robitaille	24.55	2026-10-15	\$1,000	2736-1179 Québec inc.	2% NSR (50% Buyback option)
2367330	Cooke-Robitaille	24.97	2026-10-15	\$1,000	2736-1179 Québec inc.	2% NSR (50% Buyback option)
2367331	Cooke-Robitaille	21.70	2026-10-15	\$1,000	2736-1179 Québec inc.	2% NSR (50% Buyback option)
2390109	Cooke-Robitaille	55.61	2027-09-05	\$1,800	2736-1179 Québec inc.	2% NSR (50% Buyback option)
2390110	Cooke-Robitaille	55.61	2027-09-05	\$1,800	2736-1179 Québec inc.	2% NSR (50% Buyback option)
2390111	Cooke-Robitaille	55.60	2027-09-05	\$1,800	2736-1179 Québec inc.	2% NSR (50% Buyback option)
2390112	Cooke-Robitaille	55.60	2027-09-05	\$1,800	2736-1179 Québec inc.	2% NSR (50% Buyback option)
2390113	Cooke-Robitaille	55.60	2027-09-05	\$1,800	2736-1179 Québec inc.	2% NSR (50% Buyback option)
2390114	Cooke-Robitaille	55.60	2027-09-05	\$1,800	2736-1179 Québec inc.	2% NSR (50% Buyback option)
2435594	Cooke-Robitaille	55.59	2026-01-06	\$1,800	2736-1179 Québec inc.	2% NSR (50% Buyback option)
2435595	Cooke-Robitaille	55.59	2026-01-06	\$1,800	2736-1179 Québec inc.	2% NSR (50% Buyback option)
2435596	Cooke-Robitaille	55.59	2026-01-06	\$1,800	2736-1179 Québec inc.	2% NSR (50% Buyback option)
2435597	Cooke-Robitaille	55.58	2026-01-06	\$1,800	2736-1179 Québec inc.	2% NSR (50% Buyback option)
2435598	Cooke-Robitaille	55.58	2026-01-06	\$1,800	2736-1179 Québec inc.	2% NSR (50% Buyback option)
2435599	Cooke-Robitaille	55.58	2026-01-06	\$1,800	2736-1179 Québec inc.	2% NSR (50% Buyback option)
2435600	Cooke-Robitaille	55.57	2026-01-06	\$1,800	2736-1179 Québec inc.	2% NSR (50% Buyback option)
2435601	Cooke-Robitaille	55.55	2026-01-06	\$1,800	2736-1179 Québec inc.	2% NSR (50% Buyback option)
2435602	Cooke-Robitaille	55.55	2026-01-06	\$1,800	2736-1179 Québec inc.	2% NSR (50% Buyback option)

Title	Ownership	Area (Ha)	Expiry Date	Renewal Work Required	Registration	Royalties
2435603	Cooke-Robitaille	55.55	2026-01-06	\$1,800	2736-1179 Québec inc.	2% NSR (50% Buyback option)
2435604	Cooke-Robitaille	55.55	2026-01-06	\$1,800	2736-1179 Québec inc.	2% NSR (50% Buyback option)
2435605	Cooke-Robitaille	55.55	2026-01-06	\$1,800	2736-1179 Québec inc.	2% NSR (50% Buyback option)
2435606	Cooke-Robitaille	55.53	2026-01-06	\$1,800	2736-1179 Québec inc.	2% NSR (50% Buyback option)
2435607	Cooke-Robitaille	55.51	2026-01-06	\$1,800	2736-1179 Québec inc.	2% NSR (50% Buyback option)
2436533	Cooke-Robitaille	55.50	2026-01-27	\$1,800	2736-1179 Québec inc.	2% NSR (50% Buyback option)
2436534	Cooke-Robitaille	55.50	2026-01-27	\$1,800	2736-1179 Québec inc.	2% NSR (50% Buyback option)
2436535	Cooke-Robitaille	55.50	2026-01-27	\$1,800	2736-1179 Québec inc.	2% NSR (50% Buyback option)
2438494	Cooke-Robitaille	55.60	2026-03-20	\$1,800	2736-1179 Québec inc.	2% NSR (50% Buyback option)
2438495	Cooke-Robitaille	55.60	2026-03-20	\$1,800	2736-1179 Québec inc.	2% NSR (50% Buyback option)
2438497	Cooke-Robitaille	55.58	2026-03-20	\$1,800	2736-1179 Québec inc.	2% NSR (50% Buyback option)
2438499	Cooke-Robitaille	55.57	2026-03-20	\$1,800	2736-1179 Québec inc.	2% NSR (50% Buyback option)
2438500	Cooke-Robitaille	55.57	2026-03-20	\$1,800	2736-1179 Québec inc.	2% NSR (50% Buyback option)
2438501	Cooke-Robitaille	55.57	2026-03-20	\$1,800	2736-1179 Québec inc.	2% NSR (50% Buyback option)
2438502	Cooke-Robitaille	55.57	2026-03-20	\$1,800	2736-1179 Québec inc.	2% NSR (50% Buyback option)
2438503	Cooke-Robitaille	55.56	2026-03-20	\$1,800	2736-1179 Québec inc.	2% NSR (50% Buyback option)
2438504	Cooke-Robitaille	55.56	2026-03-20	\$1,800	2736-1179 Québec inc.	2% NSR (50% Buyback option)
2438505	Cooke-Robitaille	55.55	2026-03-20	\$1,800	2736-1179 Québec inc.	2% NSR (50% Buyback option)
2438506	Cooke-Robitaille	55.55	2026-03-20	\$1,800	2736-1179 Québec inc.	2% NSR (50% Buyback option)
2465830	Cooke-Robitaille	55.56	2026-10-12	\$1,800	2736-1179 Québec inc.	2% NSR (50% Buyback option)
2517114	Cooke-Robitaille	55.50	2026-04-29	\$1,200	2736-1179 Québec inc.	2% NSR (50% Buyback option)
2517115	Cooke-Robitaille	55.50	2026-04-29	\$1,200	2736-1179 Québec inc.	2% NSR (50% Buyback option)
2517116	Cooke-Robitaille	55.49	2026-04-29	\$1,200	2736-1179 Québec inc.	2% NSR (50% Buyback option)
2517117	Cooke-Robitaille	55.49	2026-04-29	\$1,200	2736-1179 Québec inc.	2% NSR (50% Buyback option)
2517118	Cooke-Robitaille	55.49	2026-04-29	\$1,200	2736-1179 Québec inc.	2% NSR (50% Buyback option)
2542424	Cooke-Robitaille	55.61	2026-08-21	\$1,200	2736-1179 Québec inc.	2% NSR (50% Buyback option)
2542425	Cooke-Robitaille	55.61	2026-08-21	\$1,200	2736-1179 Québec inc.	2% NSR (50% Buyback option)
2542426	Cooke-Robitaille	55.61	2026-08-21	\$1,200	2736-1179 Québec inc.	2% NSR (50% Buyback option)
2542427	Cooke-Robitaille	55.61	2026-08-21	\$1,200	2736-1179 Québec inc.	2% NSR (50% Buyback option)
2542428	Cooke-Robitaille	55.61	2026-08-21	\$1,200	2736-1179 Québec inc.	2% NSR (50% Buyback option)
2542429	Cooke-Robitaille	55.61	2026-08-21	\$1,200	2736-1179 Québec inc.	2% NSR (50% Buyback option)
2542430	Cooke-Robitaille	55.60	2026-08-21	\$1,200	2736-1179 Québec inc.	2% NSR (50% Buyback option)
2542431	Cooke-Robitaille	55.60	2026-08-21	\$1,200	2736-1179 Québec inc.	2% NSR (50% Buyback option)
3816	Cooke-Robitaille	55.60	2027-09-17	\$2,500	2736-1179 Québec inc.	2% NSR (50% Buyback option)
3817	Cooke-Robitaille	55.59	2027-09-17	\$2,500	2736-1179 Québec inc.	2% NSR (50% Buyback option)
39501	Cooke-Robitaille	55.54	2026-09-22	\$2,500	2736-1179 Québec inc.	2% NSR (50% Buyback option)
39502	Cooke-Robitaille	55.54	2026-09-22	\$2,500	2736-1179 Québec inc.	2% NSR (50% Buyback option)
39503	Cooke-Robitaille	55.54	2026-09-22	\$2,500	2736-1179 Québec inc.	2% NSR (50% Buyback option)
39504	Cooke-Robitaille	55.54	2026-09-22	\$2,500	2736-1179 Québec inc.	2% NSR (50% Buyback option)
39505	Cooke-Robitaille	55.54	2026-09-22	\$2,500	2736-1179 Québec inc.	2% NSR (50% Buyback option)
39506	Cooke-Robitaille	55.53	2026-09-22	\$2,500	2736-1179 Québec inc.	2% NSR (50% Buyback option)
39507	Cooke-Robitaille	55.53	2026-09-22	\$2,500	2736-1179 Québec inc.	2% NSR (50% Buyback option)
39508	Cooke-Robitaille	55.52	2026-09-22	\$2,500	2736-1179 Québec inc.	2% NSR (50% Buyback option)
39509	Cooke-Robitaille	55.52	2026-09-22	\$2,500	2736-1179 Québec inc.	2% NSR (50% Buyback option)
85420	Cooke-Robitaille	55.61	2027-07-11	\$2,500	2736-1179 Québec inc.	2% NSR (50% Buyback option)
85421	Cooke-Robitaille	55.61	2027-07-11	\$2,500	2736-1179 Québec inc.	2% NSR (50% Buyback option)
85422	Cooke-Robitaille	55.61	2027-07-11	\$2,500	2736-1179 Québec inc.	2% NSR (50% Buyback option)
86116	Cooke-Robitaille	55.61	2027-07-14	\$2,500	2736-1179 Québec inc.	2% NSR (50% Buyback option)
86117	Cooke-Robitaille	55.61	2027-07-14	\$2,500	2736-1179 Québec inc.	2% NSR (50% Buyback option)

Title	Ownership	Area (Ha)	Expiry Date	Renewal Work Required	Registration	Royalties
86118	Cooke-Robitaille	55.61	2027-07-14	\$2,500	2736-1179 Québec inc.	2% NSR (50% Buyback option)
86119	Cooke-Robitaille	55.61	2027-07-14	\$2,500	2736-1179 Québec inc.	2% NSR (50% Buyback option)
86120	Cooke-Robitaille	55.61	2027-07-14	\$2,500	2736-1179 Québec inc.	2% NSR (50% Buyback option)
86253	Cooke-Robitaille	54.80	2027-07-11	\$2,500	2736-1179 Québec inc.	2% NSR (50% Buyback option)
86254	Cooke-Robitaille	54.76	2027-07-11	\$2,500	2736-1179 Québec inc.	2% NSR (50% Buyback option)
86255	Cooke-Robitaille	55.60	2027-07-11	\$2,500	2736-1179 Québec inc.	2% NSR (50% Buyback option)
93913	Cooke-Robitaille	55.56	2027-09-13	\$2,500	2736-1179 Québec inc.	2% NSR (50% Buyback option)
93914	Cooke-Robitaille	55.56	2027-09-13	\$2,500	2736-1179 Québec inc.	2% NSR (50% Buyback option)
93915	Cooke-Robitaille	55.55	2027-09-13	\$2,500	2736-1179 Québec inc.	2% NSR (50% Buyback option)
93916	Cooke-Robitaille	55.55	2027-09-13	\$2,500	2736-1179 Québec inc.	2% NSR (50% Buyback option)
93917	Cooke-Robitaille	55.55	2027-09-13	\$2,500	2736-1179 Québec inc.	2% NSR (50% Buyback option)
93918	Cooke-Robitaille	55.55	2027-09-13	\$2,500	2736-1179 Québec inc.	2% NSR (50% Buyback option)
93919	Cooke-Robitaille	55.54	2027-09-13	\$2,500	2736-1179 Québec inc.	2% NSR (50% Buyback option)
93920	Cooke-Robitaille	55.54	2027-09-13	\$2,500	2736-1179 Québec inc.	2% NSR (50% Buyback option)
93921	Cooke-Robitaille	55.54	2027-09-13	\$2,500	2736-1179 Québec inc.	2% NSR (50% Buyback option)
93922	Cooke-Robitaille	55.53	2027-09-13	\$2,500	2736-1179 Québec inc.	2% NSR (50% Buyback option)
93923	Cooke-Robitaille	55.53	2027-09-13	\$2,500	2736-1179 Québec inc.	2% NSR (50% Buyback option)
93924	Cooke-Robitaille	55.56	2027-09-13	\$2,500	2736-1179 Québec inc.	2% NSR (50% Buyback option)
93925	Cooke-Robitaille	55.56	2027-09-13	\$2,500	2736-1179 Québec inc.	2% NSR (50% Buyback option)
93926	Cooke-Robitaille	55.56	2027-09-13	\$2,500	2736-1179 Québec inc.	2% NSR (50% Buyback option)
93927	Cooke-Robitaille	55.56	2027-09-13	\$2,500	2736-1179 Québec inc.	2% NSR (50% Buyback option)
93928	Cooke-Robitaille	55.55	2027-09-13	\$2,500	2736-1179 Québec inc.	2% NSR (50% Buyback option)
93929	Cooke-Robitaille	55.55	2027-09-13	\$2,500	2736-1179 Québec inc.	2% NSR (50% Buyback option)
93930	Cooke-Robitaille	55.55	2027-09-13	\$2,500	2736-1179 Québec inc.	2% NSR (50% Buyback option)
93931	Cooke-Robitaille	55.55	2027-09-13	\$2,500	2736-1179 Québec inc.	2% NSR (50% Buyback option)
93932	Cooke-Robitaille	55.54	2027-09-13	\$2,500	2736-1179 Québec inc.	2% NSR (50% Buyback option)

4.3 Royalties and Encumbrances

XXIX fulfilled all its obligations under the terms of the option agreement with Ex-In on June 16, 2023, and executed the purchase agreement of 11 mining titles. As a result, these claims have now been transferred to and are 100% owned by XXIX, subject to a 2% NSR royalty, 50% of which can be purchased by XXIX for \$4.5 million. These mining titles are referred to as the XXIX mining titles in Figure 4-2. Payments to keep the claims are in good standing as of the effective date of the report.

XXIX is in an earn-in process to acquire 175 additional mining titles subject to a 2% NSR royalty, 50% of which can be purchased by XXIX before the commencement of commercial production for \$1.5 million. These mining titles are referred to as the Cooke-Robitaille Option in Figure 4-2.

4.4 Environmental Considerations

The project is located on a previously disturbed mine site. The mine included the historical underground workings, waste rock stockpiles, tailings disposal areas, the process plant site, and haulage and service roads. The project will occupy a portion of these previously disturbed areas.

All the historical mining and processing infrastructure has been dismantled and the mining operation has been decommissioned since 1991. XXIX does not have any responsibility for environmental matters arising from the historical mining and processing operations.

To the extent known, and apart from the encumbrances noted above, the Authors are not aware of any other significant factors or risks that may affect access, title, or right or ability to perform work on the Opémiska property.

The geological and geotechnical field programs recommended in Chapter 26 will require a forestry work permit and potentially other permits from the Quebec government; XXIX is actively engaged in planning for these permits.

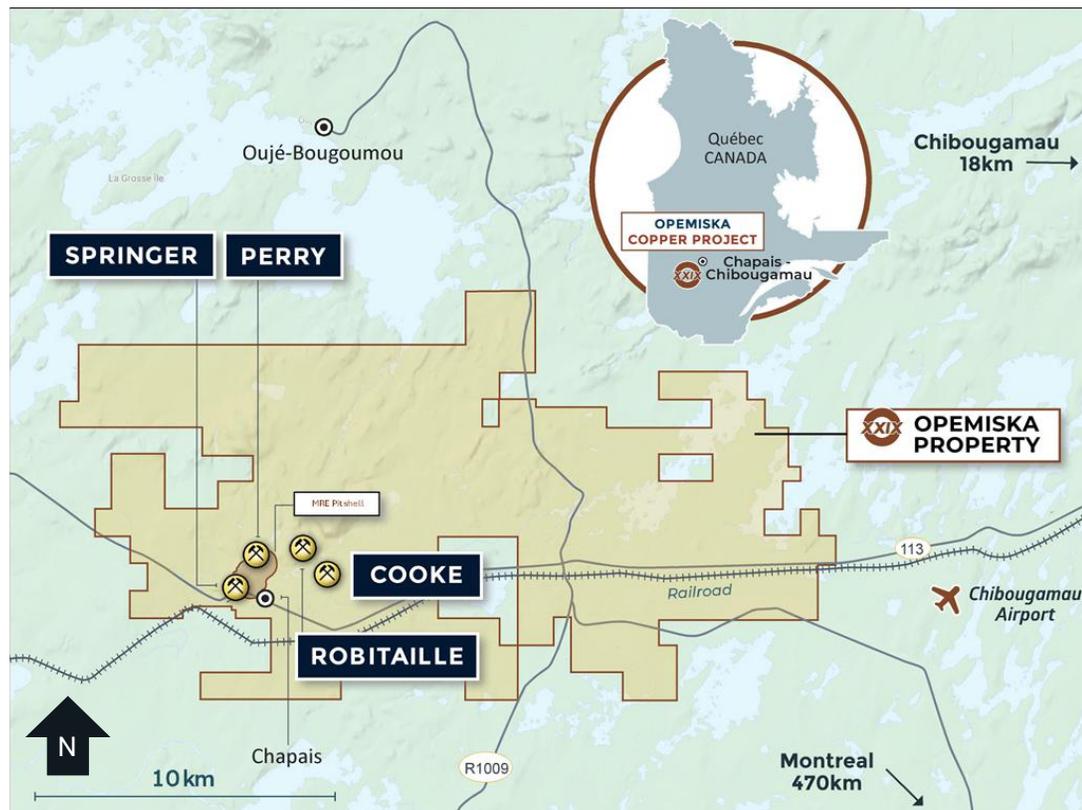
5 ACCESSIBILITY, CLIMATE, LOCAL RESOURCES, INFRASTRUCTURE AND PHYSIOGRAPHY

5.1 Accessibility

The Opémiska Project is located immediately adjacent to the municipality of Chapais and is accessible by road with paved Québec Highway 113 crossing the property (Figure 5-1). The project is located 40 km west of the town of Chibougamau, which straddles Highway 167. Highway 113 connects Chapais to the Abitibi area and Highway 167 heads south to the Saguenay - Lac St-Jean area. These all-weather paved highways are maintained year-round. The project itself is readily accessed via forestry roads and historical mine access roads.

The Chapais-Chibougamau area is serviced by the Chibougamau-Chapais Airport, located 20 km southwest of Chibougamau along Highway 113. Regularly scheduled direct flights depart from the airport three times per week to Montreal, Québec.

Figure 5-1: : Access to the Opémiska Project



Source: XXIX, 2025.

5.2 Climate

The area has a humid sub-arctic continental climate with cool summers and cold winters. Climate conditions are typical of the Canadian Shield. The temperature varies from an average minimum of -26°C in winter (January and February) to an average maximum of 22°C in the summer (July and August). Extreme temperatures below -36°C or above 27°C can be expected. Rainfall is generally common in the summer, such that there is no dry season. Snowfall is common in the winter, particularly in the early and latter part of the season. The “warm” season generally is from mid-May to mid-September. The “cold” season is from early December to early March. Exploration work can be carried out year-round.

The climatic conditions do not significantly impede the project as they do not hinder exploration or mining activities.

5.3 Local Resources and Infrastructure

5.3.1 Historical Mine Infrastructure

In addition to paved highway access, the project includes other infrastructure such as roads, buildings, transmission lines, and a past-producing underground mine.

Some infrastructure for exploration and mining operations is readily available in the project area.

5.3.2 Local Workforce

A highly specialized work force resides in the project region and within the Abitibi region. The successful mining history of Chapais–Chibougamau over the past 60 years resulted in the establishment of very experienced mining workforce and the full range of associated secondary tradesmen.

The Chapais–Chibougamau area is an active mining and forestry centre with a total population of approximately 10,000 residents (Census Canada, 2016).

5.3.3 Additional Services

Chapais has a post office, school, Centre Local de Services Communautaires (CLSC), health centre, and tourist office. Basic commercial infrastructure includes two motels, grocery store, hardware store, restaurants, and gas station.

The nearest full-service hospital is located in the town of Chibougamau.

Power and water are available at the mine site, and local housing is available in the town.

5.4 Physiography

The physiography of the general area is one of rolling hills and abundant lakes and rivers. Forest cover is variable, in part because selected areas have been harvested. Some areas are still forested with tall spruce, jack pine, birch and poplar.

The overburden cover generally consists of sand, clay and boulders, varying in thickness from 1 m to locally more than 80 m along major regional faults. There are few bedrock exposures and widespread swampy areas are found within moderately to locally densely-forested sectors.

The elevation of the lakes in the general area is approximately 390 m above mean sea level (masl). The general elevation averages approximately 400 masl, except for Mount Springer northeast of the project, which reaches an elevation of 540 masl.

Drainage in the project area is westward through the Waswanipi and Nottaway Rivers and ultimately into James Bay.

6 HISTORY

6.1 Summary of Property History

There have been three main periods of historical exploration and mining activities on the Opémiska property and surrounding area: (1) discovery and early exploration (1929 to 1953); (2) mine production (1953 to 1991); (3) recent exploration (1993 to 2016).

The activities undertaken in each of the three periods are summarized below. This section is inspired by previous reports from XXIX, in particular, the report by Yassa et Puritch (2024) which summarizes internal XXIX reports.

6.2 Discovery and Early Exploration (1929 to 1953)

The information in this section is summarized mainly from the documents listed in Table 6-1. The documents with a “GM” prefix refer to numbers in the Québec SIGEOM geoscientific archive of historical assessment work. They can be searched online and downloaded free of charge and are all georeferenced on government compilation maps.

Table 6-1: Historical Reports for the 1929-1953 Period

Assessment Report ID (Year)	Work Performed By	Work Summary
GM-03556 (1929)	Retty, J.A.	Geological report by the MRN Claims Springer. The report describes the "Lake Opémiska Copper showing" which was visited in 1929.
GM-03558 (1933)	Opémiska Copper Mines Ltd. Huston, M.B. Energy Mines and Resources Canada.	Geological report with technical evaluation, map showing original drilling (+ composites) and also trenches with assays.
GM-03559 (1935)	Opémiska Copper Mines Ltd. Taschereau, R.H.	Information report.
GM-01833 (1952)	Opémiska Copper Mines Ltd. Derry D.R.	Interim report on geology and diamond drilling results.
GM-02005 (1951)	Graham R.B. Evaluation Technique.	Summary of exploration and development activities
GM-02098 (1952)	Thompson J.M. for Opémiska Copper Mines (Que.)	Report on Opémiska copper mines.

Initial exploration between 1929 and 1953 predated mining operations. Within the area, a preliminary phase of surface exploration and discoveries occurred on the Opémiska property and surrounding area following the discovery by Mr. Leo Springer in 1929 at what would become the Springer mine.

The discovery by Mr. Springer of the Springer Syndicate, was assisted by Mr. Lloyd Rochester, a pilot of Prospectors Airways. The showing lies on high outcrops that were visible as the area was burned over by forest fires at the time. The chalcopryrite discovery was hosted in a gabbro dyke. The dimensions of the mineralized area were 1,200 ft (360 m) long and 800 ft (240 m) wide in a north-south direction.

The first development work on the project was completed in 1935 (GM 02098) and consisted of trenching and diamond drilling. Underground development was undertaken in 1936. A three-compartment shaft was sunk to 168 m (550 ft) and extensive lateral work and underground drilling was carried out on the 46, 84 and 152 m (150, 275 and 500 ft) levels. Work was suspended in 1937 due to low metal prices.

In 1951, a decision was made to re-open the mine and place it in production at an initial processing rate of 400 tons per day. This decision was facilitated by the completion of the new highway connecting Chibougamau to St-Felicien, which allowed the development of the mining industry in the Chapais–Chibougamau district to proceed. Along with new construction, the old buildings were rehabilitated, including a new concrete shaft collar. A total of 6,100 m (20,000 ft) of exploratory surface drilling was completed in 1952.

6.3 Mine Production (1953-1991)

The Chapais–Chibougamau mining district is the second largest of its kind in the Québec part of the Abitibi Greenstone Belt. From 1953 to 2008, the district produced approximately 86 Mt of mineralized material, including 1.57 Mt Cu, 176.1 t Au, 108.8 t Ag, and 72,066 t Zn (Leclerc et al., 2012).

Opémiska Copper Mines were in production from December 1953 until June 1991. Total production from the Springer and Perry mines was 22.0 Mt grading 2.40% Cu, 0.29 g/t Au, and 0.21 g/t Ag containing 527 kt Cu, 6,400 kg Au, and 4,600 kg Ag (Salmon, 2013; Salmon, 2014).

Production came from seven easterly-trending mineralized zones; specifically, the No. 1, 2, 3 (or Main Zone), 4, 5 and 6 Zones. The mineralized zones were described as being sharp-walled, except for No. 3 or the “Main Zone,” which is hosted by a shear or fault zone that contains a breccia-type mineralization with altered gabbro remnants set in a sulphide (mainly chalcopyrite) matrix.

Detailed drilling in the spring of 1956 outlined an important deposit in the Perry Zone area. The outlined deposit strikes 330° and dips 56° to the north. A fault that strikes 130° and dips to the southwest lies from 15 to 120 m southwest of the Perry Zone. The mineralization in the Perry Zone is described as heavy impregnation of sulphides in the host rocks with some massive sulphides. The alteration is partial chloritization. The sulphides present are mainly chalcopyrite and pyrite with some pyrrhotite and arsenopyrite. Quartz vein sections containing sulphides are common in the mineralized horizon.

In 1980, the original developer of the mining claims, changed its name to Corporation Falconbridge Copper (Falconbridge), and again in 1987 to Minnova Inc. (Minnova). In October 1986, an agreement between Minnova and the Québec Ministry of Energy and Resources led to an exploration program at the Springer and Perry mines, which were part of the Minnova Opémiska division.

In 1987, a low copper price meant that the secondary products gold and silver became metals of interest for exploration. Between the discovery of Springer in 1932 and closure of that mine in 1991, 612 surface and 15,287 underground diamond drill holes totalling 82,767 and 861,542 m, respectively, were completed on the Opémiska property by Falconbridge and Minnova.

Table 6-2 summarizes historical reports available.

Table 6-2: Historical Reports for the 1953-1991 period

Assessment File ID	Summary
GM-2700 (1954)	Information report. Cornwall, F.W. for the MRN. Opémiska Copper Mines Ltd.
GM-04273 (1956)	Information report. Opémiska Copper Mines Ltd. Assad, J.R. MRN Sketch Map with Campbell Lake Fault
GM-46158 (1987)	Rapport Géologique ed la Partie Nord Ouest de la Propriété Bourbeau West. Cormier J.M. Minnova Inc.
GM-87-03 (1989)	Etude métallogénique (aurifère) du Filon Couche de Bourbeau (région de Chibougamau). MRN. Dubé B., Guha J.
GM-049654 (1990)	Rapport des travaux d'exploration effectués entre le 1 ^{er} Septembre 1986 et le 31 Mars 1987 sur propriétés minières de Minnova Inc., Division Opémiska. canton Levr. Doiron G., géologue de projet. 30 Avril 1987 (numerous maps are appended to the report sections, level plans, drifts, and longitudinal sections detailing Veine 10-2S, No. 4, No. 5, No. 6 at Springer and Vein A at Perry beside work carried at adjacent Cooke Mine).

6.4 Property Exploration History (1993-2016)

Ex-In acquired the Springer and Perry mines in 1995. Table 6-3 summarizes historical reports available.

Assessment File ID (Year)	Summary
GM-55059 (1994)	Géologie et Levé au BEEP MAT effectué sur la Propriété OPEMISCA. E. Gaucher. GEOSIG Inc.
MM 91-02 (1994)	Géologie et compilation géologique de la région de Chapais. Morin R., Ressources Naturelles du Canada & Ministère des Ressources Naturelles du Québec.
DV 98-03 (1998)	Géologie et Métallogénie du District Minier de Chapais-Chibougamau. Ministère Richesses Naturelles (MRN). Nouvelle Vision du Potential de Découverte. Éditeur: Pierre Pilote.
MB 98-06 (1998)	Compilation et Répartition des Gisements Polymétalliques à Tonnage évalué dans la Sous-Province de l'Abitibi. Lacroix, S. Gouvernement du Québec, Ministère des Ressources Naturelles. Secteur Mines.
GM-60142 (2001)	Atlas des Gisements Abitibi, Fiche No 182. Springer. CONSOREM. Faure S., Gaboury D.
GM-60258 (2001)	Rentabilité de l'exploitation des piliers de surface, Projet Mine Opémiska. E. Gaucher.
GM-60259 (2001)	Métallurgie des rejets du moulin, projet Mine Opémiska. E. Gaucher, A. Laplante.
GM-60262 (2001)	Plan d'affaire d'Ex-In Inc. sur Opémiska. Gaucher E., Gaucher P.
GM-60257 (2002)	Évaluation des Ressources en Cuivre et en Or exploitables à partir de la surface, Localisation des sites prioritaires à investiguer, Projet EX-07C, Mine Opémiska. Gaucher E.
GM-60260 (2002)	Digitalisation des forages, Mine Opémiska.
GM-60261 (2002)	Validation des Ressources de Minerais exploitables à ciel ouvert, phase 2 révisée. Mine Opémiska. Gaucher E.
GM-63383 (2007)	Campagne de forage, secteur de la Mine Opémiska, Projet EX-07C. hiver 2005-2006, St-Pierre R. & Gaucher E.
GM-64969 (2009)	Rapport d'un levé de Polarisation Provoquée effectué sur la propriété Opémiska. Hubert, J.M. Explorateurs-Innovateurs de Québec Inc.
GM-64968 (2010)	Campagne d'Exploration 2009, Propriété Opémiska. Explorateurs-Innovateurs de Québec Inc. (Ex-In). Gaucher, E. & Pearson, N.
GM-65209 (2010)	Travaux de terrain 2009, Propriété Opémiska. Ex-In. Gaucher E., Pearson N.
GM-65737 (2010)	Levé de Polarisation Provoquée, propriété Opémiska (EX-07C) Block Nord. GEOSIG.
GM-65965 (2011)	Campagne d'Exploration 2010, Propriété Opémiska. Explorateurs-Innovateurs de Québec Inc. Drilling. Gaucher E., Pearson N., and Kongo J.B.
RP-2010-09A (2011)	Geology of the Chapais area (32G15-200-0101). Compilation, Geological Survey. MRNF. Leclerc F., Houle P., Rogers R.
RP-2013-02A (2014)	Geology of the Lac Simon Region (32G15-200-0102). Compilation, Geological Survey. MRNF. Leclerc F., Houle P.
GM-69674 (2016)	Campagne d'exploration 2015, Propriété Opémiska. Gaucher, F. & Gaucher P. Explorateurs-Innovateurs de Québec Inc.
GM-70399 (2016)	Report on the limited core drilling campaign completed December 2016 on the Opémiska mining property. Larouche, C. for Explorateurs-Innovateurs de Québec Inc.

6.4.1 Surface Work

Exploration work was carried out in the southwest quadrant of the Opémiska Project. Compilation, line-cutting (4.5 km), stripping, sampling, metallurgical testing, and induced polarization (IP) surveys were completed. An Ex-In report also records that in 1995, a sample weighing 15.5 t was extracted from a surface vein at Opémiska to test the recovery of surface pillars. The results were reported as being disappointing.

In 1998, Ex-In carried out an experimental gravimetric survey.

In 2000, Ex-In started a prefeasibility study to test the possibility of mining lower-grade material left behind at the closure of the mines.

In 2003, Beep-Mat prospecting was completed, along with stripping and trenching.

In 2004, a line grid was cut to guide a max-min survey. Additional sampling was conducted.

In 2005, magnetic separation tests were completed.

In 2006, a second core drilling program of 1,000 m was initiated on five separate veins to test for the presence of mineralization close to surface.

In 2009, Ex-In discovered a boulder carrying high-grade gold south of the property. The Company went back to old surface and underground maps to find a possible source for the high-grade boulder. It is reported that at levels 200 m and 400 m two zones drilled systematically at 15 m, have been previously investigated for gold. One zone is located north of the Springer No. 1 shaft and the other one is to the south of the shaft. The report also notes that certain drill holes confirm the presence of 150 m wide sections grading > 0.5% Cu and 0.3 g/t Au.

A trench 200 m long x 3.0 m wide oriented north-south, perpendicular to the mineralized structures, was completed in 2009. The overburden thickness ranged from 0.5 m to 5.0 m locally. Sampling was completed by blasting every 2.5 m along the trench. The trench exposed three separate mineralized zones. The most northerly zone corresponds to the No. 3 Vein, just east of the Glory Hole, an average value of 2.15% Cu and 0.53 g/t Au was calculated over a width of 14.55 m. This zone is in an area of previous surface drilling by Falconbridge with drill holes S-140, S-141, S-148, S-149 and S-150. The second zone of interest graded 2.99% Cu and 1.06 g/t Au over a width of 12.55 m. On the sketch provided by Ex-In (2002), this second zone of mineralization appears to be located approximately 60 m southwest of Vein No. 3 and would correspond to the “Vein 3 South” projected at surface (previous drilling is also located in this area). The third zone intersected lies due south of the previously noted zone, approximately 100 m south of No. 3 Vein South, and returned values of 0.65% Cu and 0.83 g/t Au over a width of 21.5 m on top of a recently located IP anomaly. This third zone would fall in the western extension of the No. 13, 4 and 5 Zones. Drill hole S-853 was also completed in this area. A map accompanying Ex-In (2002) shows the location of two ventilation raises. Note that the samples were collected after blasting and, therefore, that such sampling is equivalent to grab samples.

Channel sampling was also completed on Vein No. 2, south-southeast of Springer shaft no. 1. Good results were returned from the sampling, a length of 75 m was sampled every 5.0 m. A table in Ex-In (2002) summarized the results for 29 samples. The copper values are up to 26.0% Cu and the gold values are up to 11.11 g/t Au. Individual widths were not given within, except a note that the vein sampled averages 0.45 m, locally 1.0 m wide. This stripping and

sampling location was the site of the 2006 surface drilling by Ex-In. A rapid survey of the data acquired on this vein does not show a direct correlation between the higher values in copper and gold.

6.4.2 Drilling

Exploration diamond drilling campaigns were completed by Ex-In in 2006 (46 holes; 970 m), 2010 (19 holes; 1,748 m), 2015 (four holes; 537 m) and 2016 (nine holes; 708 m). The drill core size varies from BQ (earlier drilling) to NQ (recent drilling). The drill core was logged with the sampling focused on the main mineralized veins, whereas most of the remaining drill core was assayed in 3.0 to 6.0 m sections. The assaying was carried out systematically for copper (with few duplicate samples). Assaying for gold, silver and zinc has not been carried out systematically.

A report also states that in 2002, a 100 m core drilling program was completed, to test a surface vein.

7 GEOLOGICAL SETTING AND MINERALIZATION

7.1 Regional Geology

The project area is located within the Superior Structural Province of the Canadian Shield, which is present in eastern Canada and the northeastern USA. The Precambrian rock units are generally covered by glacial overburden.

The Chapais-Chibougamau Mining District (Figure 7-1) is located in the northeast part of the Abitibi Subprovince. The Abitibi Subprovince is one of the world's largest contiguous areas of Archean metavolcanic and metasedimentary rocks and hosts many significant mineral deposits (Leclerc et al., 2010, 2012). The general lithological distribution is characterized by oval-shaped granitoid batholiths surrounded by east-to-west trending "greenstone belts" that appear to wrap around and enclose the batholiths. Regional and local folding is common, and the dips of the rock units are generally sub-vertical. The region under study is located within the Northern Volcanic Zone of the Abitibi Subprovince (Guha et al., 1988; Dube and Guha, 1992).

7.2 Project Geology

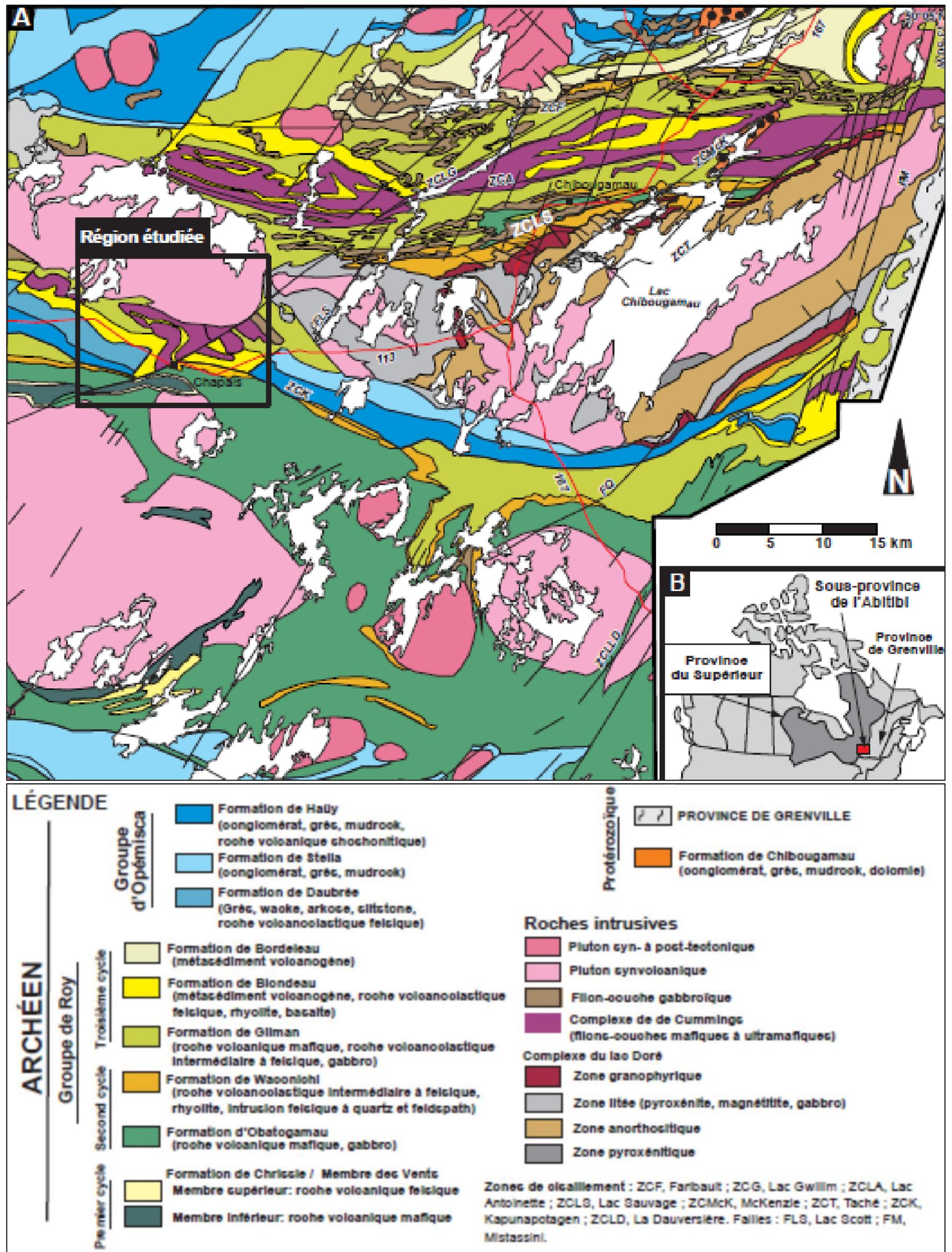
The metavolcanic stratigraphy in the Chapais-Chibougamau area is representative of deep-water deposition to submarine environments. The metavolcanic-sedimentary package is cut by mafic to ultramafic intrusions (Lac Dore Complex being the best-known example), mafic sills (Cummings Sills and gabbro), and younger plutonic intrusions that range from tonalite to carbonatite in composition.

This section is largely inspired by previous reports from XXIX, mainly Yassa et Puritch (2024), which summarizes internal XXIX reports.

Recent work by Leclerc et al. (2010, 2012) has refined the understanding of the complex geology and stratigraphy of the project area. The earlier stratigraphic interpretation has been modified, in order to take into consideration recent field observations.

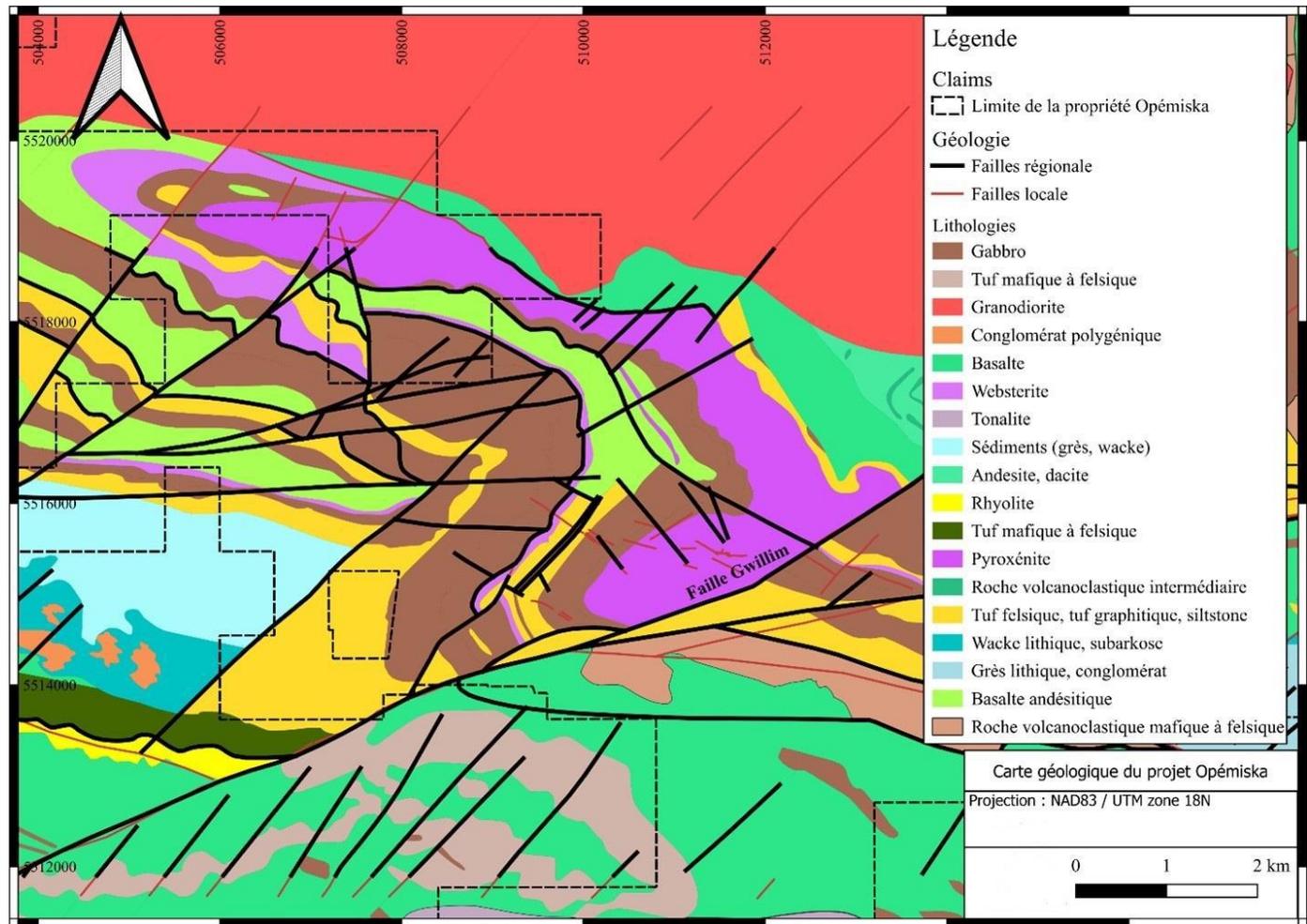
The geology of the Opémiska property is characterized by a fold affecting the Cummings Complex introduced at the lower contact of felsic volcanics of the Blondeau Formation (Figure 7-2). The Cummings Complex are comprised of three separate differentiated sills: the Roberge Sill at the base; the Ventures Sill; and the Bourbeau Sill higher-up in the Blondeau stratigraphy.

Figure 7-1: Regional Geology Plan



Source: Modified from Leclerc et al., 2010.

Figure 7-2: Project Geology Plan



Source: XXIX, 2025.

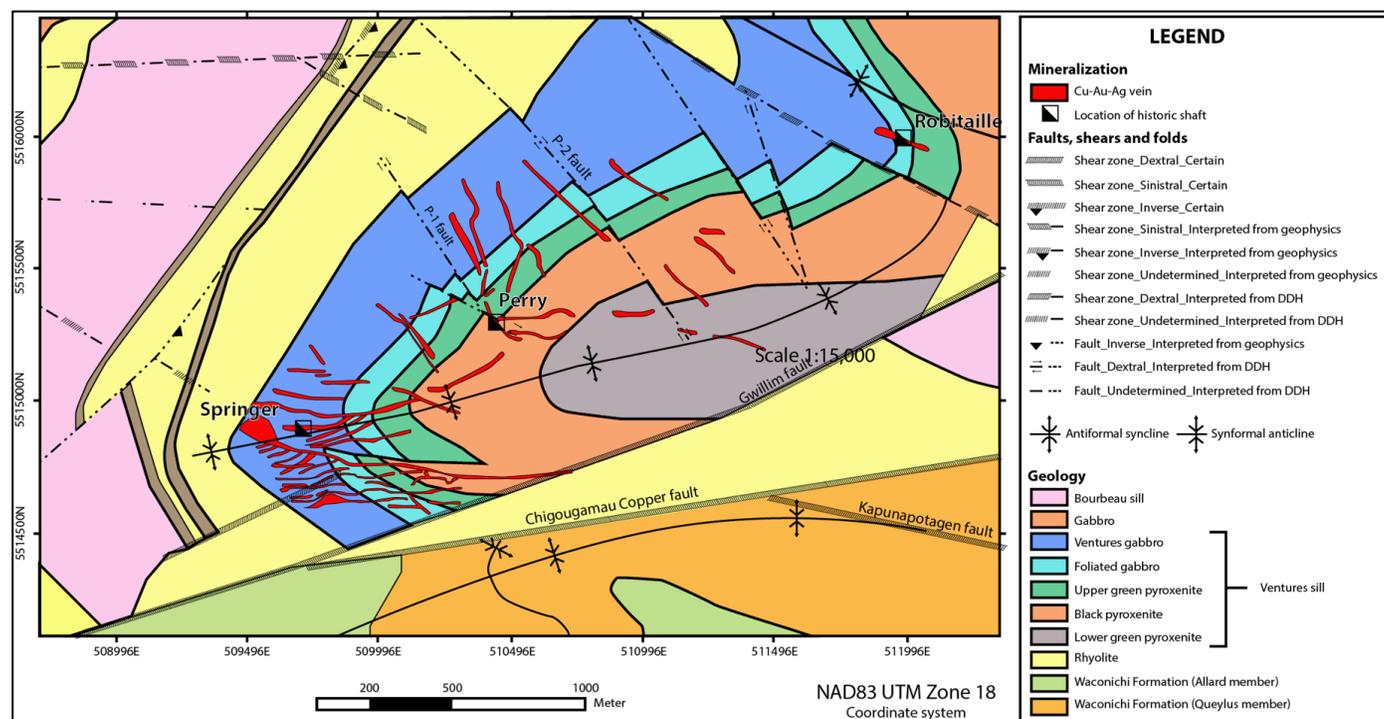
The Ventures Sill, which is approximately 1,000 m thick, is the most common host unit of the mineralization on the Opémiska property. The Ventures Sill has been divided from bottom to top into five laterally-persistent units:

- Lower Green Pyroxenite – The lower green pyroxenite (approximately 60 m thick) represents the basal layer of the Sill. It is medium-grained, dark green to black in colour and strongly magnetic, with abundant serpentinized fractures. The upper contact is commonly sharp with no evidence of “chilling”
- Black Pyroxenite with Peridotite Sills – This unit is 350 m thick, medium-grained and dark grey to black colour. Layers of serpentine–talc–magnetite (after cumulate olivine) are present. Layers containing primary chromite and magnetite are also recognized (Watkins and Riverin, 1982)

- Upper Green Pyroxenite – The Upper Green Pyroxenite is approximately 60 m thick and locally quite similar to the underlying pyroxenite. The rock is somewhat coarser-grained and interstitial feldspar is present. Contact with overlying unit is sharp, marked by cumulus plagioclase and titaniferous magnetite (McMillan, 1972)
- Foliated Gabbro – The Foliated Gabbro averages 150 m in thickness, its base is commonly marked by a 15 to 30 cm thick layer of clinopyroxene containing 30% to 40% magnetite, and layering is well developed throughout. Strong foliation is defined by alignment of pyroxenes and feldspars and the unit has a sharp upper contact marked by abrupt change in texture and grain size
- Ventures Gabbro – The Ventures Gabbro hosts the bulk of the mineralization at the Springer and Perry mines. The unit is 350 m thick and represents the top of the Ventures Sill. Its composition is similar to the underlying Foliated Gabbro, but locally carries up to 5% free quartz. It is generally coarse-grained with ophitic texture (association of lath-shaped euhedral crystals of plagioclase grouped radially or in an irregular mesh with surrounding or interstitial large anhedral crystals of pyroxene)

The distribution of these units is shown in Figure 7-3.

Figure 7-3: Deposit-Scale Geology



Source: XXIX, 2025.

7.2.1 Structure

Within the Chapais-Chibougamau region, a combination of several deformational events created structural interference patterns in certain sectors. The Opémiska property is located just south of the east-west trending Chibougamau Anticline, which is cored by the Opémiska and Chibougamau plutons. This fold structure is related to the second major phase of regional deformation.

At Opémiska, the mineralization occurs within the large composite Ventures Sill that intrudes felsic volcanics (rhyolite) of the Blondeau Formation. The Ventures Sill and the volcanic units have been overturned, folded and truncated by the Gwillim Fault/Shear Zone (Figures 7-2 and 7-3). The fold plunges 45° to 65° to the east and postdates the second phase of regional deformation, but predates a third deformation phase (Leclerc et al., 2012).

The Gwillim Fault (referred to originally as the Campbell Lake Fault) traverses the main block of claims in the project area. An apparent “sinistral” horizontal displacement of 3,300 m along this structure has been determined (Watkins and Riverin, 1982). This fault was active during several tectonic events (Dimroth et al., 1984), but its principal movement predates the third phase of deformation (Brown, 1970).

Six distinct directions of orientation were recognized by Falconbridge in the mineralized zones (GM-46158, Table 6.2):

1. N-100° represented by the main structure at Springer, vein no. 3 and western section of vein no. 1
2. N-080° represented at Springer by vein no. 2, vein no. 0, north portion of vein no. 11, east part of vein no. 1, vein no. 4, vein no. 7, and vein C, at Perry
3. N-070° represented south of Springer shaft by veins 34, 13, 5, 6 and 7 south
4. N-130° represented at Springer by south part of vein no. 11, vein no. 22 and at Perry vein D south and Gap Zone
5. N-160° to 170° represented at Perry by veins A, B, B-North, J North and K
6. The gold-bearing “arsenopyrite fault” is oriented at N-150° parallel to the P-1 Fault.

There are cross-cutting relationships between these different sets of fractures/shears, as exemplified by the Arsenopyrite Shear at surface. The relative ages of these structures remain to be determined.

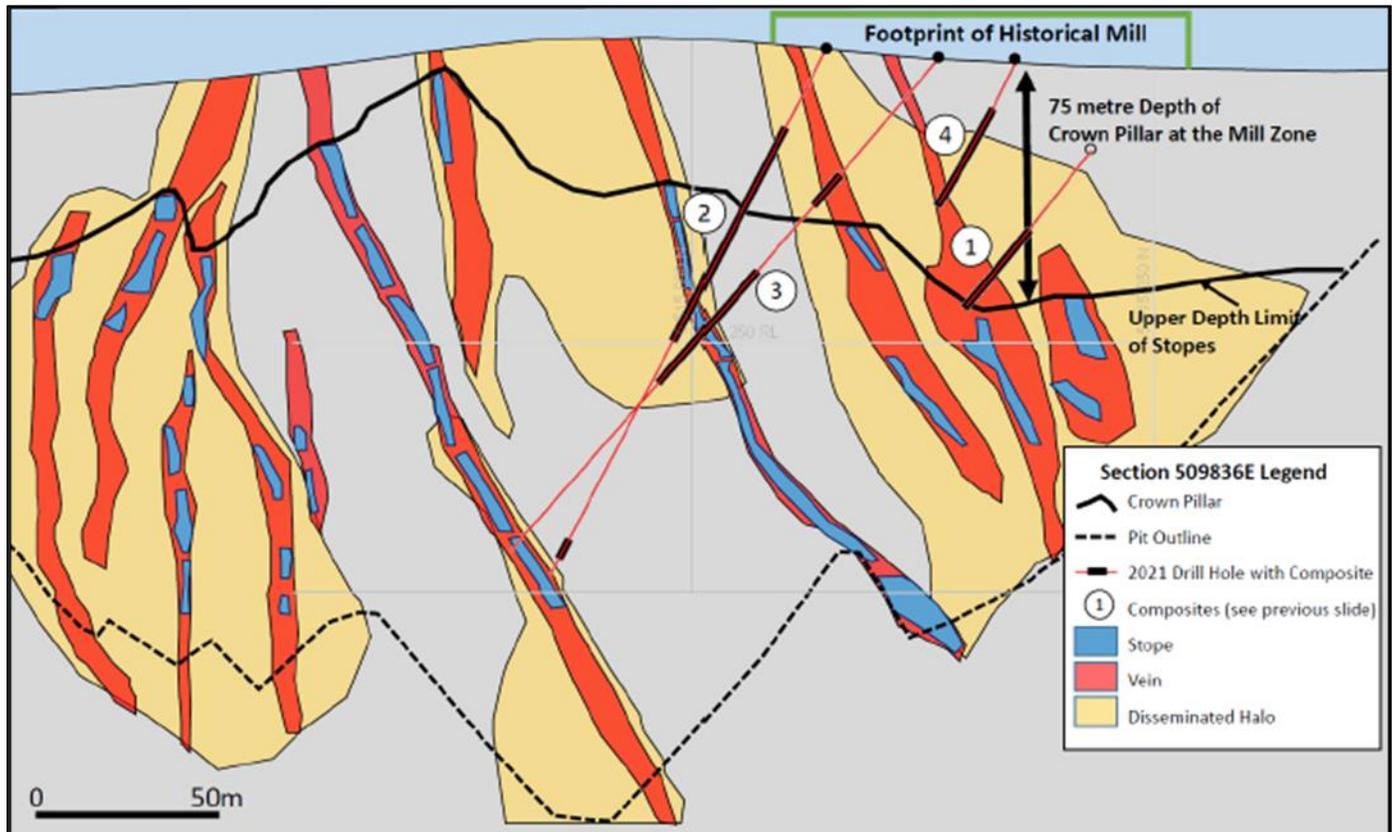
7.2.2 Mineralization

The mineralization at Opémiska consists largely of chalcopyrite-bearing quartz veins that occupy fracture systems in the folded and faulted gabbroic portions of two conformable, regionally extensive, layered Archean ultramafic-mafic sills. The veins are generally restricted to the fracture system and in lower grade halos around the main fractures/veins. The width and frequency of the veins tend to increase toward the dilated nose of the main structure at the Springer mine (Watkins and Riverin, 1982).

The mineralization at the Springer mine is associated to a series of east-trending (090°), steeply (65°) north-dipping, sets of axial plane faults and fractures with right-handed (dextral) displacement that developed in areas of maximum inflexion of folds (Watkins and Riverin, 1982). Plan and cross-section views of Springer show at least three different

orientations for the mineralized veins which could indicate a conjugate fault system or separate fracture systems. A disseminated halo (stockwork) surrounds most of the higher-grade zones (Figure 7-4).

Figure 7-4: Illustration of a Disseminated Halo (Stockwork) surrounding the Higher-Grade Zones



Source: XXIX, 2021.

The deposit extends over 1,900 m from east to west and 1,400 m from north to south, reaching 975 m below the surface.

Most of the high-grade mineralized zones are between 5 m and 15 m thick. Locally, these zones can have a minimum thickness of 2 m and a maximum thickness above 50 m. The lower-grade stockwork zones surrounding the high-grade zones can reach up to 100 m in total thickness.

Geological continuity and grade continuity are considered good in the high-grade zones, and grade continuity remains good above both the open-pit cut-off grade of 0.15% CuEq and the underground cut-off grade of 1.00% CuEq, with little internal dilution between the high-grade material. Grade continuity for the lower grade Stockwork zones is also considered good above the open pit cut-off grade of 0.15% CuEq. As expected for low-grade zones, it becomes more irregular above the underground cut-off grade of 1.00% CuEq.

In the limb of the fold at Perry mine, the mineralization is associated to northwest-trending faults and fractures, developed perpendicular to stratigraphy.

Generally, mineralization of economic interest appears within more fractured/sheared sections of the host gabbro. These sections are generally strongly chloritized and variably silicified.

A detailed description of the mineralization intersected in the 2016 drilling further classified the veins as follows:

- massive pyrite veinlets (cut by magnetite?)
- magnetite veins (minor associated disseminated chalcopyrite)
- sulphide veins (massive chalcopyrite) with magnetite-rich margins, also with disseminated fragments of massive magnetite within chalcopyrite
- high-sulphide veins with 30% to 50% quartz with massive chalcopyrite and some magnetite – anomalous W values are sometimes found associated to these veins
- quartz veins within gabbro with higher gold values and low copper
- quartz veining within felsic tuffs with associated gold and minor copper and minor arsenopyrite
- gold-rich quartz-arsenopyrite veins north of vein nos. 1 and 2 that cross-cut the copper-rich veins.

Although most of the mineralization historically mined at Springer and Perry was hosted in the upper part of the Ventures Sill, the regional and local structures are also important controls on mineralization at Opémiska. At Springer, the fold nose corresponding to the overturned anticline in the mafic-ultramafic sills controls significant amount of mineralization. A 6.0 m wide zone containing disseminated pyrrhotite and chalcopyrite occurs locally at the top of the Ventures Sill, where it is dilated at the nose of the fold (Watkins and Riverin, 1982).

The mineralized veins at Springer were described as restricted to fractures hosted in gabbro at the stratigraphic top of the Ventures Sills. The mineralization is generally massive, but locally disseminated. The main fractures trend 090° and dip 70° north. The main veins are up to 1,200 m long, average 6.0 m thick, and have been followed to >1,000 m depth. Vein No. 3 is the most important one, along with Vein No. 7 farther to the south. Additional, less important veins (six additional veins) have also been exploited. The mineralization consists mainly of chalcopyrite, pyrite, and pyrrhotite with smaller amounts of sphalerite, magnetite, galena, molybdenite, arsenopyrite, and gersdorffite (NiAsS). Native gold occurs in association with chalcopyrite and pyrite. The non-metallic gangue minerals are quartz, calcite and chlorite and minor amounts of biotite, stilpnomelane, and actinolite. Locally significant amounts of scheelite and molybdenite are present. Later cross-cutting veins carry pitchblende-uraninite and molybdenite (DV 98-03). The alteration surrounding the veins is described as chlorite and carbonate.

At the time of the start of operations at Springer (1952), five major copper-gold bearing veins or zones had been explored in the shaft area, either underground or by surface drilling. The veins consisted of chalcopyrite accompanied by quartz and magnetite. These veins generally strike east to west and dip steeply to the north. Some silver is present and locally important cobalt values have been obtained (e.g., surface drill hole S-57). In addition to the five veins, there

are many other important drill intersections that are as yet uncorrelated. Included in these intersections are some carrying important zinc, lead and gold values, but in some cases with little copper present (GM-02098).

When mapping the area in 2009, the Ministère des Ressources naturelles et de la Faune (MRNF) sampled a mineralized quartz vein in gabbro outcrop (Sigeom à la Carte, 32G15 sample No. 2009050061). This vein is likely the same one intersected in drill hole OP-16-08. The sample graded >5.0 g/t Au, 740 parts per million (ppm) Co, 60 ppm Mo, 260 ppm Ni, 0.14% Zn, 20.49% Cu, 35.65% Fe and 0.29% W. This result confirms the multi-element association in some of the veins at Springer.

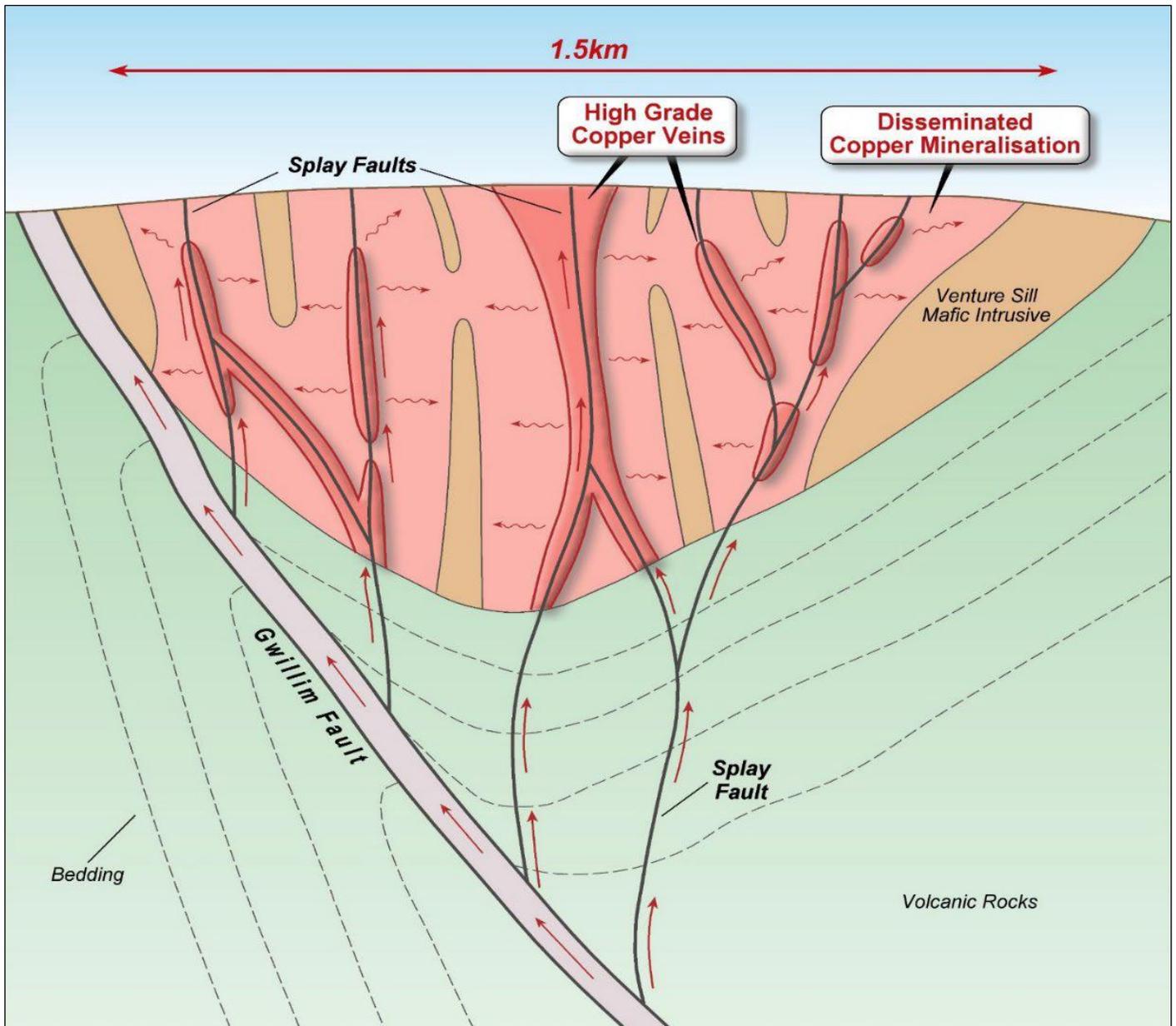
8 DEPOSIT TYPES

The mineral deposit type on the project is structurally controlled copper-gold veins. The veins occur in east-west trending axial planar faults and northwest-trending radial structures within the folded mafic-ultramafic Ventures Sill and Bourbeau Sill. The folds postdate the main east-west trending regional isoclinal folds (D2a), but predate the third phase of deformation (D3), and are interpreted to represent drag folding during sinistral movement (D2b) along the Gwillim Fault (Figure 8-1) (Leclerc et al., 2012). Progressive deformation of the Ventures Sill caused rupturing of the unit, which produced axial planar and radial fractures and faults, some of which were subsequently mineralized when the regional compression rotated from north-south to northwest-southeast, opening up the existing structures and providing pathways for circulation of mineralizing hydrothermal fluids.

In the Chibougamau mining camp, structurally-controlled copper-gold mineralization occurs in west-northwest-trending dextral shear zones (Merril, Copper Rand Mines) related to the second deformation event (D2) and in cross-cutting, northeast-trending dextral shear zones (Henderson, Portage mines).

Similitudes are interesting when comparing the veins systems at the Springer and Perry mine deposits on the Opémiska property to the Chibougamau mine deposits farther east. At the Springer mine, the veins proximal to the Gwillim Fault also contain significantly higher gold grades than the more distal veins (Salmon et al., 1984). At Chibougamau, the main copper veins are oriented northeast-southwest at Henderson-Portage mines, but the later-formed “Mines Shears” oriented at 110° appear to carry more gold.

Figure 8-1: Opémiska Deposit Model Looking West



Source: XXIX, 2021.

9 EXPLORATION

9.1 Re-Interpretation of Geological Model

XXIX carried on with the extensive compilation work started by Ex-In on the historical Springer and Perry mines. During the operation of the mines, all drill holes were logged on paper and no digital records were compiled. QC Copper and Gold built a digital database that includes drill hole collar locations, deviation tests, geology, sampling, and assay results. The compilation included all the historical surface and underground drill holes for a total of 19,471 drill holes (1,074,735 m) and 375,931 samples.

None of the drill core from the historical surface and underground drilling during the mining period was preserved and there is no means of directly validating historical assays. Assay certificates were not preserved and the samples were assayed at the on-site mine laboratory. No information is available as to the sample preparation or analytical methods used by the mine. Assay validation is discussed in Section 12. In summary, some of the historical surface drill holes were twinned in the 2019 and 2021 diamond drilling programs. The assay results and logs were compared in detail with the historical drilling, with the ultimate objective of validating all the historical mining results.

Level plans, sections, and longitudinal projections were scanned and georeferenced to confirm the location of all the drill holes and digitize all the underground mine openings. All underground drifts, veins and stopes were digitized from the available scanned maps and combined into 3D wireframe models using various software and ultimately integrated into GEOVIA GEMS™ modelling software and later converted to Micromine format. Geological contacts and faults were also digitized from level plans and the linework combined into 3D surfaces to aid interpretation. Several hundred individual 3D wireframes of the veins and stopes were constructed to approximately the -150 m elevation (approximate depth of 550 m below surface) and subsequently down to the bottom of the historical Springer mine around 650 m in depth and to the bottom of the historical Perry mine around 1,000 m in depth. Many stopes were intersected during the 2019 and 2021 drill programs typically within one or two meters of the projected downhole locations from the drill hole collars as projected in 3D.

All the work described above was performed in the original local mine grid coordinates. All the drill holes and the 3D wireframe models were subsequently converted to UTM coordinates and elevations above sea level to better integrate with GPS and surface data, such as the Chapais town site and surrounding road network, using the transformation equations in listed below:

- Mine to UTM NAD83, Zone 18 Co-ordinate Transformation Equations:
 - $X_{UTM} = (0.3048 * X_{Mine}) + 508,249.09$
 - $Y_{UTM} = (0.3048 * Y_{Mine}) + 5,513,407.36$
 - $Z_{UTM} = (Z_{Mine} - 3,676.91) * 0.3048$

These equations were generated by a qualified land surveyor in Chibougamau, based on regression analysis of a large number of mine-era surface drill collar casings that were re-surveyed using a differential GPS unit. They were validated and confirmed when new found surface drill casings were located and georeferenced. In addition, QC Copper and Gold's drilling located buried drill casings within 1 or 2 m accuracy, when bulldozing new drill setups and projected stipes were typically encountered within a few of meters of anticipated down hole depth locations.

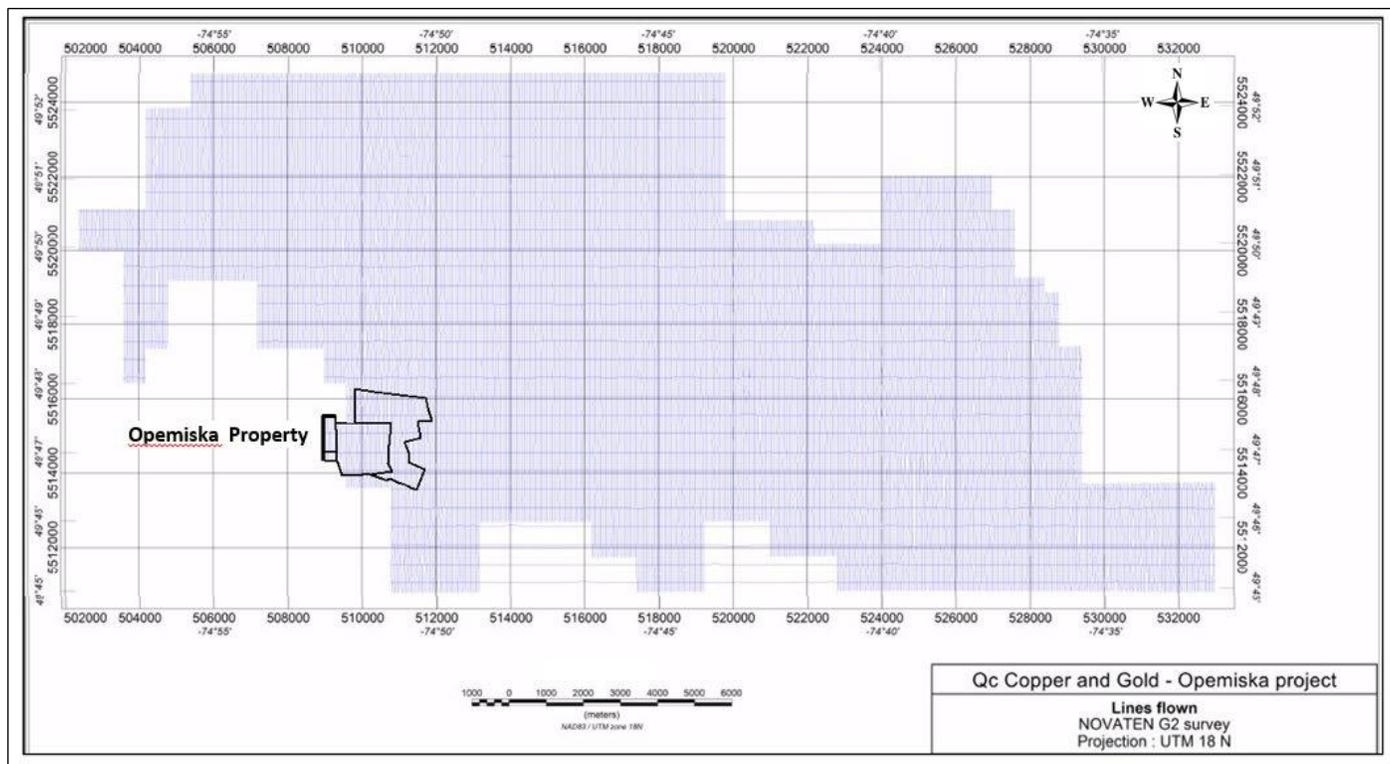
9.2 Geophysics

In 2022, QC Copper and Gold completed a high-resolution, 75 m line spacing airborne magnetic survey that covered the Opémiska property and QC Copper and Gold's adjacent properties (Figure 9-1) for a total of 6,071 km. The survey data were micro-levelled to provide the maximum resolution and interpretability. Previous magnetic susceptibility measurements on drill core were used to constrain the 3D inversions of the airborne survey data. This study was expected to define distinct geological domains for improved mineral resource modelling. The geophysical data reprocessing and 3D inversions results indicate that the Ventures Sill is variably magnetic with the most magnetic portion corresponding to the Ventures Gabbro and Green Pyroxenite units. The magnetic response within these units, where cut by northwest-trending mineralized faults is diminished, which suggests that the mineralizing fluids were magnetite destructive. Magnetite is a ubiquitous vein mineral, which suggests that either the magnetite was chemically remobilized into the veins or that the veins formed in the late stages of the hydrothermal system, when the fluid composition evolved from magnetite destructive to magnetite formative.

During the fall 2022, Géophysique TMC was commissioned by QC Copper and Gold to complete a mise-à-la-masse survey in the vicinity of the Saddle Zone. The purpose of the survey was to confirm the connection between the mineralized zones and aid interpretation of the geological model in the area. Beforehand, a field grid consisting of 13 lines ranging from 275 to 600 m long was cut to lay the wires to guide the survey.

The mise-à-la-masse survey provided two types of readings. The first is a surface survey, which consisted of putting an electrode down the drill hole to the level of a mineralized zone and reading along the surface lines. Fourteen different readings were made using six different drill holes. The second type of reading was a drillhole-to-drillhole reading, which consisted of placing one transmitting electrode in one drill hole, at a mineralized interval and the receiving electrode at the same interpreted interval in an adjacent drill hole, and reading along one line to confirm the connection of the two zones.

Figure 9-1: Geophysical Survey Location Plan



Source: Modified from Expert Geophysics, 2022.

9.3 Drill Hole Televier

A focused optical and acoustic drill hole televier surveying program was planned at the end of the 2021 drill program to obtain oriented structural measurements. A total of 16 drill holes were scanned (optic, some also with acoustic) with a Semm logging televier. Three of those drill holes were on the Bouchard Zone, four on Springer Zone, and nine on the Saddle Zone. The interpretation was completed in-house using Wellcad to help understand the orientation and the relationships of the structures to the mineralization.

9.4 Structural Geology Study

The company engaged SRK to complete a structural geological study to better understand the relationships between the different veins and mineralized faults of this structurally-hosted copper-gold-silver deposit. The purpose of the study was to integrate all the available observations, including surface stripping, drill core, underground geology level plans, longitudinal projections and cross-sections, 3D shapes of the existing stopes, and constrained inversions of the airborne magnetic data, into a comprehensive structural model for the Opémiska deposit.

10 DRILLING

10.1 Introduction

In total, 21,918 surface and underground drill holes for 1,525,074 m are recorded for project. Section 10.2 presents the drill programs completed by XXIX (and its predecessor companies) and Section 10.3 presents the parameters used during these drill programs. Further information about drill hole locations and cross-sections can be found in Section 14.

10.2 Drill Hole Programs

The drill hole programs are listed in Table 10-2 and described in the following subsections.

Table 10-1: Drill Hole Programs

Year	Season	Name of Company
2024-2025		XXIX
2023	Winter	QC Copper and Gold
2022	Summer	QC Copper and Gold
2021-2022	Autumn/winter	QC Copper and Gold
2021	Winter/spring	QC Copper and Gold
2019		PowerOre

10.2.1 2019 Program

In 2019, an initial diamond drilling program of 23 holes and 3,364 m was carried out on the Springer Zones with the primary objective of verifying that significant disseminated mineralization exists between the veins that were mined underground. Drilling focused on crown pillars and interior pillars, where these could be targeted, and results confirmed the expectations that the project could be re-evaluated as an open pit to mine pillars and the low-grade material that was left in the underground due to prevailing economics.

A series of drill holes were also completed to duplicate some of Falconbridge’s drill holes, to test favourable sections for disseminated copper mineralization adjacent to largely mined out “high-grade copper zones” and finally investigate the metavolcanic/gabbro contact for disseminated copper mineralization within both the Ventures gabbro and felsic metavolcanic rocks.

The 2019 diamond drilling program objective was to confirm the presence of wide, near-surface mineralization on the periphery of existing mined out veins. This objective was confirmed, and significant intervals were also identified in

areas previously considered to be barren. In addition, some “unnamed” veins were intersected in the drilling; these were not mined underground. Moreover, drilling in the vicinity of the historical process plant indicates that the mine left a thick crown pillar in this area, presumably to protect the mill infrastructure.

The drill program, logging and sampling were carried out under the supervision of Claude Larouche, P.Eng., ing. Samples of drill core were cut longitudinally along a line marked by the logging geologist and cut in half using a diamond drill core saw. Samples were assembled in batches with pulps of certified reference materials and blanks. Approximately 10% of the samples submitted for analyses were either certified reference materials or blanks. In addition, a suite of drill core duplicates was submitted for analysis.

10.2.2 Winter/Spring 2021 Program

From January 22 to May 16, 2021, QC Copper and Gold undertook a drilling campaign on its Opémiska Project. The work was carried out by Forage Miikan, a subsidiary of Forage Chibougamau.

A total of 78 drill holes were completed for a total of 16,411 m. The work was carried out under the supervision of Denis McNichols, P.Geo., then project Manager for QC Copper and Gold. The drill collars were set-up by a professional land surveyor, who returned to the field after the campaign to record the final position of the drill holes. All the drill hole collars were aligned using an "azimuth aligner" from Minnovare. The deviation tests were completed using a Reflex magnetic device, starting at 30 m and subsequently at 50 m intervals. Outlier azimuths and dips were removed from the dataset before plotting.

The mineralized intersections encountered in the 2021 drilling program are very similar to those in the previous drilling. Mineralization occurs in the form of shear veins (mainly Springer) or quartz-rich veins (mainly Perry), with mineralization consisting primarily of chalcopyrite with accompanying pyrite and minor pyrrhotite with magnetic, chlorite, quartz, and calcite as gangue minerals. Locally, sphalerite-rich veins are encountered that have only modest concentrations of copper, but may have high gold values. Arsenopyrite is also present locally, but it has not been possible to relate the different vein mineralogy to different vein generations.

It was also observed that the numerous mined veins in Springer and Perry mines are surrounded by low-grade stockwork halos of weakly and variably altered Ventures Sill rock with minor chalcopyrite veins a few cm to a few tens of centimetres thick separated by barren rock. The low-grade copper typically forms stockwork halos up to three to five times the thickness of the veins. In the core of the Springer mine, the veins are sufficiently numerous to create a continuous halo of mineralization over several hundreds of metres, centred on vein no. 3, the most important vein historically mined by Falconbridge.

10.2.3 Late 2021, 2022, and 2023 Programs

Subsequent to the publication of the initial MRE for the project, from October 26, 2021 to February 12, 2023, QC Copper and Gold undertook three drilling programs on its Opémiska property in fall-winter 2021-2022, summer 2022, and winter 2023.

A total of 180 drill holes totalling 47,192.2 m were completed during those programs. The diamond drilling programs were carried out on the Springer, Saddle (located between Springer and Perry), Perry, Bouchard and McNichols Zones, with the objective of verifying the presence of significant disseminated mineralization. The work was carried out under the supervision of Denis McNichols, P.Geol., then Exploration Manager for QC Copper and Gold. The drill core processing and the operation of drill core sawing equipment were carried out under the supervision of André Bouchard, employed by QC Copper and Gold.

The drill collar sites were located initially by QC Copper and Gold personnel using a handheld GPS. Prior to the start of drilling, all the drill hole collar alignments were determined with an "azimuth aligner" from Minnovare. The deviation tests were completed at every 50 m using an Axis Mining Technology gyro. Suspicious azimuths and dips were removed from the dataset before plotting. At the completion of each drill program, a professional land surveyor returned to the field to measure and record the final UTM coordinates of each collar using a differential GPS unit.

Drill core samples were cut in half using a diamond saw along a longitudinal line drawn by the logging geologist. Samples were collected in batches with insertion of certified reference material sachets and blanks. One certified reference material and one blank were sent with every 50 samples. In addition, quartered drill core duplicate samples were submitted for analysis. A total of 29,914 samples were sent to an independent commercial laboratory for analysis.

The mineralized intersections in the 2021, 2022 and 2023 drilling programs are very similar to those in the previous drilling.

10.2.4 2024-2025 Program

From November 16, 2024 to March 22, 2025, XXIX undertook a drilling campaign on its Opémiska Project. The work was carried out by Forage RJLL.

A total of 18 drill holes were completed for a total of 2,530 m. The work was carried out under the supervision of Ahcene Gaoui, P.Geol., Exploration Geologist for XXIX. The drill collars were set-up by geologists. All the drill hole collars were aligned using a TN14 instrument from Reflex. The deviation tests were completed using a gyro from Reflex, starting close after the end of the casing and subsequently at 51 m intervals. Outlier azimuths and dips were removed from the dataset before plotting.

The objectives of the drilling campaign conducted between fall 2024 and spring 2025 were as follows:

- test certain areas of the zone identified as part of the open pit in order to increase resources
- verify XXIX's new interpretation of mineralization in the Saddle Zone
- use all the drilling data obtained during this campaign to prepare an updated resource estimate.

The mineralized intersections encountered in the 2024-2025 drilling program are very similar to those in the previous drilling, with mineralization consisting primarily of chalcopyrite with accompanying pyrite and minor pyrrhotite with magnetic, chlorite, quartz, and calcite as gangue minerals. Arsenopyrite is also present locally, but it has not been possible to relate the different vein mineralogy to different vein generations.

10.3 Drill Program Parameters

10.3.1 Hole Selection

Drill holes were designed to target previously drilled mineralization, using the extensive historical database and concentrating on the Opémiska deposit.

10.3.2 Drill Hole Location and Set-up

On the project, drill collar locations are pre-surveyed by XXIX using a hand-held GPS.

A wooden stake or picket is hammered into the ground to mark the collar location. The stake is then inscribed with the predetermined drill hole identification, the intended azimuth, and the anticipated depth of the hole. In the case of inclined drill holes, a separate set of clearly marked wooden pickets mark the foresight and backsight for the alignment of the drill rig. These pickets are placed a sufficient distance from the collar location so as not to be disturbed by the drilling contractor during equipment installation and drill set-up. Foresight and backsight pickets are accurately surveyed and installed by the issuer.

The collar location is subsequently prepared to allow easy access to the drilling equipment. In some instances, this involves brushing and some tree removal.

At the completion of each drill program, a professional land surveyor returned to the field to measure and record the final UTM coordinates of each collar using a differential GPS unit.

10.3.3 Drill Hole Orientation at Start-up

Prior to the start of drilling, all the drill hole collar alignments are determined with an "azimuth aligner" from Minnovare. This method is not affected by the high magnetic susceptibility of the rocks in the Ventures Sill. The device was checked regularly to ensure proper calibration.

10.3.4 Drill Hole Orientation During Operation

Deviation tests were completed at every 50 m using a gyro from Axis Mining Technology. Suspicious azimuths and dips were removed from the dataset before plotting.

10.3.5 Drill Hole Coring

Drill cores are provided by the drilling contractor in NQ size (46 mm diameter) except for some drill holes that crossed underground stopes or mine drifts and were drilled telescopically in HQ-NQ-BQ size to pass through those openings.

The core is collected in a standard drilling tube, and the drillers place the core into wooden core boxes or trays specially manufactured for this process. The driller marks the depth in meters after each run, usually every 3 m, sometimes at shorter intervals when appropriate.

The drill hole is terminated by the XXIX site geologist once the target depth is reached. Once the drill hole is terminated and the final downhole survey reading is collected, the drill crew pulls the rods for mobilization to the next drill site.

The casing may be left in the hole and cut slightly above the surface level. It is marked with the drill hole identification number inscribed on a metal tag.

10.3.6 Core Handling at the Drill Rig

Diamond drill cores are collected in lengths up to 3 m in an NQ core barrel. The NQ core trays hold a nominal 4.5 m of cohesive core in three 1.5 m rows. After each drill run, the driller's helper loads the core into the wooden core trays at the drill rig under the driller's supervision. The driller's helper identifies the core trays with a permanent marker, indicating the drill hole number and the sequential box number, beginning with box 1 after collaring the casing into bedrock. Drill hole numbering and box numbering are also inscribed on the end piece of the core tray next to the first core placed in the row.

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

The wooden depth markers are clearly marked in meters in clean and legible writing. Additional notations can be provided on additional wooden blocks indicating if bad ground, cavities in the bedrock, or changing water conditions were encountered that resulted in core loss. Once the core tray is filled, it is secured shut using a second core box. It is then carefully stacked for transport to the core logging facility.

10.3.7 Receiving Core at the Core Logging Facility

The drill core is transported daily to the core logging facility. Care is exercised to ensure the lids are securely attached to minimize core disturbance, breakage, and loss during transport from the drill site.

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

10.3.8 Geological Logging Procedure

Detailed core logging has several components: geological (lithologies, structures, alteration, and mineralization), sampling, and photography.

All geological characteristics are described, including lithologies, structures, alteration, sulphide mineralization, assay sample numbers and intervals, density sample numbers and intervals, etc.

10.3.9 Assay Sample Selection

Assay samples are broken at major lithology contacts to represent homogeneous units. The minimum assay sample interval in the hole will be not less than 50 cm, except in unique circumstances (e.g., lithological units that are mineralized and less than 50 cm long). The maximum sample interval will not exceed 1.5 m. Procedures state that no sample will cross a major rock boundary, alteration boundary or mineralization boundary.

The geologist determines sampling intervals during logging and marks them on the core boxes or the core itself using coloured lumber pencils. Samples are numbered in consecutive order using two-way sample tag books that the laboratories provide. The sample sequence includes QC samples inserted into the sample stream using sample numbers that are in sequence with the core samples.

Sample intervals, sample numbers, and QC samples are noted in the **Assay** tab of the **Descriptions** section in the software.

10.3.10 Core Sampling

A geotechnician trained in core cutting procedures cuts the core at the core sampling facility. The logging geologist has already clearly marked out all pertinent cores for cutting and sampling. The geologist also places a paper sample tag containing a two-part sample number corresponding with the required sample interval at the beginning of the sample interval. One part will be stapled into the core box as a permanent sample reference, which will remain on the wooden core tray, and the geotechnician will remove the other part of the paper sample tag and place it inside the plastic bag.

The core is sawn with a diamond saw; one half of the core sample is placed in a sample bag, and the remaining half is returned to the core box. The cut core will be returned to the core box in the same position as it was removed so as not to rotate the core or reverse the downhole direction of the core. If the above procedure is carefully followed, the core remaining in the tray will retain its “fitted” appearance.

The bag will then be closed using a zip tie and stored in sequence before sample dispatch preparation.

A “standard” sample consisting of material of known metal content is included in the sample sequence by the trained core sampler. XXIX includes a standard after every 50 samples. Standard QC samples are selected and placed in sequence by a geologist. Standards are used to test the laboratory’s reporting for select key elements.

Similarly, a “blank” is included in the sequence as part of the QA/QC process. Blank material is technically devoid of any metals of interest.

10.3.11 Sample Shipment Preparation

Assay sample bags are packed in large rice bags. The rice bags are piled onto a pallet for transport. A waterproof bag containing the laboratory sample submission form and a hard copy of the sample dispatch sheet are included with the sample shipment.

The palletized rice bags are stored on site until shipped to the laboratory.

The laboratory is notified by email that the samples are enroute. A digital copy of the sample submission form and the sample dispatch list is emailed to the laboratory manager once the samples have left the site. Any additional instructions for processing (such as expedited service) are communicated to the laboratory manager at this time.

10.3.12 Core Storage

Following sampling, the ends of the core trays are labelled using an inscribed metal tag, which is durable and will survive weathering far longer than the permanent marker. The core tray metal tags are marked with the hole number, the tray number, and the “from-to” meterage.

The core trays are stored in metal racks on site.

11 SAMPLE PREPARATION, ANALYSES, AND SECURITY

11.1 Core Handling, Sampling, and Security

Individual cut samples were placed in poly bags with a unique bar-coded assay tag, and poly bags were placed in rice bags. They were then loaded on pallets for transport. The results were received by email in secure PDF files and Excel spreadsheets.

11.2 Laboratories Accreditation and Certification

For the 2019 drill program, samples were shipped to Laboratoire Expert in Rouyn-Noranda. Samples from the 2021-2023 drill programs were sent to ALS laboratories in Val-d'Or. Samples from the 2024-2025 drill program were sent to AGAT Laboratories in Val-d'Or. ALS and AGAT are ISO 17025 compliant.

Laboratoire Expert is a non-accredited facility that routinely performs assaying for junior mining companies. Blanks, CRMs and duplicates are inserted into the sample sequence at all sample preparation stages, as part of the laboratory's internal QA/QC protocol.

All three laboratories are independent of XXIX.

11.3 Laboratory Preparation and Assays

11.3.1 Sample Analysis Procedure (Laboratoire Expert)

Samples received at Laboratoire Expert are logged into the tracking system, weighed, dried (if necessary), crushed to 80% passing minus 10 mesh, riffle split to 250 g, then pulverized to 90% passing minus 200 mesh.

Samples are analysed for gold by lead fire assay on 30 g aliquots with AA finish. Samples returning results >5,000 ppb gold are then re-assayed by fire assay with gravimetric finish.

Copper, silver, cobalt and zinc samples are analysed by partial digestion AA, with samples assaying >10,000 ppm re-assayed by total digestion AA.

11.3.2 Sample Analysis Procedure (ALS)

Sample preparation at ALS was divided between the full-service laboratory in Val d'Or and a preparation laboratory in Lebel-sur-Quevillon. Since the laboratory has uniform protocols throughout, it is very common for the prepared pulps to be sent to more than one of their major laboratories for analysis. Gold analyses by fire assay, however, were mainly performed in Val d'Or.

Rock packages CRU-QC and PUL-QC were used for crushing and pulverizing all samples. The entire sample was crushed to 90% passing <2 mm, then 500 g was split off and pulverized to better than 85% passing 75 microns (μm).

A sample from the pulp was digested in an aqua regia leach and analysed by ICP-MS under ALS procedure codes ME-ICP41 (for Cu, Zn, Co, Ag) and Cu-OG46 (for Cu when above 10,000 ppm). Fire assay was used for gold under ALS procedure code Au-AA23.

ALS maintains ISO registrations and accreditations and all ALS geochemical hub laboratories are accredited to ISO/IEC 17025:2017 for specific analytical procedures.

11.3.3 Sample Analysis Procedure (AGAT)

The entire sample was crushed to 75% passing <2 mm, then 250 g was split off and pulverized to better than 85% passing 75 µm.

A sample from the pulp was digested in an aqua regia leach and analysed by ICP-OES for 14 elements and with ICP-MS finish for copper when above 10,000 ppm. Fire assay with an AAS finish was used for gold.

11.4 Quality Assurance and Quality Control

QA/QC on the project follows “CIM Estimation of Mineral Resources & Mineral Reserves Best Practice Guidelines” (2019).

QA/QC programs have two components. Quality assurance deals with preventing problems using established procedures, while quality control aims to detect and assess problems and take corrective actions. QA/QC programs are implemented, overseen, and reported on by a QP as defined by NI 43-101.

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

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

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

- Standards are samples of known composition inserted into sample batches to independently test the accuracy of an analytical procedure. They are acquired from a known and trusted commercial source. Standards are selected to fit the grade distribution identified in the mineralization.
- Blanks consist of material that is predetermined to be free of elements of economic interest to monitor for potential sample contamination during analytical procedures at the laboratory.
- Duplicate samples are submitted to assess assay precision (repeatability) and mineralization homogeneity. Duplicates can be submitted from all stages of sample preparation with the expectation that better precision is demonstrated by duplicates further along in the preparation process.

QC samples were inserted into the sample batches sent to the laboratory. Inserts included blank samples and standards.

11.4.1 Standards

Certified material is used by XXIX. The certified standards are purchased from OREAS, a well-known provider of such material. Certified materials used were 166, 502c, 504c, and 505.

One certified sample was inserted for every 50 samples.

All samples during the latest drill program returned values within 3SD for copper. All samples for gold and silver also returned values within 3SD, with the exception of two samples for gold and six for silver. Samples that returned values outside of the 3SD range were close to 3SD. These values are judged acceptable.

11.4.2 Duplicates

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

Several duplicates are used in the mineral industry, these being core duplicates (half core or quarter core), coarse duplicates (rejects and preparation duplicates), pulp duplicates (second split of final pulp prior to analysis) and field duplicates (double samples collected in the field, where applicable).

XXIX did not include core duplicates in their QA/QC program. The QP recommends adding core duplicates in future drilling programs and sending 5% of the 2024-2025 samples for check assay.

11.4.3 Blanks

Tested blank material, selected due to its depleted base metal geochemical signature, is used by XXIX. The blank reference material was garden rocks purchased at a local store. One blank sample was inserted for every 50 samples.

All blank samples during the latest drill program returned values of <0.005% Cu, <0.005 g/t Au, and <0.2 g/t Ag. These values are judged acceptable.

11.5 Sample Preparation Conclusions

The sample preparation, analytical procedures, and security of the samples during these procedures followed industry best practices but could be improved, mainly by inserting more blanks, more CRMs, and adding a field duplicate program. Sufficient efforts were made to identify items that were out of specification.

The QA/QC data indicate that the overall assay results of the issuer's drill program are valid and can be relied upon for the purpose of this report.

It is the QP's opinion that the sample preparation, security and analytical procedures are adequate and follow best practices.

12 DATA VERIFICATION

12.1 Drilling Database Summary

The MRE in this report is based on 21,918 drill holes for 1,525,073 m and 479,242 samples. The drill hole database includes the results of recent drilling (2002 to 2025) of 73,227 m in 382 drill holes (Ex-In, PowerOre, QC Copper and Gold, XXIX) and incorporates historical drill holes (1930 to 1990) for 1,451,846 m in 21,536 drill holes (Opémiska Copper Mines, Falconbridge, Minnova).

For this MRE, the QPs performed a basic validation of the entire database. XXIX provided all data in UTM NAD 83. The database close-out date for the resource estimate is May 16, 2025.

12.2 Historical Drill Hole Database

The historical information used in this report was taken mainly from reports produced before the implementation of NI 43-101. In most cases, little or no information about sample preparation, analytical, or security procedures is available. However, the QP assumes that exploration activities conducted by previous companies satisfied prevailing industry standards at the time. The QP consulted previous independent validation reports of the historical database and performed a series of additional validations over the course of the current mineral resource estimation.

12.3 Recent Database

12.3.1 Site Visit

QP Pierre-Luc Richard of PLR Resources Inc. (PLR) visited the project on May 1, 2025 during the course of this mandate. The site visit included a visual inspection of core, as well as a field tour (Figure 12-1) and discussions of the geological interpretations with geologists and geotechnicians employed by XXIX.

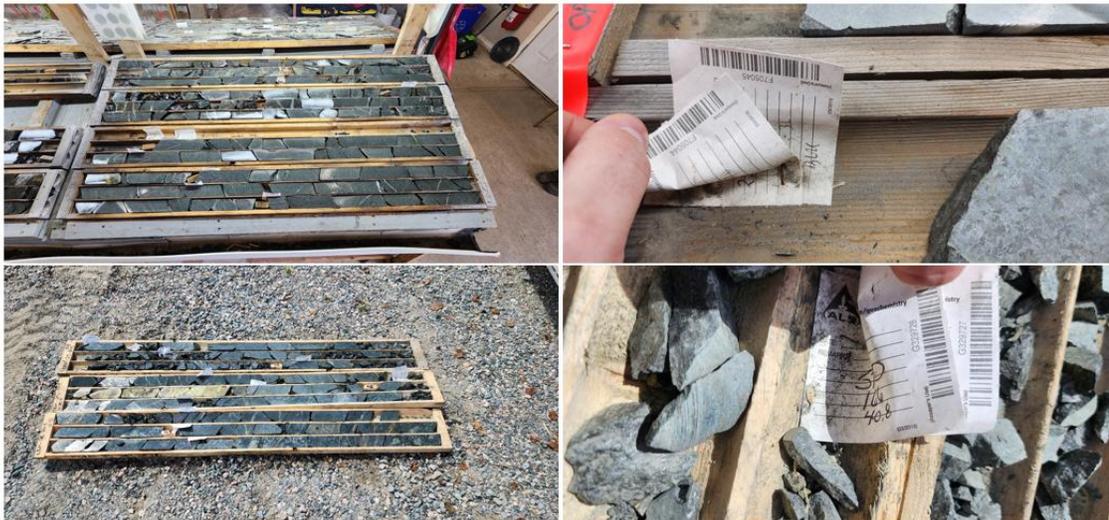
The site visit also included a review of sampling and assaying procedures, the QA/QC program, downhole survey methodologies, and the descriptions (logging) of lithologies, alteration and structures (Figure 12-2). Selected drill collars in the field were also validated using a handheld GPS (Figure 12-3).

Figure 12-1: Photos taken by the QP during the Site Visit (Mineralization on Surface and Historical Tailings)



Source: PLR, 2025.

Figure 12-2: Core Review in the Core Logging Facility and Exterior Core Storage Facility



Source: PLR, 2025.

Figure 12-3: Drill Collar Validation during the Site Visit



Source: PLR, 2025.

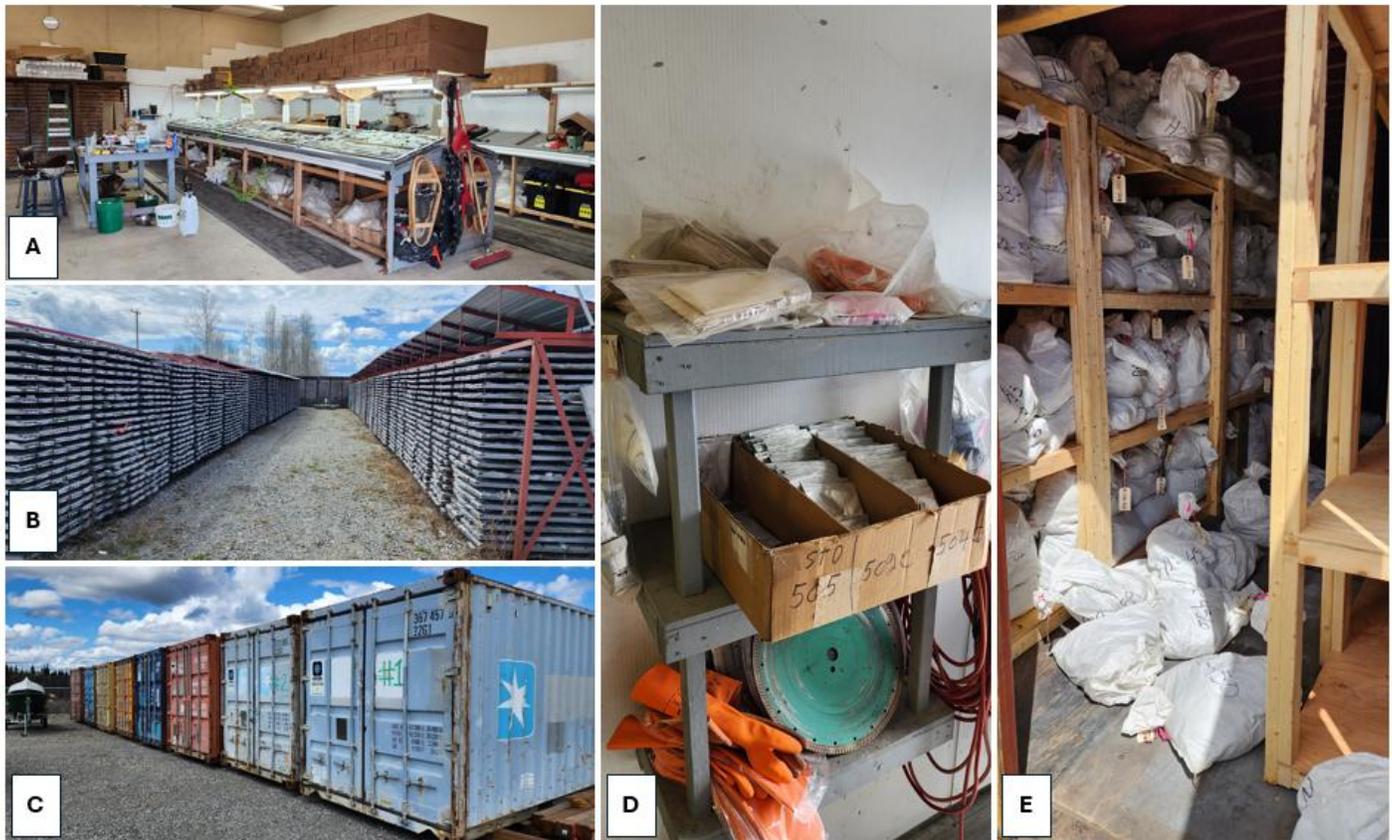
12.3.2 Drilling and Sampling Procedures

XXIX procedures are described in Section 10, Drilling, and Section 11, Sample Preparation, Analyses, and Security. Discussions with on-site geologists confirmed that said procedures were adequately applied.

The QP reviewed several sections of mineralized core while visiting the project. All core boxes were labelled and properly stored inside the core shack. The QP could also access the outdoor historical core storage facility during the site visit. In the reviewed core boxes, sample tags were present, and it was possible to validate sample numbers and confirm the presence of mineralization in witness half-core samples from the mineralized zones (Figure 12-2).

Drilling was not underway during Mr. Richard's site visit, but XXIX's employees involved during the drilling programs explained the entire path of the drill core, from the drill rig to the logging and sampling facility and finally to the laboratory (Figure 12-4).

Figure 12-4: Photographs of Core Logging and Storage Facilities



Notes: A. Core logging facility. B. Exterior core storage facility. C. Rejects and pulps storage. D. Sample preparation room. E. Stored pulps and rejects. Source: PLR, 2025.

12.3.3 Assay Validation

The issuer's procedures are described in Sections 10 and 11 of this report. Discussions held with on-site geologists confirmed that the procedures were adequately applied.

12.3.4 QA/QC Validation

The QP reviewed the QA/QC reports and found no issues.

12.4 Metallurgical Testwork Data Validation

The QP reviewed the metallurgical results that impact the metallurgical recovery model. In the opinion of the QP, the data and assumptions used to estimate the metallurgical recovery model for the overall financial model are sufficiently reliable for those purposes.

12.5 Conclusion

The QP is of the opinion that the drilling protocols in place are adequate. The database for the project is of good overall quality. Minor variations have been noted during the validation process but have no material impact on the current MRE. In the QP's opinion, the database is suitable for mineral resource estimation.

13 MINERAL PROCESSING AND METALLURGICAL TESTING

13.1 Introduction

The mineralization at Opémiska consists largely of chalcopyrite-bearing quartz veins that occupy fracture systems in the folded and faulted gabbroic portions of two conformable, regionally extensive, layered Archean ultramafic-mafic sills. The mineralized veins are generally restricted to the fracture system and in lower grade halos around the main fractures/veins. The minerals of economic interest are iron sulphides (chalcopyrite, pyrite and arsenopyrite) and oxides which are associated with copper, gold and silver. The Opémiska deposit has four identified continuous zones of mineralization, namely Perry, Springer, Springer-Gold, and Saddle.

The Opémiska deposit encompasses the historical Springer and Perry mines, operated by various entities from 1954 to 1991. The historical operations demonstrated amenability to recovery via conventional flotation processes, with reported copper recoveries consistently above 95%.

13.2 Metallurgical Testwork

Metallurgical testing was completed in 2001 by Ex-In at McGill University on historical tailings from the Opémiska deposit. The testwork program was limited to evaluating copper and gold recoveries with gravity concentration and flotation. The results of this testwork program were not considered for this assessment.

Recent metallurgical testing was completed by QC Copper and Gold (now XXIX Metal Corporation) at SGS (Quebec City) in 2023. The objective of the testwork program was to provide sufficient metallurgical data to support the design of an optimal flotation process to recover a copper concentrate with by-products of gold and silver, amenable to smelting by others.

A summary of metallurgical testwork is provided in Table 13-1.

Table 13-1: Metallurgical Testwork Summary

Year	Laboratory/Location	Testwork Performed
2001	McGill University, Montréal	Gravity recovery, flotation
2023	SGS Natural Resources, Québec City	Head analysis, mineralogy, SMC testing, rougher kinetics, open circuit cleaner

13.2.1 Legacy Testwork

The 2001 McGill University testwork program was conducted on three samples from historical tailings from the Opémiska deposit. The test program investigated the amenability of the material to gravity concentration, flotation and the impact of slurry pH on metal recoveries during flotation. The flotation tests were performed with coarse grinding to refresh particle surfaces. The results, summarized in Table 13-2, show metal recoveries to concentrate

below 57% and concentrates grades less than 5% Cu. A single gravity test was performed on a composite of the samples, using a Kelson concentrator. The results, shown on Table 13-3, demonstrated gold recoveries to concentrate below 5%.

The results of this testwork program were not considered in this analysis the samples were sourced from tailings and therefore not representative of the mineralized material to be processed. The results also did not demonstrate the potential to recover either a saleable concentrate by flotation or economic recovery of gold to concentrate via gravity separation.

Table 13-2: 2001 McGill University Program Flotation Testwork Summary

Sample ID	Feed Grade		pH	Reagents	Concentrate Grade		Recovery (%)		
	Au (g/t)	Cu (%)			Au (g/t)	Cu (%)	Mass	Au	Cu
608459	0.42	0.65	7 -7.4	Isobutyl xanthate	2.07	4.15	5.50	26.8	35.0
	0.43	0.65	10-10.3	Amyl xanthate, lime	2.20	2.63	7.50	38.9	30.3
608456	0.44	0.84	7 -7.4	Isobutyl xanthate	1.50	3.08	10.0	34.3	36.7
	0.50	0.86	10-10.3	Amyl xanthate, lime	2.27	3.04	12.6	56.6	44.3
608458	0.49	0.80	7 -7.4	Isobutyl xanthate	1.80	2.28	4.20	15.6	12.1
	0.574	0.80	10-10.3	Amyl xanthate, lime	5.90	3.88	4.20	43.2	20.5

Table 13-3: 2001 McGill University Program Gravity Testwork Summary

Size Class (µm)	Head Feed		Concentrate		Recovery	
	Grade (g/t Au)	Au Distribution (%)	Grade (g/t Au)	Au Distribution (%)	Mass (%)	Gold (%)
+212	0.47	34.5	14.3	46.0	0.38	4.0
75-212	0.50	45.6	6.1	35.8	0.56	3.1
25-75	0.48	11.0	10.4	9.1	0.33	0.8
-25	0.79	8.9	242	9.1	0.03	0.8
Total	0.50	100.0	10.0	100.0	0.44	8.7

13.2.2 XXIX Metal Corporation Testwork

The primary purpose of the 2023 SGS testwork program was to evaluate the Opémiska deposit mineralized material characteristics, environmental properties and metallurgical performance with conventional flotation processes. The testwork program, conducted at SGS in Quebec City, was commissioned by G Mining Services Inc. under the direction of QC Copper and Gold (now XXIX Metal Corporation).

The testwork was conducted on a single composite sample, referred to as composite 1, from the Opémiska deposit. This composite was made up of one-quarter core selected from intervals weighted proportionally to the Opémiska

deposit mineralized domains, and intersecting all lithologies. The scope of work included head grade characterization, mineralogical analysis, gold department, comminution (SMC), flotation and environmental testing. The results of this testwork program were used to inform the PEA design.

13.2.2.1 Head Grade Analysis

Composite 1 and samples from the Opémiska mineralized domains were submitted for screen metallics, chemical assay analysis, and an inductively coupled plasma (ICP) multi-element scan to characterize the gold, copper, iron, sulphide, and arsenic content. Gold and silver assays on composite 1 were performed by screen metallics. Silver and arsenic assays were not conducted on the domain samples.

The results are summarized in Table 13-4. Composite 1 assay values were 0.81% Cu, 1.23 g/t Au and <5 g/t Ag. Assay values for the domain samples were found to be in the range of 0.49% to 1.28% Cu and 0.21 to 0.62 g/t Au.

Table 13-4: Head Grade Characterization Summary

Sample ID	Cu (%)	Au (g/t)	Ag (g/t)	Fe (%)	As (g/t)	S (%)
Composite 1	0.81	1.23	5.00	12.7	210	1.82
Saddle	0.77	0.21		12.6		1.55
Perry	1.28	0.09		11.3		3.35
Springer	0.49	0.30		14.4		1.02
Springer-Gold	0.50	0.62		10.8		1.17

13.2.2.2 Mineralogical Analysis

Mineralogical characterization of composite 1 was completed at SGS in Lakefield using quantitative evaluation of minerals by scanning electron microscopy (QEMSCAN) and X-ray diffraction analysis (XRD). The mineralogy program was completed to understand the distribution of key minerals, clay species, gold department, including gold mineral speciation, grain size, exposure, and liberation. Samples from the Opémiska mineralized domains were submitted for mineralogical analysis by QEMSCAN using bulk material analysis (BMA). The mineralogical analysis was conducted at 80% passing (k_{80}) of 105 μm .

The modal distribution of composite 1 and the domain samples is shown in Table 13-5. The main minerals in composite 1 were identified (in order of decreasing abundance) to be feldspars, amphibole, pyroxene, stilpnomelane, quartz, iron oxides, chlorite, micas, and calcite. Minerals of economic interest are chalcopyrite (2.5%), pyrite (1.8%), and arsenopyrite (0.06%), all of which may host gold. Pyrrhotite was not observed, but some minor content has been reported from drilling programs. The major gangue minerals are feldspars, pyroxenes, iron oxides, quartz, calcite, and chlorite. The main clay mineral is chlorite, along with minor amounts of illite, and trace amounts of montmorillonite.

Table 13-5: Modal Mineral Distribution

Mineral	Mineral Mass (%)				
	Composite 1	Saddle	Perry	Springer	Springer Gold
Pyrite	1.79	1.37	3.04	0.91	1.17
Pyrrhotite	0.00	0.00	0.00	0.00	0.00
Chalcopyrite	2.49	2.26	3.79	1.47	1.55
Sphalerite	0.03	0.01	0.04	0.01	0.03
Galena	0.00	0.00	0.00	0.00	0.01
Arsenopyrite	0.06	0.00	0.00	0.00	0.05
Molybdenite	0.00	0.01	0.01	0.01	0.00
Other Sulphides	0.00	0.01	0.00	0.00	0.00
Iron-Oxides	4.62	2.10	1.96	6.06	0.39
Ilmenite	0.89	0.88	0.90	1.52	0.75
Rutile	0.71	0.57	0.50	0.34	1.16
Titanite/sphene	2.11	3.30	4.18	2.12	3.10
Quartz	7.71	1.46	8.56	3.50	10.30
Plagioclase	35.90	35.30	42.90	42.20	37.90
Potassium-Feldspars	4.83	1.26	3.46	2.41	2.76
Amphibole/Pyrox.	15.40	14.70	7.13	12.00	11.60
Micas	2.22	6.40	3.20	3.99	2.47
Stilpnomelane	8.01	12.70	7.87	11.80	8.84
Clays	0.25	0.88	0.19	0.17	0.31
Chlorite	9.40	11.00	9.05	7.26	13.70
Epidote	0.87	2.07	0.51	1.44	0.47
Other Silicates	0.02	0.01	0.01	0.01	0.03
Calcite	2.10	2.76	2.07	2.12	2.83
Ankerite	0.02	0.02	0.01	0.02	0.01
Apatite	0.41	0.41	0.49	0.48	0.49
Other	0.16	0.58	0.14	0.14	0.13
Total	100.00	100.00	100.00	100.00	100.00

Key composite 1 QEMSCAN mineralogical assays underwent assay reconciliation with chemical assays. The assays are provided in Table 13-6. Both assaying techniques yielded similar results, validating the QEMSCAN analysis.

The liberation and association characteristics of the key minerals in composite 1 is summarized on Table 13-7. Pure, free and liberated chalcopyrite, pyrite and arsenopyrite account for 63.9, 85.8 and 87.3% respectively of the total minerals mass at the target primary grind size (k_{80}) of 105 μm . The D_{50} grain size for the minerals of interest is 35 to 53 μm , indicating full exposure/liberation at the target regrind size (k_{80}) of 30 μm .

Table 13-6: Composite 1 QEMSCAN Calculated and Direct Assay Reconciliation

Element	Assay Method	
	QEMSCAN	Chemical
Al	6.54	6.08
Ca	3.82	3.29
Cu	0.86	0.81
Fe	13.00	12.70
K	1.06	0.84
Mg	2.11	1.70
Mn	0.05	0.11
S	1.85	1.82
Si	23.60	22.50
Ti	0.87	0.86

Table 13-7: Liberation and Association of Composite 1 Key Minerals

Liberation/Association Characteristic	Normalized Mineral Mass (%)		
	Chalcopyrite	Pyrite	Arsenopyrite
Pure	44.10	45.20	69.20
Free	11.50	27.90	18.10
Liberated	8.33	12.70	0.00
Other Sulphides	0.49	0.28	3.17
Mica/Chlorite/Clays/Stilpnomelane	5.73	0.66	0.45
Amphiboles/Pyroxenes	1.07	2.09	1.81
Quartz/Feldspars	5.48	1.32	3.62
Carbonates	0.25	0.02	0.00
Oxides	0.35	0.27	0.45
Other Minerals	0.25	0.00	0.00
Complex Associations	22.50	9.56	3.17

13.2.2.3 Gold Department Analysis

A gold department program was conducted on composite 1 to determine the gold mineral type, abundance and association. Gold mineral scanning was performed by optical microscopy and a Tescan integrated mineral analyser (TIMA-X) technology. Electrum is the dominant gold mineral (76.4%), followed by native gold (21.3%) and kustelite in trace amounts (2.2%).

Heavy liquid separation (HLS) at a specific gravity (SG) of 2.85 was used to obtain a sink product and a float product. The sink products were submitted for superpanning (SP) to further upgrade the gold by gravity. The results are summarized in Table 13-8: 92.1% of the gold was concentrated in the HLS sink product with a mass recovery of 30.5%. The sulphides fraction of the HLS sinks contained 54.9% of the gold in 0.50% of the total mass.

Table 13-8: Gold Distribution in HLS and SP Products

Fraction	Mass (%)	Grade (g/t Au)	Gold Distribution (%)
Feed	100	1.23	100
Total HLS Sink	30.5	3.71	92.1
HLS Sink Superpan Sulphides	0.50	13.9	54.9
HLS Float	69.5	0.14	7.91

The liberation and association characteristics of the gold minerals in composite 1 is summarized in Table 13-9. Pure, free and liberated gold minerals account for 71% of the total gold minerals mass at the target primary grind size (k_{80}) of 105 μm . The non-liberated gold is associated with sulphides (17.2%) and complex particles (11.7%), indicating a maximum recovery of gold to concentrate approximately 90%.

Approximately 40.1% of gold minerals (by mass) are greater than 75 μm , 20.2% between 30 and 75 μm , 18% between 20 and 30 μm , and 21.2% less than 20 μm . The pure, free and liberated gold in the +75 μm fraction is 92.9%, indicating a micro nugget effect. This liberation analysis and HLS results indicate the potential for gravity recoverable gold.

Table 13-9: Composite 1 Gold Minerals Liberation and Association

Liberation/Association Characteristic	Normalized Mineral Mass (%), by Size Class mm					
	Overall	+75	75 - 30	50 - 30	30 - 20	-20
Pure	4.73	0.00	0.00	8.53	2.49	69.6
Free	28.90	26.60	0.00	12.50	73.70	13.20
Liberated	37.50	66.30	20.30	15.50	2.40	9.21
Pyrite	7.83	5.74	27.10	5.59	7.28	0.72
Chalcopyrite	2.05	0.00	0.00	9.08	1.89	1.33
Arsenopyrite	5.83	0.58	45.20	7.88	4.12	1.64
Chalcopyrite/Pyrite	1.30	0.01	1.17	3.05	0.79	0.16
Chalcopyrite/Arsenopyrite	0.11	0.02	0.04	0.39	0.15	0.00
Other Sulphides	0.06	0.00	0.00	0.26	0.00	0.15
Quartz/Feldspars	0.01	0.00	0.00	0.00	0.16	0.00
Complex Associations	11.70	0.81	6.25	37.20	6.97	3.98

13.2.2.4 Grindability Testing

An SMC test was performed on composite 1 and the results are presented in Table 13-10. Composite 1 is classified as hard, with the Axb and SAG circuit specific energy (SCSE) at 92% and 94%, respectively, of the hardness percentile in the JK database. No Bond ball mill (BW_i), rod mill (RW_i), or crusher work index (CW_i) testing was conducted.

Table 13-10: Composite 1 Comminution Test Results

Parameter	Unit	Value
Specific Gravity (SG)	-	2.92
Axb	-	26.8
Drop-Weight Index (DWI)	kWh/m ³	10.9
SAG Circuit Specific Energy (SCSE)	kWh/t	12.7
t _a	-	0.24

13.2.2.5 Rougher Kinetics Testing

Six rougher kinetic tests were conducted on composite 1 to examine the effects of collector dosage and type, primary grind size, and pH level. The test conditions and results, at six- and ten-minute flotation times, are summarized in Table 13-11. The sulphide mineral collectors tested were potassium amyl xanthate (PAX), sodium diisobutylidithiophosphinate (Aerophine 3418A) and thionocarbonate (Aero 3894). Lime and methyl isobutyl carbinol (MIBC) were used as pH modifier and frother, respectively, in all tests.

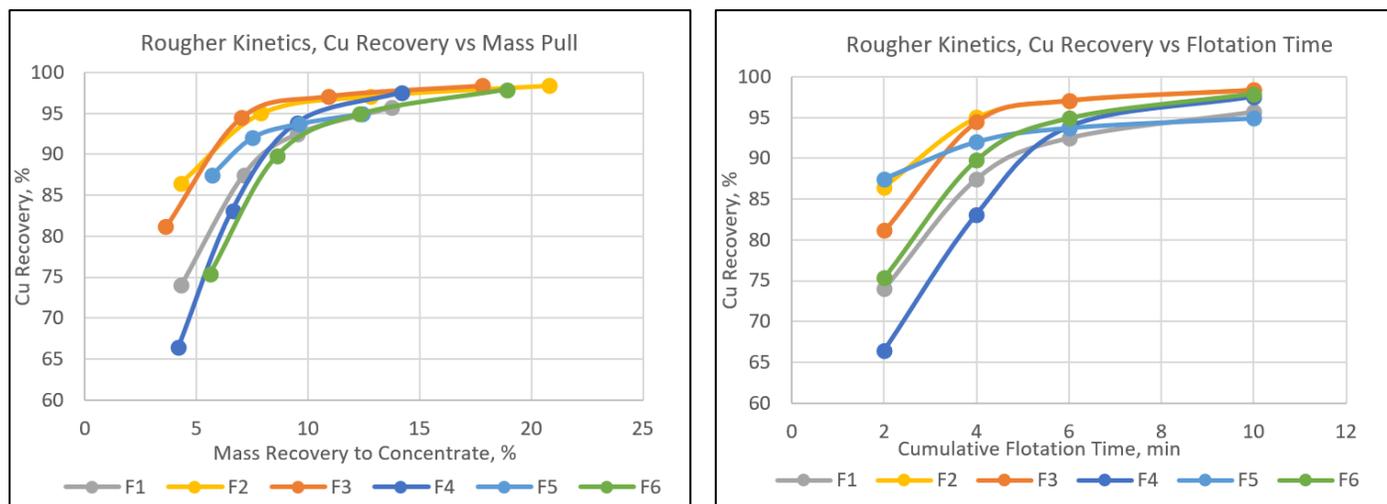
The recovery of copper to rougher concentrate versus mass recovery and flotation time is illustrated in Figure 13-1. The tests yielded copper recoveries to rougher concentrate in the range of 96% to 98% at mass pulls of 10% to 15%. Gold recoveries were in the range of 93% to 96% and silver in the range of 85% to 90%. Increasing the flotation time from 6 to 10 minutes did not result in significant metal recovery to concentrate increases.

Test F3 displayed superior kinetics, mass, and metal recovery relationships; therefore, the test conditions were selected as the optimum conditions for cleaner flotation testing as well as for the process design.

Table 13-11: Composite 1 Rougher Kinetics Tests Summary

Test ID	Calculated Head Grade, % or g/t			Primary Grind Target, k ₈₀ (µm)	Reagent, Dosage (g/t)	pH	6 min Flotation Time Recoveries, % w/w				10 min Flotation Time Recoveries, % w/w			
	Cu	Au	Ag				Mass	Cu	Au	Ag	Mass	Cu	Au	Ag
F1	0.85	1.29	4.41	105	PAX, 10 +20	10.5	9.5	92.5	89.3	85.0	13.7	95.7	94.7	90.2
F2	0.86	1.14	4.24	105	A-3418A, 30	10.5	12.8	97.1	92.5	88.0	20.8	98.4	95.2	90.7
F3	0.83	1.30	4.25	105	A-3894, 30	10.5	10.9	97.1	90.0	86.9	17.8	98.4	93.0	90.3
F4	0.82	1.22	4.38	105	PAX, 20+10	10.5	9.5	93.9	90.7	84.3	14.2	97.6	95.8	90.2
F5	0.87	1.11	4.25	150	A-3894, 30	10.5	9.6	93.7	84.5	-	12.4	94.9	86.6	-
F6	0.82	1.13	4.27	105	A-3894, 30	9.2	12.3	94.9	90.2	85.3	18.9	97.9	95.7	90.5

Figure 13-1: Composite 1 Rougher Kinetic Testing, Copper Recovery vs. Mass Pull & Flotation Time



Source: Ausenco 2025.

13.2.2.6 Cleaner Flotation Conditions Testing

Five batch, open circuit cleaner flotation tests were conducted with composite 1 to examine the effect of rougher concentrate regrind and number of cleaner stages. The primary grind (105 μm), reagents used (lime, MIBC and A-3894) and rougher flotation slurry pH (10.5) were the optimum conditions identified from rougher kinetics testing, as discussed in Section 13.2.2.5. The cleaner flotation slurry pH was maintained between 10.5 and 11.5 and the collector (A-3894) was dosed at 10 g/t in all tests. Frother (MIBC) at 5 g/t was dosed per cleaning stage.

The rougher, cleaner 1 and scavenger, and cleaner 2 flotation stage results are summarized in Table 13-12. The graded recovery curves are shown in Figure 13-2. The following can be observed:

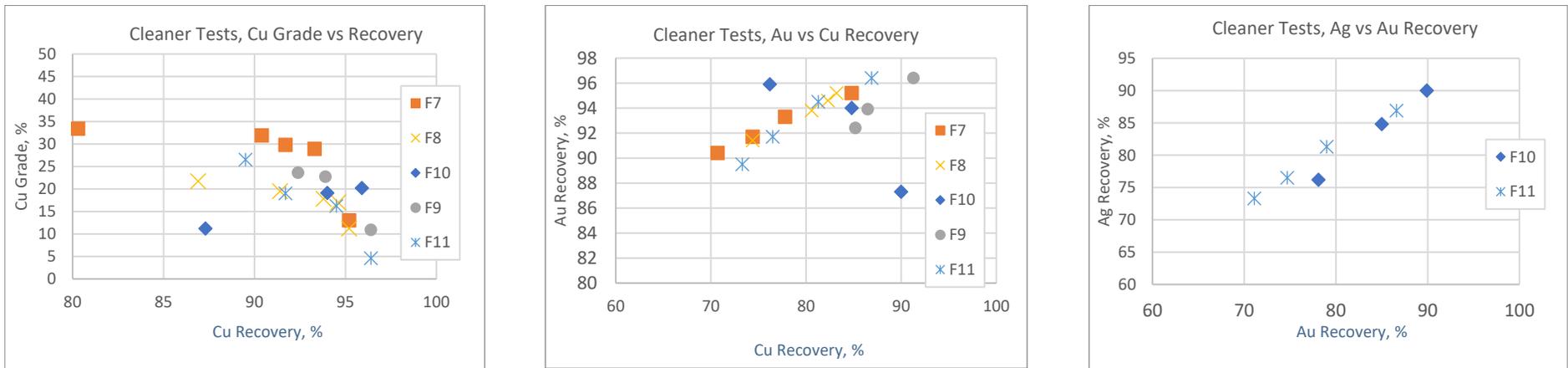
- Copper recovery decreases as grade increases. A 90% to 95% recovery equates to a concentrate grade of approximately 20%. Test F7 showed high grades and recoveries that could not be duplicated in subsequent tests. Excluding these results, a 20% concentrate grade would equate to a recovery of approximately 92%.
- There is a strong association between gold and copper recoveries, and subsequently gold and silver recoveries. It is therefore possible to calculate gold and silver recoveries to concentrate based on copper content in the concentrate.

The F11 cleaner 1 tests result (19% Cu grade, 92% Cu and 3.9% mass recoveries) were thus deemed to be the most favourable balance between metal recovery and concentrate grade. However, it should be noted that metal recoveries and concentrate grades would likely increase with the recirculation of cleaner 2 tails in locked cycle testing. The flotation concentrator flowsheet is therefore designed with two cleaning stages, and the recirculation of cleaner 2 tails.

Table 13-12: Composite 1 Batch Cleaner Test Summary

Test ID	Calculated Head Grade, % or g/t			Regrind K_{80} (mm)	Rougher Concentrate Recoveries, % w/w				Cleaner 1 Concentrate Recoveries, % w/w				Cleaner 2 Concentrate Recoveries, % w/w				Cu Grade, %	
	Cu	Au	Ag		Mass	Cu	Au	Ag	Mass	Cu	Au	Ag	Mass	Cu	Au	Ag	Clnr. 1	Clnr. 2
F7	0.85	0.99	-	22	6.2	95.2	84.8	-	2.6	91.7	74.4	-	2.4	90.4	70.7	-	29.8	31.2
F8	0.82	1.16	-	-	7.0	95.2	83.2	-	4.3	93.8	80.6	-	3.8	91.4	74.4	-	17.8	21.7
F9	0.81	1.06	-	21	7.2	96.4	91.3	-	3.2	92.4	85.2	-	-	-	-	-	23.6	-
F10	0.83	1.02	4.60	22	7.1	95.9	90.0	89.9	3.6	87.3	76.2	78.1	-	-	-	-	20.2	-
F11	0.81	1.01	4.25	28	17	96.4	86.9	86.6	3.9	91.7	76.5	74.7	2.7	89.5	73.3	71.1	19.0	26.5

Figure 13-2: Composite 1 Batch Cleaner Test Grade and Recovery Curves



Source: Ausenco, 2025.

13.3 Historical Operation Data

The historical Springer and Perry mines were operated by various entities from 1954 to 1991. The historical operations were underground mines, focusing on the high-grade quartz veins, with a reported design mill feed rate of 0.7 Mt/a, peaking at approximately 1.1 Mt/a. The process included three-stage crushing, ball mill grinding, and a conventional flotation circuit. Scavenger flotation concentrates and cleaner tailings were subject to regrinding, and the copper-gold concentrate was dewatered and shipped via rail to off-site smelters. A gravity concentration circuit was reportedly added in later years of the operation to recover free coarse-grained gold in advance of flotation.

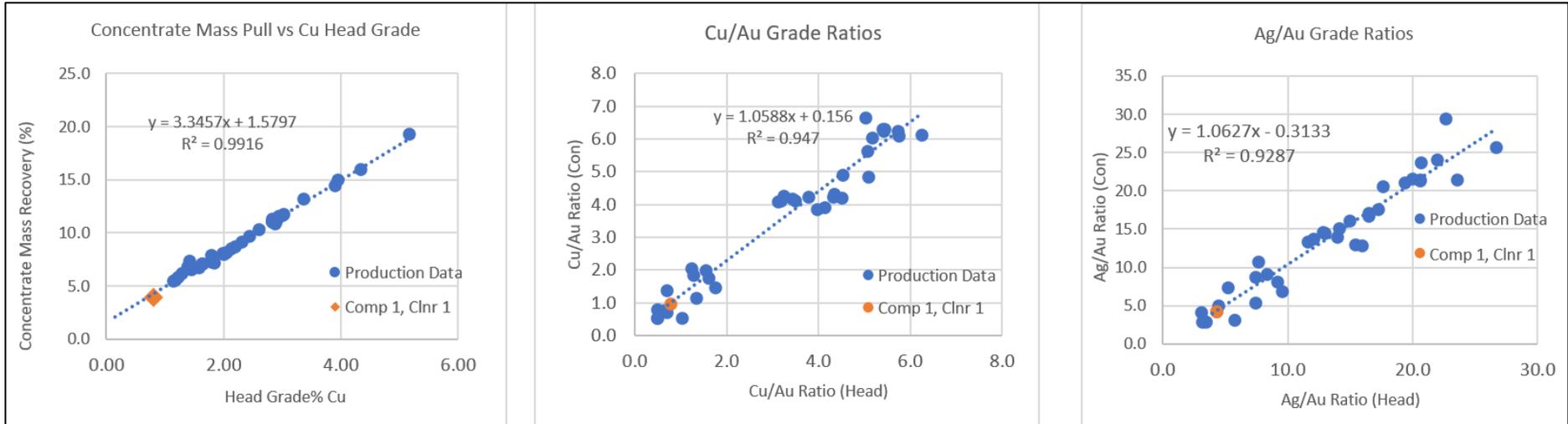
Table 13-13 shows a summary of the production data. Copper recoveries were consistently high, above 95% over most of the historical production period, despite decreasing copper head feed grades in later years. The concentrate grade was maintained between 20 and 25% over the life of mine, with lower copper head feed grades associated with the lower end of the range.

Table 13-13: Springer and Perry Mines Historical Production Data Summary

Years	Average Annual Production (Mt/a)	Head Grade, % or g/t			Recovery, %			Concentrate Grade, Cu %
		Cu	Au	Ag	Cu	Au	Ag	
1954 - 1960	0.33	3.71	1.07	14.3	93.3	81.3	80.1	23.2
1961 - 1965	0.68	2.88	0.68	10.8	95.7	81.4	82.3	24.6
1966 - 1970	0.78	2.76	0.49	9.63	96.1	82.3	84.4	24.5
1971 - 1975	1.03	2.11	0.40	9.25	95.2	79.6	84.6	24.0
1976 - 1980	1.05	1.88	1.02	10.4	95.8	83.7	81.6	23.7
1981 - 1991	0.87	1.45	1.55	8.47	95.7	86.9	79.6	21.4

Figure 13-3 shows production data regressions on concentrate mass pull versus head grade, the metal head grade, and concentrate relationships. Composite 1 F11 cleaner 1 results have been overlaid to demonstrate the strong alignment of the metallurgical testwork to the production data. Based on these regressions, it can be inferred that the production data, also demonstrating the similar metal grade and recovery relationships discussed in Section 13.2.2.6, provides a reasonable basis for the interpretation of the expected metallurgical performance.

Figure 13-3: Production Data Comparison to Composite 1 Testwork

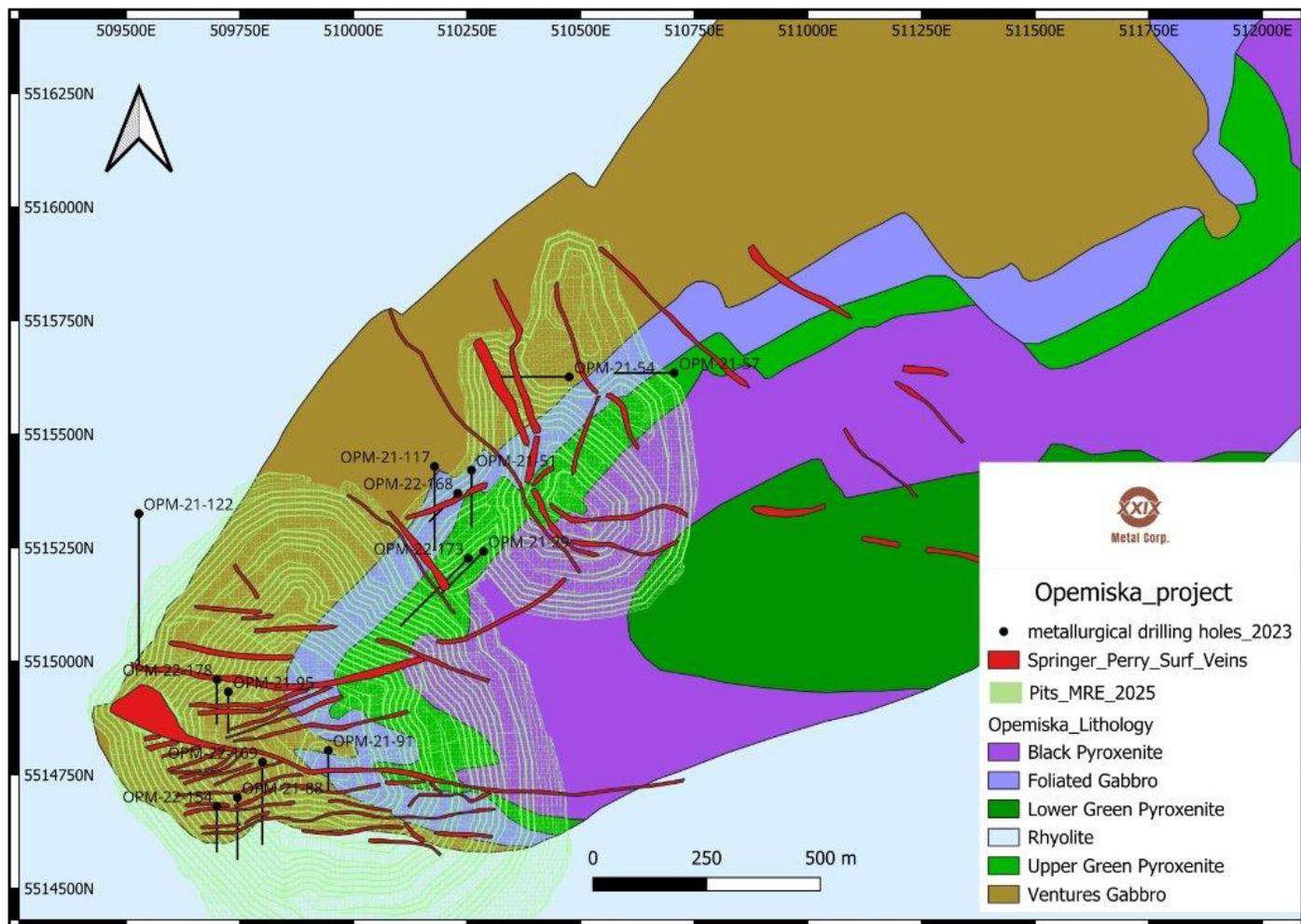


Source: Ausenco, 2025.

13.4 Metallurgical Variability

The material for composite 1 in the 2023 SGS testwork program was made up of one-quarter core selected from intervals obtained from drill holes across the Opémiska deposit mineralized domains intersecting all lithologies and mineralized area (Figure 13-4). The composite had an assayed copper grade of 0.81%, which is consistent with the expected grades in the PEA mine plan payback period. The composite 1 total sample mass was 80 kg, and its constituent material was weighted proportionally to the Opémiska deposit mineral domains (Table 13-14).

Figure 13-4: Drill Hole Locations



Source: XXIX, 2025.

Table 13-14: Composite 1 Domain, Hole IDs and Constituent Mass

Domain and Proportion of Deposit	Drill Hole IDs	Mass (kg)/% of Overall Sample
Saddle, 5%	OPM-21-29	8
	OPM-22-173	12
	OPM-22-168	8
	OPM-21-117	28
	OPM-21-51	8
	Subtotal	12/5%
Perry, 25%	OPM-21-57	4
	OPM-21-54	10
	OPM-21-54	6
	Subtotal	20/25%
Springer, 35%	OPM-22-179	14
	OPM-21-95	0.5
	OPM-21-95	0.5
	OPM-21-91	1.0
	OPM-21-122	12
	Subtotal	28/35%
Springer-Gold	OPM-22-154	8
	OPM-21-88	12
	OPM-22-169	8
	Subtotal	28/35%

13.5 Deleterious Elements

Samples of material from the mineralized domains were submitted for a multi-element chemical analysis and the results are shown in Table 13-15. Typical deleterious elements, including arsenic, mercury and lead, are present in low concentrations, with mercury below detectable limits, lead ranging from 2.6 to 17 ppm, and arsenic at 6.2 to 230 ppm. The higher concentrations of lead and arsenic are observed in the springer-gold domain, consistent with the mineralogical analysis, which showed higher galena and arsenopyrite in this domain.

There were no multi-element chemical assays conducted on concentrates. However, the concentrate is not expected to contain significant quantities of deleterious elements based on typical concentration ratios.

Table 13-15: Opémiska Domain Multi Element Chemical Analysis

Element	Symbol	Content by Domain, ppm			
		Saddle	Perry	Springer	Springer-Gold
Aluminum	Al	8,700	8,200	8,700	13,000
Arsenic	As	6.2	18	6.4	230
Barium	Ba	38	27	29	16
Beryllium	Be	0.12	0.19	0.20	0.19
Bismuth	Bi	0.40	0.59	0.25	0.23
Boron	B	2	2	2	4
Calcium	Ca	9,700	9,000	8,700	13,000
Cadmium	Cd	0.37	1.8	0.26	0.36
Cobalt	Co	37	52	32	50
Chromium	Cr	10	21	11	17
Copper	Cu	5,600	9,700	3,500	4,500
Iron	Fe	46,000	54,000	54,000	69,000
Lead	Pb	2.6	7.3	4.0	17
Lithium	Li	5	7	7	8
Magnesium	Mg	5,700	5,300	4,900	10,000
Manganese	Mn	310	330	370	610
Mercury	Hg	< 0.05	< 0.05	< 0.05	< 0.05
Molybdenum	Mo	21	9.8	11	7.0
Nickel	Ni	33	27	14	13
Phosphorus	P	240	400	310	420
Potassium	K	3,000	2,300	2,700	,1600
Antimony	Sb	< 0.8	< 0.8	< 0.8	< 0.8
Selenium	Se	1.5	3.1	< 0.7	2.4
Silver	Ag	2.5	7.6	1.7	1.9
Sodium	Na	610	640	620	640
Strontium	Sr	16	24	20	19
Tin	Sn	< 5	< 5	< 5	< 5
Tellurium	Te	< 1	< 1	< 1	< 1
Titanium	Ti	310	370	280	1100
Thorium	Th	0.85	1.1	0.93	0.79
Thallium	Tl	0.15	0.16	0.12	0.07
Uranium	U	0.49	0.75	3.7	0.21
Vanadium	V	110	45	49	81
Tungsten	W	0.39	41	74	9.1
Yttrium	Y	4.6	7.2	6.7	6.6
Zinc	Zn	77	230	67	140

13.6 Recovery Estimates

The approach to the recovery model integrates the 2023 SGS testwork results (Section 13.2) and the historical operational data (Section 13.3).

Copper recoveries are maintained at 92% over the life of mine to mirror the testwork outcomes. No recovery variability is assumed based on the production data, which showed copper recoveries to be independent of head feed grades. A constant concentrate grade of 20% is assumed, which is consistent with the testwork results at 92% recovery, as well as the production data concentrate mass pull and grade relationships.

The constant copper grade and recovery allow the concentrate mass to be calculated from copper head feed grades. The concentrate gold and silver grades can then be expressed based on the production data regressions of metal concentrate versus head feed grades (Figure 13-3). The gold and silver recoveries are then simply a function of the metal content in the known concentrate mass versus the metal content in the mill feed. The relevant mathematic expressions can be summarized as follows:

- Concentrate mass pull (fraction) = $92\% * \text{Cu head (\%)} / 20\%$
- Concentrate gold grade (g/t) = $20\% / (1.0588 * \text{Cu head (\%)} / \text{Au head (g/t)} + 0.156)$
- Concentrate silver grade (g/t) = $\text{Au concentrate (g/t)} * (1.0627 * \text{Au head (g/t)} / \text{Ag head (g/t)} - 0.3133)$
- Gold recovery (%) = $\text{Au concentrate (g/t)} * \text{mass pull (fraction)} / \text{Au head (g/t)}$
- Silver recovery (%) = $\text{Ag concentrate (g/t)} * \text{mass pull} / \text{Ag head (g/t)}$

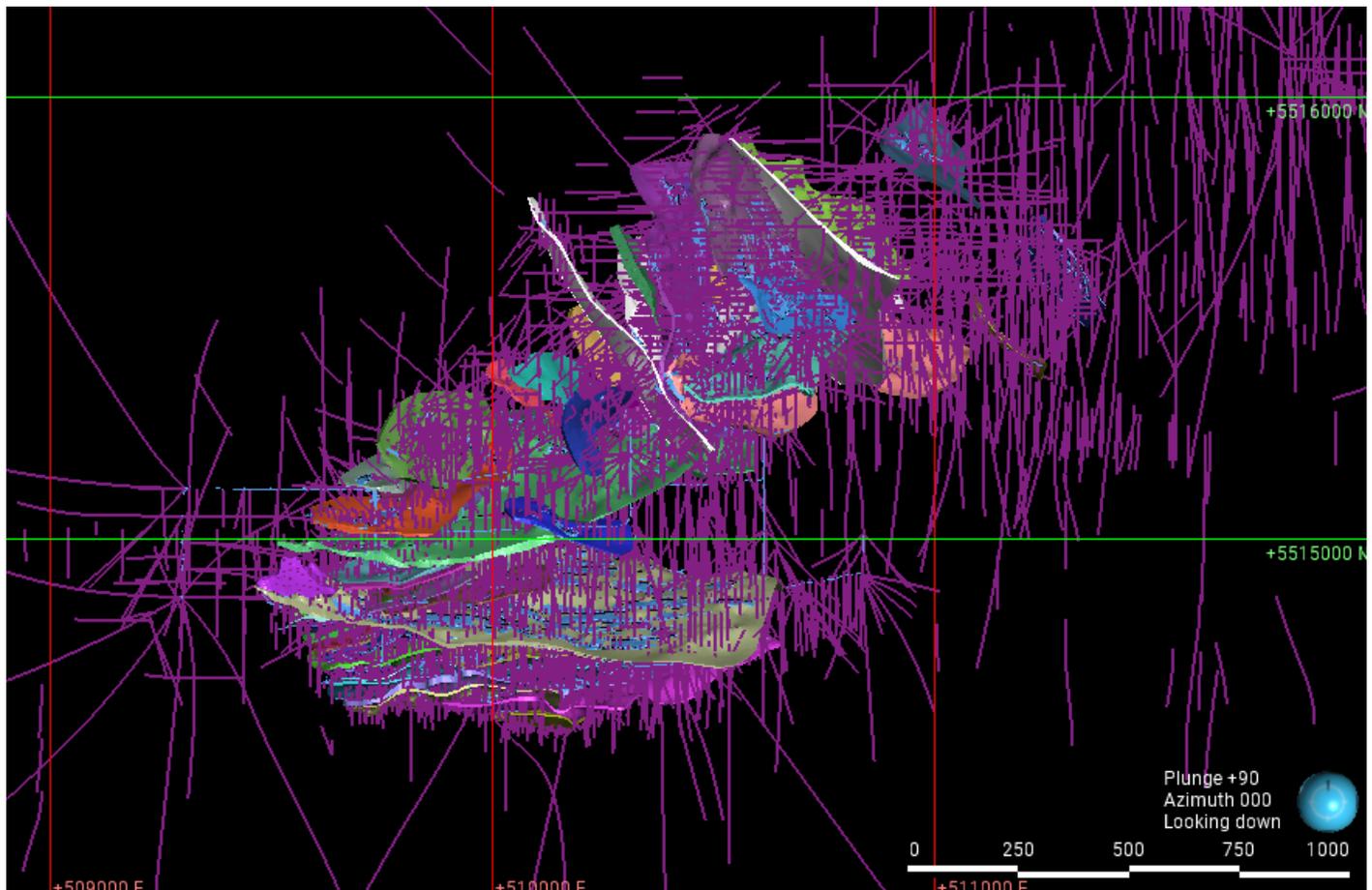
The expected life-of-mine recoveries based on these equations applied to PEA mine plan are 92.0%, 79.9% and 80.3% for copper, gold, and silver, respectively.

14 MINERAL RESOURCE ESTIMATES

14.1 Introduction

This mineral resource estimate (MRE) reflects the Opémiska deposit only (Perry, Springer, and Saddle). Other occurrences on the project were considered exploration targets at the time of writing and additional exploration work is required before those occurrences can be considered mineral resources. Figure 14-1 shows the Opémiska deposit in plan view.

Figure 14-1: Overall 3D View Looking Down showing the High-Grade Zones (Multiple Colours) and the Drill Holes (in Purple)



Source: PLR, 2025.

14.2 Methodology

Leapfrog Geo™ and Edge™ v.2024.1.3 (Leapfrog) was used to update the geological and mineralized zones and to generate the drill hole intercepts for each solid. Leapfrog was used for compositing, 3D block modelling, and interpolation. Statistical studies were conducted using Excel and Snowden Supervisor.

The methodology for the mineral resource estimation involved the following steps:

- database verification
- 3D modelling of the geological zones
- 3D modelling update of the mineralized zones
- 3D modelling of a stockwork zones
- drill hole intercept and composite generation
- basic statistics
- capping
- geostatistical analysis including variography
- block modelling and grade interpolation
- block model validation
- mineral resource classification
- cut-off grade calculation
- pit shell optimization
- DSO optimization
- preparation of the mineral resource statement.

14.3 Resource Database

The MRE wireframes are based on 21,918 drill holes for 1,525,073 meters and 479,242 samples. The drill hole database includes recent drilling (2002 to 2025) for 73,227 meters in 382 drill holes (Ex-In, PowerOre, QC Copper and Gold, XXIX) and incorporates historical drill holes (1930 to 1990) of 1,451,846 meters in 21,536 drill holes (Opémiska Copper Mines, Falconbridge, Minnova). The cut-off date for the drill hole database is May 16, 2025. The QP validated the database.

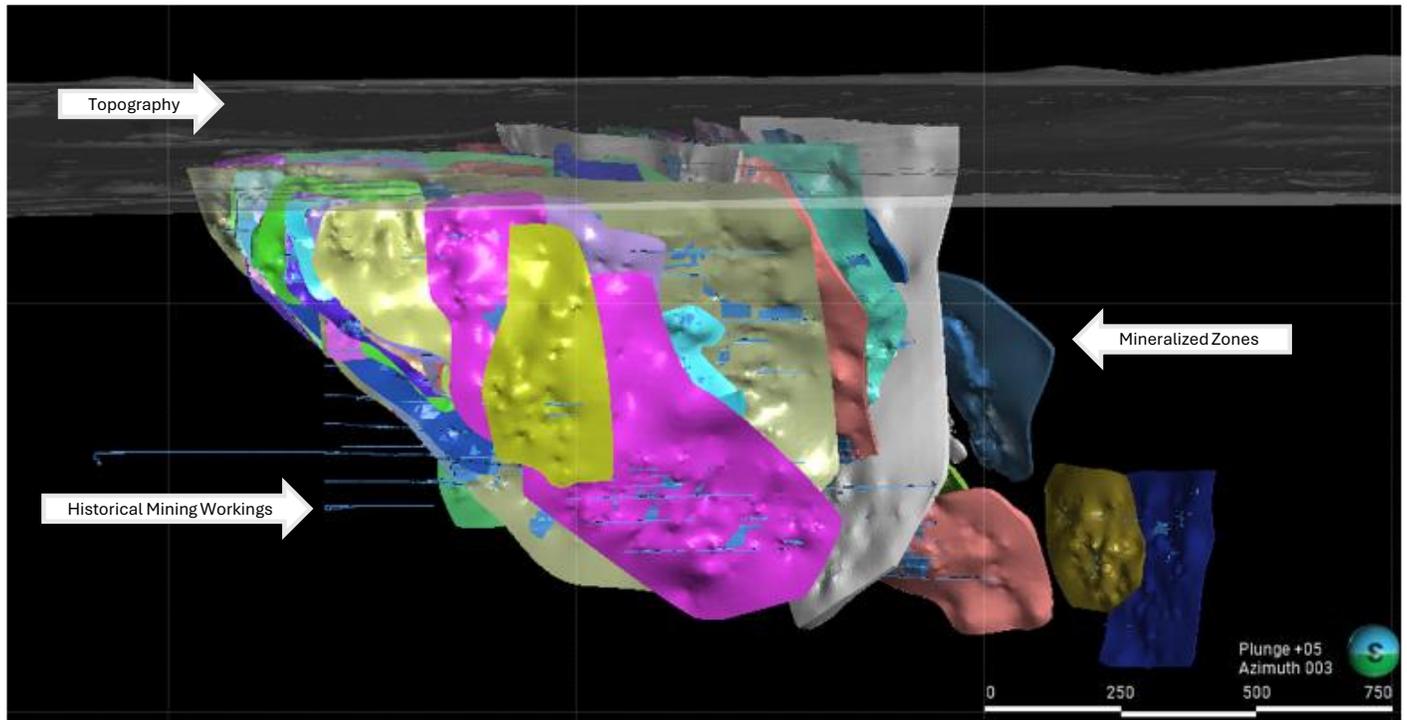
14.4 Geological Model

Geological and high-grade zone wireframes were provided by XXIX. The QP concentrated his efforts in updating the 3D model mainly by adding the stockwork zones surrounding the high-grade zones.

In total, 19 high-grade zones in the Perry sector, 50 in the Springer sector, five in the Saddle sector, and six lithologies were modelled, as were the overburden, the historical underground voids, and the topography. The geological model, mineralized zone, and dilution envelope were clipped to the overburden/bedrock interface when necessary.

Figure 14-2 shows a 3D view of the geological model.

Figure 14-2: 3D Geological Model of the Opémiska Deposit (looking North)



Source: PLR, 2025.

14.5 Voids Model

The Opémiska deposit has seen significant underground mining activity. Blocks affected by historical underground workings were sterilized.

The QP and XXIX are aware that the currently modelled voids lack precision. They are well-representative of the volume of material historically mined out, but galleries were modelled perfectly horizontal and did not take into consideration they were built with a small slope towards the shafts; therefore, drifts, stopes, and underground drill hole collars are likely inaccurate by up to a few meters. This is in part why the QP did not declare any measured resources despite openings and very tight drill spacing.

14.6 Historical Database Validation

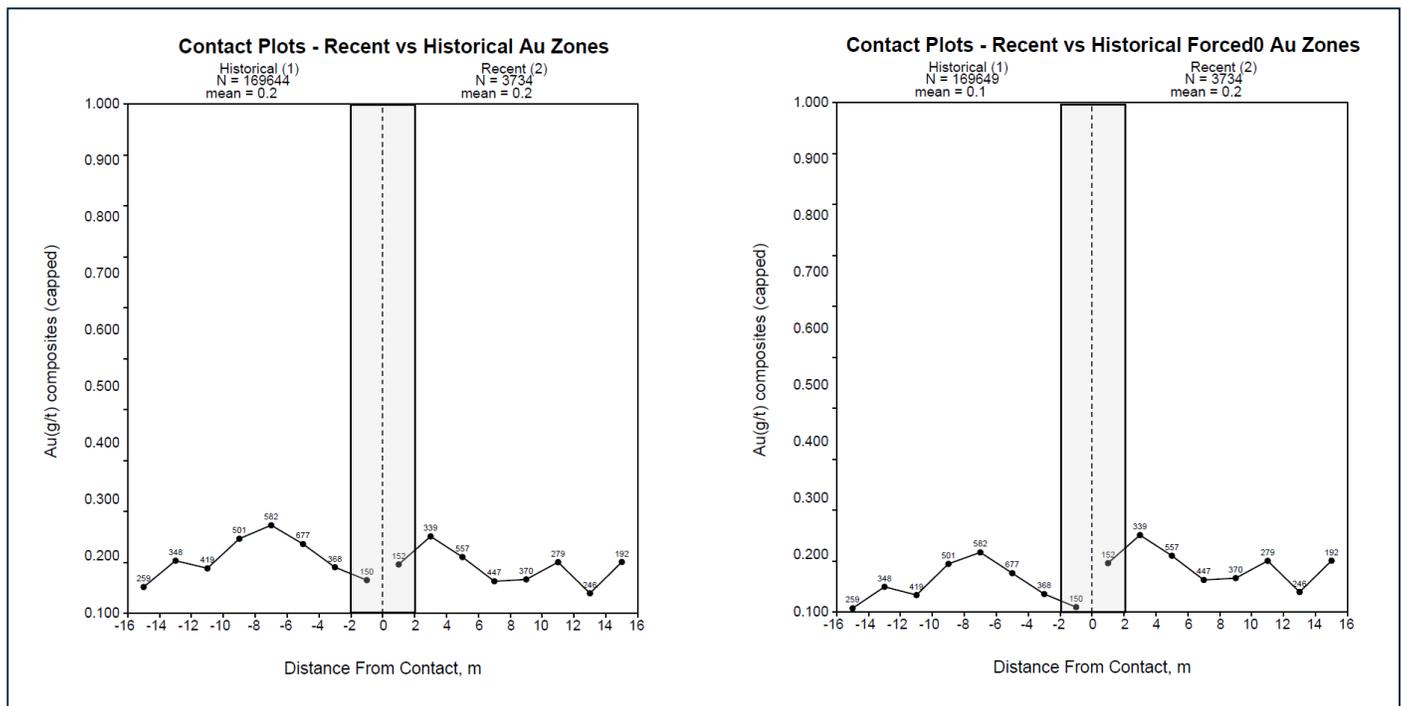
Basic statistics have shown that multiple historical drill hole assays had lower detection limits significantly high for gold, such as 0.0857 g/t (0.0025 oz/t) and 0.1714 g/t (0.005 oz/t). The QP geostatistically reviewed the data by creating 3D grade-distance paired plots with the following two populations: historical holes with high lower-detection limits, and

recent holes with low lower-detection limits. This method allows a comparison of every composite from a database (historical holes) to composites from a second database (recent holes) based on grades and the distance between each individual sample.

The purpose of creating these charts was to identify possible bias and understand the impact of using the historical database as-is versus applying a conservative approach, such as forcing all samples at the lower detection limit to 0.00 g/t or any arbitrary values in between. Comparing two non-biased databases would show similar average grade in the centre of the graph, whereas biased databases would show a disconnect in the centre of the graph.

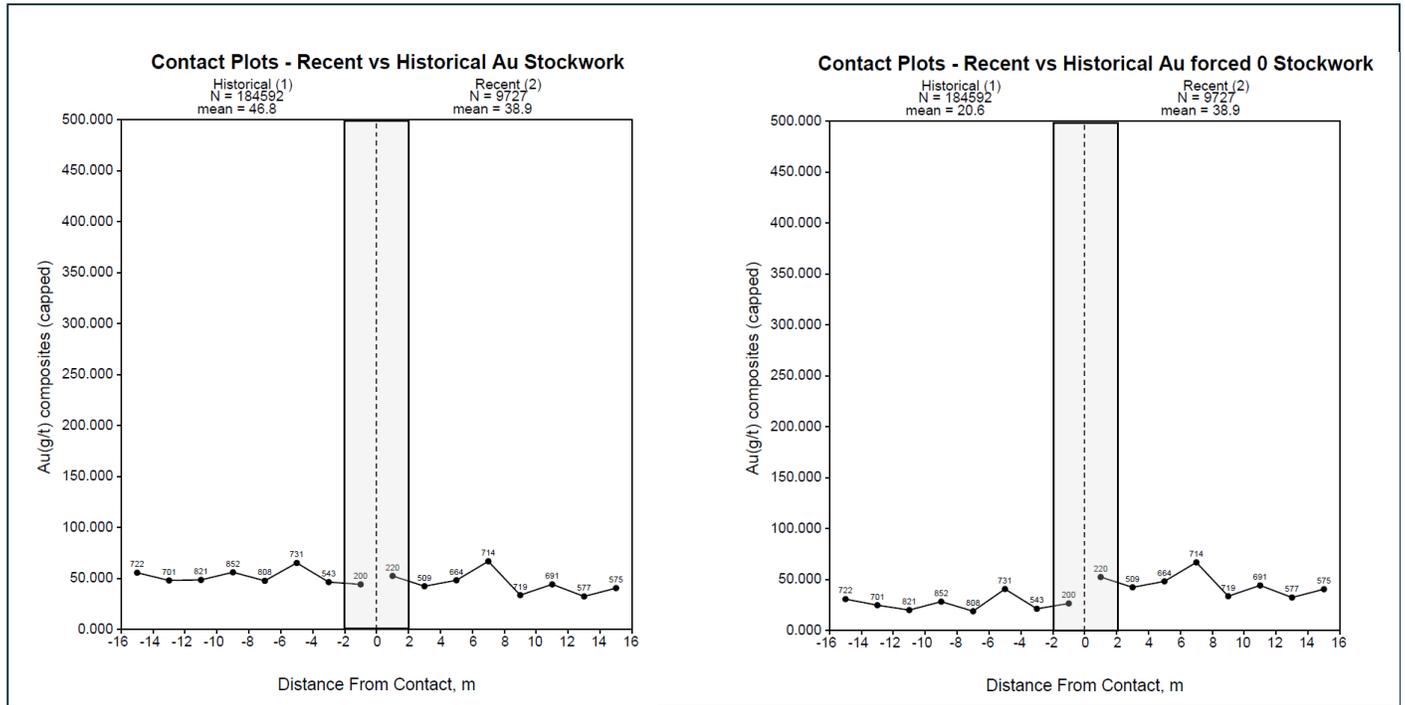
After running numerous tests, it was demonstrated that using the gold value presented in the historical database did not bias the database. In fact, lowering these values would negatively bias the database and block model. The 3D pair-distance plots supporting these actions are presented in Figure 14-3 for the high-grade zones and Figure 14-4 for the stockwork zones.

Figure 14-3: 3D Grade-Distance Paired Plots for Gold in the High-Grade Zones



Note: The graph on the left shows the historical database as-is being geostatistically compared to the recent database. The graph on the right shows all samples where the lower detection limit is either 0.0025 or 0.005 oz/t being forced to 0.00 g/t. These graphs clearly show that altering the database is introducing a negative bias not supported by recent drillholes. Source: PLR, 2025.

Figure 14-4: 3D Grade-Distance Paired Plots for Gold in the Stockwork Zones

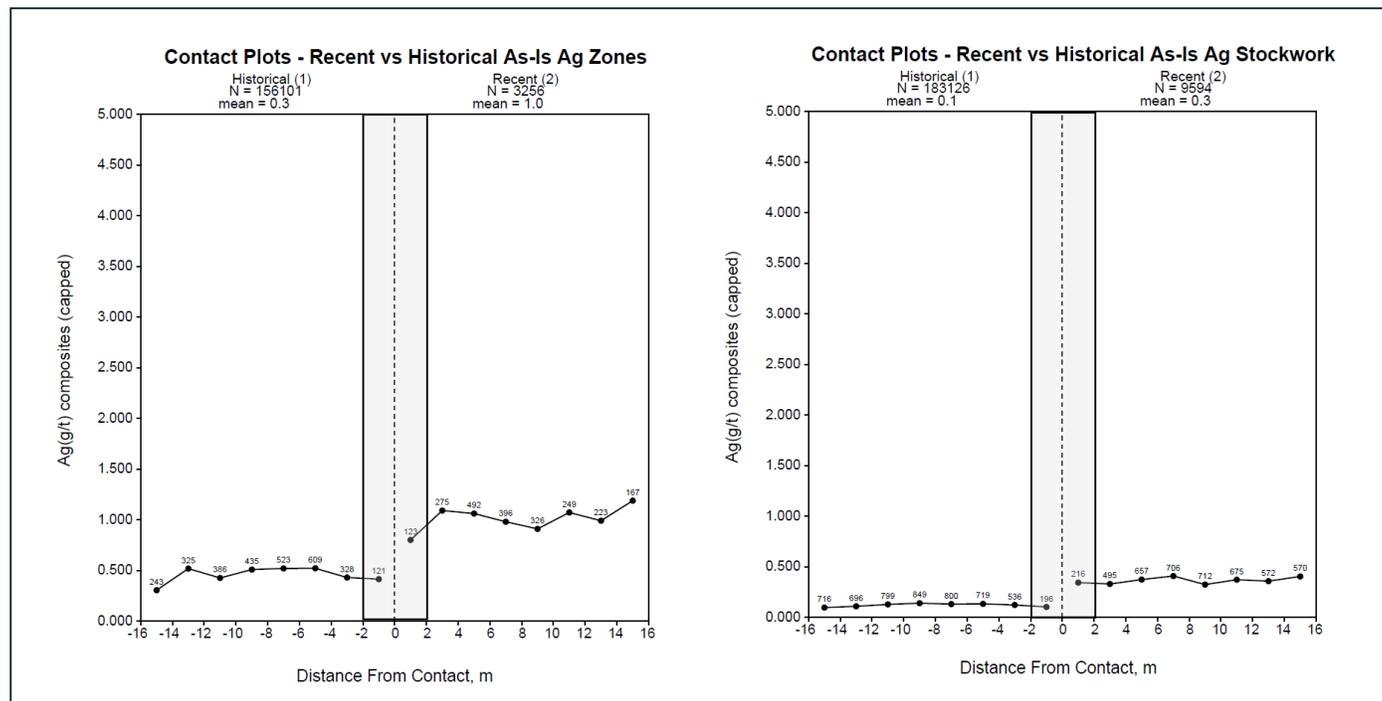


Note: The graph on the left shows the historical database as-is being geostatistically compared to the recent database. The graph on the right shows all samples where the lower detection limit is either 0.0025 or 0.005 oz/t being forced to 0.00 g/t. These graphs clearly show that altering the database is introducing a negative bias not supported by recent drillholes. Source: PLR, 2025.

As part of the historical database validation, the QP also looked for any bias in silver and copper.

The graphs in Figure 14-5 show the historical silver database being geostatistically compared to the recent database. Both graphs show the historical database to be negatively biased for silver grades at the Opémiska deposit. This represents an upside for the project.

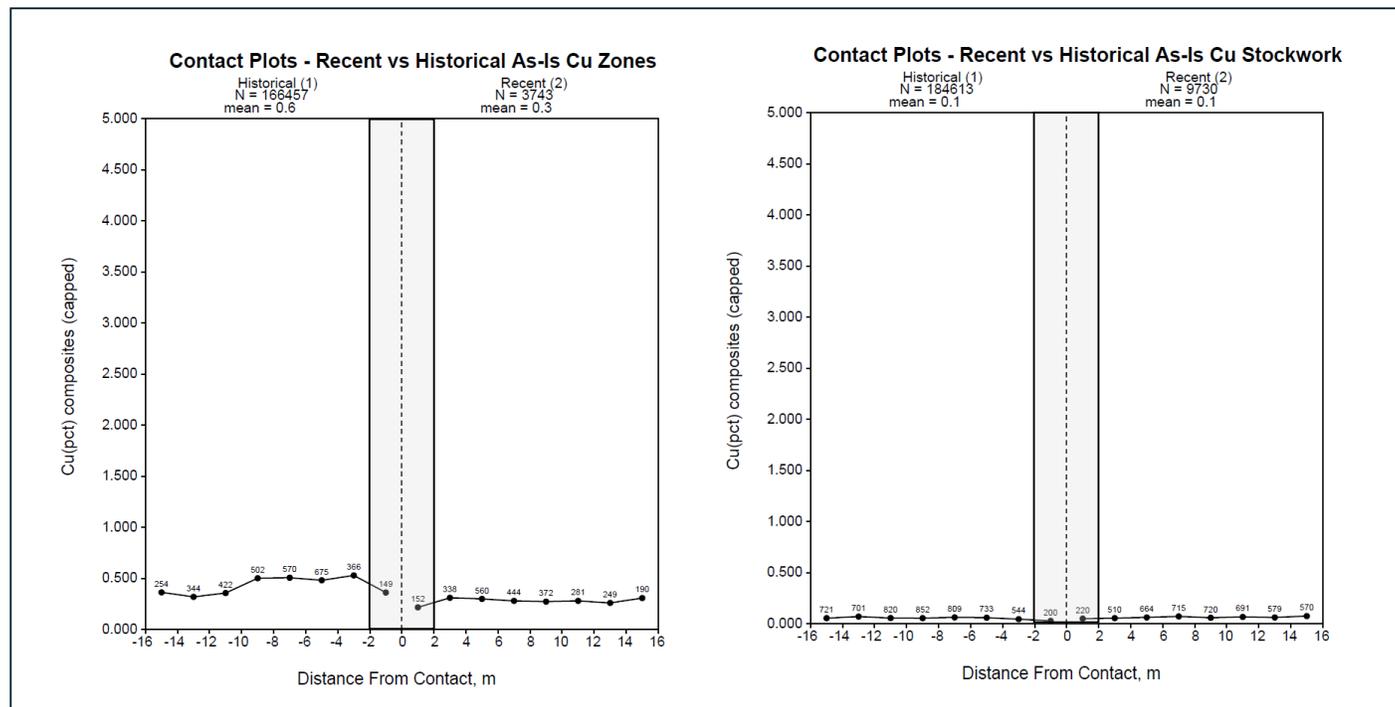
Figure 14-5: 3D Grade-Distance Paired Plots for Silver in the High-Grade Zones (Left) and Stockwork Zones (Right)



Note: Both graphs show the historical database as-is being geostatistically compared to the recent database. Both graphs show that the historical database is negatively biased for silver grades at the Opémiska deposit. This represents an upside for the project. Source: PLR, 2025.

Copper is presented in Figure 14-6. The graphs show the historical database being geostatistically compared to the recent database. The graph on the left (high-grade zones) shows that the historical database is positively biased for copper grades. This is what one could expect from an historical mine after significant depletion. Recent drilling did not crosscut the higher-grade material that was depleted, which explains the disconnect shown in the graph. The graph on the right which shows the stockwork zones (i.e., outside historical mine depletion) presents no bias.

Figure 14-6: 3D Grade-Distance Paired Plots for Copper in the High-Grade Zones (Left) and Stockwork Zones (Right)



Note: Both graphs show the historical database as-is being geostatistically compared to the recent database. The graph on the left (high-grade zones) shows that the historical database is positively biased for copper grades. This is what one could expect from an historical mine that saw significant mining depletion. The graph on the right (stockwork zones; outside historical mine depletion) shows no bias. Source: PLR, 2025.

14.7 Compositing

All raw assay data intersecting the mineralized zone, the stockwork zones, and the various lithological units were assigned individual rock codes. These coded intercepts were used to produce basic statistics on sample lengths and grades. A total of 201,805 assays is included in the high-grade mineralized zones and 224,095 in the stockwork zones.

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

In the mineralized zones, 86% of the samples are less than 1.53 m long, with the average sample length being 1.26 m. In the stockwork zones, 88% of the samples are less than 3.05 m long, with the average sample length being 2.50 m. Based on these statistics and geological considerations, 363,902 composites were generated with an average length of 1.5 m in the high-grade zones and 3 m in the stockwork zones, after redistributing the tails.

14.8 Capping

It is common practice to statistically examine the higher grades within a population and trim them to a lower grade value based on the results of a statistical study. Capping is performed on high-grade values considered to be outliers. An outlier is an observation that appears inconsistent with most of the data. High-grade capping was done on the composited assay.

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

Basic statistics for copper, gold, and silver composites, as well as capped composites, are summarized in Tables 14-1 to 14-3, respectively. Figures 14-7 to 14-9 are graphs supporting the capping threshold decisions for one of the high-grade zones. Capping was defined on all 76 zones (both high-grade and stockwork) individually for copper and gold, and on sectors for silver.

Table 14-1: Basic Statistics on Composites and High-Grade Capping Values for Copper

Zones	Raw (Uncapped)						Capping	Restricted Search (RS)	Capped						
	Count	Min	Max	Mean	Median	COV			Min	Max	Mean	Median	COV	COV (with RS)	
Perry	Perry-FLT_B	2,757	0.00	26.17	1.19	0.34	1.76	17.50	Nil	0.00	17.50	1.19	0.34	1.72	1.72
	Perry-FLT_J	1,133	0.00	16.54	1.53	0.67	1.50	15.00	Nil	0.00	15.00	1.53	0.67	1.49	1.49
	Perry-FLT_P-1	1,667	0.00	21.07	0.69	0.11	2.37	15.00	Nil	0.00	15.00	0.68	0.11	2.32	2.32
	Perry-FLT_P-2	3,505	0.00	15.61	0.58	0.13	2.23	15.00	Nil	0.00	15.00	0.58	0.13	2.23	2.23
	Perry-FLT_P-3	1,682	0.00	11.89	0.93	0.34	1.42	7.50	Nil	0.00	7.50	0.92	0.34	1.39	1.39
	Perry-VA	4,322	0.00	18.59	0.59	0.20	1.90	15.00	Nil	0.00	15.00	0.59	0.20	1.88	1.88
	Perry-VA-1	1,553	0.00	9.27	0.33	0.10	1.86	4.00	Nil	0.00	4.00	0.32	0.10	1.72	1.72
	Perry-VA-2	903	0.00	13.73	0.74	0.20	2.14	12.50	Nil	0.00	12.50	0.74	0.20	2.13	2.13
	Perry-VB	18,099	0.00	27.63	1.18	0.35	1.98	25.00	Nil	0.00	25.00	1.18	0.35	1.98	1.98
	Perry-VB-2	217	0.00	26.45	0.89	0.15	2.59	6.00	Nil	0.00	6.00	0.76	0.15	1.80	1.80
	Perry-VC	5,948	0.00	20.05	0.87	0.21	2.03	17.50	Nil	0.00	17.50	0.86	0.21	2.02	2.02
	Perry-VD	11,750	0.00	21.36	0.84	0.27	1.91	17.50	Nil	0.00	17.50	0.84	0.27	1.90	1.90
	Perry-VE	4,322	0.00	18.67	0.59	0.15	2.13	12.50	Nil	0.00	12.50	0.58	0.15	2.10	2.10
	Perry-VJ	3,353	0.00	15.07	0.87	0.31	1.68	15.00	Nil	0.00	15.00	0.87	0.31	1.68	1.68
	Perry-VJ-2	1,592	0.00	15.74	0.84	0.29	1.87	10.00	Nil	0.00	10.00	0.83	0.29	1.79	1.79
	Perry-VJ3	2,426	0.00	28.23	0.99	0.34	2.01	20.00	Nil	0.00	20.00	0.99	0.34	1.94	1.94
	Perry-VJ4	4,065	0.00	16.00	0.87	0.35	1.50	12.50	Nil	0.00	12.50	0.87	0.35	1.48	1.48
	Perry-VJ5	252	0.00	3.84	0.47	0.19	1.53	3.00	Nil	0.00	3.00	0.46	0.19	1.49	1.49
Perry-VK	870	0.00	17.99	0.55	0.13	2.81	12.50	Nil	0.00	12.50	0.54	0.13	2.64	2.64	
Perry-STKW	30,821	0.00	23.97	0.21	0.04	3.24	15.00	1.50	0.00	15.00	0.21	0.04	3.16	2.02	
Springer	Springer_V02	14,515	0.00	27.05	0.99	0.18	2.18	25.00	Nil	0.00	25.00	0.99	0.18	2.18	2.18
	Springer-V00	1,026	0.00	15.41	0.51	0.26	1.79	7.50	Nil	0.00	7.50	0.50	0.26	1.63	1.63
	Springer-V01	2,305	0.00	24.77	0.78	0.24	2.17	17.50	Nil	0.00	17.50	0.77	0.24	2.13	2.13
	Springer-V03	21,122	0.00	22.61	1.12	0.50	1.57	20.00	Nil	0.00	20.00	1.12	0.50	1.57	1.57
	Springer-V03_Ext	1,374	0.00	16.22	0.73	0.24	1.96	12.50	Nil	0.00	12.50	0.72	0.24	1.90	1.90
	Springer-V04	2,340	0.00	19.43	0.89	0.22	2.09	17.50	Nil	0.00	17.50	0.89	0.22	2.08	2.08
	Springer-V05	2,359	0.00	8.70	0.53	0.19	1.57	7.50	Nil	0.00	7.50	0.52	0.19	1.56	1.56
	Springer-V05_B	2,065	0.00	18.00	0.59	0.15	2.34	12.50	Nil	0.00	12.50	0.59	0.15	2.26	2.26
	Springer-V06	3,754	0.00	15.57	0.51	0.12	2.11	10.00	Nil	0.00	10.00	0.51	0.12	2.06	2.06
	Springer-V06_S	1,213	0.00	5.69	0.23	0.04	2.03	4.00	Nil	0.00	4.00	0.23	0.04	1.94	1.94
	Springer-V07	6,580	0.00	16.14	0.51	0.23	1.55	10.00	Nil	0.00	10.00	0.50	0.23	1.53	1.53
	Springer-V07_S	3,830	0.00	9.34	0.31	0.09	1.95	6.00	Nil	0.00	6.00	0.30	0.09	1.87	1.87
	Springer-V08	861	0.00	24.38	0.96	0.29	1.98	15.00	Nil	0.00	15.00	0.95	0.29	1.88	1.88
	Springer-V09	4,562	0.00	14.63	0.59	0.22	1.76	12.50	Nil	0.00	12.50	0.59	0.22	1.76	1.76
	Springer-V10	1,067	0.00	10.29	0.35	0.01	2.21	4.00	Nil	0.00	4.00	0.34	0.01	1.97	1.97
	Springer-V11-1	855	0.00	10.13	0.50	0.21	1.59	5.00	Nil	0.00	5.00	0.49	0.21	1.48	1.48
	Springer-V11-2	973	0.00	8.24	0.43	0.14	1.92	7.50	Nil	0.00	7.50	0.43	0.14	1.91	1.91
	Springer-V13	891	0.00	22.55	0.55	0.01	2.91	8.00	Nil	0.00	8.00	0.51	0.01	2.39	2.39
	Springer-V20	689	0.00	12.46	0.64	0.41	1.33	4.00	Nil	0.00	4.00	0.62	0.41	1.06	1.06
	Springer-V21	866	0.00	12.52	0.90	0.27	1.80	10.00	Nil	0.00	10.00	0.89	0.27	1.77	1.77
	Springer-V22	327	0.00	11.90	0.64	0.17	2.15	7.50	Nil	0.00	7.50	0.62	0.17	1.94	1.94
	Springer-V23	6,791	0.00	19.72	0.75	0.28	1.90	17.50	Nil	0.00	17.50	0.75	0.28	1.90	1.90
	Springer-V28	41	0.01	12.10	0.86	0.23	2.29	3.50	Nil	0.01	3.50	0.65	0.23	1.48	1.48
	Springer-V29	281	0.00	9.29	0.53	0.20	1.90	5.00	Nil	0.00	5.00	0.51	0.20	1.72	1.72
	Springer-V30	659	0.00	9.42	0.75	0.27	1.80	8.00	Nil	0.00	8.00	0.75	0.27	1.77	1.77
	Springer-V31	989	0.00	18.64	0.64	0.24	2.03	8.00	Nil	0.00	8.00	0.63	0.24	1.81	1.81
	Springer-V31A	34	0.00	0.98	0.28	0.22	0.77	1.00	Nil	0.00	0.98	0.28	0.22	0.77	0.77
	Springer-V32	1,808	0.00	18.06	0.76	0.31	1.88	12.50	Nil	0.00	12.50	0.75	0.31	1.82	1.82
	Springer-V32_B	188	0.00	14.64	0.69	0.30	1.99	7.50	Nil	0.00	7.50	0.66	0.30	1.61	1.61
	Springer-V33	1,254	0.00	11.26	0.56	0.21	1.92	10.00	Nil	0.00	10.00	0.56	0.21	1.91	1.91
	Springer-V33_B	1,406	0.00	15.05	0.63	0.33	1.59	7.50	Nil	0.00	7.50	0.62	0.33	1.49	1.49
	Springer-V34	1,043	0.00	22.01	0.96	0.25	2.31	15.00	Nil	0.00	15.00	0.94	0.25	2.21	2.21
	Springer-V34_B	278	0.00	5.06	0.27	0.09	2.03	3.00	Nil	0.00	3.00	0.25	0.09	1.76	1.76
	Springer-V34_C	111	0.00	4.92	0.39	0.16	1.81	3.00	Nil	0.00	3.00	0.37	0.16	1.63	1.63
	Springer-V34_D	101	0.00	2.12	0.29	0.19	1.26	2.50	Nil	0.00	2.12	0.29	0.19	1.26	1.26
	Springer-V34_E	277	0.00	16.44	0.79	0.16	2.85	15.00	Nil	0.00	15.00	0.79	0.16	2.82	2.82
	Springer-V34_F	28	0.00	1.86	0.23	0.10	1.68	2.50	Nil	0.00	1.86	0.23	0.10	1.68	1.68
	Springer-V34_North	382	0.00	12.90	0.57	0.24	2.07	6.00	Nil	0.00	6.00	0.55	0.24	1.82	1.82
	Springer-V39-40	2,745	0.00	16.22	0.59	0.25	1.78	10.00	Nil	0.00	10.00	0.59	0.25	1.73	1.73
	Springer-V41	1,725	0.00	17.02	0.65	0.24	1.85	12.50	Nil	0.00	12.50	0.64	0.24	1.80	1.80
	Springer-V41_B	2,297	0.00	21.40	0.81	0.23	2.18	17.50	Nil	0.00	17.50	0.81	0.23	2.15	2.15
	Springer-V42	1,833	0.00	13.77	0.75	0.25	1.85	12.50	Nil	0.00	12.50	0.74	0.25	1.84	1.84
Springer-V42_A	586	0.00	9.56	0.46	0.13	2.30	7.50	Nil	0.00	7.50	0.45	0.13	2.26	2.26	
Springer-V43	450	0.00	15.28	0.52	0.15	2.70	6.00	Nil	0.00	6.00	0.47	0.15	2.18	2.18	
Springer-V60	1,396	0.00	10.95	0.34	0.16	1.86	5.00	Nil	0.00	5.00	0.33	0.16	1.66	1.66	
Springer-V61	663	0.00	15.87	0.45	0.20	2.45	6.00	Nil	0.00	6.00	0.41	0.20	1.76	1.76	
Springer-V62	1,099	0.00	12.30	0.39	0.17	1.93	3.50	Nil	0.00	3.50	0.37	0.17	1.52	1.52	
Springer-V63	561	0.00	8.85	0.40	0.14	2.01	6.00	Nil	0.00	6.00	0.39	0.14	1.91	1.91	
Springer-V64-65	619	0.00	17.09	0.36	0.10	2.76	7.50	Nil	0.00	7.50	0.34	0.10	2.28	2.28	
Springer-V72	552	0.00	7.13	0.35	0.16	1.65	3.00	Nil	0.00	3.00	0.34	0.16	1.46	1.46	
Springer-STKW	62,514	0.00	13.34	0.12	0.02	3.28	1.00	0.25	0.00	1.00	0.09	0.02	2.12	1.43	
Saddle	Saddle_00	449	0.00	24.27	0.89	0.11	2.71	12.50	Nil	0.00	12.50	0.84	0.11	2.43	2.43
	Saddle_01	90	0.00	7.40	0.50	0.01	2.15	3.00	Nil	0.00	3.00	0.42	0.01	1.60	1.60
	Saddle_02	111	0.00	12.33	0.79	0.72	1.65	4.00	Nil	0.00	4.00	0.71	0.72	1.06	1.06
	Saddle_03	59	0.00	8.25	0.79	0.47	1.61	2.50	Nil	0.00	2.50	0.66	0.47	1.08	1.08
	Saddle_05	38	0.00	6.05	1.36	1.00	0.99	3.50	Nil	0.00	3.50	1.29	1.00	0.89	0.89
	Saddle_STKW	2,712	0.00	3.10	0.08	0.01	2.47	1.00	Nil	0.00	1.00	0.07	0.01	2.05	2.05

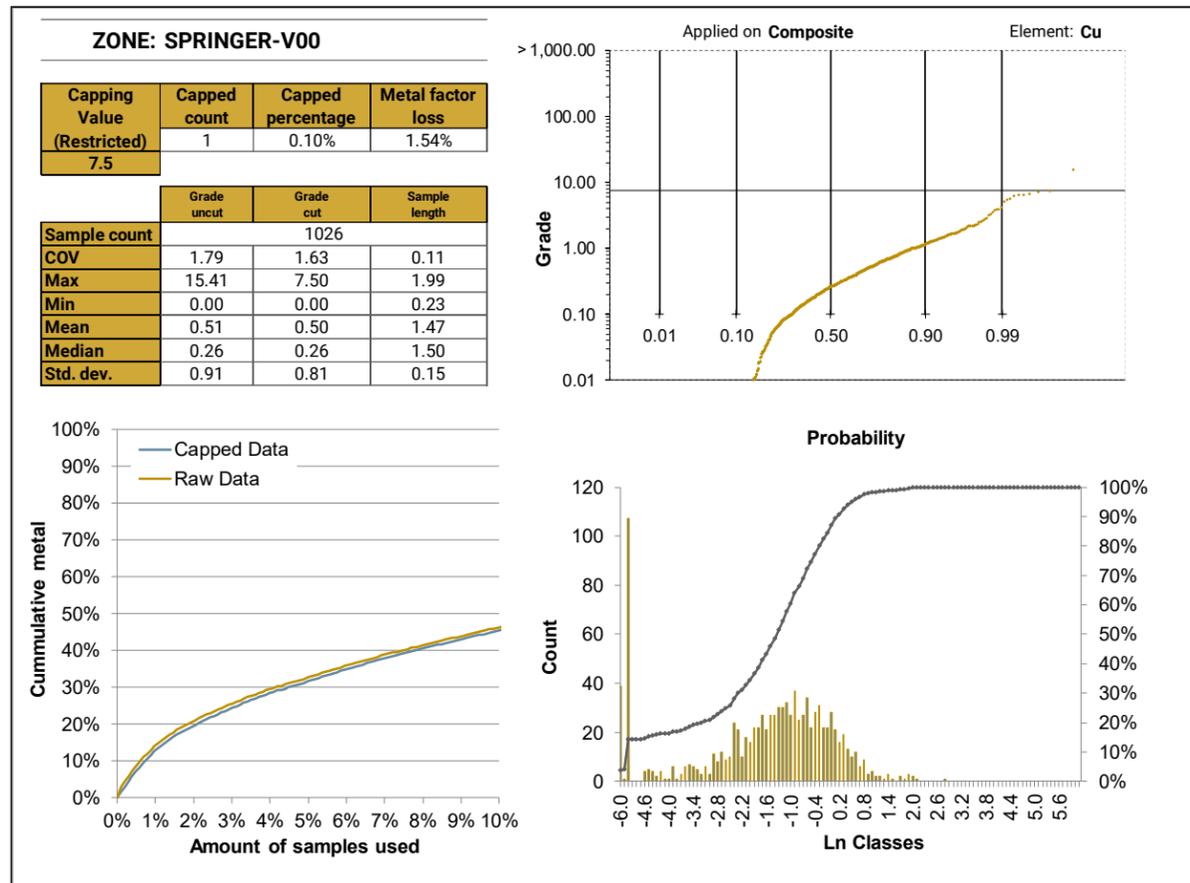
Table 14-2: Basic Statistics on Composites and high-Grade Capping Values for Gold

Zones	Raw (Uncapped)						Capping	Restricted Search (RS)	Capped						
	Count	Min	Max	Mean	Median	COV			Min	Max	Mean	Median	COV	COV (with RS)	
Perry	Perry-FLT_B	2,757	0.00	18.51	0.14	0.01	3.94	3.50	Nil	0.00	3.50	0.12	0.01	2.76	2.76
	Perry-FLT_J	1,133	0.00	4.82	0.15	0.09	2.02	5.00	Nil	0.00	4.82	0.15	0.09	2.02	2.02
	Perry-FLT_P-1	1,667	0.00	14.78	0.16	0.09	3.26	3.50	Nil	0.00	3.50	0.15	0.09	2.05	2.05
	Perry-FLT_P-2	3,505	0.00	21.35	0.08	0.01	6.35	3.00	1.50	0.00	3.00	0.07	0.01	3.53	2.71
	Perry-FLT_P-3	1,682	0.00	4.66	0.08	0.06	1.94	5.00	Nil	0.00	4.66	0.08	0.06	1.94	1.94
	Perry-VA	4,322	0.00	41.93	0.15	0.09	5.93	3.00	Nil	0.00	3.00	0.13	0.09	2.10	2.10
	Perry-VA-1	1,553	0.00	2.87	0.04	0.01	3.27	1.00	Nil	0.00	1.00	0.04	0.01	2.33	2.33
	Perry-VA-2	903	0.00	11.65	0.23	0.09	2.81	3.00	Nil	0.00	3.00	0.21	0.09	1.96	1.96
	Perry-VB	18,099	0.00	92.50	0.14	0.01	6.32	3.50	Nil	0.00	3.50	0.12	0.01	2.58	2.58
	Perry-VB-2	217	0.00	5.49	0.11	0.01	3.90	1.00	Nil	0.00	1.00	0.08	0.01	2.17	2.17
	Perry-VC	5,948	0.00	8.87	0.08	0.01	3.25	3.50	Nil	0.00	3.50	0.08	0.01	2.80	2.80
	Perry-VD	11,750	0.00	15.71	0.10	0.01	4.03	5.00	Nil	0.00	5.00	0.09	0.01	2.77	2.77
	Perry-VE	4,322	0.00	12.52	0.14	0.09	3.26	5.00	Nil	0.00	5.00	0.13	0.09	2.73	2.73
	Perry-VJ	3,353	0.01	1.71	0.03	0.01	3.29	1.50	Nil	0.01	1.50	0.03	0.01	3.25	3.25
	Perry-VJ-2	1,592	0.00	7.09	0.11	0.09	2.92	2.50	Nil	0.00	2.50	0.10	0.09	2.34	2.34
	Perry-VJ3	2,426	0.00	0.91	0.02	0.01	2.59	0.50	Nil	0.00	0.50	0.02	0.01	2.50	2.50
	Perry-VJ4	4,065	0.00	1.57	0.03	0.01	3.40	1.00	0.35	0.00	1.00	0.02	0.01	3.21	2.64
	Perry-VJ5	252	0.01	1.04	0.05	0.01	2.47	0.50	Nil	0.01	0.50	0.04	0.01	1.95	1.95
	Perry-VK	870	0.00	3.14	0.05	0.01	3.27	1.00	Nil	0.00	1.00	0.05	0.01	2.42	2.42
Perry-STKW	19,614	0.00	16.49	0.10	0.09	2.07	3.50	Nil	0.00	3.50	0.09	0.09	1.46	1.46	
Springer	Springer_V02	14,515	0.00	50.03	0.13	0.01	6.21	12.50	1.00	0.00	12.50	0.13	0.01	4.91	2.61
	Springer-V00	1,026	0.00	11.83	0.30	0.09	2.79	4.00	Nil	0.00	4.00	0.27	0.09	2.11	2.11
	Springer-V01	2,305	0.00	43.22	0.41	0.07	4.28	7.50	Nil	0.00	7.50	0.35	0.07	2.90	2.90
	Springer-V03	21,122	0.00	55.54	0.29	0.17	3.92	10.00	Nil	0.00	10.00	0.27	0.17	2.76	2.76
	Springer-V03_Ext	1,374	0.00	13.44	0.28	0.17	2.51	7.50	Nil	0.00	7.50	0.28	0.17	2.28	2.28
	Springer-V04	2,340	0.00	30.47	0.38	0.01	3.70	7.50	Nil	0.00	7.50	0.34	0.01	2.74	2.74
	Springer-V05	2,359	0.00	14.09	0.35	0.09	2.50	9.00	Nil	0.00	9.00	0.35	0.09	2.41	2.41
	Springer-V05_B	2,065	0.00	96.35	0.78	0.09	5.29	35.00	7.50	0.00	35.00	0.69	0.09	3.96	2.47
	Springer-V06	3,754	0.00	232.42	1.32	0.09	5.70	35.00	12.50	0.00	35.00	1.08	0.09	3.51	2.66
	Springer-V06_S	1,213	0.00	33.53	0.46	0.09	4.20	10.00	5.00	0.00	10.00	0.39	0.09	3.09	2.56
	Springer-V07	6,580	0.00	94.67	0.94	0.14	3.29	25.00	Nil	0.00	25.00	0.90	0.14	2.79	2.79
	Springer-V07_S	3,830	0.00	69.80	0.70	0.09	4.41	35.00	7.50	0.00	35.00	0.68	0.09	4.03	2.68
	Springer-V08	861	0.00	23.53	0.16	0.01	5.38	2.50	Nil	0.00	2.50	0.13	0.01	2.55	2.55
	Springer-V09	4,562	0.00	28.14	0.32	0.09	3.04	12.50	Nil	0.00	12.50	0.32	0.09	2.77	2.77
	Springer-V10	1,067	0.00	10.35	0.17	0.09	2.70	5.00	Nil	0.00	5.00	0.17	0.09	2.26	2.26
	Springer-V11-1	855	0.00	11.66	0.22	0.09	2.66	4.00	Nil	0.00	4.00	0.21	0.09	2.14	2.14
	Springer-V11-2	973	0.00	5.45	0.15	0.01	2.51	3.50	Nil	0.00	3.50	0.15	0.01	2.35	2.35
	Springer-V13	891	0.00	24.70	0.33	0.01	4.48	10.00	3.50	0.00	10.00	0.29	0.01	3.73	2.84
	Springer-V20	689	0.00	20.79	0.40	0.10	3.17	10.00	Nil	0.00	10.00	0.37	0.10	2.59	2.59
	Springer-V21	866	0.00	11.05	0.22	0.09	3.35	6.00	Nil	0.00	6.00	0.21	0.09	2.83	2.83
	Springer-V22	327	0.00	16.49	0.13	0.01	7.42	1.00	Nil	0.00	1.00	0.07	0.01	2.56	2.56
	Springer-V23	6,791	0.00	38.68	0.35	0.09	3.76	20.00	7.50	0.00	20.00	0.35	0.09	3.31	2.56
	Springer-V28	41	0.01	1.03	0.12	0.01	2.03	1.50	Nil	0.01	1.03	0.12	0.01	2.03	2.03
	Springer-V29	281	0.01	3.37	0.17	0.09	2.52	2.50	Nil	0.01	2.50	0.17	0.09	2.40	2.40
	Springer-V30	659	0.00	9.58	0.30	0.09	2.64	7.50	Nil	0.00	7.50	0.30	0.09	2.54	2.54
	Springer-V31	989	0.00	20.03	0.25	0.09	4.03	3.50	Nil	0.00	3.50	0.20	0.09	2.55	2.55
	Springer-V31A	34	0.07	2.34	0.25	0.09	2.03	2.50	Nil	0.07	2.34	0.25	0.09	2.03	2.03
	Springer-V32	1,808	0.00	17.18	0.28	0.09	3.55	5.00	Nil	0.00	5.00	0.25	0.09	2.54	2.54
	Springer-V32_B	188	0.00	25.12	0.50	0.09	4.21	6.00	3.00	0.00	6.00	0.38	0.09	2.78	2.25
	Springer-V33	1,254	0.00	15.64	0.30	0.09	3.14	12.00	4.00	0.00	12.00	0.30	0.09	3.03	2.27
	Springer-V33_B	1,406	0.00	48.65	0.43	0.09	4.00	15.00	6.00	0.00	15.00	0.41	0.09	2.97	2.36
	Springer-V34	1,043	0.00	12.27	0.29	0.09	2.99	9.00	3.00	0.00	9.00	0.28	0.09	2.89	2.16
	Springer-V34_B	278	0.00	2.06	0.15	0.09	1.76	2.50	Nil	0.00	2.06	0.15	0.09	1.76	1.76
	Springer-V34_C	111	0.01	12.32	0.56	0.09	3.06	3.50	Nil	0.01	3.50	0.38	0.09	2.00	2.00
	Springer-V34_D	101	0.00	1.68	0.18	0.09	1.61	2.00	Nil	0.00	1.68	0.18	0.09	1.61	1.61
	Springer-V34_E	277	0.00	16.21	0.66	0.09	3.14	15.00	4.00	0.00	15.00	0.65	0.09	3.11	2.12
	Springer-V34_F	28	0.01	1.92	0.18	0.09	2.03	2.00	Nil	0.01	1.92	0.18	0.09	2.03	2.03
	Springer-V34_North	382	0.00	13.65	0.43	0.09	2.76	7.00	Nil	0.00	7.00	0.39	0.09	2.26	2.26
	Springer-V39-40	2,745	0.00	43.89	0.37	0.09	4.59	20.00	4.50	0.00	20.00	0.35	0.09	3.80	2.34
	Springer-V41	1,725	0.00	31.51	0.34	0.09	3.90	6.00	Nil	0.00	6.00	0.30	0.09	2.49	2.49
	Springer-V41_B	2,297	0.00	51.04	0.41	0.09	3.97	7.50	Nil	0.00	7.50	0.36	0.09	2.59	2.59
	Springer-V42	1,833	0.00	46.42	0.36	0.09	4.03	17.00	5.00	0.00	17.00	0.35	0.09	3.08	2.35
	Springer-V42_A	586	0.00	52.25	0.41	0.09	7.11	4.50	Nil	0.00	4.50	0.23	0.09	2.61	2.61
Springer-V43	450	0.00	26.60	0.29	0.09	5.34	15.00	2.00	0.00	15.00	0.26	0.09	4.34	2.16	
Springer-V60	1,396	0.00	49.88	0.77	0.09	3.77	30.00	7.50	0.00	30.00	0.76	0.09	3.58	2.35	
Springer-V61	663	0.00	44.47	0.82	0.09	3.60	20.00	7.00	0.00	20.00	0.76	0.09	2.97	2.16	
Springer-V62	1,099	0.00	59.90	1.02	0.12	3.53	35.00	10.00	0.00	35.00	0.97	0.12	3.07	2.18	
Springer-V63	561	0.00	61.10	1.19	0.09	3.40	25.00	10.00	0.00	25.00	1.11	0.09	2.84	2.32	
Springer-V64-65	619	0.00	124.98	1.61	0.09	4.40	30.00	6.00	0.00	30.00	1.37	0.09	3.34	2.14	
Springer-V72	552	0.00	53.31	0.69	0.09	4.37	12.00	6.50	0.00	12.00	0.57	0.09	2.45	2.15	
Springer-STKW	55,762	0.00	85.18	0.17	0.09	6.38	12.50	2.50	0.00	12.50	0.16	0.09	3.91	2.18	
Saddle	Saddle_00	449	0.00	16.87	0.32	0.02	4.65	3.50	Nil	0.00	3.50	0.21	0.02	2.88	2.88
	Saddle_01	90	0.00	2.61	0.11	0.01	2.59	0.50	Nil	0.00	0.50	0.09	0.01	1.28	1.28
	Saddle_02	111	0.00	3.24	0.26	0.10	1.73	3.00	Nil	0.00	3.00	0.26	0.10	1.69	1.69
	Saddle_03	59	0.01	4.34	0.46	0.17	1.53	4.50	Nil	0.01	4.34	0.46	0.17	1.53	1.53
	Saddle_05	38	0.00	1.47	0.28	0.17	1.15	1.50	Nil	0.00	1.47	0.28	0.17	1.15	1.15
	Saddle_STKW	2,584	0.00	3.06	0.05	0.01	2.95	1.00	Nil	0.00	1.00	0.05	0.01	2.18	2.18

Table 14-3: Basic Statistics on Composites and High-Grade Capping Values for Silver

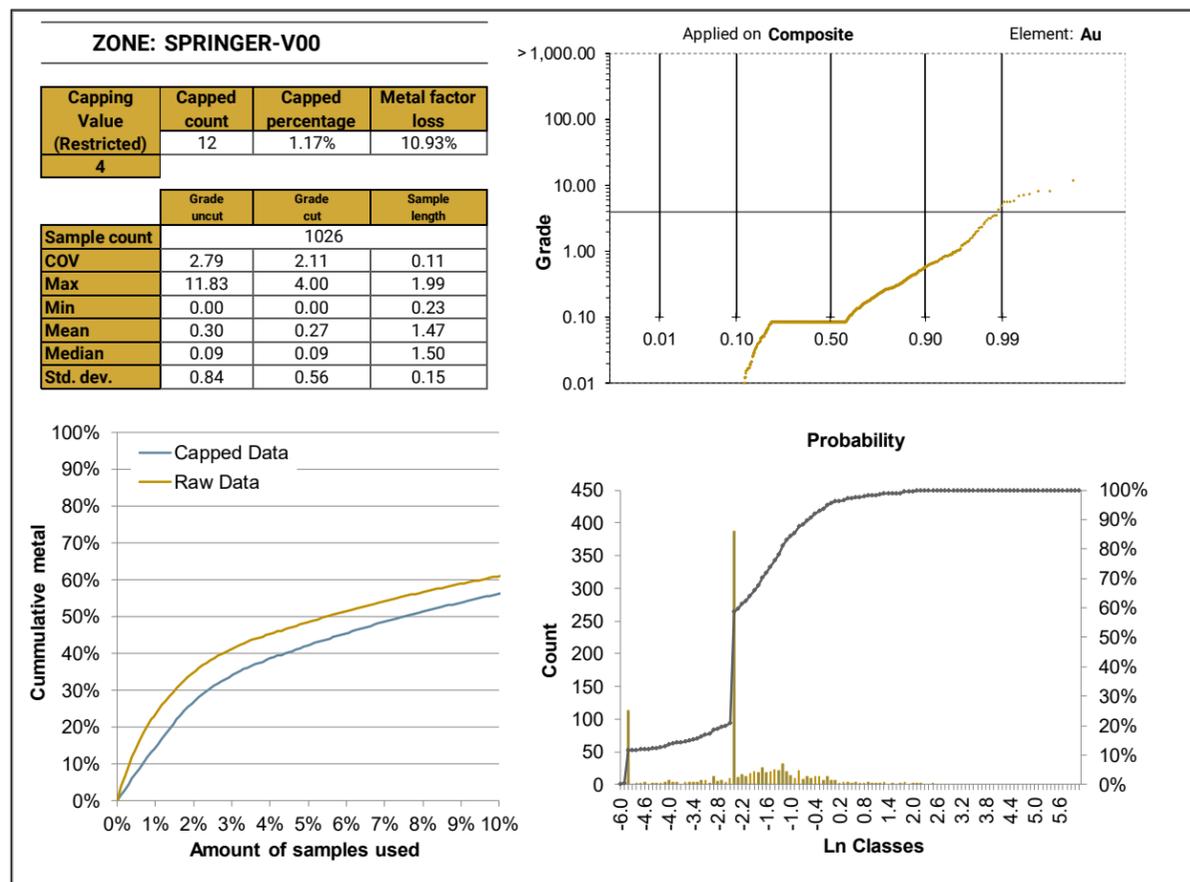
Zones	Raw (Uncapped)						Capping	Restricted Search (RS)	Capped				
	Count	Min	Max	Mean	Median	COV			Min	Max	Mean	Median	COV (with RS)
Perry High-Grade Zones	19,222	0.00	18.11	2.69	0.59	1.47	17.50	Nil	0.00	17.50	2.69	0.59	0.59
Perry Stockwork Zones	1,875	0.00	44.61	0.71	0.16	2.75	12.50	Nil	0.00	12.50	0.68	0.16	0.16
Springer High-Grade Zones	46,643	0.00	243.54	3.76	1.04	2.12	120.00	Nil	0.00	120.00	3.75	1.04	1.04
Springer Stockwork Zones	47,442	0.00	128.65	0.61	0.10	3.74	10.00	Nil	0.00	10.00	0.54	0.10	0.10
Saddle High-Grade Zones	537	0.00	77.46	6.20	2.47	1.67	40.00	Nil	0.00	40.00	5.82	2.47	2.47
Saddle Stockwork Zones	1,875	0.00	44.61	0.71	0.16	2.75	12.50	Nil	0.00	12.50	0.68	0.16	0.16

Figure 14-7: Graphs Supporting Copper Capping on Composites in the High-Grade Zone Springer-V00



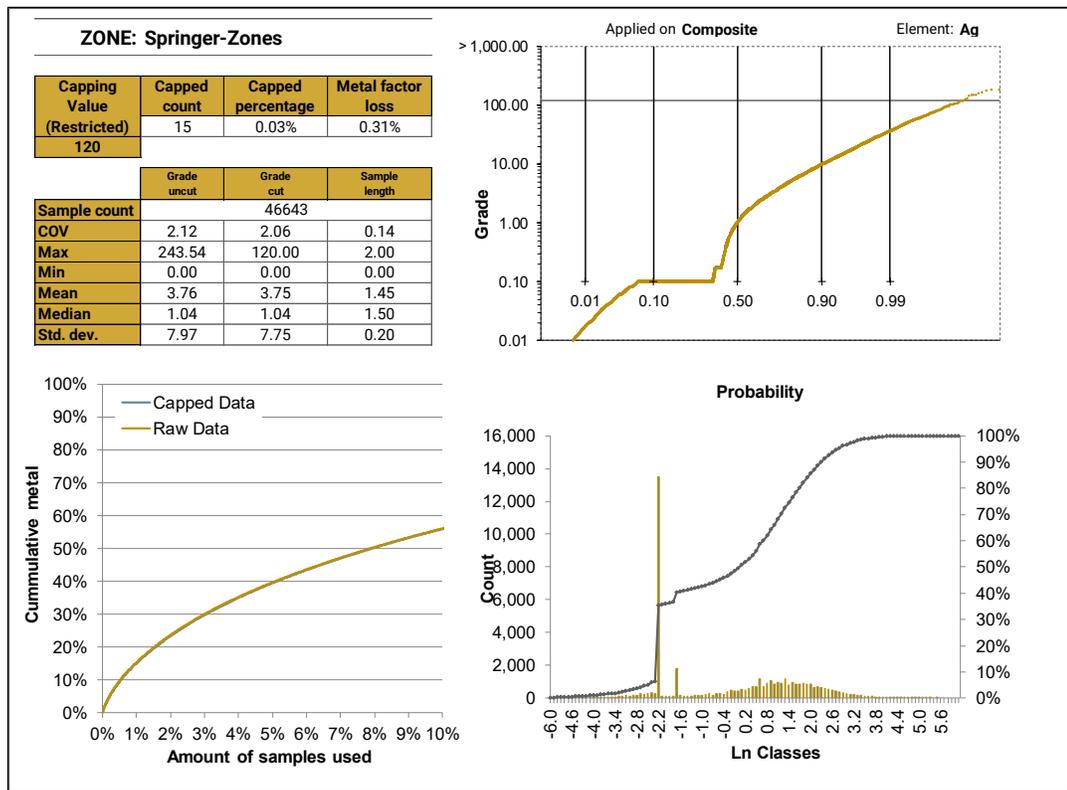
Source: PLR, 2025.

Figure 14-8: Graphs Supporting Gold Capping on Composites in the High-Grade Zone Springer-V00



Source: PLR, 2025.

Figure 14-9: Graphs Supporting Silver Capping on Composites in the Springer High-Grade Zones



Source: PLR, 2025.

14.9 Density

Bulk density is an important parameter to calculate tonnages for the estimated volumes derived from the block model.

Density measurements were collected on the project by previous operators and, more recently, by XXIX. A total of 1,149 measurements are within the geological model used for the current MRE. The samples span all the different lithologies and mineralized domains, although some domains contain very few data.

The number and distribution of density measures made available are not sufficient for density to be interpolated. Therefore, fixed density values were assigned to lithological units and mineralized domains, corresponding to the average density of each unit when data was statistically sufficient and corresponding to a similar domain when data was statistically insufficient.

Table 14-4 shows the basic statistics for the density database. The method of assigning densities to the block model is identified in the last column of the table. A fixed density of 2.00 g/cm³ was assigned to the overburden. A fixed density of 0.00 g/cm³ was assigned to the underground voids.

Table 14-4: Density Basic Statistics

Lithology	Mineralized Domain	Blockcode	Count	Min	Max	Average	Standard Deviation	CoV	Assigned	Comment
Gabbro	Country Rock	0	180	2.59	4.56	3.00	0.17	0.06	3.00	
	Perry_Zone	100	1	2.88	2.88	2.88	0.00	0.00	2.99	Same as Gabbro - Stockwork_Perry (not enough measures in Gabbro - Saddle Zone)
	SaddleZone	300	4	2.89	3.11	2.98	0.08	0.03	3.01	Same as Gabbro - Stockwork_Saddle (not enough measures in Gabbro - Saddle Zone)
	Springer_Zone	200	105	2.71	4.05	2.97	0.19	0.06	2.97	
	Stockwork_Perry	1100	18	2.79	3.17	2.99	0.11	0.04	2.99	
	Stockwork_Saddle	1300	7	2.79	3.19	3.01	0.13	0.04	3.01	
	Stockwork_Springer	1200	440	2.65	4.12	2.94	0.11	0.04	2.94	
Foliated Gabbro	Country Rock	0	1	3.11	3.11	3.11	0.00	0.00	3.00	Same as Gabbro (not enough measures in Foliated Gabbro - Country Rock)
	Springer_Zone	200	8	2.93	3.13	3.02	0.07	0.02	3.02	
	Stockwork_Springer	1200	68	2.77	3.22	3.01	0.09	0.03	3.01	
Peridotite	Country Rock	0	23	2.78	4.01	2.91	0.25	0.08	2.91	
	Springer_Zone	200	1	3.17	3.17	3.17	0.00	0.00	3.05	Same as Peridotite - Stockwork_Springer (not enough measures in Peridotite - Springer Zone)
	Stockwork_Perry	0	2	2.82	3.18	3.00	0.18	0.06	3.05	Same as Peridotite - Stockwork_Springer (not enough measures in Stockwork_Perry)
	Stockwork_Springer	1200	9	2.82	3.20	3.05	0.16	0.05	3.05	
Pyroxenite	Country Rock	0	109	2.81	4.12	3.15	0.15	0.05	3.15	
	Perry_Zone	100	3	3.14	4.01	3.44	0.40	0.12	3.14	Same as Pyroxenite - Stockwork_Perry (not enough measures in Pyroxenite - Perry Zone)
	Springer_Zone	200	3	2.87	3.32	3.03	0.21	0.07	2.98	Same as Pyroxenite - Stockwork_Springer (not enough measures in Pyroxenite - Springer Zone)
	Stockwork_Perry	1100	10	3.04	3.22	3.14	0.07	0.02	3.14	
	Stockwork_Springer	1200	45	2.75	3.36	2.98	0.16	0.05	2.98	
Rhyolite	Country Rock	0	28	2.64	2.93	2.71	0.07	0.03	2.71	
	Springer_Zone	200	3	2.68	2.80	2.74	0.05	0.02	2.70	Same as Rhyolite - Stockwork_Springer (not enough measures in Rhyolite - Springer Zone)
	Stockwork_Saddle	1300	1	2.81	2.81	2.81	0.00	0.00	2.70	Same as Rhyolite - Stockwork_Springer (not enough measures in Rhyolite - Stockwork_Saddle)
	Stockwork_Springer	1200	75	2.63	2.93	2.70	0.06	0.02	2.70	
Basalt	Country Rock		5	3.13	3.21	3.17	0.03	0.01		None in the Model

14.10 Variogram Analysis and Search Ellipsoids

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

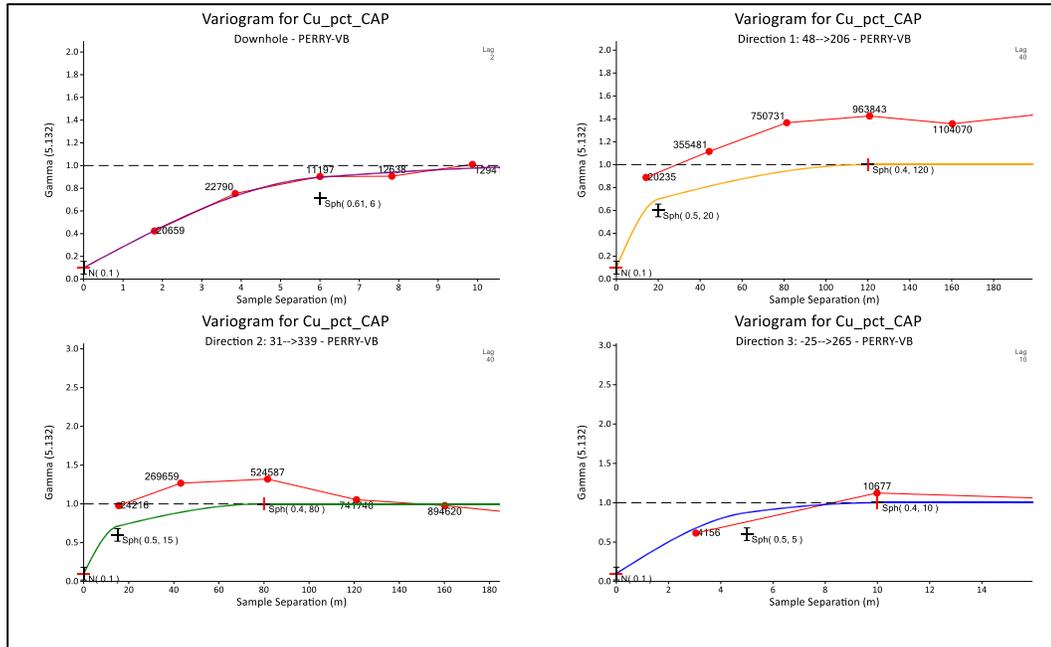
Three-dimensional directional variography was carried out on the composites using the Snowden Supervisor software. Variograms were modelled in the three orthogonal directions to define a 3D ellipsoid for the mineralized zone. The three directions of ellipsoid axes were set by using the variogram fans and visually confirmed using the geological knowledge of the deposit. A mathematical model was then interpreted to best fit the shape of the calculated variogram for each direction. Three components were defined for the mathematical model: the nugget effect, the sill, and the range. In all cases where a normal score transformation was used, the results were back-transformed before using them to define the ellipsoids and interpolation parameters,

Table 14-5 presents the chosen variogram model parameters, and Figures 14-10 to 14-12 illustrate the variograms for copper, gold, and silver, respectively, for high-grade zone Perry-VB, one of the zones containing a significant number of composites.

Table 14-5: Variogram Model Parameters

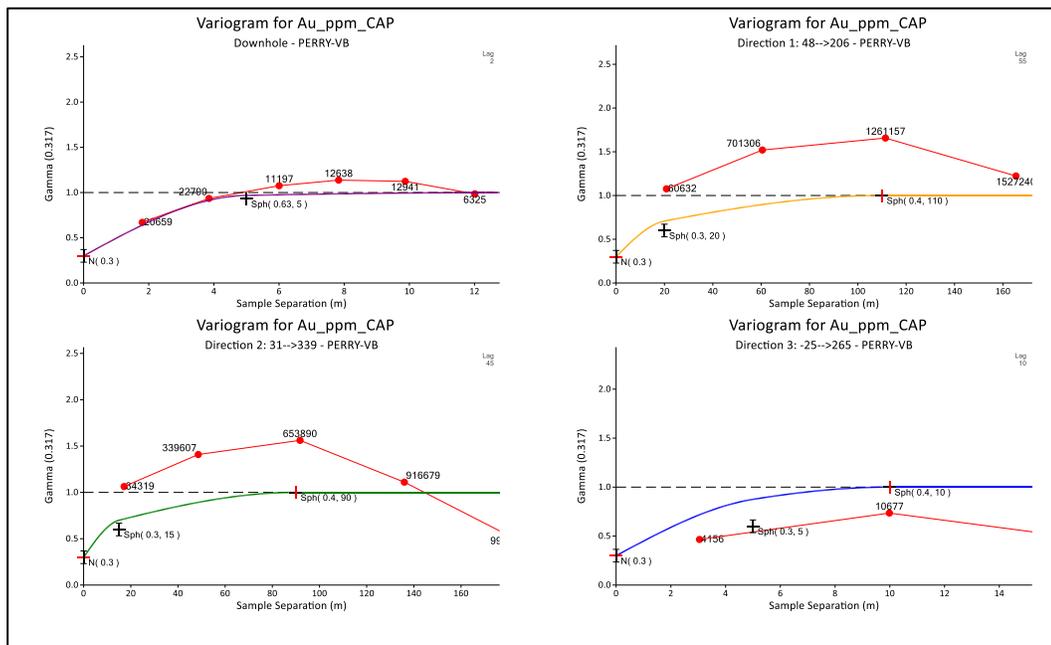
Zones	Element	Units	Nugget	First Structure			Second Structure				Leapfrog Orientation			
				Sill	Range X (m)	Range Y (m)	Range Z (m)	Sill	Range X (m)	Range Y (m)	Range Z (m)	Dip Azimuth	Dip	Pitch
Perry High-Grades	Cu	%	0.100	0.500	20	15	5	0.400	120	80	10	Variable Orientation		
	Au	g/t	0.300	0.300	20	15	5	0.400	110	90	10	Variable Orientation		
	Ag	g/t	0.100	0.500	20	20	5	0.400	90	90	10	Variable Orientation		
Springer High-Grades	Cu	%	0.300	0.360	15	10	5	0.340	60	45	10	Variable Orientation		
	Au	g/t	0.410	0.250	15	10	10	0.340	60	45	30	Variable Orientation		
	Ag	g/t	0.100	0.280	15	10	5	0.620	60	45	10	Variable Orientation		
Saddle High-Grades	Cu	%	0.300	0.150	40	20	5	0.550	90	35	10	Variable Orientation		
	Au	g/t	0.100	0.350	40	20	5	0.550	80	35	10	Variable Orientation		
	Ag	g/t	0.310	0.140	40	20	5	0.550	90	35	10	Variable Orientation		
Perry Stockwork	Cu	%	0.200	0.500	15	15	15	0.300	60	60	70	Variable Orientation		
	Au	g/t	0.200	0.480	30	15	75	0.320	80	80	180	Variable Orientation		
	Ag	g/t	0.250	0.450	30	50	15	0.300	100	120	50	Variable Orientation		
Springer Stockwork	Cu	%	0.350	0.470	15	15	15	0.180	65	50	70	Variable Orientation		
	Au	g/t	0.400	0.420	15	15	20	0.180	60	40	80	Variable Orientation		
	Ag	g/t	0.350	0.470	15	15	15	0.180	60	40	70	Variable Orientation		
Saddle Stockwork	Cu	%	0.200	0.230	25	75	40	0.570	130	180	105	Variable Orientation		
	Au	g/t	0.300	0.130	50	75	40	0.570	150	190	60	Variable Orientation		
	Ag	g/t	0.200	0.230	25	75	40	0.570	130	210	80	Variable Orientation		

Figure 14-10: Variography Study for Copper within the Perry-VB High-Grade Zone



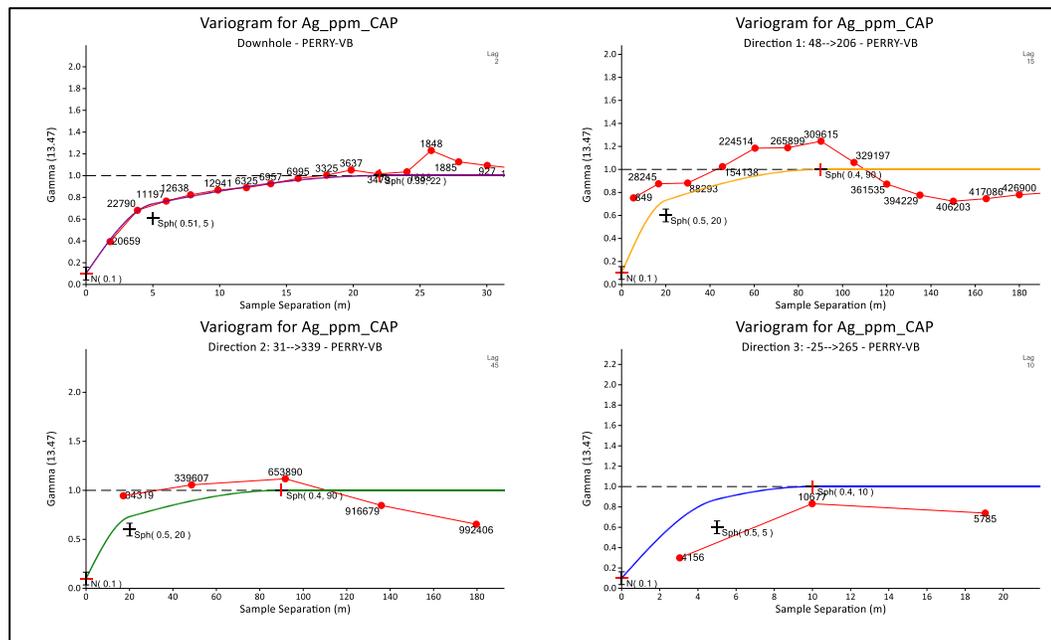
Source: PLR, 2025.

Figure 14-11: Variography Study for Gold within the Perry-VB High-Grade Zone



Source: PLR, 2025.

Figure 14-12: Variography Study for Silver within the Perry-VB High-Grade Zone



Source: PLR, 2025.

14.11 Block Model

The block model was constructed in Leapfrog for the current MRE using the block model parameters provided in Table 14-6. Individual block cells have dimensions of 5 m long (X-axis) by 5 m wide (Y-axis) by 5 m vertical (Z-axis). The size of the blocks was chosen to best match the drilling pattern, the thickness of the zones, the complexity of the geological model, and plausible future mining methods. The block size was discussed with engineers working on the project.

The block model was coded using the Octree sub-block method, down to 0.625 m, reflecting the proportion of each solid inside every block. All blocks falling within a solid were assigned the corresponding solid block code. Table 14-7 shows the various attributes in the block model.

Table 14-6: Block Model Parameters

Properties	X (Column)	Y (Row)	Z (Level)
Origin Coordinates	509,250	5,514,200	450
Number of Blocks	459	394	227
Block Size (m)	5	5	5
Sub-Block Size (down to)	0.625	0.625	0.625
Rotation	0		

Table 14-7: Block Model Coding

Attribute	Description
Blockcode	Code attributed to individual lithological units, mineralized zones, historical underground voids, and overburden.
CuEq	CuEq calculated using the CuEq formula
Classification	Classification (3 = Measured; 4 = Indicated; 5 = Inferred; 6 = Potential)
Copper	Cu interpolated with OK
Gold	Au interpolated with OK
Silver	Ag interpolated with OK
Density	Density (fixed)

14.12 Search Ellipsoid Strategy

The range and orientation of the ellipsoids used for interpolation were established using the variography study. Other interpolation parameters are derived from combining kriging neighbourhood analyses and the QP's professional experience.

Based on geostatistical analysis and general geological knowledge of the project, the following parameter was chosen for this mandate: the ranges of the ellipsoids correspond to the range of the variogram for the first pass and twice the range of the variogram for the second pass (Table 14-8). It should be mentioned that the classification was mostly based on geological confidence, grade continuity, the presence of recent drill holes, and drill hole spacing. For this reason, some interpolated blocks could not be classified as either inferred or indicated. Refer to Section 14.16, Mineral Resource Classification, for more details.

Table 14-8: Search Ellipsoids Range and Orientation by Interpolation Passes

Zones	Element	Units	Leapfrog orientation			First Pass			Second Pass		
			Dip Azimuth	Dip	Pitch	Range X (m)	Range Y (m)	Range Z (m)	Range X (m)	Range Y (m)	Range Z (m)
Perry High-Grades	Cu	%	Variable Orientation			120	80	20	240	160	40
	Au	g/t	Variable Orientation			110	90	20	220	180	40
	Ag	g/t	Variable Orientation			90	90	20	180	180	40
Springer High-Grades	Cu	%	Variable Orientation			120	80	20	240	160	40
	Au	g/t	Variable Orientation			60	45	30	120	90	60
	Ag	g/t	Variable Orientation			75	90	20	150	180	40
Saddle High-Grades	Cu	%	Variable Orientation			120	80	20	240	160	40
	Au	g/t	Variable Orientation			90	35	20	180	70	40
	Ag	g/t	Variable Orientation			90	35	20	180	70	40
Perry Stockwork	Cu	%	Variable Orientation			60	60	70	120	120	140
	Au	g/t	Variable Orientation			80	80	180	160	160	360

Zones	Element	Units	Leapfrog orientation			First Pass			Second Pass		
			Dip Azimuth	Dip	Pitch	Range X (m)	Range Y (m)	Range Z (m)	Range X (m)	Range Y (m)	Range Z (m)
	Ag	g/t	Variable Orientation			100	120	50	200	240	100
Springer Stockwork	Cu	%	Variable Orientation			65	50	70	130	100	140
	Au	g/t	Variable Orientation			60	40	80	120	80	160
	Ag	g/t	Variable Orientation			60	40	70	120	80	140
Saddle Stockwork	Cu	%	Variable Orientation			130	180	105	260	360	210
	Au	g/t	Variable Orientation			150	190	60	300	380	120
	Ag	g/t	Variable Orientation			130	210	80	260	420	160

14.13 Interpolation Method

The interpolation was run on a set of points extracted from the capped composited data. The block model grades were estimated using the ordinary kriging (OK) method. Hard boundaries were applied between the mineralized zones and surrounding country rocks to prevent grades from adjacent lithologies from being used during interpolation. As a block was estimated, it was tagged with the corresponding pass number, slope of regression, kriging efficiency, number of composites used, number of drill holes used, and drill spacing.

For comparison purposes, an additional grade model was generated (Table 14-9) using ID2.

Table 14-9: Interpolation Methods

Interpolation Method	Comments	Discretization (m)
Ordinary Kriging (OK)	Negative weights set to zero	3 x 3 x 3
Inverse Distance (ID2)	Anisotropic using variography ellipsoids	3 x 3 x 3

14.14 Interpolation Parameters

A kriging neighbourhood analysis (KNA) was conducted on one of the mineralized zones using Snowden Supervisor software. KNA provides a quantitative method of testing different estimation parameters (i.e., block size, discretization and min/max of composites used for the interpolation) by evaluating their impact on the quality of the results. This analysis helps select the optimal value for each parameter.

Following this study, the parameters provided in Table 14-10 were chosen for the interpolation of the block model. Although the interpolation parameters are largely inspired by the KNA study, they may differ slightly to accommodate certain interpolation needs, such as having a minimum number of drill holes or avoiding smearing effects. Multiple tests were made using different interpolation parameters.

Table 14-10: Interpolation Parameters

Zones	Element	Units	First Pass				Second Pass			
			Min Composite	Max Composite	Max Composite per DDH	Variography Ratio	Min Composite	Max Composite	Max Composite per DDH	Variography Ratio
Perry High-Grades	Cu	%	4	8	3	1.00	2	8	Not Limited	2.00
	Au	g/t	4	8	3	1.00	2	8	Not Limited	2.00
	Ag	g/t	4	8	3	1.00	2	8	Not Limited	2.00
Springer High-Grades	Cu	%	4	8	3	1.00	2	8	Not Limited	2.00
	Au	g/t	4	8	3	1.00	2	8	Not Limited	2.00
	Ag	g/t	4	8	3	1.00	2	8	Not Limited	2.00
Saddle High-Grades	Cu	%	4	8	3	1.00	2	8	Not Limited	2.00
	Au	g/t	4	8	3	1.00	2	8	Not Limited	2.00
	Ag	g/t	4	8	3	1.00	2	8	Not Limited	2.00
Perry Stockwork	Cu	%	4	8	3	1.00	2	8	Not Limited	2.00
	Au	g/t	4	8	3	1.00	2	8	Not Limited	2.00
	Ag	g/t	4	8	3	1.00	2	8	Not Limited	2.00
Springer Stockwork	Cu	%	4	8	3	1.00	2	8	Not Limited	2.00
	Au	g/t	4	8	3	1.00	2	8	Not Limited	2.00
	Ag	g/t	4	8	3	1.00	2	8	Not Limited	2.00
Saddle Stockwork	Cu	%	4	8	3	1.00	2	8	Not Limited	2.00
	Au	g/t	4	8	3	1.00	2	8	Not Limited	2.00
	Ag	g/t	4	8	3	1.00	2	8	Not Limited	2.00

14.15 Block Model Validation

The block model was validated using several methods, including statistical analyses and a visual review of the grades in the associated drill hole. Based on these visual and statistical reviews, it is the QP’s opinion that the Opémiska block model provides a reasonable estimate of in-situ mineral resources.

14.15.1 Visual validation

Block model grades were visually compared against drill hole composite grades and raw assays in cross-section, plan, longitudinal, and 3D views. This visual validation process also confirmed that the proper coding was done within the various domains. The visual comparison shows a good correlation between the values without excessive smoothing. Visual comparisons were also conducted between OK and ID2 interpolation scenarios. The OK scenario used for the MRE produced a grade distribution honouring drill hole data and the style of mineralization observed at the Opémiska deposit.

14.16 Mineral Resource Classification

The mineral resources were classified according to the “CIM Definition Standards for Mineral Resources & Mineral Reserves” (2014) published by the Canadian Institute of Mining, Metallurgy and Petroleum (CIM Definition Standards).

14.16.1 Mineral Resource Definition

The CIM Definition Standards (2014) clarify the following:

Inferred Mineral Resource:

- An inferred mineral resource is that part of a mineral resource for which quantity and grade or quality are estimated on the basis of limited geological evidence and sampling. Geological evidence is sufficient to imply but not verify geological and grade or quality continuity.
- An inferred mineral resource has a lower level of confidence than that applying to an indicated mineral resource and must not be converted to a mineral reserve. It is reasonably expected that the majority of inferred mineral resources could be upgraded to indicated mineral resources with continued exploration.

Indicated Mineral Resource:

- An indicated mineral resource is that part of a mineral resource for which quantity, grade or quality, densities, shape and physical characteristics are estimated with sufficient confidence to allow the application of modifying factors in sufficient detail to support mine planning and evaluation of the economic viability of the deposit.
- Geological evidence is derived from adequately detailed and reliable exploration, sampling and testing and is sufficient to assume geological and grade or quality continuity between points of observation.
- An indicated mineral resource has a lower level of confidence than that applying to a measured mineral resource and may only be converted to a probable mineral reserve.

Measured Mineral Resource:

- A measured mineral resource is that part of a mineral resource for which quantity, grade or quality, densities, shape, and physical characteristics are estimated with confidence sufficient to allow the application of modifying factors to support detailed mine planning and final evaluation of the economic viability of the deposit.
- Geological evidence is derived from detailed and reliable exploration, sampling and testing and is sufficient to confirm geological and grade or quality continuity between points of observation.
- A measured mineral resource has a higher level of confidence than that applying to either an indicated mineral resource or an inferred mineral resource. It may be converted to a proven mineral reserve or to a probable mineral reserve.

14.16.2 Opémiska Mineral Resource Classification

The mineral resources were classified according to CIM Definition Standards (2014). The estimated block grades were classified as either inferred or indicated using the drill spacing, geological continuity of mineralization, grade continuity, presence of recent drilling, and overall confidence level. Although a significant portion of the block model shows tight drill spacing, no measured mineral resources were defined for this phase of the project due, in part, to the modelled historical voids needing additional work.

Inferred mineral resources were defined for blocks within the mineralized zones within 60 m of a drill hole (120 m of drill spacing). Indicated mineral resources were defined where the following criteria were met:

- drill spacing of 50 m or less
- demonstrated geological continuity
- grade continuity at the reported cut-off grade
- recent drill holes confirming the model (geologically and grade-wise).

All remaining estimated but unclassified blocks were not reported.

The QP validated that no clipping boundaries were needed to either upgrade or downgrade classification to avoid issues caused by automatically generated classification.

14.17 Pit Optimization and DSO Parameters and Cut-off Grades

Resourced were constrained by both economic parameters represented by a value cut-off and geometrical parameters represented by pit shells for the open pit resource or stopes shapes for the underground resource. Table 14-11 presents the economic and geometrical optimization parameters used to constrain the resource.

Table 14-11: Optimization Parameters

Parameter	Unit	Open Pit	Underground
Revenue			
Royalty	%	1.00	1.00
Operating Costs			
Mining Cost	US\$/t mined	3.00	70.00
Process Cost	US\$/t milled	9.00	9.00
General & Administration Cost	US\$/t milled	2.25	2.25
Mineralization-Based Costs	US\$/t milled	11.25	81.25
Mining			
Block Size	m	5 x 5 x 5	Sub-blocked
Slope Angle – Rock	°	55	-
Slope Angle – Overburden	°	30	-
Minimum Mining Width	m	-	2.0
Stope Height	m	-	25.0
Cut-off Grade			
NSR Cut-off	US\$/t milled	11.36	82.07
CuEq Cut-off	%	0.15	1.00

Resources are presented as undiluted and in situ for the open pit scenario and include internal dilution for the underground scenario. The pit optimization to develop the resource-constraining pit shells was done using Deswik Pseudoflow. The stope optimization to develop the resource-constraining stope shapes was done using Deswik SO.

Revenue calculations were done on a block-by-block basis and were based on a metallurgical recovery model developed for the project and on customary smelter contract provisions that consider current market conditions. Parameters used for revenue calculation are presented in Table 14-12. Based on the revenue assumptions, the equation used to calculate CuEq% equals $Cu\% + 0.8531 Au\text{ g/t} + 0.0083 Ag\text{ g/t}$.

Table 14-12: Revenue Calculation

Parameter	Unit	Value
Price Assumptions		
Copper	US\$/lb	4.25
Gold	US\$/oz	2500.00
Silver	US\$/oz	27.00
Concentrate Specifications		
Mass Pull	Decimal	$0.0335 * HG_{Cu} + 0.0158$
Copper Concentrate Grade	%	$IF(Cu\text{ Head Grade} < 2.89, 23.205 + 23.32 * HG_{Cu} - 576.7 * MP - 3.276 * HG_{Cu} * HG_{Cu} + 2070 * MP * MP, 24.5)$
Gold Concentrate Grade	g/t	$IF(HG_{Cu}/HG_{Au} < 2, Con\ gr\ Cu / (1.0643 * HG_{Cu}/HG_{Au} + 0.0542), 19.779 * (HG_{Cu}/HG_{Au})^{-0.905})$
Silver Concentrate Grade	g/t	$(1.0893 * (HG_{Ag}/HG_{Au}) - 1.0645) * Con\ gr\ Au$
Recovery Calculation		
Copper	%	$mp * conc\ gr/HG$, capped at 0.95
Gold	%	$mp * conc\ gr\ HG$, capped at 0.88
Silver	%	$mp * conc\ gr/HG$, capped at 0.86
Concentrate Considerations		
Copper Deduction	%	1.0
Gold Payable	%	96.0
Silver Payable	%	90.0
Moisture	%	8.0
Treatment Charges	US\$/dt	40.00
Transportation	US\$/wt	125.00
Copper Refining	US\$/lb	0.04
Gold Refining	US\$/oz	5.00
Silver Refining	US\$/oz	0.50

Note: MP = Mass pull in fraction, HG_{Cu} = Copper head grade in percentage points, HG_{Au} = Gold head grade in g/t, HG_{Ag} = Silver head grade in grams per tonne, Con gr Cu = Copper head grade in percentage points, Con gr Au = Gold concentrate grade in grams per tonne.

14.18 Opémiska Mineral Resource Estimate

The 2025 Opémiska MRE is constrained within a pit shell developed from the above-mentioned pit optimization and DSO shapes using appropriate cut-off grades. The effective date of the MRE is May 30, 2025. Table 14-13 presents the results of the estimate. Mineral resources that are not mineral reserves do not have demonstrated economic viability. The QP is not aware of any known environmental, permitting, legal, title-related, taxation, socio-political or marketing issues or any other relevant issues that could materially affect this MRE.

Table 14-13: Opémiska Mineral Resource Estimate

Pit Constrained	Tonnes (k)	CuEq (%)	Cu (%)	Ag (g/t)	Au (g/t)	Cu (M lbs)	Ag (koz)	Au (koz)
0.15% CuEq Cut-Off								
Indicated	62,706	1.04	0.76	1.71	0.31	1,047	3,450	634
Inferred	78,485	0.41	0.26	0.61	0.17	457	1,530	419
Out of Pit	Tonnes (k)	CuEq (%)	Cu (%)	Ag (g/t)	Au (g/t)	Cu (M lbs)	Ag (koz)	Au (koz)
1.00% CuEq Cut-Off								
Indicated	6,947	1.85	1.59	2.76	0.28	243	617	64
Inferred	2,130	0.88	0.69	1.20	0.21	33	82	15
Total	Tonnes (k)	CuEq (%)	Cu (%)	Ag (g/t)	Au (g/t)	Cu (M lbs)	Ag (koz)	Au (koz)
0.15% & 1.00% CuEq Cut-Off								
Indicated	69,653	1.12	0.84	1.82	0.31	1,290	4,067	697
Inferred	80,615	0.42	0.28	0.62	0.17	490	1,613	433

Notes: **1.** The independent qualified persons for the MRE, as defined by National Instrument (NI) 43-101 guidelines, is Pierre Luc Richard, P.Geo., of PLR Resources Inc. with contributions from Stephen Coates, P.Eng., of Evomine for value cut-off, open pit and optimization solids, and Christian Laroche, P.Eng., from Synectiq, for metallurgical parameters. The effective date of the MRE is May 30, 2025. **2.** These mineral resources are not mineral reserves as they do not have demonstrated economic viability. The quantity and grade of reported inferred resources in this MRE are uncertain in nature, and there has been insufficient exploration to define these inferred resources as indicated or measured. However, it is reasonably expected that the majority of inferred mineral resources could be upgraded to the indicated category with continued exploration. **3.** The MRE wireframe was prepared using Leapfrog Edge v. 2024.1.3 and is based on 21,918 drill holes for 1,525,073 meters and 479,242 samples. The drill hole database includes recent drilling (2002 to 2025) of 73,227 meters in 382 drill holes (Ex-In, PowerOre, QC Copper and Gold, XXIX) and also incorporates historical drill holes (1930 to 1990) for 1,451,846 meters in 21,536 drill holes (Opémiska Copper Mines, Falconbridge, Minnova). The cut-off date for the drill hole database was May 16, 2025. **4.** Resources are presented as undiluted and in situ for the open-pit scenario and include internal dilution for the underground scenario and are considered to have reasonable prospects for economic extraction. The constraining pit shell was developed using overall pit slopes of 55 degrees in bedrock and 30 degrees in overburden. The pit optimization to develop the resource-constraining pit shells was done using Deswik Pseudoflow 2024.2. **5.** Composites of 1.5 meters were created inside the high-grade zones and 3.0 meters inside the stockwork zones. High-grade capping was done on the composited assay data; composites were capped at variable grades ranging from 1.00 to 25.00% for Cu, 0.50 to 35.00 g/t for Au, and 10.00 to 120.00 g/t for Ag. **6.** Mineral resources are reported at a cut-off grade of 0.15% CuEq for open-pit resources and 1.00% CuEq for underground resources. All material within the underground stopes is being reported, including internal dilution. The cut-off grades will be re-evaluated in light of future prevailing market conditions and costs. **7.** Specific gravity values were estimated using data available in the drill hole database. Values assigned per zone and per host rock. Surrounding barren lithologies were assigned the average specific gravity value from all measured samples available. **8.** Grade model resource estimation was calculated from drill hole data using an ordinary kriging (OK) interpolation method in a sub-blocked model using blocks measuring 5 x 5 x 5 m in size and sub-blocks down to 0.625 x 0.625 x 0.625 m. Both OK and inverse square distance (ID2) interpolation methods were tested, resulting in no material difference in the mineral resource estimates. **9.** The indicated and inferred mineral resource categories are constrained to areas where drill spacing is less than 50 m and 120 m, respectively, and show reasonable geological and grade continuity. **10.** Calculations used metric units (meters, tonnes). Metal contents are presented in percent or pounds. Metric tonnages were rounded, and any discrepancies in total amounts are due to rounding errors. **11.** Canadian Institute of Mining, Metallurgy and Petroleum (CIM) definitions and guidelines for mineral resource estimates have been followed. **12.** The QPs are not aware of any known environmental, permitting, legal, title-related, taxation, socio-political or marketing issues or any other relevant issues that could materially affect this MRE.

Table 14-14 shows the sensitivity of the block model to grade cut-off for the Indicated in-pit MRE. Table 14-15 shows the sensitivity of the block model to grade cut-off for the inferred in-pit mineral resource estimate. Higher cut-off grades significantly increase the average grade of the deposit, as expected, with a complementary drop in tonnage.

The reader is cautioned that, although they represent reasonable RPEEE scenarios under plausible future economic scenarios, the numbers in the tables should not be misconstrued with a mineral resource statement.

Figure 14-13 shows a 3D view of the mineralized zones within the MRE pit shell. Figure 14-14 shows a cross-section of the mineralized zones and block model within the MRE pit shell.

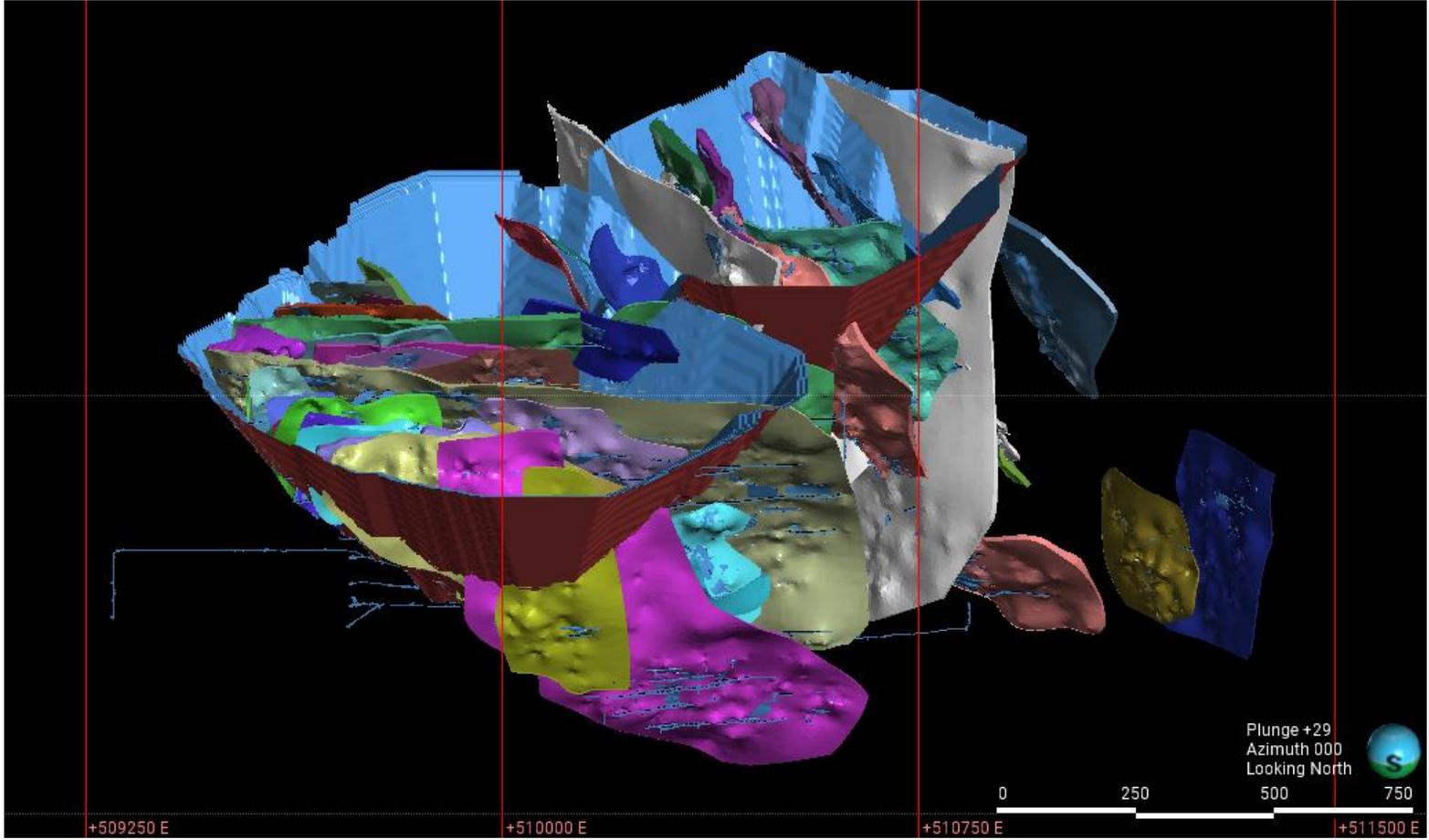
Table 14-14: Pit-Constrained Indicated Resources at Various Cut-off Grades

Pit Constrained	Tonnes (k)	CuEq (%)	Cu (%)	Ag (g/t)	Au (g/t)	Cu (M lbs)	Ag (koz)	Au (koz)
Indicated Resources								
0.10% CuEq Cut-Off	66,028	0.99	0.72	1.64	0.30	1,053	3,481	639
0.15% CuEq Cut-Off	62,706	1.04	0.76	1.71	0.31	1,047	3,450	634
0.20% CuEq Cut-Off	59,350	1.09	0.79	1.79	0.33	1,038	3,411	627
0.25% CuEq Cut-Off	56,098	1.14	0.83	1.86	0.34	1,027	3,361	619
0.30% CuEq Cut-Off	52,875	1.19	0.87	1.94	0.36	1,013	3,300	611

Table 14-15: Pit-Constrained Inferred Resources at Various Cut-off Grades

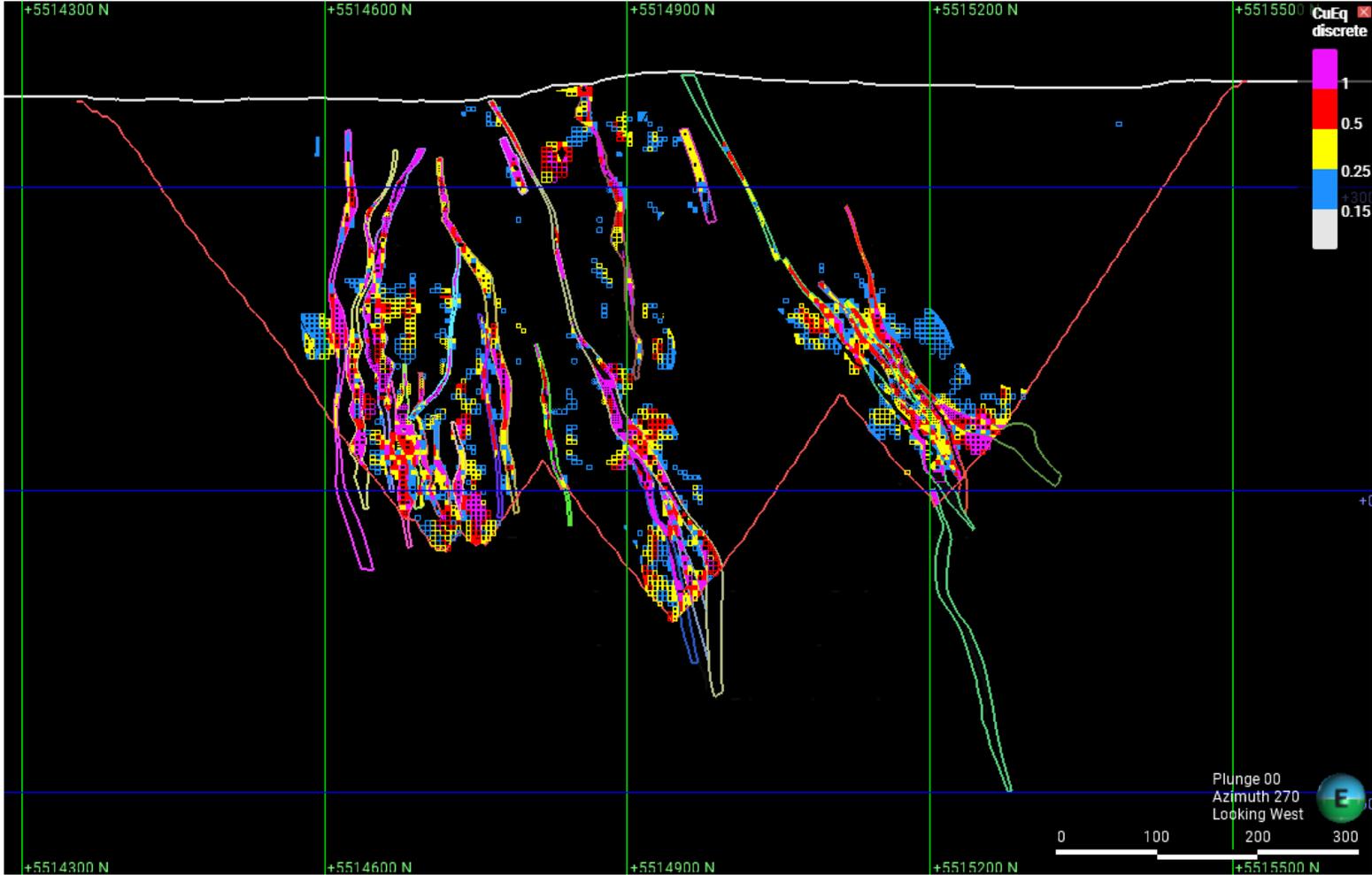
Pit Constrained	Tonnes (k)	CuEq (%)	Cu (%)	Ag (g/t)	Au (g/t)	Cu (M lbs)	Ag (koz)	Au (koz)
Inferred Resources								
0.10% CuEq Cut-Off	115,913	0.32	0.20	0.50	0.13	512	1,849	494
0.15% CuEq Cut-Off	78,485	0.41	0.26	0.61	0.17	457	1,530	419
0.20% CuEq Cut-Off	57,452	0.50	0.32	0.70	0.20	408	1,289	367
0.25% CuEq Cut-Off	44,126	0.58	0.38	0.77	0.23	367	1,097	328
0.30% CuEq Cut-Off	35,006	0.66	0.43	0.84	0.26	331	945	296

Figure 14-13: 3D View of the Mineralized Zones and The Pit Shell (Down Plunge looking North)



Source: PLR, 2025.

Figure 14-14: Cross-Section View of the CuEq Grade within the Pit Shell



Note: Only blocks within the pit shell are shown. Source: PLR, 2025.

15 MINERAL RESERVE ESTIMATES

This section is not applicable to the technical report.

16 MINING METHODS

16.1 Mining Overview

The preliminary economic assessment is preliminary in nature. 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 PEA would be realized.

The project is planned as a conventional truck-and-shovel open pit mining operation. The nominal processing rate is set at 12,500 t/d over a 17-year mine life, with an average strip ratio of 3.7 to 1. Two ultimate pits will be mined over the life of mine, the Springer pit and the Perry pit, with interim pits designed within these ultimate pits to optimize the mineralized material grade and strip ratio extraction profile. Mined physical quantities represent 77.2 Mt of mineralized material, 270.7 Mt of waste and 15.0 Mt of overburden segregated by a block model reblocked to 5 m x 5 m x 5 m dimensions to adequately consider selectivity and associated mining dilution for the envisioned mining equipment. Four mining phases are planned and detailed as follows: starter pits in both Springer and Perry (Phase 1), an intermediate pushback in Springer (Phase 2), the depletion of Perry (Phase 3), and the depletion of Springer (Phase 4). The 17-year life of mine incorporates 13 years of direct mill feed from open pit operations and four years of stockpile rehandling. The mining rate is expected to peak at 116,000 t/d and average 76,000 t/d over the 13 years of mining. The open pit operation has also been optimized to push any impact to the neighbouring town of Chapais to the end of Phase 3 and beginning of Phase 4. Also considered are areas dedicated to overburden, waste rock, and mineralized material stockpiling.

16.2 Geotechnical Considerations

No geotechnical studies have been completed to date to evaluate rock mass conditions and establish pit slope design criteria. A geotechnical site investigation program is recommended as part of the next steps in developing the project. Prior to initiating this study, a detailed survey of a historical underground stope breaking through the surface was conducted. This survey illustrates long term stability for quasi vertical walls over a height of over 60 m. Pit design criteria was established based on benchmarking of open pit operations within the region of the project and assumes favourable ground conditions.

16.3 Hydrogeological Considerations

No hydrogeological studies have been completed to date to assess groundwater conditions at this stage of the project. A hydrogeological site investigation program is recommended as part of the next steps in developing the project. Hydrogeological recommendations and guidelines should be provided based on these findings. It has been assumed that groundwater infiltrations into the open pit will not add significant dewatering requirements relative to the precipitation expected. Provisions have been included at the beginning of operation to consider the dewatering of historical underground voids connected to the open pit.

16.4 Pit Optimization

Open pit optimization was conducted to determine the optimal economic shape of the open pit and guide the open pit design process. This task was performed utilizing Deswik.CAD’s pseudoflow algorithm. The algorithm progressively constructs lists of blocks that should or should not be mined, based on their economic value. The optimization process defines an open pit outline that maximizes total economic value while adhering to the required open pit slopes and other parameters. The optimizations performed to generate optimal limits to guide the ultimate open pit design were based on valuing measured, indicated, and inferred mineral resource category blocks.

By varying the economic parameters while keeping inputs for metallurgical recoveries and pit slopes constant, various generated pit cases are evaluated to determine if incremental pit shells produce marginal or negative economic returns. This drop-off is due to increasing waste mining ratios, decreasing metal grades, increased mining costs associated with the larger or deeper pit shells, and the value of discounting costs before revenues. The economic margins from the expanded cases are evaluated on a relative basis to provide payback on capital and produce a return for the project. At some point, further expansion does not provide significant added value. A pit limit can then be chosen that has suitable economic return for the deposit.

16.4.1 Key Assumptions/Basis of Estimate

A summary of the open pit optimization parameters for a nominal processing rate of 12,600 t/d are presented in Tables 16-1 and 16-2.

Table 16-1: Open Pit Optimization Parameters

Parameter	Unit	Value
Revenue		
Royalty	%	1.00
Operating Cost		
Mining Cost	US\$/t mined	3.00
Process Cost	US\$/t milled	9.00
General & Administration Cost	US\$/t milled	2.25
Mineralized Material-Based Costs	US\$/t milled	11.25
Mining		
Block Size	m	5 x 5 x 5
Slope Angle – Rock	degrees	55
Slope Angle – Overburden	degrees	30
Processing		
Capacity	Mt/a	4.6
Capacity	t/d	12,600
Cut-off Grade		
NSR Cut-off	US\$/t milled	11.36

Table 16-2: Open Pit Optimization NSR Parameters

Parameter	Unit	Value
Price Assumptions		
Copper	US\$/lb	4.00
Gold	US\$/oz	2500.00
Silver	US\$/oz	27.00
Recovery Assumptions		
Copper	%	82.6
Gold	%	75.5
Silver	%	63.8
Copper Concentrate		
Copper Concentrate Grade	%	25.0
Copper Deduction	%	1.0
Gold Payable	%	96.0
Silver Payable	%	90.0
Moisture	%	8.0
Treatment Charges	US\$/dt	40.00
Transportation	US\$/wt	125.00
Copper Refining	US\$/lb	0.04
Gold Refining	US\$/oz	5.00
Silver Refining	US\$/oz	0.50
NSR Value by Grade		
Copper	US\$/%	63.45
Gold	US\$/g/t	58.14
Silver	US\$/g/t	0.49

Metal prices were set at US\$4.00/lb of copper, US\$2,500.00/oz of gold and US\$27.00/oz of silver. Recoveries were assumed to be 82.6% for copper, 75.5% for gold and 63.8% for silver. Other parameters including concentrate grade assumptions, copper deductions, precious metal payabilities, concentrate transport costs, treatment charges, and metal refining costs were considered to calculate an NSR value per block. Mining costs were benchmarked at \$3.00/t. The total mineralized material-based cost that was applied, inclusive of processing, general and administrative expenses, was \$11.25/t. The cut-off grade was estimated at an NSR value of \$11.36/t. Revenues were varied from 30% to 110% of base case values.

16.4.2 Dilution

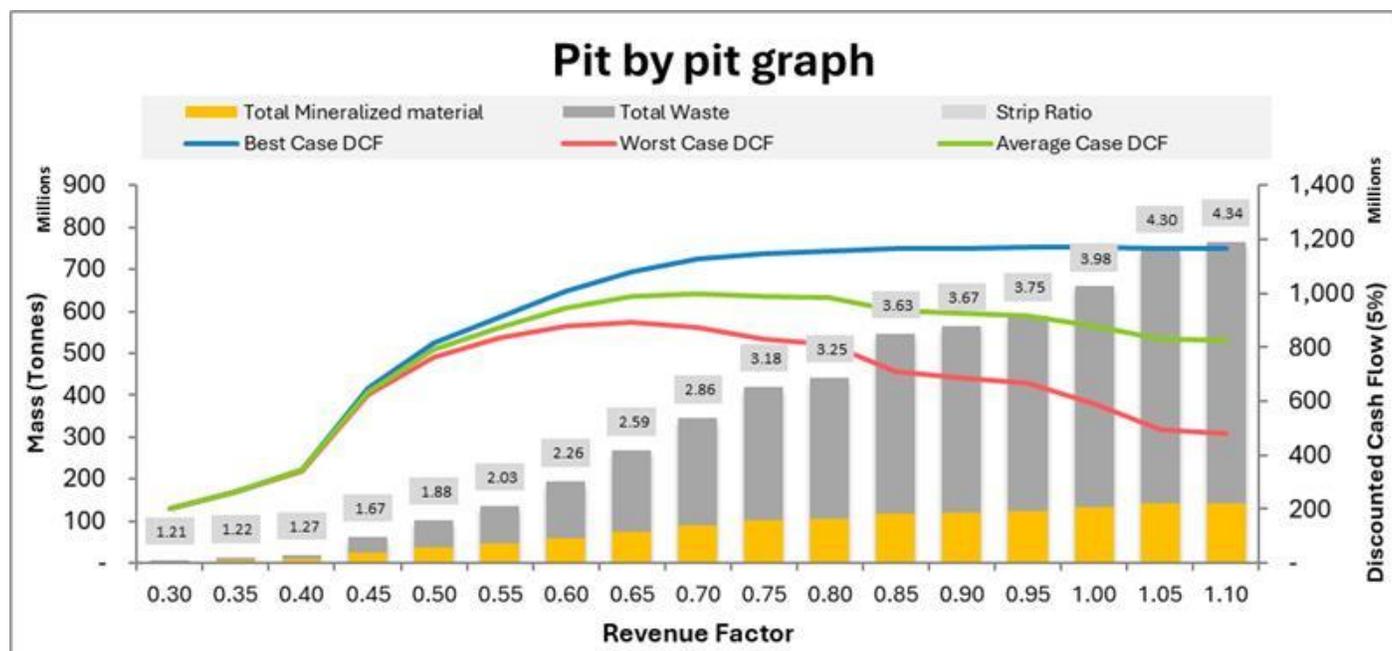
The mineral resource block model was imported to the Deswik.CAD™ software as a single block model. The model provided was regularized and reblocked into a 5 x 5 x 5 m block model to consider mining dilution and mineralized material loss. The evaluation of the potentially economic portion of the MRE, referred to as mineralized material, includes all categories of mineral resources: measured, indicated, and inferred.

16.4.3 Ultimate Pit Shell

Figure 16-1 and Table 16-3 show the contents of the generated Pseudoflow nested pit shells for the project. The figure includes the cumulative mineralized material, waste material, best case discounted cashflow (DCF), worst case DCF

and average DCF. The best case DCF considers that the pits are mined shell by shell, while the worst case DCF considers that the pits are mined bench by bench. The average DCF is the average of the best and worst case DCF. The revenue factor (RF) 0.75 pit is selected as the PEA project limits, which represents a 22-year project life at 12,600 t/d mill throughput. Shells beyond the 0.75 RF case do not produce increased project value, but as the project develops, a larger pit shell could generate additional value to the project. Once several variables were assessed, including overall pit size, stripping ratio metrics, and discounted economics performance, the pit shell generated from case RF 0.75 was selected as the ultimate pit shell for the project. This pit shell selection was used for further mine planning as a target for detailed open pit designs with berms and ramps and represents two pits, the Springer pit and the Perry pit.

Figure 16-1: Open Pit Optimization Results



Source: Evomine, 2025.

Table 16-3: Open Pit Optimization Results

Open Pit Shell	Revenue Factor	Best Case Value @ 8% (\$M)	Average Case Value @ 8% (\$M)	Worst Case Value @ 8% (\$M)	Mineralized Material (kt)	Waste Tonnage (kt)	Total Tonnage (kt)	Mineralize Material NSR (US\$/t)	Strip Ratio
1	0.30	203.1	203.1	203.1	3,466	4,188	7,714	78.65	1.21
2	0.35	266.2	265.2	264.2	5,358	6,534	11,964	70.00	1.22
3	0.40	346.2	343.4	340.5	8,273	10,524	18,871	62.75	1.27
4	0.45	645.9	634.9	623.9	23,271	38,870	62,236	52.69	1.67
5	0.50	815.7	790.0	764.3	35,672	67,098	102,870	49.77	1.88
6	0.55	912.6	873.2	833.7	45,218	91,894	137,221	48.27	2.03
7	0.60	1,009.5	945.1	880.8	59,312	134,314	193,752	46.51	2.26
8	0.65	1,081.5	988.2	894.9	74,517	193,076	267,731	45.50	2.59
9	0.70	1,126.5	999.3	872.1	89,270	255,381	344,797	44.66	2.86
10	0.75	1,149.2	988.6	828.1	100,614	319,692	420,462	44.41	3.18
11	0.80	1,154.7	981.8	809.0	104,016	338,375	442,552	44.27	3.25
12	0.85	1,166.7	937.9	709.1	118,092	428,531	546,793	43.86	3.63
13	0.90	1,168.6	928.7	688.7	121,186	444,629	565,988	43.63	3.67
14	0.95	1,169.5	917.3	665.2	123,980	464,866	589,020	43.51	3.75
15	1.00	1,169.5	880.5	591.5	132,311	526,639	659,130	43.14	3.98
16	1.05	1,167.6	832.2	496.7	141,658	609,266	751,110	42.95	4.30
17	1.10	1,167.2	824.3	481.3	143,473	622,790	766,451	42.84	4.34

16.4.4 Interim Pit Shells and Strategic Optimization

To determine the phasing or interim pits within the ultimate pit, a pre-design schedule was built using the Deswik.GO software, which relies on mixed-integer linear programming to solve a multi-period plan for maximum net present value (NPV). The process below was followed:

1. generate shells based on revenue factors (typical Pseudoflow shells)
2. select a final shell and generate rough mining phases (ramp-less top-down projections)
3. select grade bins, define project constraints and run a bench-by-bench life-of-mine plan
4. output physicals and confirm NPV/IRR in an Excel financial model
5. re-iterate steps 2 to 4 until a mine plan is satisfactory (maximal NPV with realistic constraints).

Some of the most relevant assumptions and constraints used in the optimization were:

- minimum phase mining width of 150 m; maximum vertical advance rate of 80 m/a
- total maximum mining rate of 102,700 t/d, and a mill process rate of 12,600 t/d
- three long-term stockpiles available for grade segregation, with unlimited capacity (30 Mt stored at the peak)
- total housing relocation once mining comes within 150 m of any house, which the mine plan delays as much as possible.

The strategic optimization allowed the definition of three interim pits before reaching the ultimate Springer pit and two interim pits before reaching the ultimate Perry pit. It also allowed the determination of a grade binning strategy described in Table 16-4.

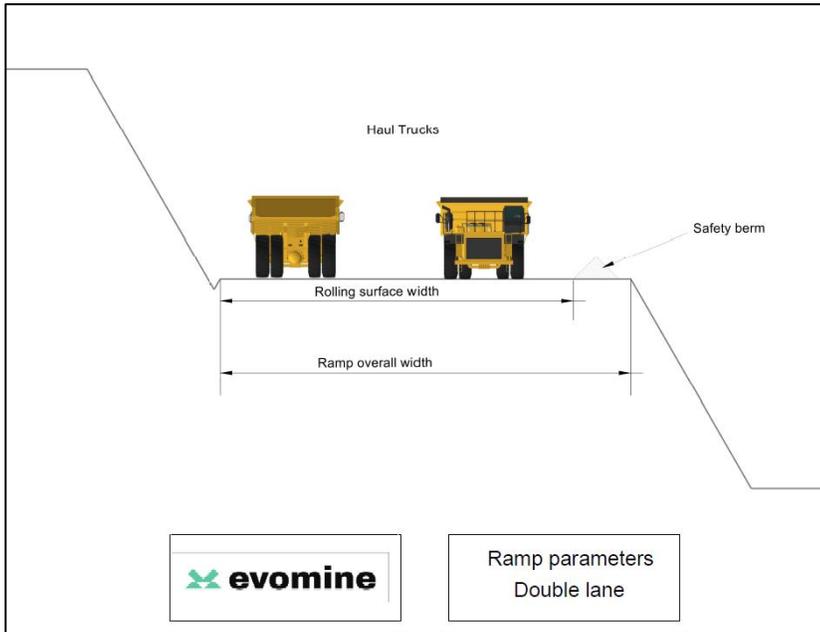
Table 16-4: Grade Binning

Material	Abbreviation	NSR Cut-off (US\$/t)
Super High-Grade Mineralized Material	SHG	30.00+
High-Grade Mineralized Material	HG	20.00-30.00
Medium-Grade Mineralized Material	MG	15.00-20.00
Low-Grade Mineralized Material	LG	12.30-15.00

16.5 Pit Design

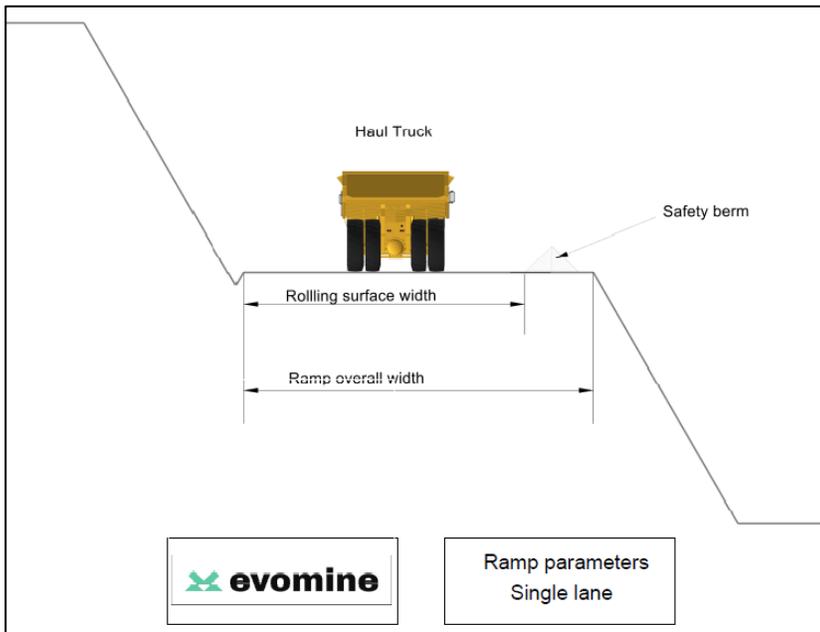
Detailed pit designs were executed based on the selected ultimate pit shells and interim pit phases. Ramp configurations were designed to accommodate 139-tonne class off-highway trucks. The ramps were designed for double-lane traffic, except for the six final benches of the pits, 30 vertical meters, where it transitions to one-way access. Single-lane ramps have an overall width of 19.1 m and double-lane ramps have an overall width of 26.7 m. Figures 16-2 and 16-3 illustrate the single-lane and double-lane ramp configurations applied to the mine designs. The ramp gradient considered is 10% with a minimum turning radius of 20 m.

Figure 16-2: Double-Lane Ramp Configuration



Source: Evomine, 2025.

Figure 16-3: Single-Lane Ramp Configuration



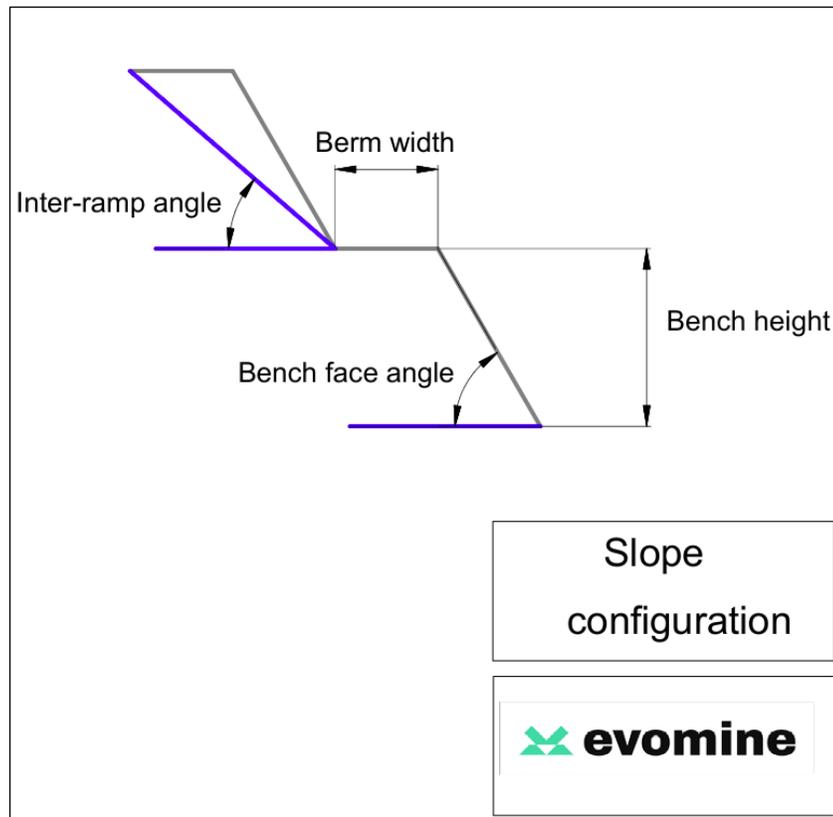
Source: Evomine, 2025.

The preliminary pit slope configuration established for open pit mining are highlighted in Table 16-5 and Figure 16-4. Pit design slope design criteria were established based on benchmarking of open pit operations within the region of the project and assumes favourable ground conditions.

Table 16-5: Pit Slope Configuration

Wall	Bench Height (m)	Bench Face Angle (°)	Berm Width (m)	Inter-ramp Angle (°)
Rock	20	80	10.2	55
Overburden	20	25	7.1	22

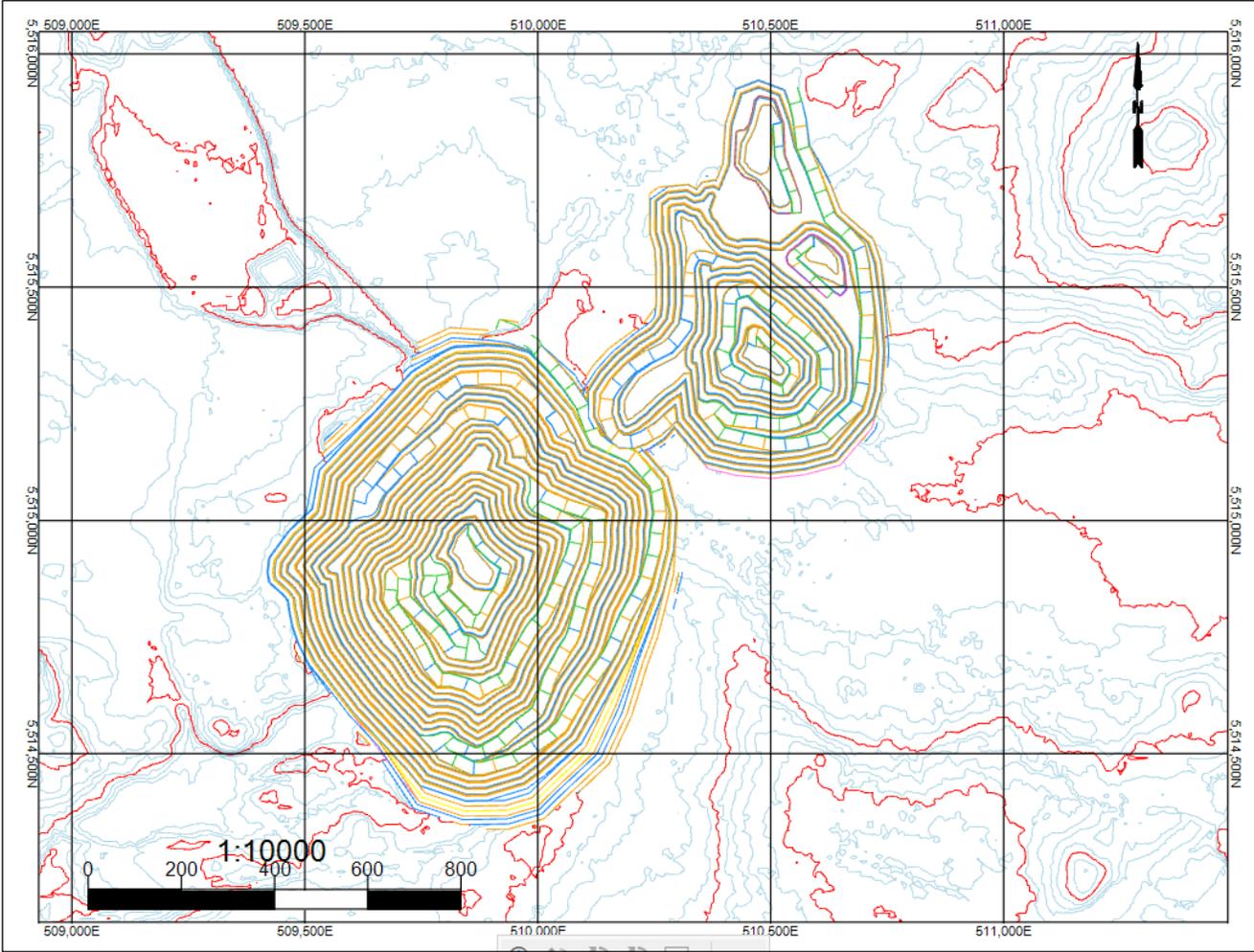
Figure 16-4: Slope Configuration



Source: Evomine, 2025.

Figure 16-5 illustrates the ultimate pit design for both Perry and Springer; Table 16-6 highlights the pit physicals within these ultimate pit designs. The ultimate Springer pit is expected to encroach on the current town of Chapais.

Figure 16-5: Ultimate Pit Design



Source: Evomine, 2025.

Table 16-6: Pit Physical Quantities

Pit	Total (Mt)	ROM (Mt)	ROM Cu Grade (%)	ROM Au Grade (g/t)	ROM Ag Grade (g/t)	Waste Rock (Mt)	Overburden (kt)	Strip Ratio
Springer	283.6	62.5	0.45	0.27	1.19	209.3	11.8	3.5
Perry	79.2	14.7	0.61	0.10	0.83	61.4	3.1	4.4
Total	362.9	77.2	0.48	0.23	1.12	270.7	15.0	3.7

16.6 Production Schedule

The life-of-mine production schedule for the open pit operations was developed in Deswik.Sched with the following considerations:

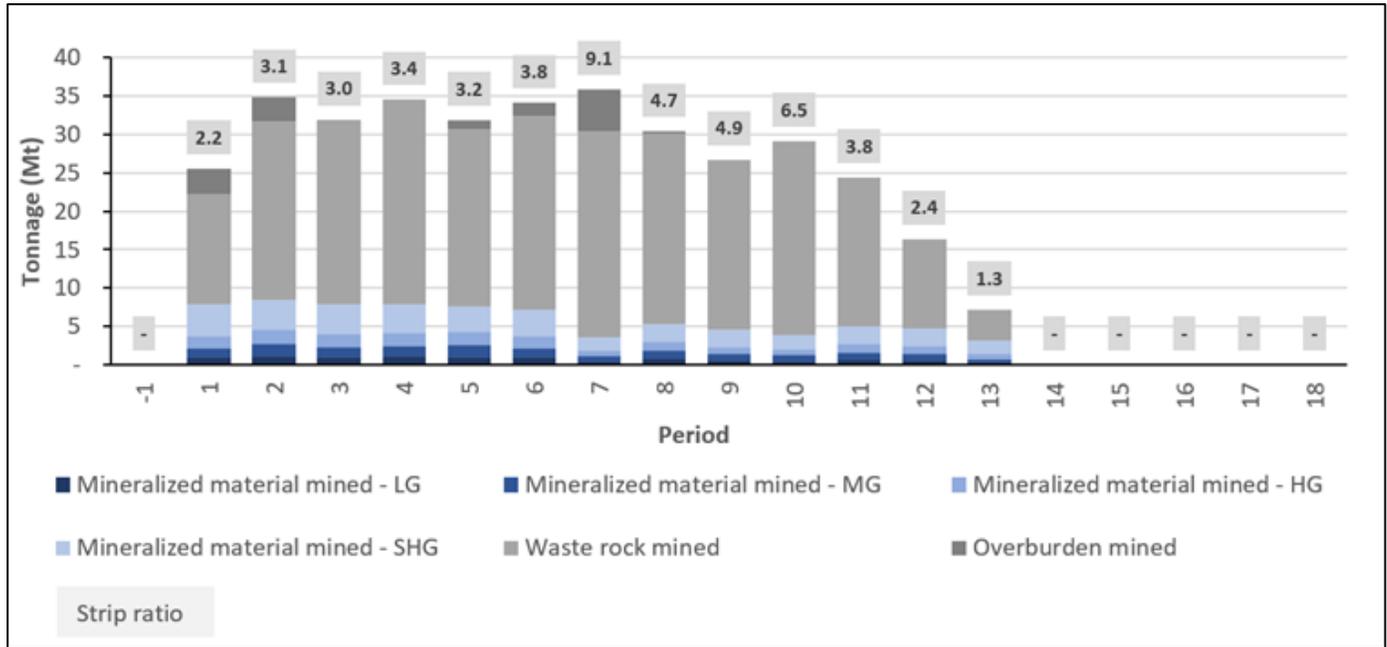
- scheduled on monthly periods
- targeted mill feed rate of 12,500 t/d
- pit vertical progression limited to 80 m annually
- material is segregated and stockpiled by grade bin
- material is fed to the mill to process higher value material upfront.

Four mining phases are planned and detailed as follows: starter pits in both Springer and Perry (Phase 1), an intermediate pushback in Springer (Phase 2), the depletion of Perry (Phase 3), and the depletion of Springer (Phase 4).

The 17-year life of mine incorporates 13 years of direct mill feed from open pit operations and four years of stockpile rehandling. The mining rate is expected to peak at 116,000 t/d and average 76,000 t/d over the 13 years of mining. The open pit operation has also been optimized to push any impact to the neighbouring town of Chapais to the end of Phase 3 and beginning of Phase 4.

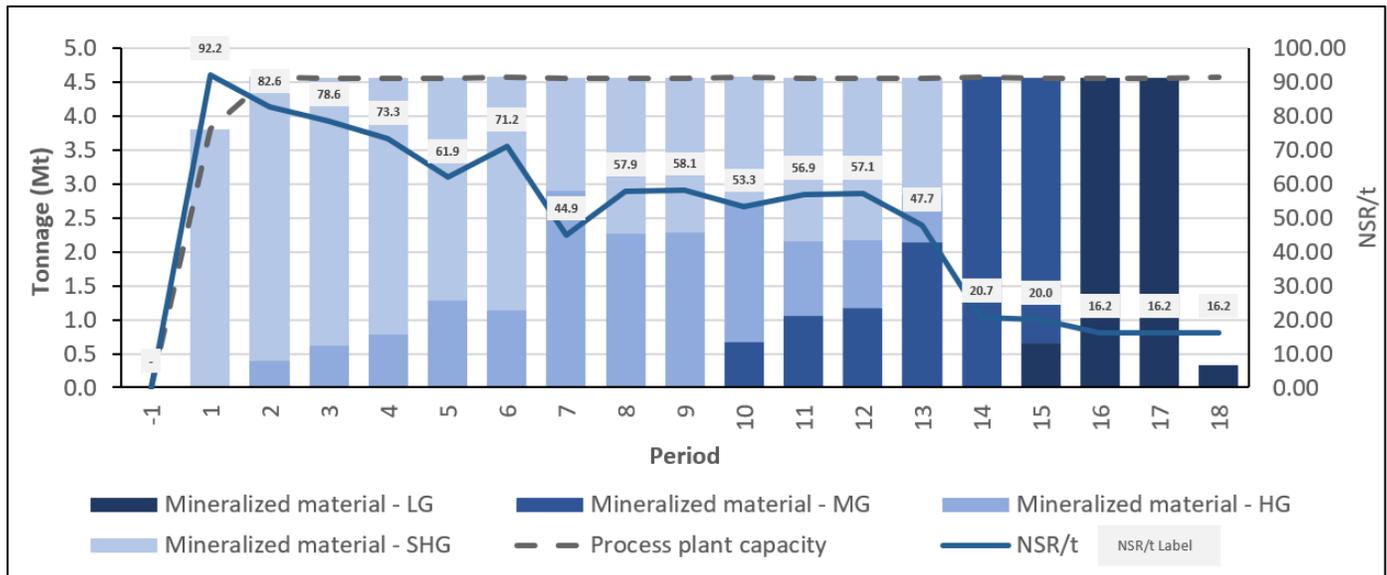
Figure 16-6 and Table 16-7 highlight the material movement in the mine plan. The project has been optimized by sequencing the open pit mining extraction schedule in four distinct phases and by segregating mineralized material to process higher value material upfront as highlighted in the mill feed schedule presented in Figure 16-7.

Figure 16-6: Mining Schedule by Material Type



Source: Evomine, 2025.

Figure 16-7: Mill Feed Schedule



Source: Evomine, 2025.

Table 16-7: Annual Mining Schedule

Mine Production	Units	LOM	Y1	Y2	Y3	Y4	Y5	Y6	Y7	Y8	Y9	Y10	Y11	Y12	Y13	Y14	Y15	Y16	Y17	Y18
Total Material Mined	Mt	362.9	25.5	34.8	31.9	34.6	31.9	34.1	35.9	30.4	26.7	29.1	24.4	16.4	7.2	-	-	-	-	-
Mineralized Material Mined – Low-Grade	Mt	10.1	0.9	1.2	1.0	1.1	1.1	0.9	0.5	0.9	0.6	0.5	0.7	0.6	0.3	-	-	-	-	-
Mineralized Material Mined – Medium-Grade	Mt	13.5	1.2	1.5	1.3	1.4	1.5	1.3	0.6	1.0	0.7	0.7	0.9	0.8	0.5	-	-	-	-	-
Mineralized Material Mined – High-Grade	Mt	16.5	1.6	1.8	1.7	1.6	1.8	1.6	0.8	1.1	0.9	0.8	1.1	1.0	0.7	-	-	-	-	-
Mineralized Material Mined – Super High-Grade	Mt	37.0	4.1	3.9	3.9	3.8	3.3	3.4	1.7	2.3	2.3	1.9	2.4	2.4	1.7	-	-	-	-	-
Total Mineralized Material Mined	Mt	77.2	7.9	8.4	8.0	7.9	7.6	7.2	3.5	5.3	4.5	3.9	5.1	4.8	3.2	-	-	-	-	-
Total Mineralized Material Copper Grade	%	0.48	0.54	0.46	0.45	0.42	0.43	0.55	0.55	0.54	0.55	0.36	0.42	0.46	0.58	-	-	-	-	-
Total Mineralized Material Gold Grade	g/t	0.23	0.25	0.25	0.29	0.26	0.18	0.17	0.08	0.14	0.20	0.41	0.30	0.28	0.22	-	-	-	-	-
Total Mineralized Material Silver Grade	g/t	1.12	1.85	1.44	1.14	1.28	1.02	0.62	0.74	0.95	1.07	1.17	1.16	0.96	0.27	-	-	-	-	-
Waste Rock Mined	Mt	270.7	14.3	23.3	23.8	26.6	23.2	25.3	26.9	24.9	22.2	25.2	19.3	11.6	4.0	-	-	-	-	-
Overburden Mined	Mt	15.0	3.3	3.1	0.1	-	1.1	1.6	5.5	0.2	-	-	-	-	-	-	-	-	-	-
Strip Ratio	ratio	3.7	2.2	3.1	3.0	3.4	3.2	3.8	9.1	4.7	4.9	6.5	3.8	2.4	1.3	-	-	-	-	-
Mine to Mill Feed	Mt	45.9	3.5	4.2	4.1	4.5	4.0	3.6	2.4	2.9	2.7	3.0	4.2	3.9	2.9	-	-	-	-	-
Mine to Stockpile	Mt	31.4	4.4	4.2	3.9	3.5	3.6	3.6	1.1	2.4	1.8	0.9	0.8	0.9	0.3	-	-	-	-	-
Stockpile to Mill Feed	Mt	31.4	0.4	0.4	0.5	0.1	0.6	1.0	2.1	1.7	1.9	1.6	0.3	0.7	1.7	4.6	4.6	4.6	4.6	0.3
Mill Feed	Mt	77.2	3.8	4.6	4.6	4.6	4.6	4.6	4.6	4.6	4.6	4.6	4.6	4.6	4.6	4.6	4.6	4.6	4.6	0.3
Mill Feed NSR	US\$/t	52.93	92.15	82.59	78.57	73.25	61.91	71.16	44.95	57.94	58.14	53.29	56.91	57.10	47.71	20.69	20.05	16.24	16.24	16.24
Feed Copper Grade	%	0.48	0.84	0.75	0.65	0.61	0.59	0.74	0.50	0.62	0.58	0.36	0.45	0.48	0.46	0.18	0.17	0.13	0.13	0.13
Feed Gold Grade	g/t	0.23	0.40	0.36	0.41	0.38	0.24	0.23	0.11	0.16	0.21	0.37	0.32	0.29	0.19	0.10	0.10	0.08	0.08	0.08
Feed Silver Grade	g/t	1.12	2.69	2.12	1.53	1.71	1.32	0.84	0.86	1.20	1.18	1.15	1.25	1.00	0.39	0.58	0.57	0.47	0.47	0.47

16.7 Mining Operation

16.7.1 Drilling and Blasting

A single drilling pattern has been assumed at this stage of the project. Table 16-8 highlights the main drilling and blasting parameters. Diameter holes at 8 inches will be drilled for both mineralized material and waste rock by a blasthole rotary drill operated by the mine. A peak of six drill rigs is expected over the life of mine. Holes will be drilled at 10 m, with an additional subdrill of 1.5 m. The drilling pattern was established at 5.5 m burden and a 6.5 m spacing.

Table 16-8: Drilling and Blasting Parameters

Item	Unit	Value
Explosive Density	g/cm ³	1.20
Hole Diameter	in	8.00
Burden	m	5.50
Spacing	m	6.50
Subdrill	m	1.50
Stemming	m	4.00
Bench Height	m	10.00
Blasthole Length	m	11.50
Rock Density	t/BCM	2.70
Yield per Hole	BCM/hole	358
Yield per Hole	t/hole	965
Yield per Meter Drilled	t/m drilled	84
Explosive Column	m	7.50
Weight of Explosive per Hole	kg	291.86
Powder Factor	kg/t	0.30
Powder Factor	kg/BCM	0.82
Re-drills	%	5
Pure Penetration Rate	m/hr	31.6
Overall Drilling Factor	%	50
Overall Penetration Rate	m/h	15.8
Drilling Efficiency	t/h	1,263
Drilling Efficiency	Holes/h	1.31

Bulk emulsion will be used with powder factors of 0.30 kg/t in or 0.82 kg/BCM. The blast holes will be initiated with electronic detonators paired with 450 g boosters. Explosives will be supplied by a third-party provider who will be responsible for supplying and delivering explosives into emulsion trucks. A contractor-operated blasting team will oversee the loading and blasting activities. The technical services department will be responsible for designing blast patterns.

16.7.2 Loading

The loading fleet consists of 12.4 m³ diesel hydraulic excavators in backhoe configuration operated by the mine. Auxiliary front-end loaders will also be available to assist with loading activities when necessary. The shovels will be used to load both waste and mineralized material, while the front-end loader will be primarily used for rehandling the mineralized stockpile material and to supplement mineralized material loading as needed. The shovel fleet is expected to peak at three units, and the front-end loader fleet is expected to peak at three units over the life of mine.

16.7.3 Hauling

Haulage will be performed by 140-tonne class off-highway mining trucks for waste and mineralized material operated by the mine. The mineralized material will be hauled to the crusher or the mineralized material stockpile located near the process plant. The waste and overburden material will either be hauled and stockpiled in their respective storage areas or used for construction purposes.

The truck requirements have been estimated using a set of assumptions that were based on similar operations:

- When loaded, the average speed will be 10 km/h when going uphill and 30 km/h when on a flat road.
- When empty, the average speed will be 40 km/h both when on a flat road and when going downhill.
- The average duration for queuing, spotting, loading, and dumping will be 2.0 min, 0.5 min, 5.0 min and 1.0 min, respectively.
- The number of truck units is expected to peak at 18 over the life of mine.

16.7.4 Support Equipment

Support equipment requirements are based on typical open pit mine operation and maintenance requirements to safely support the drilling, loading, and hauling fleets. Support equipment is planned for maintaining dump areas, stockpiles, pit floors, ditches, and mine roads. The mine will be providing the support equipment, which include track dozers, excavators, front-end wheel loaders, fuel trucks, water trucks.

16.7.5 Dewatering

Due to the characteristics of the open pits and the current lack of site-specific hydrogeological data available, dewatering requirements in line with those of large, open pit mines in the Abitibi region have been considered.

16.7.6 Fleet Requirements

Table 16-9 summarizes the number of gross operating hours per year that were used to calculate equipment fleet requirements for major equipment. The mine is expected to operate 24 hours per day, 365 days per year. This accounts for shift changes and 10 days of delay related to weather. Additional delays and applied factors are described in productivity calculations for each fleet as calculated in the table. These factors have been applied to production and support equipment.

Table 16-9: Equipment Usage Assumptions

Item	Unit	Shovels	Loaders	Trucks	Drills
Days in Period	days	365	365	365	365
Weather, Schedule Outages	days	10	10	10	10
Shifts per Day	shifts/day	2	2	2	2
Hours per Shifts	hours/shift	12	12	12	12
Availability	-	0.82	0.80	0.85	0.80
Use of Availability	-	0.90	0.80	0.90	0.90
Utilization	-	0.74	0.64	0.77	0.72
Effectiveness	-	0.87	0.80	0.87	0.87
Overall Equipment Effectiveness	-	0.64	0.51	0.67	0.63
Total Hours	h	8,760	8,760	8,760	8,760
Scheduled hours	h	8,520	8,520	8,520	8,520
Down Hours	h	1,534	1,704	1,278	1,704
Delay Hours	h	817	1,363	817	797
Standby Hours	h	699	1,091	724	682
Operating Hours	h	6,288	5,453	6,518	6,134
Ready Hours	h	5,470	4,362	5,670	5,337

Additional equipment will be procured for maintenance activities and to support the operation (e.g., forklift, telehandler, low-boy trailer, and tractor for moving the tracked equipment). Other small equipment, such as mechanical service trucks, generators, and welding machines, is also included. Table 16-10 presents the maximum open pit and surface equipment requirements for the life of mine.

Table 16-10: Equipment Requirements

Equipment	Maximum Quantity
Definition Drill	1
Pre-Shear Drill	1
Production Drill	6
Explosive Truck	2
Shovel	3
Haul Truck	18
Tailings Truck	2
Front-end Wheel Loader	3
Auxiliary Shovel	1
Track Dozer	2
Motor Grader	2
Articulated Water Truck	1
Articulated Fuel/Lube Truck	1
Lowboy and Tractor	1
Boom Truck	1
Telehandler	2
Forklift	2
Pickup	29
Mechanical Service Truck	1
Personnel Carrier	1
Mobile Welding Machine	1
Lighting Towers	8
6 kW Genset	4
60 kW Genset	2
Dewatering Pump	24
Trash Pump	8
Diesel-Powered Air Heater	4
Bulk Emulsion Depot	1

16.7.7 Maintenance

Maintenance will be performed by the mine’s personnel. The maintenance department has been structured to fully manage this function, performing maintenance planning and training of employees. However, reliance on dealer and manufacturer support will be key for the initial years of the project and major component rebuilds will be supported by the equipment dealer throughout the life of mine. Tire monitoring, rotation and/or replacement will also be conducted internally.

16.7.8 Mine Management and Technical Services

The operating team is responsible for achieving production targets in a safe and efficient manner. The engineering and geology team will support the operations team by providing short-term and long-term planning, drilling, and blasting design, geotechnical engineering, grade control, resource estimation, surveying, and other technical functions.

16.8 Surface Mining Infrastructure

16.8.1 Haul Roads

A road network links open pits, mineralized material stockpiles, tailings and waste co-disposal facility, process plant, and other significant mine infrastructure. The surface haulage roads were designed for double-lane traffic to accommodate 139-tonne class off-highway trucks and have a width of 28.7 m.

16.8.2 Stockpiles

The preliminary stockpile slope configuration established are highlighted Table 16-11 and Figure 16-4. Stockpile slope design criteria were established based on benchmarking of similar operations within the region of the project.

Table 16-11: Stockpile Slope Configuration

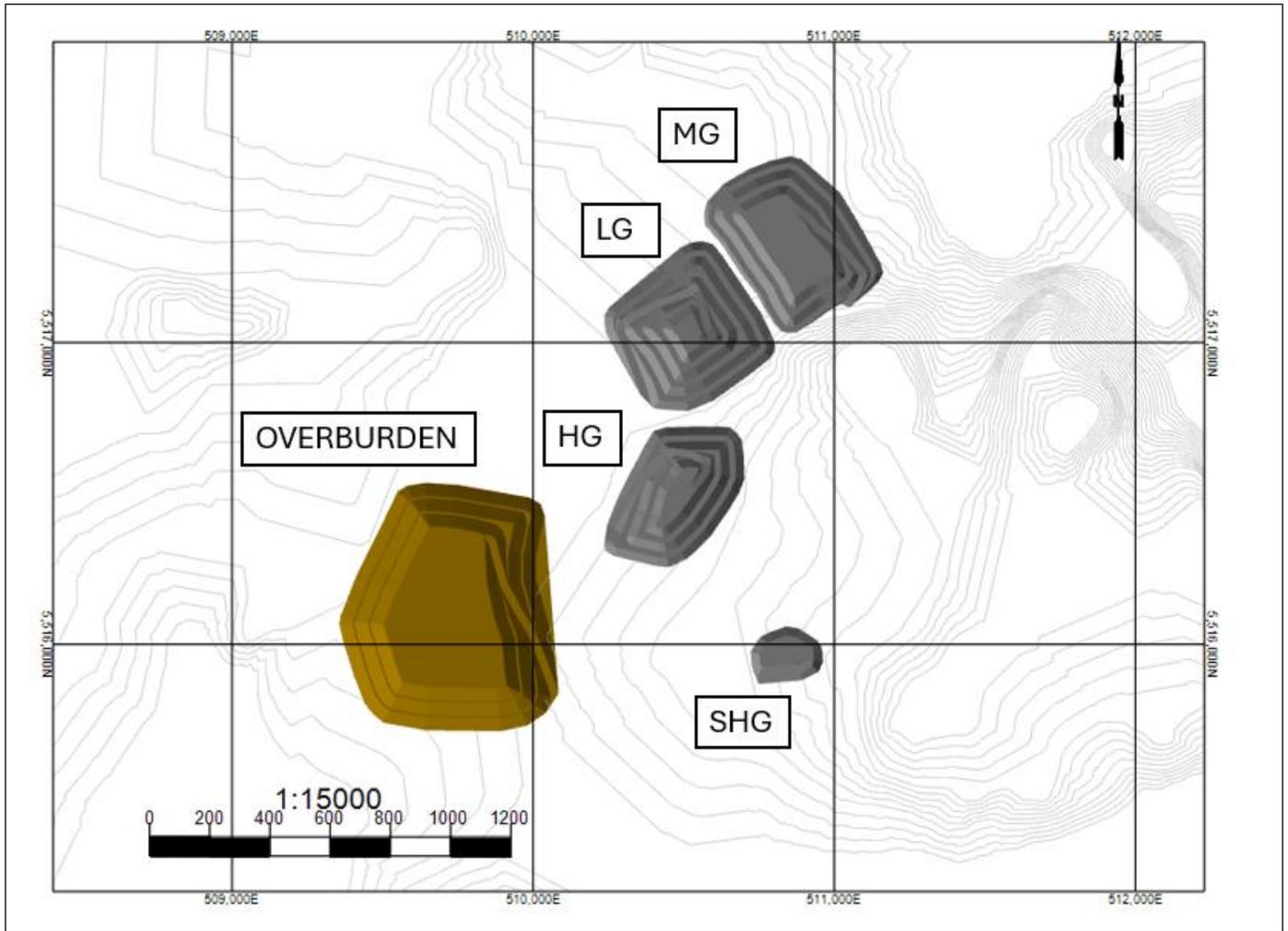
Type	Bench Height (m)	Bench Face Angle (°)	Berm Width (m)	Inter-ramp Angle (°)
Mineralized Material	20	35	20	22
Overburden	20	25	7.1	22

Five different stockpiles were designed for the project:

- An SHG mineralized material stockpile, which can accommodate a volume or 0.4 Mm² of rock, whereas the mine plan shows this stockpile peaking at 0.3 Mt.
- An HG mineralized material stockpile, which can accommodate a volume or 3.9 Mm² of rock, whereas the mine plan shows this stockpile peaking at 5.9 Mt.
- An MG mineralized material stockpile, which can accommodate a volume or 6.5 Mm² of rock, whereas the mine plan shows this stockpile peaking at 10.7 Mt.
- An LG mineralized material stockpile, which can accommodate a volume or 5.9 Mm² of rock, whereas the mine plan shows this stockpile peaking at 10.1 Mt.
- An overburden stockpile, which can accommodate a volume or 16.4 Mm² of rock, whereas the mine plan shows this stockpile peaking at 15.0 Mt.

Figure 16-8 illustrates the design for the different stockpiles. No waste stockpile was designed as waste will be managed in a co-disposal facility along with tailings.

Figure 16-8: Stockpile Design



Source: Evomine, 2025.

16.9 Mining Workforce

Table 16-12 summarizes the peak mining personnel requirements, which represents a total of 291 employees.

Table 16-12: Mining Workforce Requirements

Description	Personnel
Mine Operations Mine Superintendent & General Foreperson	2
Mine Operations Supervisor	12
Mine Operations Shovel Operator	12
Mine Operations Haul Truck Operator	72
Mine Operations Excavator, Loader, Dozer, Grader and Other Support Equipment Operators	30
Mine Operations Drill & Blast	56
Mine Operations Clerks, Trainers, Labour	8
Mine Operations Surface Tailings Truck Operator	8
Grade Control Laborers/Samplers	4
Mine Maintenance – Superintendent & General Foreperson	2
Mine Maintenance – Superintendent, Planner, Clerk	7
Mechanic	40
Electrician	4
Welder/Machinist	4
Fuel and Lube Technician	2
Tool Crib Attendants & Maintenance Helpers	6
Geology – Chief, Geologists & Technician	5
Engineering and Survey including Chief Engineer	17
Total	291

17 RECOVERY METHODS

The process flowsheet developed for the project was selected based on preliminary testwork, historical operating data, and subsequent economic modelling. The process route chosen is the best suited, with unit operations that are standard technologies typically used in concentrator plants. The proposed process plant consists of the following areas:

- comminution circuit consisting of two-stage crushing, a semi-autogenous (SAG) mill, and a ball mill with cyclone classification
- rougher flotation with regrind
- two stages of cleaner flotation
- concentrate handling
- tailings handling.

17.1 Process Design Criteria

The key process design criteria for the plant are listed in Table 17-1. The process plant design is based on a typical metallurgical flowsheet developed for optimum recovery.

17.2 Process Plant Description

The process design has the following major components:

- primary crushing of run of mine mineralized material
- SAG mill grinding
- ball mill grinding with cyclone classification
- rougher flotation with concentrate cyclone classification and regrind
- two stages of cleaner flotation
- concentrate thickening, filtration, drying and stockpiling
- concentrate load-out
- final tailings thickening, filtration, stockpiling, and storage in the tailings storage facility (TSF).

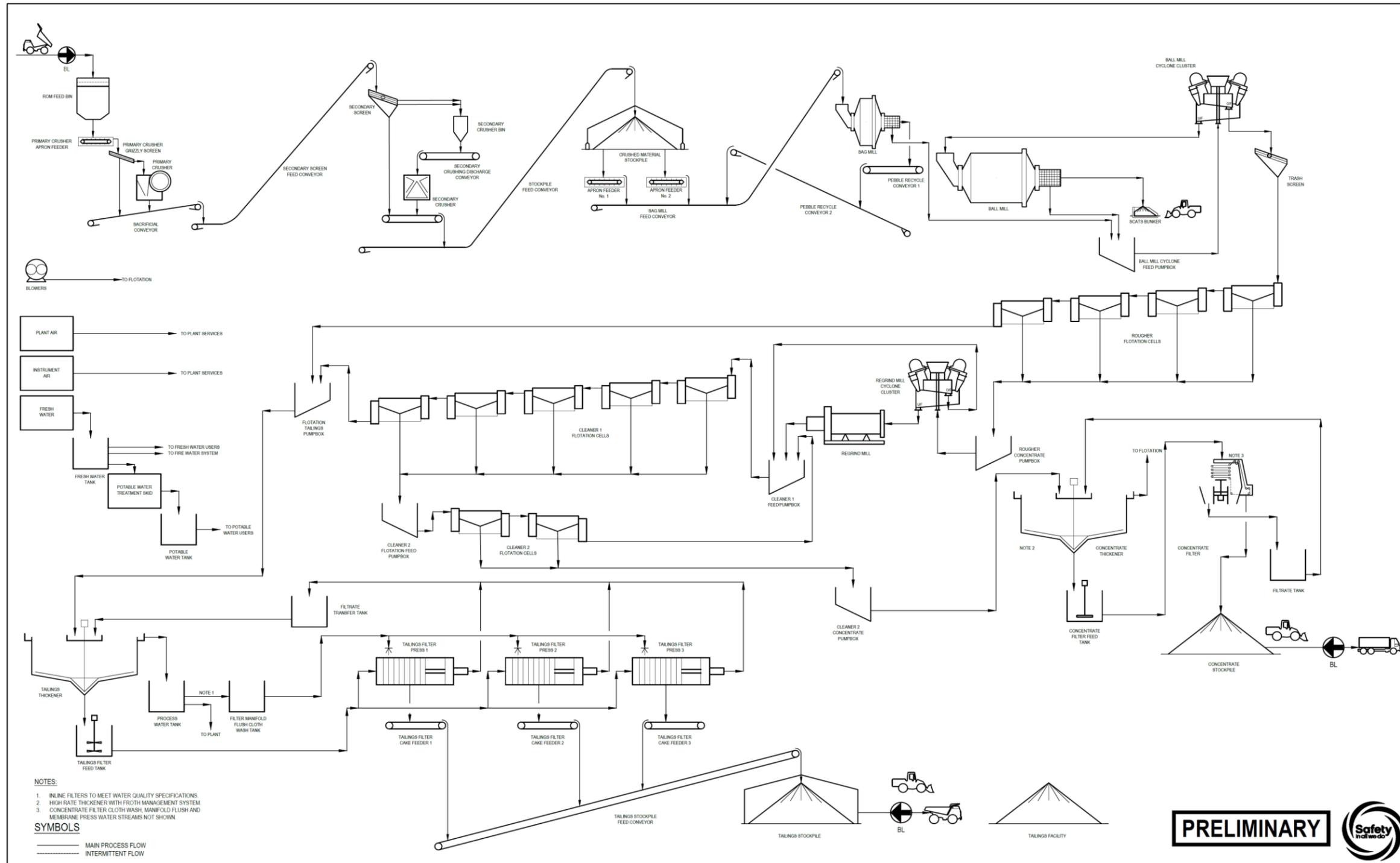
17.2.1 Process Flowsheet

The overall process flow diagram is shown in Figure 17-1.

Table 17-1: Process Design Criteria

Parameter	Units	Value
Annual Process Plant Throughput	Mt/a	4.6
Daily Process Plant Throughput	t/d	12,500
Copper Head Grade, Design	%	0.92
Gold Head Grade, Design	g/t	0.32
Silver Head Grade, Design	g/t	1.41
Life of Mine	y	18
Operating Availability, Crushing	%	65
Operating Availability, Grinding	%	92
Operating Availability, Filtration	%	84
Copper Recovery, Design	%	92
Gold Recovery, Design	%	83
Silver Recovery, Design	%	82
JK SMC Axb, Design	-	26.8
Bond Crushing Work Index (CWi), Design	kWh/t	26.1
Bond Rod Mill Work Index (RWi), Design	kWh/t	23.0
Bond Ball Mill Work Index (BWi), Design	kWh/t	20.0
Bond Abrasion Index, Design	g	0.403
Specific Gravity	-	2.92
Crushing Feed Size, F ₈₀	mm	423
Crushing Product Size, P ₈₀	mm	36
SAG Mill Pebble Recycle Rate, Design	% fresh feed	14
Ball Mill Circulating Load, Design	%	350
Grinding Product Size, P ₈₀	µm	105
Primary Cyclone Overflow Density	% w/w solids	35
Rougher Flotation Stage Recovery, Design	%	11
Regrind Product Size, P ₈₀	µm	30
Regrind Specific Energy	kWh/t	13
Cleaner 1 Flotation Stage Recovery, Design	%	48
Cleaner 2 Flotation Stage Recovery, Design	%	70
Concentrate Thickening Rate, Design	t/m ² /h	0.25
Concentrate Specific Filtration Rate	kg/m ² /h	500
Concentrate Filter Cake Moisture, Target	% w/w	9.5
Tailings Thickening Rate, Design	t/m ² /h	0.75
Tailings Specific Filtration Rate	kg/m ² /h	149
Tailings Filter Cake Moisture, Target	% w/w	15

Figure 17-1: Process Flowsheet



Source: Ausenco, 2025.

17.2.2 Crushing & Coarse Material Stockpiling

The crushing circuit consists of a two-stage crushing process that is intended to reduce the run-of-mine mineralized material from an F_{80} of 423 mm to a P_{80} of 36 mm. The crushing circuit is designed for an annual operating time of 5,694 hours, or 65% availability, at a processing capacity of 808 dry t/h.

The run-of-mine material is hauled from the mine and dumped into a feed bin where it is withdrawn via apron feeder and discharged onto a vibrating grizzly screen. Coarse oversize reports directly to the primary jaw crusher. The vibrating grizzly feeder undersize discharges onto the onto the sacrificial conveyor, where it is combined with the jaw crusher product and transferred to the secondary screen feed conveyor. A rock breaker is employed to fracture material larger than the crusher opening into smaller, more manageable pieces.

The secondary screen feed conveyor feeds the double-deck vibrating secondary screen with top and bottom deck apertures of 60 mm and 28 mm, respectively. The secondary screen oversize feeds the secondary crusher bin, from where the material is conveyed by a belt feeder to the secondary cone crusher. The cone crusher discharge is combined with the secondary screen undersize onto the secondary crushing discharge conveyor for deposition on the crushed material stockpile via the stockpile feed conveyor. The crushed material stockpile has a live capacity, residence time equivalent to 12 hours of mill feed.

Major equipment in this area includes the following:

- primary jaw crusher
- sacrificial conveyor and secondary screen feed conveyors
- secondary screen and cone crusher
- secondary crushing discharge and stockpile feed conveyors

17.2.3 Grinding Circuit

The grinding circuit consists of a SAG mill followed by a ball mill with cyclone classification in closed circuit to reduce the mineralized material to a P_{80} of 105 μm . The grinding circuit is designed to operate at a nominal processing rate of 566 dry t/h at 92% availability.

Apron feeders withdraw material from the stockpile and feed the SAG mill feed conveyor where it is fed into the SAG mill. The SAG mill discharges onto the trommel screen, with the oversize pebbles returning to the SAG mill via transfer conveyors. The undersize material is transferred to the ball mill cyclone feed pumpbox where it is diluted to the appropriate feed density with process water and pumped to the ball mill cyclone cluster for classification. The cyclone overflow, at the target P_{80} of 105 μm and 35% solids w/w, is sent through a trash screen before being pumped to the rougher flotation circuit.

The cyclone underflow feeds the ball mill, which discharges onto a trommel screen. The trommel screen undersize discharges into the cyclone feed pumpbox for classification, and the oversize material, along with grinding media

fragments, is discharged to the scats bunker for disposal. The circulating load within the ball mill hydrocyclone circuit is designed to be 350%.

Hydrated lime is added to the ball mill to increase the slurry pH to 10.5 ahead of rougher flotation.

Major equipment in this area includes the following:

- SAG mill feed and pebble transfer conveyors
- 28 ft diameter x 23 ft effective grinding length (EGL) SAG mill, 8.3 MW
- 22 ft diameter x 33 ft EGL ball mill, 8.3 MW
- ball mill classification cyclone cluster
- trash screen.

17.2.4 Rougher Flotation and Regrind

The rougher flotation circuit includes four consecutive, conventional 130 m³ rougher flotation cells. The rougher tailings report to the flotation tailings pumpbox and the concentrate is collected in the rougher concentrate pumpbox before being pumped to the open circuit regrind mill cyclone cluster for classification. The regrind mill cyclone overflow, at the target P₈₀ of 30 µm, bypasses the regrind mill and reports directly to the first cleaner feed pumpbox.

The cyclone underflow is fed to the regrind mill for reduction to the target grind size. The regrind mill has an internal classification system that allows the mill to operate in an open-circuit configuration, retaining the grinding media inside and discharging the finished product at the correct size specification. The regrind mill discharge is combined with the regrind mill cyclone overflow in the first cleaner feed pumpbox.

Lime is added to maintain the slurry pH at the 10.5 target. Aero 3894 (thionocarbonate) and methyl isobutyl carbinol (MIBC) is added as a collector and frother to support sulphide mineral collection and froth stability.

Major equipment in this area includes the following:

- four conventional 130 m³ rougher flotation cells
- regrind mill cyclone cluster
- horizontal regrind mill, 1.12 MW.

17.2.5 Cleaner Flotation

Rougher concentrate is subjected to two stages of cleaner flotation to upgrade the concentrate to the target final product quality. The rougher concentrate is first pumped into the first cleaner flotation bank, which consists of five conventional 20 m³ flotation cells. The tailings from the first cleaner flotation is combined with the rougher tailings in the flotation tailings pumpbox and pumped to the tailings thickener.

The first cleaner concentrate is fed to the second cleaner flotation circuit which consists of two conventional 20 m³ cleaner flotation cells. The tailings from the second cleaner flotation circuit is recycled to the first cleaner feed pumpbox to prevent losses from misplacement. The second cleaner concentrate is sent to concentrate dewatering.

Lime is added to the cleaner flotation circuit as a pH modifier, and Aero 2894 and MIBC is added as a collector and frother, respectively.

Major equipment in the cleaner flotation area includes the following:

- five 20 m³ conventional first cleaner flotation cells
- two 20 m³ conventional second cleaner flotation cells.

17.2.6 Concentrate Dewatering & Handling

The concentrate handling circuit consists of a high-rate thickener followed by a filter press to dewater the concentrate prior to loadout. The concentrate thickener overflow reports back to the flotation circuit while the underflow is pumped to a vertical plate and frame filter press. The filtrate is returned to the concentrate thickener to minimize losses. The dewatered concentrate is stockpiled and loaded onto covered trucks for delivery to the contracted smelters via local roads.

Flocculant is introduced to the concentrate thickener feed to aid in settling the materials.

Major equipment in this area includes the following:

- concentrate thickener
- vertical concentrate filter press.

17.2.7 Tailings Dewatering

The flotation tailings pumpbox feeds the tailings thickener. The overflow of the tailings is recycled to the process water system, and the underflow is pumped to the tailings filter feed tank, from where it is pumped through one of three horizontal plate-and-frame filter presses. The filters dewater the tailings to approximately 15% moisture before the filter cake is discharged onto cake feed conveyors and a stockpile feed conveyor. The filter cake is stockpiled and hauled to the tailings facility for long-term containment. The wash water used in the filter presses, as well as water removed from the filter cake, is recycled to the tailings thickener via a filtrate transfer tank.

Flocculant is introduced to the tailings thickener feed to aid in settling the materials.

Major equipment in this area includes the following:

- tailings thickener
- three horizontal plate and frame filter presses
- tailings cake and stockpile feed conveyors.

17.3 Reagent Handling and Storage

The reagent handling systems include unloading and storage facilities, mixing and storage tanks, and feed equipment as needed for each of the required plant reagents. Each set of compatible reagents is contained in a dedicated area to prevent environmental contamination and mixing of incompatible reagents. Storage tanks are equipped with level indicators, instrumentation, and alarms to ensure spills do not occur during normal operation, and appropriate ventilation, fire and safety protection, eyewash stations, and safety data sheet (SDS) stations are located throughout the facilities. Sumps and sump pumps are provided for spillage control.

The required reagents are listed in Table 17-2.

Table 17-2: Reagents

Reagent	Use	Handling
Hydrated lime	pH modifier	Received as powder; dissolved with process water; distributed to ball mill, rougher flotation, cleaner 1 flotation, and cleaner 2 flotation circuits.
Aero 3894 (Thionocarbonate)	Sulphides collector	Received as liquid; added directly into rougher flotation, cleaner 1 flotation, and cleaner 2 flotation circuits.
Methyl Isobutyl Carbinol (MIBC)	Frother	Received as liquid; distributed to rougher flotation, cleaner 1 flotation and cleaner 2 flotation circuits.
Flocculant	Settling aid	Received as powder; mixed with raw water, dissolved in process water and transferred to a storage tank; distributed to concentrate and tailings thickeners.

17.4 Energy, Water, and Process Materials Requirement

17.4.1 Process Materials

Reagent and consumable usage rates were estimated from expected material properties, industry benchmarks, and preliminary testwork data. Table 17-3 shows the expected average annual reagent and consumables usage rates.

Table 17-3: Annual Reagent and Consumables Requirements

Item	Unit	Usage Rate
Reagents		
Hydrated Lime	t/a	5,745
Aero 3894 (Thionocarbonate)	t/a	161
Methyl Isobutyl Carbinol (MIBC)	t/a	138
Flocculant	t/a	91
Operating Consumables		
SAG Mill Media	t/a	2,760
Ball Mill Media	t/a	2,806
Regrind Mill Media	t/a	110
Concentrate Filtration Cloths	unit (roll)/a	1
Tailings Filtration Cloths	unit/a	4,320

Item	Unit	Usage Rate
Maintenance Consumables		
SAG Mill Liners	set/a	1
Ball Mill Liners	set/a	1
Regrind Mill Liners	set/a	2
Jaw Crusher Liners	set/a	4
Cone Crusher Liners	set/a	8
Secondary Screen Media	set/a	6

17.4.2 Water Requirements

17.4.2.1 Raw Water

Raw water is supplied from lakes in the area to the fresh/firewater tank. Raw water is used for all purposes requiring clean water with low dissolved solids and low salt content. The estimated raw water usage requirement is 70 m³/h. Areas of the plant requiring raw water are as follows:

- flocculant dosing
- concentrate filter cloth washing
- reagent mixing
- fire and gland water.

17.4.2.1.1 Fire Water

Fire water for the process plant is stored as a dedicated volume within the raw water tank. A dedicated pump skid consisting of an electrical pump, jockey pump, and diesel pump will supply water from the fire water reserve volume to the distribution system

17.4.2.1.2 Gland Water

Gland seal water for the plant is sourced from the raw water tank and pumped to the various end-users across the plant site. The estimated gland water requirement is 24 m³/h.

17.4.2.2 Process Water

Process water for the plant consists of concentrate and tailings thickener overflows, while fresh water provides any additional make-up water requirements. Process water is stored in a tank before being distributed via pumps across the plant to various end-users.

The estimated process water usage is 1,665 m³/h, supplied from internal recycle streams with 11 m³/h of make-up water required. Areas requiring process water are as follows:

- grinding
- rougher flotation and regrinding
- cleaners 1 and 2 flotation
- concentrate filter flushing
- tailings filter flushing
- tailings filter cloth washing
- reagent mixing.

17.4.2.3 Potable Water

Potable water is produced by an on-site potable water plant which will treat water from the raw water tank. Treated potable water is stored in a dedicated storage tank before being distributed to the various end-users in the process plant.

17.4.3 Power Requirements

The total installed load for the process plant is estimated at 31.2 MW with an estimated annual power consumption of 165.5 MWh/a. Further discussion regarding the power supply and distribution system is available in Section 18.

An outline of power requirements for each section of the process plant is provided in Table 17-4.

Table 17-4: Power Requirements by Process Plant Area

Area	Installed Power (kW)	Nominal Demand (kW)
Crushing	1,632	496
Grinding	17,286	14,394
Flotation and Regrind	2,837	1,662
Reagents	30	17
Water & Air Services	1,177	395
Tailings Dewatering	6,364	3,139
HVAC, Lighting & Buildings	1,875	788
Total	31,201	20,891

18 PROJECT INFRASTRUCTURE

18.1 Introduction

The infrastructure required for this project includes:

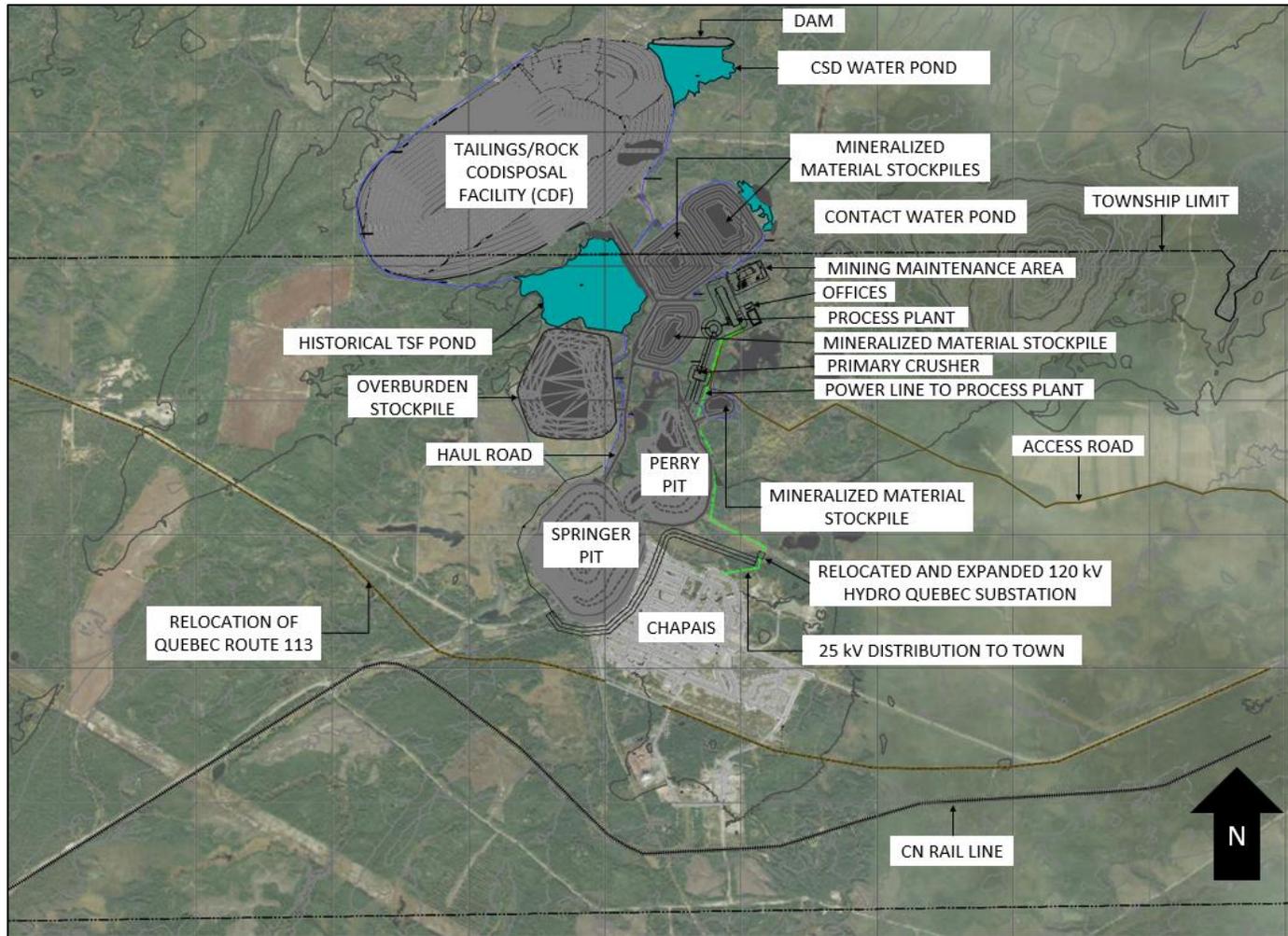
- mine maintenance facilities (truckshop, mine workshop and warehouse, mine dry, and miscellaneous facilities)
- process facilities (process plant, crusher facilities, crushed material stockpile, process plant warehouse and workshop)
- co-disposal storage facility (CDSF) to store filtered tailings and waste rock
- mineralized material stockpiles

The locations of the site facilities have been based on the following factors and observations:

- locate the waste rock storage facility near the mine pits to reduce haul distance
- locate the primary crushing and run-of-mine pad to reduce hauling from all pits over the life of mine
- locate the process plant in an area within the township of Chapais
- locate the CDSF near the process plant and outside of local flood plains
- locate administration, processing plant, and offices close to each other to limit walking distances (important during cold weather).

The overall site layout is shown in Figure 18-1.

Figure 18-1: Infrastructure Layout Plan



Source: Ausenco, 2025.

18.2 Site Access

The Opémiska property is located near the town of Chapais, approximately 44 km west from Chibougamau. Overland site access is available by travelling south from Chibougamau on Quebec Route 167 for 12 km, and then west on Quebec Route 113 for 32 km. The project site is connected to Quebec Route 113 via an access road that will be upgraded (Figure 18-1). These routes are well-maintained in all seasons.

A CN railway line passes to the south of the property and the town of Chapais, connecting east to Chibougamau and eventually connecting through to Quebec City and Montreal to the south. Chibougamau and Chapais have a regional airport with weekly scheduled flights to and from other Northern Quebec regional airports, served mainly by Air Creebec. This airport is a 20-minute drive from the project site at Chapais. Val-d'Or, which is a bigger regional airport in the area, is a four-hour drive from Chapais. It has regularly scheduled flights to and from Montreal and acts as a hub for other flights to the north. Val-d'Or is a six-hour drive from Montreal, and there is daily bus service between Montreal and the other cities in the Abitibi region.

The powerlines and telecommunication systems can be easily accessed with the powerline directly feeding the town of Chapais. There will be efforts to relocate and upgrade this powerline by Hydro-Québec to accommodate the Opémiska mines and project needs.

There are multiple site access methods for personnel and equipment outside the project region, and additional local supplies, labour, and service providers are available in the town of Chibougamau and other neighbouring towns and communities. Materials (such as reagents) will be transported to site via Quebec Route 113 or by rail. The mineralized material will be hauled from site in the same way.

Development plans for the pits will require a section of the existing Quebec Route 113 to be relocated, since it currently passes through the ultimate pit footprint. The highway is planned to be reconstructed as per regulations from the Ministère des Transports du Québec.

18.3 Built Infrastructure

18.3.1 Plant Earthworks

The typical method of clearing, topsoil removal, and excavation will be employed, incorporating drains, safety bunds, and backfilling with granular material and aggregates for road structure. The existing road that connects the site to Quebec Route 113 will be upgraded and extended. It is expected that forest clearing and topsoil removal will be required to allow the process plant and other buildings and facilities to be constructed.

Site civil work includes designs for the following infrastructure:

- light vehicle and heavy equipment roads
- access roads
- overburden stockpile area

- mineralized material stockpiles
- mine facility platforms and process facility platforms
- water management ponds and contact and non-contact water channels and dams
- CDSF
- waste rock storage facilities.

As mentioned in Section 18.2, the southern portions of the mining pits will encroach into the town of Chapais and Quebec Route 113. As such, a portion of the highway will be relocated and reconstructed.

18.3.2 Buildings

Project buildings are summarized in Table 18-1.

Table 18-1: Buildings

Description	Location	Building Construction	Length (m)	Width (m)	Height (m)	Area (m ²)
Primary Crusher Building	Plant	Stick-Build	28.0	8.0	16.0	224
Secondary Crusher Building	Plant	Stick-Build	18.0	10.0	25.0	180
Stockpile Cover Building	Plant	Fabric	66.0	66.0	25.0	4,356
Process Plant Building	Plant	Pre-Engineered	200	33.0	22.0	10,247
Truck Shop & Wash	Plant	Pre-Engineered	80.0	18.0	15.0	1,440
Truck Shop Warehouse	Plant	Fabric	21.0	18.0	6.0	378
Truck Shop Office	Plant	Modular	7.4	12.2	3.0	90
Primary Crusher Control Room	Plant	Modular	6.0	2.4	2.7	14
Secondary Crusher Control Room	Plant	Modular	6.0	2.4	2.7	14
Process Plant Control Room	Plant	Modular	6.0	2.4	2.7	14
Plant Warehouse and Maintenance Building	Plant	Fabric	40.0	18.0	6.0	720
Plant Office	Plant	Modular	14.8	12.2	3.0	181
Security Gatehouse	Plant	Modular	6.0	2.4	2.7	14
Mine Office and Change Rooms	Plant	Modular	33.3	12.2	3.0	406

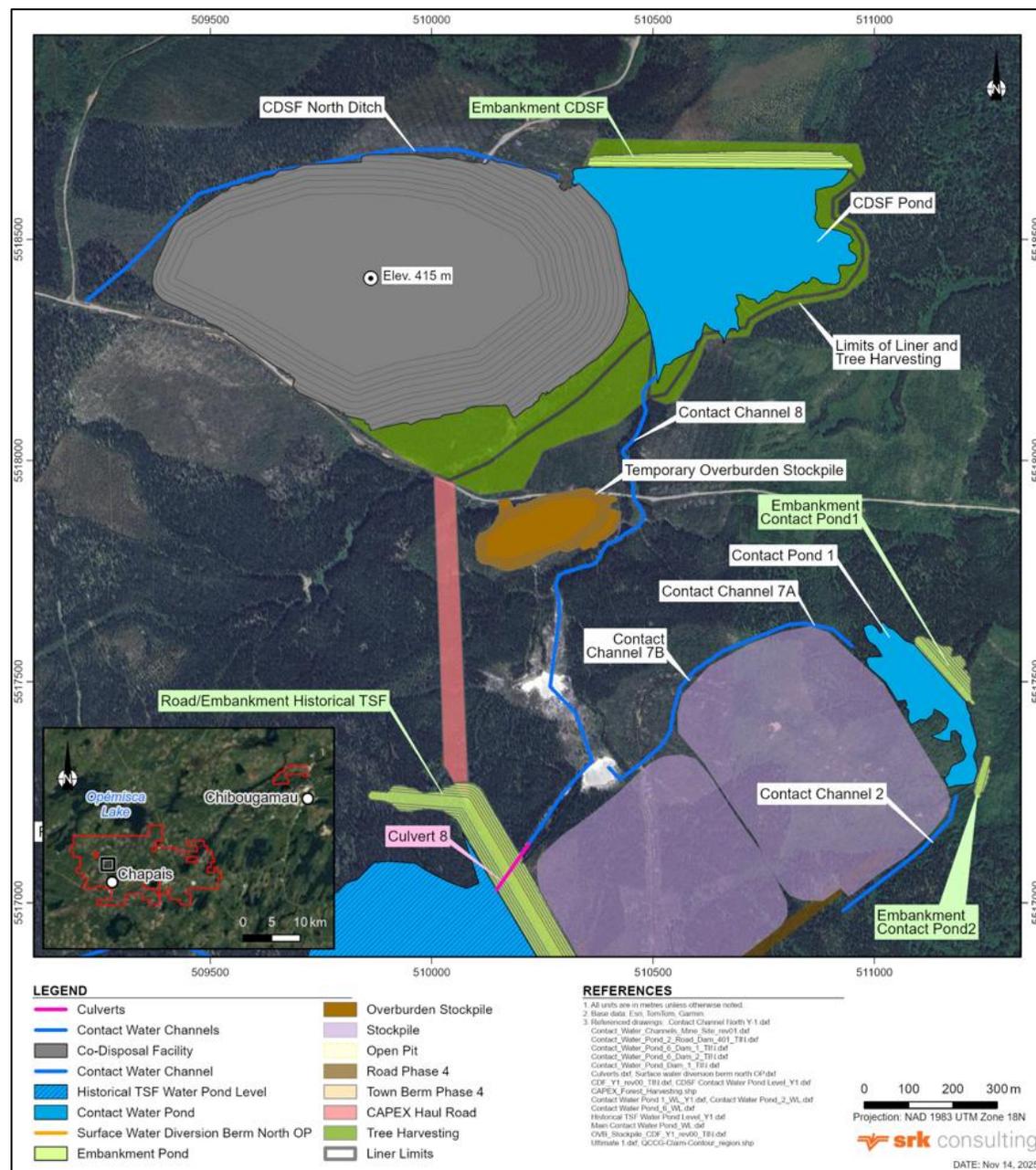
18.3.3 Accommodation

The project is located next to the town of Chapais and near other northern Quebec towns, like Chibougamau. It is planned to employ workers from local communities as much as possible. No separate accommodation facilities are anticipated at this time.

18.4 Stockpiles

The organic material beneath the co-disposal storage facility (CDSF) footprint will be excavated and hauled, and the surface will be compacted within a temporary overburden stockpile south of the CDSF Phase 1 (Figure 18-2).

Figure 18-2: Phase 1 of the CDSF and Temporary Overburden Stockpile



Source: SRK, 2025.

The configuration of the temporary overburden stockpile will comply with the applicable regulations and current practice recommendations. The temporary overburden stockpile will reach a maximum elevation of 395 m (± 15 m in height) according to the current engineering stage.

The stockpile will be built with an external slope of 3H:1V and its current configuration allows stockpiling of 190,782 m³ ($\pm 300,000$ tons, assuming a placed dry density of 1.6 t/m³).

A comprehensive QA/QC program will be implemented during construction. Material from this temporary overburden stockpile will be excavated and placed on the slope of the CDSF as part of progressive reclamation activities. Water management related to the temporary overburden stockpile and CDSF is discussed in Section 18.6.

The overburden and organic material thickness beneath the CDSF is unknown and assumed to be variable. The organic material deposit thickness was considered to vary from 0 cm to 50 cm, depending on the sector. An average thickness of 30 cm across the CDSF footprint was considered for this study.

18.5 Tailings and Waste Rock Co-Disposal Storage Facility (CDSF)

The filtration process is expected to dewater the tailings to a gravimetric water content of 15% to 18%. Filtered tailings will be temporary stockpiled, and protected against precipitation and wind in a dome at the industrial site until transported to the CDSF. This targeted moisture content from the filter press allows for proper transportation of tailings with 140-tonne haul trucks, and placement and compaction using CAT D9 dozers. The water recovered from the CDSF will be directed to the CDSF pond and pumped back to the mill, as needed.

The CDSF will be constructed on a lined platform, and designed to (1) promote basal drainage toward peripheral ditches and the CDSF contact water pond; and (2) avoid creating a preferential slip surface at the contact with the geomembrane. The CDSF will be located north of the pit and of the historical TSF (Figure 18-1).

The projected CDSF configuration has been developed to comply with applicable regulations and recommended practice. The CDSF is currently configured with a maximum elevation of 525 m (± 154 m above ground and ± 50 m less than Mont Springer). The CDSF will be constructed with 10 m high benches with a minimum 10 m setback in between. Each bench will be built with an external slope of 3H:1V, leading to an average slope of approximately 4.3H:1V.

The co-disposal of filtered tailings and waste rock within the same facility will minimize the facility footprint, and strategic placement of waste rock will increase the facility's overall stability. Dilative behaviour of tailings will be promoted through compaction using CAT D9 (or equivalent) and by enabling the haul trucks to circulate over the full facility surface. A blanket drain at the base of the stockpile has been included to evacuate any potential water that accumulates at the base of the facility and to promote a low phreatic level. Comprehensive QA and QC programs will be implemented during construction of the CDSF, including installing instrumentation that is connected to a data logger for continuous recording, and installing monitoring wells. A comprehensive surveillance program will be implemented to monitor the performance of the facility over time.

Engineering of the CDSF will comply with the following applicable regulation and guidelines:

- Directive 019 sur l'industrie minière, February 2025 (D019). Ministère du Développement durable, de l'Environnement et des Parcs (2025).
- Guide de préparation du plan de réaménagement et de restauration des sites miniers au Québec. Ministère de l'Énergie et des Ressources naturelles (MERN, 2024) (Le Guide).

- Mining Association of Canada (MAC), 2017. A Guide to the Management of Tailings Facilities.

A stability assessment was not performed during this project phase. The geotechnical stability of the overall facility is expected to comply with the current recommendations and applicable regulation listed above. Also, considering that 80% of the CDSF will be waste rock, the comprehensive deposition plan, yet to be developed, will enable strategically placing tailings away from critical slip surfaces as a measure to improve geotechnical stability. Because both streams of mine waste will be placed and compacted simultaneously, this will allow the operation to work in multiple areas, improving worker safety and providing flexibility with placement activities. The compaction process (i.e., lift thickness and number of passes) is yet to be determined, as it will be dependent on test-pad scale-tests to ensure that the targeted dry densities are achieved.

Off-specification material (i.e., tailings with a moisture content that is too high) will be placed at locations strategically identified during every stage of construction. Dewatering the tailings during commissioning is typically variable and higher-than-expected moisture contents have been assumed for that period. Compacting wet tailings will be more challenging. The wet tailings will be placed within the inner part of the CDSF, outside the structural zones of the facility.

Snow will be managed during winter, and proactive surface water management will be carried out to keep working areas dry. Road graders and water/sand trucks will be used to maintain the roads at the CDSF.

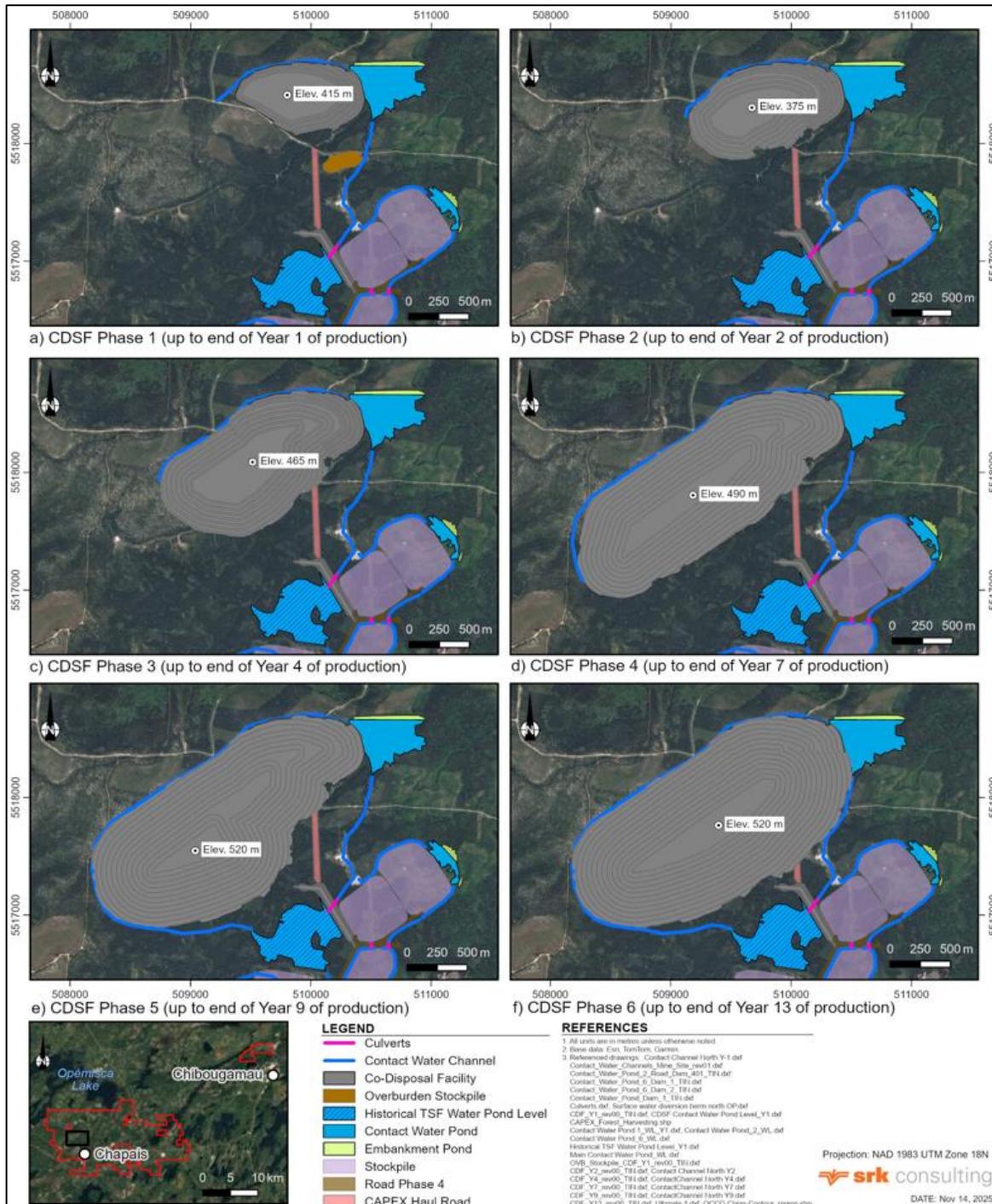
The CDSF will be built over 13 years (including two years for construction and pre-mining phases) according to the current mine plan. Construction and operation of the CDSF have been developed in six phases that minimize its footprint and costs. The CDSF construction sequence will be from east to west until Year 7. The final two phases will be built from west to east (Figure 18-3). Slurry tailings in-pit deposition will begin in Year 14 according to the current mine plan and tailings management strategy, as a measure to reduce the CDSF footprint. The six phases are summarized below.

- Phase 1 (lined platform to be built by Year 0; can store up to one year of operation)
- Phase 2 (lined platform construction completed by Year 1, can store up to one year of operation)
- Phase 3 (lined platform construction completed by Year 2, can store up to two years of operation)
- Phase 4 (lined platform construction completed by Year 4, can store up to three years of operation)
- Phase 5 (lined platform construction completed by Year 7, can store up to two years of operation)
- Phase 6 (lined platform construction completed by Year 9; can store up to four years of operation)

Surface water management phases for the perimeter of the CDSF will be developed concurrently and in accordance with the schedule shown in Section 18.6.

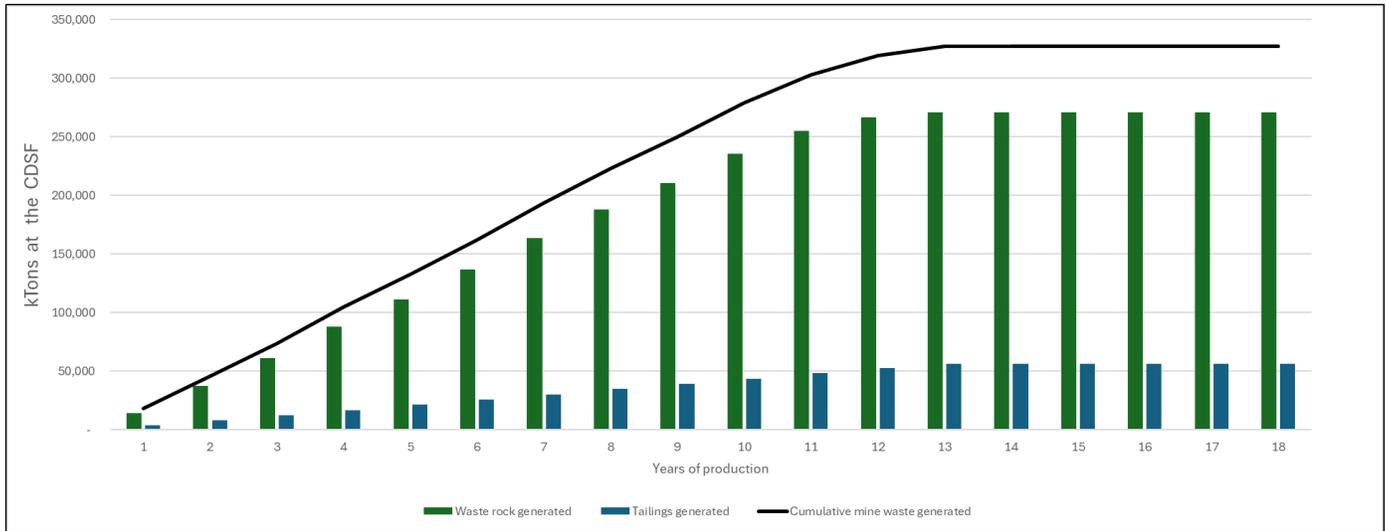
CDSF construction activities will follow the mine operations schedule. The CDSF final configuration, presented in Figure 18-3 shows the evolution of the CDSF through time. The final CDSF allows for stockpiling of approximately 162 Mm³ or 326 Mt of material. Figures 18-4 and 18-5 show the anticipated production of mine waste to be managed at the CDSF per the life of mine plan as well as the CDSF available storage capacity in cubic meters versus time for the same period, assuming in-situ placed and compacted dry densities of 1.7 t/m³ for tailings and 2.1 t/m³ for waste rock. Figure 18-6 shows a typical cross-section of the CDSF during operation/construction of the CDSF.

Figure 18-3: CDSF Final and Phases Configuration



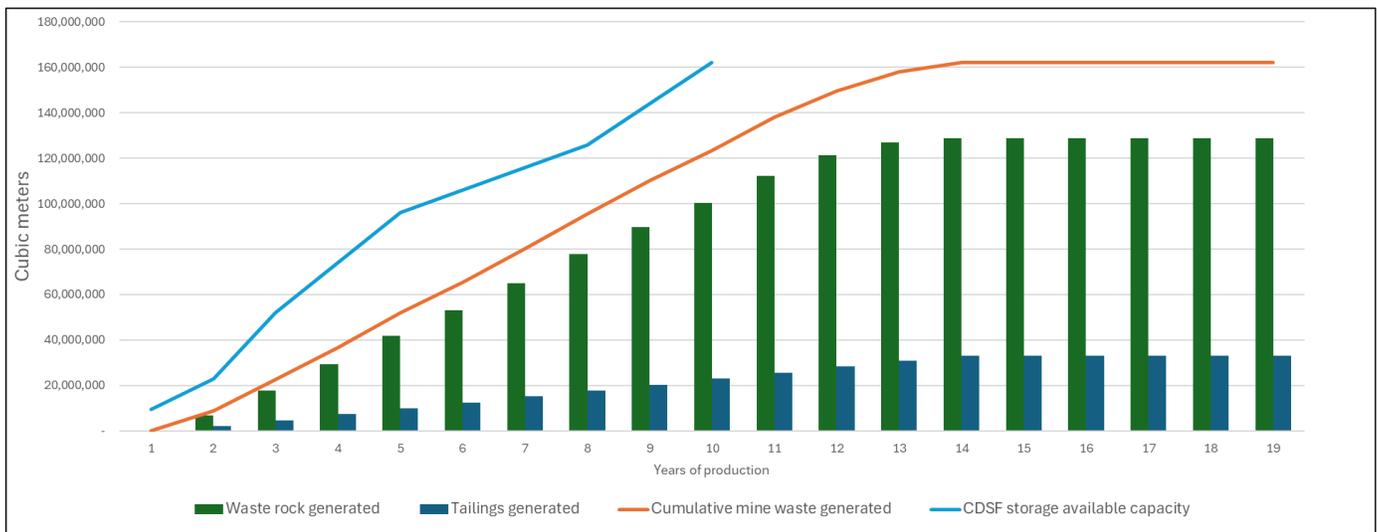
Source: SRK, 2025.

Figure 18-4: CDSF Storage Capacity vs. Life of Mine – Tonnage to be Managed at the CDSF



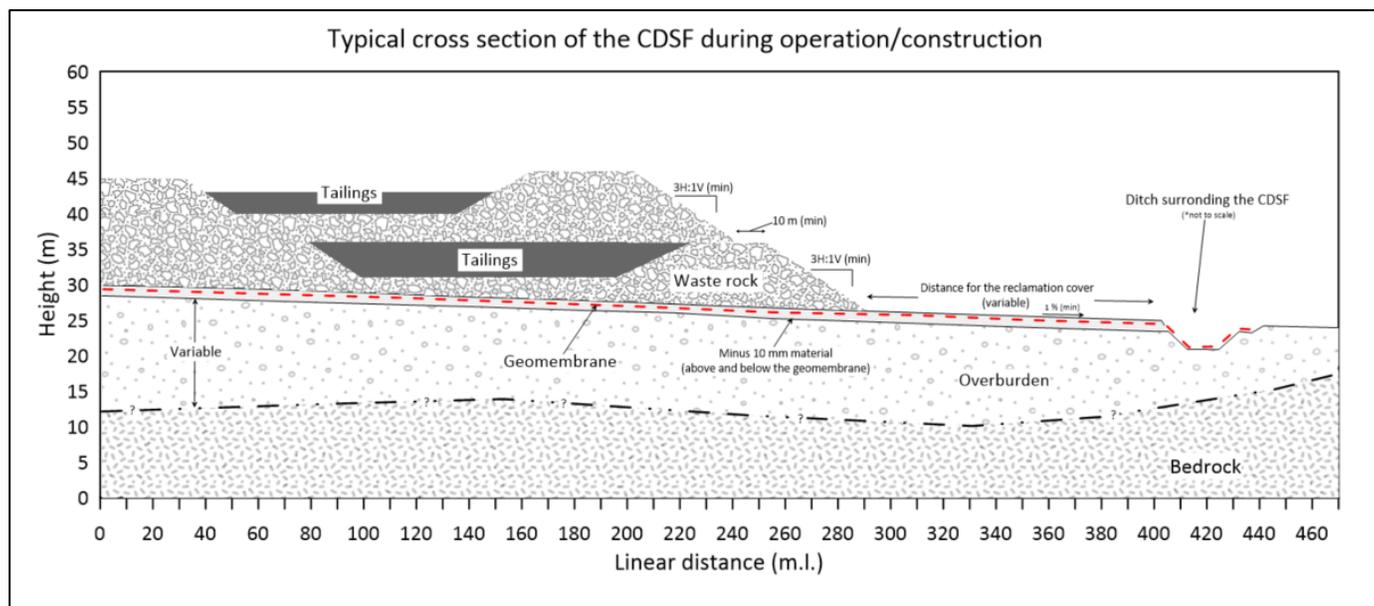
Source: SRK, 2025.

Figure 18-5: CDSF Storage Capacity vs. Life of Mine – Tonnage to be Managed at the CDSF



Source: SRK, 2025.

Figure 18-6: Typical Cross-Section of the CDSF



18.6 Surface Water Management

18.6.1 Design Criteria

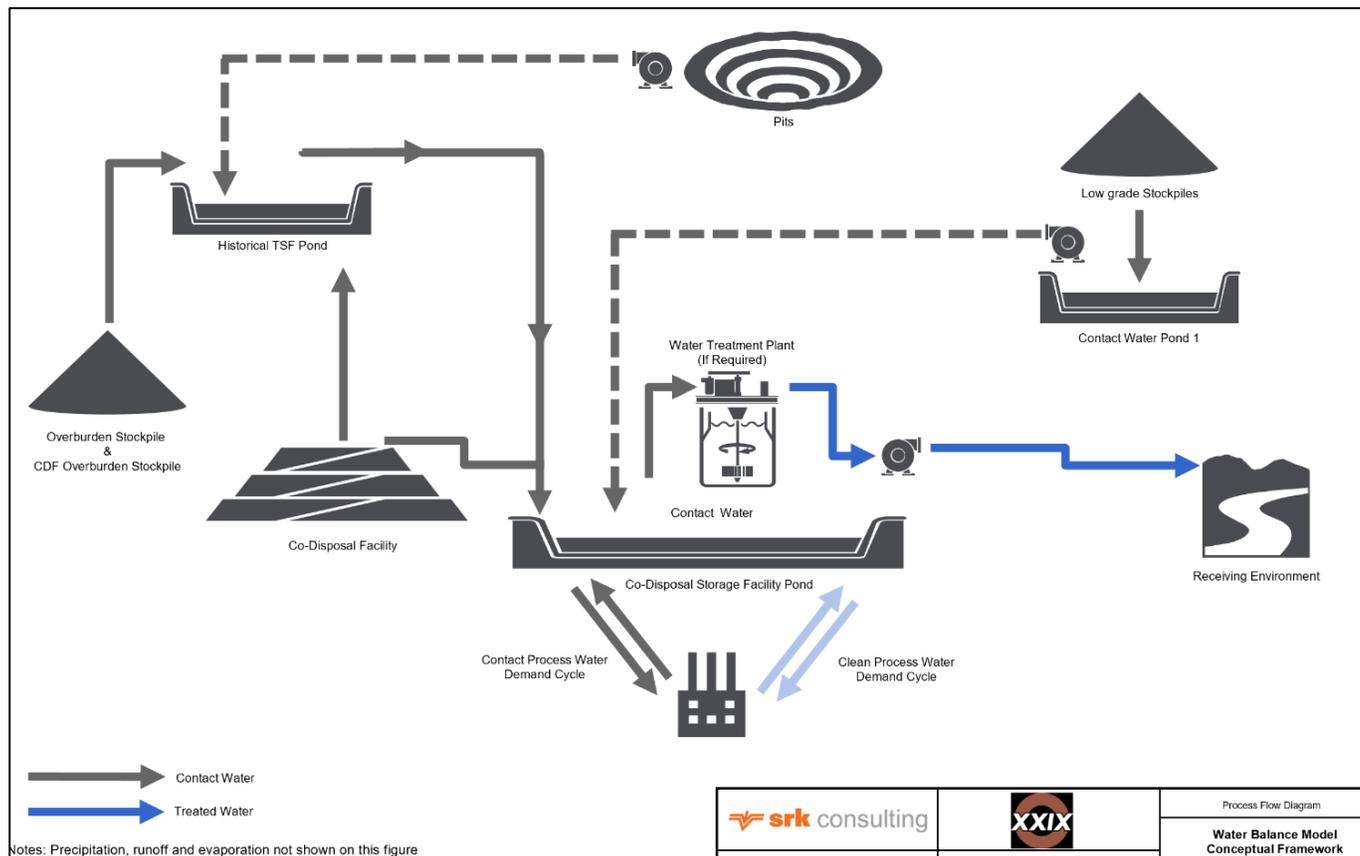
The design criteria for contact surface water management are based on the Mining Industry Directive 019, published by the Ministère de l'Environnement, de la Lutte contre les changements climatiques, de la Faune et des Parcs (MELCCFP) in February 2025. Surface water collection ponds (CDSF pond, historical TSF pond, and contact pond 1), and pumping stations (Section 18.6.5), are designed to manage the spring runoff, which is a combination of a 100-year snow accumulation melting over a 30-day period and a 1,000-year, 24-hour rainfall event. It is noted that additional geochemical characterization of waste rock, mineralized rock, tailings, and overburden will be conducted in the next phase of engineering, which may inform further refinements of the water management system. A schematic process flow diagram (PFD) is presented in Figure 18-7.

The collection ditches surrounding the CDSF are designed to handle the same storm event as the contact water ponds: the combination of a 1-in-1000-year rainfall event and snowmelt over 30 days with a 1-in-100-year recurrence interval.

The design criterion for calculating runoff from the industrial area and the stockpiles of topsoil, overburden, and mineralized material, on the other hand, is the combination of a 1-in-100-year rainfall event and snowmelt over 30 days with a 1-in-100-year recurrence interval.

The Chapais airport climate station, located 7 km northwest of the Opémiska project site, was used to obtain environmental weather and hydrological data.

Figure 18-7: Schematic Process Flow Diagram



Source: SRK, 2025.

18.6.2 Ditches and Runoff Diversion Berm

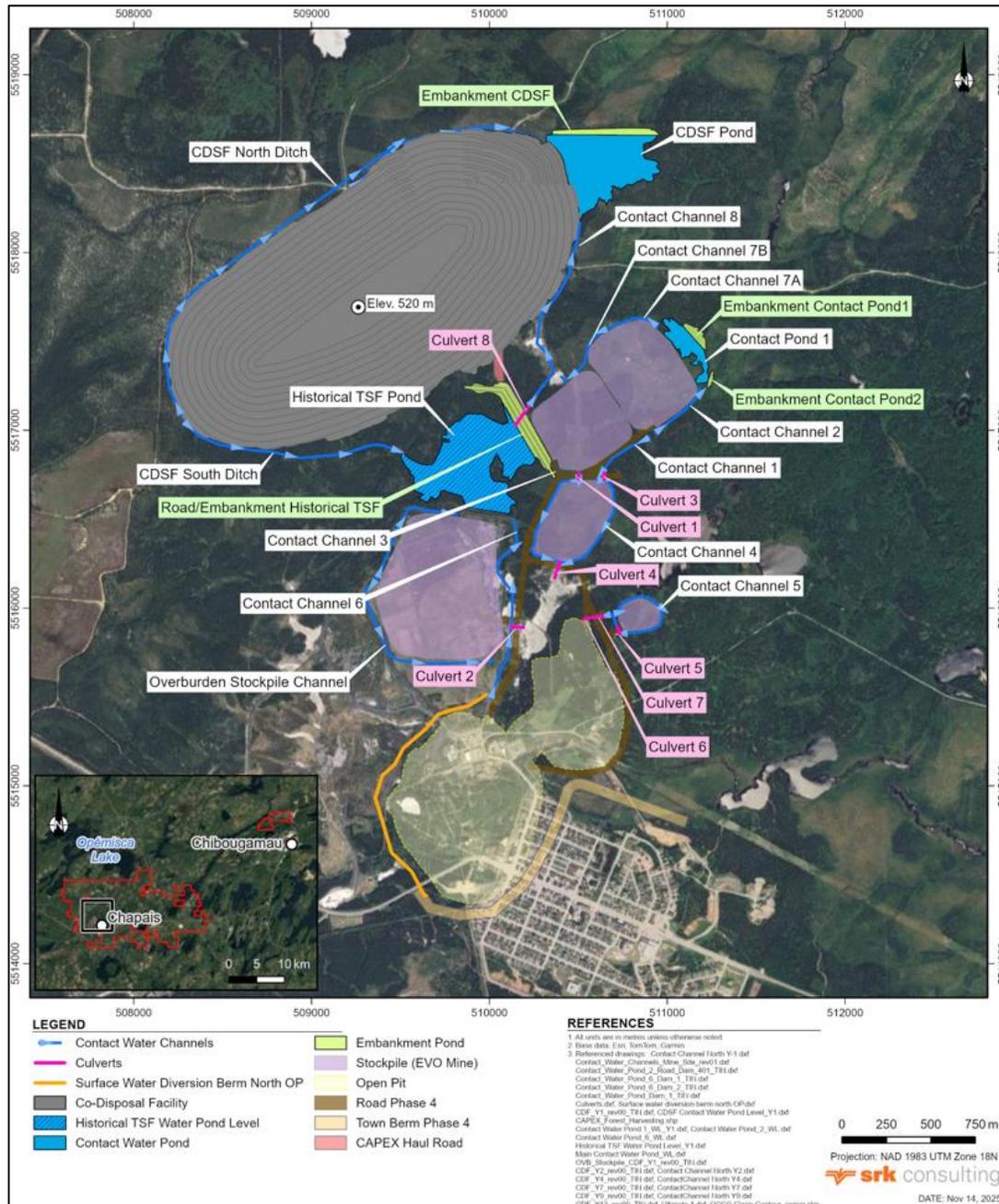
Contact water collection ditches surrounding the CDSF will be lined using 2 mm high-density polyethylene (HDPE) geomembrane. The geomembrane will be exposed, so no protection riprap is included within the cost estimation.

The CDSF deposition plan with one collection ditch, north of the CDSF that will be required until Year 7, with an additional ditch south of the CDSF will be required. Prior to Year 7, a ditch surrounding the CDSF deposition will collect runoff water from the north slope of the CDSF and drain it to the collection pond to the northeast (CDSF pond). After Year 7, an additional ditch south of the CDSF will be required and will extend progressively toward the historical TSF pond as construction progresses. The natural topography on the south side of the CDSF enables the contact water to flow by gravity towards the historical TSF pond and the CDSF pond.

In addition to the two CDSF ditches, ten collection ditches will intercept and direct runoff from the industrial site and various stockpiles to the collection ponds. These collection ditches have been placed to optimize the existing

topography and current surface water runoff management scheme of the historical TSF. Figure 18-8 and Table 18-2 present the proposed 12 collection ditches. The ditches will be armoured with a properly designed riprap.

Figure 18-8: Proposed Collection Ditches



Source: SRK, 2025.

Table 18-2: Collection Ditches/Channels

Collection Ditch Name	Length (m)
Ditches Relating to CDSF	
CDSF North Ditch	3,600
Contact Channel 8	1,200
Ditches Relating to Historical TSF Pond	
CDSF South Ditch	550
OVB_Stockpile_Channel	1,500
Contact Channel 6	1,120
Channels Relating to Historical Contact Pond 1	
Contact Channel 1	385
Contact Channel 2	375
Contact Channel 3	825
Contact Channel 4	695
Contact Channel 5	370
Contact Channel 7A	365
Contact Channel 7B	470

18.6.3 Culverts

The following criteria were retained for the design of the culverts surrounding the industrial zone, the CDSF, and the various stockpiles:

- The culverts are designed to be consistent with the design criteria of the ditch in which they will be installed.
- The minimum slope is 1% to avoid sedimentation and to allow self-cleaning.

The proposed culvert lengths and locations are presented in Table 18-3.

Table 18-3: Culvert Summary

Culvert ID	Length (m)
Culvert_1	36
Culvert_2	70
Culvert_3	41
Culvert_4	95
Culvert_5	68
Culvert_6	31
Culvert_7	60
Culvert_8	126

18.6.4 Collection Ponds and Embankments

The criteria below were used to design the new collection ponds. In total, there are three proposed collection ponds, one at the east extremity of the CDSF (CDSF pond), one to the southeast of the CDSF (historical TSF pond), and one to the northeast of the medium-grade stockpile (contact pond 1). These ponds will be lined with a 2 mm HDPE geomembrane. The geomembrane will be anchored at the bottom, walls, and the crest of each basin. The ponds will be built above natural ground level, using topographic high and embankments. Figure 18-8 (above) highlights the location of the collection ponds.

The CDSF pond receives water from the historical TSF pond via a controlled sluice gate to be installed upstream of culvert 8. Additionally, the CDSF pond receives water from contact pond 1 via a sump pump. Any water captured within the mine pits will be pumped into contact channel 6 in a controlled manner.

Embankments surrounding these ponds will have a minimum slope of 3H:1V. The anticipated embankment details are presented in Table 18-4.

Table 18-4: Embankment Summary

Pond Associated with Embankment	Embankment Name	Crest Width (m)	Embankment Length (m)	Embankment Crest Elevation (m)
CDSF	Embankment_CDSF	10	1 223	372
Historical TSF	Embankment_HistoricalTSF	20	778	410
Contact Pond 1	Embankment_CPond1	10	190	383
Contact Pond 1	Embankment_CPond2	10	89	383

The site water management plan (SWMP) addresses the surface runoff to be collected from the industrial areas, including the open pit dewatering, the main overburden stockpiles, mineralized material stockpiles, and the CDSF. Surface runoff and process water will be collected through a series of collection ditches that will discharge into the historical TSF pond and CDSF collection pond. As per the current understanding, settling of total suspended solids (TSS) will occur in those ponds and the collected water will be released toward the final effluent through a pumping station.

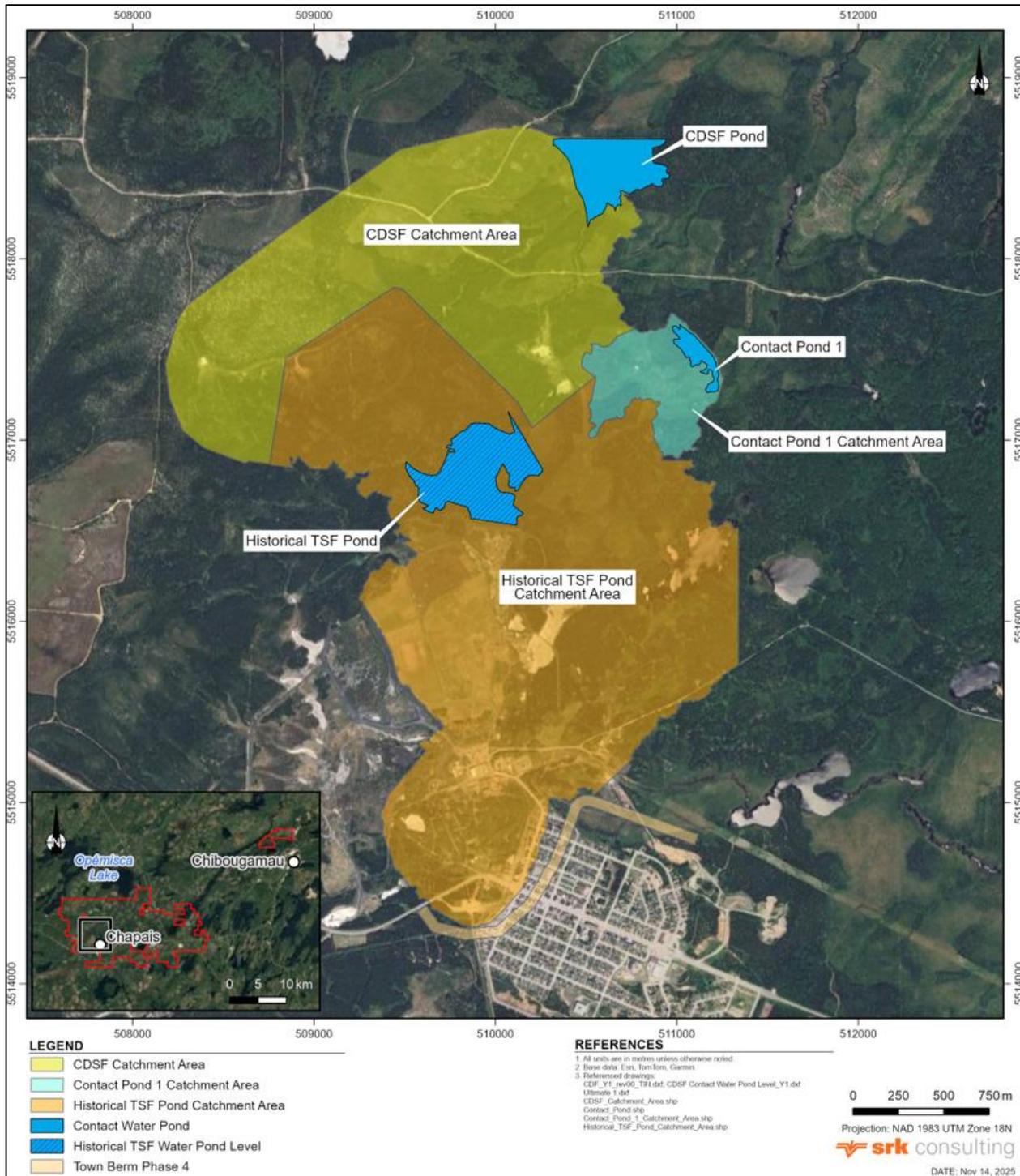
XXIX will prioritize reusing and recycling contact water in the process water make-up to reduce freshwater intake. The suspended solids collected in the pond will be managed on site at the CDSF collection pond.

Three main watersheds have been defined at this stage, as summarized in Table 18-5 and shown in Figure 18-9.

Table 18-5: Main Watershed Areas

Watershed ID	Area (m ²)
CDSF Pond Catchment	7,800,000
Historical TSF Pond Catchment	5,140,000
Contact Pond 1	400,000

Figure 18-9: Main Watersheds for the Project Site



Source: SRK, 2025.

18.6.5 Pumping Stations and Pipelines

Two pump systems on barge will be required for the water management operations. The first one will be used to pump the water from collection pond 1 to the CDSF pond, and the second one from the CDSF pond toward the final effluent.

18.6.6 Water Treatment

Currently, there is no indication that contact water at the site will require treatment beyond the sedimentation of suspended solids within the collection pond. Water management infrastructure has been designed to promote effective settling of particulate matter prior to any discharge. Should geochemical investigations in subsequent engineering phases identify a need for further water treatment—such as the removal of dissolved metals or other contaminants—a dedicated water treatment plant will be designed and constructed adjacent to the collection pond. The final design and operation of this facility will be determined based on the results of ongoing geochemical characterization and regulatory requirements.

18.6.7 Rainfall and Snowmelt

Rainfall and snowmelt within the pit will be collected in pit sumps and pumped to surface, to the contact water ditches system and will flow toward the CDSF pond.

18.6.8 Groundwater

Groundwater is an important consideration in open pit mining operations. Not only must the water be collected and pumped out of the pit, but it also significantly impacts the stability of the pit slopes. Water pressures act in direct opposition to stabilizing forces and must be accounted for to ensure realistic stability modelling results. A pit dewatering system must be considered in the mine design and mine planning, as groundwater drawdown is vital for a safe and efficient mining operation. Vertical piezometers will be installed behind the pit walls to evaluate the efficiency and success of the drainage system on groundwater drawdown. The ground water will be pump within the contact water ditches and flow directly to the CDSF pond.

18.7 Power and Electrical

The estimated total connected load at the Opémiska site (including process plant, mining, and site buildings) is 38 MW, with the operating load varying depending on and the blend of plant feed and the tailings deposition strategy.

18.7.1 Facility Power Supply

Primary power to the Opémiska site will be provided by Hydro-Québec via the existing 120 kV distribution network. An existing 120 kV powerline runs near the project site.

The 120 kV substation that currently supplies the town of Chapais (and also provides power to the HQ grid from Chapais Energy) will be reconstructed to a new location, north of the town and outside of the ultimate pit footprint. The new substation will have a larger capacity to support the power demand of the town and project. It will be branched off

perpendicularly from substation tie-in point in two directions: north to supply the process plant, and south to supply the town. Both directions will have new 25 kV overhead lines.

The voltage will be stepped down from 120 kV to 25 kV at the relocated substation which will have utility metering. This substation will have 100% redundancy in transformer capacity. Two 50/66 MVA oil-filled with forced-air cooled substation transformers are proposed to carry the maximum power required by the site and town. This incorporates future growth and redundancy in the event a single transformer is temporarily out of service.

Power factor correction equipment will be installed to improve the power factor to 0.9 or better at the point of interconnection with the utility.

18.7.2 Site Power Reticulation

Site-wide power distribution to the process plant and maintenance facilities will be via the 25 kV overhead powerline using wood pole structures.

18.8 Impact on town of Chapais

The project will require a portion of Route 113 to be redirected as part of initial construction. A town berm to the northwest of Chapais will also need to be constructed.

The town berm was designed with a height of 15 m to limit the visual and environmental impacts of the mining operation and will be vegetated with local flora. Mining of the ultimate Springer pit phase during Year 7 of project life is expected to encroach on the town of Chapais, which will require approximately 120 homes and 20 commercial buildings to be relocated, as well as the town berm.

19 MARKET STUDIES AND CONTRACTS

19.1 Market Studies

The project is expected to produce a copper concentrate. Neither XXIX nor its consultants have undertaken a formal market study regarding the sale of this concentrate. Accordingly, the marketing assumptions in this study are based on prevailing market conditions, discussions with XXIX, and terms reported in comparable recent studies and projects.

The QP considers the marketing and pricing assumptions reasonable for use in PEA-level cash flow analyses. For the purposes of this study, it is assumed that the concentrate will be transported from the mine location by road to a smelter located in the Abitibi Region of Québec. Considering that the project is a previously producing operation and that a copper concentrate will be produced, the product is expected to be marketable.

19.2 Concentrate Marketing Assumptions

Table 19-1 shows the market assumptions applicable to the project's copper concentrate used for PEA-level cashflow analysis. Concentrate losses of 0.2% were applied and expected due to transportation. Deductions and payability assumptions were applied based on option yielding the lowest return as in customary smelter contracts.

Table 19-1: Copper Concentrate Assumptions

Copper Concentrate	Units	Value
Copper Concentrate Grade	% Cu	20.0
Copper Concentrate Deduction	% Cu	1.0
Gold Concentrate Deduction	g/t Au	1.0
Silver Concentrate Deduction	g/t Ag	-
Copper Payable	%	96.5
Gold Payable	%	96.0
Silver Payable	%	90.0
Copper Concentrate Moisture	%	8.0
Treatment Charges	US\$/dt	40.00
Copper Concentrate Transportation	US\$/wt	115.00
Copper Refining	US\$/lb	0.04
Gold Refining	US\$/oz	5.00
Silver Refining	US\$/oz	0.50

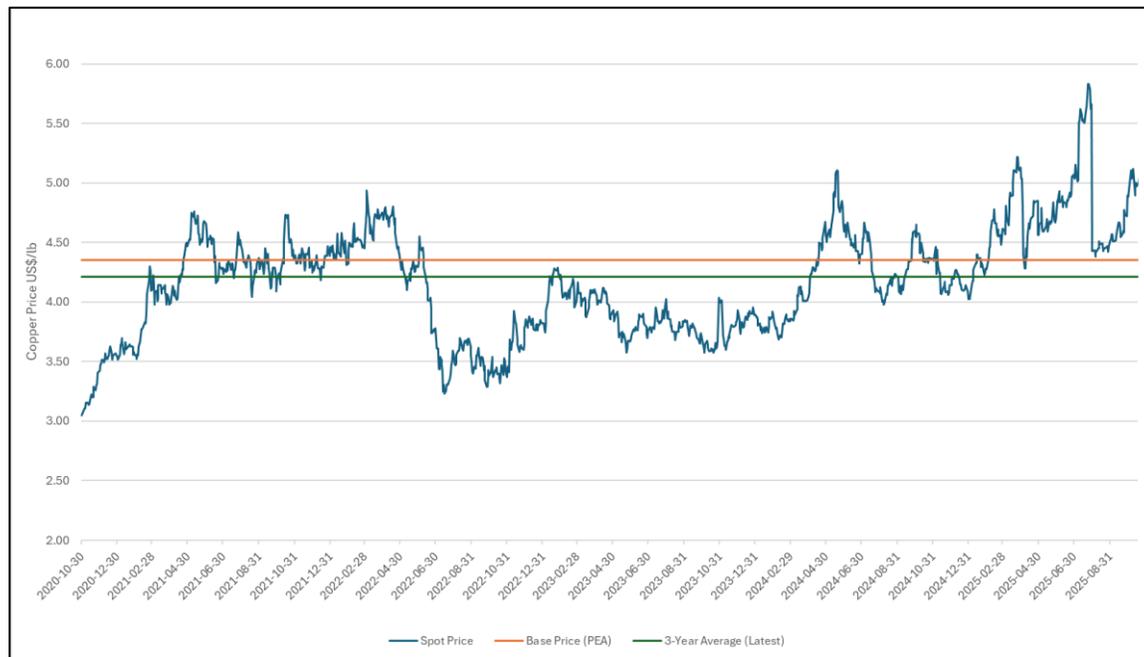
19.3 Commodity Price Projections

The metal prices selected for the economic evaluation in this technical report are presented in Table 19-2. A constant long-term price of US\$ 4.35/lb for copper, US\$ 3,000.00/oz for gold, and US\$ 30.00/oz for silver has been assumed. Figures 19-1, 19-2 and 19-3 illustrate the historical price performance, the three-year average, and the PEA base metal price assumptions, respectively. There is no guarantee that the prices of copper, gold, and silver used in this study will be realized at the time of production.

Table 19-2: Metal Price Assumptions

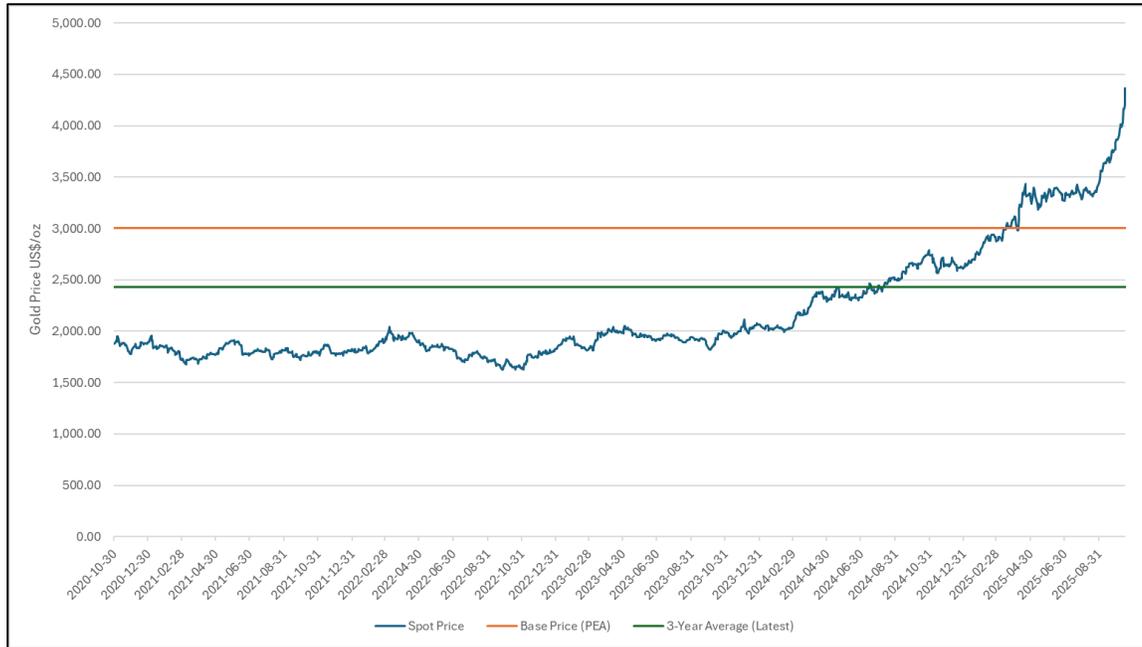
Metal	Units	Value
Copper	US\$/lb	4.35
Gold	US\$/oz	3,000.00
Silver	US\$/oz	30.00

Figure 19-1: Copper Historical Price (2020-2025)



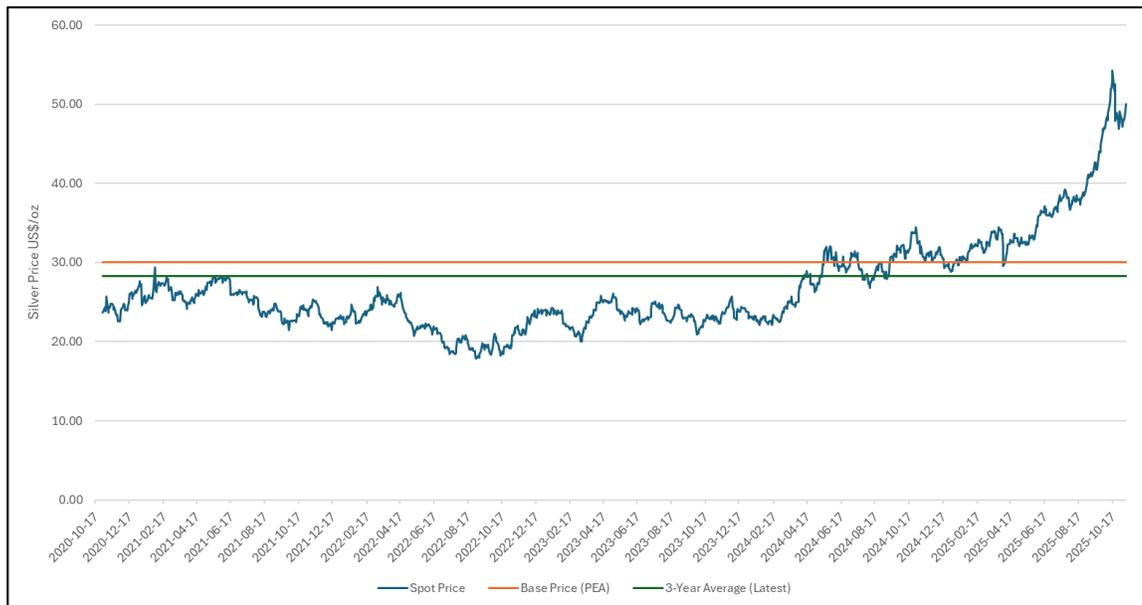
Source: Evomine, 2025.

Figure 19-2: Gold Historical Price (2020-2025)



Source: Evomine, 2025.

Figure 19-3: Silver Historical Price (2020-2025)



Source: Evomine, 2025.

19.4 Contracts

At present, there are no mining, concentrating, smelting, refining, transportation, handling, marketing, or hedging arrangements, nor any forward sales contracts associated with the project. Such an absence is typical for projects at this level of development, which are a few years from potential commercial production.

20 ENVIRONMENTAL STUDIES, PERMITTING, AND SOCIAL OR COMMUNITY IMPACT

20.1 Environmental Studies

The environmental permitting process requires an understanding of the physical, biological and social environments by completing numerous environmental studies. The first step is to define the studies area, based on the different components, such as:

- Physical environment includes, but is not limited to, topography, geology and geochemistry of rocks, geomorphology and soil quality, hydrology and surface water quality, sediments quality, hydrogeology and groundwater quality, air quality, noise and vibrations, greenhouse gases and climate.
- Biological environment includes, but is not limited to, vegetation and wetlands, fauna (fish, mammals, avifauna, herpetofauna) and their habitats, species at risk, and protected areas.
- Social environment includes, but is not limited to, socio-economic profile of the area, health, administrative framework, first nations, land uses, archaeology and patrimony, and landscape.

Early in the permitting process, it is recommended to start baseline studies to identify any fatal flaw or environmental issues (e.g., protected areas, species at risk, waterbodies, fish habitats). Identification of the sensitive elements on the receiving environment must also be carried out by these studies (e.g., sources of drinking water, major waterbodies, wetlands, archaeology sites). The results of environmental studies will have to be considered to identify project alternatives.

Once the baseline has been defined by the latter studies, an impact assessment study must be carried out to identify and analyse both the positive and negative impacts of the mining project. Mitigation measures are then proposed to attenuate the negative effects on the environment.

Regarding the project, the presence of protected areas in the vicinity of the project was investigated. Biological refuges are present on the shores of the Opémiska Lake, and in the middle of the Presqu'île Lake. They are approximately 10 km away from the project and outside the mine layout area. The Mount Springer, even if it was not classified as a protected area, was excluded from the alternatives for the mine infrastructures because of its recreational uses by the population. Finally, a preliminary review was conducted with the Centre de données sur le patrimoine naturel du Québec (CDPNQ) for species at risk. Some occurrences have been identified within a 10 km radius of the project, such as bank swallow, northern myotis and little brown myotis (CDPNQ, 2025). The territory of woodland caribou also expands to the project area. Baseline studies and inventories will be able to confirm the presence, or not, of these species in the study area. Based on the closure plan of the former Opémiska mine (Minnova, 1991), the most common wildlife species in the area are the moose, black bear, beaver, fox, marten, hare, grouse, duck and willow ptarmigan. Fish species of interest are the northern pike, the walleye and the brook trout.

The next step for the Opémiska Project is the execution of the baseline studies to be able to conduct an impact assessment study afterward.

20.2 Waste Rock, Tailings, Overburden, Mineralized Material and Water Management

The layout of the project is shown in Figure 18-1. Mine operations involve four phases, with two open pits, named Springer and Perry pits. One overburden pile and four mineralized material stockpiles are located north of the pits. The filtered tailings and waste rock CDSF is located approximately 3 km north of the town of Chapais. Water ponds are located south and east of the CDSF, and a small water pond is located east of the mineralized material stockpile. Site selection criteria are as follows:

- avoid areas highlighted as a potential mineralized zone
- reduce the distance to the pits and to the plant
- build the plant in the limits of the municipality of Chapais
- use the natural slope for drainage
- avoid major watercourses and wetlands
- avoid Mount Springer and protected areas.

At this phase of the project, several assumptions were made about the geotechnical and geochemical aspects of the waste rock, mineralized material, tailings and water management. Full studies will be completed in a subsequent phase.

20.2.1 Geochemical Characterization of Waste Rock, Tailings and Mineralized Material

A preliminary geochemical characterization was conducted on 29 samples of waste rock, four samples of mineralized material, and two samples of tailings to evaluate potential metal leaching and acid rock drainage (ML-ARD). The samples consisted of core from recent drillholes and from tailings produced by the metallurgical testing (Turgeon, 2023). The mineralization (copper, gold, silver) is associated with sulphides (chalcopyrite, pyrite, arsenopyrite) and iron oxides (magnetite). The host rock is a gabbro, mainly composed of plagioclase, amphibole/pyroxene, chlorite, stilpnomelane, quartz, K-feldspar, with traces of mica, titanite and calcite (SGS, 2023). Lithological units of waste rock are gabbro, pyroxenite and rhyolite. Sulphides are present in the mineralized material, but only in traces for the waste rock near the mineralized zones. Carbonates content in waste rock varies from 0.035% to 5.58%.

This study was done according to the “MELCCFP Guide de caractérisation des résidus miniers et du minerai” applicable in Quebec (MELCC, 2020). According to this Guide’s criteria, the mineralized material is classified as acid-generating and metal-leachable; waste rock units are potentially acid-generating and potentially metal-leachable; and tailings are non-acid-generating and non-metal-leachable (Turgeon, 2023).

A full geochemical characterization study for tailings, mineralized material, waste rocks, and overburden needs to be conducted to confirm the ML-ARD classification of these materials. More samples are needed to cover the entire area, achieving a good representativity of the waste rock material. Other testing, such as kinetic tests, could also be carried

out to get more information about the geochemical behaviour of these materials. The Opémiska site was built with waste rock from the former mine and there is no evidence of ML-ARD on the site (Figure 20-1). There is no treatment (active or passive) for the water flowing out of the old tailings facilities. Baseline study for surface water and groundwater will bring more information about water quality around the site.

Figure 20-1: Old Tailings Facilities Built with Waste Rock from the Former Opémiska Mine



Source: Geodoz, 2025.

20.2.2 Waste Rock, Tailings and Overburden Management

The CDSF was designed for the storage of filtered tailings (56.3 Mt) and waste rock (270.7 Mt). Tailings will be dewatered to the range of 15% to 18% solids on a weight basis. Dams are not required for the stability of the CDSF. Waste rock will be sent to the CDSF. Based on the uncertainty about the ML-ARD potential of waste rock, the CDSF was designed with a liner to collect seepage and runoff water into a basin. In the case that some lithological units of waste rock would be non-acid-generating, a certain amount could be used for the construction of the mine site or sent to

Perry pit after its depletion (end of mining Phase 3). Starting in Year 14, the tailings will be sent directly to Springer pit in slurry form (19.2 Mt). Superficial overburden below the CDSF will be stripped and stored next to the CDSF for the first years of operations. During the expansion of the CDSF, overburden stripped will be placed directly on the CDSF. An overburden pile will be placed north of the pits for the storage of overburden from the mine site (15.0 Mt).

The old tailings facilities from the former Opémiska mine do not have the capacity to receive more tailings or waste rock. The area was reclaimed, and it is advisable to leave the old tailings in their current conditions.

20.2.3 Mineralized Material Management

Four mineralized material stockpiles are located near the plant. Material of low-, medium-, high- and very high-grades will be stored into these piles. During the last years of mine life, it is planned to process the mineralized material from stockpiles. Accordingly, the stockpiles will be temporary, and their volumes will be variable during the operations.

20.2.4 Water Management

All mining waters will be flowing in a closed system. That means that every water in contact with the mine site will be collected, sent into basins or recirculated for the plant operations. The final effluent will meet water quality criteria, and the water will be treated, if needed, before being released in the receiving environment.

20.3 Permitting Requirements

The construction, production and closure of a mine is regulated by several acts and regulations at three different levels: federal, provincial and municipal. The overview of the regulatory context described in this section is based on acts and regulations in place at the time of the preparation of this PEA. The process for permitting related to the environmental and social impact assessment has not yet begun.

20.3.1 Federal Regulations

20.3.1.1 Impact Assessment Act (S.C. 2019, c.28, s.1) and Physical Activities Regulations (SOR/2019-285)

Under the *Impact Assessment Act* (IAA), the projects designated by the Physical Activities Regulations are subjected to the environmental assessment procedure. Consequently, an environmental assessment under the IAA is required for a project that involves the construction, operation, decommissioning, and abandonment of a new metal mine, other than a rare earth element mine, placer mine or uranium mine, with a production capacity of 5,000 t/d or more. This situation is applicable for the Opémiska Project with a production capacity of 12,500 t/d.

The federal impact assessment process consists of five phases (IAAC, 2025):

- Planning – Submittal by the proponent of the initial project description to the IAAC, followed by a detailed project description. The IAAC will provide the Permitting Plan and the Tailored Impact Statement Guidelines.
- Impact Statement – The proponent collects information and conducts studies to present a detailed technical document as per the requirements set out in the Guidelines.

- Impact Assessment – The IAAC begins its analysis, considering potential environmental, health, social and economic impacts of proposed projects, including benefits.
- Decision-making – The impact assessment report and Crown consultation outcomes inform the Minister or Governor in Council decision on whether a project's adverse impacts are in the public interest. If yes, the Minister must establish conditions for the proponent.
- Post-Decision – IAAC will be active in verifying compliance with Decision Statements and correcting non-compliance.

All these phases include consultation and consideration of the public and Indigenous peoples concerns and interests. Indigenous peoples is a collective name recognized in Canada for First Nations, Inuit, and Métis.

20.3.1.2 Fisheries Act (R.S.C., 1985, c. F-14) and Metal and Diamond Mining Effluent Regulations (SOR/2002-222)

Under the *Fisheries Act*, the Metal and Diamond Mining Effluent Regulations (MDMER) provide the framework for mining activities regarding the protection of fish and fish habitats. The MDMER sets criteria for mining effluents.

20.3.1.3 Other Federal Acts or Regulations

A mining project must comply with other regulations. For example, they are subject to the *Species at Risk Act* (S.C. 2002, c. 29), the *Migratory Birds Convention Act* (S.C. 1994, c. 22), the *Explosives Act* (R.S.C., 1985, c. E-17), and all regulations under these acts that may apply.

20.3.2 Provincial Regulations

20.3.2.1 Environmental Quality Act (CQLR, c. Q-2)

The *Environmental Quality Act* (EQA) is the main environmental protection law in Quebec. It leads to numerous regulations that aim to protect the environment, including its ecological, social, and economic dimensions. Provisions under Title II of the EQA are applicable to the James Bay and Northern Québec region. Schedule 1 of Section 22 of the James Bay and Northern Quebec Agreement (JBNQA) provides a list of projects that are subject to the assessment and review procedure (Schedule A of the EQA). All mining developments, including the additions to, alterations of modifications of existing mining developments, are automatically subject to the procedure. This situation is applicable for the Opémiska Project.

The social and environmental impacts assessment and review procedure involves several phases (COMEX, 2025):

- Preliminary Information Statement – Submittal of a notice of intent and preliminary information on the project to the Administrator.
- Assessment and Directive – Documentation is sent to the evaluating committee (COMEV) that will issue a directive. The nature and scope of the impact study (to be carried out by the proponent) are described in this document.

- Preparation of the Impact Study – The proponent undertakes an impact study that complies with the directive and meets the expectations of the review committee (COMEX).
- Review – Analysis of the documents by the COMEX. Questions, comments, further research or additional studies could be asked to the proponent. The COMEX may also hold public hearings or other forms of consultation. Recommendation from the COMEX is submitted to the administrator.
- Decision – Issue of a certificate of authorization if the project is approved. Monitoring and control are planned as a post-decision phase.

After the issue of the certificate of authorization, other authorizations under Article 22 of the EQA must be obtained for the project activities. These authorizations are reviewed by the Ministère de l'Environnement, de la Lutte contre les changements climatiques, de la Faune et des Parcs (MELCCFP), through the Direction régionale de l'Abitibi-Témiscamingue et du Nord-du-Québec.

The Ministère des Ressources naturelles et des Forêts (MRNF) published a document that provides an exhaustive list of permits required (MRNF, 2023). The project will have to conform with the requirements of the latest version of Directive 019 on the Mining Industry (MELCCFP, 2025), since the authorizations of the former Opémiska mine are no longer effective.

20.3.2.2 Mining Act (CQLR, c. M-13.1)

The *Mining Act* provides the regulatory framework for the mining lease, the rehabilitation and restoration plan (closure plan), and the financial guarantee. To obtain the mining lease, the project's survey plan must be formally approved by the Office of the Surveyor-General of Quebec, the closure plan must be approved by the MRNF and the authorization under the Environmental Quality Act (EQA) must be issued. More details about the closure plan are described in Section 20.5.

20.3.2.3 Act Respecting the Preservation of Agricultural Land and Agricultural Activities (CQLR, c. P-41.1)

It is the function of the Commission de protection du territoire agricole du Québec (CPTAQ) to secure the preservation of the agricultural land, and to promote the preservation and development of agricultural activities and enterprises. Agricultural activities (e.g., blueberries, potatoes) are active in Chapais and its surroundings, thus the mining project needs to be developed in accordance with this Act and its regulations.

20.3.3 Municipal Regulations

The municipal regulations include local municipalities and regional county municipality (RCM) authorities. The Opémiska Project is located in the town of Chapais, which is part of the territory of Eeyou Istchee James Bay. Since 2014, the area is governed by the Eeyou Istchee James Bay Regional Government (EIJBRG), which was established through an agreement between the Cree Nation and the Quebec government, combining the former municipalities of Baie-James and Jamésie. The EIJBGR has the authority of jurisdiction as an RCM. This territorial organization is complemented by the James Bay and Northern Quebec Agreement (JBNQA) signed in 1975. The territorial regime

established under the JBNQA divides these lands into three categories (I, II and III). The Opémiska Project lies on Category III lands.

20.4 Social or Community Requirements

One of the early steps of social requirements involves identifying the various stakeholders who are likely to be affected or concerned with the Opémiska Project. Stakeholders are individuals, groups, or organisations who could be directly or indirectly affected by the project or has an interest in its outcome. Consultations with stakeholders should be initiated early in the project development to ensure that the concerns and issues that could be raised are clearly identified to take them into consideration in the project. Consultation meetings with stakeholders and First Nation authorities are an opportunity for participants to ask questions, share their comments and express their concerns about the project.

An engagement process will be implemented by XXIX to meet various stakeholders concerned by the Opémiska Project, First Nations communities, land users, development economic organizations, environmental groups and academic community. In 2024, XXIX and the town of Chapais have established an official working group named “La Table Ville/Mine”. This group was created to engage a constructive dialogue between both parties. The development of the Opémiska Project will be carried out in accordance with the values and expectations of the population of Chapais. In October 2025, XXIX met the citizens of Chapais by offering an information and discussion meeting and a “Café-citoyen” where each person had the opportunity to share their comments and concerns on the Opémiska Project. Thematic meetings are planned for the next months. XXIX has also adopted a local purchasing policy to maximize the positive impact of its activities on the local economy. In parallel, XXIX will also meet the First Nations authorities of Oujé-Bougoumou and Waswanipi.

20.5 Mine Closure and Reclamation

A land rehabilitation and restoration plan (closure plan) and a financial guarantee are required under the provincial Mining Act (m-13.1, article 232). Guidelines for preparing mine closure plan in Quebec are provided by the Ministry of Natural Resources and Forests (MRNF). Rehabilitation involves all activities after mining operations to return the site affected by mining activity to a satisfactory condition. The latest version of this guide (MRNF, 2024) defines the aim to a satisfactory condition by:

- eliminating unacceptable health hazards and ensuring public safety
- limiting the production and circulation of substances that could damage the receiving environment, and trying to eliminate any form of maintenance and monitoring in the long term
- restoring the site to a condition that approaches the initial or surrounding ecosystem, including tailings and mine waste rock accumulation areas where possible
- restoring the infrastructure areas to a condition compatible with future use.

The closure plan must be approved by the minister before the issue of the mining lease. A financial guarantee to fully implement the closure plan must be provided in three payments in the first two years following its approval. The closure plan must be revised every five years, unless a shorter period is fixed by the minister.

Finally, monitoring is required during the period specified by the approved plan to ensure that the rehabilitation has been successfully completed. When the minister declares to be satisfied with the rehabilitation and restoration work carried out, a certificate attesting that the operator has been released from its obligations is issued.

The closure plan is not yet available for the Opémiska Project, although some conceptual designs have already been incorporated into the project to reduce the footprint and impact on the landscape. The use of filtered tailings eliminates the risks of dam failure, since no dam is needed for the CDSF. General design of the CDSF allows progressive reclamation with the overburden stocked next to the CDSF. The overburden from the mine site will be stocked next to the pits and will be used at the end of the mine life for reclamation. Starting in Year 14, the Springer pit will be backfilled with tailings from the plant instead of being sent to the CDSF. At the end of mine life, all buildings will be dismantled, the soil will be levelled, the topsoil will be replaced, and it will be revegetated. Estimated closure costs have been benchmarked against recent mining projects in similar jurisdictions. Refer to Section 21 for a review of closure costs.

Closure plan of the former Opémiska mine was prepared by Minnova in 1991. Closure work was conducted by Inmet Mining Corporation and the certificate of release was issued by the MRNF in 2001. Closure work has been carried out in accordance with the requirements of the MRNF, and the site was considered free of risks for the environment, human health and safety, particularly of acid mine drainage. Unfortunately, a dam failure occurred in 2008 from basin no. 3. Water and tailings were spread in Slam Creek, and damage was caused to Route 113 and the railway. Emergency work was carried out, and corrective measures were implemented by the MRNF. This kind of disaster will not happen again by using filtered tailings for future operation.

21 CAPITAL AND OPERATING COSTS

21.1 Introduction

The capital and operating cost estimates of the PEA were prepared to support the development of the project on and off site. The capital and operating cost estimates are based on an open pit mining operation, the processing of mineralized material on site at a rate of 4.6 Mt/a, and the shipment of copper concentrate to customers off site. All cost figures reported are in Canadian dollars (CAD, C\$), unless specified otherwise.

21.2 Capital Costs

21.2.1 Overview

The capital cost estimate developed in this PEA was prepared to a Class 5 estimate with an accuracy of $\pm 50\%$ as defined by the Association for the Advancement of Cost Engineering International (AACE International). Generally, engineering performed to date is between 1% to 5% of full project definition.

The capital cost estimate of the project includes initial capital costs and sustaining capital costs and is estimated at \$1,048 million, as summarized in Table 21-1.

Table 21-1: Capital Cost Estimate

Cost Area	Initial Capital Cost (\$M)	Sustaining Cost (\$M)	Total Cost (\$M)
Mining	45.6	230.3	276.0
Processing	271.0	-	271.0
Waste and Water Management	21.4	60.6	82.0
On-Site Infrastructure	16.2	-	16.2
Off-Site Infrastructure	27.0	-	27.0
Town Relocation	-	100.0	100.0
Indirect Costs	114.6	-	114.6
Contingency	121.4	-	121.4
Closure	-	40.0	40.0
Total	617.3	430.9	1,048.2

Ausenco, Evomine, SRK and Geodoz prepared the capital cost estimate. Their areas of contribution are summarized in Table 21-2.

Table 21-2: Capital Cost Estimate Responsibilities

Cost Area	Company Responsible
Mining	Evomine
Processing	Ausenco
Waste and Water Management	SRK
On-Site Infrastructure	Ausenco
Off-Site Infrastructure	Ausenco
Town Relocation	Evomine
Indirect Costs	Ausenco
Capitalized Operating Costs	Evomine
Contingency	Ausenco
Closure Costs	Geodoz

21.2.2 Basis of Estimate

The capital cost estimate is based on the following considerations:

- The estimate is expressed in Canadian dollars using Q4 2025 as a base period, with no provisions for cost escalation.
- Labour productivities were benchmarked against other projects in similar environments.
- Cost components were converted to Canadian dollars from US dollars with a fixed CAD/USD exchange rate of 1.35.
- Each capital cost input to the estimate was assigned with specific accuracies based on estimating methodology, pricing, and quantity estimation.
- Diesel fuel cost was estimated at \$1.50 per litre (L).

21.2.3 Mine Capital Costs

The initial capital cost for mining was estimated at \$45.6 million and is summarized in Table 21-3. Initial capital costs for this area represent the purchase of mobile equipment as well as the costs associated capital spares purchases, shipping, and assembly. A capital lease on the acquisition cost of major equipment was considered, limiting cash payments for mobile equipment prior to commercial production.

Table 21-3: Mining Initial Capital Costs

Cost Element	Initial Capital Cost (\$M)
Acquisitions	102.0
Capital Spares	8.0
Capital Lease	-64.3
Total	45.6

21.2.4 Process Capital Costs

Data for the process capital cost estimate have been obtained from numerous sources, including the following:

- final mine plan and process plant throughput
- conceptual process design criteria and flowsheet (refer to Section 17)
- mechanical equipment costs derived from first principles and priced using Ausenco’s database of recent Canadian studies and projects
- costs for concrete, steel, instrumentation, electrical distribution, in-plant piping, and platework were factored by benchmarking against similar projects with equivalent technologies and unit operations.

The initial capital cost for processing was estimated at \$271 million and is summarized in Table 21.4.

Table 21-4: Processing Initial Capital Costs

Cost Element	Initial Capital Cost (\$M)
Crushing	31.9
Grinding	108.5
Flotation and Re grind	43.3
Reagents	2.9
Water and Air Services	5.4
Tailings Dewatering	79.2
Total	271.0

21.2.5 Waste and Water Management Capital Costs

The initial capital cost for waste and water management was estimated at \$21.4 million and is summarized in Table 21.5.

Table 21-5: Waste and Water Initial Capital Costs

Cost Element	Initial Capital Cost (\$M)
Tailings Management Facility	14.5
Surface Water Management	6.9
Total	21.4

21.2.6 Infrastructure Capital Costs

On-site and off-site infrastructure costs include the following:

- earthworks costs for the site access road, process plant, and maintenance area earthworks
- general site buildings, including truck shop and wash, plant warehouse, plant office, security gatehouse, truck shop warehouse, truck shop office, mine office, and changerooms
- new 120 kV substation to replace the existing Chapais town substation (substation was sized based on the estimated project electrical loads, plus an allowance for the loads for the town of Chapais)
- 25 kV power distribution lines from the new substation location to the town and to the process plant.

Costs for the above items have been estimated using Ausenco’s database for recent Canadian mining projects.

Costs for relocating Route 113 are not included.

The initial capital cost for infrastructure was estimated at \$43.2 million and is summarized in Table 21.6.

Table 21-6: Infrastructure Initial Capital Costs

Cost Element	Initial Capital Cost (\$M)
On-Site Infrastructure	6.0
On-Site Buildings	10.2
Off-Site Power Supply	27.0
Total	43.2

21.2.7 Indirect Capital Costs

Indirect initial capital costs were estimated at \$236 million and are summarized in Table 21.7.

Table 21-7: Indirect Initial Capital Costs

Cost Element	Initial Capital Cost (\$M)
Field Indirects	15.7
Project Delivery (EPCM)	62.8
Commissioning Operations Readiness	1.1
Vendor Representatives	2.3
Spares	5.7
First fills	2.8
Owners Project Management	15.7
Mining Capitalized Operating Costs	8.4
Contingency	121.4
Total	236.0

21.2.8 Sustaining Capital

Total sustaining capital costs were estimated at \$431 million and are summarized in Table 21-8. Sustaining capital costs include capital lease and replacement costs for the mining fleet, tailings management facility expansion costs, the costs associated with relocating part of the town of Chapais for Phase 4 of the project, as well as closure costs.

Table 21-8: Sustaining Capital Costs

Cost Element	Sustaining Capital Cost (\$M)
Mining – Equipment Acquisitions	35.7
Mining – Equipment Rebuilds	101.3
Mining – Capital Lease Payments	93.4
Waste and Water Management	60.6
Town Relocation	100.0
Closure	40.0
Total	430.9

21.3 Operating Costs

21.3.1 Overview

The operating costs estimate developed in this PEA was prepared to a Class 5 estimate with an accuracy of $\pm 50\%$ as defined by the Association for the Advancement of Cost Engineering International (AACE International). Generally, engineering performed to date is between 1% to 5% of full project definition.

Operating costs are summarized in Table 21-9. These include mining, processing, waste and water management, and general and administration (G&A) costs. The average life-of-mine operating cost estimate is \$34.52/tonne processed.

Table 21-9: Operating Cost Summary

Cost Area	Life-of-Mine Total (\$M)	\$/t Processed	% of Total
Mining	1,594.7	20.66	59.8
Process	819.9	10.62	30.8
Water Management	6.5	0.08	0.2
G&A	244.0	3.16	9.2
Total	2,665.0	34.52	100.0

21.3.2 Basis of Estimate

The basis of estimate for the operating cost estimate is summarized below according to cost area:

- Mining – Based on recent benchmarks for open pit iron ore projects in Canada and the USA (assuming current diesel and consumable prices).

- Process – Based on first principles, estimated consumptions, and benchmark unit costs from Canadian projects.
- Waste and Water Management – Based on recently quoted equipment prices and estimated material volumes from conceptual designs multiplied by benchmarked unit costs taken from other recent and similar DRA studies.
- G&A – Based on Ausenco benchmark estimates for similar projects in Canada.

21.3.3 Mine Operating Costs

Mining operating costs over the life of mine were estimated at \$1,594.7 million, representing \$20.66/tonne processed. These are summarized in Table 21-10 and presented on an annual basis in Table 21-11.

A detailed mine cost build-up was developed from basic cost elements such as remuneration costs, consumable prices, fuel prices, and equipment productivities. Equipment operating costs were estimated for each type of equipment used, which includes operation and maintenance labour, parts (maintenance and repairs), fuel consumption, lubricant consumption, ground-engaging tools and tires if applicable. Equipment operating costs were determined from various sources including information from the major suppliers and benchmarked costs from operations in similar environments.

Activities reflected in the costs include drilling and blasting waste rock and mineralized material, and mucking and hauling material from the open pit mine to the crusher, mineralized material stockpile, or waste management facility. Included in mucking and hauling costs are the rehandling of stockpile material and the transportation of tailings to the tailings management facility. Also considered are mine services such as dewatering and material placement at the overburden, waste rock, and tailings management facilities, as well as supervision, grade control, and technical services activities.

Table 21-10: Mining Operating Costs Summary

Cost Element	Total (\$M)	Avg/Year (\$M)	\$/t Mined	\$/t Processed	% of Total
OP Definition Drilling	21.5	1.26	0.06	0.28	1.3
OP Pre-Shear Drilling	26.2	1.53	0.07	0.34	1.6
OP Production Drilling	182.7	10.68	0.50	2.37	11.5
OP Blasting	248.4	14.52	0.68	3.22	15.6
OP Mucking	139.9	8.18	0.39	1.81	8.8
OP Hauling	524.3	30.66	1.44	6.79	32.9
OP Mine Services	62.7	3.67	0.17	0.81	3.9
Surface Services	123.6	7.23	0.34	1.60	7.7
Surface Tailings Transportation	60.4	3.53	0.17	0.78	3.8
Overhead	70.3	4.11	0.19	0.91	4.4
Grade Control	58.3	3.41	0.16	0.76	3.7
Technical Services	76.4	4.47	0.21	0.99	4.8
Total	1,594.7	93.26	4.39	20.66	100.0

Table 21-11: Annual Mining Operating Costs

Cost Element	Unit	Total (\$M)	Y1	Y2	Y3	Y4	Y5	Y6	Y7	Y8	Y9	Y10	Y11	Y12	Y13	Y14	Y15	Y16	Y17	Y18
OP Definition Drilling	M\$	21.5	2.0	2.1	2.0	2.0	1.9	1.9	1.2	1.5	1.4	1.3	1.5	1.5	1.1	-	-	-	-	-
OP Pre-Shear Drilling	M\$	26.2	1.8	2.3	2.3	2.4	2.2	2.3	2.2	2.2	2.0	2.1	1.9	1.5	0.9	-	-	-	-	-
OP Production Drilling	M\$	182.7	12.4	16.0	16.1	17.1	15.6	16.3	15.5	15.5	14.2	15.0	13.1	10.0	5.8	-	-	-	-	-
OP Blasting	M\$	248.4	16.5	23.0	22.9	23.4	22.2	23.4	21.5	21.7	19.1	19.8	18.4	11.7	4.8	-	-	-	-	-
OP Mucking	M\$	139.9	9.2	11.0	10.5	10.9	10.5	10.9	11.6	10.5	10.0	10.2	9.2	8.1	6.3	2.7	2.7	2.7	2.7	0.2
OP Hauling	M\$	524.3	27.1	41.7	42.9	45.8	44.7	46.8	42.1	41.5	39.3	46.1	46.0	38.1	22.3	-	-	-	-	-
OP Mine Services	M\$	62.7	4.4	4.5	4.6	4.7	4.8	4.8	4.8	4.8	4.8	4.8	4.8	4.8	4.0	0.5	0.5	0.5	0.5	0.0
Surface Services	M\$	123.6	9.8	9.5	9.5	9.5	9.5	9.5	9.6	9.6	9.7	9.5	9.5	9.8	8.6	-	-	-	-	-
Surface Tailings Transportation	M\$	60.4	4.1	4.7	4.7	4.7	4.7	4.7	4.7	4.7	4.8	4.8	4.8	4.8	4.2	-	-	-	-	-
Overhead	M\$	70.3	5.5	5.5	5.5	5.5	5.5	5.5	5.5	5.5	5.5	5.5	5.5	5.5	4.6	-	-	-	-	-
Grade Control	M\$	58.3	4.7	5.5	5.4	5.4	5.2	5.2	4.2	4.6	4.2	4.1	4.3	3.5	2.1	-	-	-	-	-
Technical Services	M\$	76.4	6.0	5.9	5.9	5.9	5.9	5.9	5.9	5.9	6.0	5.9	5.9	6.0	5.0	-	-	-	-	-
Total	M\$	1,594.7	103.4	131.6	132.3	137.3	132.9	137.3	128.9	128.1	120.9	129.2	124.8	105.2	69.9	3.2	3.2	3.2	3.2	0.3
Total	\$/t mined	4.39	4.05	3.78	4.15	3.97	4.17	4.02	3.59	4.22	4.52	4.43	5.12	6.42	9.72	-	-	-	-	-

21.3.4 Processing Operating Costs

The total operating costs for processing were estimated at \$820 million, representing \$10.62/t processed, and are summarized in Table 21-12.

Table 21-12: Processing Operating Costs

Cost Element	Total (\$M)	\$/t Processed	Avg/year (\$M)
Labour	251.8	3.26	14.8
Power	156.0	2.02	9.2
Reagents	97.2	1.26	5.7
Consumable	250.9	3.25	14.8
Lab/ Assays	10.0	0.13	0.6
Mobile Equipment	6.2	0.08	0.4
Maintenance	49.4	0.64	2.9
Total	819.9	10.62	48.2

21.3.4.1 Process Plant Labour

The personnel requirements for the Opémiska Project were estimated by benchmarking against similar projects in the same region with access to similar labour pools. The labour costs incorporate personnel requirements for plant operations, such as management, metallurgy, operations, maintenance, site services, assay lab, and contractor allowance. The total process plant labour averages 102 employees through the life of the project.

Individual personnel were divided into their positions and classified as either eight-hour or 12-hour shift employees. Salaries were estimated using benchmarks for similar projects in similar regions. An organizational staffing plan outlining the labour requirement for the process plant is shown in Table 21-13. Costs include all benefits and bonuses.

Table 21-13: Process Plant Staffing Plan and Labour Cost Summary

Position	Persons per Shift	No. Shifts	Total Persons	Cost per Person (C\$/person/a)	Total Cost (C\$/a)
Chief Metallurgist/Operations Superintendent	1	1	1	255,969	255,969
Senior Metallurgist	1	1	1	173,109	173,109
Chief Assayer	1	1	1	197,691	197,691
Maintenance Superintendent	1	1	1	292,110	292,110
Management Subtotal	4	4	4	918,879	918,879
Shift Foreperson	1	4	4	197,691	790,764
Control Room Operator	1	4	4	173,742	694,970
Crusher Operator	1	4	4	151,686	606,744
Crusher Labourer	1	4	4	115,354	461,417
Grinding Operator	1	4	4	157,190	628,760
Grinding Labourer	1	4	4	115,354	461,417
Flotation Operator	1	4	4	157,190	628,760

Position	Persons per Shift	No. Shifts	Total Persons	Cost per Person (C\$/person/a)	Total Cost (C\$/a)
Flotation Labourer	1	4	4	115,354	461,417
Reagents Labourer	1	4	4	115,354	461,417
Concentrate Load out Operator	1	4	4	151,686	606,744
Tailings Filter Plant Operator	1	4	4	157,190	628,760
Filter Plant/Utility Labourer	1	4	4	115,354	461,417
Tailings Load Out Operator	1	4	4	115,354	461,417
Operations Subtotal	13	52	52	1,838,501	7,354,002
Plant Metallurgist	2	1	2	128,159	256,319
Metallurgical Tech	2	1	2	122,541	245,082
Assay Laboratory Tech	2	2	4	122,541	490,163
Process Plant Trainer	1	2	2	174,163	348,325
Technical Services Subtotal	7	6	10	547,404	1,339,889
Maintenance Foreperson	1	2	2	148,154	296,309
Maintenance Planner	1	2	2	143,238	286,477
Electrician	2	2	4	137,559	550,237
Electrical Apprentice	1	2	2	120,300	240,600
Welders	2	2	4	132,769	531,076
Instrument Tech	2	2	4	132,769	531,076
Millwright/Fitter	4	4	16	137,559	2,200,949
Mechanical Apprentice	1	2	2	120,300	240,600
Maintenance Subtotal	14	18	36	1,072,649	4,877,324
Allowance – Mill relining	-	-	-	500,000	500,000
Process Plant Labour Total	38	80	102	4,877,433	14,990,094

21.3.4.2 Power

The processing power draw was based on the average power utilization of each motor on the electrical load list for the process plant. An estimated 166 GWh are nominally required annually. Table 21-14 shows a breakdown of the power consumption and cost summary by plant area.

Table 21-14: Process Plant Power Cost Summary

Area	Installed Power (kW)	Power Consumption (kWh/y)	Power Cost (C\$/a)
Crushing	1,632	2,826	158,280
Grinding	17,286	116,006	6,496,313
Flotation and Re grind	2,837	13,396	750,151
Reagents	30	138	7,743
Water and Air Services	1,177	3,183	178,230
Tailings Dewatering	6,364	23,100	1,293,624
HVAC, Lighting and Buildings	1,875	6,899	386,316
Total	31,200	165,547	9,270,657

21.3.4.3 Reagents and Consumables

Individual reagent consumption rates were estimated based on the metallurgical testwork results, Ausenco’s in-house database and experience, industry practice, and peer-reviewed literature. Consumables consumption rates were estimated based on engineering calculations, Ausenco’s in-house database and experience, standard industry practice, and recently received Canadian vendor quotes. Reagent and consumables unit costs were obtained through benchmarking from recently received Canadian quotes and Ausenco’s unit cost database.

Table 21-15 summarizes the annual and unit costs for all reagents and consumables.

Table 21-15: Reagent and Consumables Costs

Reagents and Consumables	Annual Cost (C\$/k/y)	Unit cost (C\$/t milled)
Reagents		
Hydrated Lime	3,103	0.67
Aero 3894	1,510	0.33
MIBC	689	0.15
Flocculant	475	0.10
Operating Consumables		
SAG Mill Media	5,030	1.09
Ball Mill Media	5,303	1.15
Regrind Mill Media	635	0.14
Concentrate Filtration Cloths	10	0.002
Tailings Filtration Cloths	672	0.15
Maintenance Consumables		
SAG Mill Liners	1,398	0.30
Ball Mill Liners	655	0.14
Regrind Mill Liners	395	0.09
Jaw Crusher Liners	171	0.04
Cone Crusher Liners	591	0.13
Secondary Screen Media	71	0.02

21.3.4.4 Laboratory

Operating costs associated with laboratory and assay activities were estimated by Ausenco according to the anticipated number of assays per day per year. These costs are related to sample preparation and assays for processing plant samples.

21.3.4.5 Mobile Equipment

Vehicle costs are based on a scheduled number of light vehicles and mobile equipment (including fuel, maintenance, spares, times and annual registration, and insurance fees). Mobile equipment required includes light vehicles, forklifts, front-end loaders, a bulldozer, and a flat-bed truck.

21.3.4.6 Equipment Maintenance

Annual maintenance costs were calculated based on the total installed mechanical capital cost by area using weighted average factors based on industry benchmarks ranging between 2% and 5%, as shown in Table 21-16. The factors were applied to the cost of the mechanical equipment as shown in the capital cost estimate.

Table 21-16: Process Plant Maintenance Cost Summary

Area	Maintenance Factor (%)	Maintenance Cost (C\$/y)
Crushing	5.0	0.4
Grinding	4.0	1.2
Flotation & Regrind	4.0	0.5
Reagents	4.0	0.0
Water & Air Services	2.0	0.0
Tailings Dewatering	4.0	0.8
Total		2.9

21.3.5 Water Management Operating Costs

The total operating costs for water management activities were estimated at \$6.5 million, representing \$0.08/tonne processed, and are summarized in Table 21-17.

Table 21-17: Water Management Operating Costs

Cost Element	Total (\$M)	\$/t Processed	Avg/Year (\$M)
Water Management	6.5	0.08	0.35
Total	6.5	0.08	0.35

21.3.6 General and Administrative Operating Costs

General and administrative (G&A) costs are expenses not directly related to the operation of the process plant but required to support safe and effective operation of the facility and satisfy legislative requirements in some cases.

These costs were developed using Ausenco’s in-house data on existing operations, and include costs such as the following:

- human resources, including training, recruiting, and community relations
- site administration, maintenance, and security, including subscriptions, memberships, advertisement, office supplies and garbage disposal
- health and safety, including personal protective equipment, hospital service cost, and first aid
- environmental, including water sampling and tailings management facility operating costs
- IT & telecommunications, including hardware and support services
- contract services, including insurance, consulting, sanitation and cleaning, licence fees, and legal fees.

The total operating costs for general and administrative activities were estimated at \$244 million, representing \$3.16/t processed, and are summarized in Table 21-18.

Table 21-18: General and Administrative Operating Costs

Cost Element	Total (\$M)	\$/t Processed	Avg/Year (\$M)
G&A Labour	140.50	1.82	8.26
Expenses	98.82	1.28	5.81
Site Services	3.86	0.05	0.23
Total	244.0	3.16	14.35

22 ECONOMIC ANALYSIS

22.1 Forward-Looking Information

The economic and financial evaluation presented in this technical report utilizes a discounted cash flow method, both on a pre-tax and after-tax basis. The metal pricing used in the evaluation was determined in Section 19. The financial model provides results in terms of NPV, payback period, and IRR for the project. The economic analysis is conducted in real terms, without considering inflation factors, using Q4 2025 Canadian dollars. The analysis does not take into account project financing.

The economic model estimates cash flows on an annual basis for the life of the project, based on the level of engineering and design appropriate for a PEA.

Cash flow projections for the life of the project are based on sales revenue, operating costs, capital expenditures and other cost estimates. Capital expenditures are estimated in five categories: initial capital, growth capital, sustaining capital, closure and reclamation cost, and working capital. Operating cost estimates include labour, reagents, maintenance, supplies, services, fuel, and power. Other costs, such as royalties, depreciation, and taxes, are estimated based on the current mine and processing plans.

The economic results are calculated from the start of the initial capital expenditures.

The results of the economic analyses discussed in this section represent forward-looking information as defined under the Canadian securities law. These results are subject to known and unknown risks, uncertainties, and other factors that may cause actual results to differ materially from those presented here. The forward-looking information includes, but is not limited to, the following:

- assumed metal prices
- cost inflation
- proposed mine production plan
- assumptions regarding mining dilution and mining recovery
- metallurgical recovery
- proposed sustaining and operating costs
- labour and materials availability
- labour and materials costs being approximately consistent with the assumptions in the report
- assumptions regarding closure costs
- assumptions regarding environmental, social, and licensing risks

- changes to tax rates or tax credit policies
- unexpected variations in the amount of mineralized material and material grade
- geotechnical or hydrogeological considerations during mining that differ from the assumptions
- ability to maintain a social license to operate
- unrecognized environmental risks
- unforeseen reclamation expenses
- failure of plant, equipment, and processes to operate as anticipated
- absence of significant disruptions affecting the development and operation of the project
- availability of certain consumables and services, and the prices for electricity and other key supplies being approximately consistent with the assumptions in the technical report.

22.2 Methodologies Used

The project has been evaluated using a discounted cash flow (DCF) analysis based on an 8% discount rate. Cash inflows consist of annual revenue projections. Cash outflows consist of capital expenditures including pre-production costs, operating costs, taxes, and royalties. These are subtracted from the inflows to arrive at the annual cash flow projections. Cash flows are 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 metals price, discount rate, head grade, recovery, total operating cost, exchange rate, and total capital costs.

The capital and operating cost estimates developed specifically for this project are presented in Section 21 of this report. The economic analysis has been run on a constant dollar basis with no inflation.

22.3 Financial Model Parameters

22.3.1 Assumptions

The economic analysis was performed assuming a copper price of US\$4.35/lb, a gold price of US\$3,000/oz, and a silver price of US\$30.00/oz. These metal prices were based on a three-year trailing average and consensus analyst estimates. 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 taken into account. 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
- total mine life of 17.1 years

- cost estimates in constant Q4 2025 Canadian dollars with no inflation or escalation factors considered, with an exchange rate of 1.35 CAD to 1.00 USD used.
- results based on 100% ownership with a 1% net smelter return (NSR) royalty
- capital costs funded with 100% equity (no financing cost except as stated in mining equipment)
- cash flows discounted to start of construction period using middle of period discounting
- project revenue is derived from the sale of copper concentrate with gold and silver credits
- no contractual arrangements for refining currently exist.

22.3.2 Taxes

The project is subject to three levels of taxation: federal corporate income tax, provincial corporate income tax, and provincial mining taxes. The taxation calculations for the project were compiled by Evomine.

The current Canadian tax system applicable to mineral resource income was used to assess the annual tax liabilities for the project. This consists of federal and provincial corporate income taxes, as well as provincial mining taxes. The federal and provincial (Québec) corporate income tax rates currently applicable over the operating life of the project are 15.0% and 11.5% of taxable corporate income, respectively. The marginal tax rates applicable under the *Mining Tax Act* in Quebec are 16%, 22% and 28% of taxable income and are dependent on the profit margin.

The taxation calculations resulted in estimated total payments of \$740.7 million over the life of mine. Some of the capital costs for the project were also assumed to be eligible for the Clean Technology Manufacturing Investment Tax Credit (CTM ITC), resulting in a refundable tax credit of \$149.6 million, for net tax payments of \$591.1 million over the life of mine.

The CTM ITC (enacted on June 20, 2024) provides a refund of up to 30% of the cost of eligible property used for eligible activities through a refundable investment credit mechanism. A 30% refund of processing-related capital costs via the CTM ITC was considered in the tax calculations; however, there is no guarantee that the credit will be eligible within the timeframe of the project execution.

22.3.3 Royalties

XXIX fulfilled all its obligations under the terms of the option agreement with Ex-In on June 16, 2023 and executed the purchase agreement of 11 mining titles. As a result, these claims have now been transferred to and are 100% owned by XXIX, subject to a 2% NSR royalty, 50% of which can be purchased by XXIX for \$4.5 million.

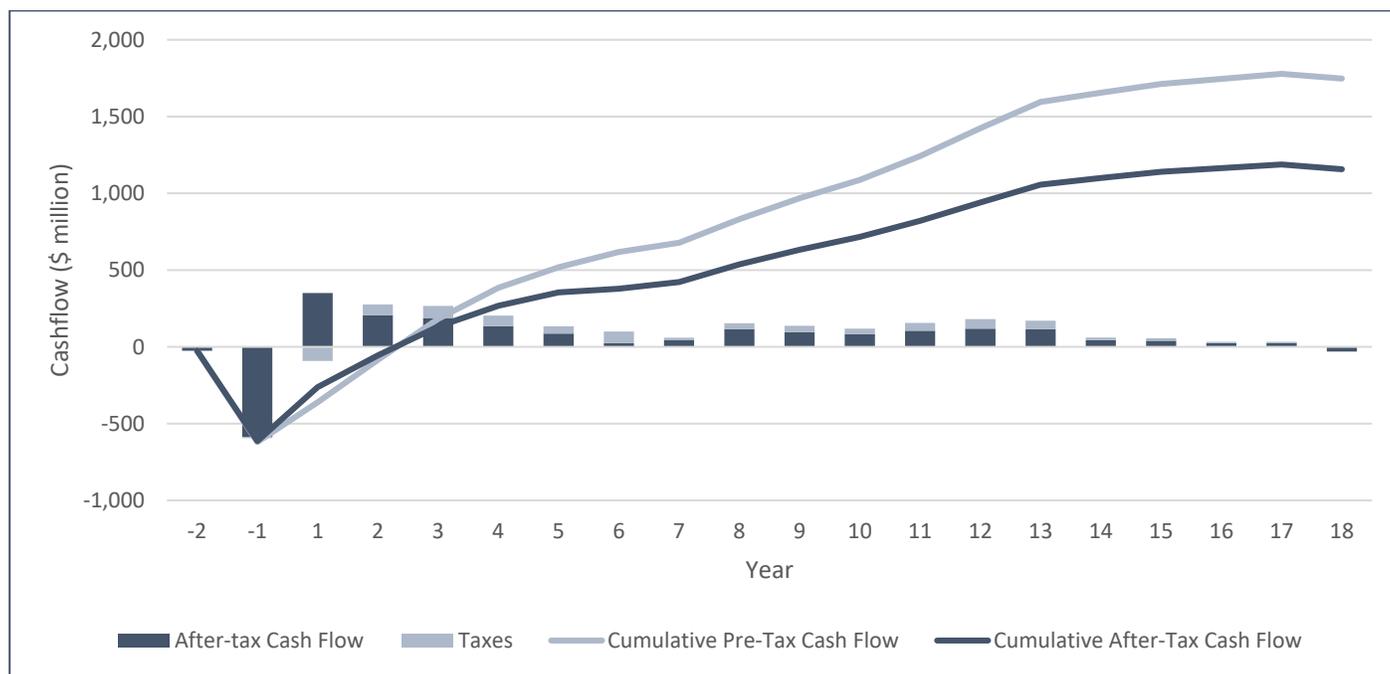
XXIX is in an earn-in process to acquire 175 additional mining titles subject to a 2% NSR royalty, 50% of which can be purchased by XXIX before the commencement of commercial production for \$1.5 million.

For the PEA, it was considered that XXIX would exercise the option to repurchase 50% of the royalty applicable to the claims on which mining is planned. As such, a 1% NSR royalty was considered. The cost of this repurchase was considered to be incurred prior to a final investment decision and was not considered in the PEA.

22.4 Economic Analysis

The economic analysis was performed assuming an 8% discount rate. The pre-tax NPV (8%) is \$793.0 million; the internal rate of return IRR is 32.1%, and payback period is 2.3 years. On a post-tax basis, the NPV (8%) is \$505.2 million; the IRR is 27.2%, and the payback period is 2.3 years. The analysis was done on an annual cashflow basis; the cashflow output is shown graphically in Figure 22-1.

Figure 22-1: Life-of-Mine Cashflow



Source:

A summary of project economics is shown in Table 22-1. Table 22-2 details the main physical quantities on an annual basis, while Table 22.3 presents the detailed annual cash flows.

Readers are cautioned that the PEA is preliminary in nature. 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 PEA will be realized.

Table 22-1: Economic Analysis Summary

Description	Units	Value
General		
Copper Price	US\$/lb	4.35
Gold Price	US\$/oz	3,000
Silver Price	US\$/oz	30.00
Exchange Rate	USD/CAD	1.35
Mine Life	years	17.1
Production		
Mill Feed Tonnage	kt	77,201
Mill Feed Average Grade – Cu	%	0.481
Mill Feed Average Grade – Au	g/t	0.234
Mill Feed Average Grade – Ag	g/t	1.119
Average Metallurgical Recovery – Cu	%	92.0
Average Metallurgical Recovery – Au	%	79.9
Average Metallurgical Recovery – Ag	%	80.3
Total Metal Recovered – Cu	Mlbs	752.7
Total Metal Recovered – Au	koz	464
Total Metal Recovered – Ag	koz	2,231
Average Annual Production – Cu	Mlbs	44.1
Average Annual Production – Au	koz	27
Average Annual Production – Ag	koz	131
Gold Payable	koz	409
Silver Payable	koz	2,008
Capital Costs		
Initial Capital Costs	\$ million	617.3
Sustaining Capital Costs	\$ million	390.9
Closure Capital Costs	\$ million	40.0
Operating Costs		
Mining Cost	\$/t mined	4.39
Mining Cost	\$/t milled	20.66
Processing Cost	\$/t milled	10.62
Waste and Water Management Cost	\$/t milled	0.08
General and Administrative Cost	\$/t milled	3.16
Total Operating Costs	\$/t milled	34.52
C1 Cash Cost (Net of By-products)	US\$/lb Cu	1.40
C3 Cash Cost (Net of By-products)	US\$/lb Cu	2.50
Pre-Tax Valuation Indicators		
Undiscounted Cashflow	\$ million	1,747.8
NPV (8%)	\$ million	793.0
Payback Period (from Start of Operations)	years	2.3
IRR	%	32.1
After-Tax Valuation Indicators		
Undiscounted Cashflow	\$ million	1,156.8
NPV (8%)	\$ million	505.2
Payback Period (from Start of Operations)	years	2.3
IRR	%	27.2

Table 22-2: Physical Quantities Summary per Year

Description	Units	Total/Avg.	Y-2	Y-1	Y1	Y2	Y3	Y4	Y5	Y6	Y7	Y8	Y9	Y10	Y11	Y12	Y13	Y14	Y15	Y16	Y17	Y18
Overburden Mined	Mt	-	-	-	3.3	3.1	0.1	-	1.1	1.6	5.5	0.2	-	-	-	-	-	-	-	-	-	-
Waste Rock Mined	Mt	-	-	-	14.3	23.3	23.8	26.6	23.2	25.3	26.9	24.9	22.2	25.2	19.3	11.6	4.0	-	-	-	-	-
Mineralized Material Mined	Mt	-	-	-	7.9	8.4	8.0	7.9	7.6	7.2	3.5	5.3	4.5	3.9	5.1	4.8	3.2	-	-	-	-	-
Mineralized Material Processed	Mt	-	-	-	3.8	4.6	4.6	4.6	4.6	4.6	4.6	4.6	4.6	4.6	4.6	4.6	4.6	4.6	4.6	4.6	4.6	0.3
Mineralized Material Grade – Cu	%	0.48	-	-	0.84	0.75	0.65	0.61	0.59	0.74	0.50	0.62	0.58	0.36	0.45	0.48	0.46	0.18	0.17	0.13	0.13	0.13
Mineralized Material Grade – Au	g/t	0.23	-	-	0.4	0.4	0.4	0.4	0.2	0.2	0.1	0.2	0.2	0.4	0.3	0.3	0.2	0.1	0.1	0.1	0.1	0.1
Mineralized Material Grade – Ag	g/t	1.12	-	-	2.7	2.1	1.5	1.7	1.3	0.8	0.9	1.2	1.2	1.1	1.2	1.0	0.4	0.6	0.6	0.5	0.5	0.5
Cu Concentrate Produced (Dry)	kt	1,707	-	-	146.7	158.0	136.9	127.9	124.1	156.1	104.7	130.6	120.9	76.7	94.6	101.2	96.8	37.5	36.0	28.1	28.1	2.1
Cu Concentrate Produced (Wet)	kt	1,844	-	-	158.5	170.7	147.8	138.1	134.0	168.6	113.1	141.0	130.6	82.9	102.1	109.3	104.6	40.5	38.9	30.3	30.3	2.2
Cu Concentrate Grade	%	0.20	-	-	0.20	0.20	0.20	0.20	0.20	0.20	0.20	0.20	0.20	0.20	0.20	0.20	0.20	0.20	0.20	0.20	0.20	0.20
Au Concentrate Grade	g/t	8.46	-	-	8.31	8.34	10.80	10.67	7.25	5.44	4.01	4.67	6.42	16.52	12.05	10.39	7.18	9.67	9.80	10.82	10.82	10.82
Ag Concentrate Grade	g/t	40.65	-	-	57.72	50.29	39.61	48.30	39.87	20.04	31.96	35.46	36.48	49.55	46.61	34.72	13.69	57.73	58.19	61.84	61.84	61.84
Payable Cu	Mlb	715	-	-	61	66	57	54	52	65	44	55	51	32	40	42	41	16	15	12	12	1
Payable Au	koz	409	-	-	34	37	43	40	25	22	10	15	21	38	34	31	19	10	10	9	9	1
Payable Ag	koz	2,008	-	-	245	230	157	179	143	91	97	134	128	110	128	102	38	63	61	50	50	4
Payable CuEq	Mlb	1,043	-	-	90	96	91	85	72	83	52	67	68	62	66	66	56	24	23	19	19	1

Table 22-3: Cashflow Summary per Year

Description	Units	Total/Avg.	Y-2	Y-1	Y1	Y2	Y3	Y4	Y5	Y6	Y7	Y8	Y9	Y10	Y11	Y12	Y13	Y14	Y15	Y16	Y17	Y18
Gross Revenue – Cu	\$M	4,199.0	-	-	361.0	388.8	336.7	314.5	305.2	384.0	257.6	321.2	297.4	188.7	232.7	248.9	238.2	92.2	88.7	69.1	69.1	5.1
Gross Revenue – Au	\$M	1,657.0	-	-	139.6	151.1	174.7	161.0	100.9	90.2	41.0	62.3	85.3	155.0	136.1	123.7	77.9	42.3	41.3	35.9	35.9	2.7
Gross Revenue – Ag	\$M	81.3	-	-	9.9	9.3	6.4	7.2	5.8	3.7	3.9	5.4	5.2	4.5	5.2	4.1	1.6	2.5	2.5	2.0	2.0	0.2
Gross Revenue – Total	\$M	5,937.3	-	-	510.5	549.1	517.8	482.8	411.9	477.8	302.6	389.0	387.8	348.2	373.9	376.7	317.7	137.1	132.4	107.0	107.0	7.9
Transportation cost	\$M	286.2	-	-	24.6	26.5	23.0	21.4	20.8	26.2	17.6	21.9	20.3	12.9	15.9	17.0	16.2	6.3	6.0	4.7	4.7	0.3
Treatment Charge	\$M	92.2	-	-	7.9	8.5	7.4	6.9	6.7	8.4	5.7	7.1	6.5	4.1	5.1	5.5	5.2	2.0	1.9	1.5	1.5	0.1
Refining Charge – Cu	\$M	38.6	-	-	3.3	3.6	3.1	2.9	2.8	3.5	2.4	3.0	2.7	1.7	2.1	2.3	2.2	0.8	0.8	0.6	0.6	0.0
Refining Charge – Au	\$M	2.8	-	-	0.2	0.3	0.3	0.3	0.2	0.2	0.1	0.1	0.1	0.3	0.2	0.2	0.1	0.1	0.1	0.1	0.1	0.0
Refining Charge – Ag	\$M	1.4	-	-	0.2	0.2	0.1	0.1	0.1	0.1	0.1	0.1	0.1	0.1	0.1	0.1	0.0	0.0	0.0	0.0	0.0	0.0
Total Net Revenue	\$M	5,516.2	-	-	474.3	510.1	483.9	451.2	381.3	439.5	276.8	356.9	358.1	329.1	350.5	351.7	293.9	127.8	123.5	100.1	100.1	7.4
Royalty	\$M	55.2	-	-	4.7	5.1	4.8	4.5	3.8	4.4	2.8	3.6	3.6	3.3	3.5	3.5	2.9	1.3	1.2	1.0	1.0	0.1
Capital Cost – Initial	\$M	617.3	25.7	591.6	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
Capital Cost – Sustaining	\$M	390.9	-	-	31.7	36.0	17.2	40.6	50.6	136.6	22.5	7.4	33.0	14.5	-	0.9	-	-	-	-	-	-
Capital Cost – Closure	\$M	40.0	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	40.0
Capital Cost - Total	\$M	1,048.2	25.7	591.6	31.7	36.0	17.2	40.6	50.6	136.6	22.5	7.4	33.0	14.5	-	0.9	-	-	-	-	-	40.0
Operating Cost – Mining	\$M	1,594.7	-	-	103.4	131.6	132.3	137.3	132.9	137.3	128.9	128.1	120.9	129.2	124.8	105.2	69.9	3.2	3.2	3.2	3.2	0.3
Operating Cost – Processing	\$M	819.9	-	-	40.5	48.6	48.5	48.5	48.5	48.6	48.5	48.5	48.5	48.6	48.5	48.5	48.5	48.6	48.5	48.5	48.5	3.6
Operating Cost – Waste and Water Mgmt	\$M	6.5	-	-	0.5	0.5	0.5	0.5	0.5	0.5	0.5	0.5	0.5	0.5	0.5	0.5	0.5	-	-	-	-	-
Operating Cost – G&A	\$M	244.0	-	-	12.0	14.5	14.4	14.4	14.4	14.5	14.4	14.4	14.4	14.5	14.4	14.4	14.4	14.5	14.4	14.4	14.4	1.1
Operating Cost – Total	\$M	2,665.0	-	-	156.5	195.2	195.7	200.6	196.3	200.8	192.2	191.5	184.2	192.7	188.2	168.6	133.3	66.2	66.0	66.0	66.0	4.9
Changes in Working Capital	\$M	0.0	-	2.7	22.0	-1.9	-0.1	2.1	-3.4	-1.9	-0.3	0.8	0.2	0.3	2.8	-2.4	-13.2	-0.0	-0.5	-	-	-7.3
Taxes – Federal Corporate Income Tax	\$M	216.4	-	-	17.0	21.5	23.3	20.3	13.3	21.6	1.4	11.6	12.0	9.3	14.0	17.9	16.4	5.4	5.4	2.8	3.1	0.2
Taxes – Quebec Corporate Income Tax	\$M	151.7	-	-	3.2	13.6	16.9	15.2	10.1	16.5	1.1	8.9	9.2	7.1	10.7	13.7	12.5	4.1	4.1	2.2	2.4	0.2
Taxes – Quebec Mining Tax	\$M	271.4	-	-	24.7	25.8	29.4	23.3	15.3	28.7	5.5	9.7	12.8	9.9	18.3	24.3	22.4	6.8	6.8	3.6	4.0	0.3
Taxes – Quebec Carbon Tax	\$M	101.1	-	0.4	6.1	8.4	8.4	9.0	8.6	9.0	8.3	8.1	7.5	8.5	8.1	6.4	3.6	0.2	0.2	0.2	0.2	0.0
Taxes – Investment Tax Credit	\$M	-149.6	-	-7.1	-142.5	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
Pre-Tax Unlevered Free Cashflow	\$M	1,747.8	-26	-594	259	276	266	203	134	100	60	154	137	118	156	181	171	60	57	33	33	-30
Cumulative	\$M	-	-26	-620	-361	-85	181	385	519	618	678	832	969	1,087	1,243	1,424	1,595	1,655	1,712	1,745	1,778	1,748
Post-Tax Unlevered Free Cashflow	\$M	1,156.8	-26	-588	351	206	188	136	87	24	43	115	96	84	105	119	116	44	40	24	23	-31
Cumulative	\$M	-	-26	-613	-262	-56	132	268	355	379	422	537	633	716	821	940	1,056	1,100	1,140	1,164	1,188	1,157

22.5 Sensitivity Analysis

The project financial performance is most sensitive to commodity prices, exchange rate, head grade, and metallurgical recovery. It is significantly less sensitive to capital costs and operating costs. Tables 22-4 to 22-9 summarize the sensitivity of the main valuation metrics to the exchange rate, commodity prices, head grade, metallurgical recovery, operating costs, and capital costs, respectively. Tables 22-8 and 22-9 summarize the same sensitivities, but for the after-tax internal rate of return and the after-tax payback period, respectively.

Table 22-4: Exchange Rate Sensitivity Analysis

Variation in Parameter	Pre-Tax NPV (8%) (\$M)	Pre-Tax IRR (%)	Pre-Tax Payback Period (Years)	Post-Tax NPV (8%) (\$M)	Post-Tax IRR (%)	Post-Tax Payback Period (Years)
-50%	-743.3	0.0	-	-705.8	0.0	-
-40%	-436.0	0.0	-	-401.3	0.0	-
-30%	-128.8	2.8	11.9	-111.8	2.3	12.4
-20%	178.5	14.2	4.1	117.8	13.1	3.5
-10%	485.8	23.6	2.7	316.3	20.7	2.6
0%	793.0	32.1	2.3	505.2	27.2	2.3
10%	1,100.3	40.0	2.0	692.1	33.2	2.0
20%	1,407.6	47.5	1.7	877.1	38.7	1.8
30%	1,714.9	54.7	1.5	1,061.3	43.9	1.6
40%	2,022.1	61.7	1.3	1,243.6	48.8	1.4
50%	2,329.4	68.4	1.2	1,425.0	53.6	1.3

Table 22-5: Commodity Pricing Sensitivity Analysis

Variation in Parameter	Pre-Tax NPV (8%) (\$M)	Pre-Tax IRR (%)	Pre-Tax Payback Period (Years)	Post-Tax NPV (8%) (\$M)	Post-Tax IRR (%)	Post-Tax Payback Period (Years)
-50%	-861.0	0.0	-	-821.8	0.0	-
-40%	-530.2	0.0	-	-494.9	0.0	-
-30%	-199.4	0.0	-	-173.9	0.0	-
-20%	131.4	12.6	4.4	86.0	11.8	3.8
-10%	462.2	22.9	2.7	301.7	20.2	2.6
0%	793.0	32.1	2.3	505.2	27.2	2.3
10%	1,123.9	40.6	1.9	706.1	33.6	2.0
20%	1,454.7	48.6	1.7	904.8	39.5	1.7
30%	1,785.5	56.3	1.4	1,101.8	45.0	1.5
40%	2,116.3	63.7	1.3	1,296.8	50.2	1.4
50%	2,447.1	70.9	1.2	1,490.7	55.2	1.3

Table 22-6: Head Grade Sensitivity Analysis

Variation in Parameter	Pre-Tax NPV (8%) (\$M)	Pre-Tax IRR (%)	Pre-Tax Payback Period (Years)	Post-Tax NPV (8%) (\$M)	Post-Tax IRR (%)	Post-Tax Payback Period (Years)
-50%	-743.3	0.0	-	-705.8	0.0	-
-40%	-436.0	0.0	-	-401.3	0.0	-
-30%	-128.8	2.8	11.9	-111.8	2.3	12.4
-20%	178.5	14.2	4.1	117.8	13.1	3.5
-10%	485.8	23.6	2.7	316.3	20.7	2.6
0%	793.0	32.1	2.3	505.2	27.2	2.3
10%	1,100.3	40.0	2.0	692.1	33.2	2.0
20%	1,407.6	47.5	1.7	877.1	38.7	1.8
30%	1,714.9	54.7	1.5	1,061.3	43.9	1.6
40%	2,022.1	61.7	1.3	1,243.6	48.8	1.4
50%	2,329.4	68.4	1.2	1,425.0	53.6	1.3

Table 22-7: Metallurgical Recovery Sensitivity Analysis

Variation in Parameter	Pre-Tax NPV (8%) (\$M)	Pre-Tax IRR (%)	Pre-Tax Payback Period (Years)	Post-Tax NPV (8%) (\$M)	Post-Tax IRR (%)	Post-Tax Payback Period (Years)
-20%	136.2	12.8	4.4	89.2	11.9	3.8
-10%	464.6	23.0	2.7	303.2	20.2	2.6
0%	793.0	32.1	2.3	505.2	27.2	2.3
10%	1,121.5	40.5	1.9	704.6	33.5	2.0
20%	1,449.9	48.5	1.7	902.0	39.4	1.8

Table 22-8: Operating Costs Sensitivity Analysis

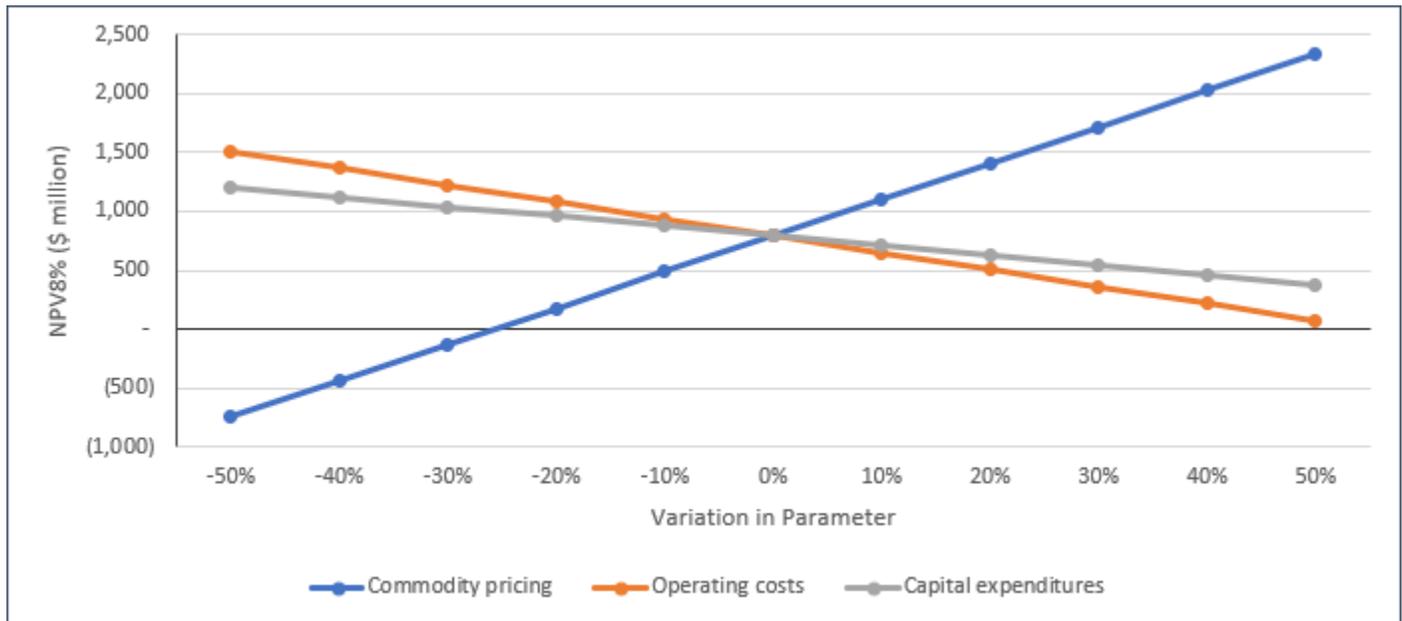
Variation in Parameter	Pre-Tax NPV (8%) (\$M)	Pre-Tax IRR (%)	Pre-Tax Payback Period (Years)	Post-Tax NPV (8%) (\$M)	Post-Tax IRR (%)	Post-Tax Payback Period (Years)
-50%	1,510.8	47.6	1.8	926.2	38.8	1.8
-40%	1,367.2	44.7	1.8	844.2	36.7	1.9
-30%	1,223.7	41.7	1.9	761.0	34.5	2.0
-20%	1,080.1	38.6	2.0	676.6	32.2	2.1
-10%	936.6	35.4	2.2	591.6	29.8	2.2
0%	793.0	32.1	2.3	505.2	27.2	2.3
10%	649.5	28.5	2.4	418.3	24.5	2.4
20%	505.9	24.8	2.6	329.6	21.5	2.5
30%	362.4	20.7	2.7	231.1	18.0	2.6
40%	218.8	16.3	3.2	127.2	14.0	3.1
50%	75.3	11.1	3.9	13.7	8.7	5.0

Table 22-9: Capital Costs Sensitivity Analysis

Variation in Parameter	Pre-Tax NPV (8%) (\$M)	Pre-Tax IRR (%)	Pre-Tax Payback Period (Years)	Post-Tax NPV (8%) (\$M)	Post-Tax IRR (%)	Post-Tax Payback Period (Years)
-50%	1,205.8	74.6	1.1	787.8	66.5	0.9
-40%	1,123.2	61.2	1.3	731.9	53.8	1.1
-30%	1,040.7	51.2	1.6	675.6	44.4	1.4
-20%	958.1	43.4	1.8	619.1	37.3	1.7
-10%	875.6	37.2	2.1	562.3	31.7	2.0
0%	793.0	32.1	2.3	505.2	27.2	2.3
10%	710.5	27.8	2.5	447.8	23.5	2.5
20%	628.0	24.2	2.7	390.2	20.4	2.7
30%	545.4	21.1	3.1	332.5	17.7	3.2
40%	462.9	18.4	3.4	274.5	15.5	3.6
50%	380.3	16.0	3.9	216.1	13.5	4.2

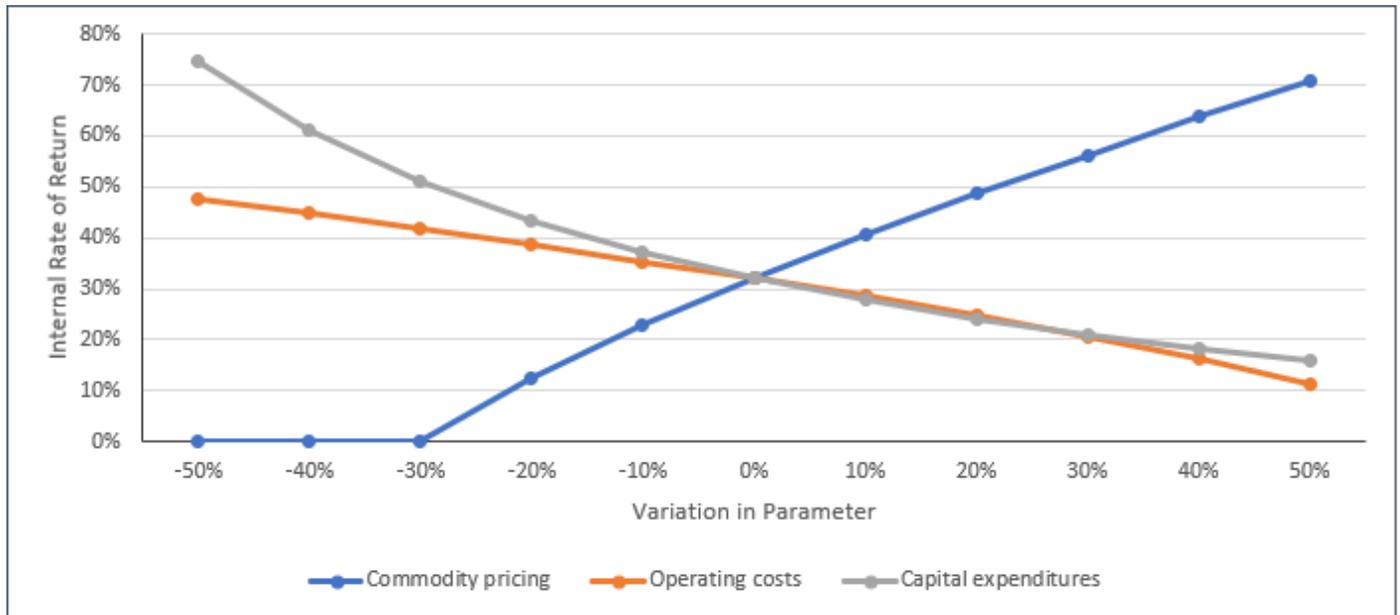
Figures 22-2 to 22-4 illustrate the sensitivities on the pre-tax NPV 8%, IRR, and payback period, respectively. Figures 22-5 to Figure 22-7 illustrate the sensitivities on the post-tax NPV 8%, IRR, and payback period, respectively.

Figure 22-2: Pre-Tax Net Present Value 8% Sensitivity



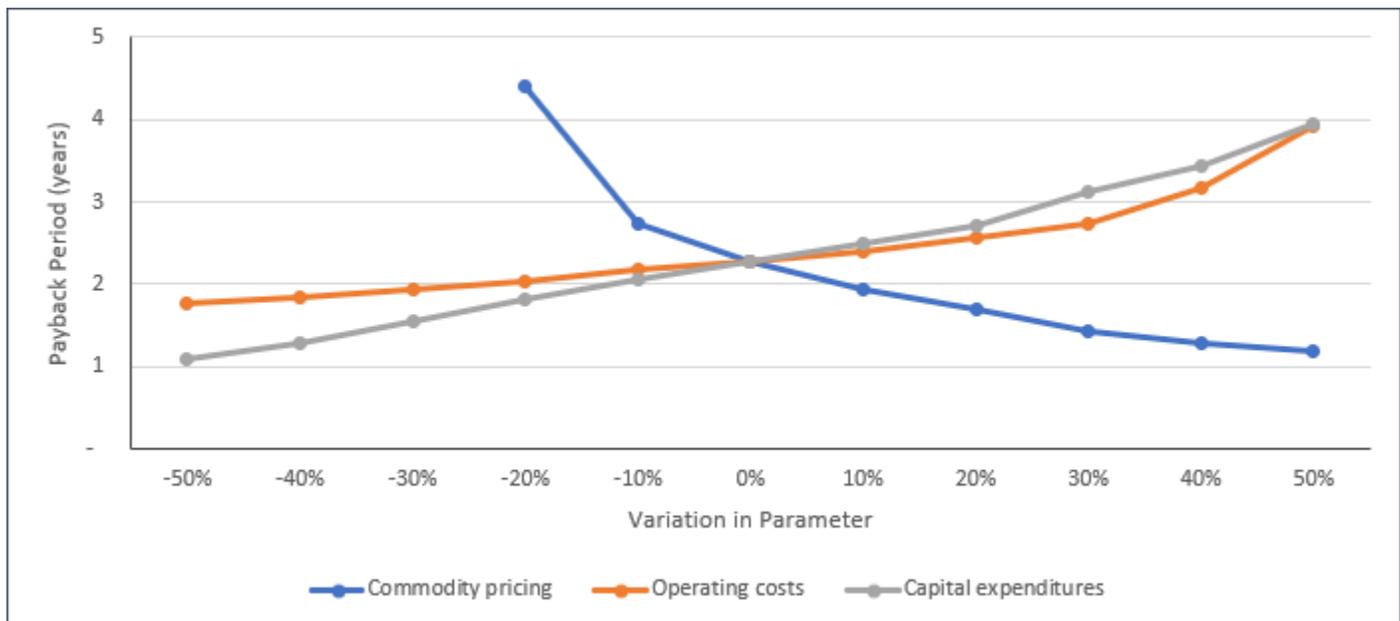
Source: Evomine, 2025.

Figure 22-3: Pre-Tax Internal Rate of Return Sensitivity



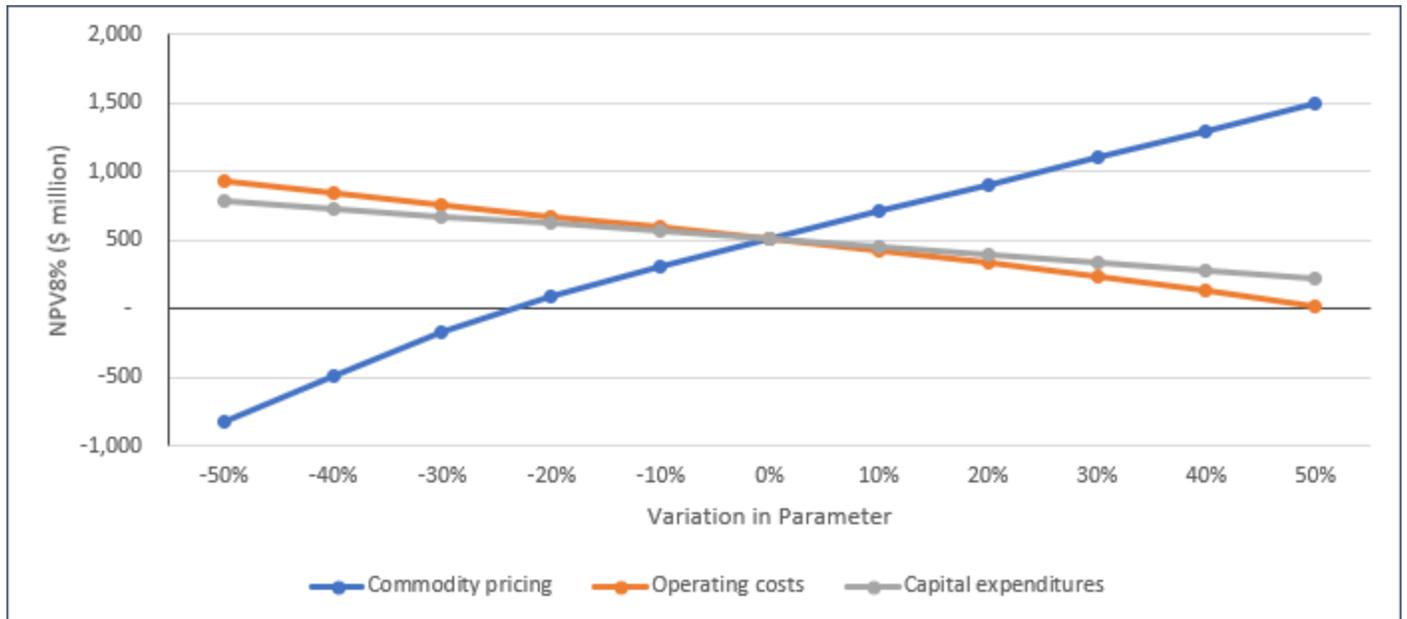
Source: Evomine, 2025.

Figure 22-4: Pre-Tax Net Payback Period Sensitivity



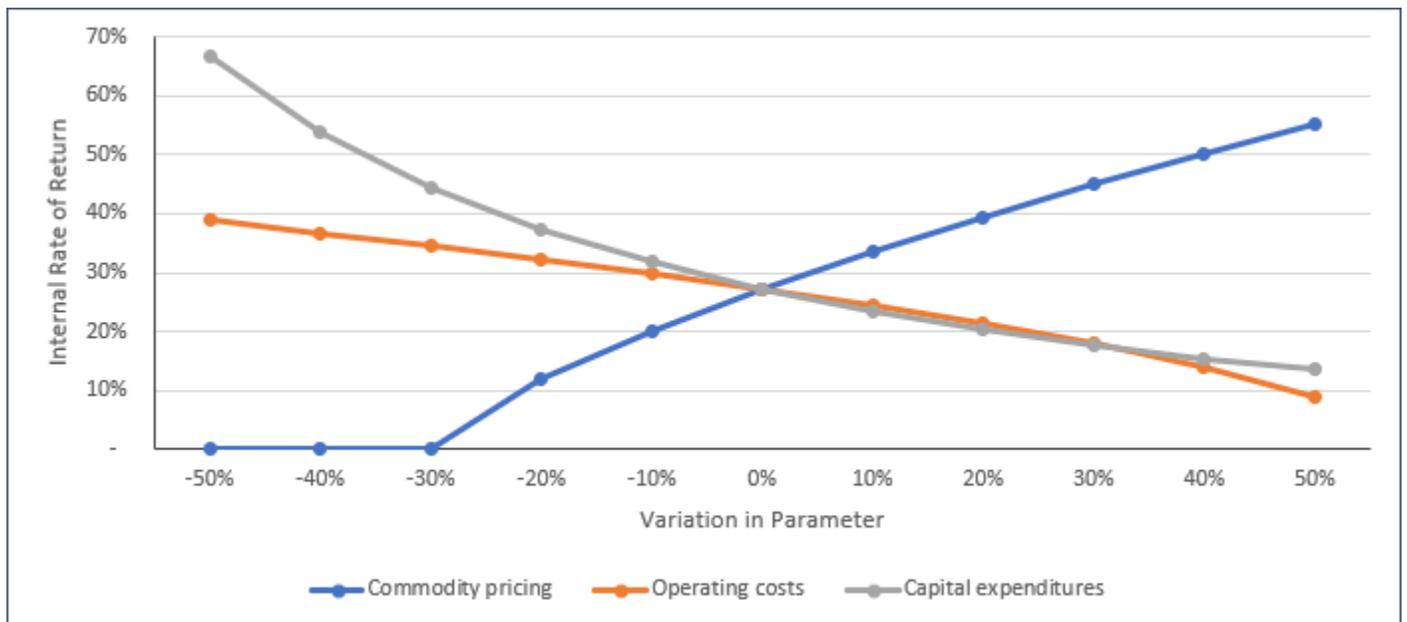
Source: Evomine, 2025.

Figure 22-5: Post-Tax Net Present Value 8% Sensitivity



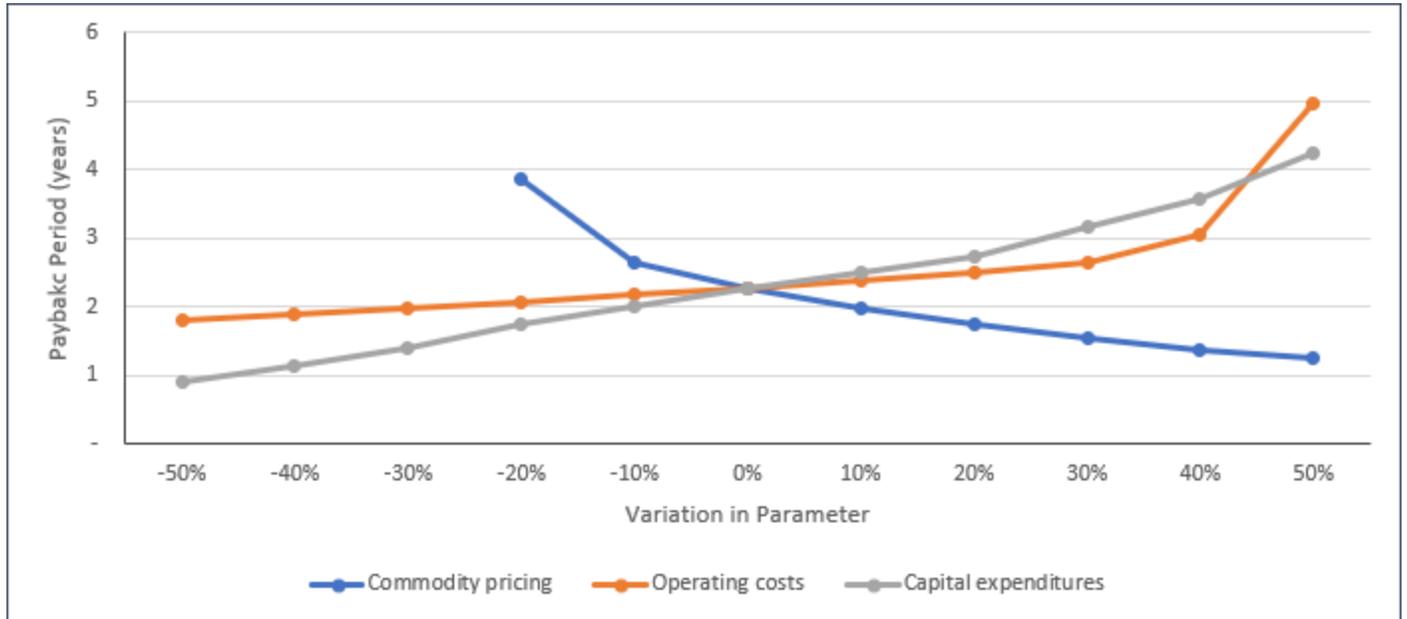
Source: Evomine, 2025.

Figure 22-6: Post-Tax Internal Rate of Return Sensitivity



Source: Evomine, 2025.

Figure 22-7: Post-Tax Payback Period Sensitivity



Source: Evomine, 2025.

23 ADJACENT PROPERTIES

There are no significant projects in the close vicinity of the project.

24 OTHER RELEVANT DATA AND INFORMATION

This section is not applicable to the technical report

25 INTERPRETATION AND CONCLUSIONS

25.1 Introduction

The QPs note the following interpretations and conclusions in their respective areas of expertise, based on the review of data available for this report.

25.2 Mineral Tenure, Surface Rights, Water Rights, Royalties and Agreements

Information from XXIX supports that the mining tenure held is valid and is sufficient to support declaration of mineral resources and mineral reserves.

25.3 Geology and Mineralization

The understanding of the regional geology, lithological, and structural controls of the mineralization at Opémiska are sufficient to support the mineral resource estimate in this report.

25.4 Exploration and Drilling

XXIX and previous owners have completed geophysical surveys and diamond drilling programs for the project between 1929 and 2025. In total, 21,918 surface and underground drill holes for 1,525,074 m have been completed and recorded for the project.

25.5 Analytical Data Collection in Support of Mineral Resource Estimation

The sample preparation, analytical procedures, and security of the samples during these procedures followed industry best practices but could be improved, mainly by inserting more blanks, more CRMs, and adding a field duplicate program. Sufficient efforts were made to identify items that were out of specification.

The QA/QC data indicate that the overall assay results of the issuer's drill program are valid and can be relied upon for the purpose of this report.

It is the QP's opinion that the sample preparation, security and analytical procedures are adequate and follow best practices.

25.6 Metallurgical Testwork

The quantity of metallurgical testing is suitable for this level of study and provides a comprehensive assessment of the metallurgical response of the resource materials to the developed process. The recovery model used for the financial analysis is based on a reasonable interpretation of the metallurgical testing, as well as available historical operation data.

25.7 Mineral Resource Estimate

The Opémiska mineral resource estimate in this report was prepared by Pierre-Luc Richard (P.Geo.) of PLR, with contributions from Stephen Coates, P.Eng., of Evomine for value cut-off, open pit, and optimization solids, and Christian Laroche, P.Eng., from Synectiq, for metallurgical parameters.

Mineral resources are not mineral reserves as they do not have demonstrated economic viability. The resources in the estimate are categorized as inferred and indicated based on data density, search ellipse criteria, drill hole density, specific interpolation parameters, geological continuity, and grade continuity above the cut-off grade. The effective date of the estimate is May 30, 2025, based on the compilation status and cut-off grade parameters.

The QP considers the estimate reliable and based on quality data, reasonable hypotheses, and parameters that follow CIM Definition Standards. After completing the mineral resource estimate and performing a detailed review of all pertinent information, the QP reached the following conclusions:

- Using a cut-off grade of 0.15% CuEq, the indicated mineral resources amount to 69.7 Mt grading 1.12% CuEq (0.84% Cu, 0.31 g/t Au, and 1.82 g/t Ag).
- Using a cut-off grade of 0.15% CuEq, the inferred mineral resources amount to 80.6 Mt grading 0.42% CuEq (0.28% Cu, 0.17 g/t Au, and 0.62 g/t Ag).

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

25.8 Mining Methods

The project is planned as a conventional truck-and-shovel open pit mining operation. The nominal processing rate is set at 12,500 t/d over a 17-year mine life, with an average strip ratio of 3.7 to 1. Two ultimate pits—the Springer and Perry pits—will be mined over the life of mine, with interim pits designed within these ultimate pits to optimize the mineralized material grade and strip ratio extraction profile. Mined physical quantities represent 77.2 Mt of mineralized material, 270.7 Mt of waste, and 15.0 Mt of overburden segregated by a block model reblocked to 5 m x 5 m x 5 m dimensions to adequately consider selectivity and associated mining dilution for the envisioned mining equipment.

Four mining phases are planned and detailed as follows:

- Phase 1 – Springer and Perry starter pits
- Phase 2 – Intermediate pushback in Springer
- Phase 3 – Depletion of Perry pit
- Phase 4 – Depletion of Springer pit.

The 17-year life of mine incorporates 13 years of direct mill feed from open pit operations and four years of stockpile rehandling. The mining rate is expected to peak at 116,000 t/d and average 76,000 t/d over the 13 years of mining. The open pit operation has also been optimized to push any impact to the neighbouring town of Chapais to the end of

Phase 3 and beginning of Phase 4. Also considered are areas dedicated to overburden, waste rock, and mineralized material stockpiling.

25.9 Recovery Methods

The plant is designed to process material at a nominal rate of 4.6 Mt/a, operating two shifts per day, 365 days per year, with an overall plant availability of 92%. The project flowsheet was developed based on 2023 metallurgical testwork and historical operating data, and uses conventional unit operations for copper concentration plants. There are no significant elements of technological innovation in this project.

25.10 Infrastructure

25.10.1 Infrastructure

The main site infrastructure includes open pit mining, mineralized material stockpiles, a CDSF, water management structures, administrative buildings and offices, access roads connecting the pits to the main plant site and highways, and a truckshop. The main plant access area will be gated for security, and the access road from the highway will be expanded and upgraded. Additionally, the local provincial highway will be relocated to accommodate the ultimate mining pits. The strategic positioning of all the infrastructure facilities takes into account various factors such as site boundaries, operational efficiency, and accessibility.

25.10.2 Co-Disposal Facility and Water Management Infrastructure

This PEA suggests that building and operating a co-disposal facility is technically feasible. Both streams of mine waste will be placed within the same footprint, minimizing environmental liability. The contact water management scheme minimizes the use of pumps and allows contact water to be managed in a series of water ponds located downstream of the mining operation and Chapais. The selected final discharge location is a stream located northeast of the site that eventually drains into Opémiska lake.

25.11 Markets and Contracts

The QP considers the marketing and pricing assumptions reasonable for use in PEA-level cash flow analyses. For the purposes of this study, it is assumed that the concentrate will be transported from the mine location by road to a smelter located in the Abitibi Region of Quebec. Considering that the Opémiska is a previously producing operation and that a copper concentrate will be produced, the product is expected to be marketable.

25.12 Environmental, Permitting and Social Considerations

The construction, production, and closure of a mine is regulated by several acts and regulations at three different levels: federal, provincial, and municipal. The project will be subject to the federal and provincial impact assessment procedure. It is recommended to start baseline studies and consultations with stakeholders early in the development of the project.

The environmental permitting process requires an understanding of the physical, biological, and social environments. It includes an evaluation of the potential impacts of the mining project and proposes mitigation measures. Environmental baseline studies will start in the next phase of the project.

The project is in the municipality of Chapais, part of the administrative territory of Eeyou Istchee James Bay. Information and consultation meetings have been initiated by XXIX with the town of Chapais, and will start with the First Nation authorities, other stakeholders, and land users.

25.13 Capital Cost Estimate

The capital cost estimate developed in this PEA was prepared to a Class 5 estimate standard with an accuracy of $\pm 50\%$ as defined by the Association for the Advancement of Cost Engineering International (AACE International). Generally, engineering performed to date is between 1% to 5% of full project definition.

The total capital cost estimate for the project includes initial capital costs and sustaining capital costs and is estimated at \$1,048 million. The capital costs are summarized in Section 21.

25.14 Operating Cost Estimate

Operating costs are summarized in Section 21. These include mining, processing, waste and water management, and general and administration (G&A) costs. Operating costs were estimated at \$2,665 million over the life of mine, which represents \$34.52/tonne processed.

25.15 Economic Analysis

The PEA is preliminary in nature. 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 PEA would be realized.

The economic and financial evaluation presented in this technical report utilizes a discounted cash flow method, both on a pre-tax and after-tax basis. The metal pricing used in the evaluation was determined in Section 19. The financial model provides results in terms of NPV, payback period, and IRR for the project. The economic analysis is conducted in real terms, without considering inflation factors, using Q4 2025 Canadian dollars. The analysis does not take into account project financing.

The economic analysis was performed assuming an 8% discount rate. The pre-tax NPV discounted at 8% is \$793.0 million; the internal rate of return IRR is 32.1%, and payback period is 2.3 years. On a post-tax basis, the NPV discounted at 8% is \$505.2 million; the IRR is 27.2%, and the payback period is 2.3 years.

25.16 Risks and Opportunities

25.16.1 Risks

25.16.1.1 Mineral Resource

It is possible that some grade smearing occurs in the current stockwork model near localized higher-grade intervals. Capping is currently significantly low and also mitigate high-grade smearing in the stockwork zones. An improved 3D model would help mitigate such risk.

The underground historical workings need additional work to improve its accuracy.

25.16.1.2 Metallurgy and Recovery Methods

The flowsheet was designed with limited metallurgical testing available, potentially impacting the flowsheet design, equipment sizing, recovery, concentrate saleability and therefore overall project economics. Specific risks include:

- Equipment selection for the PEA was based on available benchmarks. Bond work index, specific grinding index, and solid-liquid separation testwork is required to confirm equipment selection throughout the flowsheet, and may result in additional capital cost if testwork results and therefore equipment sizing differ significantly from the benchmarks.
- The precious metals recovery circuits and expected performance were based on limited open circuit testing. Future testwork could show reduced recovery, impacting the overall recovery metals for the project.

Neither concentrate characterization data nor agreements with potential off-takers were available during the PEA. Concentrate payability, and therefore project revenue, could be reduced due to potential deleterious elements not identified at this stage.

25.16.1.3 Mining Methods

The main project risks are listed below:

- differences between current geological model and actual mineralization
- worse hydrogeological conditions than expected
- worse geotechnical conditions than expected
- limited labour availability
- limited goods and services availability
- delays in obtaining permits
- downturn in commodity pricing

- limited project financing ability
- inability to obtain authorization for town relocation.

25.16.1.4 Infrastructure

25.16.1.4.1 Power Availability

The process plant demands significant electrical loads, and there is a risk that Hydro-Québec may not have sufficient generation or transmission capacity available to supply the required load. This can lead to plant operation and production delays and issues. This risk could be mitigated by starting early engagement with Hydro-Québec and confirming load availability with them for the project.

25.16.1.4.2 Co-Disposal Facility and Water Management Infrastructure

The location of the CDSF was selected based on its proximity to the mill and favourable topography. However, there is currently insufficient geotechnical data to confirm the adequacy of the foundation at this site.

While XXIX has proposed using filtered tailings and a co-disposition strategy with waste rock, it should be noted that local communities previously experienced a tailings dam failure in 2008 at the historical TSF facility. This past experience may heighten concerns regarding the use of the historical TSF as the foundation for the main overburden stockpile and the management of contact water.

The absence of comprehensive geotechnical and hydrogeological studies could impact the configuration of the stockpiles and the water retaining embankments. There is consequently a risk for non-optimum configurations and locations, and potential quantity and cost variations that could impact the operational and financial aspects of the project.

Lack of site-specific climatic data may have led to underestimating the quantity of surface water runoff, consequently the water ponds may have been underestimated

25.16.1.5 Environmental, Permitting, Social or Community Impact

General risks to the forecast information are as follows:

- unanticipated environmental risks
- geochemical or hydrogeological predictions differing from those used in mining plans
- risks to maintaining the social licence to operate
- delays in permitting
- unforeseen rehabilitation and restoration costs.

25.16.2 Opportunities

25.16.2.1 Metallurgy and Recovery Methods

There may be opportunities to optimize the flowsheet once additional metallurgical testing is completed. Future programs should include the following engineering studies or testwork programs:

- locked cycle testing to optimize concentrate grades and metal recoveries
- reagent dosage optimization
- gravity recovery testing to quantify the potential recovery of coarse gold with gravity concentration and impact on final concentrate saleability.
- regrind of first cleaner versus rougher concentrate to reduce regrind mill sizing. Preliminary open circuit testing indicated equivalent metal recovery to cleaner 1 concentrate without regrind.

25.16.2.2 Co-Disposal Facility and Water Management Infrastructure

- The current historical tailings storage facility (TSF) has not been fully reclaimed. XXIX could consider assuming full environmental liability for this historical TSF to enhance the social acceptability of the project. This approach may positively contribute to community relations and the project's social license to operate.
- There is potential to substitute the sand layer above the liner with tailings, provided that their geotechnical and geochemical properties are deemed suitable. This substitution could lead to a reduction in sustaining capital expenditure.
- The CDSF is currently designed to store all waste rock generated by the project. However, if the geochemical assessment shows that certain types of waste rock are both NPAG and non-metal leaching (NML), the CDSF design could be optimized to reduce its size and related costs.

25.16.2.3 Mining Methods

The main project opportunities are as follows:

- extension or expansion of the project through the inclusion of mineral resources that are currently excluded in the mine plan
- exploration upside
- increase of commodity pricing.

25.16.2.4 Infrastructure

There is an opportunity to refine the earthworks quantities in the next project phase when the results from more boreholes in the process plant area become available.

25.16.2.5 Environmental, Permitting, Social or Community Impact

General opportunities related to early environmental studies and public consultations in the development of the mining project are as follows:

- consideration of the biological and social environments in the mine design
- optimization of the mine footprint based on environmental studies
- implementation of appropriate mitigation measures to reduce environmental and social impacts
- consideration of the stakeholders' concerns in the project development (e.g., road deviation, relocation)
- local economic benefits and support the economic development of Chapais.

26 RECOMMENDATIONS

26.1 Introduction

It is recommended to continue developing the project through the prefeasibility study (PFS) stage based on the current mineral resource estimate. Table 26-1 summarizes the estimated cost of recommended future work on the project.

Table 26-1: Recommended Work Program

Area	Program Component	Estimated Cost (\$M)
Geology and Mineral Resource	Update modelling of historical underground workings, additional drilling on the Saddle zone between the two pits, infill drilling to upgrade inferred resources [as required post update of underground workings model] and drilling program (5,000 m) on Cooke deposit.	3.5
Mineral Processing and Metallurgical Testing	JK Axb, Abrasion, Bond work indices, concentrate regarding signature plot, gravity amenability testing, open circuit flotation, locked-cycle testing, dynamic thickening and pressure filtration, concentrate characterization, material flow properties, slurry rheology	1.0
Mining Methods	Geotechnical data collection and study, hydrogeological testing and analysis, prefeasibility study and associated trade-off studies	2.25
Co-Disposal Storage Facility and Water Management Infrastructure	PFS Engineering, including: CDSF foundation material stratigraphy, new tailings properties, geotechnical investigations and data collection, environmental geochemistry characteristics, develop a conceptual hydrogeological model, develop a detailed overall site water balance	0.70
Environmental, Permitting, Social or Community Impact	Stakeholder consultation, baseline studies, geochemical characterization, federal and provincial environmental assessments	2.0
Total		9.45

26.2 Geology and Mineral Resources

The following recommendations are noted regarding the geology and mineral resource:

- Drilling is warranted on the Cooke deposit, and a 5,000 m drilling program is recommended. Following a drilling program, the mineral resource should be estimated to evaluate its potential.
- Additional drilling in the saddle zone between the Springer and Perry pits is recommended as well.
- Improvement to the spatial surveying and modelling of the underground historical workings is recommended to improve the block-level accuracy of the resource.

The estimated cost of the recommended future work is C\$3.5 million.

26.3 Metallurgical Testwork

It is recommended to complete the metallurgical testwork program prior to the PFS to confirm PEA design assumptions. This testwork program should be conducted on a representative production composite with variability testing on individual domain samples. The proposed testwork program, summarized in Table 26-2 would cost an estimated C\$1.0 million.

Table 26-2: Metallurgical Testwork Program

Activity	Purpose
JK Axb, Abrasion, Bond Crushing, Ball and Rod Mill Work Indices Concentrate Regrind Signature Plot	Provide design assumptions for comminution equipment sizing
Gravity Amenable Testing (GAT)	Testing to determine the amenability of gold to gravity recovery and the impact of gravity gold on overall metal recovery and concentrate quality
Open Circuit Flotation	Optimization of reagent regime, regrind size and retention times
Locked Cycle Testing	Confirmation of open circuit indication, effect of circulating loads on circuit performance
Dynamic Thickening and Pressure Filtration (Concentrate and Tailings)	Provide design assumptions for thickener and filter sizing
Concentrate Characterization	Confirm transport moisture limit (TML) and deleterious elements content
Material Flow Properties	Provide design inputs for material handling systems design
Slurry Rheology	Provide design inputs for thickened slurry pumping

26.4 Mining Methods

Several studies should be performed to further optimize and define the mine designs, mine schedule, physical quantity estimates for open pit areas, and projected extraction costs. The following work is recommended:

- A geotechnical data collection program to better define soil and rock mass characteristics and an associated prefeasibility level geotechnical study to better understand potential pit slope configurations.
- Hydrogeological testing and analysis to validate potential water inflows and associated water quality in the projected open pit mine.

A prefeasibility study, including relevant trade-off studies, should be conducted to properly assess strategic alternatives. Detailed costing should be supported by appropriate data sources

26.5 Co-Disposal Facility and Water Management Infrastructure

The following additional information and studies are required to progress prefeasibility engineering, address project design refinements, and confirm the assumptions made as part of the CDSF PEA design and related water management infrastructure.

Seepage and stability analyses should be performed during the PFS engineering phase with the intent to demonstrate that the facility design complies with industry standards, current practice recommendations, and applicable regulations.

Complementary field and laboratory investigations should be performed with the intent to:

- define the CDSF foundation material stratigraphy and material properties and characteristics
- define the new tailings properties and characteristics
- define the historical tailings properties and characteristics, as benchmark for the new tailings
- define and understand the prevailing hydrogeological context of the property, especially beneath the CDSF footprint, historical TSF, and at the open pits
- gather geotechnical data on the historical TSF
- gather information about potential borrow pit for construction materials
- define the environmental geochemistry characteristics and properties of both tailings and waste rock
- collect climatic data and calculate climate extreme events for design input.

The next engineering phase will be performed with the intent to:

- assess the geotechnical and hydrogeological behaviour of the CDSF and water-retaining dams
- confirm that the historical TSF can behave as the main overburden stockpile foundation
- develop a conceptual hydrogeological model of the site
- establish the quantity of groundwater to be dealt with during the open pit dewatering of mining phases
- assess the likelihood of dusting and surface erosion during dry periods and heavy rainfall events
- assess the likelihood of geomembrane perforation
- develop a detailed overall site water balance
- develop a detailed CDSF reclamation cover
- develop a detail material take off estimation.

26.6 Environmental, Permitting, Social or Community Impact

Environmental work is recommended to support the project as it advances through economic and development studies. By submitting the project to the federal and provincial impact assessment procedure, the main following studies or activities should be carried out during the next phase of the project. These studies or activities will also help to ensure that project development limits impacts to the receiving environment.

The following activities should be undertaken for the next phase of the project:

- continue information and consultation activities with stakeholders, including First Nations, to ensure that their expectations and concerns are considered throughout the development of the project
- document the land uses, including the land uses for traditional purposes by First Nations
- start the baseline studies on the physical, biological, and social environments
- start the hydrogeological study and its modelling
- continue the geochemical characterization of waste rock, mineralized rock, tailings, and overburden
- initiate federal and provincial environmental assessment procedures and involve government authorities to ensure that their expectations and concerns are considered throughout the development of the project.

The estimated cost of the above work is \$2 million.

27 REFERENCES

- Centre de données sur le patrimoine naturel du Québec (CDPNQ), 2025. Extractions du système de données pour des occurrences fauniques sensibles à la diffusion pour le projet Opémiska. Ministère de l'Environnement, de la Lutte contre les changements climatiques, de la Faune et des Parcs, Gouvernement du Québec, 5 p.
- Compilation of Québec Laws and Regulations (CQLR). Environmental Quality Act, chapter Q-2.
[\[https://www.legisquebec.gouv.qc.ca/en/document/cs/q-2\]](https://www.legisquebec.gouv.qc.ca/en/document/cs/q-2)
- Compilation of Québec Laws and Regulations (CQLR). Mining Act, chapter M-13.1.
[\[https://www.legisquebec.gouv.qc.ca/en/document/cs/m-13.1\]](https://www.legisquebec.gouv.qc.ca/en/document/cs/m-13.1)
- Dimroth, E., Mueller, W., Rocheleau, M., Archer, P., Jutras, M., Piche, M., Simoneau, P., Carignan, J., Chown, E. H., Guha, J., Goulet, N., Allard, G. O., Franconi, A., and Gobeil, A. 1984: Stratigraphie et Évolution du Bassin de Transition entre les Groupes de Roy et d'Opémiska, Région de Chibougamau-Chapais. Stratigraphie des Ensembles Volcano-sédimentaires Archeens de l'Abitibi et des Connaissances, Québec 11. E.R DV83-11, p. 21–33.
- Dubé B., Guha J., 1992. Relationship between northeast-trending regional faults and Archean mesothermal gold-copper mineralization: Cooke mine, Abitibi greenstone belt, Quebec, Canada; *Economic Geology*; volume 87, pages 1525-1540.
- Guha et al., 1988, Structural and Stratigraphic Controls on Magmatic, Volcanogenic, and Shear Zone-Hosted Mineralization in the Chapais-Chibougamau Mining Camp, Northeastern Abitibi, Canada. *Economic Geology* 107(5):963-989
- IAAC, 2025. Impact Assessment Process Overview. Impact Assessment Agency of Canada.
[\[https://www.canada.ca/en/impact-assessment-agency/services/policy-guidance/impact-assessment-process-overview.html\]](https://www.canada.ca/en/impact-assessment-agency/services/policy-guidance/impact-assessment-process-overview.html)
- Leclerc F. Harris L.B. Bédard J.H. Van Breemen O. Goulet N., 2012, Structural and stratigraphic controls on magmatic, volcanogenic, and shear zone-hosted mineralization in the Chapais-Chibougamau mining camp, northeastern Abitibi, Canada: *Economic Geology* , v. 107, p. 963–989.
- MELCC, 2020. Guide de caractérisation des résidus miniers et du minerai. Ministère de l'Environnement et de la Lutte contre les changements climatiques, Gouvernement du Québec, 52 p.
[\[https://www.environnement.gouv.qc.ca/Industriel/secteur-minier/guide-caracterisation-minerai.pdf\]](https://www.environnement.gouv.qc.ca/Industriel/secteur-minier/guide-caracterisation-minerai.pdf)
- MELCCFP, 2025. Directive 019 sur l'industrie minière. Ministère de l'Environnement, de la Lutte contre les changements climatiques, de la Faune et des Parcs, Gouvernement du Québec, 96 p.
[\[https://www.environnement.gouv.qc.ca/milieu_ind/directive019/directive-019-2025.pdf\]](https://www.environnement.gouv.qc.ca/milieu_ind/directive019/directive-019-2025.pdf)
- Minnova, 1991. Division Opémiska – Fermeture et restauration des sites miniers. 113 p.
- MRNF, 2024. Guide de préparation du plan de réaménagement et de restauration des sites miniers au Québec. Ministère des Ressources naturelles et des Forêts, Gouvernement du Québec, 125 p.
[\[https://mrnf.gouv.qc.ca/wp-content/uploads/GM_restauracion_sites_miniers_MERN.pdf\]](https://mrnf.gouv.qc.ca/wp-content/uploads/GM_restauracion_sites_miniers_MERN.pdf)

- MRNF, 2023. Cadre normatif s'appliquant au domaine minier. Ministère des Ressources naturelles et des Forêts, Gouvernement du Québec, 66 p. [<https://mrnf.gouv.qc.ca/wp-content/uploads/cadre-normatif-domaine-minier.pdf>]
- Salmon. B., 2013. Determination of Exploration Potential at the Springer mine – Opémiska property; B. Salmon; Roscoe Postle Associates Inc.; February 2013; Internal Document.
- Salmon. B., 2014. Determination of Exploration Potential at the Perry mine – Opémiska property; B. Salmon; Roscoe Postle Associates Inc.; July 2014; Internal Document
- SGS, 2023. Gold Department of One Composite Sample from the Opémiska Project. Prepared for G Mining Services, Project 19318-01 – Mineralogy Report, 27 July 2023, 109 p.
- SGS, 2023. The Recovery of Copper and Gold from the Opémiska Project. Prepared for G Mining Services, Project 19318-01 – Final Report, 24 November 2023, 396 p.
- Turgeon Consult, 2023. Résultats d'analyses de géochimie environnementale – Projet Opémiska, QC Copper. Prepared for G Services Miniers, 19 October 2023, 94 p.
- Yassa and Puritch, 2024. Mineral Resource Estimate and Technical Report on the Opémiska Copper-Gold Property, 275p.
- Watkins D.H., Riverin G., 1982. Geology of the Opémiska Copper – Gold Deposits at Chapais, Quebec; in Precambrian Sulphide Deposits, H.S. Robinson Memorial Volume, edited by R.W. Hitchinson, C.D. Spence and J.M. Franklin; Geological Association of Canada; Special Paper 25.