Sisson Tungsten-Molybdenum Project

NI 43-101 Technical Report & Feasibility Study Update

New Brunswick, Canada

Effective Date: September 23, 2026

Report Date: October 1, 2026

Prepared for:

Northcliff Resources Ltd
1040 West Georgia Street
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Prepared by:

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Daniel Friedman, P.Eng., Knight Piésold Consulting

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AJ MacDonald, P.Eng., Integrated Sustainability Consultants Ltd

Paul Chamois, P.Geo., SLR Consulting (Canada) Ltd

Philip A. Geusebroek, M.Sc., P.Geo., SLR Consulting (Canada) Ltd

Ausenco

Certificate of Qualified Person
Kevin Murray, P.Eng.

I, Kevin Murray, P.Eng., certify that

1) I am employed as a Principal Process Engineer with Ausenco Engineering Canada ULC, (Ausenco), with an office address of 1050 West Pender, Suite 1200, Vancouver BC, Canada V6E 3S7.
2) This certificate applies to the technical report titled "Sisson Tungsten-Molybdenum Project NI 43-101 Technical Report & Feasibility Study Update, New Brunswick, Canada" (the "Technical Report"), prepared for Northcliff Resources Ltd (the "Company"), with a report date of September 23, 2026 and an effective date of October 1, 2026 (the "Effective Date").
3) I graduated from University of New Brunswick with a Bachelor of Science in Chemical Engineering in 1995.
4) I am a member in good standing of Engineers and Geoscientists British Columbia (No. 32350), Northwest Territories Association of Professional Engineers and Geoscientists (No. L4940) and Association of Professional Engineers and Geoscientists of Saskatchewan (No. 82404).
5) I have practiced my profession continuously for 25 years. I have been directly involved in all levels of engineering studies from preliminary economic assessments (PEAs) to feasibility studies. I have led preliminary test work design, test work analysis and flowsheet development as well involvement in detailed design and commissioning. I have also developed operating cost estimates and contributed to and reviewed capital cost estimates. I have been involved with flotation studies including Skeena's Eskay Creek gold concentrate project and lead-zinc flotation studies including Prairie Creek.
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 fulfill the requirements to be a "Qualified Person" for the purposes of this instrument.
7) I visited the project site on September 18, 2025.
8) I am responsible for Sections 1.1, 1.2, 1.9, 1.13, 1.14.1–1.14.3, 1.14.6, 1.14.7, 1.15, 1.17–1.19, 2.1–2.3, 2.4.4, 2.5–2.7, 3.1, 3.4, 3.5, 12.3, 13, 17, 18.1–18.4, 18.6.2, 18.6.3, 18.6.8, 18.8, 18.9, 18.10, 19, 21.1, 21.2.1, 21.2.2, 21.2.4, 21.2.5–21.2.9, 21.3.1, 21.3.4, 21.3.6, 22, 24, 25.1, 25.5, 25.8, 25.9.1, 25.11–25.13, 25.14.1.2, 25.14.1.6, 25.14.1.7.1, 25.14.1.9, 25.14.2.2, 25.14.2.6, 26.1, 26.5, 26.6.1–26.6.4, 26.7, and 27 of the Technical Report.
9) I am independent of the Company as independence is defined in Section 1.5 of NI 43-101.
10) I have not been previously involved with the Sisson 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: October 1, 2026

"*Signed and sealed*"

Kevin Murray, P.Eng.

Certificate of Qualified Person
Jennifer Lynn McPhail, P.Eng.

I, Jennifer Lynn McPhail, P.Eng., certify that:

1. I am employed as Principal with Stantec Consulting Ltd., (Stantec), with an office address of 845 Prospect St, Fredericton, NB, Canada E3B 2T7.
2. This certificate applies to the technical report titled "Sisson Tungsten-Molybdenum Project NI 43-101 Technical Report & Feasibility Study Update, New Brunswick, Canada" (the "Technical Report"), prepared for Northcliff Resources Ltd (the "Company"), with a report date of October 1, 2026 and an effective date of September 23, 2026 (the "Effective Date").
3. I graduated from University of New Brunswick with a Master of Engineering in Chemical Engineering in 2006.
4. I am a professional engineer registered with the Association of Professional Engineers and Geoscientists of New Brunswick (No. M7036).
5. I have practiced my profession continuously for 20 years with experience in environmental impact assessments of resource development and other projects in New Brunswick and throughout Atlantic Canada.
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 fulfill the requirements to be a "Qualified Person" for the purposes of this instrument.
7. I visited the project site on multiple occasions in the past in support of the environmental impact assessment of the project. My most recent visit to the site was September 8, 2026.
8. I am responsible for Sections 1.16.1–1.16.3, 2.4.1, 3.3, 12.7, 20.1–20.4, 25.10, 25.14.1.8, 25.14.2.8, 26.8 and 27 of the Technical Report.
9. I am independent of the Company as independence is defined in Section 1.5 of NI 43-101.
10. I have been previously involved in a consulting engineering capacity with the Sisson Tungsten-Molybdenum Project. I have supported the Environmental Impact Assessment, permitting, and follow up processes since approximately 2011.
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: October 1, 2026

"*Signed and sealed*"

Jennifer Lynn McPhail, P.Eng.

Certificate of Qualified Person
Daniel Friedman, P.Eng.

I, Daniel Friedman, P.Eng., certify that:

1. I am employed as a Specialist Civil Engineer with Knight Piésold Ltd., (KP), with an office address of Suite 1400-750 West Pender Street, Vancouver, British Columbia, V6C 2T8, Canada.
2. This certificate applies to the technical report titled "Sisson Tungsten-Molybdenum Project NI 43-101 Technical Report & Feasibility Study Update, New Brunswick, Canada" (the "Technical Report"), prepared for Northcliff Resources Ltd (the "Company"), with a report date of October 1, 2026 and an effective date of September 23, 2026 (the "Effective Date").
3. I am a graduate of McGill University, Montreal, Canada with a B.Eng. (Civil), in 2003.
4. I am a registered professional engineer, in good standing, in New Brunswick, Canada (No. L5001), British Columbia, Canada (No. 32571), Yukon, Canada (No. 3404) and Arizona, USA (No. 53722).
5. I have practiced my profession continuously since 2004. My principal experience is in the areas of water and waste management for mining projects, including open pit and underground hard-rock projects, and hydrotechnical engineering. My experience covers all phases of mine development including design, construction, operation, and closure.
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 fulfill the requirements to be a "Qualified Person" for the purposes of this instrument.
7. My most recent personal inspection of the Property was September 13, 2011.
8. I am responsible for Sections 1.14.3, 1.14.5, 1.16.4, 2.4.5, 12.4, 18.5, 18.6.1, 18.6.4–18.6.7, 18.7, 20.5, 20.6, 21.2.4.1, 21.2.10, 21.3.1, 21.3.5.1, 25.9.2, 25.14.1.7.2, 25.14.7.3, 26.6.5, and 27 of the Technical Report.
9. I am independent of the Company as independence is defined in Section 1.5 of NI 43-101.
10. I have been previously involved in a consulting engineering capacity with the Sisson Tungsten-Molybdenum Project. I have developed various engineering studies related to tailings and water management for Northcliff Resources Ltd. and its affiliates ("Northcliff") from 2010 through 2026, with most of the prior engineering study work carried out in 2011 to 2014, and 2025 to 2026. I was a qualified person for the technical report entitled "Canadian National Instrument 43-101 Technical Report on the Sisson Project, New Brunswick, Canada", prepared for Northcliff Resources Ltd., and dated January 22, 2013.
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: October 1, 2026

"*Signed and sealed*"

Daniel Friedman, P.Eng.

Certificate of Qualified Person
Robert Gray, P.Eng.

I, Robert Gray, P.Eng., certify that:

1. I am employed as a Manager Engineer for Civil and Mining projects, with Moose Mountain Technical Services, (MMTS), with an office address of #210 - 1510 2 St N, Cranbrook, BC, Canada, V1C 6H4.
2. This certificate applies to the technical report titled "Sisson Tungsten-Molybdenum Project NI 43-101 Technical Report & Feasibility Study Update, New Brunswick, Canada" (the "Technical Report"), prepared for Northcliff Resources Ltd (the "Company"), with a report date of October 1, 2026 and an effective date of September 23, 2026 (the "Effective Date").
3. I graduated from University of Calgary with a Bachelor of Science in Civil Engineering in June, 2006.
4. I am a Professional Engineer (P.Eng.) registered with Engineers and Geoscientists British Columbia (No. 232577), the Association of Professional Engineers and Geoscientists of Alberta (No. 86997), Engineers Nova Scotia (No. 20260258)
5. I have practiced my profession continuously since 2006 and have 18 years of experience in open-pit mine planning, production scheduling, mine cost estimating and Reserve estimating for a variety of precious and strategic metals, including but not limited to Atlantic Canada.
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 fulfill the requirements to be a "Qualified Person" for the purposes of this instrument.
7. I visited the project site on September 18, 2025.
8. I am responsible for Sections 1.11, 1.12, 2.4.3, 12.2, 15, 16, 21.2.3, 21.3.1–21.3.3, 25.6, 25.7, 25.14.1.4, 25.14.1.5, 25.14.2.4, 25.14.2.5, 26.4 and 27 of the Technical Report.
9. I am independent of the Company as independence is defined in Section 1.5 of NI 43-101.
10. I have been previously involved with the Sisson Project, beginning in 2011 through to 2018, and then again in 2022 through to present. During this time I was involved in a consulting capacity with mine planning, design, engineering studies and reports, including the technical report entitled "Canadian National Instrument 43-101 Technical Report on the Sisson Project, New Brunswick, Canada", prepared for Northcliff Resources Ltd., and dated January 22, 2013
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: October 1, 2026

"*Signed and sealed*"

Robert Gray, P.Eng.

Certificate of Qualified Person
AJ MacDonald, P.Eng.

I, AJ MacDonald, P.Eng., certify that:

1. I am employed as Vice President Engineering with Integrated Sustainability Consulting Ltd. (ISC), with an office address of Suite 400, 1305 11th Avenue SW, Calgary, Alberta Canada T3C 3P6.
2. This certificate applies to the technical report titled "Sisson Tungsten-Molybdenum Project NI 43-101 Technical Report & Feasibility Study Update, New Brunswick, Canada" (the "Technical Report"), prepared for Northcliff Resources Ltd (the "Company"), with a report date of October 1, 2026 and an effective date of September 23, 2026 (the "Effective Date").
3. I graduated from Queen's University in Kingston, Ontario, Canada, with a Bachelor of Science in 2005, and from Carleton University in Ottawa, Ontario, Canada, with a Master of Applied Science in Environmental Engineering in 2007.
4. I am a Professional Engineer and member in good standing of the Association of Professional Engineers Yukon (No. 2697), Engineers and Geoscientists British Columbia (No. 45872), Association of Professional Engineers and Geoscientists of Alberta (No. 95330), Association of Professional Engineers and Geoscientists of Saskatchewan (No. 64847), Professional Engineers Ontario (No. 100536555), Association of Professional Engineers and Geoscientists of New Brunswick (No. L6636), and Professional Engineers and Geoscientists Newfoundland and Labrador (No. 10454).
5. I have practiced my profession continuously since 2007, with experience in dozens of mining and other resource-sector projects, primarily in Western Canada and with a particular focus on water treatment. My experience spans preliminary analysis, conceptual design, detailed design, construction, commissioning, and optimization of infrastructure at industrial water treatment facilities in Canada and internationally.
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 fulfill the requirements to be a "Qualified Person" for the purposes of this instrument.
7. I have not visited the project site.
8. I am responsible for Sections 1.14.4, 12.6, 18.6.9, 20.6, 21.2.4.2, 21.2.10, 21.3.1, 21.3.5.2, 25.9.3, 25.14.1.7.4, 25.14.2.7, 26.1, 26.6.6 and 27 of the Technical Report.
9. I am independent of the Company as independence is defined in Section 1.5 of NI 43-101.
10. I have not been previously involved with the Sisson 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: October 1, 2026

"*Signed and sealed*"

AJ MacDonald, P.Eng.

Certificate of Qualified Person
Paul Chamois, M.Sc.(A), P. Geo.

I, Paul Chamois, M.Sc.(A), P. Geo., certify that:

1. I am employed as an Associate Principal Geologist with SLR Consulting (Canada) Ltd., (SLR), with an office address of 55 University Avenue, Suite 501, Toronto, Ontario, Canada, M5J 2H7.
2. This certificate applies to the technical report titled "Sisson Tungsten-Molybdenum Project NI 43-101 Technical Report & Feasibility Study Update, New Brunswick, Canada" (the "Technical Report"), prepared for Northcliff Resources Ltd (the "Company"), with a report date of October 1, 2026, and an effective date of September 23, 2026 (the "Effective Date").
3. I graduated from Carleton University with a Bachelor of Science (Honours) degree in Geology in May 1977 and from McGill University with a Master of Science (Applied) degree in Mineral Exploration in May 1979.
4. I am registered as a Professional Geoscientist in the Province of Ontario (Reg.#0771), in the Province of Newfoundland and Labrador (Reg.#03480), in the Province of Saskatchewan (Reg.#14155) and in the Northwest Territories and Nunavut (Reg.#L4088).
5. I have practiced my profession continuously for 47 years. My relevant experience for the purpose of this Technical Report is:
    a. Review and report on exploration and mining projects for due diligence and regulatory requirements.
    b. Vice President – Exploration with Canadian mineral exploration and development company responsible for technical aspects of exploration programs and evaluation of new property submission.
    c. District Geologist with a major Canadian mining company in charge of technical and budgetary aspects of exploration programs in Eastern Canada.
    d. Project Geologist with a major Canadian mining company responsible for field mapping and sampling, area selection and management of drilling programs across Ontario and Québec.
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 fulfill the requirements to be a "Qualified Person" for the purposes of this instrument.
7. I visited the project site on June 9, 2022.
8. I am responsible for Sections 1.3–1.8, 2.4.2, 3.2, 4–11, 12.1, 23, 25.2, 25.3, 25.4, 25.14.1.1, 25.14.2.1, 26.2, 26.3.1, and 27 of the Technical Report.
9. I am independent of the Company as independence is defined in Section 1.5 of NI 43-101.
10. I have not been previously involved with the Sisson Tungsten-Molybdenum 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: October 1, 2026

"*Signed and sealed*"

Paul Chamois, M.Sc.(A), P. Geo.

Certificate of Qualified Person
Philip A. Geusebroek, M.Sc., P.Geo.

I, Philip A. Geusebroek, M.Sc., P.Geo., certify that:

1. I am employed as a Consultant Resource Geologist with SLR Consulting (Canada) Ltd, (SLR), with an office address of Suite 501, 55 University Avenue, Toronto, Ontario, Canada.
2. This certificate applies to the technical report titled "Sisson Tungsten-Molybdenum Project NI 43-101 Technical Report & Feasibility Study Update, New Brunswick, Canada" (the "Technical Report"), prepared for Northcliff Resources Ltd (the "Company"), with a report date of October 1, 2026 and an effective date of September 23, 2026 (the "Effective Date").
3. I graduated from University of Western Ontario with a Master of Science in Geology in October, 2008. Previously, I graduated from the University of Alberta with a Bachelor of Science with Specialization in Geology in June 1995.
4. I am a Professional Geologist registered with The Association of Professional Engineers and Geoscientists of New Brunswick (No. 2659), and Professional Geoscientists Ontario (No. 1938), and The Association of Professional Engineers and Geoscientists of Alberta (No. 57911).
5. I have practiced my profession continuously for 30 years, with the following experience:
    a. Mineral Resource estimation, geological modelling, geological database implementation and management, mineral exploration workflows and QA/QC experience.
    b. Reviewed and reported as a consultant on numerous exploration, development, and production mining projects around the world for due diligence and regulatory requirements.
    c. Approximately a decade as a mine and exploration geologist with Echo Bay Mines Ltd., Kinross Gold Corporation, and Western Mining Company.
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 fulfill the requirements to be a "Qualified Person" for the purposes of this instrument.
7. I have not visited the project site.
8. I am responsible for Sections 1.10, 12.5, 14, 25.3, 25.14.1.3, 25.14.2.3, 26.3.2, and 27 of the Technical Report.
9. I am independent of the Company as independence is defined in Section 1.5 of NI 43-101.
10. I have not been previously involved with the Sisson 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: October 1, 2026

"*Signed and sealed*"

Philip A. Geusebroek, M.Sc., P.Geo.

Important notice

This report was prepared as National Instrument 43-101 Technical Report for Northcliff Resources, Ltd. (Northcliff) by Ausenco Engineering Canada ULC (Ausenco), Knight Piésold Consulting (KP), Moose Mountain Technical Services (MMTS), SLR Consulting (Canada) Ltd (SLR), Stantec Consulting Ltd. (Stantec), and Integrated Sustainability Consultants Ltd. (ISC), 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 Northcliff subject to terms and conditions of its contracts with each of the Report Authors. Except for the purposes legislated under Canadian provincial and territorial securities law, any other uses of this report by any third party is at that party's sole risk.

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1 Summary

1.1 Introduction

Northcliff Resources Ltd. ("Northcliff", the "Company", or the "Owner") commissioned Ausenco Engineering Canada ULC (Ausenco) to manage the preparation of a Feasibility Study Update (FSU) and associated technical report for the Sisson Tungsten-Molybdenum Project (the "Sisson Project", the "Project", or the "Property") located in New Brunswick, Canada (the "Technical Report"). The Technical Report was prepared in accordance with the Canadian disclosure requirements of National Instrument (NI) 43-101 – Standards and Disclosure for Mineral Projects (NI 43-101) and the requirements of Form 43-101 F1.

The responsibilities of the engineering consultants and firms contracted by Northcliff to prepare this Technical Report are as follows:

- Ausenco was responsible for managing and coordinating the work related to the Technical Report. Ausenco managed the metallurgical testwork program, developed the FS-level design, capital, and operating costs estimates for the process plant and general site infrastructure. Ausenco compiled the overall capital cost estimate and is also responsible for auditing the economic analysis.
- SLR Consulting (Canada) Ltd. (SLR) prepared the Mineral Resource Estimate (MRE) for the Project and was responsible for completing the work related to the property and geological setting, deposit type, drilling, exploration works, sample preparation and analysis. SLR acquired RPA Consulting (RPA) in 2019 and references to RPA in this Technical Report should be viewed in this context.
- Moose Mountain Technical Services (MMTS) was responsible for completing the reserve estimate, open pit mine design, quarry design, mine production schedule, mine related infrastructure, and mine and quarry capital and operating cost estimates.
- Knight Piésold Consulting (KP) was responsible for completing the tailings storage facility, water management design, site wide water balance, TSF and water management capital and operating cost estimates, and closure costs.
- Integrated Sustainability Consultants Ltd. (ISC) was responsible for water treatment plant design and cost estimate and WTP closure cost.
- Stantec Consulting Ltd. (Stantec) was responsible for completing environmental and permitting works.

1.2 Terms of reference

The purpose of this Technical Report is to present the results of the FSU and to support the disclosure by Northcliff in a news release dated August 31, 2026 entitled "Northcliff Announces Results of Feasibility Study Update for Sisson Project."

All measurement units used in this Technical Report are SI units unless otherwise noted. Currency is expressed in Canadian dollars (C$ or CAD) unless otherwise noted. This Technical Report has been prepared using Canadian English spelling conventions. Metric tonne units (MTU) are used to express contained tungsten trioxide (WO_3), with 1 MTU equivalent to 10 kg of contained WO_3.

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

1.3 Project description and location

The Property, covering an area of approximately 14,140 hectares (ha), is located approximately 60 kilometres (km) northwest of Fredericton, New Brunswick. The claims lie within National Topographic Series map sheets 21J2, 3, 6, and 7. The approximate centre of the Property is located at grid coordinates 650000E, 5135500N, within UTM Zone 19T (NAD 83).

1.4 Mineral tenure and ownership

Tenure for the mineral rights is held via three contiguous claim groups comprising a total of 707 units. In New Brunswick, claims are staked online as blocks of units which measure 500 x 500 metres (m) each. A unit is the smallest measure within the New Brunswick mineral grid. Sisson claims were originally staked prior to the implementation of the map-based system currently in place. The Sisson claims were converted to the current system on February 3, 2011.

Northcliff acquired the Sisson claims through a joint venture (JV) agreement with Geodex Minerals Ltd (Geodex), which was entered into on October 21, 2010. Under the terms of the JV, Northcliff could acquire a 70% interest in the Project by funding up to C$17 million in exploration and development costs. Northcliff was appointed the operator of the Project. Northcliff completed the earn-in obligations to secure a 70% interest in June 2012.

In May 2012, Northcliff and Geodex entered into a definitive agreement by which Northcliff could acquire the remaining 30% interest in the Property. Geodex agreed to sell its interest for 16,003,700 common shares of Northcliff, C$1 million in cash, and the return of 3,333,333 Geodex common shares which had been purchased by Northcliff in 2010. Northcliff shareholders unanimously approved the acquisition at the company's Annual General Meeting held on May 30, 2012, and Geodex shareholder approval was received on June 18, 2012. The transaction closed on June 21, 2012.

On October 3, 2013, Northcliff announced that it had entered into a C$5 million private placement financing agreement with Todd Sisson (NZ) Limited, a subsidiary of Todd Corporation of New Zealand ("Todd"). As a result, Todd acquired ownership and control of 13,888,889 shares in Northcliff and became Northcliff's largest shareholder with an approximate 15% interest in Northcliff.

As of 2026, Todd participated in several private placements and provided loans to Northcliff for which it ultimately received shares in Northcliff as payment. As of the effective date of this Technical Report, Todd owns a total of 543,213,298 shares in Northcliff and controls 81.9% of Northcliff.

In addition, in October 2013, Todd and Northcliff entered into a limited partnership agreement under the name of the Sisson Project Limited Partnership (Sisson LP) in which Todd acquired an initial 11.5% interest for an initial contribution of C$14 million, which was funded in stages and completed in October 2014. Todd has the option to acquire an additional 10% interest in Sisson LP by investing an additional C$20 million upon a final investment decision to commence construction.

Northcliff owns a collective 88.5% interest in the Sisson LP, through its 100% interest in Sisson Resources Ltd and Northcliff Holdings (Canada) Ltd. (88.49%) and through its 88.5% interest in Sisson Mines Ltd, (0.01%). Sisson Mines Ltd is the general partner of Sisson LP.

The Sisson claims are held by Sisson LP.

1.5 History

The first significant work in the Sisson area was carried out in the late 1950s by Nashwaak Pulp and Paper Co. Twelve holes were completed in 1955 and 43 holes in 1959 to 1960 which resulted in the discovery of the Nashwaak polymetallic vein deposit.

From 1967 to 1969, Penarroya Canada Ltée conducted geological mapping, a ground magnetic survey, and soil sampling mostly south of the Sisson deposit. Texasgulf Inc. and Kidd Creek Mines Ltd. (Kidd Creek) carried out exploration work from 1973 to 1983 comprising soil sampling, geological mapping, trenching, ground geophysical surveys, and drilling. Relatively limited work was conducted by various operators from 1977 and 2001. From 2004 to 2009, Geodex, initially in JV with Champlain Resources Inc., carried out ground and airborne geophysical surveys, compilation of historical data, trenching, reanalysis of historical drill core, geological mapping and prospecting, and extension of previous soil and till sampling grids over and around the Sisson deposit. Approximately 210 drill holes were completed. Preliminary economic assessments (PEAs) with positive conclusions were completed by Wardrop Engineering Inc. in 2007 and Geodex in 2009. Northcliff initially signed a JV agreement with Geodex in October 2010 and eventually earned a 100% interest in the Project in June 2012, through completion of its earn-in obligations and purchase of Geodex's remaining interest.

1.6 Geology and mineralization

The Sisson deposit can be defined as an intrusion related, structurally controlled, bulk tonnage tungsten-molybdenum deposit. Deposits of this type have general hydrothermal similarities to porphyry copper deposits. They form in convergent margin to collisional tectonic environments and are related to highly evolved granitic melts formed from continental crust.

The Sisson deposit is centred on a north-trending contact between Acadian intrusions to the west and older metavolcanic and metasedimentary rocks to the east. Mineralization occurs in four contiguous zones. Zones I and II are narrow, structurally controlled zones that extend north from Zone III, which hosts most of the mineralization. The Ellipse Zone extends northwest from the southwest corner of Zone III. Metavolcanic and metasedimentary host rocks at Sisson formed during the Taconic Orogeny and are of Cambrian to Ordovician age. They include the predominantly clastic sedimentary sequences in the Miramichi Group overlain by Ordovician felsic to mafic volcanic strata and clastic sedimentary rocks of the Tetagouche Group. The rocks within and near the Sisson deposit consistently strike north-northeast and dip steeply to the east.

Mineralization in the Sisson deposit is hosted by:

- quartz diorite and gabbro phases of the Howard Peak Granodiorite
- felsic, mafic, and mafic crystal tuffs in the western part of the Turnbull Mountain Formation
- biotite wacke with minor interbeds of tuff in the eastern part of the Turnbull Mountain Formation
- volumetrically minor granite dykes and very rare mafic dykes
- Low-grade mineralization on the eastern edge of the Sisson deposit is hosted by more siliceous biotite-sericite wackes that may be part of the Miramichi Group.

Mineralization at Sisson occurs almost exclusively in quartz veins, fractures and, locally, in their alteration envelopes. Tungsten and molybdenum are the metals of principal economic interest, whereas several other metals, including copper, zinc, lead, arsenic, and bismuth, occur in geochemically anomalous but subeconomic concentrations.

1.7 Exploration and drilling

Since 1979, 69,321 m of diamond drilling has been completed on the property in 407 drill holes by three operators.

Diamond drilling to delineate the W-Mo ± Cu mineralization in Zones I, II, and III on the Sisson property was completed by Kidd Creek during the 1979 to 1982 period and Geodex in the 2005 to 2009 period. Kidd Creek completed 40 drill holes on the property in three campaigns. Geodex drilled 213 additional holes totalling 46,655 m in five subsequent campaigns to the end of 2009.

Northcliff completed drill campaigns in 2010, 2011, 2013, 2014, 2023 and most recently in 2025-2026. These campaigns were designed to obtain exploration data including metallurgical, hydrogeological, and geotechnical information within both the resource area and Tailings Storage Facility impoundment. An aggregate of 154 holes were drilled totaling 15,161 metres.

The Sisson Deposit has been essentially closed off by resource drilling to the northeast, northwest, and south. At the present time, with mineralization being constrained by distinct geological and alteration boundaries, Northcliff considers the exploration potential in the immediate vicinity to be limited and has no plans for additional exploration activity.

1.8 Sample preparation and analyses

During the Kidd Creek campaigns (1979-1982), half-core samples were collected and sent to Bondar-Clegg (Ottawa, ON) for analysis. Sample preparation procedures were poorly documented, but assays employed conventional methods such as colorimetric analysis for tungsten, atomic absorption spectrometry (AAS) for copper, molybdenum and bismuth, fire assay for silver, and wet-chemistry techniques for arsenic. Sample or site security measures were not described in the Kidd Creek reports, and it is assumed that security measures consistent with industry standards of the day were in place during these campaigns.

During the Geodex campaigns (2005-2009), sample preparation became standardized. Half-core samples were shipped to Actlabs (Fredericton, NB and Ancaster, ON), where they were crushed, split and pulverized to 95%, passing 105 µm (150 mesh). Analytical methods transitioned to modern techniques, with Instrumental Neutron Activation Analysis (INAA) used for tungsten and molybdenum and ICP-AES methods used for copper and multi-element analyses. Independent check assays were performed by ALS Chemex in 2006, and by Global Discovery/Acme from 2007-2009.

A system of drill core security was implemented during the 2006 program that began at the drill site and continued through transportation to the sampling and logging facility, and then onward through shipment of samples to the laboratory. Designated field transfer points were established at which responsibility for core security passed from the drilling contractor to Geodex.

As part of Northcliff's 2010 due diligence program, archived core, reject and pulp samples were re-sampled and submitted to Acme Laboratories (Vancouver). Samples were crushed to 75% passing 2 mm and pulverized to 85% passing 75 µm, then analyzed using four-acid digestion with ICP-MS for copper and multi-elements and phosphoric acid leach with ICP-AES for tungsten and molybdenum. This program successfully verified the earlier Geodex results.

During the Northcliff drilling campaigns, more rigorous procedures were implemented. Diamond-sawn half-core samples were prepared at Actlabs Fredericton, NB, crushed to 75% passing 2 mm, and pulverized

to 95% passing 105 µm before analysis at Actlabs, Ancaster, ON. Tungsten and molybdenum were analyzed by INAA method, while copper, silver, sulphur and a broad multi-element suite were analyzed using aqua regia digestion with ICP-OES/MS. Independent check assays continued at Acme Laboratories. At this stage, the project implemented comprehensive chain-of-custody controls, including matrix-matched certified reference standards, blanks, and duplicates, resulting in a robust geochemical database suitable for Mineral Resource estimation.

1.9 Metallurgical testwork

Several metallurgical test programs have been conducted on representative composites from the Sisson deposit since 2008. A smaller portion of the testing was completed by the previous project owner, while the bulk of the testing has been completed by Northcliff Resources. Aside from one small test program, all metallurgical testing has been completed by SGS, most of which took place at their Lakefield facility.

Comminution testing on the Sisson materials indicate that it is quite competent with respect to impact breakage, the 75th percentile of the Axb values measured is 24.8. For this reason, testing with high pressure grinding rolls has also been completed which demonstrated that the material is amenable to this crushing technology. The material also has a moderately high ball mill work index, the measured 75th percentile is 17.4 kWh/t.

Preconcentration methods have been investigated to reduce the feed mass advancing to the comminution and flotation circuits. Particle ore sorting using dual energy x-ray transmission sensors on broken ½ core material indicated that up to 60% of the sorter feed mass can be rejected with WO_3 and Mo losses of approximately 6% and 15%, respectively. Heavy liquid separation was tested but did not appear to be as effective, as it was sensitive to small incremental changes to the slurry SG.

Gravity concentration was evaluated by various methods on a number of test programs. Approximately 40-50% of the feed tungsten can be recovered to a low mass concentrate at primary grind sizes of about 200 µm P_{80}.

A flotation only process flowsheet was selected to carry forward and received most of the metallurgical testing. This flowsheet was selected as it has the lowest complexity and was considered to be well suited to a high throughput operation. Initial testing evaluated global composites that were stage ground to -348 µm, and later to the selected closing screen of 212 µm. The 80% passing value at the -212 µm grind size was 140 µm.

The mineral assemblage of the material is suited to a sequential flotation process, where molybdenum is recovered first, followed by sulphide minerals dominated by pyrite and pyrrhotite. The bulk sulphide flotation stage is simply to remove sulphides from the tungsten flotation feed, the sulphide rougher concentrate reports to tailings. The molybdenum rougher circuits consistently achieve about 90% recovery to a mass pull of about 2%. Locked cycle testing on the molybdenum cleaner circuit indicates that final concentrates grading 48% Mo can be produced after six cleaning stages at a circuit recovery of 93%. An overall molybdenum recovery of 82% is estimated.

Bulk sulphide flotation tailings were advanced to tungsten flotation testing. Due to the high concentration ratio required in the tungsten flotation circuit, large feed masses were tested. Two pilot programs were conducted in the 2013 test program to generate suitable quantities of molybdenum rougher concentrate and sulphide tailings for downstream testing. Flowsheet development testing for the tungsten circuit was conducted on both crushed feed that went through Mo and S flotation, as well as portions of pilot plant tailings. Oxide mineral flotation chemistry was applied to the tungsten flotation feed, which included

conditioning with sodium hydroxide and sodium silicate, and often other reagents to adjust the pulp chemistry. A fatty acid collector was used to recover scheelite.

The initial tungsten flotation strategy was to generate a concentrate with about 30-35% WO_3, that would be a suitable feed for hydrometallurgical upgrading in an APT process. Locked cycle testing with this conventional 3-stage cleaning circuit demonstrated that 80% of the feed tungsten could be recovered to a concentrate grading 32% WO_3. Hydrometallurgical testing was also conducted on concentrates generated in the 2013 program, which demonstrated that the concentrate was amenable to high pressure leaching, impurity removal, solvent extraction and ultimately production of commercial-grade ammonium paratungstate crystals (APT).

In 2014 an alternate flotation strategy was introduced to the project, a modified Petrov method, which involved heating an intermediate flotation concentrate prior to final cleaning stages. This hot conditioning process demonstrated that concentrates grading 5-10% WO_3 generated in ambient temperature conventional flotation could be upgraded to over 60% WO_3 at high cleaner circuit recoveries. The testing conducted in 2014 and 2017 used feed sample and pilot plant tailings remaining from the 2013 test program. Additional composites were assembled in 2025 to confirm the process and test variability samples. The recent testing identified parameters that required increased control and the 2017 metallurgical performance was ultimately replicated. Variances in performance are now understood to be a result of optimizing collector dosages and working with somewhat low feed sample masses. The results indicate that a tungsten recovery of 78% to a concentrate grading 50% WO_3 can be achieved.

1.10 Mineral Resource Estimate

The Mineral Resource Estimate (MRE) was prepared by running a new pit optimization on the block model (the 2012 model) supporting the 2012 MRE. For this Technical Report, current metal prices, mining costs, and updated processing parameters were used as input to a new Net Smelter Return (NSR) equation and cut-off value determination and applied to the 2012 model. Tungsten trioxide (WO_3) and molybdenum (Mo) values from the 2012 model were unchanged.

Drill hole information collected in 2013, 2014, and 2023, after the 2012 block model was prepared, was reviewed against estimated block values and found to show general accord. All but two are beyond the deposit extents, and the results are not incorporated.

With exception of the updated NSR equation and values, the core of the 2012 model is unchanged. The model extents were expanded to remove spatial constraints during pit optimization.

NSR calculation revisions align with the processing parameters derived from the FSU commissioned by Northcliff in 2025, and the cost considerations derived from the 'No APT' WO_3 concentrate scenario developed by Moose Mountain Technical Services (MMTS).

The December 31, 2025, MRE is summarized in Table 1-1. The Canadian Institute of Mining, Metallurgy and Petroleum (CIM) Definition Standards for Mineral Resources and Mineral Reserves definitions (CIM, 2014) were used for Mineral Resource classification. Mineral Resources that are not Mineral Reserves do not have demonstrated economic viability.

Table 1-1: Summary of Mineral Resources -effective date: December 31, 2025

Category	Tonnage	Grade WO_3	Grade Mo	Contained WO_3	Contained Mo	Grade WO_3EQ	Average NSR
	(Mt)	(%)	(%)	(M MTU)	(M lb)	(%)	(C$/t)
Measured	117.9	0.068	0.022	8.0	57.2	0.098	39.13
Indicated	303.7	0.061	0.019	18.5	127.2	0.088	34.65
Total Measured and Indicated	**421.6**	**0.063**	**0.020**	**26.6**	**185.9**	**0.091**	**35.90**
Inferred	182.9	0.048	0.020	8.8	80.6	0.078	28.59

Notes:
1. CIM (2014) definitions were followed for Mineral Resources.
2. $WO_3Eq = WO_3\% + ((Mo\% \times NSP\_Mo \times REC\_Mo)/(NSP\_WO_3 \times REC\_WO_3))$.
3. Mineral Resources are estimated at an NSR cut-off grade of C$12.02/t.
4. Mineral Resources are estimated using a long-term WO_3 price of US$500/MTU, and a long-term Mo price of US$23/lb.
5. Recovery factors are based on a conditional regression formula for WO_3, and set to 82% for Mo.
6. Mineral Resources are constrained within a Whittle shell, generated using the NSR cut-off value.
7. Bulk density is interpolated into blocks where proximal density sample information is present and set to 2.77 t/m^3 otherwise.
8. Mineral Resources are inclusive of Mineral Reserves.
9. Mineral Resources that are not Mineral Reserves do not have demonstrated economic viability.
10. To meet reasonable prospects for eventual economic extraction, Mineral Resources assume metallurgical recoveries and processing costs to be the same as for Mineral Reserves but use higher long term metal prices to calculate NSR.
11. Numbers may not add due to rounding.

The QP is not aware of any environmental, permitting, legal, title, taxation, socio-economic, marketing, political, or other relevant factors that could materially affect the MRE.

1.11 Mineral reserve estimate

The Mineral Reserve estimates for the Sisson Project, with an effective date of August 11, 2026, were prepared by independent Qualified Person, Robert Gray, P.Eng. The estimates were prepared in accordance with the 2014 CIM Definition Standards and the 2019 CIM Best Practices Guidelines.

The Mineral Reserves are a subset of the physically contained Measured and Indicated Mineral Resources and are supported by the FSU Open-pit Life-of-mine (LOM) Plan. Proven and Probable Mineral Reserves are modified from the Measured and Indicated Mineral Resources, while all Inferred Mineral Resources within the pit are treated as barren rock.

The reference point for the Mineral Reserves is the mill feed at the primary crusher. The estimate incorporates operational modifying factors based on a 10 m x 10 m x 10 m block size, including a variable application of dilution that averages 2.56% over the model at a 0% diluting grade, alongside a 98.76% mining recovery. The Mineral Reserves are reported using a variable net smelter return (NSR) cut-off by period, yielding an effective LOM cut-off of $16.31/t. The NSR calculation assumes concentrate pricing of US$330/MTU for WO_3 and US$20/lb for Mo, offsite costs of $0.12/lb WO_3 and $3.39/lb Mo, alongside a variable WO_3 metallurgical recovery capped at 82% and a fixed Mo metallurgical recovery of 82%.

The total Proven and Probable Mineral Reserves are estimated at 276.8 Mt of run-of-mine (ROM) material. This total is comprised of:

- **Proven Mineral Reserves:** 85.6 Mt ROM at an average NSR of $37.81/t and average metal grades of 0.079% WO_3 and 0.025% Mo. Proven reserves contain 6.77 million MTU of WO_3 and 47.79 million pounds of Mo.
- **Probable Mineral Reserves:** 191.2 Mt ROM at an average NSR of $33.78/t and average metal grades of 0.073% WO_3 and 0.021% Mo. Probable reserves contain 13.92 million MTU of WO_3 and 90.58 million pounds of Mo.
- **Total Proven and Probable:** 276.8 Mt ROM at an average NSR of $35.03/t and average metal grades of 0.075% WO_3 and 0.023% Mo. Total Proven and Probable reserves contain 20.69 million MTU WO_3 and 138.37 million pounds of Mo.

Factors that may materially affect the Mineral Reserve estimate include changes in metal prices, variations in geological interpretations of mineralization, and updated geotechnical or hydrogeological assumptions. Additionally, changes to operating and capital costs, fluctuations in process plant recoveries, or the inability to maintain necessary environmental permits and social licence could impact these reserve estimates.

1.12 Mining methods

The Sisson Project will utilize conventional, owner-operated open-pit mining methods utilizing a truck-and-shovel fleet. The operation is designed for year-round production in a Canadian climate. The mine plan provides a nominal mill feed rate of 30,000 t/d, which equates to 10,500 kt/a.

The ultimate pit limits were determined using the Pseudoflow algorithm to optimize project net present value. The open pit is designed in seven distinct phases to target consistently higher economic margin material early in the mine life. Over the 27-year life of mine (LOM), the operation will yield the following key quantities:

- Total material mined from the open pit is 564 Mt.
- Total ore mined is 286 Mt, yielding 268 Mt of direct mill feed.
- Total barren rock mined from the pit is 278 Mt.
- The overall open-pit strip ratio (barren rock to ore mined) is 0.97 to 1.

Production loading will be primarily executed by 300-tonne electric-hydraulic front shovels, supported by a 200-tonne diesel hydraulic backhoe and a 15-cubic-metre wheel loader. Material will be hauled using a fleet of 147-tonne payload off-highway trucks. To improve safety and optimize unit costs, the Sisson Project will integrate an Autonomous Haul System (AHS) for its hauling fleet. Total material movement is constrained by shovel availability, peaking at 36,000 kt/a and averaging approximately 22,000 kt/a over the LOM. Production drilling utilizes diesel hydraulic down-the-hole (DTH) tracked drills, while all blasting operations are managed by a full-service contractor

Due to the presence of potentially acid-generating (PAG) material within the open pit, all barren rock and stockpiled ore must be carefully managed. A Barren Rock Storage Area (BRSA) and a Mid-Grade Stockpile (MGSP) will operate within the footprint of the Tailings Storage Facility (TSF), where they will be progressively encapsulated by tailings. Because barren rock from the pit cannot be used for the tailings embankment, non-PAG construction material will be sourced from a separate, four-phase quarry located within the TSF footprint. The LOM schedule includes 73,480 kt of total rock-fill quarried. By Year 19, targeted areas of the

open pit will reach their final floor, allowing barren rock to be directly backfilled into the pit and eliminating the need for further BRSA expansion.

1.13 Recovery methods

The Sisson process plant flowsheet is designed to process 30,000 t/d and considers crushing, grinding, multiple stages of flotation, and dewatering. Two stages of conventional crushing will be employed prior to tertiary crushing by HPGR. The tertiary crushing circuit operates in closed circuit with a wet screen, with undersize passing to the grinding area. Grinding will be conducted via ball milling in closed circuit with cyclones.

Cyclone overflow will report to a multi-stage flotation process. Molybdenum minerals will be collected in the first stage concentrate which will consist of rougher flotation, regrinding, and cleaner flotation. Additional cells at the end of the molybdenum rougher flotation bank will be included to remove any residual sulphides as waste prior to tungsten flotation.

Tailings from the first flotation stage will be thickened prior to additional processing to create a separate process water loop, isolating the tungsten flotation reagents within the tungsten flotation area. Following repulping with tungsten process water, the tailings will then proceed through tungsten flotation, consisting of rougher and cleaner stages, to produce an intermediate tungsten concentrate. The concentrate will be thickened and heated to undergo a high-density hot condition step prior to additional flotation stages to generate a final tungsten concentrate.

Both the tungsten and the molybdenum concentrates will be dewatered in dedicated circuits consisting of high-rate thickeners, plate-and-frame filters, and indirect heated dryers. The final products will be bagged and weighed for transport offsite.

Tailings from the process will be separated within the process into potentially acid-generating (PAG) and non-acid-generating (NAG) materials. PAG tailings will be collected from the first stage of flotation, including the residual sulphide waste, and pumped directly to the tailings facility for subaqueous deposition. NAG tailings will be thickened prior to leaving the process plant to recycle dissolved reagents back to the tungsten flotation area. Thickened tailings will then be pumped to the TSF.

1.14 Project infrastructure

1.14.1 Infrastructure requirements

The proposed Sisson Mine is a greenfield site. The site is accessible from Highway 102/104 from the South (Primary Site Access) or from Highway 107 from the North (Secondary Site Access).

Infrastructure that will be required for the open pit mining and mineral processing operations will include the following facilities:

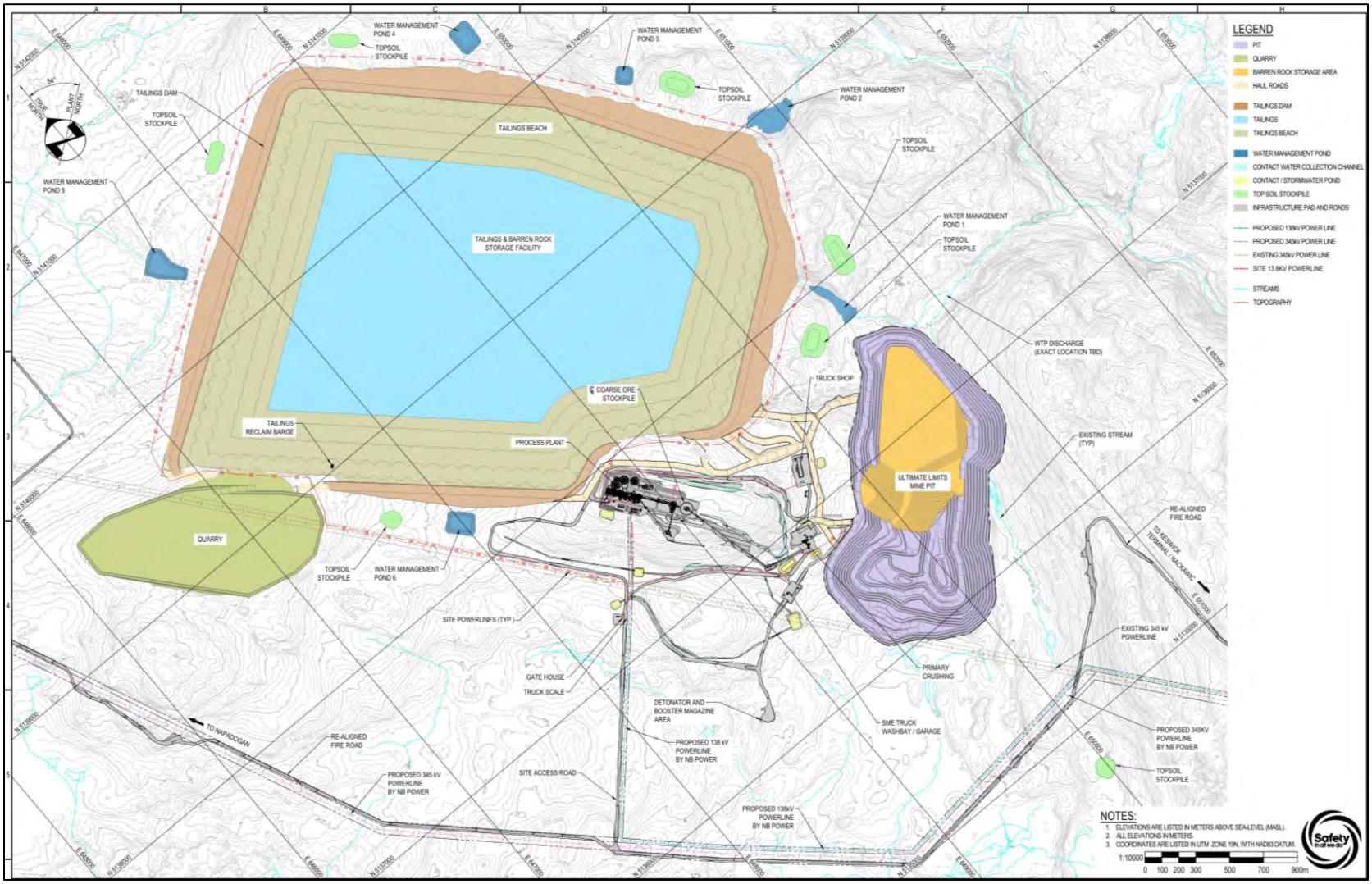
- mining facilities including truck shop, warehouse, offices, truck wash bay and garage, detonator and booster magazine, fuel storage and distribution;
- mineral processing facilities including process plant, crushing facilities, warehouse and maintenance shop, assay laboratory, freshwater infrastructure;
- quarry for the production of non-PAG rock for tailings impoundment;
- water treatment plant;

- tailings storage facility (TSF);
- barren rock storage facility;
- general facilities including administration building, change house, potable water treatment plant, sewage treatment plant, gatehouse, weigh scale; and
- stormwater and contact water ponds, diversion ditches, and other site water management infrastructure.

The site facilities are generally arranged in accordance with the layout developed for the 2013 Feasibility Study (FS), which formed the basis of the approved Project Development Area (PDA) and Environmental Impact Assessment (EIA) approval. Consequently, the current site layout closely reflects the configuration established in the 2013 FS, with only minor refinements incorporated where required.

An overall site layout is provided in Figure 1-1.

Figure 1-1: Overall site layout



Source: Ausenco, 2026

1.14.2 Access

The Project will be accessed through a series of unpaved existing public and forestry roads and new Project-specific access roads. Consistent design criteria have been applied to both upgraded and newly constructed roads to safely accommodate construction and operational traffic while maximizing the use of existing infrastructure to minimize disturbance, environmental impacts, and capital costs. Upgrades to existing roads will include localized widening, geometric improvements, resurfacing, drainage enhancements, culvert upgrades, subgrade improvements, erosion protection, and safety improvements. Existing bridges along the access routes will be assessed to confirm their suitability for Project traffic.

The access road network comprises:

- Primary Site Access (PSA) Road – 39.2 km of existing road to be retained and upgraded through resurfacing, drainage improvements, widening, and geometric corrections.
- Secondary Site Access (SSA) Road – 12.9 km of existing and proposed route sections to be upgraded or reconstructed where required.
- Re-Aligned Fire Road – 11 km of new road construction with a minimum 7 m running-surface width.
- Site Access Road – 3 km of new Project access road connecting the external road network to the Sisson Mine site.

In total, the Project road network includes approximately 66.1 km of access roads, consisting of upgrades to 52.1 km of forestry roads and construction of 14 km of new gravel roads.

1.14.3 Water management

The objective of surface water management is to protect groundwater and surface water resources. The surface water management system has been designed to maintain natural drainage patterns where practical, protect project infrastructure, control erosion and sediment transport, and safeguard downstream water quality throughout construction, operations, and closure. The system accounts for local climatic and hydrological conditions, including extreme rainfall events, spring freshet, snowmelt, freeze-thaw cycles, and rain-on-snow events that are characteristic of New Brunswick. An integrated site-wide drainage strategy will separate contact water from non-contact water.

Contact water and non-contact water will be managed in accordance with applicable permits, approvals, and regulatory requirements. The water management objectives are to divert non-contact runoff from undisturbed catchments around project facilities and routed through stormwater ponds that provide flow attenuation, sediment settlement, and erosion protection before discharge to the receiving environment.

Runoff from the process plant, ore handling facilities, stockpiles, and other potentially affected areas will be managed as contact water and collected in dedicated lined contact-water ponds for storage, reuse, or treatment, as required.

1.14.4 Water treatment plant

The Water Treatment Plant (WTP) will be located north of the process plant facility near the tailings pond and will receive mine-impacted water from the tailings storage facility via a dedicated pipeline. The WTP will support both operations and post-closure active care and will be designed for a treatment capacity of 2,083 m^3/h. The treatment process is based on High Density Sludge (HDS) technology for the removal of metals, bicarbonates, and total suspended solids. Lime and ferric sulphate will be used to promote metals precipitation, while hydrogen peroxide will be added to oxidize thiosalts to sulfate. Generated solids will be flocculated, settled in a clarifier, and transferred to the tailings thickener for co-disposal with tailings.

Following HDS treatment, the treated water will undergo pH neutralization using sulphuric acid prior to discharge to Sisson Brook, with a small portion recycled to the Mineral Processing facility. Neutralization will support compliance with effluent pH requirements, reduce the potential for downstream calcite scaling, and decrease the fraction of un-ionized ammonia. The WTP will be housed within a centralized WTP building with the majority of equipment located indoors. Large tanks and clarifiers will be located outdoors and designed for the site climate conditions.

1.14.5 Tailings storage facilities

A tailings storage facility (TSF) will manage tailings and barren rock, including the subaqueous disposal of potentially acid-generating materials to mitigate the onset of acidic conditions. The TSF comprises an approximately 8 km long ring-dam structure located adjacent to the process plant and open pit. The TSF is situated in the headwaters of Bird Brook and other small tributaries to West Branch Napadogan Brook and therefore does not require extensive surface water diversions. The TSF is the primary water management facility for the Project. Water will be impounded in the TSF during the pre-production period for mill start-up. The water balance indicates that the Sisson Project will operate in a surplus condition over the 27-year mine life.

The TSF embankment comprises of zoned earthfill and rockfill materials. Rockfill for the downstream shell of the dam will be sourced from a dedicated rock quarry, partially located within the northwest quadrant of the TSF along the watershed divide between McBean Brook and West Branch Napadogan Brook. Material for the upstream shell will be sourced from barren rock generated during mining. Thickened tailings will be selectively deposited from the crest of the embankments to develop tailings beaches which function as an extensive low permeability zone to mitigate seepage through the embankments. The TSF reclaim water system will pump water from the TSF supernatant pond to the process plant for reuse. Surplus barren rock will be stored within the Barren Rock Storage Area (BRSA) situated towards the south of the TSF and will be progressively encapsulated by tailings and water.

1.14.6 Power supply and distribution

Power for the Project will be supplied by NB Power under a negotiated 58 MW power supply arrangement. Electrical service will be delivered to the Project substation through a new 42 km, 138 kV transmission line extending from NB Power's Keswick Terminal. Upgrades will be completed at the Keswick Terminal to support the new connection and ensure reliable delivery of power to the Project.

To accommodate mine development, approximately 10 km of the existing 345 kV transmission line that currently crosses the property will be relocated to provide a safe setback from the open pit.

The site power reticulation system will distribute electrical power through a network of 13.8 kV overhead lines originating from the process plant's 13.8 kV switchgear at the outdoor substation.

Emergency power for process plant critical loads will be provided by on-site diesel-powered generators. The emergency generators will be connected to the 13.8 kV switchgear.

The total connected load for the Project will be 81 MW with a normal operating load of 59 MW. Operational restrictions have been applied to maintain a maximum load requirement of 58 MW power supply by NB Power.

1.14.7 Concentrate transportation

Tungsten and molybdenum concentrates will be packaged in 2-tonne bulk bags at the mine site and transported primarily to customers in the United States, with targeted smelters located in Ohio, Iowa, and Pennsylvania. Conventional highway trucking will be used for North American deliveries, while a portion of production may be directed to European customers in Austria and Germany, subject to market conditions and smelter capacity. For European shipments, concentrates will be loaded into containers at the mine and transported via the Port of Saint John for ocean shipment to major European ports.

Transportation costs for direct trucking to U.S. customers are estimated to range from C$201.55/t to C$264.25/t to Tungsten smelters, and C$264 to C$358/t to the farther Molybdenum smelters. These costs include bags and pallets as consumables.

Transportation costs for shipment to a European port (Austria / Germany customers), on a CIF basis for pick up by the off-taking party, range from C$128/t to C$141/t, inclusive of bags and pallets as consumables. These costs also include container rental, drayage and terminal/port fees.

Seasonal road access constraints, including a potential winter shutdown period of six to eight weeks and reduced spring thaw road capacity, will be managed through on-site concentrate stockpiling to ensure continuity of product delivery.

1.15 Markets and contracts

An independent market study completed by Argus Media indicates positive long-term market fundamentals for both tungsten and molybdenum concentrates. Demand for tungsten is forecast to increase significantly through 2040, driven by industrial manufacturing, mining equipment, aerospace, electrification technologies, and particularly growth in defense-related applications. Global tungsten demand is projected to increase from approximately 90,000 t in 2026 to 129,000 t by 2040.

The tungsten market continues to be characterized by concentrated global supply and significant dependence on Chinese production. Export controls implemented by China in 2025, combined with declining domestic mine production and growing strategic concerns regarding supply security, have increased the importance of new tungsten supply from stable jurisdictions. Tungsten has been designated a critical mineral in several jurisdictions, including Canada, the United States, and the European Union, supporting long-term demand for non-Chinese supply sources.

Forecast tungsten concentrate prices average approximately US$2,215/MTU in 2026, declining to long-term levels near US$1,880/MTU by 2040. The pricing outlook reflects ongoing supply constraints, geopolitical considerations, and increasing defense-sector demand. For economic evaluation purposes, the 2040 forecast price was maintained throughout the remaining life of mine.

Molybdenum demand is expected to grow from approximately 305,000 t in 2025 to 432,000 t by 2040, supported by demand from stainless and alloy steel production, transportation, energy infrastructure, petrochemical applications, and defense industries. Supply growth is expected to remain constrained due to the predominance of molybdenum as a by-product of copper mining operations and declining grades at several major deposits.

For the economic analysis, molybdenum concentrate prices are based on the Argus forecast for 45% molybdenum concentrate, increasing to US$744/MTU (US$33.75/lb) in 2026 and stabilizing at approximately US$627/MTU (US$28.44/lb) by 2040. The 2040 forecast price was applied for the remainder of the Project life.

The proposed tungsten and molybdenum concentrate products are expected to meet established industry specifications and are considered readily marketable. Potential customers include tungsten processors, ammonium paratungstate producers, carbide manufacturers, specialty metals producers, molybdenum roasters, alloy steel producers, and strategic consumers seeking secure non-Chinese supply sources. Europe and North America are considered particularly attractive markets due to supply diversification initiatives and reliance on imported tungsten units.

Offsite costs assumed in the study include average tungsten concentrate transportation costs of approximately US$264/t concentrate, together with a 0.15% deduction for handling, insurance and metal losses. Molybdenum treatment and refining charges were estimated at 12% of contained molybdenum value.

No concentrate offtake, transportation, reagent supply, utility, or other material contracts are currently in place. Any future agreements are expected to be negotiated under customary commercial terms consistent with industry practice.

Based on the independent Argus market assessment, there is reasonable evidence that future tungsten and molybdenum concentrate production from the Project can be marketed and sold into established global markets at commercially achievable terms, supporting the economic assumptions used in this Technical Report.

1.16 Environmental, permitting and social considerations

1.16.1 Environmental considerations

The Project site is on provincial Crown land southwest of the community of Napadogan, New Brunswick, and is approximately 60 km directly northwest of the city of Fredericton. The Project lies within the traditional territory of Wolastoqiyik. The Project area is sparsely populated and relatively undeveloped, with a long history of natural resource development and use. While the area is important for resource-based economic activities, it is also important for hunting, fishing, and outdoor recreation undertaken by the people of New Brunswick.

Many baseline environmental studies have been conducted for the Project, and the available baseline technical reports include ambient air quality, sound quality, aquatics, wildlife and wildlife habitat, vegetated and wetland environments, metal concentrations in soil and biota, socioeconomics, heritage resources, hydrogeology, water quality, and archaeology. These are described in more detail in Section 20.

Environmental monitoring will be governed by the Project's Environmental and Social Management System and Environmental Management Plan, planned to be completed in 2027. An Environmental Monitoring Plan will also be developed to meet regulatory requirements. An Environmental Protection Plan will be developed to guide construction of the Project in accordance with applicable legislation and regulations, and will include Project-specific environmental protection, mitigation, and response measures.

1.16.2 Permitting considerations

A Final Environmental Impact Assessment (EIA) Report was submitted in February 2015 to meet the requirements of the provincial EIA Regulation (Stantec Consulting Ltd., 2015). The Project received EIA approvals from the provincial government in 2015 (NBDELG 2015), and from the federal government in 2017 (Government of Canada 2017). Additional information is provided in Section 20, including a summary of the results of the Final EIA Report.

The provincial EIA approval included 40 conditions of approval, and Northcliff is currently working to fulfill these. Of the 40 conditions of approval, there are 28 that must be completed before the start of construction. The remaining 12 conditions of approval are tied to operations and compliance. The conditions of approval include major permits such as approval to construct, air quality approval to operate and water quality approval to operate. There have been several extensions of the Project's EIA approval, the most recent was issued in November 2025, extending the approval to December 3, 2030. There are numerous other approvals, authorizations, and permits that are required prior to initiating construction (see Section 20). Compliance with terms and conditions of approvals, standards contained in federal and provincial legislation and regulations, and commitments made during the environmental assessment processes (including application of mitigation measures and monitoring and follow-up requirements), will need to be assured throughout all Project phases.

1.16.3 Social considerations

Northcliff is committed to engaging with Indigenous communities, the public, and stakeholders in an open, transparent, and responsive manner in respect of the Sisson Project. Northcliff established a public, stakeholder and Indigenous engagement program concurrent with the EIA Report process to develop positive, long-term relationships and to facilitate the success of the Project. Additional information about engagement with First Nations communities, public engagement, and public participation activities undertaken by Sisson is provided in Section 20.

1.16.4 Closure and reclamation considerations

Following completion of mining, decommissioning, reclamation and closure of the Sisson Project will be undertaken to establish the physical, chemical and biological stability of the site, to the extent practicable, and to meet desired end land uses, in accordance with applicable provincial and federal legislation and regulations.

Financial security is required by the Province of New Brunswick to ensure acceptable closure of the Project. The required security will increase over the life of the mine to an estimated value of $210 million at closure in Year 27, in today's dollars. The estimated closure costs are based on the conceptual-level closure and reclamation plan developed for the Environmental Assessment (EvEco, 2013).

1.17 Capital and operating cost estimates

1.17.1 Capital cost estimate

The capital cost estimate was developed in Q2-2026 CAD based on budgetary quotations for equipment and construction contracts, as well as Ausenco's in-house database of projects and studies including experience from similar operations.

The estimate includes mining, crushing, processing, on-site infrastructure, tailings storage facility and water management, off-site infrastructure, project indirect costs, project delivery, owners' costs, and contingency.

The following parameters and qualifications were considered:

- No allowance has been made for exchange rate fluctuations.
- There is no forward escalation added to the estimate.
- A growth allowance has been included.

Data for the estimates have been obtained from numerous sources, including:

- mine schedules;
- engineering design and cost estimates by Ausenco, MMTS, KP, and ISC;
- Facility Study cost estimate for the power transmission lines by NB Power;
- topographic information obtained from the site survey;
- geotechnical investigations;
- budgetary equipment quotes from suppliers based in North America;
- budgetary unit costs from several local contractors for civil, concrete, steel, electrical, piping, and mechanical works; and
- data from similar recently completed studies and projects.

Major cost categories (permanent equipment, material purchase, installation, subcontracts, indirect costs, and Owner's costs) were identified and analyzed. A contingency was applied in the cost estimate and was based on ranging the accuracy of the data by discipline and WBS level 3 and applying a probabilistic method (Monte Carlo Simulation). An overall contingency amount was derived in this fashion.

The capital cost summary is presented in Table 1-2 by WBS and Table 1-3 by discipline. The initial capital cost for the Sisson Project is $1,528 million and the sustaining capital cost is $571 million; resulting in a total capital cost of $2,099 million.

Table 1-2: Capital cost estimate summary by WBS Level 1

WBS LVL 1 Description	WBS LVL 1	Initial Cost (C$M)	Sustaining Cost (C$M)	Total Cost (C$M)
Mining	1000	193.3	215.4	408.7
Crushing	2000	210.6	-	210.6
Process Plant	3000	248.6	1.7	250.3
Tailings Storage Facility & Water Management	5000	118.3	256.0	374.3
Utilities	6000	71.7	-	71.7
On-Site Infrastructure	7000	114.2	-	114.2
Off-Site Infrastructure	8000	125.3	-	125.3
Total Directs		**1,082.0**	**473.2**	**1,555.2**
Indirect Costs & Contingency	9000	445.6	97.9	543.6
Total Indirects		**445.6**	**97.9**	**543.6**
Project Total		**1,527.7**	**571.1**	**2,098.8**

Note: Values may not sum due to rounding. Total direct and indirect costs may differ between presentations by WBS and by discipline due to differences in cost allocation methodology. Aggregate values are consistent with those disclosed in the August 31, 2026 press release.

Table 1-3: Capital cost estimate summary by discipline

Disc.	Discipline Description	Initial Cost (C$M)	Sustaining Cost (C$M)	Total Cost (C$M)
B	Earthworks	46.5	-	46.5
C	Concrete	67.5	-	67.5
S	Structural Steelwork	41.2	-	41.2
A	Architectural	65.3	-	65.3
F	Platework	44.9	-	44.9
M	Mechanical Equipment	235.7	1.7	237.4
P	Piping	58.5	-	58.5
E	Electrical Equipment	72.1	0.06	72.1
L	Electrical Bulks	40.0	-	40.0
I	Instrumentation	16.6	-	16.6
R	Third-party Packages	408.1	477.3	885.4
	Total Direct Costs	**1,096.4**	**479.0**	**1,575.5**
U	Field Indirects	53.5	44.9	98.3
O	Owner's Costs	31.4	-	31.4
V	Other (Spares, Fills, Vendors)	33.6	-	33.6
T	Project Delivery	112.7	5.1	117.8
Y	Contingency	200.1	42.1	241.9
	Total Indirect Costs	**431.3**	**92.1**	**521.8**
Project Total		**1,527.7**	**571.1**	**2,098.8**

Note: Values may not sum due to rounding. Total direct and indirect costs may differ between presentations by WBS and by discipline due to differences in cost allocation methodology. Aggregate values are consistent with those disclosed in the August 31, 2026 press release.

1.17.2 Operating cost estimate

The operating cost estimates use Canadian dollars as the base currency and are reflective of Q2 2026 pricing, with the exception of mining operating costs and fuel rates which are reflective of Q4 2025. Allowance for inflation has not been included. An average operating cost was estimated for the Sisson Project based on the proposed mining schedule. Operating costs include the ongoing cost of operations related to mining, processing, quarry development, tailings disposal, water treatment, and general administration activities.

The average LOM unit operating cost for the Sisson Project is estimated to be C$23.09/t milled. A summary of the operating costs is presented below in Table 1-4.

The peak operational workforce is estimated at 352 employees, including personnel associated with general and administrative (G&A), mining, process plant, and water treatment plant (WTP) functions.

Table 1-4: Operating cost summary

Description	C$M/a (LOM Average)*	C$/t milled (LOM Average)
Mining	77.8	7.50
Process	131.7	12.72
Quarry	8.7	0.83
G&A	12.0	1.17
Tailings	0.7	0.07
WTP	8.1	0.79
Total	**239**	**23.09**

Note: Values may not sum due to rounding.
* Production-weighted average operating cost over life of mine

1.18 Economic analysis

The economic analysis was performed assuming an 8% discount rate. On a pre-tax basis the net present value (NPV) discounted at 8% is C$12,290 million; the internal rate of return (IRR) is 68.3%, and payback period is 1.3 years. On a post-tax basis, the NPV discounted at 8% is C$6,915 million; the IRR is 49.8%, and the payback period is 1.6 years. The analysis was done on an annual cashflow basis; the cashflow output is shown graphically in Figure 1-2 and summarized in Table 1-5.

Table 1-5: Economic analysis summary

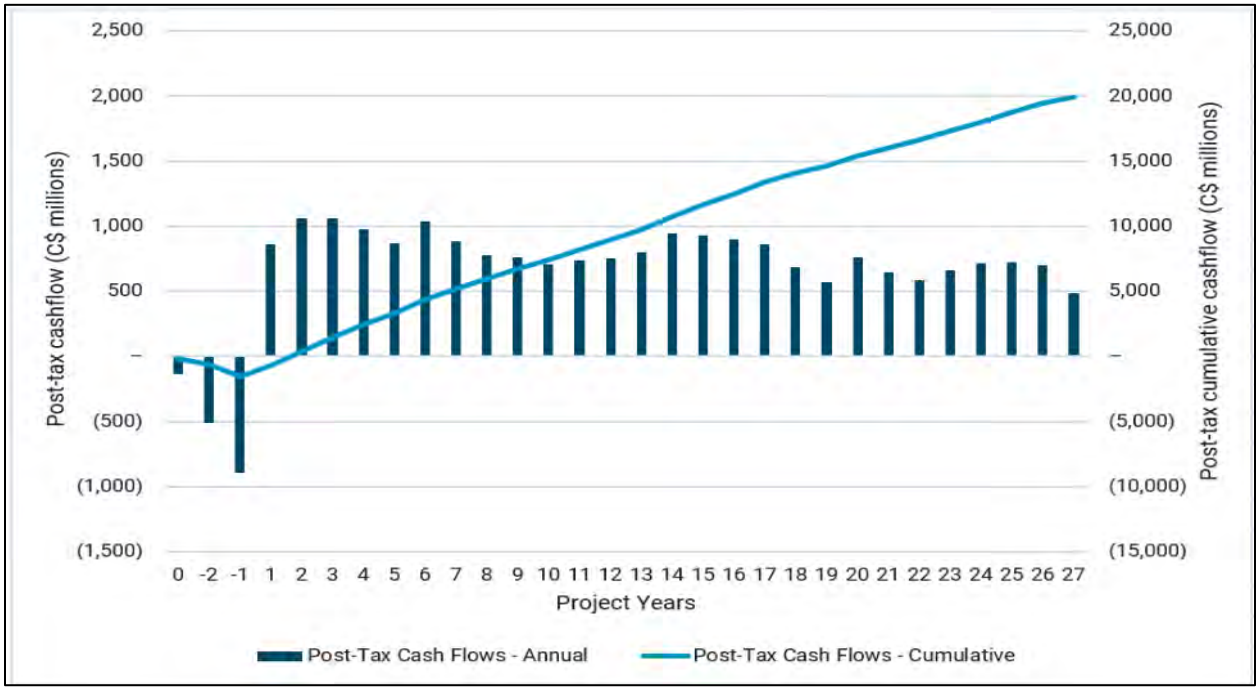
Economic Summary	Units	LOM	Annual Average	Years 1 to 5 Average
Metal Price - Tungsten (Concentrate)(1)	US$/MTU WO_3*	1,821		1,634
Metal Price - Molybdenum(1)	US$/lb	28.46		28.51
Foreign Exchange Rate	C$:US$	1.35		1.35
Production				
Mining	Mt	638	22	18.5
Milling	Mt	277	10.3	10.2
Strip Ratio	waste:ore	0.97		0.47
Tungsten (WO_3) Grade	%	0.0747%		0.096%
Molybdenum Grade	%	0.0227%		0.0217%
Tungsten (WO_3) Recovery	%	78.0%		78.0%
Molybdenum Recovery	%	82.0%		82.0%
Tungsten (WO_3) Production	MMTU WO_3*	16.1	0.598	0.767
Molybdenum Production	Mlbs	113.5	4.2	4
Gross Revenue	C$M	43,699	1,618	1,845
Operating Costs				
Total Operating Costs	C$M	6,391	239	221

Economic Summary	Units	LOM	Annual Average	Years 1 to 5 Average
Operating Costs - Mining	$/t milled	7.50		6.52
Operating Costs - Milling	$/t milled	12.72		12.84
Operating Costs - Waste Management	$/t milled	1.69		1.04
Operating Costs - G&A	$/t milled	1.17		1.21
Total operating costs to concentrate	$/t milled	23.09		21.62
Operating Costs - Less: Molybdenum by-product credit	$/t milled	(13.85)		(13.26)
Total operating cash costs to concentrate, net of by-product credits[2]	$/t milled	9.23		8.35
Concentrate All-in Sustaining Costs, net of by-product credits[3]	$/t milled	14.24		14.21
Total operating costs to concentrate	$/MTU WO_3*	396		289
Operating Costs - Less: Molybdenum by-product credit	$/MTU WO_3*	(238)		(177)
Total operating cash costs to concentrate, net of by-product credits[2]	$/MTU WO_3*	158		112
Concentrate All-in Sustaining Costs, net of by-product credits[3]	$/MTU WO_3*	244		190
Total operating costs to concentrate	US$/MTU WO_3*	293		215
Operating Costs - Less: Molybdenum by-product credit	US$/MTU WO_3*	(176)		(130)
Total operating cash costs to concentrate, net of by-product credits[2]	US$/MTU WO_3*	117		85
Concentrate All-in Sustaining Costs, net of by-product credits[3]	US$/MTU WO_3*	181		143
Capital and Other Costs				
Initial Capital Costs	C$M	1,528	na	na
Sustaining Capital Costs	C$M	571	21	29
Reclamation Funding	C$M	147	5	6
Taxes and Royalties	C$M	14,499	537	545
Pre Tax Economics				
NPV 8, pre-tax (calculated at mid-year)	C$M	12,290	-	-
IRR, pre-tax (calculated at mid-year)	%	68.3%	-	-
Payback, pre-tax (years of production)	year(s)	1.3	-	-
Post Tax Economics				
Post-Tax Cash Flows	C$M	19,896	686	965
NPV 8, post-tax (calculated at mid-year)	C$M	6,915	-	-
IRR, post-tax (calculated at mid-year)	%	49.8%	-	-
Payback, post-tax (years of production)	year(s)	1.6	-	-

Notes:
* MTU = metric tonne unit = 1% of a metric tonne = 10kg

** reclamation funding over construction and operations period of 29 years combined.
1) Metal prices for the economic analysis were applied based on Argus' independent market study forecasts on an annual basis until 2040 and thereafter the 2040 forecast prices were applied for the remainder of the Project life.
2) Operating cash costs includes mine, mill, waste management and G&A costs less by-product credits .
3) All-in sustaining cost includes operating cash cost, offsite costs, reclamation costs and sustaining capital costs.
Values may not sum due to rounding.
Source: Northcliff, 2026

Figure 1-2: LOM post-tax free cash flow



Source: Northcliff, 2026

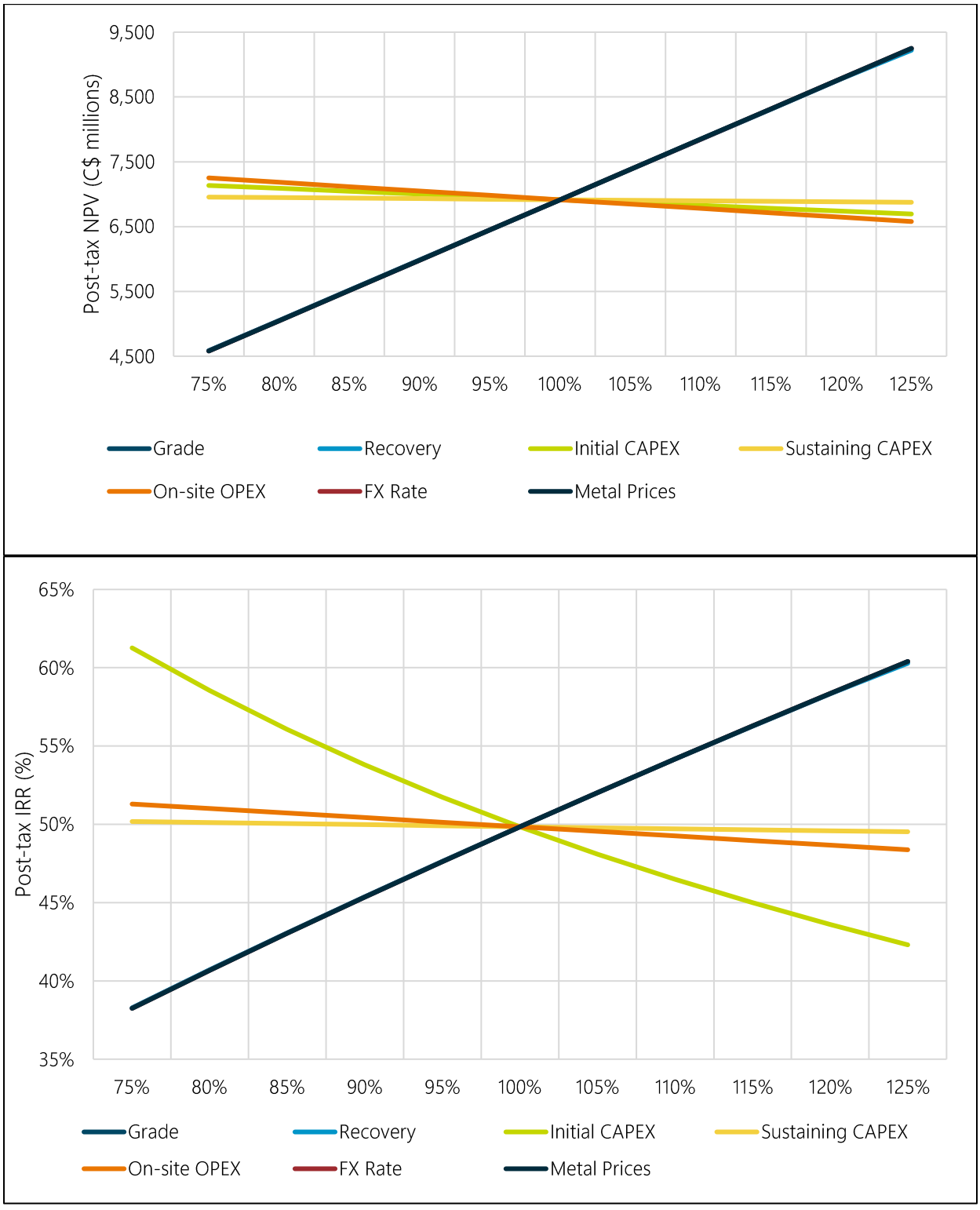
1.18.1 Sensitivity analysis

A sensitivity analysis was conducted on the base case pre-tax and post-tax NPV and internal rate of return of the project, using the following variables: commodity prices, discount rates, operating costs, initial capital costs, sustaining capital costs, metal recovery, head grade and exchange rate.

A summary of post-tax economic sensitivities to commodity prices, operating costs, initial capital costs, sustaining capital costs, metal recovery, head grade and exchange rate is shown in Figure 1-3.

The project is most sensitive to changes in foreign exchange rate, commodity price, head grade and recovery; it is less sensitive to total operating cost, sustaining capital cost and initial capital cost.

Figure 1-3: Post-tax sensitivity summary



Note: Recovery, head grade, metal price and FX rate overlap. Source: Northcliff, 2026

1.19 Conclusions and recommendations

The Sisson Project shows positive economics, as demonstrated by the results presented in this Technical Report. It is recommended that the Project be advanced through to the basic engineering phase. The following sections detail the recommended future work for the Project. The estimated costs are summarized in Table 1-6.

Table 1-6: Recommendations cost summary

Program component	Estimated total cost (C$M)
Geology and mineral resource	
• Geology and mineralization	0.05
• Mineral Resource	0.15
Mine planning	
• Barren Rock and PAG Characterization	0.3
• Mine geotechnical drilling	0.0*
• Mine planning and trade-off studies	0.4
• Operational readiness and cost validation	0.35
Metallurgical testing and process recovery	
• Metallurgical testing and process recovery	0.95
• Particle Ore Sorting	0.2
Infrastructure	
• Site access, logistics, and targeted geotechnical investigation	1.3
• Site optimization and value engineering	0.35
• Fresh water and potable water supply	0.45
• Power supply and electrical infrastructure development	0.75
• Tailing management facility	2.9
• Water treatment facilities	0.8
Project execution and procurement	
• Procurement, contracting, and long-lead equipment strategy	0.4
• Early works definition and execution readiness	0.55
Environmental studies	1.0
Total	**10.9**

*No cost is shown because the geotechnical program is currently underway.

1.19.1 Sisson project execution plan

The Sisson Project Execution Plan (PEP) establishes the FS-level execution basis, delivery strategy, and governance framework for advancing the Sisson Project from the Feasibility Study (FS) stage through subsequent engineering and execution phases. The PEP covers the plan for engineering, procurement, construction, and commissioning of the Project and identifies key interfaces, constraints, and execution priorities to support effective project delivery. The Project is currently planned to be implemented using an

engineering, procurement, construction management (EPCM) delivery model, subject to subsequent Project authorization and commercial arrangements.

A peak construction workforce of approximately 500 direct workers has been estimated based on benchmark data from comparable projects and Ausenco's experience in delivering similar projects. Construction methodologies, workforce requirements, and execution planning will be further developed during the subsequent phase of the Project.

2 Introduction

2.1 Overview

Northcliff Resources Ltd. ("Northcliff", the "Company", or the "Owner") commissioned Ausenco Engineering Canada ULC (Ausenco) to manage the preparation of a Feasibility Study Update (FSU) and associated technical report for the Sisson Tungsten-Molybdenum Project (the "Sisson Project", the "Project", or the "Property") located in New Brunswick, Canada (the "Technical Report"). The Technical Report was prepared in accordance with the Canadian disclosure requirements of National Instrument (NI) 43-101 – Standards and Disclosure for Mineral Projects (NI 43-101) and the requirements of Form 43-101 F1.

The responsibilities of the engineering consultants and firms contracted by Northcliff to prepare this Technical Report are as follows:

- Ausenco was responsible for managing and coordinating the work related to the Technical Report. Ausenco managed the metallurgical testwork program, developed the FS-level design, capital, and operating costs estimates for the process plant and general site infrastructure. Ausenco compiled the overall capital cost estimate and is also responsible for auditing the economic analysis.
- SLR Consulting (Canada) Ltd. (SLR) prepared the Mineral Resource Estimate (MRE) for the Project and was responsible for completing the work related to the property and geological setting, deposit type, drilling, exploration works, sample preparation and analysis. SLR acquired RPA Consulting (RPA) in 2019 and references to RPA in this Technical Report should be viewed in this context.
- Moose Mountain Technical Services (MMTS) was responsible for completing the reserve estimate, open pit mine design, quarry design, mine production schedule, mine related infrastructure, and mine and quarry capital and operating cost estimates.
- Knight Piésold Consulting (KP) was responsible for completing the tailings storage facility, water management design, site wide water balance, TSF and water management capital and operating cost estimates, and closure costs.
- Integrated Sustainability Consultants Ltd. (ISC) was responsible for water treatment plant design and cost estimate and WTP closure costs.
- Stantec Consulting Ltd. (Stantec) was responsible for completing environmental and permitting works.

2.2 Terms of reference

The purpose of this Technical Report is to present the results of the FSU and to support the disclosure by Northcliff in a news release dated August 31, 2026, entitled "Northcliff Announces Results of Feasibility Study Update for Sisson Project."

All measurement units used in this Technical Report are SI units unless otherwise noted. Currency is expressed in Canadian dollars (C$ or CAD) unless otherwise noted. This Technical Report has been prepared using Canadian English spelling conventions. Metric tonne units (MTU) are used to express contained tungsten trioxide (WO_3), with 1 MTU equivalent to 10 kg of contained WO_3.

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

2.3 Qualified persons

The Qualified Persons (QPs) for this Technical Report are listed in Table 2-1. By virtue of their education, experience, and professional association membership, they are considered QPs, as defined in National Instrument 43-101, Standards of Disclosure for Mineral Projects, and in compliance with Form 43-101 F1.

Table 2-1: Report contributors

Qualified person	Professional designation	Position	Employer	Independent of Northcliff Resources Ltd
Kevin Murray	P.Eng.	Principal Process Engineer	Ausenco	Yes
Jennifer Lynn McPhail	P.Eng.	Principal	Stantec	Yes
Daniel Friedman	P.Eng.	Specialist Civil Engineer	Knight Piésold	Yes
Robert Gray	P.Eng.	Manager Engineer for Civil and Mining Projects	MMTS	Yes
AJ MacDonald	P.Eng.	Vice President Engineering	ISC	Yes
Paul Chamois	M.Sc.(A), P.Geo.	Associate Principal Geologist	SLR	Yes
Philip A. Geusebroek	M.Sc., P.Geo.	Resource Geologist	SLR	Yes

2.4 Personal inspections

2.4.1 Personal inspection details by Jennifer Lynn McPhail, P.Eng.

Jennifer Lynn McPhail, P.Eng., Principal with Stantec, visited the site on September 8, 2026. During her inspection, Ms. McPhail inspected general environmental existing conditions in the Project area. Previously, she had visited the site on multiple occasions during the environmental assessment process, in approximately 2011 to 2013, to collect environmental samples and observe existing environmental conditions.

2.4.2 Personal inspection details by Paul Chamois, M.Sc.(A), P.Geo.

A site visit to the Property was carried out by Paul Chamois, M.Sc.(A), P. Geo., SLR Associate Principal Geologist, on June 9, 2022. During the visit, Mr. Chamois examined drill core from historical drilling programs, confirmed the local geological setting, assessed logistical aspects related to access and conducting work in the area, and investigated factors that might affect the Project.

2.4.3 Personal inspection details by Robert Gray, P.Eng.

Robert Gray visited the Property on September 18, 2025. During the site visit he observed the general topography of the planned open pit, mine haul roads, ore and barren rock storage areas, process plant, as well as the natural water features and existing site access roads throughout the site. During the site visit he also observed the historical test pit and examined drill core at a nearby storage facility.

2.4.4 Personal inspection details by Kevin Murray, P.Eng.

Kevin Murray, P.Eng., Principal Process Engineer with Ausenco, visited the site on September 18, 2025. During the site visit he observed the general topography of the planned open pit, mine haul roads, ore and barren rock storage areas, process plant area, as well as the natural water features and existing site access roads throughout the site and existing electrical transmission line. During the site visit he also observed the historical test pit and examined drill core at a nearby storage facility.

2.4.5 Personal inspection details by Daniel Friedman, P.Eng.

Daniel Friedman, P. Eng., visited the Sisson Project site on September 13, 2011. The purpose of the site visit was to gain a firsthand understanding of the site characteristics. Geotechnical site investigation work was ongoing at the time of the visit, which allowed a brief review of the equipment and procedures, as well as some of the samples and test results.

2.5 Effective dates

The Technical Report contains the following effective dates:

- Effective date of Mineral Resource Estimate: December 31, 2025.
- Effective date of Mineral Reserve Estimate: August 11, 2026.
- Effective date of Financial Model: August 21, 2026.
- Effective date of the Technical Report is: September 23, 2026.

2.6 Information sources and references

2.6.1 Previous reports

Northcliff has filed the following technical reports on the Project:

- Technical Report on a Mineral Resource Update, Sisson Tungsten-Molybdenum Deposit, York County, New Brunswick, by Michael P. Cullen and Matthew Harrington. Report prepared by Mercator Geological Services Limited for Northcliff. Effective date: February 4, 2011.
- Technical Report on the Sisson Project, New Brunswick, Canada, by David W. Rennie. Report prepared by RPA Inc. for Northcliff. Effective date: June 29, 2012.
- Canadian National Instrument 43-101 Technical Report on the Sisson Project, by David W. Rennie, Daniel Friedman, Jim Gray, Matt Bolu, Steven Pozder, & Gene Greskovich. Report prepared by Samuel Engineering, Inc. for Northcliff. Effective Date: January 22, 2013.

2.7 Currency, units, abbreviations and definitions

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

Table 2-2: Abbreviations and acronyms

Abbreviation	Description
AA	atomic absorption spectroscopy
AACE	Association for the Advancement of Cost Engineering International
AHS	Autonomous Haul System
Ar-Ar	$^{40}Ar/^{39}Ar$ (Argon-Argon isotopic dating method)
APT	ammonium paratungstate
Ausenco	Ausenco Engineering Canada ULC
Bi	Bismuth
BRSA	Barren Rock Storage Area
BSF	Bulk Sulfide Flotation
CAD:USD	Canadian-American exchange rate
CAGR	compound annual growth rate
CIM	Canadian Institute of Mining, Metallurgy and Petroleum
CIM Definition Standards	CIM Definition Standards for Mineral Resources and Mineral Reserves 2014
CoG	cut-off grade
CRMA	Critical Raw Materials Act
Cu	Copper
DDH	diamond drill hole
DTH	down the hole
EM	electromagnetic
EoP	end of period
E-HEX	electric-hydraulic excavator
FA	fire assay
FET	federal excise tax
FS	feasibility study
FSU	feasibility study update
HEX	hydraulic excavator
G&A	general and administration
GCVW	gross combined vehicle weight
Geodex	Geodex Minerals Ltd.
GPR	gross production royalty
HDI	Hunter Dickinson Inc.
HDPE	high-density polyethylene
IBCs	intermediate bulk containers
ICP-MS	inductively coupled plasma-mass spectrometry

Abbreviation	Description
ICP-OES	inductively coupled plasma - optical emission spectrometry
IFA	Issued for Approval
INAA	instrumental neutron activation analysis
IR	information request
ISC	Integrated Sustainability Consultants Ltd
ISO	International Organization for Standardization
JV	joint venture
Kidd Creek	Kidd Creek Mines Ltd.
KP	Knight Piésold Consulting
Lantech	Lantech Drilling Limited
LOM	life of mine
Maritime	Maritime Diamond Drilling Limited
MGSP	mid-grade stockpile
Mo	molybdenum
MMTS	Moose Mountain Technical Services
MRE	Mineral Resource Estimate
NAD 83	North American Datum of 1983
Nashwaak	Nashwaak Pulp and Paper Co.
NBDELG	New Brunswick Department of Environment and Local Government
NBDNR	New Brunswick Department of Natural Resources
NBDTI	New Brunswick Department of Transportation and Infrastructure
NI 43-101	National Instrument 43-101 (Regulation 43-101 in Quebec)
NSR	net smelter return
OK	ordinary kriging
PAG	potentially acid generating
PAX	potassium amyl xanthate
Pb	Lead
PEA	preliminary economic assessment
PFS	prefeasibility study
Progressive	Progressive Diamond Drilling
PSA	primary site access
PSJ	Port Saint John
QA/QC	quality assurance/quality control
QP	qualified person (as defined in National Instrument 43-101)
RC	reverse circulation

Abbreviation	Description
ROM	run-of-mine
RPA	RPA Consulting
RQD	rock quality designation
S	sulphur
SCC	Standards Council of Canada
SD	standard deviation
S_d-BWI	micro hardness or bond ball mill work index on SAG ground material
SG	specific gravity
SME	site mixed explosives
SMU	Service meter unit
Sn	tin
Sisson LP	Sisson Project Limited Partnership
SLR	SLR Consulting (Canada) Ltd
SSA	secondary site access
Stantec	Stantec Consulting Ltd
The Ministry	New Brunswick Ministry of Natural Resources and Energy Development
Todd	Todd Minerals Ltd.
TSF	tailings storage facility
TSS	total suspended solids
UG	underground
UTM	Universal Transverse Mercator coordinate system
UV	ultraviolet
VC	valued component
W	Tungsten
WBS	work breakdown structure
WO_3	Tungsten trioxide
Zn	Zinc

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 tonne
°	angular degree
°C	degree Celsius
µm	micron (micrometre)
cm	centimetre
cm^3	cubic centimetre
ft	foot (12 inches)
g	gram
g/cm^3	gram per cubic centimetre
g/L	gram per liter
g/t	gram per metric ton (tonne)
h	hour (60 minutes)
h/a	hours per year
ha	hectare
kg	kilogram
kg/t	kilogram per tonne
km	kilometre
km^2	square kilometre
kt	kilo tonnes
kt/a	kilo tonnes per year
kV	kilovolt
kW	kilowatt
kWh/t	kilowatt-hour per tonne
L	liter
lb	pound
m, m^2, m^3	metre, square metre, cubic metre
m^3/a	cubic metre per year
M	million
Ma	million years (annum)
masl	metres above mean sea level

Abbreviation	Description
mm	millimetre
Moz	million (troy) ounces
Mt	million tonnes
MTU	Metric tonne unit
MW	megawatt
M/a	million per year
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

3.1 Introduction

The QPs have relied upon the following other expert reports, which provided information regarding mineral rights, surface rights, property agreements, royalties, environmental, social and community impacts, taxation, and marketing for sections of this Technical Report.

3.2 Property agreements, mineral tenure, surface rights, and royalties

The QPs have not independently reviewed ownership of the Project area and any underlying property agreements, mineral tenure, surface rights, or royalties. The QPs have fully relied upon information derived from Northcliff and legal experts retained by Northcliff for this information through the following documents:

- Thomas, T, 2026: Letter to Paul Chamois dated September 23, 2026 on the title, mineral tenure, property agreements, surface rights and royalties Re: NI 43-101 Technical Report on Sisson Tungsten-Molybdenum Project 43-101 Technical Report & Feasibility Study Update New Brunswick, Canada", effective date August 31, 2026.

This information is used in Sections 1.4, 1.10, 4, 14, 25.2, and 25.3 of the Technical Report.

3.3 Environmental, Social and community impacts

The QPs have fully relied upon information supplied by Northcliff and One-Eighty Consulting for social and community impacts.

- Quigley, J., Ing., A., and Davidson, G., Letter to Jennifer McPhail, dated September 21, 2026 on social and community information Re: NI 43-101 Technical Report on Sisson Tungsten-Molybdenum Project 43-101 Technical Report & Feasibility Study Update New Brunswick, Canada", effective date August 31, 2026.

This information is used in Sections 1.16.3, 20.4, and 25.10 of the Technical Report.

The QPs have fully relied upon information supplied by Cultural Resource Management Group (CRM Ltd., 2026) on archaeological investigations and mitigation.

- CRM (Cultural Resource Management Group) Ltd. (2026). Letter to: Jennifer McPhail, dated February 20, 2026, Re: Sisson Project: Archaeological Summary for Feasibility Study.

This information is used in Sections 1.16.1, 20.1, and 25.10 of the Technical Report.

3.4 Taxation

The QPs have fully relied upon information supplied by experts retained by Northcliff for information related to taxation as applied to the financial model as follows:

- Peters, Mark, Letter to Kevin Murray, dated September 24, 2026, Re: NI 43-101 Technical Report on Sisson Tungsten-Molybdenum Project 43-101 Technical Report & Feasibility Study Update New Brunswick, Canada", effective date September 23, 2026, confirming that he reviewed the taxation model in:
- "NCF.Sisson.FSU26.finmodel.draft260820" Excel workbook received August 20th, 2026, from Simon Beller and reviewed by Mark Peters

This information is used in Sections 1.18, 22, and 25.13 of the Technical Report.

3.5 Markets

The QPs have not independently verified the marketing information. The QPs have fully relied upon information derived from Northcliff and experts retained by Northcliff for this information through the following documents:

- Argus Media, July 27, 2026, "Sisson Project Tungsten Market Outlook Study – Update," prepared for Northcliff.

Argus Media has a team of over 600 commodity market experts working with partners across multiple industries and markets to monitor commodity prices including tungsten and molybdenum. The qualified person believes it is reasonable to rely on this information as Argus Media have access to information which may not be publicly available. Risks included would be similar to any that can potentially impact commodity prices around the world including but not limited to market disruption, supply and demand, political and governmental changes.

This information is used in Sections 1.15 and 19 of the Technical Report. The information is also used in support of Sections 1.18, 22, and 25.13.

4 Property description and location

4.1 Introduction

The Property is located approximately 60 km northwest of Fredericton, New Brunswick (Figure 4-1). The claims lie within National Topographic Series map sheets 21J2, 3, 6, and 7. The approximate centre of the Property is at grid coordinates 5135500N, 650000E, within UTM Zone 19T (NAD 83).

Figure 4-1: Location map



Source: SLR, 2025

4.2 Land tenure

Tenure for the mineral rights is held via three contiguous claim groups comprising a total of 707 units totaling approximately 14,140 ha (Table 4-1) and are held by the Limited Partnership.

Todd acquired an initial 11.5% interest in the Limited Partnership for an initial contribution of C$14 million. Todd has the option to acquire an additional 10% interest in the Limited Partnership by investing an additional C$20 million upon a final investment decision to commence construction.

Northcliff owns a collective 88.5% interest in the Sisson Project Limited Partnership (Sisson LP), through its 100% interest in Sisson Resources Ltd. and Northcliff Holdings (Canada) Ltd. (88.49%), and through its 88.5% interest in Sisson Mines Ltd. (0.01%). Sisson Mines Ltd. is the general partner of the Limited Partnership (Figure 4-2). Sisson Mines Ltd. is an incorporated entity. Northcliff and all entities owned by within the Northcliff Group are in good corporate standing.

In New Brunswick, claims are staked online as blocks of units which measure 500 x 500 m each. The list of claim groups for the Property as per the New Brunswick Ministry of Natural Resources and Energy Development (the Ministry) web site (http://www.gnb.ca/) at the time of writing is provided in Table 4-1 and a claim map is provided in Figure 4-3. SLR accepted the data as reported and cannot guarantee that the information is either accurate or current.

Northcliff reported that the Sisson claims were originally staked prior to the implementation of the map-based system currently in place. The Sisson claims were converted to the current system on February 3, 2011.

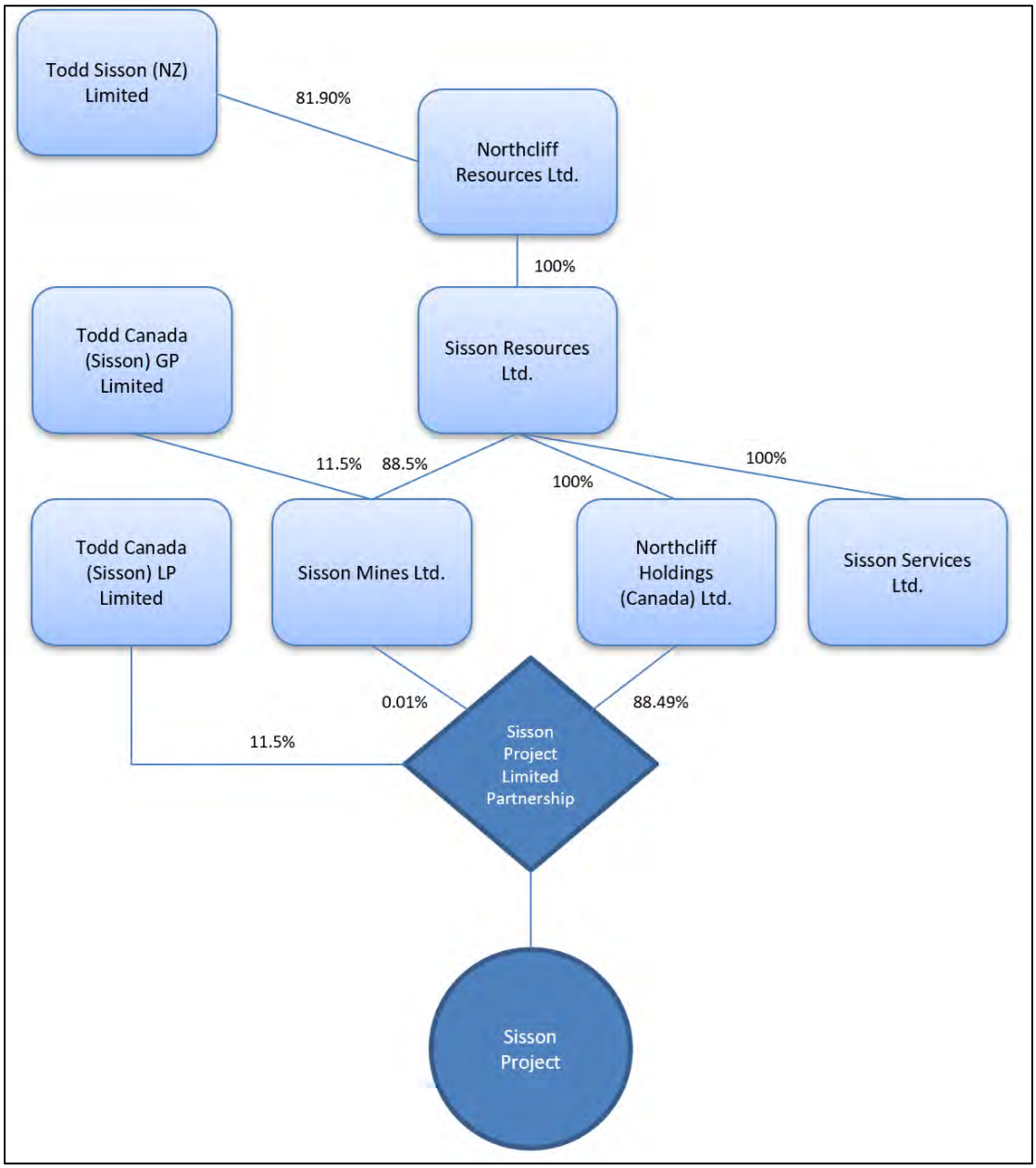
Northcliff provided a number of documents supporting the land tenure. These documents included a letter dated March 24, 2011, from the legal firm Barry Spalding of Saint John, New Brunswick, which confirmed the ownership of the Sisson claims. Also provided were copies of the digital confirmations of the most recent assessment work filed on the Sisson claims to maintain them in good standing.

The Sisson Mineral Resources are all located within Claim Group No. 3270. A total expenditure of $168,101.60 is required to retain the claims in 2026. There are no underlying royalties or back-in right on the Project. Sisson LP is the holder of the Sisson claims.

Table 4-1: Mineral claims list

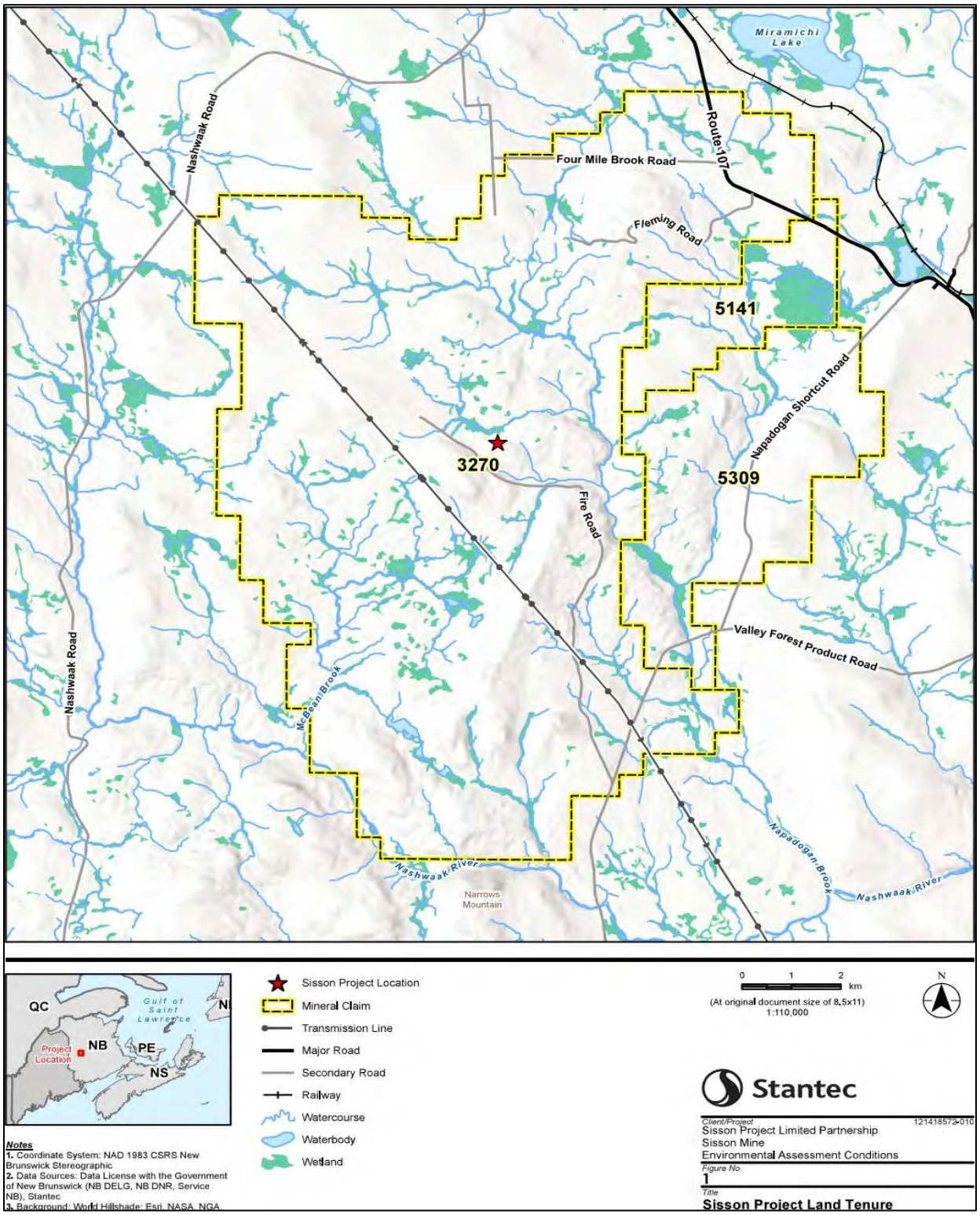
No.	Mineral Claim Name	Mineral Claim Type	Mineral Claim Sub Type	Issue Date	Expiry Date	Status	Units
5141	Turnbull Mountain	Mineral	Claim	June 14, 2007	June 14, 2027	Active	40
5309	Napadogan Brook	Mineral	Claim	November 28, 2007	November 28, 2026	Active	106
3270	Sisson Brook	Mineral	Claim	September 4, 1997	September 4, 2027	Active	561
Total							**707**

Figure 4-2: Inter-corporate relationships



Source: Northcliff, 2026

Figure 4-3: Land tenure map



Source: Stantec, 2026

4.3 Agreements

Northcliff acquired the Sisson claims through a joint venture (JV) agreement with Geodex Minerals Ltd. (Geodex), which was entered into on October 21, 2010. Under the terms of the JV, Northcliff could acquire a 70% interest in the Project by funding up to C$17 million in exploration and development costs. Northcliff was appointed the operator of the Project. Northcliff completed the earn-in obligations to secure a 70% interest in June 2012.

In May 2012, Northcliff and Geodex entered into a definitive agreement by which Northcliff could acquire the remaining 30% interest in the Property. Geodex agreed to sell its interest for 16,003,700 common shares of Northcliff, C$1 million in cash, and the return of 3,333,333 Geodex common shares which had been purchased by Northcliff in 2010. Northcliff shareholders unanimously approved the acquisition at the company's Annual General Meeting held on May 30, 2012, and Geodex shareholder approval was received on June 18, 2012. The transaction closed on June 21, 2012.

On October 3, 2013, Northcliff announced that it had entered into a C$5,000,000 private placement financing agreement with Todd Minerals Ltd. (Todd), a subsidiary of Todd Corporation of New Zealand. As a result, Todd acquired ownership and control of 13,888,889 shares in Northcliff and became Northcliff's largest shareholder with an approximate 15% interest in Northcliff.

In addition, Todd and Northcliff entered into a limited partnership agreement under the name of the Sisson LP in which Todd acquired an initial 11.5% interest for an initial contribution of C$14 million, which contribution was funded in stages and completed in October 2014. Todd has the option to acquire an additional 10% interest in the Limited Partnership by investing an additional C$20 million upon a final investment decision to commence construction.

As of 2026, Todd (through Todd Sisson (NZ) Limited) participated in several private placements and provided convertible loans to Northcliff for which it ultimately received shares in Northcliff as payment. As of the effective date of the Technical Report, Todd owns a total of 543,213,298 shares in Northcliff and controls 81.90% of Northcliff.

Northcliff and all entities owned by Northcliff are in good corporate standing.

4.4 Mineral rights

In Canada, natural resources fall under provincial jurisdiction. In the Province of New Brunswick, the management of mineral resources and the granting of exploration and mining rights for mineral substances and their use are regulated by the *New Brunswick Mining Act* that is administered by the New Brunswick Department of Energy and Mines. Mineral rights are owned by the Crown and are distinct from surface rights.

In April 2010, the Government of New Brunswick implemented NB e-CLAIMS, an internet-based electronic mineral claim acquisition and administration system. The new claim acquisition system is based on a predefined New Brunswick grid that required conversion of existing ground staked claims to conform to the new format. A map-designated mineral claim is valid for one year from the date of registration. Mineral claims can be renewed and transferred. To maintain a mineral claim in good standing, exploration work must be conducted, and reports of work must be submitted.

Claims are staked online as blocks of units which measure 500 x 500 m each.

4.5 Surface rights

Northcliff does not hold any surface rights. New Brunswick mining law allows for access and use of the surface for mining through the permitting process.

4.6 Encumbrances

As of the effective date of the Technical Report, the property is free and clear of any mortgages, liens, pledges, or third-party encumbrances that could materially affect the potential development of the Mineral Resources.

4.7 Environmental Liabilities, Permitting, and Other Significant Factors and Risks

The QPs are not aware of any environmental liabilities associated with the property. Northcliff has obtained all permits required to conduct the drilling and site work currently underway on the property. The QPs are not aware of any other significant factors or risks that may affect access, title, or the right or ability to carry out the proposed work program on the property. All necessary permits to carry out the recommended programs will be routinely applied for and obtained prior to commencing those activities.

5 Accessibility, climate, local resources, infrastructure, and physiography

5.1 Accessibility

Access to the Sisson property is gained via paved highway and good quality gravel forestry roads. The nearest large centre is Fredericton, which is the provincial capital of New Brunswick (Figure 4-1). From Fredericton, the property is accessible by travelling for approximately 60 km along Highway 1 (Trans-Canada Highway), and then via approximately 45 km of secondary highway and Valley Forest Products forestry roads. Forestry roads in the Sisson area are on Crown land hence fall under the stewardship of the New Brunswick Department of Natural Resources. These roads provide access to most areas of the Property and are licensed to AV Nackawic Inc. with Valley Forest Products being this company's woodlands subsidiary (Figure 5-1).

5.2 Local resources

Fredericton is the closest major centre, and third largest city in New Brunswick with a population of just over 63,000 according to the 2021 Canadian census. The city is able to supply all necessary supplies and commercial services for exploration and mining. Daily international commercial air service operates out of Fredericton, as well as rail, bus, courier, and truck transport.

Several smaller towns and villages are located in the Project area, and these can provide labour and some support services. The largest is Nackawic, a town of approximately 7,000 people, located 35 km from the property. Nackawic has been the site of a pulp mill since 1970 and is a likely source of skilled labour and heavy industrial services.

The property is traversed by power lines, and as stated, is readily accessible by paved and good gravel roads. The CN rail line is located approximately 15 km north of the Project.

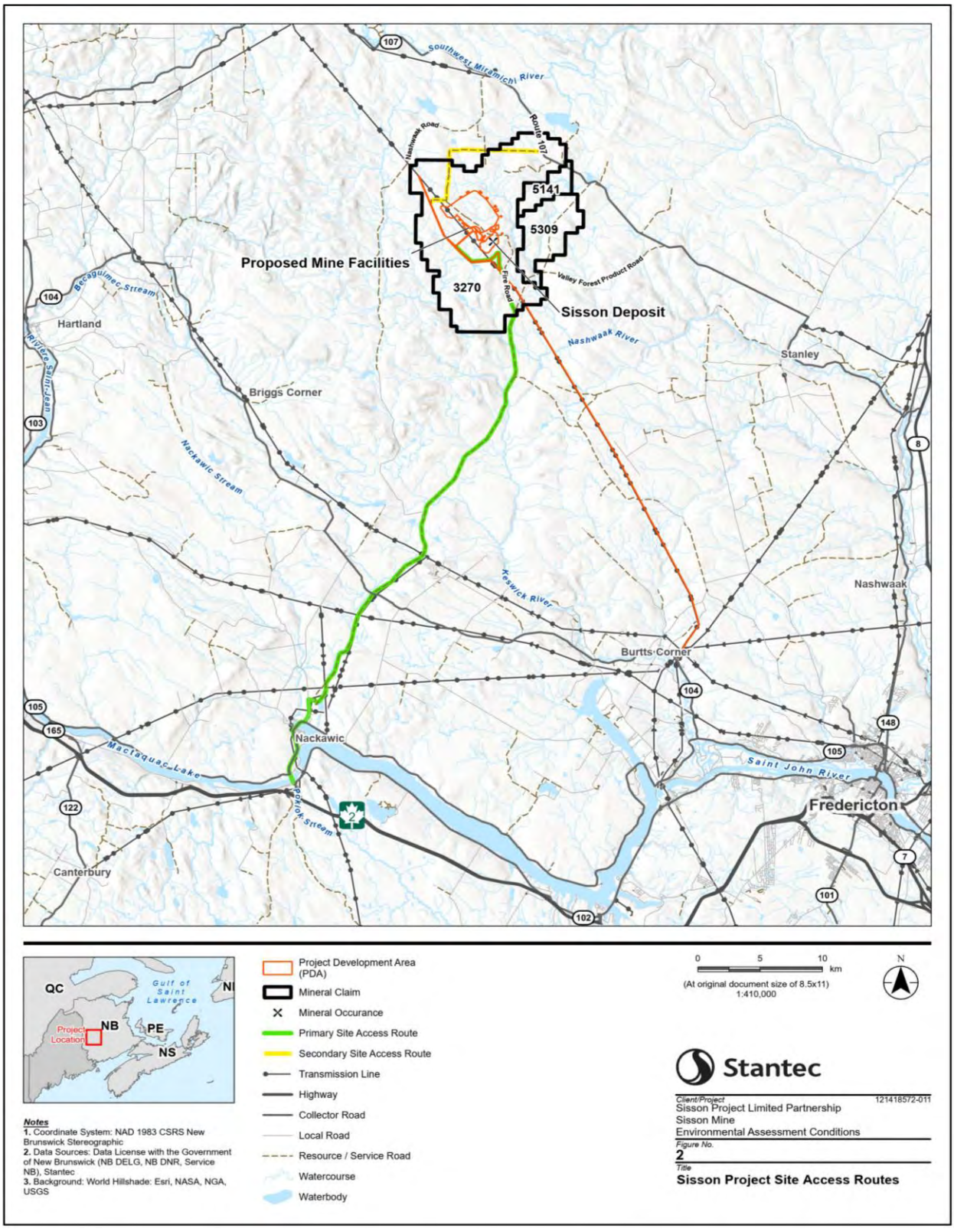
5.3 Physiography

The property lies in an area of moderate relief with surface elevations generally within the range of 200 to 350 metres above mean sea level (masl). Hilltops are strongly glaciated and rounded, and drainage valleys tend to be broad and open. Most of the countryside is timbered, and forestry is the primary land use in the Project area at present. Clear-cut logging and reforestation operations are active on the property. Other land uses for the surrounding region include agriculture and recreation.

Local vegetation comprises deciduous and coniferous forests typical of Canadian boreal environments.

In the QP's opinion, the area and physiography do not present any impediment to exploration or the development of an open pit mining operation and its site infrastructure.

Figure 5-1: Project access



Source: Stantec, 2026

5.4 Infrastructure

Infrastructure consists of several gravel and dirt roads providing access to the interior of the claim group. No conventional mining infrastructure exists on the property.

5.5 Climate

Fredericton is the closest Environment Canada maintained weather station to the Property.

Fredericton has a warm-summer humid continental climate (Dfb under the Köppen climate classification system) with short, warm summers and long, cold winters. The city has high humidity and precipitation year-round.

The mean annual temperature for the Project area is estimated to be 3.3 °C, with minimum and maximum monthly temperatures of –16.6 °C and 20.0 °C occurring in January and July, respectively. The Mean Annual Precipitation for the Project area is estimated to be 1,350 mm, with 1,013 mm falling as rain and 337 mm falling as snow. Snow can generally be expected from November to March, with accumulations remaining on the ground from December to February. Exploration and mining activities can be carried out year-round.

6 History

6.1 Prior ownership

The ownership of the Property prior to Northcliff's involvement is well documented. Work performed by various companies over time (see below) is a first approximation of historical property ownership. The project has historically been referred to as the Sisson Brook Project due to its mineral claim nomenclature and location in New Brunswick. In February 2011, Northcliff renamed the project to the Sisson Project.

6.2 Exploration and development history

This section is summarized from RPA (2012) and is a compilation from various assessment reports filed with the New Brunswick Department of Natural Resources (DNR) and from a previous compilation by Giggi (2006) and updated by Dr. James Lang, P. Geo., Senior Vice President Geology, Hunter Dickinson Inc. (HDSI) in 2012. HDSI is a wholly owned subsidiary of Hunter Dickinson Inc.; a diversified mineral development company. HDSI provides services to a range of its associated exploration and development companies, including Northcliff.

The QP has reviewed and accepted this information.

6.2.1 Nashwaak Pulp and Paper Co.

The first significant work in the Sisson area was completed in the late 1950s. The focus of this early work was east of Sisson Deposit and included prospecting, line cutting, geological mapping, trenching, ground (horizontal loop electromagnetic (EM) and magnetometer) and airborne (EM and magnetometer) geophysical surveys, and diamond drilling. A total of 12 holes were completed in 1955 and 43 holes in 1959-1960 which resulted in the discovery of the Nashwaak polymetallic vein deposit.

6.2.2 Penarroya Canada Ltée. (1967–1969)

Geological mapping, a ground magnetic survey, and soil sampling were completed, mostly south of the Sisson deposit. This work identified local zinc and copper soil anomalies. Four holes were drilled, including one near the Sisson deposit, but results were not encouraging.

6.2.3 Texasgulf Inc./ Kidd Creek Mines Ltd. (1973, 1978–1983)

The work in 1973 focused on the area of the Nashwaak vein deposit east of Sisson and included soil sampling, geological mapping, trenching, ground geophysical surveys (magnetic and horizontal loop EM), and five short drill holes. In 1978, soil surveys in the area of the Sisson deposit returned strong geochemical anomalies for Zn, Cu, W, Bi, Pb, and Sn. Subsequent drilling of 40 holes between 1979 and 1983 led to discovery of Zones I and II in 1979 and Zone III in 1981. This work is summarized by Mann (1981, 1982) and is further discussed in Section 7.2.

6.2.4 Various operators (1977–2001)

Between 1977 and 2001, additional, comparatively limited work was completed by Canadian Nickel Company Ltd. (1977-1981), Riocanex Inc., and Rio Tinto Canadian Exploration Ltd. (1980-1982), Shell Canada Resources (1981), K.D.A. Whaley (1981-1982; 1989), Phelps Dodge Corporation of Canada Ltd. (1992-1993), Noranda Exploration Company Ltd. (1992-1996), and Nicon Holdings Ltd./Freegrant Silver Corporation

Ltd./Champlain Resources Inc. (1999-2001). The work by these groups was mostly surface exploration and geophysical surveys, accompanied by very limited drilling. Most of this work was completed outside the Sisson deposit and did not result in any significant discoveries of mineralization.

6.2.5 Geodex and Champlain Resources Inc. (2004–2009)

The next period of significant exploration in the Sisson deposit was completed by Geodex, initially in JV with Champlain Resources Inc. Work included ground and airborne geophysical surveys, compilation of historical data, trenching, reanalysis of historical drill core, geological mapping and prospecting, and extension of previous soil and till sampling grids over and around the Sisson deposit. Approximately 210 drill holes were completed. The results of this work largely outlined the extent of mineralization in Zones I, II and III, and led to the discovery of the Ellipse Zone in 2008. Several NI 43-101 technical reports, MREs, and Mineral Resource updates, using various tungsten trioxide (WO_3) equivalent (WO_3Eq) cut-off values, were produced between 2007 and 2010. Preliminary economic assessments (PEAs) with positive conclusions were completed by Wardrop Engineering Inc. (Wardrop) in 2007 and Geodex in 2009.

6.2.6 Northcliff and Geodex (2010 to present)

Northcliff signed a JV agreement with Geodex in October 2010. Under the terms of the agreement, Northcliff would be the operator with the opportunity to earn 70% of the Project for expenditures of about C$17 million (see details in Section 4.3 of this Technical Report entitled Agreements). Northcliff completed the earn-in obligations to secure 70% interest in June 2012. In May 2012, Northcliff and Geodex entered into a definitive agreement by which Northcliff could acquire the remaining 30% interest in the Property and the transaction closed on June 21, 2012.

The work completed by Northcliff following the JV agreement is described in Sections 9and 10of this Technical Report.

6.3 Historical mineral resource estimates

This section is summarized from RPA (2012) and documents historical resource estimates on the Property prepared by previous owners. Inclusion of the historical estimates is deemed relevant as they give an indication of mineralization on the Property. However, all historical estimates are superseded by the mineral resource presented in Section 14 of this Technical Report. A QP has not done sufficient work to classify the historical estimates as current mineral resources or mineral reserves and the issuer is not treating the historical estimates as current mineral resources or mineral reserves.

The earliest MRE referenced in the Geodex technical reports was compiled by Kidd Creek Mines Ltd. (Kidd Creek) in or around 1980 (Mercator Geological Services Ltd. [Mercator], 2007). This estimate was carried out for Zone I, and totalled 7.5 million tons of material grading 0.21% WO_3 and 0.35% Cu. This estimate was completed prior to the implementation of NI 43-101, and the material was classified as “Drill Indicated”. The use of terms characterizing the confidence of estimates was, at that time, unregulated and largely discretionary. The term “Drill Indicated” does not necessarily demonstrate the same level of confidence as the term “Indicated” as defined in NI 43-101.

Some time prior to 2006 Geodex prepared an in-house estimate for Zone II which totalled 1.6 million tons grading 0.215% WO_3 and 0.41% Cu. No public report was issued, and details of dates, methodology and classification are unavailable. This is a historical estimate and should not be relied upon.

In 2006, Geodex prepared another in-house estimate for what was then termed a "potential mineral deposit that is the target of further exploration" within Zone III. This estimate comprised 26.0 million tonnes (Mt) of inferred mineral resources grading 0.05% MoS and 0.102% WO_3. The results of this analysis were announced in a Geodex Press Release on March 24, 2006; however, it was not characterized as a formal MRE and no report was issued. This is an historical estimate and should not be relied upon.

The first public disclosure of a MRE for the Project under NI 43-101 was made in March 2007 (Mercator, 2007). The estimate was compiled for Zone III on behalf of Geodex by Mercator and is summarized in Table 6-1. Mercator prepared the estimate using a block model constrained by three-dimensional (3D) wireframe solids, and grades were interpolated using inverse distance squared (ID^2) weighting.

The QP notes that the resources were reported at a range of WO_3Eq cut-offs, however, the base case cut-off grade was not specified, nor were the metal prices used to derive the WO_3Eq values and cut-offs.

Table 6-1: Historical MRE to March 28, 2007

Cut-off	Tonnage	WO_3	MoS_2	WO_3Eq
(%WO_3Eq)	(Mt)	(%)	(%)	(%)
0.025	167.4	0.059	0.021	0.122
0.075	126.3	0.069	0.025	0.143
0.125	64.4	0.084	0.034	0.186
0.175	28.9	0.098	0.045	0.233
0.225	12.3	0.110	0.057	0.281

An update of the MRE, using data from an additional 18 holes drilled that year, was released in October 2007. All Mineral Resources were classified as Inferred. This estimate formed the basis for a PEA of the Project conducted by Wardrop in 2007 (Ozturk et al., 2007). The updated estimate is summarized in Table 6-2.

Table 6-2: Historical MRE to October 15, 2007

Cut-off	Tonnage	WO_3	Mo	WO_3Eq
(%WO_3Eq)	(Mt)	(%)	(%)	(%)
0.025	290.8	0.059	0.020	0.118
0.075	215.0	0.069	0.024	0.140
0.125	109.0	0.084	0.032	0.179
0.175	43.4	0.100	0.043	0.227
0.225	15.8	0.121	0.054	0.281

Again, the base case cut-off grade for the resource estimate was not specified in Ozturk et al. (2007). The QPs note that the cut-off criteria were not explicitly stated either. The financial model was based on a total of 207 Mt of mill feed grading 0.063% WO_3 and 0.024% Mo. In the QP's opinion, this implies that the cut-off grade would be somewhere between the 0.025% WO_3Eq and 0.075% WO_3Eq cut-offs in Table 6-2.

Geodex continued with diamond drilling, which resulted in the discovery of additional Mineral Resources as well as upgrades in classification to both Measured and Indicated categories (Mercator, 2008). An update was prepared in June 2008 and is summarized in Table 6-3.

Table 6-3: Historical MRE to June 5, 2008

Cut-off (%WO_3Eq)	Tonnage (Mt)	WO_3 (%)	Mo (%)
Measured			
0.025	18.6	0.067	0.025
0.075	15.2	0.075	0.029
0.125	10.3	0.086	0.036
0.175	5.6	0.099	0.047
0.225	3.0	0.109	0.060
Indicated			
0.025	225.3	0.061	0.019
0.075	160.0	0.073	0.025
0.125	92.5	0.088	0.033
0.175	47.4	0.100	0.043
0.225	22.6	0.111	0.055
Inferred			
0.025	154.4	0.049	0.020
0.075	103.4	0.059	0.027
0.125	55.4	0.072	0.036
0.175	25.7	0.085	0.047
0.225	11.4	0.094	0.060

Mercator concluded that the increase in tonnes from the previous estimate was due to additional resources discovered by drilling.

Following a 2008 drilling program, Mercator prepared an updated estimate, which included both Zone III and the Ellipse Zone (Mercator, 2009). A summary of this estimate, which was current to January 28, 2009, is provided in Table 6-4. The QPs note that for the first time, a base case cut-off was assigned to the estimate. This cut-off grade was 0.125% WO_3Eq and was derived using metal prices of US$9.00/lb WO_3 and US$15.00/lb Mo, plus recoveries of 70% for WO_3 and 85% for Mo. The tonnes and grade at the base case cut-off are highlighted in Table 6-4.

Table 6-4: Historical MRE to January 28, 2009

Zone III			
Cut-off	Tonnage	WO_3	Mo
($\%WO_3Eq$)	(Mt)	(%)	(%)
Measured			
0.025	26.6	0.072	0.025
0.075	19.4	0.086	0.030
0.125	**11.0**	**0.109**	**0.037**
0.175	5.1	0.135	0.044
0.225	1.8	0.168	0.053
Indicated			
0.025	257.3	0.062	0.019
0.075	151.1	0.082	0.026
0.125	**69.1**	**0.107**	**0.034**
0.175	26.7	0.131	0.045
0.225	9.3	0.152	0.058
Inferred			
0.025	223.6	0.055	0.018
0.075	118.7	0.073	0.027
0.125	**47.8**	**0.097**	**0.036**
0.175	15.9	0.120	0.049
0.225	5.4	0.138	0.064

Ellipse Zone			
Cut-off	Tonnage	WO_3	Mo
($\%WO_3Eq$)	(Mt)	(%)	(%)
Measured			
0.025	1.10	0.082	0.023
0.075	0.90	0.092	0.027
0.125	**0.50**	**0.112**	**0.034**
0.175	0.20	0.135	0.044
0.225	0.10	0.146	0.068
Indicated			
0.025	28.3	0.072	0.024
0.075	22.5	0.081	0.027
0.125	**10.4**	**0.105**	**0.038**
0.175	4.60	0.121	0.050
0.225	1.50	0.139	0.067
Inferred			
0.025	36.5	0.055	0.018
0.075	23.4	0.073	0.027
0.125	**8.60**	**0.097**	**0.036**
0.175	3.40	0.120	0.049
0.225	0.90	0.138	0.064

Compared to the previous estimate, the January 2009 estimate was somewhat smaller in terms of tonnes in some categories and cut-off grades, but larger in others. Grades for WO_3 were moderately to significantly higher in all categories. Molybdenum grades were generally slightly higher with some categories showing a modest reduction. Mercator concluded that the change in the Mineral Resources was due to an increase in drilling information plus changes to the cut-off criteria applied to the model.

The January 2009 MRE was used by Geodex in an update of the PEA for the Project, conducted during 2009 (Marr, 2009). The financial model for the updated PEA used metal prices of US$10.00/lb WO_3 and US$15.00/lb Mo, and an exchange rate of CAD 1: USD 0.85. Metallurgical recoveries were 73.9% for WO_3 and 70.4% for Mo. The cut-off grade derived from the PEA was reported to be 0.100% WO_3Eq, and this cut-off was applied to subsequent MREs (see Table 6-5).

Geodex continued the drilling in 2009 and released another updated resource estimate in December of that year (Mercator, 2010). This estimate is summarized in Table 6-5.

Table 6-5: Historical MRE to December 17, 2009

Zone III			
Cut-off	Tonnage	WO_3	Mo
($\%WO_3Eq$)	(Mt)	(%)	(%)
Measured			
0.025	47.4	0.073	0.025
0.075	34.6	0.088	0.031
0.100	**27.5**	**0.097**	**0.034**
0.125	20.2	0.110	0.038
0.175	9.50	0.133	0.048
0.225	3.80	0.157	0.061
Indicated			
0.025	308.5	0.061	0.019
0.075	179.5	0.082	0.026
0.100	**125.4**	**0.094**	**0.030**
0.125	83.0	0.107	0.035
0.175	32.7	0.133	0.045
0.225	11.7	0.153	0.060
Inferred			
0.025	168.1	0.055	0.020
0.075	91.3	0.075	0.029
0.100	**61.3**	**0.086**	**0.034**
0.125	39.4	0.098	0.040
0.175	15.8	0.117	0.055
0.225	6.10	0.133	0.072

Ellipse Zone			
Cut-off	Tonnage	WO_3	Mo
($\%WO_3Eq$)	(Mt)	(%)	(%)
Measured			
0.025	2.70	0.069	0.021
0.075	1.80	0.086	0.027
0.100	**1.30**	**0.095**	**0.031**
0.125	0.90	0.104	0.037
0.175	0.40	0.124	0.048
0.225	0.10	0.134	0.071
Indicated			
0.025	58.9	0.062	0.019
0.075	34.1	0.081	0.026
0.100	**23.2**	**0.092**	**0.032**
0.125	15.3	0.102	0.038
0.175	6.20	0.117	0.054
0.225	2.30	0.127	0.076
Inferred			
0.025	22.9	0.057	0.017
0.075	13.0	0.073	0.024
0.100	**7.70**	**0.085**	**0.029**
0.125	4.60	0.092	0.036
0.175	1.30	0.093	0.060
0.225	0.40	0.088	0.085

As explained above, the cut-off grade for the December 2009 resource estimate was reduced from 0.125% WO_3 to 0.100% WO_3. This was reportedly done to reflect the base case cut-off grade developed by Geodex in the updated PEA.

The tonnage for Measured and Indicated Mineral Resources increased for all cut-off grades and decreased for the Inferred. Grades for WO_3 tended to remain unchanged from the January 2009 estimate in Zone III but were generally lower in the Ellipse Zone. This was particularly true for the higher cut-off grades (i.e., 0.175% WO_3 and 0.225% WO_3). Molybdenum grades were similar for both estimates in the Measured and Indicated categories in Zone III, but significantly higher for the Inferred. In the Ellipse Zone, molybdenum grades varied in both directions, ranging from an increase of 32.8% in the Inferred at the 0.225% WO_3 cut-off grade to a decrease of 20.8% in the Indicated at the 0.025% WO_3 cut-off grade. There were no opinions offered in the Mercator December 2009 technical report as to the reasons for the changes, but the QP assumes it was due primarily to the drilling done during 2009.

In 2011, Mercator was retained to prepare a NI 43-101 Technical Report on the property to support a reverse take-over agreement between Northcliff and Cabre (Mercator, 2011). The MRE in this Technical Report was unchanged from the December 2009 report. The only modification made was the effective date, which was revised to February 4, 2011. All these historical estimates have been superseded by the MRE in Section 14 of this Technical Report.

6.4 Past production

There has been no historical production from the Property.

7 Geological setting and mineralization

7.1 Regional geology

The regional, local and property geology, and mineralization sections of this Technical Report are summarized from RPA (2012).

The QP has reviewed and accepted this information.

The regional geological setting of southwest New Brunswick is illustrated in Figure 7-1 and Figure 7-2. The summary, which follows, is based mostly upon work by Fyffe et al. (2008), Fyffe and Thorne (2010), and references contained therein.

Mesoproterozoic to Earliest Cambrian basement rocks are exposed in southwest New Brunswick (Figure 7-1). They include carbonate and clastic sedimentary rocks of Mesoproterozoic to Neoproterozoic age in the Green Head Group and Neoproterozoic to earliest Cambrian mafic and felsic volcanic rocks and clastic sedimentary rocks in the Belleisle Bay Group. The tectonic setting comprised a comparatively stable volcanic and sedimentary continental margin that continued into the Cambrian. Intrusions related to the basement sequences were emplaced between 622 and 528 Ma and have calc-alkalic, continental margin geochemical signatures. These basement rocks are overlain by variably deformed and metamorphosed stratified sequences that were deposited in continental to marine environments between Early Cambrian and Early Devonian time.

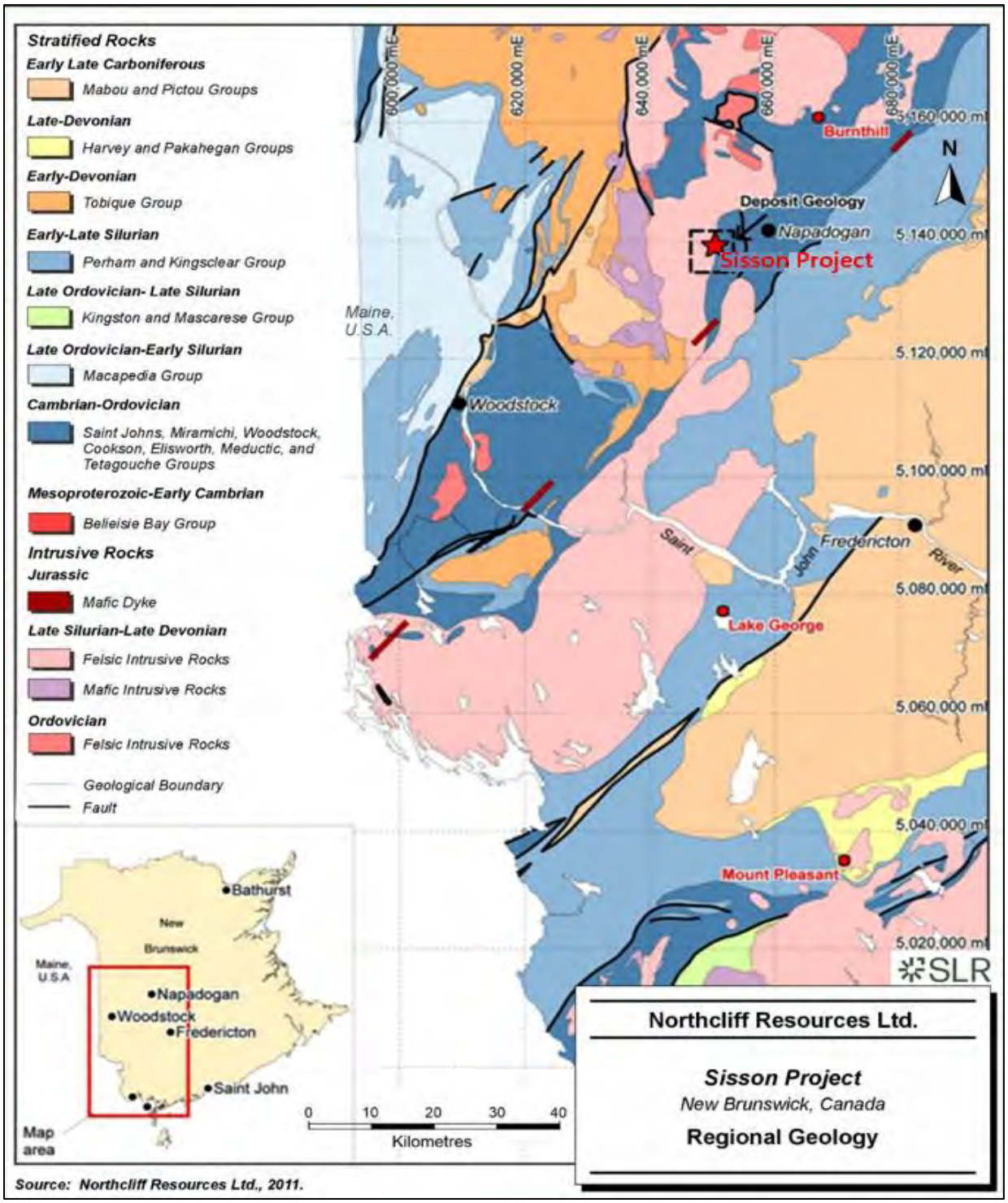
The Taconic Orogeny began by about 490 Ma, in the Early Ordovician, and continued into the Silurian. Tectonism was characterized by amalgamation of volcanic arcs during subduction of Iapetus (proto-Atlantic) oceanic crust. Sedimentary and volcanic rocks were deposited in volcanic arc and/or back-arc tectonic settings. Early Cambrian to Silurian sequences (Figure 7-1) are dominated by fine to coarse grained, clastic sedimentary rocks and lesser carbonate sequences which are, in many places, interlayered with mafic and felsic volcanic rocks. Metavolcanic and metasedimentary host rocks at Sisson formed during the Taconic Orogeny and are of Cambrian to Ordovician age. They include the predominantly clastic sedimentary sequences in the Miramichi Group overlain by Ordovician felsic to mafic volcanic strata and clastic sedimentary rocks of the Tetagouche Group.

The Acadian orogeny occurred from Late Silurian to Late Devonian and was a response to closure of the Iapetus Ocean and subsequent crustal thickening. Stratified sequences include both volcanic and clastic sedimentary rocks. The most important manifestation of the Acadian Orogeny is a north-northeast trending belt of plutons and batholiths that were emplaced between about 425 and 360 Ma (Figure 7-1 and Figure 7-2; Fyffe et al., 2008). These intrusions have geochemical signatures consistent with volcanic arc and, less commonly, collisional tectonic settings (W. Zhang, pers. comm., 2011). Younger intrusions of this group have a spatial, and an interpreted genetic, relationship to base and lithophile element mineralization, including tungsten deposits (Figure 7-2). The most significant deposits in this group include Sisson (Re-Os dates of 377 Ma on molybdenite; Zhang, unpublished), Burnt Hill (Ar-Ar isotopic date on muscovite alteration of 383 Ma; Taylor et al., 1987; MacLellan & Taylor, 1989), Mount Douglas (Ar-Ar date on alteration of 362 Ma; McLeod, 1991), Lake George (alteration not dated; Seal et al., 1987, 1988), and Mount Pleasant (associated rock types between 360 and 365 Ma; Kooiman et al., 1986; Hunt & Roddick, 1990; Tucker et al., 1998).

Regional deformation during the Taconic and Acadian orogenies includes folding, thrust faults and normal, reverse and strike-slip faults. Regional metamorphic facies are typically greenschist or lower. Wide contact metamorphic aureoles surround many of the Late Silurian to Late Devonian intrusions.

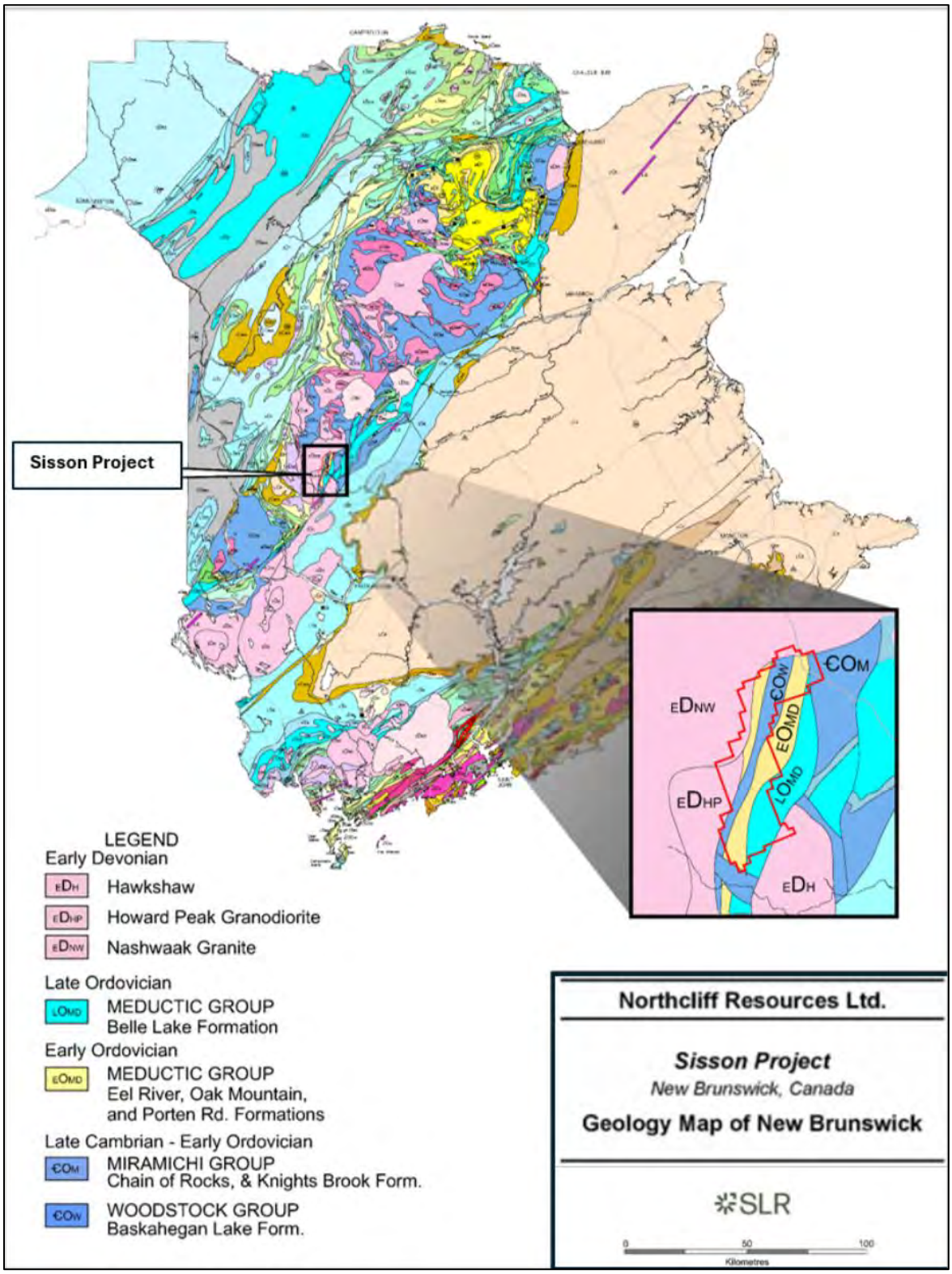
The youngest rocks in the region form the nearly flat lying, Late Devonian to Carboniferous strata of the "Carboniferous platform", which extends across most of central and eastern New Brunswick (Figure 7-1).

Figure 7-1: Regional geology



Source: Northcliff, 2011 modified from New Brunswick Department of Natural Resources Maps NR-4, 5

Figure 7-2: Geology map of New Brunswick, Canada



Source: Northcliff 2011 modified from New Brunswick Department of Natural Resources, Map NR-1(2000)

7.2 Local and property geology

The Sisson deposit is centred on a north-trending contact between Acadian intrusions to the west and older metavolcanic and metasedimentary rocks to the east (Figure 7-3; the "meta" prefix is hereafter omitted for simplicity). Mineralization occurs in four contiguous zones (Figure 7-3). Zones I and II are narrow, structurally controlled zones that extend north from Zone III, which hosts the bulk of the deposit. The Ellipse Zone extends northwest from the southwest corner of Zone III.

There is very little outcrop in the area of the Sisson deposit, and the geological interpretation is based primarily on results of drilling, exploration pits and trenches, and regional interpolation. From west to east (Figure 7-3), rock types progress through the following units, which use the formational assignments of Fyffe et al. (2008).

Nashwaak Granite – A massive, probably multiphase, equigranular biotite granite batholith of Acadian age. The batholith is poorly dated with ages of 422±4 Ma (Rb-Sr) and 386±5 Ma (K-Ar) from Whalen and Theriault (1990) and an Ar-Ar date of 379 Ma by Taylor et al. (1987).

Howard Peak Granodiorite - granodiorite phase. An undated, equigranular biotite granodiorite that grades into quartz diorite to the east and is intruded by Nashwaak Granite to the west.

Howard Peak Granodiorite - quartz diorite phase. This intrusion hosts mineralization in the western part of the Ellipse Zone at Sisson. It is an undated, medium-grained, sub-porphyritic hornblende quartz diorite.

Howard Peak Granodiorite - gabbro phase. This intrusion hosts mineralization in the eastern part of the Ellipse Zone and the western part of Zone III at Sisson. It is an undated, medium-grained, weakly porphyritic, pyroxene bearing hornblende gabbro. Its eastern contact with the Turnbull Mountain Formation is a vertical fault.

Turnbull Mountain Formation of the Ordovician Tetagouche Group - This unit comprises bimodal tuffaceous volcaniclastic rocks and biotite wackes. It is the main host rock to mineralization in Zone III at Sisson and is further described in this section.

Cambrian to Early Ordovician Miramichi Group - These rocks are dominated by siliceous wackes interbedded with siltstones and quartzites, with minor interbeds of intermediate volcaniclastic rocks. This rock sequence hosts only low-grade mineralization on the eastern margin of the Sisson deposit.

Hayden Lake Formation of the Ordovician Tetagouche Group - This unit contains black shales, flow banded felsic volcanic rocks, and fragmental mafic volcanic rocks which unconformably overlie the Miramichi Group. It is located east of the Sisson deposit.

Push and Be Damned Formation of the Ordovician Tetagouche Group - These are clastic sedimentary rocks located east of the Sisson deposit.

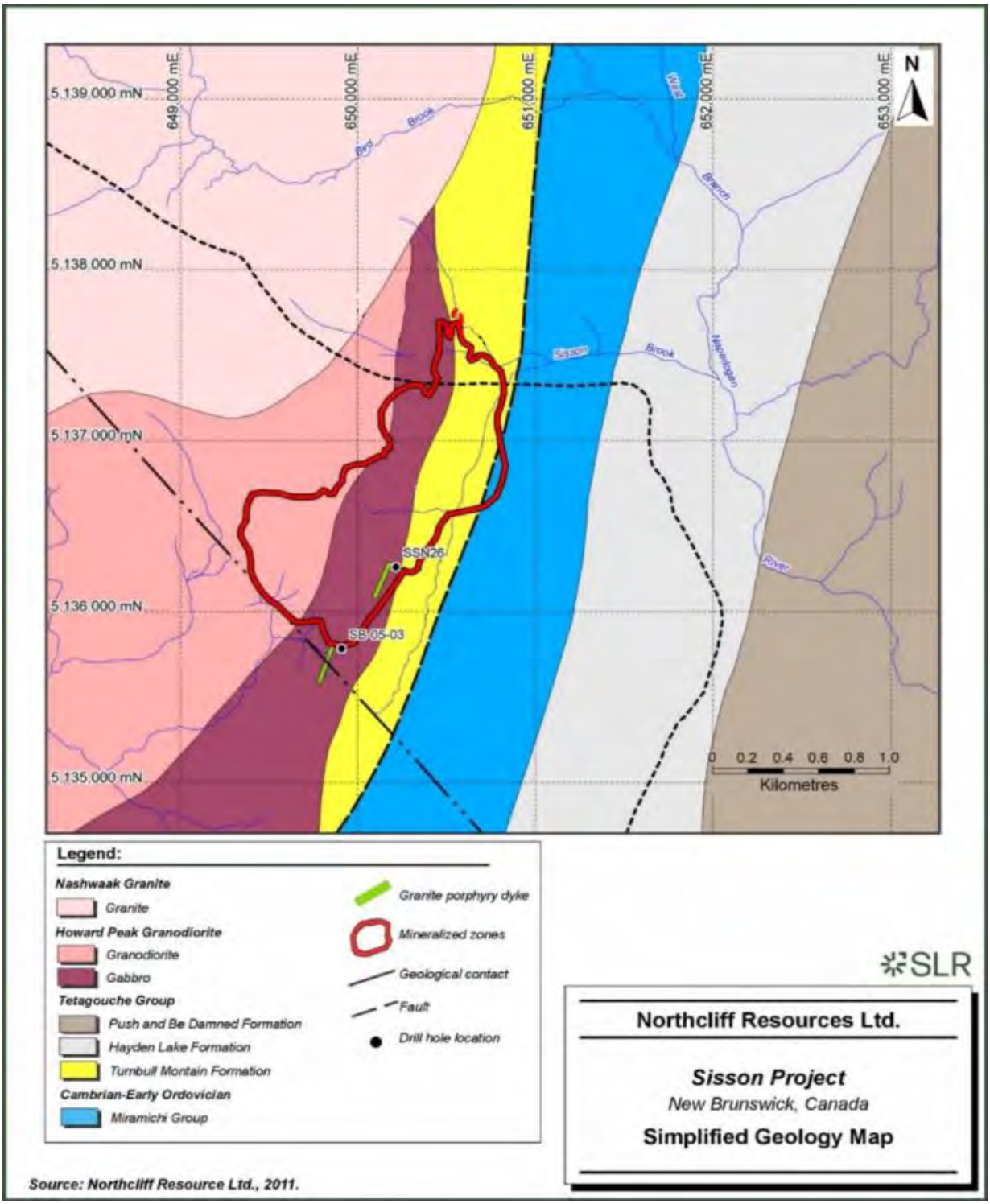
Stratified rocks within and near the Sisson deposit consistently strike north-northeast and dip steeply to the east.

Mineralization in the Sisson deposit is hosted by:

- the quartz diorite and gabbro phases of the Howard Peak Granodiorite
- felsic, mafic, and mafic crystal tuffs in the western part of the Turnbull Mountain Formation
- biotite wacke with minor interbeds of tuff in the eastern part of the Turnbull Mountain Formation
- volumetrically minor granite dykes and very rare mafic dykes (Figure 7-3).

- low-grade mineralization on the eastern edge of the deposit is hosted by more siliceous biotite-sericite wackes that may be part of the Miramichi Group.

Figure 7-3: Simplified geology map

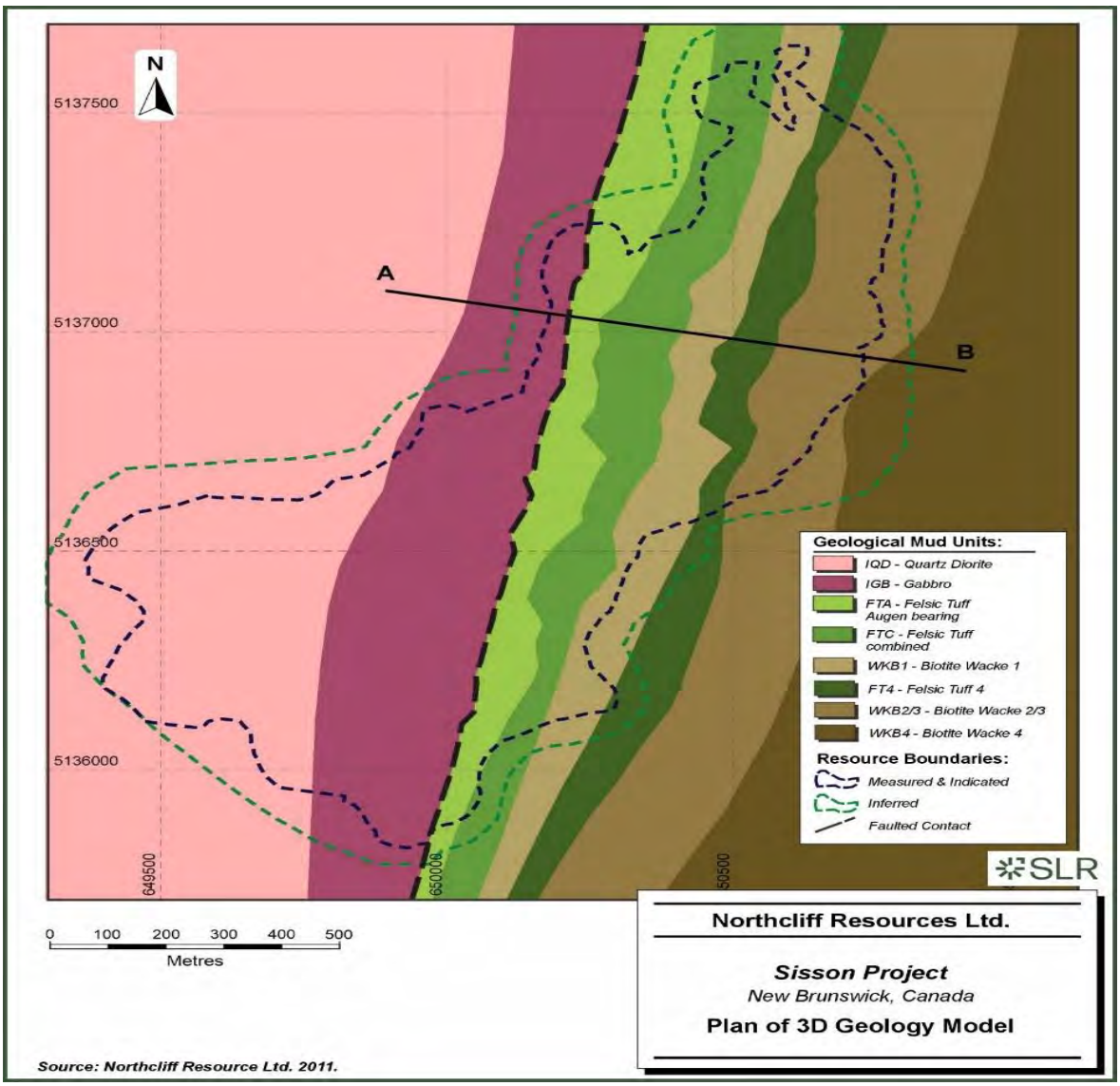


Source: Modified by Northcliff from Fyffe et al., 2008.

7.2.1 Rock types within and proximal to the Sisson Deposit

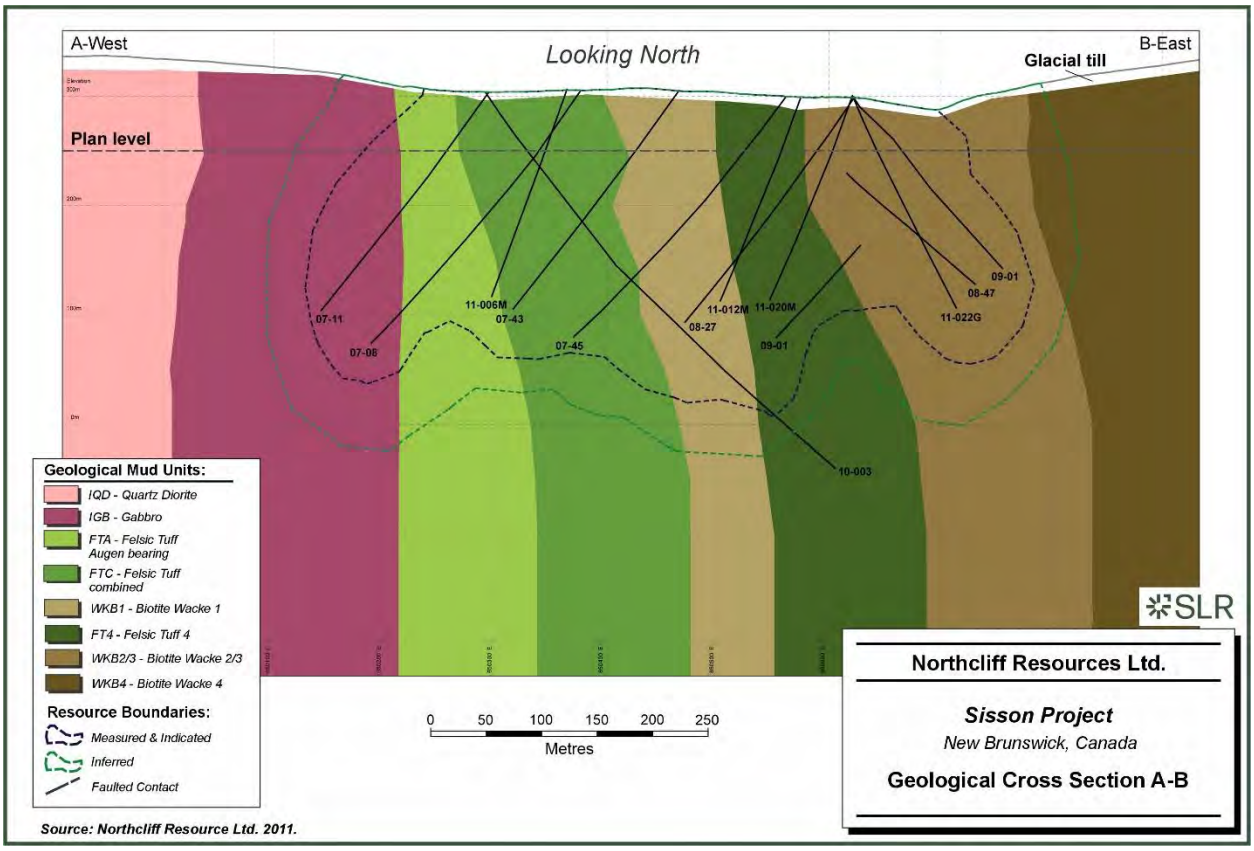
Rock types within the Sisson deposit are described in this section in the context of the 3D geological model which has been constructed from drilling information. The geological model for the Sisson deposit comprises eight units, which are summarized in Table 7-1 and shown in plan and section on Figure 7-4 and Figure 7-5. The gabbro and quartz diorite intrusive units and the biotite and biotite-sericite wacke units form comparatively thick, internally coherent units that can be traced laterally across the deposit. In contrast, felsic, mafic, and mafic crystal tuffs in the western part of the deposit comprise narrow, complexly interlayered beds, which may manifest significant lateral facies changes and whose lateral continuity has been further disrupted by deformation. In this light, individual rock types typically cannot be correlated confidently between sections, and these sequences therefore have been subdivided into model units which contain distinct combinations of rock types that can be traced across the deposit.

Figure 7-4: Plan of 3D geology model



Source: Northcliff, 2011

Figure 7-5: Geological cross-section A-B

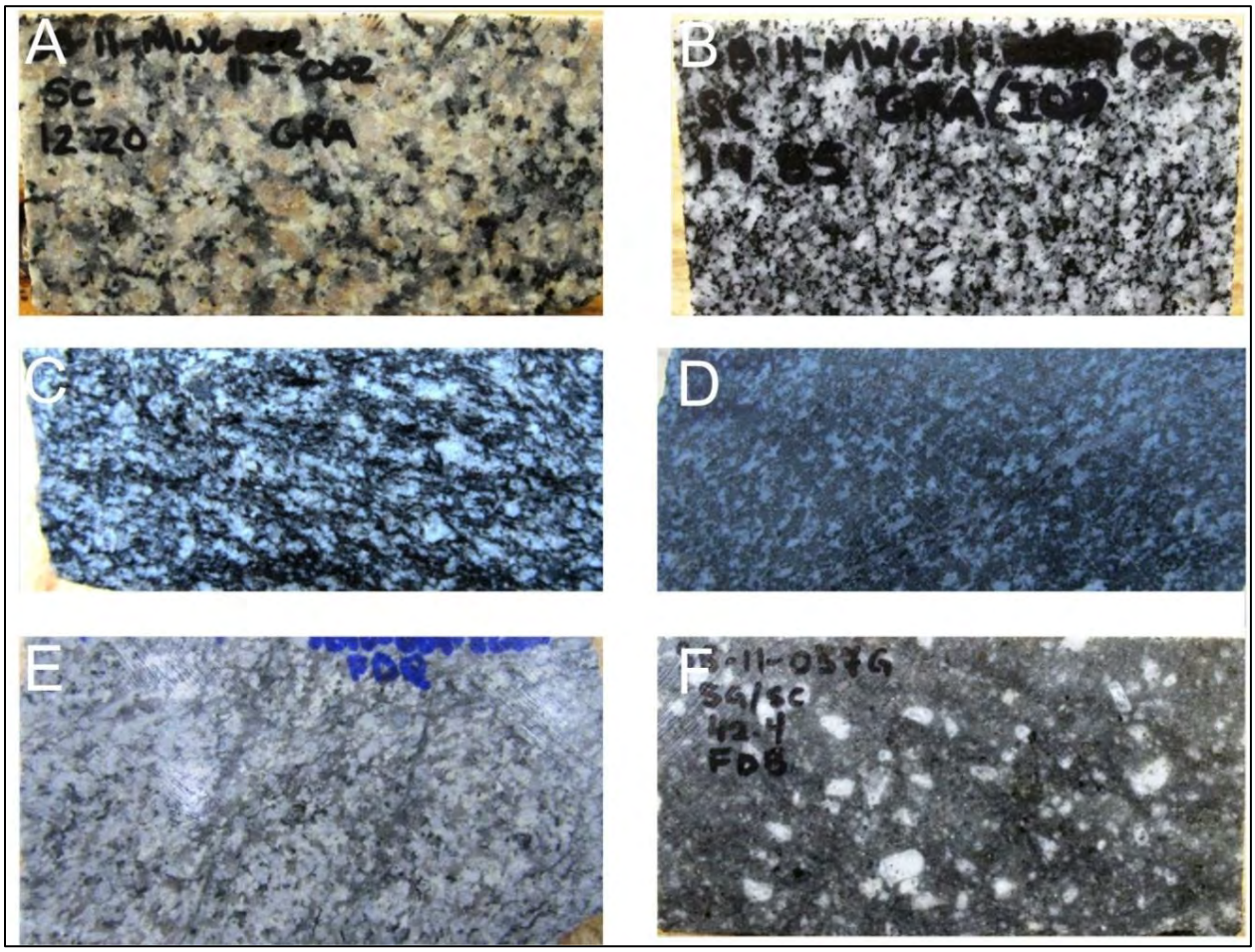


Source: Northcliff, 2011

7.2.1.1 Quartz Diorite and Gabbro (Model Units IQD and IGB)

The quartz diorite phase of the Howard Peak Granodiorite (Figure 7-6C) is an important host to mineralization in the western part of the Ellipse Zone, whereas the gabbro (Figure 7-6D) hosts the western part of Zone III and the eastern part of the Ellipse Zone. The quartz diorite is dark grey to mottled, sub-porphyritic, and medium grained. It is dominated by subhedral plagioclase and approximately 25% to 35% subhedral hornblende, accompanied by up to 5% small, anhedral grains of interstitial quartz, minor K-feldspar, 3% hematitized magnetite, and accessory titanite, apatite, and zircon. The gabbro is very similar but can be distinguished from quartz diorite by its higher concentration of hornblende, rare accessory pyroxene, and a near absence of quartz. Both intrusions can be well foliated and marked by segregations of metamorphic biotite. The quartz diorite grades to the biotite granodiorite phase of the Howard Peak intrusion (Figure 7-6B) to the east, and both are intruded by the Nashwaak Granite (Figure 7-6A). Quartz diorite and gabbro are intermingled near their contact: however, possible chilled contacts suggest that the quartz diorite intrudes the gabbro (Oliver, 2010). The contact between gabbro and volcanic rocks of the Turnbull Mountain Formation to the east is faulted and sheared and was a major locus for hydrothermal activity and tungsten mineralization. Dykes of gabbro have not been conclusively identified within the Turnbull Mountain Formation.

Figure 7-6: Photographs of igneous rock types

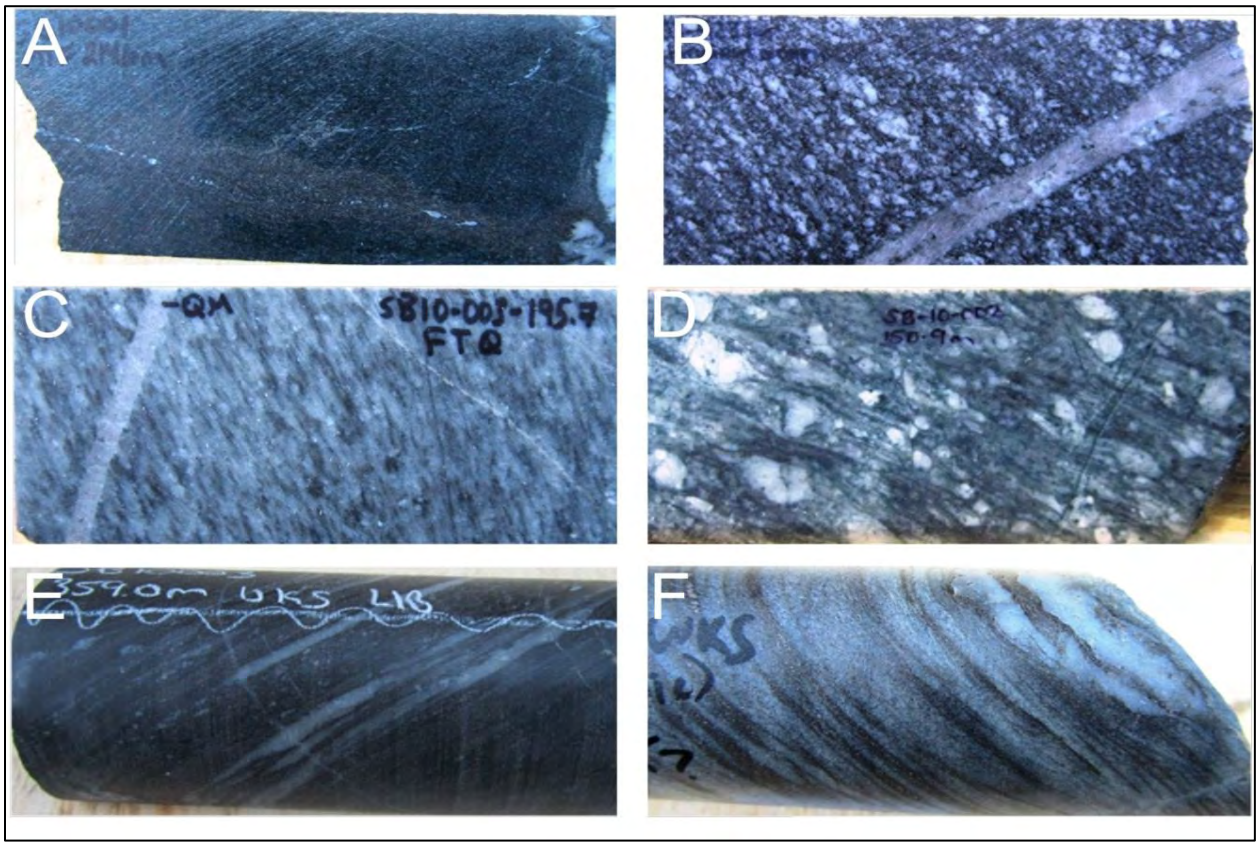


Note: Photographs of igneous rock types within and proximal to the Sisson deposit. A: Nashwaak Granite; B: Howard Peak intrusion, biotite granodiorite phase; C: Howard Peak intrusion, quartz diorite phase; D: Howard Peak intrusion, gabbro phase; E: granite dyke; F: biotite granite porphyry dyke. Source: RPA, 2012

7.2.1.2 Mafic Tuff

Mafic tuff (Figure 7-7A) is interbedded with felsic (Figure 7-7C) and mafic crystal tuff (Figure 7-7B) throughout the volcanic sequence, and minor interbeds are locally present in biotite wacke intervals in the central part of the deposit. It is present primarily in geology model units FTA, FTC, WKB1, and FT4. Drill intersections of individual mafic tuffs are rarely more than 1 to 2 m in length and range down to sub-centimetre scale; as such, individual beds of mafic tuff cannot be confidently correlated between drill holes or cross-sections. Mafic tuff is dark grey to dark green and is very fine-grained to aphanitic. It is dominated by amphibole and plagioclase, along with accessory titanite. The rock is typically massive but locally has a weak laminar fabric.

Figure 7-7: Volcanic and sedimentary rock types



Note: Photographs of volcanic and sedimentary rock types within and proximal to the Sisson deposit. A: mafic tuff; B: mafic crystal tuff; C: felsic tuff; D: felsic tuff with augen; E: biotite wacke; F: wacke with biotite- and sericite-rich interbeds. Source: RPA, 2012

7.2.1.3 Mafic Crystal Tuff

Mafic crystal tuff (Figure 7-7B) is a texturally variable rock type that forms the largest part of the volcanic sequence within the deposit. It occurs primarily in geology model units FTC and FT4 and, to a lesser extent, in units FTA and WKB1. It is black, dark brown, or mottled in colour and is typically interbedded at all scales with mafic and felsic tuffs. Narrow intersections of this rock type are locally present in biotite wacke intervals. The rock contains plagioclase, quartz, amphibole, minor alkali feldspar, and accessory magnetite and/or ilmenite, zircon and apatite; metamorphic biotite is abundant and replaces hornblende. The abundance of quartz indicates an intermediate composition. This rock type ranges from massive to strongly foliated.

7.2.1.4 Felsic Tuff and Augen-Bearing Felsic Tuff

Felsic tuff (Figure 7-7C) occurs throughout the volcanic portions of the stratigraphy and is an important host to mineralization. It occurs primarily in geology model units FTA, FTC, and FT4 and, to a lesser extent, in unit WKB1. It is typically interbedded with mafic tuff and mafic crystal tuff; however, internally homogenous beds locally range up to 20 m or more in true thickness. Narrow interbeds are locally present within wacke intervals. The rock is white to light grey and is dominated by quartz and feldspars. It ranges from massive and medium grained to strongly foliated and fine grained. Augen-bearing felsic tuff (Figure 7-7D) occurs only in the western part of the volcanic stratigraphy. Geology model unit FTA is defined by the

western and eastern extents of augen-bearing felsic tuff. It is distinguished from felsic tuff by up to 15% feldspar crystals or crystal aggregates up to 2 centimetres (cm) in size[1].

7.2.1.5 Biotite Wacke (model units WKB1, WKB2/3 and WKB4)

The eastern part of the Sisson deposit is hosted predominantly by three geological model units defined by the extent of biotite wacke (Figure 7-7E). Intersections of biotite wacke range from nearly black, to dark brown to medium grey-brown. They are locally medium grained but are mostly fine-grained and have a laminar fabric. All are characterized by high concentrations of metamorphic biotite, abundant feldspar, and lesser granular quartz; metamorphic sericite variably replaces feldspar. Quartz segregations occur parallel to the laminar fabric. The westernmost zone of biotite wacke, defined as geology model unit WKB1, contains andalusite formed during contact metamorphism. Central model unit WKB2/3 is similar to WKB1 but lacks andalusite. The easternmost intersections, which form geology model unit WKB4, may be part of the Miramichi Group and contain staurolite, have a higher concentration of sericite (Figure 7-7F), and include narrow interbeds of siliceous siltstone and quartzite. Narrow interbeds of mafic, felsic, and mafic crystal tuff occur in the western parts of the biotite wacke section.

7.2.1.6 Minor Intrusions

The Sisson deposit is cut by several types of volumetrically minor dykes.

Granite dykes - Light grey, fine-grained to pegmatoidal, locally granophyric granite dykes (see Figure 7-6E) are very common within the gabbro and quartz diorite intrusions and, to a much lesser extent, in volcanic rocks immediately east of the gabbro. True width for most dykes is less than 3 m. All of these dykes contain mineralization, veins, and/or alteration. Paragenetic relationships of hydrothermal features suggest that some of these dykes may be syn-hydrothermal; this is supported by overlap in their preliminary U-Pb ages from some dykes and Re-Os dates on molybdenite (Zhang, pers. comm., 2011).

Biotite granite porphyry dykes - These intrusions (Figure 7-6F) have been intersected in several drill holes, mostly from the south part of Zone III. They are grey, massive and unfoliated, fresh and commonly truncate mineralized veins. They are distinguished by phenocrysts of plagioclase, alkali feldspar, quartz, and 8% biotite in a fine-grained matrix. Fyffe et al. (2008) obtained a U-Pb date on zircon of 364.5±1.3 Ma, which is significantly younger than the ~377 Ma Re-Os dates on molybdenite (Zhang, unpublished data).

Mafic dykes - These are rare at Sisson and occur mostly in gabbro in the southwest part of the deposit. They are altered and mineralized but are volumetrically insignificant.

[1] These features probably represent volcaniclastic lapilli, but the term "augen" has been used historically and is retained here.

Table 7-1: Rock types or groups of rock types used to model the Sisson deposit

Model Unit	Unit Name	Mineralization	Constituent Rock Types
IQD	Quartz diorite	Western part of the Ellipse Zone	Quartz diorite phase, Howard Peak Granodiorite; internally homogenous
IGB	Gabbro	Eastern part of the Ellipse Zone, Zone II and western part of Zone III	Gabbro phase, Howard Peak Granodiorite; internally homogeneous
FTA	Felsic tuff – augen-bearing	Western part of Zone III	Thinly interbedded felsic > mafic > mafic crystal tuff; defined by eastern limit of augen-bearing felsic tuff
FTC	Felsic tuff combined	Central part of Zone III	Moderately to thinly interbedded mafic crystal > felsic > mafic tuff
WKB1	Biotite wacke 1	Zone II and central part of Zone III	Medium grained biotite wacke, commonly with contact metamorphic andalusite; contains minor mafic and felsic tuff near eastern and western contacts
FT4	Felsic tuff 4	Central and eastern parts of Zone III	Interbedded mafic crystal > mafic > felsic tuff; increasing biotite wacke interbeds toward eastern and western margins
WKB2/3	Biotite wacke 2/3	Eastern part of Zone III	Fine-grained biotite wacke with minor interbeds of mafic tuff; lacks andalusite
WKB4	Biotite wacke 4	Low-grade mineralization near the eastern margin of Zone III	Fine-grained biotite wacke with interbeds of siliceous siltstone and quartzite; may be the western margin of the Miramichi Group; locally contains staurolite

7.2.2 Structure

The Miramichi Group is believed to occupy the core of a north-northeast trending, south plunging anticline (Lutes, 1981) and to be bounded to the east and west by younger volcanic and sedimentary rocks of the Ordovician Tetagouche Group (Figure 7-3). The interpretation and/or specific characteristics of a major anticline cored by Miramichi Group, however, continues to be debated. Fyffe et al. (2008) note that rocks of the Tetagouche Group do not correlate on the east and west sides of the Miramichi Group, and Mann (1980, 1981) suggests that the volcanic rocks to the west of the Miramichi Group may young to the east, which precludes an antiformal structure. Small isoclinal folds, however, have been observed in drill core, and measurements in oriented core confirm that they plunge steeply to the south (Duncan, 2011), which is consistent with the interpretation of Lutes (1981). The western contact of the Miramichi Group against the Turnbull Mountain Formation may be a fault (Figure 7-3; Fyffe et al. 2008), which may further complicate local structural interpretation, although a definite structure has not been identified during the work described in this Technical Report. To the east, the Miramichi Group is overlain unconformably by the Hayden Lake Formation of the Tetagouche Group, which yields to the younger Push and Be Damned Formation of the Tetagouche Group. Regardless of the uncertainties in structural interpretation and regional correlation of rock types, these effects occur at scales larger than the zone of mineralization and do not impact on exploration and development considerations.

The north trending contact between the gabbro and the Turnbull Mountain Formation is a vertical fault zone, which ranges from a few to approximately 20 m in width. It is marked by strong fracturing, local brecciation, and minor gouge seams. The strongest alteration and tungsten mineralization and most

abundant quartz-sulphide veins occur along and adjacent to this important fault, although it did not apparently influence later molybdenum mineralization. Kinematic indicators are compatible with sinistral strike-slip movement, but absolute displacement is not constrained. Large quartz-sulphide veins in a trench in Zone I, described in a later section, also formed along north-trending, sinistral faults. Brittle faults with gouge are largely absent from the rest of the deposit but a few rubble zones are present.

Most rock types exhibit a weak to strong foliation, defined by biotite and/or muscovite, which is parallel to stratigraphic orientation. A northwest trending, steeply dipping cleavage is also commonly present (Fyffe and Thorne 2010).

7.3 Mineralization

Geometrically, Sisson Zone III forms an elongate shape with a maximum strike dimension of approximately 1,700 m oriented in a north-northeast direction. Zone III is approximately 600 m wide in the cross-strike direction with a maximum drilled depth extent of 450 m. The Ellipse Zone is located at the southern extent of Zone III and is oriented nearly perpendicular to it in a north-northwest direction. The dimensions of the Ellipse Zone are approximately 750 m long, 250 m wide and 315 m deep.

Mineralization in the Sisson deposit is related to several types of alteration and veins, each of which has a distinct relationship to metal introduction.

Hydrothermal features of the Sisson deposit are described below from early to late effects, to the extent that their timing has been constrained by paragenetic relationships. To summarize, early barren to weakly mineralized alteration was followed sequentially by an early stage of tungsten mineralization, an intermediate stage of molybdenum ± tungsten mineralization, and a late stage of polymetallic mineralization in which tungsten is accompanied by numerous base and trace elements. Most alteration and nearly all mineralization can be directly related to specific types of veins. The overall vein density at Sisson is only approximately 3% and pervasive alteration zones are spatially limited, which indicates that the hydrothermal system at Sisson had a low fluid flux.

7.3.1 Early sodic-calcic alteration and amphibole veins

The earliest hydrothermal feature at Sisson comprises amphibole veins and associated albite-actinolite (sodic-calcic) alteration (Figure 7-8A). These features occur almost exclusively in gabbro and, to a lesser extent, in quartz diorite. They are most abundant in the west-central part of Zone III and extend into the Ellipse Zone. The AA veins and sodic-calcic alteration contain a few percent pyrrhotite-pyrite but completely lack molybdenum and contain only trace scheelite. The veins are mostly between two and 20 mm in width, have diffuse contacts with host rocks and contain mostly actinolite, lesser albite, minor calcite, and trace to minor pyrrhotite and/or pyrite. Alteration envelopes are mineralogically similar and can be up to several times the width of the associated vein. Where vein density is high, the alteration envelopes coalesce to pervasive alteration.

7.3.2 Early biotite and biotite-sulphide alteration

This alteration mostly spans the contact between gabbro and volcanic rocks on the west side of Zone III. This alteration is commonly intense and texture-destructive (Figure 7-8B). Similar alteration is erratically distributed through gabbro west of the contact however is rare in quartz diorite. Sulphide concentration ranges from trace to 10% and manifests cubiform pyrite locally accompanied by pyrrhotite. Biotite-sulphide alteration typically contains neither scheelite nor molybdenite.

7.3.3 Calc-silicate alteration

Drilling has intersected mineralized calc-silicate alteration (Figure 7-8C) over lengths between 0.5 and 3 m. This alteration occurs both in gabbro and in volcanic rocks but mostly at depths greater than approximately 300 m. The timing of this alteration relative to other hydrothermal effects has not been constrained. The alteration is massive and pervasive and associated veins have not been observed. The mineralogy comprises various combinations of red-brown garnet, green pyroxene, calcite, epidote, and possibly amphibole, and up to a few percent pyrrhotite and lesser pyrite. Disseminated scheelite is ubiquitous but molybdenite has not been observed. Relicts of granite dyke within this alteration suggest that it may be endoskarn.

7.3.4 Quartz-scheelite veins (Type QW)

Quartz-scheelite (QW) veins represent the earliest stage of significant tungsten mineralization at Sisson. These veins are typically less than 5 mm in width, have sharp to locally diffuse contacts with host rocks, and range from planar to strongly curvi-planar (Figure 7-8D). They contain clear to milky quartz, scheelite, low concentrations of pyrite and/or pyrrhotite, and rare molybdenite. Alteration envelopes are mostly less than 1 cm in width and comprise biotite in mafic host rocks (Figure 7-8D) and, more rarely, sericite in felsic host rocks. The alteration envelopes commonly contain disseminated scheelite, particularly when hosted by gabbro or quartz diorite. The density of these veins diminishes markedly below approximately 300 m depth in the centre of Zone III, and at shallower and deeper levels in the north and south parts of the deposit, respectively.

7.3.5 Quartz-molybdenite (Type QM) and quartz-feldspar (Type QF) veins

Quartz-molybdenite (QM, Figure 7-8E) veins consistently cut QW veins. They represent the main stage of molybdenum mineralization and, except in the north part of Zone III, also contain high concentrations of scheelite. They are most abundant in the centre of Zone III, of intermediate abundance in the south part of Zone III and the Ellipse Zone, and are rare to absent in Zones I and II, the north part of Zone III, and in gabbro from the west-central part of Zone III. Type QM veins extend to much greater depth than QW veins and, for example, remain abundant at 525 m true depth in DDH-SB10-004.

The QM veins are mostly 1 to 5 cm in width, but veins greater than 20 cm wide are common. They are planar to curviplanar and have sharp contacts with wall rocks (Figure 7-8E). The QM veins are dominated by clear, locally epitaxial quartz. Molybdenite can form up to 10% of these veins but has a clotty distribution along their length. Most QM veins contain less than 3% pyrite, rarely accompanied by pyrrhotite. They contain low to locally high concentrations of scheelite (Figure 7-8F), although in some cases tungsten mineralization is related to younger type SX veins which exploited fractured QM veins. Sericite alteration envelopes less than 1 cm in width are only locally present and rare biotite envelopes occur in mafic host rocks.

Type quartz-feldspar (QF) veins are very similar to QM veins but occur mostly at depths below approximately 250 m. They are distinguished by selvages of white K-feldspar, contain a dark green mineral that may be grossular garnet, have a significantly lower concentration of molybdenite and scheelite than normal QM veins, and are mostly less than 3 cm in width. Vein types QM and QF are generally contemporaneous.

7.3.6 Quartz shear (Type QS) and sulphide-rich (Type SX) veins

Late quartz-shear (QS, Figure 7-8G) and sulphide-rich (SX, Figure 7-8H) veins are closely related and represent the youngest stage of significant mineralization at Sisson. The QS veins are the main style of mineralization in Zones I and II and in the strongly mineralized contact between gabbro and volcanic rocks in Zone III. They are common throughout the Turnbull Mountain Formation in Zone III but are less common in gabbro in Zone III and in the Ellipse Zone. The QS veins and associated alteration commonly attain grades between 0.25% WO_3 and 1.00% WO_3, although they constitute only a very small portion of the overall deposit. Drill hole intersections of QS veins are mostly less than 2 m in length, although intersections up to 10 m have been observed and swarms of smaller QS veins in rare cases span zones 20 m or more in apparent width. They exhibit crack-seal textures and formed by repeated reopening. As such, their formation was plausibly protracted and may have spanned much of the life of the Sisson hydrothermal system.

The QS veins range from planar to curviplanar and contacts with host rock are typically sharp (Figure 7-8G). Lensoidal forms occur in outcrop in Trench 1. The QS veins contain mostly white quartz cut by myriad brittle fractures that reflect structural disruption during and after vein formation. Sulphide concentration can attain 15% and comprises mostly pyrrhotite and lesser pyrite, 1% to 3% chalcopyrite, and highly variable concentrations of arsenopyrite, galena, sphalerite, and bismuth minerals. These veins host almost all of the wolframite in the deposit (Figure 7-8G), which occurs as black, prismatic crystals that are variably to completely replaced by scheelite. Molybdenite has not been observed.

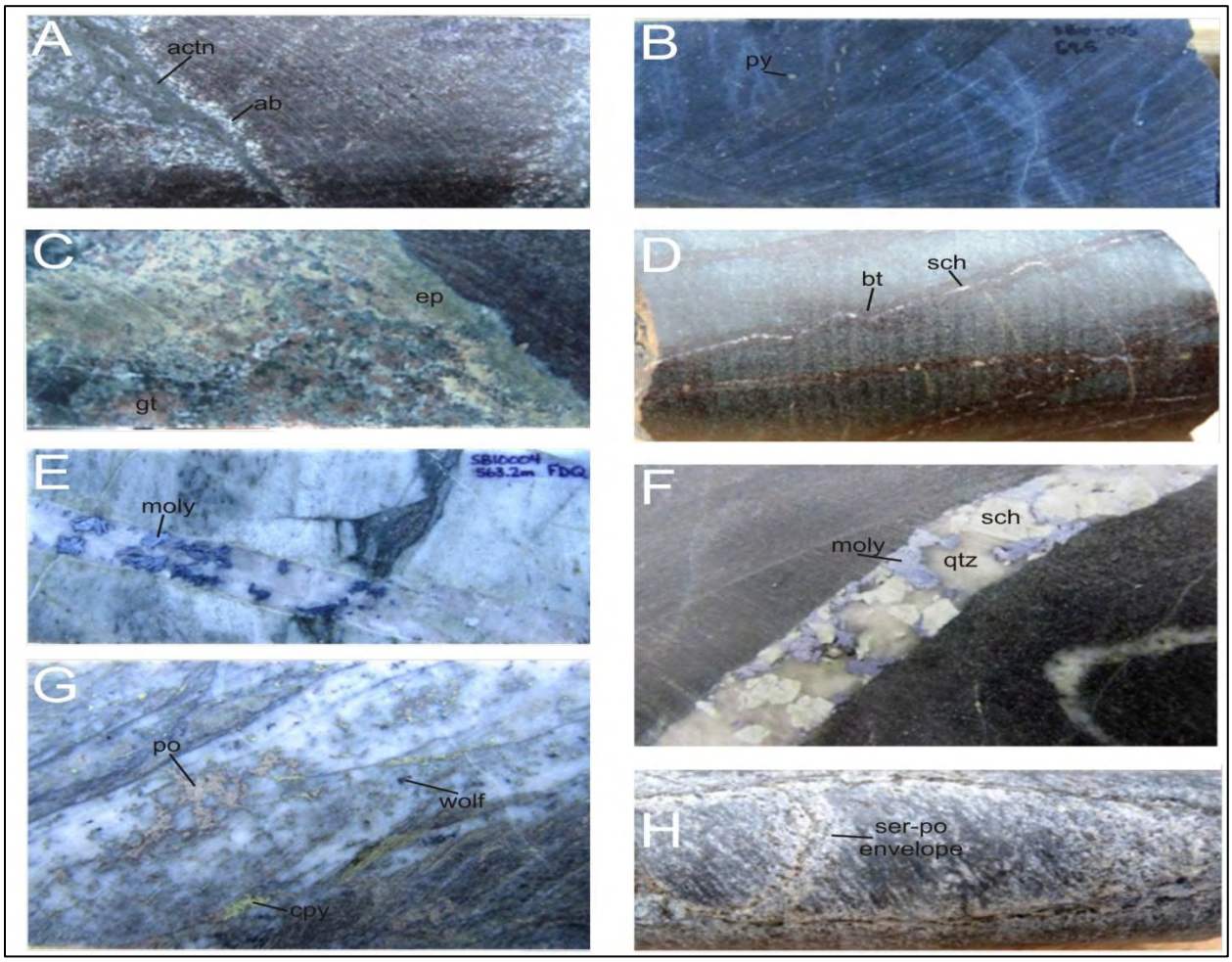
The QS veins are enclosed by envelopes of intense, texture-destructive sericite-quartz-sulphide alteration. The envelopes coalesce where multiple QS veins are closely spaced. The envelopes are dominated by sericite and quartz and contain up to 15% pyrrhotite, which is locally intergrown with lesser pyrite. The metal assemblage in veins and envelopes is similar. A few larger QS veins, mostly from the far eastern side of the deposit, have an outer, poorly mineralized carbonate envelope. These envelopes manifest swarms of narrow fractures that are filled with ankerite or ferroan dolomite, which cut and replace the proximal sericite-sulphide envelopes but extend up to 10 m from the associated veins.

The SX veins (Figure 7-8H) are widely distributed through the Turnbull Mountain Formation but are rare in the Ellipse Zone and gabbro in Zone III. Relationships exposed in Trench I suggests that SX veins may occur as sheeted swarms of planar fractures oriented at conjugate angles to and surrounding QS veins. Individual SX veins are fracture fills that rarely exceed 3 mm in width, and which are enclosed by intense sericite-quartz-sulphide alteration envelopes (Figure 7-8H) identical to those related to QS veins. The SX veins are mineralogically similar to QS veins.

7.3.7 Carbonate veins (Type QC)

Late carbonate veins are planar, up to 1 cm in width, and lack alteration envelopes. They contain calcite or dolomite, quartz, pyrite, and minor green to purple fluorite. In a few rare cases, they contain minor scheelite, but molybdenite has not been observed. This same assemblage locally forms open-space infill of small voids in late SX and QS veins. These veins are widespread but volumetrically and economically insignificant.

Figure 7-8: Examples of alteration and veins types



Note: A: actinolite veins with albite envelopes in gabbro. B: Pervasive biotite-pyrite alteration in mafic tuff. C: Calc-silicate alteration in either granite dyke and/or gabbro. D: Early quartz-scheelite (QW) veins with biotite alteration envelopes in mafic tuff. E: Intermediate stage quartz-molybdenite (QM) veins in felsic tuff. F: High-grade type QM or QW vein in mafic tuff. G: Late quartz-shear (QS) vein. H: Late sulphide-rich (SX) veins with prominent sericite-pyrrhotite alteration envelopes in mafic crystal tuff. Source: RPA, 2012

7.3.8 Styles of mineralization

Mineralization at Sisson occurs almost exclusively in quartz veins, fractures, and their alteration envelopes (Figure 7-9). Tungsten and molybdenum are the metals of principal economic interest, whereas several other metals, including copper, zinc, lead, arsenic, and bismuth, occur in geochemically anomalous but subeconomic concentrations.

Mineralization occurs in contiguous Zones I, II, and III and in the Ellipse Zone (Figure 7-3). Zones I and II trend approximately north from the northern end of Zone III. Zone I is hosted by the Turnbull Mountain Formation, whereas Zone II occurs in gabbro. Zones I and II are structurally controlled, up to tens of metres in width, and extend several hundred metres along strike. They are dominated by QS and SX veins and contain tungsten, copper, and associated trace elements, but lack significant molybdenum. Zone III contains the bulk of the tungsten and molybdenum resource at Sisson. It is an ovoid zone that obliquely spans the contact between gabbro and the Turnbull Mountain Formation and mineralization occurs in gabbro,

volcanic, and sedimentary rocks. The highest grades of molybdenum occur in both volcanic and sedimentary rocks in the central part of Zone III, whereas the highest grades of tungsten occur on either side of the contact between gabbro and volcanic rocks in Zone III. The Ellipse Zone and the southern part of Zone III are very similar to each other and contain moderate grades of both tungsten and molybdenum. Host rocks in the southern part of Zone III are gabbro and volcanic rocks, whereas the Ellipse Zone is hosted by gabbro and quartz diorite. Mineralization diminishes very abruptly at the south end of Zone III.

The minerals of economic interest at Sisson are scheelite, molybdenite, and minor wolframite. Scheelite precipitated during formation of QW, QM, QS, and SX veins, whereas most molybdenite is found in QM and, to a lesser extent, QF veins. Overall, the deposit contains very minor wolframite, which occurs almost exclusively in QS and SX veinlets where it is replaced, in whole or in part, by scheelite (Figure 7-9C). The highest concentration of copper, zinc, lead, arsenic, bismuth, and other trace elements is directly related to late QS and SX veins and their sericite-sulphide envelopes.

Tungsten and molybdenum mineralization precipitated mostly within quartz-sulphide veins (Figure 7-9A) and, in some cases, in their alteration envelopes (Figure 7-9B). Scheelite ranges from very fine (less than 100 µm) to coarse grained (less than 1 cm). Scheelite occurs both as selvages and along the central axes of veins. Fine-grained scheelite is also commonly disseminated through biotite alteration envelopes to QW veins (Figure 7 9B), particularly in gabbro and quartz diorite host rocks. Disseminated scheelite is less common in sericite alteration envelopes to QW, QS, and SX veins. Molybdenite occurs almost primarily within QM veins where it commonly forms coarse grained selvages (Figure 7-8E).

Metal zoning has only been established empirically at Sisson. The concentration of scheelite decreases with depth and grades diminish rapidly below approximately 250 m and 400 m in the north and south parts of Zone III, respectively. Above average grades of tungsten are most consistently present along the contact between gabbro and volcanic rocks and, to a much lesser extent, in narrow, steeply dipping zones on the eastern side of the deposit. The highest concentration of molybdenum occurs in the centre of Zone III, where type QM veins are most abundant. Moderate grades of molybdenum occur in the south part of Zone III and in the Ellipse Zone. Zones I and II and the north part of Zone III contain almost no molybdenum. Significant intersections of molybdenum mineralization extend to a much greater depth than tungsten mineralization. The distribution of both tungsten and molybdenum mineralization are consistent with a south plunge to the Sisson deposit. The highest concentrations of copper and associated elements are most consistently present in Zones I and II, whereas elevated concentrations of these elements elsewhere are related to more widely distributed QS and SX veins.

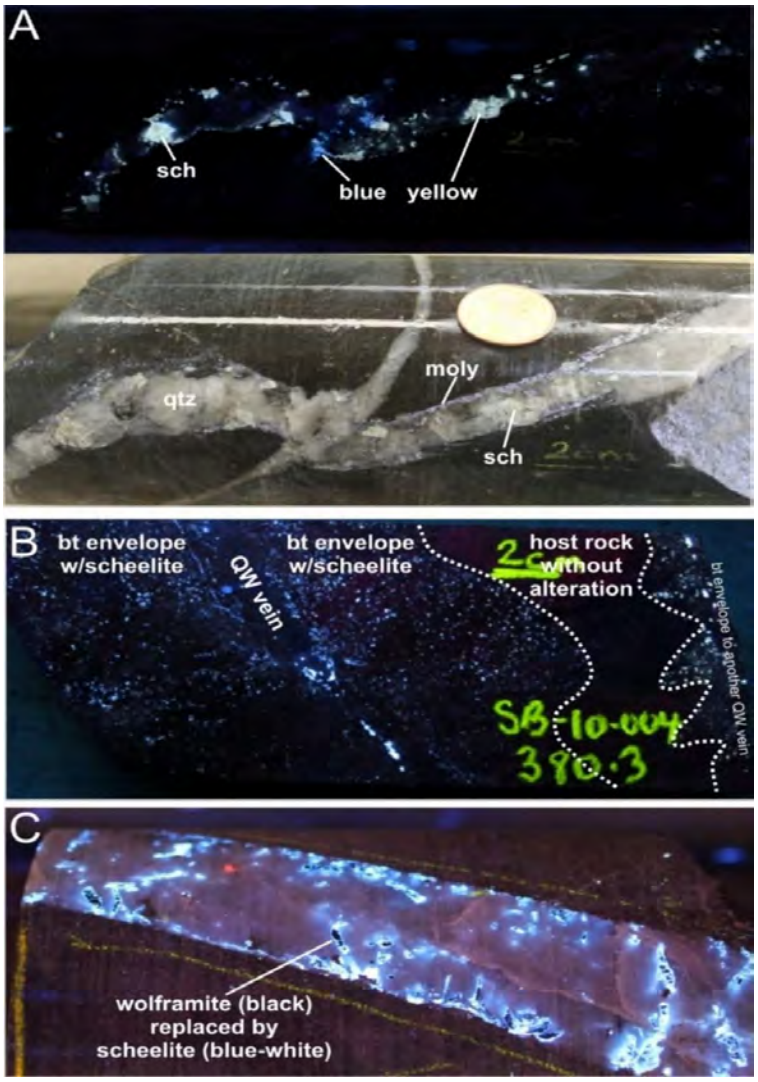
The grade of tungsten correlates only weakly with the composition of host rocks. Nast (1985) and Nast and Williams-Jones (1991) have suggested that the paucity of pervasive alteration and weak development of alteration envelopes to veins is compatible with a low flux of hydrothermal fluids. In such a rock-buffered hydrothermal system, they further suggest that scheelite preferentially precipitated in more mafic, calcium-rich host rocks. Empirically, there is a tendency for higher concentrations of scheelite to occur in mafic tuff, gabbro, and mafic crystal tuff than in adjacent felsic tuffs. The biotite wacke located along the central axis of Zone III (geological model unit WKB1; Figure 7-5) commonly has lower tungsten grade than adjacent volcanic host rocks. Host rock does not exhibit any apparent control on the grade of molybdenum.

Scheelite exhibits both blue and yellow fluorescence (Figure 7-9A) when exposed to shortwave ultraviolet (UV) light. Yellow fluorescence can indicate anything from end-member powellite ($CaMoO_4$) to minor substitution of molybdenum for tungsten in scheelite (referred to as molybdoscheelite). Electron microprobe analyses indicate that yellow fluorescence at Sisson reflects molybdoscheelite which contains a maximum of approximately 2 wt% molybdenum (Gregory, 2011). These results are consistent with Nast and Williams-Jones (1991) who report a maximum of approximately 4 wt% molybdenum in scheelite. These

results indicate that less than three percent of the total molybdenum inventory at Sisson might be sequestered in molybdoscheelite, using average deposit grades and a concentration of 1 wt% molybdenum, based on the distribution of the available data, in molybdoscheelite.

Oxidation related to surface weathering is mostly weak to moderate and confined to the upper 20 m of the deposit. Oxidation along fractures locally extends to greater depths, particularly along the structurally disrupted contact between gabbro and the Turnbull Mountain Formation. Where oxidation is strongest, primary rock textures remain visible and hypogene pyrite and pyrrhotite are, at most, only partially converted to iron oxides. Scheelite, wolframite, and molybdenite are essentially non-reactive in surface settings and are not significantly affected.

Figure 7-9: Examples of mineralization styles



Note: A. Vein-hosted scheelite and molybdenite mineralization. Scheelite includes both blue-fluorescent grains of pure scheelite and yellow-fluorescent grains of molybdoscheelite which contain a maximum of about 2 wt% molybdenum. Note the absence of fluorescent grains outside the vein itself. Drill hole SB11-019M. B. A narrow QW vein with blue-fluorescent scheelite surrounded by a biotite alteration envelope with abundant, very fine-grained, disseminated scheelite. This effect is most common in gabbro. Drill hole SB10-004 at 380.3 m depth. C. A late QS vein which contains wolframite (dark, elongate grains) partially to completely replaced by fluorescent scheelite. Drill hole SB11-019M. Source: RPA, 2012

8 Deposit types

8.1 Deposit model

The following is summarized from RPA (2012). The QP has reviewed and accepted this information.

The Sisson deposit can be defined as an intrusion related, structurally controlled, bulk tonnage tungsten-molybdenum deposit. Deposits of this type have general hydrothermal similarities to porphyry copper deposits. These types of deposit form in convergent margin to collisional tectonic environments and are related to highly evolved granitic melts formed from continental crust.

8.2 Provisional magmatic-structural-hydrothermal model

Tungsten-molybdenum deposits in New Brunswick have been linked genetically and temporally to hydrothermal fluids generated by Late Devonian felsic batholiths and intrusions formed during the Acadian Orogeny (e.g., Fyffe & Thorne, 2010, pp. 7-2, 7-5). The overlap between U-Pb ages on granite dykes and Re-Os dates on molybdenite support a similar link at Sisson (W. Zhang, pers. comm., 2011). A larger granitic pluton as a source for mineralizing hydrothermal fluids at Sisson has not been intersected by drilling thus far, but the numerous, albeit narrow, granitic dykes allow the presence of such a body to be confidently inferred at depth below the deposit.

Structures focused the ascent from depth of the intrusion derived hydrothermal fluids which formed the Sisson deposit. The physical relationship between QS and SX veins exposed in exploration Trench 1 in Zone I and the concentration of mineralization along the disrupted contact between gabbro and the Turnbull Mountain Formation suggest the possible nature of this control. Trench 1 is cut by several broadly north trending, approximately vertical QS veins that are between 0.3 m and 1.5 m in width. Kinematic indicators support long-lived to episodic, sinistral movement along the structures which host the QS veins. Sheeted arrays of northwest trending, steeply dipping, narrow SX veins, as well as fewer but larger, lensoidal quartz extension veins, form an envelope at approximately conjugate angles to the larger QS veins. Orientations obtained on 1,631 veins of all types in seven oriented geotechnical drill holes (Duncan, 2011) also show that most veins throughout the deposit form a sheeted array with northwest strike and steep southwest dip.

A provisional, highly simplified genetic model for Sisson therefore comprises: (1) emplacement of granitic magma at depth; (2) magmatic fractionation and release of hydrothermal fluids; (3) ascent of these fluids along north trending, sinistral fault zones; and (4) lateral distribution of mineralizing fluids along northwest trending, sheeted fracture arrays.

9 Exploration

9.1 Introduction

This section is taken from RPA (2012) and was prepared by Dr. James Lang, P. Geo., Senior Vice President Geology, HDI.

The QP has reviewed and accepted this information.

Early exploration programs, conducted prior to Northcliff's acquisition of the property, are discussed in Section 6, History of this Technical Report.

9.2 2011 Test Pit Program

In 2011, Northcliff conducted a test pitting program comprising 349 pits dug to bedrock using a Caterpillar 320 excavator. The purpose of the program was to assess exploration potential in areas of possible infrastructure development. A total of 409 till samples and 276 bedrock samples were collected from these pits for geochemical analysis. The locations of the pits were chosen by measuring approximately equidistant sections along pre-existing roads at spacings of between 50 m and 100 m. In addition to the samples collected in 2011, 147 till and 44 bedrock samples collected by Geodex in 2008 were also submitted for analysis. These sites were located between 2011 sample locations to ensure geochemical coverage of the target areas.

The physical characteristics of the till profile were documented as each pit was dug. Where bedrock was encountered, a sample was broken off by the excavator and rock type, grain size, foliation, mineralization, weathering, oxidation, veining, alteration, and colour were described. Bedrock samples were only collected where the material was angular and interpreted to be in place.

Other information recorded at each test pit site included UTM coordinates, the depth of the pit, depth to where the till began (below BC horizon), depth of till sample(s) taken, depth of bedrock sample taken, photo number, and whether any water seepage within the pit occurred. UTM coordinates were marked/saved in a GPS unit (Garmin 62s).

Till and bedrock samples were prepared for analysis by Activation Laboratories Ltd. (Actlabs) in Fredericton, New Brunswick. Full chain of custody control was maintained for all analytical samples from collection through delivery to the analytical laboratory. The till samples were dried and screened to 230 mesh. The concentration of tungsten was analyzed by instrumental neutron activation analysis (INAA). A suite of 58 additional elements was analyzed by a combination of INAA and inductively coupled plasma-mass spectrometry (ICP-MS) methods following four-acid digestion (Code UT-5 INAA(INAAGEO)/Total Digestion ICP/MS). All geochemical analyses were completed by Actlabs in Ancaster, Ontario. Bedrock samples were prepared and analyzed using protocols identical to those described in Section 11 of this Technical Report for drill core geochemical analysis.

Principal geological observations from the 2011 till and bedrock sampling program include the following:

- Analytical results for both bedrock and tills yield only a few scattered values for elements of economic interest at very low concentrations throughout the area of infrastructure.
- Slightly higher, but still weak, results were encountered over and immediately to the north of known mineralization in Zones I and II. These values decrease rapidly to the north.

- The results are fully consistent with the lack of mineralization and alteration in the hydrogeological and engineering drill holes completed in the area of potential infrastructure.
- The rock types encountered in bedrock samples generally conform to the distribution of major units as defined by published DNR regional geology maps.
- The results indicate that exploration potential for significant deposits of tungsten, molybdenum, or other metals in the area of potential infrastructure is very low.

9.3 Exploration potential

The Sisson deposit has been essentially closed off by drilling to the northeast, northwest, and south. At the present time, Northcliff considers the exploration potential in the immediate vicinity to be limited and has no plans for additional exploration work.

10 Drilling

10.1 Introduction

Since 1979, 69,321 m of diamond, reverse circulation (RC) and sonic drilling have been completed on the property from 407 drill holes (Table 10-1, Figure 10-1). For 2023, Northcliff drilled 10 (including one abandoned) successful reverse circulation geotechnical holes over the planned tailings area for hydrogeological purposes. Recently, for the 2025-2026 campaign, Northcliff successfully drilled 21 sonic overburden characterization and 19 (Inc. two abandoned) diamond, geotechnical and geomechanical drillholes for a total of 1,574 m as well as 36 Test Pits to an average overburden thickness of ~1.66 m. This work was completed across the planned Tailings Storage Facility peripheral abutment boundary to acquire both hydrogeological and geotechnical data (Figure 10-2). Drill data encompassing and ending on the Q1 2026 period was both validated and prepared by Cain Saint Merat, P. Geo., Senior Manager Resource Database & Geology, HDI.

Representative sections and a deposit plan view are shown in Figures 14-8 and 14-9.

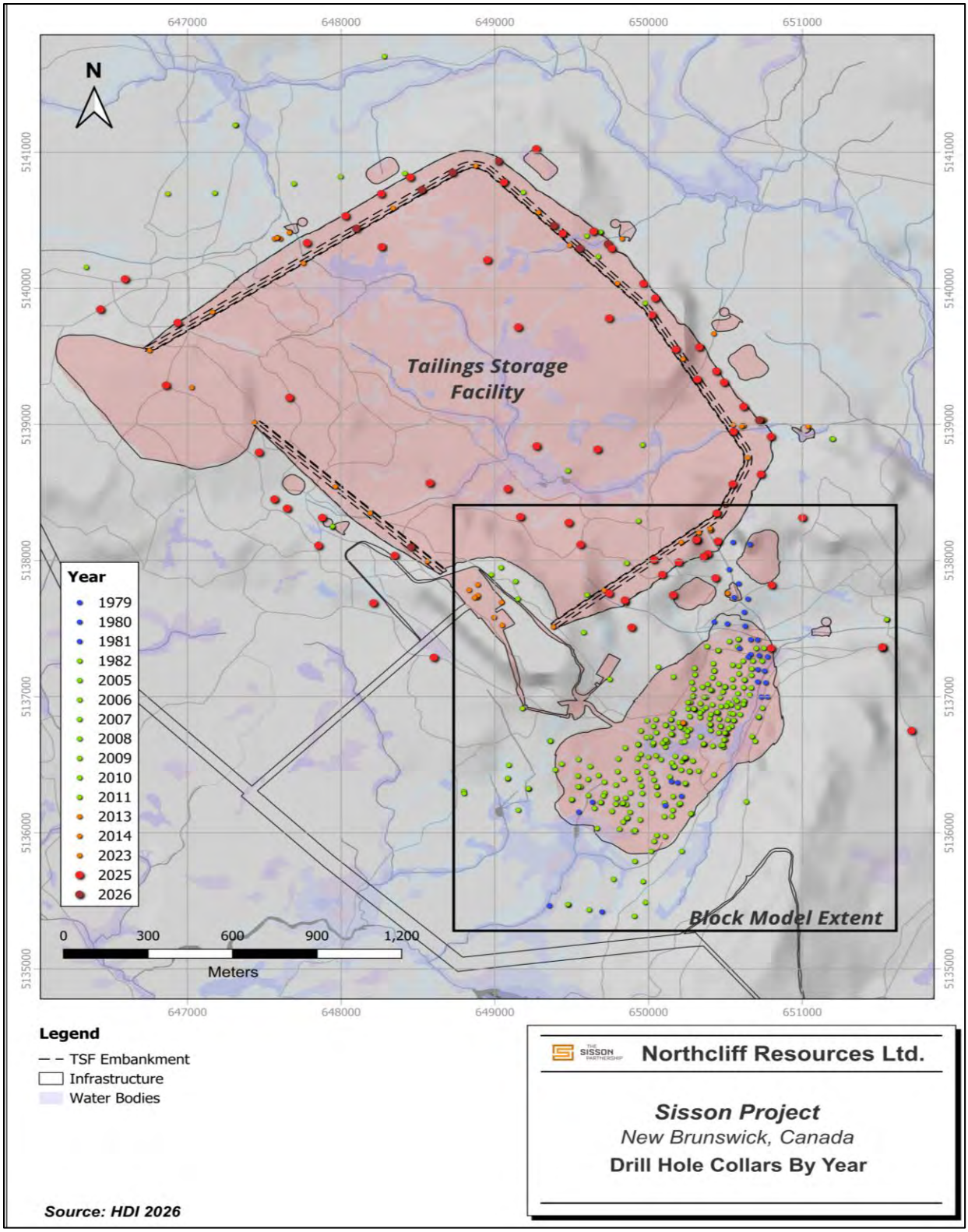
Table 10-1: Drilling summary all years

Operator	Year	Drill Hole ID	No. of Holes	Core Size[3]	Metres
Kidd Creek Mines Ltd.	1979	SSN01 to SSN11	11	BQ	1,663
	1980	SSN12 to SSN23	12	BQ	2,419
	1981	SSN24 to SSN32	9	BQ	1,727
	1982	SSN33 to SSN40	8	BQ	1,697
Geodex Minerals Ltd.	2005	SB-05-01 to SB-05-09	9	NQ	1,220
	2006	SB-06-01 to SB-06-30	30	NQ	7,787
	2007	SB-07-01 to SB-07-75 [1]; SBM-07-01	83	BTW, NQ	20,178
		SE-07-01 to SE-07-05			
		SP-07-01			
	2008	SB-08-01 to SB-08-47 [2]; BB08-01 to BB08-03; GZ-08-01 to GZ-08-02; MP-08-02 to MP-08-09; & PP-08-01	63	BTW, HQ	12,570
	2009	SB-09-1 to SB-09-28	28	NQ	4,901
Northcliff Resources Ltd.	2010	SB-10-001 to SB-10-005	5	HQ	2,700
	2011	SB-11-006M to SB-11-037G	61	PQ and HQ	8,474
		SB-11-MW-001D to SB-11-MW-006D			
		SB-11-MW-001S to SB-11-MW-006S			
		SB-11-MWG-001 to SB-11-MWG-018			

Operator	Year	Drill Hole ID	No. of Holes	Core Size[3]	Metres
	2013	SB-MWG13-018 to SB-MWG13-046	29	HQ	1,522
	2014	SB-14-038M to SB-14039M	2	PQ	325
	2014	SB-MWG14-049 to SB-MWG14-055	7	HQ	150
	2023[4]	SB-MWG23-056 to SB-MWG23-064	10	RC	443
	2025[5]	SB-OVB25-001 to SB-OVB25-021; SB-GTC25-001 to SB-GTC25-003; & SB-GMC25-001 to SB-GMC25-006	30	HQ3 (Sonic & Diamond)	884
	2026[5]	SB-GTC26-001 to SB-GTC26-005; & SB-GMC26-001 to SB-GMC26-005	10	HQ3 (Diamond)	663
Total			**407**	-	**69,321**

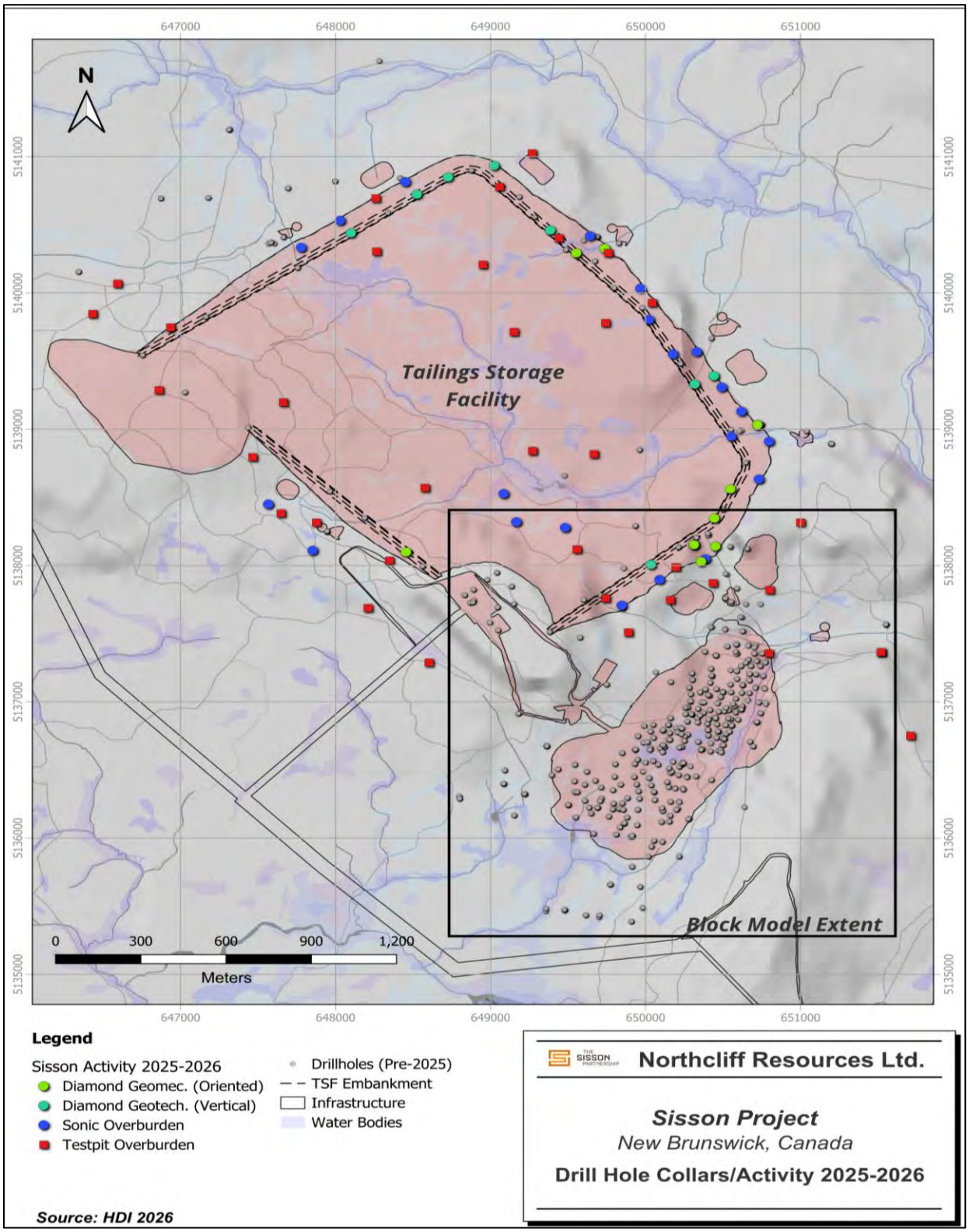
Notes:
1. Including hole SB-07-10A
2. Including hole SB-08-7A, SB-08-23A and SB-08-34A
3. Diameters: BQ – 36.5 mm, NQ – 47.6 mm, HQ – 63.5 mm, PQ – 85.0 mm
4. Including hole SB-MWG23-058B
5. Holes: SB-GMC25-001 (4.85 m) & SB-GMC26-001 (3.00 m) were abandoned due to both a core barrel malfunction and an unfavourable azimuth alignment, respectively.

Figure 10-1: Diamond drill hole locations by year



Source: HDI, 2026

Figure 10-2: Drill hole locations by 2025–2026 hole type and program



Source: HDI, 2026

The following describes the drilling activities carried out by Kidd Creek, Geodex, and Northcliff from 1979 to 2026 on the Sisson property.

10.2 Geodex and Kidd Creek programs (1979-2009)

Diamond drilling to delineate the W-Mo ± Cu mineralization in Zones I, II, and III on the Sisson property was completed by Kidd Creek during the 1979 to 1982 period and Geodex in the 2005 to 2009 period. Kidd Creek completed 40 drill holes totaling 7,506 m on the property in three campaigns. Geodex drilled 213 additional holes totaling 46,655 m in five subsequent campaigns to the end of 2009.

The initial digital compilation of the historical drilling data was carried out by Geodex staff. Drilling information, such as lithologic and sampling logs, assay results, collar survey data and downhole survey data, was compiled from hard copy assessment reports filed with the New Brunswick government and from in-house records and reports. The Geodex compilation was provided to Northcliff, and additional information was added where available.

Ideally, Drilling Ltd. provided contract drilling services to Kidd Creek for the 1979 through 1981 drilling campaigns, and Petro Drilling Limited provided services in 1982. In all cases, BQ-size equipment, producing drill core measuring approximately 36.5 mm in diameter, was used. Kidd Creek staff supervised the on-site geological work and also carried out core logging, sampling, and interpretive and reporting functions. Kidd Creek programs were coordinated by the company's Fredericton exploration office.

The 2005 drilling program by Geodex was contracted to Maritime Diamond Drilling Limited (Maritime) in Hilden, Nova Scotia. Maritime recovered NQ-size drill core measuring approximately 47.6 mm in diameter. In 2006, Maritime initiated the Zone III program using two drill rigs, both recovering NQ core. Geodex also contracted Lantech Drilling Limited (Lantech) of Moncton, New Brunswick, to assist in the completion of the 2006 drill program. Lantech also recovered NQ core. Geodex staff supervised all aspects of the Project, including on-site supervision, core logging, sampling, interpretive and reporting functions. Lantech was retained by Geodex to carry out all 2007 and 2008 drilling on the Sisson property, and Maritime carried out the 2009 program. The Geodex programs were directed by staff operating out of their Fredericton office.

With the exception of one track-mounted drill rig operated by Maritime in 2006, conventional skid-mounted drilling equipment was utilized.

Drill core from all programs completed prior to 2006 is stored at the New Brunswick DNR core storage facility near Sussex, New Brunswick. Core from 2006 through 2009 programs were held in a secure storage facility by Geodex. Northcliff took over this storage facility in 2011.

Drill hole collar locations for all Kidd Creek holes were originally surveyed and coordinated to Mining Lease survey pins in the area. Most of these drill collars in the Zone III area were relocated by Geodex in 2006 and subsequently resurveyed along with all Geodex holes of the 2006 through 2009 programs. All Geodex surveying was carried out by suitably certified technical staff employed by an independent, third-party surveying firm. UTM Zone 19T grid coordinates based on North American Datum 1983 (NAD 83) were generated from this program. Holes were typically tested for inclination and azimuthal variation using downhole survey instruments, and these data were incorporated into the Project database for use in the deposit model. Early holes were tested using mechanical instrumentation (Tropari) and those subsequent to 2005 used FlexIt (Reflex Instruments) downhole orientation tools. Mercator independently checked drill hole collar location coordinates for selected drill holes completed by Geodex through September 2009 and consistently found an excellent agreement with the Geodex coordinates.

10.3 Northcliff drilling

10.3.1 2010 Program

Northcliff contracted Lantech to carry out a seven-hole drilling program that was initiated in October 2010. Conventional skid-mounted drilling equipment recovering HQ-size core (63.5 mm diameter) was used, and the program was terminated in mid-December 2010 after completion of only five holes (2,700 m). Professional and technical staff acting on behalf of Northcliff managed all aspects of the Project, including on-site supervision, core logging, sampling, interpretive, and reporting functions. Field activities were coordinated from offices in Fredericton, New Brunswick, and the Geodex core logging facility located at Keswick Ridge, New Brunswick, was used for logging and sampling purposes.

Northcliff hole locations were set out by suitably certified technical staff, and locations were surveyed by an independent, third-party surveying firm. UTM Zone 19T grid coordinates based on NAD 83 were generated from this program and incorporated into the Project drilling database. Holes were tested for inclination and azimuthal variation using electronic (Reflex) downhole survey instruments, and these data (Table 10-2) were incorporated in the Northcliff database. Remaining half-core and coarse rejects are stored at the Northcliff core logging facility located on Kingsley Road north-northwest of Fredericton. The sample pulps are stored at a secure warehouse separate from the core logging facility.

Table 10-2: Northcliff 2010 drill hole coordinates and orientations

Hole ID	Easting [1]	Northing	Elevation (m)	Length (m)	Azimuth (°)2	Dip (°)	Purpose
SB-10-001	650334.1	5136881.7	303.2	528	270	-70	Exploration
SB-10-002	650161.7	5137146.5	327.8	356	90	-50	Exploration
SB-10-003	650286.7	5137002.4	303.7	570	90	-55	Exploration
SB-10-004	649827.4	5136215.4	300.8	674	90	-59	Exploration
SB-10-005	650092.4	5136660.2	305.5	573	90	-55	Exploration

Notes:
1. Coordinates are in NAD83 Zone19
2. Azimuth and dip are for the collar

10.3.2 2011 Program

In 2011, 61 drill holes totaling 8,474 m were completed by Northcliff. The purpose of this program was to provide additional geological, metallurgical, hydrological, and geotechnical information for a feasibility study to be completed in the third quarter of 2012 (Northcliff News Release dated March 05, 2012). The following information was obtained:

- Metallurgical and comminution sample material (15 holes, 3,048 m).
- Hydrogeological information (12 holes, 275 m).
- Engineering data in areas of proposed infrastructure (17 holes, 608 m).
- Pit wall geotechnical data in oriented drill holes (seven holes, 1,650 m).
- Additional deposit delineation and exploration to depth (ten holes, 2,901 m).

Northcliff contracted Layne Christiansen Canada Ltd. to carry out the 2011 drilling program initiated in June of that year. Conventional skid-mounted drilling equipment recovering both PQ-size drill core measuring approximately 85.0 mm in diameter and HQ-size core was used, and the program was successfully completed in October 2011. Professional and technical staff acting on behalf of Northcliff managed all aspects of the Project, including on-site supervision, core logging, sampling, and interpretive and reporting functions. Field activities were coordinated from offices located in Fredericton, New Brunswick.

Drill hole locations were set out by suitably certified technical staff, and locations were surveyed by an independent, third-party surveying firm. UTM coordinates (NAD 83 Zone 19T) generated from this program were incorporated in the Project drilling database developed by Northcliff. Holes were tested for downhole inclination and azimuth variation using Reflex electronic survey instruments, and these data were also incorporated in the Northcliff database. Holes collared to provide additional deposit delineation and exploration to depth were oriented to intersect the mineralization as close to perpendicular as possible. However, given the steep nature of mineralization this was difficult to achieve, and most holes intersected with the mineralization obliquely. The 2011 drill hole coordinates and orientations are listed in Table 10-3.

Table 10-3: Northcliff 2011 drill hole coordinates and orientations

Hole ID	Easting [1]	Northing	Elevation (m)	Length (m)	Azimuth (°)2	Dip (°)	Purpose
SB-11-006M	650,361.5	5,136,996.5	305.7	201.0	275	-70	Metallurgy
SB-11-007M	650,502.6	5,136,730.7	300.1	150.0	95	-55	Metallurgy
SB-11-008M	650,498.5	5,137,074.5	305.3	189.1	275	-70	Metallurgy
SB-11-009M	650,405.1	5,136,871.3	306.7	252.0	275	-70	Metallurgy
SB-11-010M	650,469.1	5,136,782.0	301.1	198.0	90	-55	Metallurgy
SB-11-011M	649,832.2	5,136,217.7	300.8	147.0	50	-70	Metallurgy
SB-11-012M	650,574.0	5,136,968.5	299.2	201.0	275	-70	Metallurgy
SB-11-013M	649,714.0	5,136,291.9	298.8	200.7	50	-70	Metallurgy
SB-11-014M	650,471.0	5,136,928.3	307.1	201.0	275	-70	Metallurgy
SB-11-015M	650,296.2	5,136,910.7	304.3	201.0	250	-70	Metallurgy
SB-11-016M	650,550.3	5,136,874.6	299.2	252.0	275	-70	Metallurgy
SB-11-017M	650,168.3	5,136,487.7	304.9	201.0	275	-70	Metallurgy
SB-11-018M	650,252.7	5,136,746.6	304.0	252.0	275	-70	Metallurgy
SB-11-019M	650,147.1	5,136,725.8	302.9	201.0	275	-70	Metallurgy
SB-11-020M	650,617.2	5,136,944.0	299.4	201.0	275	-70	Metallurgy
SB-11-021	649,810.5	5,136,027.3	293.2	508.4	45	-60	Exploration
SB-11-022G	650,617.2	5,136,944.8	299.4	216.0	90	-65	Geotechnical
SB-11-023	649,803.2	5,136,543.8	313.0	282.0	95	-55	Exploration
SB-11-024	649,990.0	5,136,825.0	319.6	129.1	275	-65	Exploration
SB-11-025G	650,495.4	5,136,734.6	300.8	303.0	130	-65	Geotechnical
SB-11-026	649,838.4	5,136,393.8	303.6	405.0	95	-65	Exploration

Hole ID	Easting[1]	Northing	Elevation (m)	Length (m)	Azimuth (°)2	Dip (°)	Purpose
SB-11-027	650,424.6	5,136,427.5	301.6	204.0	95	-55	Exploration
SB-11-028	650,205.2	5,136,216.3	310.6	141.0	95	-75	Exploration
SB-11-029G	650,497.1	5,137,071.7	305.4	219.0	320	-65	Geotechnical
SB-11-030	650,274.9	5,136,139.7	311.9	150.0	95	-55	Exploration
SB-11-031	649,542.5	5,136,338.9	300.5	507.0	45	-55	Exploration
SB-11-032G	650,294.9	5,136,910.4	304.3	321.0	280	-65	Geotechnical
SB-11-033	650,693.5	5,136,673.4	308.9	135.0	95	-55	Exploration
SB-11-034	649,912.0	5,136,018.1	296.4	441.0	100	-55	Exploration
SB-11-035G	649,931.5	5,136,648.5	310.2	180.0	280	-65	Geotechnical
SB-11-036G	649,706.1	5,136,272.1	298.3	201.0	230	-65	Geotechnical
SB-11-037G	650,238.4	5,136,364.8	314.8	210.0	100	-65	Geotechnical
SB-11-MW-001D	647,944.0	5,138,245.0	300.0	38.0	0	-90	Hydrogeology
SB-11-MW-001S	647,944.4	5,138,252.5	345.5	24.4	0	-90	Hydrogeology
SB-11-MW-002D	649,477.8	5,135,470.4	293.3	37.2	0	-90	Hydrogeology
SB-11-MW-002S	649,476.1	5,135,475.1	293.6	8.0	0	-90	Hydrogeology
SB-11-MW-003D	651,193.7	5,138,895.3	287.5	33.2	0	-90	Hydrogeology
SB-11-MW-003S	651,200.0	5,138,893.7	287.8	6.0	0	-90	Hydrogeology
SB-11-MW-004D	651,551.8	5,137,567.0	261.3	35.0	0	-90	Hydrogeology
SB-11-MW-004S	651,550.1	5,137,567.6	262.0	9.0	0	-90	Hydrogeology
SB-11-MW-005D	647,315.3	5,141,200.4	308.8	35.4	0	-90	Hydrogeology
SB-11-MW-005S	647,312.0	5,141,200.1	309.0	9.1	0	-90	Hydrogeology
SB-11-MW-006D	649,690.8	5,140,410.8	305.9	28.9	0	-90	Hydrogeology
SB-11-MW-006S	649,687.2	5,140,414.2	306.1	9.2	0	-90	Hydrogeology
SB-11-MWG-001	648,977.7	5,137,896.8	382.4	39.7	0	-90	Engineering
SB-11-MWG-002	649,135.0	5,137,847.7	386.3	29.9	0	-90	Engineering
SB-11-MWG-003	649,040.2	5,137,948.9	382.2	39.7	0	-90	Engineering
SB-11-MWG-004	649,149.8	5,137,718.7	388.4	29.8	0	-90	Engineering
SB-11-MWG-005	646,872.2	5,140,695.7	328.3	42.9	0	-90	Geotechnical
SB-11-MWG-006	647,692.3	5,140,769.7	320.0	35.3	0	-90	Geotechnical
SB-11-MWG-007	648,414.3	5,140,845.2	323.5	50.5	0	-90	Geotechnical
SB-11-MWG-008	649,184.6	5,140,705.9	332.8	32.5	0	-90	Geotechnical
SB-11-MWG-009	649,672.4	5,140,233.9	314.7	23.1	0	-90	Geotechnical
SB-11-MWG-010	649,978.2	5,139,892.0	340.6	22.5	0	-90	Geotechnical
SB-11-MWG-012	649,960.8	5,138,850.0	306.0	33.9	0	-90	Geotechnical
SB-11-MWG-013	649,934.3	5,138,292.2	317.6	39.6	0	-90	Engineering

Hole ID	Easting[1]	Northing	Elevation (m)	Length (m)	Azimuth (°)2	Dip (°)	Purpose
SB-11-MWG-014	649,600.4	5,137,748.0	336.6	36.9	0	-90	Engineering
SB-11-MWG-015	649,577.4	5,137,472.3	358.6	30.8	0	-90	Geotechnical
SB-11-MWG-016	646,341.6	5,140,156.2	354.3	35.3	0	-90	Geotechnical
SB-11-MWG-017	647,996.0	5,140,821.4	321.7	38.3	0	-90	Geotechnical
SB-11-MWG-018	647,178.0	5,140,700.6	320.8	39.7	0	-90	Geotechnical

Notes:
1. Coordinates are in NAD83 Zone19
2. Azimuth and dip are for the collar

10.3.3 2013 Program

Northcliff contracted Lantech to complete a 29-hole drilling program totaling 1,522 m from October 16, 2013, to December 4, 2013, using HQ wireline equipment. The program consisted of a series of relatively short (generally from 30 m to 70 m), vertical holes drilled in the area of the proposed tailings disposal area for geotechnical and condemnation purposes.

As with the previous drilling programs undertaken by Northcliff, professional and technical staff acting on behalf of Northcliff managed all aspects of the drilling, including on-site supervision, core handling and logging, sampling, and interpretive and reporting functions. Field activities were coordinated from offices located in Fredericton, New Brunswick. Drill hole locations were set out by suitably certified technical staff, and locations were surveyed by an independent, third-party surveying firm. UTM coordinates (NAD 83 Zone 19T) generated from this program were incorporated in the Project drilling database developed by Northcliff. Holes were tested for downhole inclination and azimuth variation using Reflex electronic survey instruments, and these data were also incorporated into the Northcliff database. The 2013 program was sampled using full standard company QA/QC protocols.

The 2013 drill hole coordinates and orientations are listed in Table 10-4.

Table 10-4: Northcliff 2013 drill hole locations and orientation

Hole ID	Easting[1]	Northing	Elevation (m)	Length (m)	Azimuth (°)2	Dip (°)	Purpose
SB-MWG13-018	647962.3	5138543.53	349.97	50.29	0	-90	Geotechnical
SB-MWG13-019	647754.8	5140185.47	321.38	60.96	0	-90	Geotechnical
SB-MWG13-020	649380.7	5137512.43	380.60	32.00	0	-90	Geotechnical
SB-MWG13-021	647663.6	5140410.00	318.87	30.48	0	-90	Geotechnical
SB-MWG13-022	648560.1	5137994.82	370.93	65.63	0	-90	Geotechnical
SB-MWG13-023	649485.9	5140317.49	309.06	70.10	0	-90	Geotechnical
SB-MWG13-024	648885.5	5137729.92	392.06	30.79	0	-90	Geotechnical
SB-MWG13-025	649827.8	5140365.49	299.91	54.87	0	-90	Geotechnical
SB-MWG13-026	648337.7	5140589.21	324.14	51.81	0	-90	Geotechnical
SB-MWG13-027	648876.1	5140900.11	339.87	53.65	0	-90	Geotechnical

Hole ID	Easting [1]	Northing	Elevation (m)	Length (m)	Azimuth (°)2	Dip (°)	Purpose
SB-MWG13-028	649284.2	5140558.03	322.40	62.29	0	-90	Geotechnical
SB-MWG13-029	650550.1	5138997.41	304.05	96.01	0	-90	Geotechnical
SB-MWG13-030	646754.3	5139544.30	380.33	55.37	0	-90	Geotechnical
SB-MWG13-031	647028.1	5139271.89	399.26	30.68	0	-90	Geotechnical
SB-MWG13-032	647435.5	5139014.68	380.36	38.10	0	-90	Geotechnical
SB-MWG13-033	647163.4	5139825.42	355.15	41.15	0	-90	Geotechnical
SB-MWG13-034	649797.3	5140036.75	327.78	51.81	0	-90	Geotechnical
SB-MWG13-035	650327.2	5138205.25	304.27	67.06	0	-90	Geotechnical
SB-MWG13-036	650225.5	5139481.63	321.22	60.96	0	-90	Geotechnical
SB-MWG13-037	650424.7	5139668.07	310.67	47.55	0	-90	Geotechnical
SB-MWG13-038	649711.9	5137781.43	324.37	57.91	0	-90	Geotechnical
SB-MWG13-039	650751.50	5139034.28	295.06	44.19	0	-90	Geotechnical
SB-MWG13-040	651038.6	5138986.11	292.41	29.46	0	-90	Geotechnical
SB-MWG13-041	650212.4	5138135.08	323.35	70.10	0	-90	Geotechnical
SB-MWG13-042	649855.6	5137686.95	319.17	42.67	0	-90	Geotechnical
SB-MWG13-043	650514.4	5137760.30	289.18	44.20	0	-90	Geotechnical
SB-MWG13-044	648188	5138349.86	337.96	70.11	0	-90	Geotechnical
SB-MWG13-045	650502.2	5138410.38	324.49	59.43	0	-90	Geotechnical
SB-MWG13-046	650645.9	5138755.70	324.25	52.12	0	-90	Geotechnical
Total	-	-	-	**1,521.75**	-	-	-

Notes:
1. Coordinates are in NAD83 Zone19. 2. Azimuth and dip are for the collar.

10.3.4 2014 Program

In 2014, Northcliff completed two separate diamond drilling programs.

Northcliff contracted Progressive Diamond Drilling (Progressive) to complete two PQ-diameter holes totaling 325 m during September to provide material for additional metallurgical test work.

Simultaneously, Northcliff contracted Lantech to complete an additional seven HQ diameter holes totaling 150 m.

As with the previous drilling programs undertaken by Northcliff, professional and technical staff acting on behalf of Northcliff managed all aspects of the drilling, including on-site supervision, core handling and logging, sampling, and interpretive and reporting functions. Field activities were coordinated from offices located in Fredericton, New Brunswick. Drill hole locations were set out by suitably certified technical staff, and locations were surveyed by an independent, third-party surveying firm. UTM coordinates (NAD 83 Zone 19T) generated from this program were incorporated into the Project drilling database developed by Northcliff. Holes were tested for downhole inclination and azimuth variation using Reflex electronic survey instruments, and these data were also incorporated into the Northcliff database.

The 2014 drill hole coordinates and orientations are listed in Table 10-5.

Table 10-5: Northcliff 2014 drill hole locations and orientation

Hole ID	Easting [1]	Northing	Elevation (m)	Length (m)	Azimuth (°)[2]	Dip (°)	Purpose
SB-14-038M	650222.56	5136810	304.06	179.00	35	-70	Metallurgy
SB-14-039M	650225.58	5136807	303.94	146.00	270	-80	Metallurgy
SB-MWG14-049	649047.77	5137525	397.21	16.72	0	-90	Engineering
SB-MWG14-050	648994.05	5137582	395.64	24.32	0	-90	Engineering
SB-MWG14-051	649044.91	5137694	399.49	24.24	0	-90	Engineering
SB-MWG14-052	648889.90	5137744	393.43	26.02	0	-90	Engineering
SB-MWG14-053	648867.96	5137728	390.95	17.95	0	-90	Engineering
SB-MWG14-054	648832.04	5137784	390.06	22.74	0	-90	Engineering
SB-MWG14-055	648889.41	5137823	391.32	18.10	0	-90	Engineering
Total	-	-	-	**475.09**	-	-	-

Notes:
1. Coordinates are in NAD83 Zone19
2. Azimuth and dip are for the collar.

10.3.5 2023 Program

In 2023, Northcliff contracted ER Steeves Well Drilling to complete 10 RC holes totaling 443 m to collect hydrogeological information.

As with the previous drilling programs undertaken by Northcliff, professional and technical staff acting on behalf of Northcliff managed all aspects of the drilling, including on-site supervision, core handling and logging, and interpretive and reporting functions. Field activities were coordinated from offices located in Fredericton, New Brunswick. Drill hole locations were set out by suitably certified technical staff, and locations were surveyed by an independent, third-party surveying firm. UTM coordinates (NAD 83 Zone 19T) generated from this program were incorporated in Northcliff's Project drilling database. Holes were tested for downhole inclination and azimuth variation using Reflex electronic survey instruments, and these data were also incorporated into the Northcliff database.

The 2023 RC holes were not sampled for assaying.

The 2023 RC drill hole coordinates and orientations are listed in Table 10-6.

Table 10-6: Northcliff 2023 drill hole locations and orientation

Hole ID	Easting [1]	Northing	Elevation (m)	Length (m)	Azimuth (°)[2]	Dip (°)	Purpose
SB-MWG23-056	647566.10	5140365.03	322.34	48.16	0	-90	Hydrogeology
SB-MWG23-057	647604.43	5140364.37	321.92	48.16	0	-90	Hydrogeology
SB-MWG23-058A	647586.57	5140374.92	321.90	5.49	0	-90	Hydrogeology
SB-MWG23-058B	647582.60	5140371.74	321.82	47.85	0	-90	Hydrogeology
SB-MWG23-059	650612.56	5138995.18	298.23	53.95	0	-90	Hydrogeology
SB-MWG23-060	650618.45	5138990.88	297.82	47.85	0	-90	Hydrogeology
SB-MWG23-061	650604.84	5138986.54	298.40	48.01	0	-90	Hydrogeology
SB-MWG23-062	650408.21	5138226.52	302.60	47.85	0	-90	Hydrogeology
SB-MWG23-063	650402.78	5138222.04	302.24	47.85	0	-90	Hydrogeology
SB-MWG23-064	650400.89	5138236.61	302.68	47.85	0	-90	Hydrogeology
Total	-	-	-	**443.03**	-	-	-

Notes:
1. Coordinates are in NAD83 Zone19. 2. Azimuth and dip are for the collar.

10.3.6 2025-2026 Program

Between 2025-2026, a total of 40 drill holes totaling 1,547 m were completed by Northcliff. The purpose of this program was to provide additional geological, hydrological, and geotechnical information for the Tailings Storage Facility peripheral abutment and better understand both surface and bedrock characteristics to aid in overall engineering design. The drilling was part of a broader designed program segregated into 4 principal activities:

- Geotechnical drill program including both geotechnical (vertical) and geomechanical (inclined) drillholes to define bedrock (near surface) structures including down-hole, in situ packer hydrological testing (21 planned holes, of which 19 were executed including 2 abandoned holes for 1,266 m).
- Sonic drill program, to obtain undisturbed samples to characterize both sub-surface till and soil horizons (i.e. overburden) (21 holes for 282 m).
- Widely distributed excavation (test-pit) program to support and complement the detailed sub-surface characterization drill works (52 planned Test-Pits, of which 36 were executed to an average of 1.66 m).
- Seismic program to support the primary three penetrative works with the intent to determine any obscured and covered geological bedrock features (58 station points for 11,350 lined metres).

Northcliff contracted Lantech Drilling Ltd. to carry out the geotechnical program, 403 Drilling Ltd. to carry out the sonic program, Geophysics GPR to conduct the seismic program, and a local contractor was utilized to excavate the test pitting. All four work activities were executed simultaneously with 403 Drilling commencing on the 28th of October 2025 and continuing across into a winter program with Lantech Drilling finishing the program on the 18th of February 2026. All drilling equipment was mounted on tracked rigging recovering HQ-size drill core. Due to safety concerns, two geotechnical drillholes were deferred for the later part of 2026 (pending) and are planned to be executed as part of the early fall 2026 work activities. Professional and technical staff acting on behalf of both Northcliff and Knight Piésold managed all aspects

of the Project, (except for the seismic program). This includes on-site supervision, core logging, point-test sampling, core orientation, detailed geotechnical logging, packer testing, interpretive and reporting functions. Field activities were coordinated between both the Knight Piésold and Northcliff Vancouver head offices.

Drill hole locations were set out by suitably certified Knight Piésold technical staff, with locations still pending accurate surveying at the time this Technical Report was being compiled. UTM coordinates (NAD 83 Zone 19T) generated from this program were incorporated into the Project drilling database developed by Northcliff. Holes were tested for downhole inclination and azimuth variation using Reflex (IMDEX) electronic survey instruments (i.e. OMNIx42-Multi device). Holes were collared to provide orientation of water bearing structures and data was collected to better understand hydrogeological and permeability characteristics from intermittent packer testing throughout each of the geotechnical and geomechanical holes. The 2025-2026 geotechnical, geomechanical and sonic drillhole coordinates and orientations are listed in Table 10-7.

Table 10-7: Northcliff 2025-2026 drill hole locations and orientation

Hole ID	Easting[1]	Northing	Elevation (m)	Length (m)	Azimuth (°)[2]	Dip (°)	Purpose
SB-OVB25-001	649083.964	5138527.823	310.541	18.20	0.00	-90.00	Overburden Characterization
SB-OVB25-002	650090.306	5137897.150	325.259	7.50	0.00	-90.00	Overburden Characterization
SB-OVB25-003	650386.608	5138049.223	306.830	15.00	0.00	-90.00	Overburden Characterization
SB-OVB25-004	649846.408	5137708.574	321.049	7.50	0.00	-90.00	Overburden Characterization
SB-OVB25-005	650491.798	5139309.747	315.680	10.50	0.00	-90.00	Overburden Characterization
SB-OVB25-006	650618.579	5139132.108	306.916	13.50	0.00	-90.00	Overburden Characterization
SB-OVB25-007	650553.272	5138947.745	293.906	12.00	0.00	-90.00	Overburden Characterization
SB-OVB25-008	650330.622	5139569.277	315.342	30.00	0.00	-90.00	Overburden Characterization
SB-OVB25-009	650177.587	5139552.241	325.458	22.50	0.00	-90.00	Overburden Characterization
SB-OVB25-010	650023.297	5139805.359	338.276	6.00	0.00	-90.00	Overburden Characterization
SB-OVB25-011	650796.801	5138911.663	291.219	3.00	0.00	-90.00	Overburden Characterization
SB-OVB25-012	650731.663	5138634.385	331.140	6.00	0.00	-90.00	Overburden Characterization
SB-OVB25-013	647565.868	5138451.310	356.004	7.50	0.00	-90.00	Overburden Characterization

Hole ID	Easting[1]	Northing	Elevation (m)	Length (m)	Azimuth (°)[2]	Dip (°)	Purpose
SB-OVB25-014	647852.312	5138111.178	346.999	18.00	0.00	-90.00	Overburden Characterization
SB-OVB25-015	648028.689	5140532.418	326.707	16.50	0.00	-90.00	Overburden Characterization
SB-OVB25-016	648449.957	5140816.841	325.110	25.90	0.00	-90.00	Overburden Characterization
SB-OVB25-017	647778.233	5140335.137	319.594	9.00	0.00	-90.00	Overburden Characterization
SB-OVB25-018	649642.263	5140419.082	307.838	18.00	0.00	-90.00	Overburden Characterization
SB-OVB25-019	649166.000	5138321.000	TBA	16.76	0.00	-90.00	Overburden Characterization
SB-OVB25-020	649481.000	5138279.000	TBA	9.14	0.00	-90.00	Overburden Characterization
SB-OVB25-021	649967.000	5140035.000	TBA	9.00	0.00	-90.00	Overburden Characterization
SB-GTC25-001	650035.258	5138007.172	335.067	40.85	0.00	-90.00	Geotechnical (Vertical)
SB-GTC25-002	650441.484	5139392.274	315.853	41.15	0.00	-90.00	Geotechnical (Vertical)
SB-GTC25-003	650317.363	5139330.793	318.650	60.37	0.00	-90.00	Geotechnical (Vertical)
SB-GTC26-001	649385.086	5140461.434	322.000	61.79	0.00	-90.00	Geotechnical (Vertical)
SB-GTC26-002	649023.067	5140935.983	339.404	30.20	0.00	-90.00	Geotechnical (Vertical)
SB-GTC26-003	648725.683	5140851.431	332.536	60.12	0.00	-90.00	Geotechnical (Vertical)
SB-GTC26-004	648524.780	5140725.058	326.629	60.55	0.00	-90.00	Geotechnical (Vertical)
SB-GTC26-005	648099.371	5140440.996	326.000	60.65	0.00	-90.00	Geotechnical (Vertical)
SB-GMC25-001	650312.979	5138152.752	310.134	4.85	265.00	-54.00	Geomechanical (Oriented)
SB-GMC25-002	650312.979	5138152.752	310.134	122.60	262.00	-58.00	Geomechanical (Oriented)
SB-GMC25-003	650359.000	5138029.000	312.000	62.20	282.19	-54.53	Geomechanical (Oriented)
SB-GMC25-004	650547.810	5138562.252	326.399	74.30	276.00	-56.00	Geomechanical (Oriented)
SB-GMC25-005	650443.961	5138347.535	TBA	101.40	272.27	-54.56	Geomechanical (Oriented)

Hole ID	Easting[1]	Northing	Elevation (m)	Length (m)	Azimuth (°)[2]	Dip (°)	Purpose
SB-GMC25-006	650451.135	5138141.914	300.999	95.20	274.90	-58.98	Geomechanical (Oriented)
SB-GMC26-001	650721.741	5139033.223	296.620	3.00	276.51	-58.51	Geomechanical (Oriented)
SB-GMC26-002	650721.741	5139033.223	296.620	61.17	297.50	-58.30	Geomechanical (Oriented)
SB-GMC26-003	649736.354	5140326.943	308.319	100.93	345.72	-59.55	Geomechanical (Oriented)
SB-GMC26-004	649552.626	5140293.659	309.737	101.50	139.95	-59.56	Geomechanical (Oriented)
SB-GMC26-005	648455.467	5138100.085	361.508	122.70	126.76	-64.65	Geomechanical (Oriented)
Total	-	-	-	1,547	-	-	-

Notes:
1. Coordinates are in NAD83 Zone19.
2. Azimuth and dip are for the collar. All holes were recorded using a handheld GPS at the time this Technical Report was compiled and are pending RTK surveying.

10.4 Core recovery

Table 10-8 summarizes the core recovery for the diamond drilling programs conducted from 2006 onwards. The QP notes that the recovery appears to be good.

Table 10-8: Core recovery

Year	Number of Intervals	Mean Core Recovery (%)
2006	2,341	98.0
2007	6,632	97.1
2008	3,871	97.6
2009	1,561	97.4
2010	894	98.0
2011	2,415	98.6
2013	985	90.3
2014	149	96.3
2025	203	97.2
2026	221	97.8
Total	**19,272**	**97.23**

10.5 Bulk density

In previous estimates, density values were calculated using a water immersion method. A summary by year is presented in Table 10-9.

The procedures of the water immersion method are as follows:

- Whole core samples which are typical of the surrounding rock and free of visible moisture were selected. They ranged from 7 to 20 cm in length, and averaged 15 cm.
- The sample was weighed in air on a digital scale and the mass in air (Ma) recorded to the nearest >0.1 g in an MS Excel spreadsheet.
- The sample was then suspended in water below the scale and the mass in water (Mw) was entered into the same sheet.
- Bulk density (BD) was calculated by the following formula: BD = Ma/(Ma – Mw).

Table 10-9: Bulk density sampling summary

Year	Number	BD Min (t/m³)	BD Max (t/m³)	Mean (t/m³)	Median (t/m³)
2006	205	2.61	3.51	2.79	2.77
2007	172	2.59	3.02	2.76	2.76
2008	40	2.61	2.92	2.82	2.84
2009	60	2.59	2.94	2.76	2.76
2010	224	2.59	3.17	2.76	2.76
2011	1,042	2.56	3.37	2.77	2.77
Total	**1,743**	**2.56**	**3.51**	**2.77**	**2.77**

10.6 QP opinion

The QP has reviewed Northcliff's current drill hole database and the information summarized in the sections above and is of the opinion that the drilling was performed according to industry standard practices and is of sufficient quality to support the estimation of Mineral Resources for public disclosure.

The QP is not aware of any drilling, sampling, or recovery factors that could materially impact the accuracy and reliability of the results.

11 Sample preparation, analyses, and security

11.1 Introduction

This section was largely summarized from information prepared by Eric Titley, P. Geo., Senior Manager Resource Geology, HDI, to reflect the drilling to end of 2011 and was updated by SLR to include the results of the 2013 and 2014 drilling.

Table 11-1 is a summary of the sampling and sample preparation work performed on the Project since 1979. The analytical history is summarized in Table 11-2.

Table 11-1: Sampling and sample preparation summary

Year	No. of Drill Holes	Operator	Method	Sample Length		Sample Preparation Laboratory
				Range	Typical	
1979	11	Kidd Creek Mines Ltd	Half Split	0.05 - 4.1 m Selected Intervals	2.0 - 4.1 m Selected Intervals	Bondar-Clegg Ottawa, ON
1980	12					
1981	9					
1982	8					
2005	9	Geodex Minerals Ltd	Half Split	0.12 - 2.56 m Continuous Intervals	1.5 m Continuous Intervals	Actlabs Ancaster ON or Fredericton, NB
2006	29		Half Split Mechanical	0.5 - 1.5 m Continuous Intervals	1.5 m Continuous Intervals	
2007	82					
2008	49					
2009	28					
2010	n/a	Northcliff Due Diligence	Pulp Split Reject Split 1/4 Core Split	1.5 m Selected Intervals	1.5 m Selected Intervals	Acme Vancouver, BC
2010	5	Northcliff	Half Split Diamond Saw	0.7 – 5.0 m Continuous Intervals	3.0 m Continuous Intervals	Actlabs Fredericton, NB
2011	61	Northcliff	Half Split Diamond Saw	0.7 – 5.0 m Continuous Intervals	3.0 m Continuous Intervals	Actlabs Fredericton, NB
2013	29	Northcliff	Half Split Diamond Saw	0.7 – 5.0 m Continuous Intervals	3.0 m Continuous Intervals	Actlabs Fredericton, NB
2014	7	Northcliff	Half Split Diamond Saw	0.7 – 5.0 m Continuous Intervals	3.0 m Continuous Intervals	Actlabs Fredericton, NB

Table 11-2: Assay summary

Year	No. of DDH	Assay Laboratory	Elements Assayed	Method and Code	Digestion or Activation	Finish
1979	11	Bondar-Clegg Ottawa, ON	WO_3, Cu, Mo	Unknown	Unknown	Unknown
1980	12					
1981	9					
1982	8		WO_3, Cu, Mo, Ag, As, Bi	Colorimetric; titration; AAS; FA	Thiocyanate for WO_3; HF + HNO_3 for Cu and Bi; FA for Ag; Na_2O_2 fusion for As	Colorimetric for WO_3; I2 titration for As; AAS for Cu, Mo, Ag, Bi
2005	9	Actlabs Ancaster, ON	W, Mo, Au, Ag, Cd, Cu, Mn, Ni, Pb, Zn, S, As, Ba, Hg, Sb	Multi-element analytical package IEPI AQUAGEO	Neutron activation for W, Au, As, Ba, Sb, and Hg; AR digestion for Ag, Cd, Cu, Mn, Mo, Pb, Zn, and S	W, Au, As, Ba, Sb, and Hg by INAA; Ag, Cd, Cu, Mn, Mo, Pb, Zn, and S by ICP
2006	29		W, Mo, Cu	INAA INAAGEO	Neutron activation for W, Mo; aqua regia digestion for Cu	W and Mo by INAA; Cu by ICP
2007	82					
2008	49					
2009	28					
2010	Due Dil.	Acme Vancouver, BC	W, Mo, Cu and 38 additional elements	1EX 7KP2	Phosphoric acid digest for W and 4 additional elements, 4 acid digestion for Cu and 40 additional elements	ICP-AES for W and 4 additional elements, ICP-MS for Cu and 40 additional elements.
2010	5	Actlabs Ancaster, ON	W, Mo, Cu and 61 additional elements	Geodex-INAA (INAAGEO) UT-2-0.5 g	Neutron activation for W, Mo; aqua regia digestion for Cu and 61 elements	W and Mo by INAA; Cu and 61 elements by ICP-OES/MS
2011	61	Actlabs Ancaster, ON	W, Mo, Cu and 61 additional elements	Geodex-INAA (INAAGEO) UT-2-0.5 g	Neutron activation for W, Mo; aqua regia digestion for Cu and 61 elements	W and Mo by INAA; Cu and 61 elements by ICP-OES/MS
2013	29	Actlabs Ancaster, ON	W, Mo, Cu and 61 additional elements	Geodex-INAA (INAAGEO) UT-2-0.5 g	Neutron activation for W, Mo; aqua regia digestion for Cu and 61 elements	W and Mo by INAA; Cu and 61 elements by ICP-OES/MS
2014	7	Actlabs Ancaster, ON	W, Mo, Cu and 61 additional elements	Geodex-INAA (INAAGEO) UT-2-0.5 g	Neutron activation for W, Mo; Aqua Regia digest for Cu and 61 elements	W and Mo by INAA; Cu and 61 elements by ICP-OES/MS

Notes: AAS – atomic absorption spectrometry; FA – fire assay; INAA - instrumental neutron activation analysis; ICP-AES - inductively coupled plasma- atomic emission spectroscopy; ICP-OES/MS - inductively coupled plasma- optical emission spectroscopy/mass spectrometry.

11.2 Kidd Creek samples (1979-1982)

According to Mercator (2011), documentation of the Kidd Creek drilling programs does not provide much detail on the sample preparation methodologies, analytical procedures, or security considerations. The bagged samples of half core material were shipped from Kidd Creek's Fredericton office to Bondar-Clegg laboratories in Ottawa, Ontario, where sample preparation and assay were conducted. Assays from 1979 to 1981 were done for copper and WO_3 and in 1982 for copper, WO_3, silver, arsenic, and bismuth. The analytical methods used were as follows:

- WO_3 – thiocyanate digestion with colorimetric finish
- Mo – nitric and hydrochloric acid (HNO_3+HCl) digestion with atomic absorption spectrometry (AAS) finish
- Cu – hydrofluoric and nitric acid (HF+HNO_3) digestion with AAS finish
- Ag – fire assay (FA) fusion with AAS finish
- Bi – HF + HNO_3 digestion with AAS finish
- As – Na_2O_2 fusion and then titrated by I2 solution.

In addition, certain samples were selected for geochemical analysis. Table 11-3 lists the analytical methods used in the 1982 program.

Table 11-3: Analytical methods 1982 program

Type	Element	Fraction	Digestion or Extraction	Finish
Assay	WO_3	-200 Mesh	Thiocyanate	Colorimetric
Assay	Cu	-200 Mesh	HF + HNO_3	Atomic Absorption
Assay	Ag	-200 Mesh	Fire Assay	Atomic Absorption
Assay	As	-200 Mesh	Na Peroxide Fusion	Iodometric Titration
Assay	Bi	-200 Mesh	HF + HNO_3	Atomic Absorption
Geochemical	Pb, Zn, Bi	-200 Mesh	HNO_3	Atomic Absorption
Geochemical	Mo	-200 Mesh	HNO_3 + HCl	Atomic Absorption
Geochemical	Sb, Sn	-200 Mesh		X-Ray Fluorescence
Geochemical	F	-200 Mesh	Fusion	Specific Ion Electrode
Geochemical	As	-200 Mesh	HNO_3 + $HClO_4$	Atomic Absorption

No mention is made of sample or site security measures in the Kidd Creek reports. It is assumed that security measures consistent with industry standards of the day were in place during the 1979 through 1982 drilling programs.

11.3 Geodex samples (2005-2009)

Actlabs performed sample preparation and analytical work for the 2005 through 2009 drilling programs. Two Actlabs facilities, one in Ancaster, Ontario, and the other in Fredericton, New Brunswick, were used. Actlabs is an independent commercial laboratory. Core samples were placed in plastic sample bags, checked for sample sequence integrity, and shipped by courier to the laboratory. Core samples were crushed, split, and then pulverized to 95% passing 105 µm (150 mesh).

In 2005, INAA was used to determine gold, arsenic, barium, antimony, mercury, and tungsten and inductively coupled plasma atomic emission spectroscopy (ICP-AES) was used to determine silver, cadmium, copper, manganese, molybdenum, nickel, lead, zinc, and tin after aqua regia digestion.

From 2006 and onward, Geodex focused on tungsten and molybdenum, as the main metals of economic interest. Tungsten and molybdenum were determined by the INAA method with included tungsten and molybdenum standards. Copper in selected samples was determined by aqua regia digestion with ICP-AES finish. In addition, some samples were selected for multi-element analysis using a variety of assay methods (Table 11-2).

In addition to the mainstream samples analyzed by Actlabs, check duplicates from the 2006 drilling program were analyzed by ALS Chemex (ALS) in Vancouver, British Columbia, using X-ray fluorescence (XRF) on pressed pellets. In the 2007, 2008, and 2009 drilling programs, check duplicate analyses were carried out at Global Discovery Laboratory in Vancouver, British Columbia (acquired by Acme Analytical Laboratories [Acme] in 2008).

Mercator (2009a) reported that a system of drill core security was implemented during the 2006 program that began at the drill site and continued through transportation to the sampling and logging facility, and then onward through shipment of samples to the laboratory. At the drill site, the security of core was the responsibility of the contractor under the direction of Geodex. Drill crews were instructed that only they and designated Geodex personnel were to access core at the drill sites and that core boxes should be covered immediately after drilling. Designated field transfer points were established at which responsibility for core security passed from the drilling contractor to Geodex. This was coordinated daily between the drilling contractor crews and Geodex personnel. Core delivered to the facility was placed in a secure location prior to logging and sampling, and Geodex personnel were responsible for security of all core and samples at this facility. Prior to sampling, a digital photograph of each box of core was taken and archived for future reference. After core logging and sampling were completed, core boxes containing half-core archives were labelled with weather-proof aluminum tags and were bound with steel strapping and stored at the logging facility on pallets. Core sample numbers were recorded on sample shipment forms, and after checking against sample listings, were placed in sealed plastic buckets for shipment to the laboratory. Details of the shipment were recorded, including date shipped, sample numbers, client account number, laboratory name, and name of the individual responsible for preparation of the documents and security of the sample shipment preparation process.

11.4 Northcliff due diligence (2010)

As part of a due diligence review, a sampling program of core, rejects, and pulps was carried out by Darrel Johnson, P.Geo., of HDI. Eighteen core samples, 25 coarse rejects, and 237 pulp samples from site were collected on behalf of Northcliff and submitted to Acme Analytical Laboratories (Vancouver) Ltd., an independent commercial laboratory in Vancouver, British Columbia. The samples were prepared and analysed at Acme and results were approved by a British Columbia Certified Assayer. All core samples were

crushed to 75% passing a 2 mm screen (10 mesh) and a 250 g subsample was taken. Pulverizing was performed on the 250 g splits to 85% passing 75 µm (200 mesh).

Molybdenum, copper, and 39 additional elements were determined by four acid digestion with an ICP-MS finish (Acme code: 1EX). Molybdenum, tungsten, and three additional elements were re-assayed by phosphoric acid leach with ICP-AES finish (Acme code: 7KP2). In addition, 27 external standards and five blanks were inserted and assayed along with the mainstream samples.

11.5 Northcliff samples (2010)

Sampling during Northcliff's 2010 drilling program was carried out under the supervision of Dr. James Lang, P.Geo., of HDI.

Core logging and sampling activities were conducted by Northcliff at the Geodex logging facility located in Keswick Ridge, New Brunswick. Procedures for core handling and processing were generally similar to those employed by Geodex. Drill cores were transported from the Project site to the core logging facility by company truck. Core received from the field was washed and prepared for initial inspection under UV light to define major rock units and associated mineralized intervals. This information was used to provide a preliminary assessment of mineralization intensity. Core was then stacked for subsequent detailed logging and sampling.

After logging by a Northcliff geologist, but prior to sampling, all core was photographed using a standardized layout format and digital camera to provide a permanent pre-sampling record of core from each drill hole. Continuous rock quality and core recovery determinations were also recorded for all cores and entered into the digital project database. Conventional core logging procedures were standardized through use of a project lithological legend and lithocode system with log information entered directly into a digital logging utility accessed via the logging geologist's laptop computer and later integrated into the digital project database.

Core sampling by Northcliff was conducted using diamond saws. One half of each split core sample was placed in a pre-labelled plastic bag along with a corresponding sample book tag and then sealed and laid out for checking prior to insertion of quality assurance/quality control (QA/QC) materials into the associated sample sequence. A corresponding tag was secured to the core box for archive purposes. Tracking of sample intervals and QA/QC materials was integrated with the logging function and thereby within the digital project database.

The core samples were collected by site geological staff. Most sample intervals were from continuous 3 m lengths of core. Some interval lengths varied depending on the geologic features encountered. The sample intervals were marked and then the core samples were cut in half lengthwise with a diamond saw. The samples were sealed in plastic bags and sent to Actlabs sample preparation laboratory in Fredericton, New Brunswick.

Core samples and associated QA/QC samples were prepared at Actlabs sample preparation laboratory in Fredericton, New Brunswick, and analyzed at Actlabs in Ancaster, Ontario.

At Actlabs in Fredericton, the samples were sorted, dried, weighed, and then crushed to 75% passing 2 mm (10 mesh). After crushing, an 800 g subsample was split and pulverized to 95% passing 105 µm (150 mesh). The pulverized subsample pulps were sent to Actlabs in Ancaster, Ontario, for assay. Actlabs in Ancaster is ISO 17025 accredited for specific registered tests. In addition, it is also accredited to CAN-P-1579, specific to mineral analysis laboratories.

At Actlabs in Ancaster, tungsten and molybdenum were determined by the Geodex INAA method (Actlabs code: INAAGEO); copper, silver, sulphur, and an additional 59 elements were analyzed by aqua regia digestion with inductively couple plasma optical emission spectroscopy/mass spectrometry (ICP-OES/MS) finish (Actlabs code: UT-2-0.5g).

In-line duplicates were assayed by Actlabs along with the mainstream samples using the same method. Inter-laboratory check assays were performed by Acme. Acme is ISO certified (since 1996) and is currently registered to the ISO 9001:2000 quality system standards. Tungsten, molybdenum, and three additional elements were determined by phosphoric acid leach with ICP-AES finish (Acme code: 7KP2). Molybdenum, copper, and 39 additional elements were determined by four-acid digestion with ICP-MS finish (Acme code: 1EX).

A similar system of drill core security, as used by Geodex in 2005 through 2009, was implemented by Northcliff for the 2010 drill program.

11.6 Northcliff samples (2011)

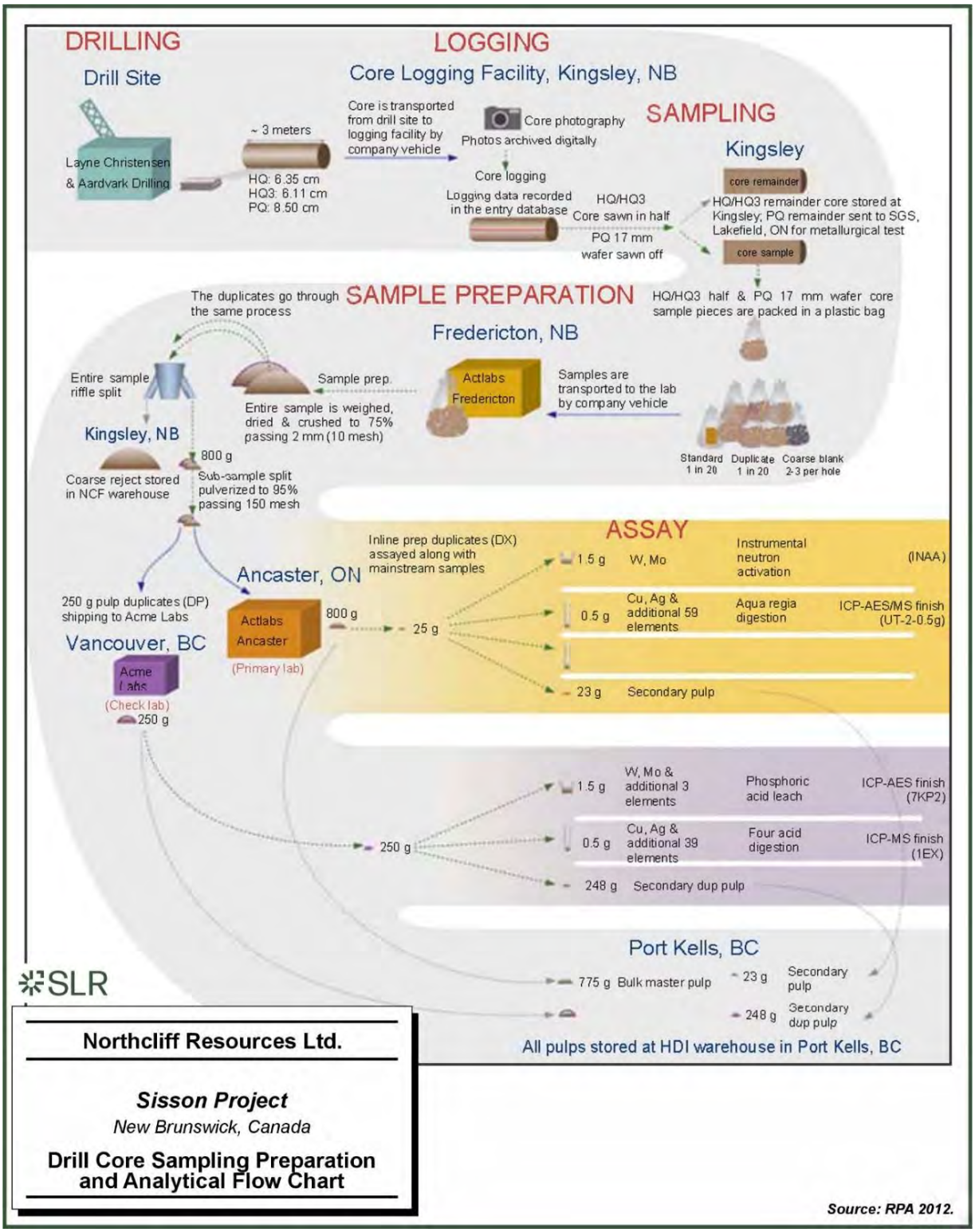
Sampling during the 2011 drilling program was carried out under the supervision of Dr. James Lang, P.Geo., of HDI, using essentially the same procedures as those used in 2010. Mainstream sample preparation and assay work continued to be performed by Actlabs using the same procedures as for the 2010 drill program. Sample preparation and test work on metallurgical samples was performed by SGS of Lakefield, Ontario.

Figure 11-1 is a flow chart illustrating the 2011 drill core sampling, sample preparation, and assay analysis.

11.7 Northcliff samples (2013-2014)

No documentation describing the core handling, logging, and sampling procedures during the 2013 and 2014 drilling programs was available, however, the QPs understand that these procedures were largely consistent with those adhered to by Northcliff during the 2010 and 2011 drilling programs.

Figure 11-1: 2011 Drill core sampling, sample preparation, and analysis flow chart



Source: RPA, 2012

11.8 Quality Assurance/Quality Control

This section is based on RPA (2012) and was largely prepared under the direction of Eric Titley, P. Geo., Senior Manager Resource Geology, HDI, with assistance from Weidong Yang, Ph.D., and Catherine Sidwell.

11.8.1 Earlier operators

11.8.1.1 Kidd Creek (1979-1982)

Reports documenting the Kidd Creek Mines drilling programs do not specifically address QA/QC issues and no evidence was noted of any external QA/QC samples inserted with core samples. However, it was reported that 67 original sample pulp splits from drill hole SSN-33 were reassayed and overall reproducibility of assay values between sample splits was found to be consistent and acceptable.

The overall impact of the Kidd Creek drill holes on the current Sisson resource estimate is limited as many of the holes are outside the current study area. Of the 31,900 core samples assayed on the Project from 1979 to the end of 2011, only 2,700, or about 8.5% of the total, are from the Kidd Creek drill holes. In 2007 and 2008, Geodex sampled and assayed 769 intervals from the historic Kidd Creek drill core, which had never previously been assayed.

11.8.1.2 Geodex (2005-2009)

During the 2005 drill program, no external QA/QC samples were applied by Geodex. Only internal standards, blanks, and duplicates administered by Actlabs were used to assess the analytical results. From 2006 onward, Geodex implemented a QA/QC protocol which included the use of external standards and blanks, which were inserted and analyzed with the mainstream core samples. Table 11-4 is a summary of the regular mainstream samples and additional QA/QC samples analyzed in the years 2005 through 2009.

In the 2006 drill program, Geodex completed 29 holes and collected 4,876 core samples. The company adopted a QA/QC program that included systematic insertion of certified standards and barren rock blanks and analysis of duplicate samples. As shown in Table 11-4, 52 standards and 193 blanks were inserted and assayed along with mainstream samples. A total of 161 check duplicates including 147 mainstream duplicates and 14 blanks were submitted to the ALS Chemex laboratory in Vancouver, British Columbia, for check analysis.

Table 11-4: Assay QA/QC summary for 2005-2009

Year	MS	DP	ST	BL	ST %
2005	515	-	-	-	-
2006	4,876	147	52	193	4
2007	11,940	320	498	522	4
2008	5,114	186	213	219	4
2009	2,945	121	156	129	4
ALL	25,390	774	919	1,063	4

Note: MS – mainstream sample; DP – duplicate; ST- standard; BL – blank; ST% - standards as a percent of the total mainstream samples

In the 2007 drill program, Geodex drilled 83 holes with a total length of 20,194 m. A total of 11,940 core samples were collected and 498 standards and 522 blanks were applied. In addition, 417 check duplicates (320 mainstream duplicates and 97 blank duplicates) were assayed at Actlabs.

Geodex completed 49 holes in 2008 with a total length of 12,122 m and collected 5,114 core samples. They submitted 213 external standards and 219 external blanks with these samples for analysis as Actlabs. A total of 190 check duplicates (186 mainstream duplicates and four blank duplicates) were also assayed.

A 4,899 m, 28-hole program was completed by Geodex in 2009 in which 2,945 core samples were collected, and 156 standards and 129 blanks were included for analysis at Actlabs. In addition, 121 mainstream duplicates were selected for check assay.

In 2007 and 2008, Geodex also assayed 769 core samples from the Kidd Creek program, from core intervals that had not previously been assayed.

11.8.1.2.1 Standards

Table 11-5 lists the standard reference samples used in 2006 to 2009 exploration programs.

Table 11-5: Standards used by Geodex 2006 through 2009

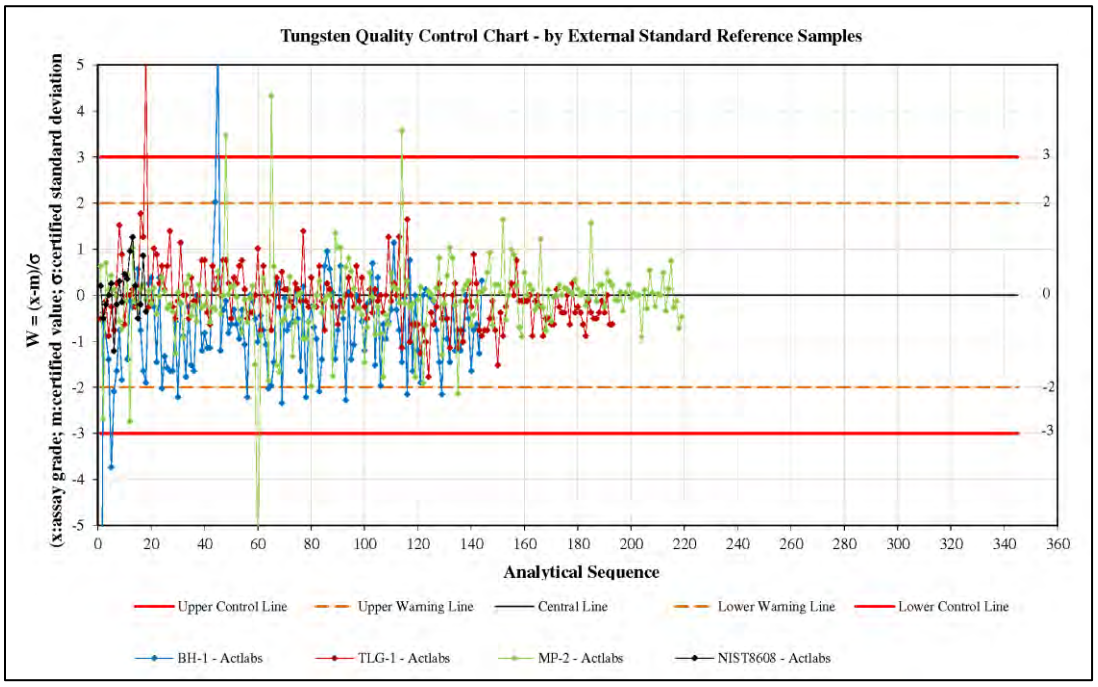
Standard	Times Used	W (%)	Mo (%)	Cu (%)
BH-1	144	0.422	0.02	-
HV-2	345	-	0.048	0.57
MP-2	219	0.65	0.281	-
NIST8608	18	0.22	0.098	0.096
TLG-1	193	0.083	-	-

Note: Value in italics is not certified

Figure 11-2 and Figure 11-3 show the combined standard control charts based on Geodex 2006 and 2009 assay QA/QC results.

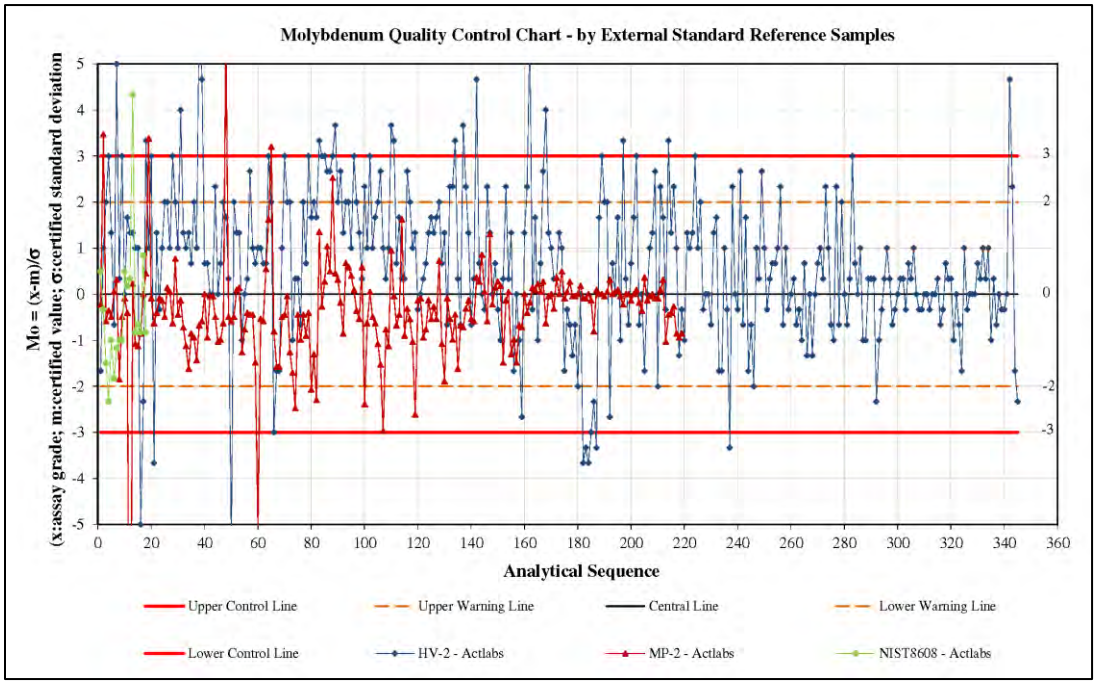
In the QP's opinion, the standards results from the Geodex assays show a fairly broad scatter but are generally within acceptable limits and unbiased.

Figure 11-2: Tungsten quality control chart 2006–2009 drill programs



Source: RPA, 2012

Figure 11-3: Molybdenum quality control chart 2006–2009 drill programs

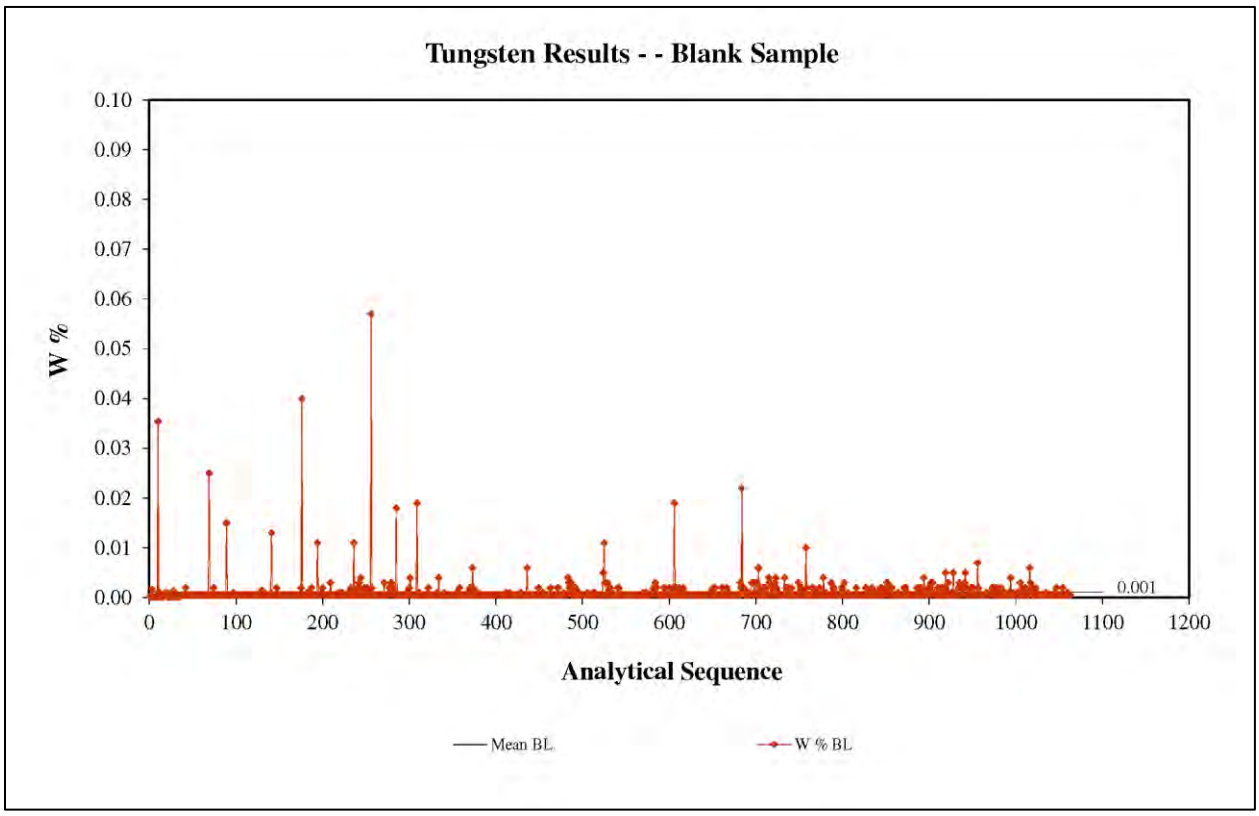


Source: RPA, 2012

11.8.1.2.2 Blanks

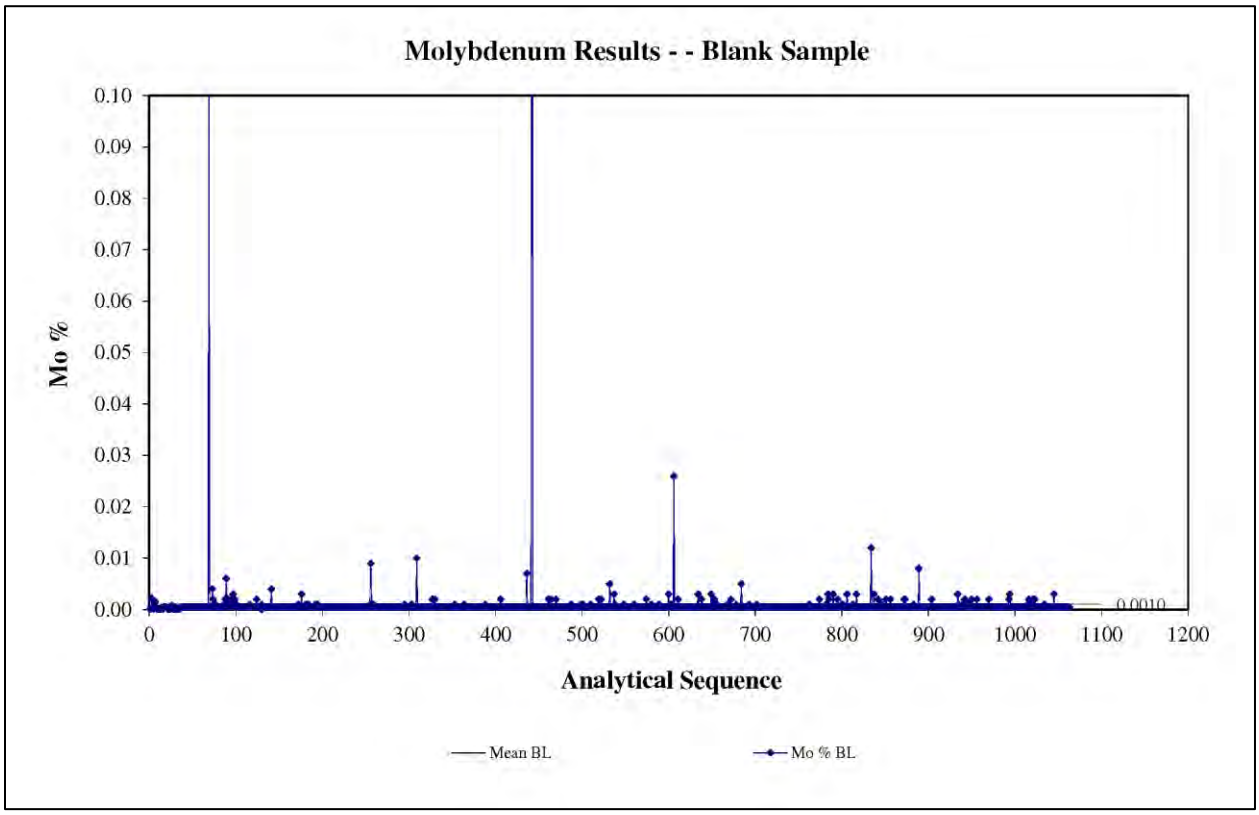
A total of 1,063 external blanks were included with the regular samples during the 2006 through 2009 drill programs. Figure 11-4 and Figure 11-5 show the results of tungsten and molybdenum respectively for these blanks. The assay grades of the blanks are generally low (≤0.001%). However, some are obviously higher than normal, which Northcliff personnel think may be due to some degree of cross-contamination during sample preparation, perhaps during the sample crushing process. In the QP's opinion, this is a plausible explanation. The total number of apparent failures relative to the number of determinations is within acceptable tolerance, and the failure rate has improved with time.

Figure 11-4: Tungsten results of blank samples 2006–2009 drill programs



Source: RPA, 2012

Figure 11-5: Molybdenum results of blank samples 2006–2009 drill programs

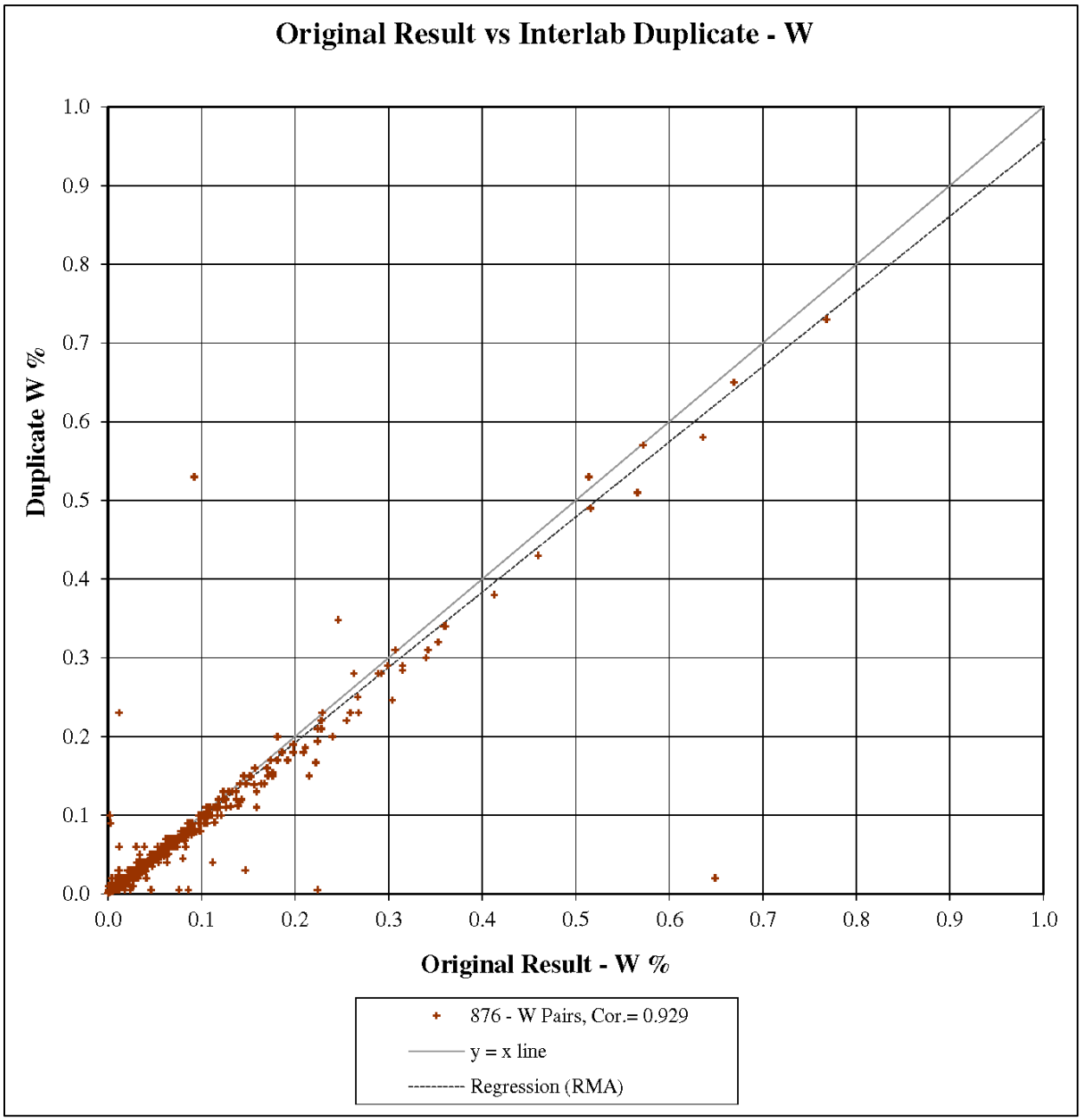


Source: RPA, 2012

11.8.1.2.3 Duplicates

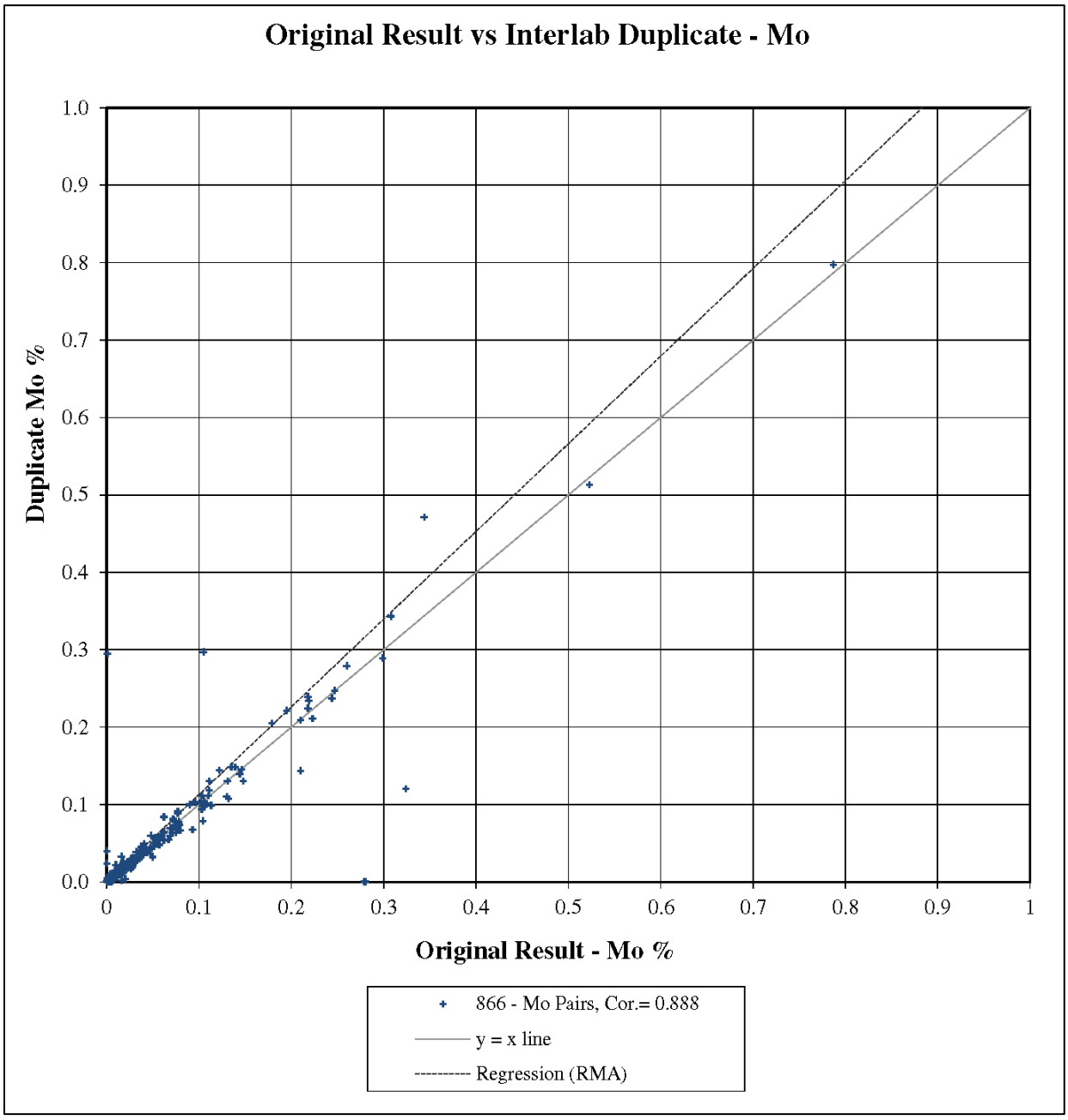
A total of 889 inter-laboratory pulp duplicates (774 mainstream duplicates and 115 blank duplicates) were assayed at a second laboratory during the 2006 through 2009 Geodex drilling programs. Figure 11-6 and Figure 11-7 are scatterplots of the matched pair duplicate results for tungsten and molybdenum respectively comparing Actlabs results on the X axis with the check laboratory results from ALS Chemex (2006) and Global Discovery Labs (2007-2009) on the Y axis.

Figure 11-6: Tungsten duplicates scatterplot 2006–2009 drill programs



Source: RPA, 2012

Figure 11-7: Molybdenum duplicates scatterplot 2006–2009 drill programs



Source: RPA, 2012

In the QP's opinion, the results for molybdenum demonstrate a small bias between the original and duplicate assays. On inspection of the data, this apparent bias was found to be due to one outlier, which plots outside the range of Figure 11-7. When this outlier is removed, the apparent bias disappears.

11.8.2 Northcliff due diligence sampling (2010)

As part of a due diligence review conducted by Northcliff in 2010, 18 core samples, 25 coarse rejects, and 237 pulps were selected and sent for reanalysis. Northcliff included 27 standards and five blanks with the samples and sent them to Acme in Vancouver for assay. The results from this resampling program confirmed the original results obtained by Geodex.

A total of 27 external standards and five blanks were included with this suite of samples. Table 11-6 is a summary of the due diligence sample types.

Table 11-6: Summary of 2010 QA/QC samples

Type	MS	ST	BL	Sum
¼ Core	18	2	1	21
Reject	25	2	0	27
Pulp	237	23	4	264
ALL	280	27	5	312

11.8.2.1.1 Standards

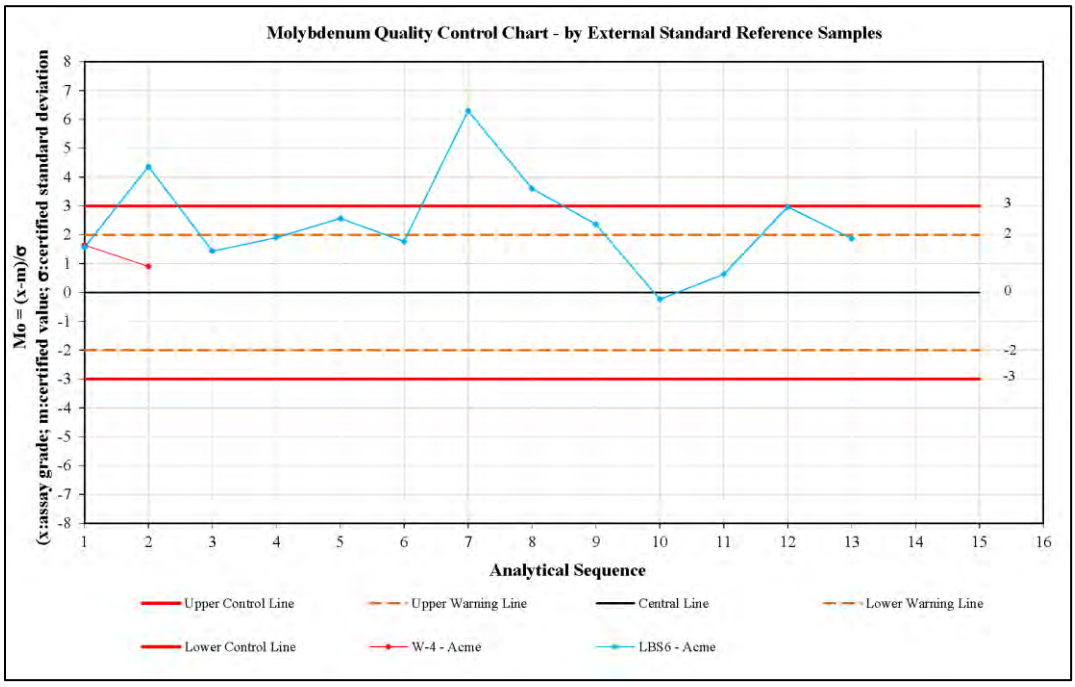
Three different assay standards were submitted as part of the due diligence program. Two commercially available standards W-4 and TLG-1 were submitted along with a proprietary copper-molybdenum standard. Table 11-7 lists the key parameters of these standards.

Table 11-7: Standards used in due diligence analyses

Standard	Times Used	W (%)	Mo (%)	Cu (%)
W-4	2	0.366	0.11	0.139
Cu-Mo	13	-	0.0146	1.23
TLG-1	12	0.083	-	-

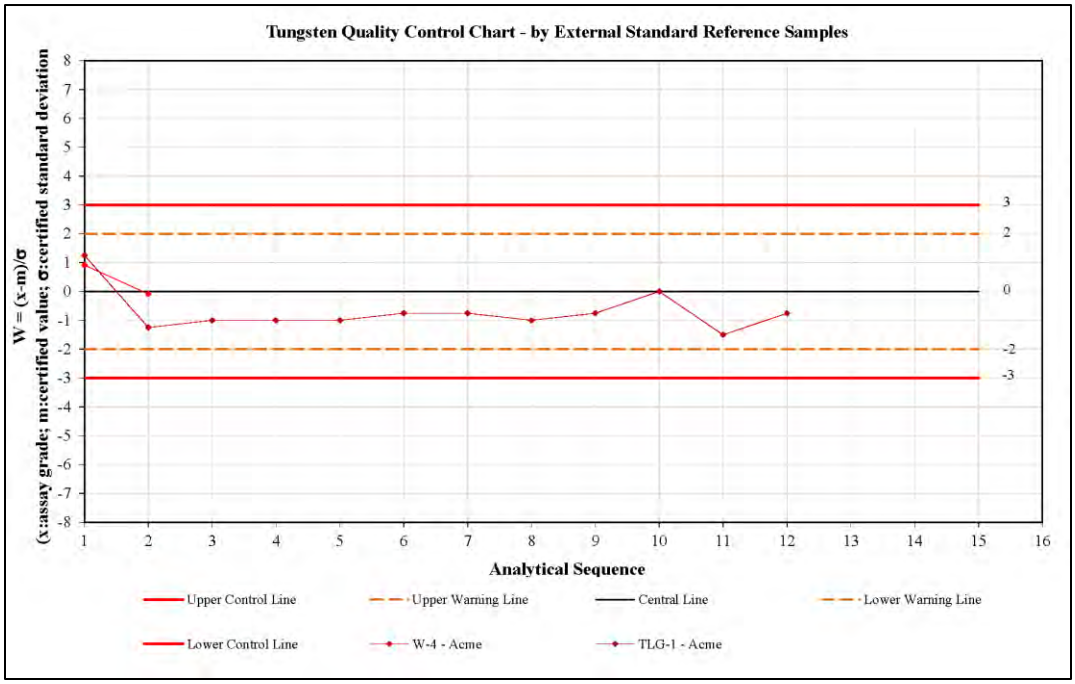
Figure 11-8 and Figure 11-9 show the assay results of these standards in the due diligence program. The assayed values of tungsten and molybdenum for W-4 and the assayed values of tungsten for TLG-1 are within the control limits. Three of the thirteen molybdenum values for the copper-molybdenum standard returned above upper control limit.

Figure 11-8: Molybdenum quality control chart for due diligence



Source: RPA, 2012

Figure 11-9: Tungsten quality control chart for due diligence



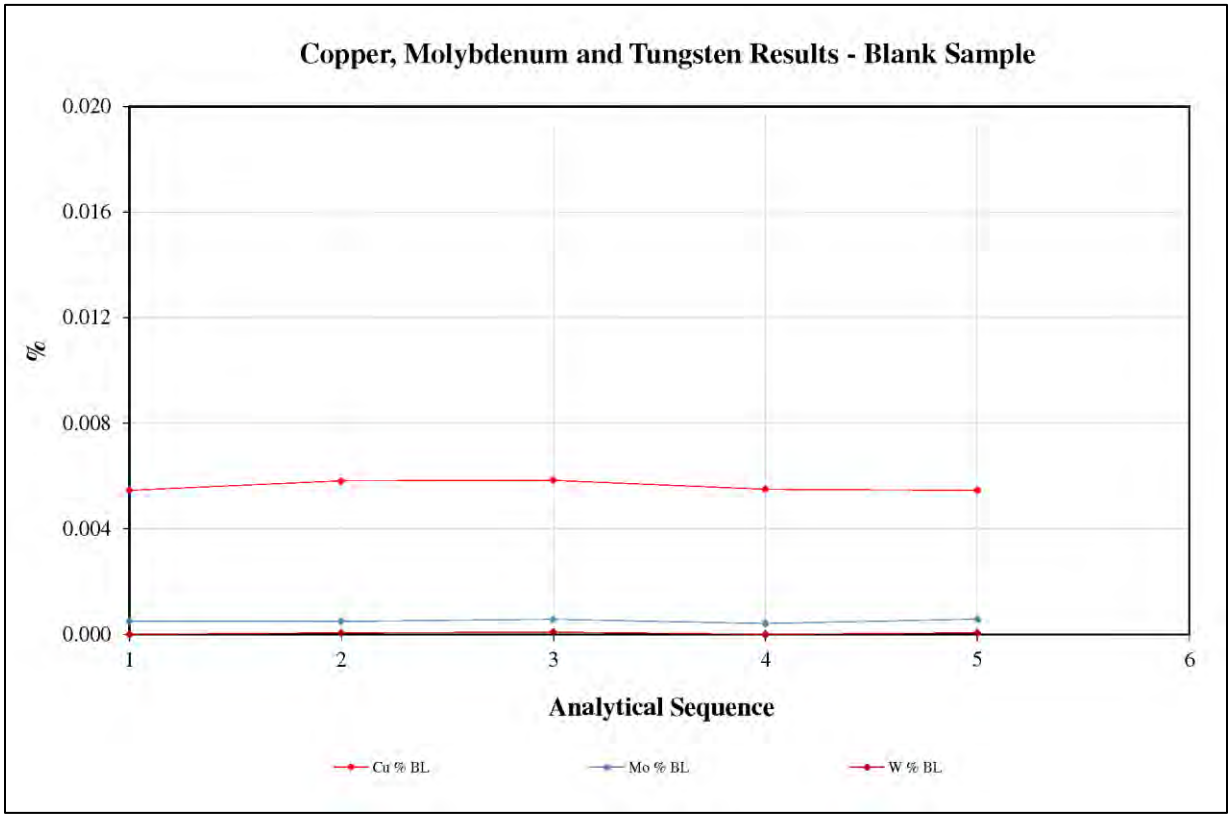
Source: RPA, 2012

The QP notes that the molybdenum standards results show a persistent positive bias. However, since these samples are not part of the resource estimation database, there is no concern with respect to the block grade interpolations.

11.8.2.1.2 Blanks

A total of five pulp blanks, comprising commercial blank BL-6, were included with the due diligence samples to monitor for potential contamination during sample preparation and assay. BL-6 is certified as a standard with very low (<0.01 ppm) precious metal content, however, as it had been analyzed extensively on another HDI project with low tungsten (0.6 ppm) and molybdenum (4.7 ppm) results, it was considered suitable for this purpose. Figure 11-10 shows the copper, molybdenum, and tungsten results. In Northcliff's opinion, no contamination is apparent based on these results, and the QP concurs.

Figure 11-10: Copper, molybdenum and tungsten results of blank sample BL-6 for due diligence

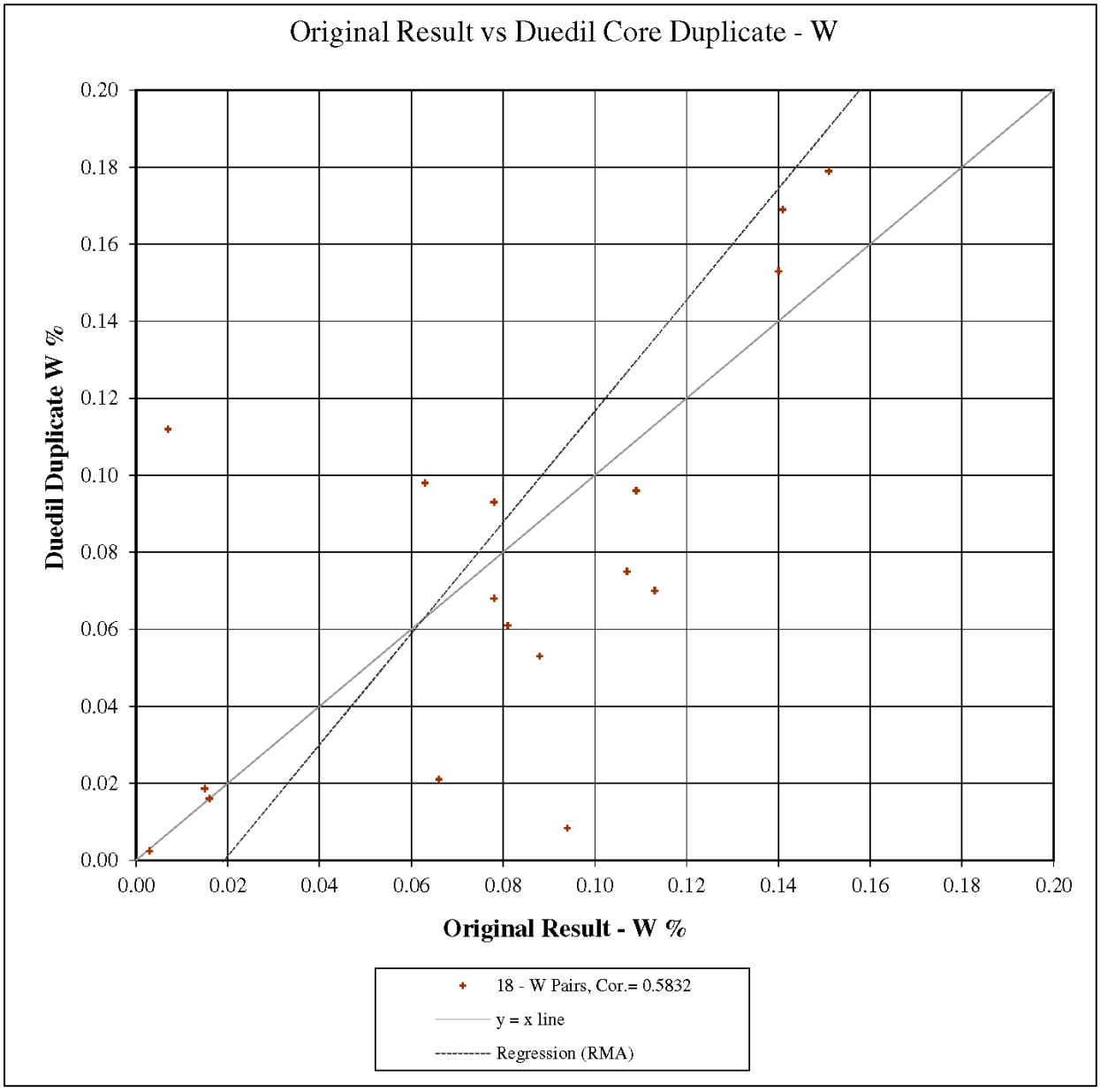


Source: RPA, 2012

11.8.2.1.3 Duplicates

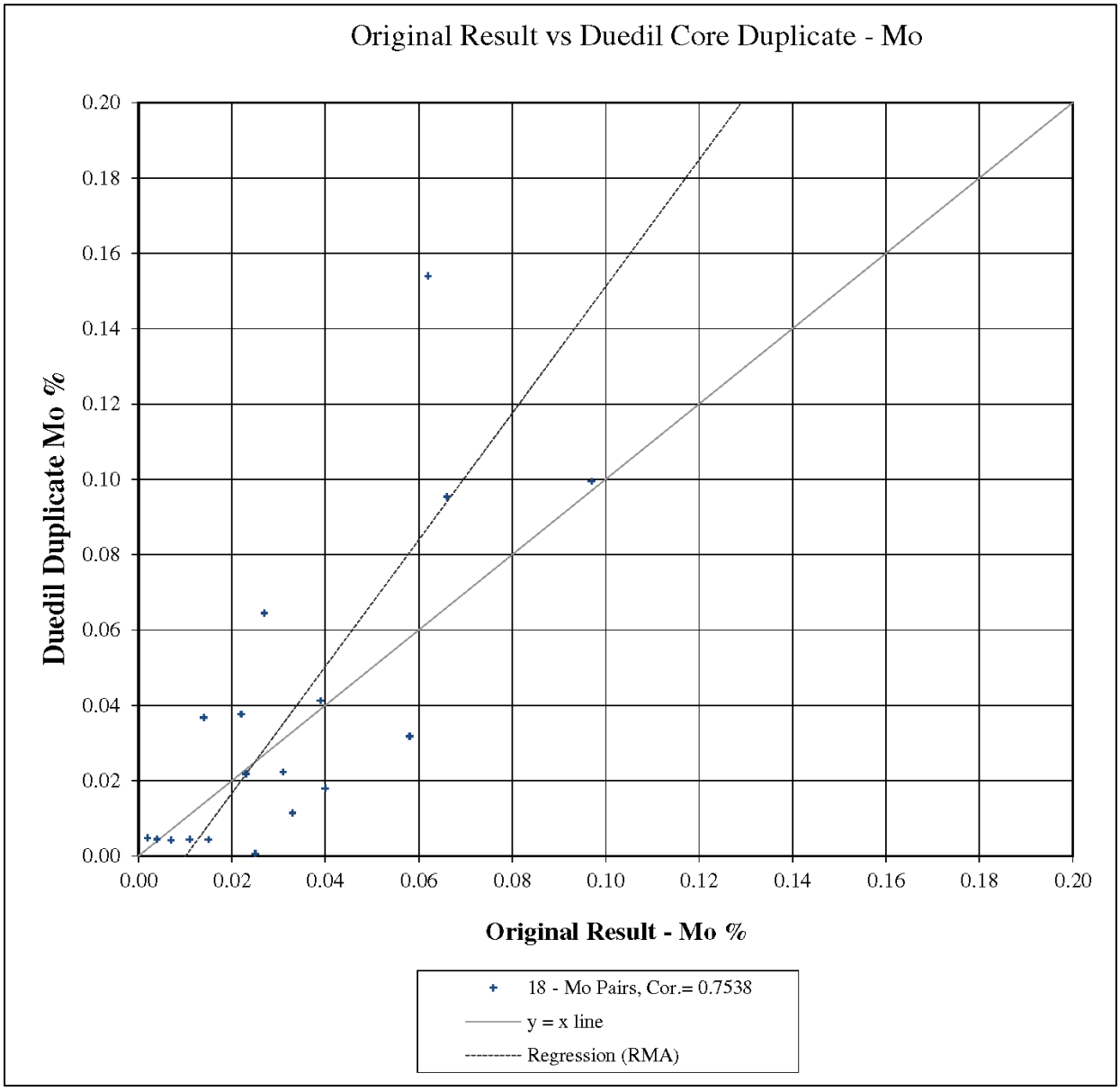
The 280 due diligence samples were compared with the original results obtained by Geodex. Figure 11-11 to Figure 11-16 are comparisons of the matched pairs duplicate results in a series of scatterplots by sample type. Generally speaking, the pulp and reject duplicates match the original results reasonably well. Although the results of the quarter core duplicates do not match the original half core sample results very well, the presence of tungsten and molybdenum mineralization is confirmed in the samples. Northcliff personnel are of the opinion that it is likely that a significant portion of the difference between these samples is attributable to the fact that it is often not possible to obtain a truly representative sample of vein-type mineralization in quarter core. In QP's opinion, this is a plausible explanation.

Figure 11-11: Tungsten scatterplot of ¼ core duplicates for due diligence



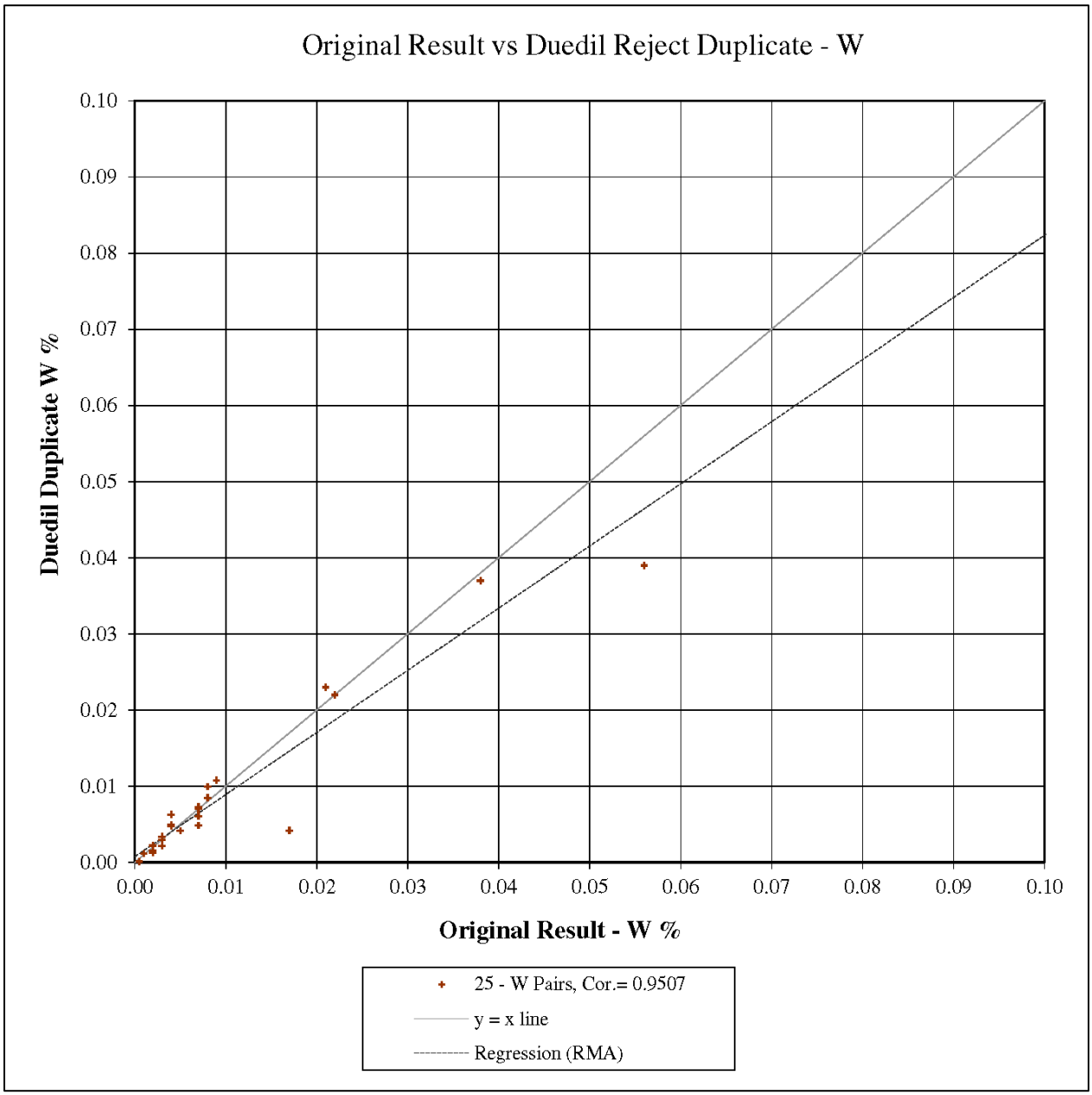
Source: RPA, 2012

Figure 11-12: Molybdenum scatterplot of ¼ core duplicates for due diligence



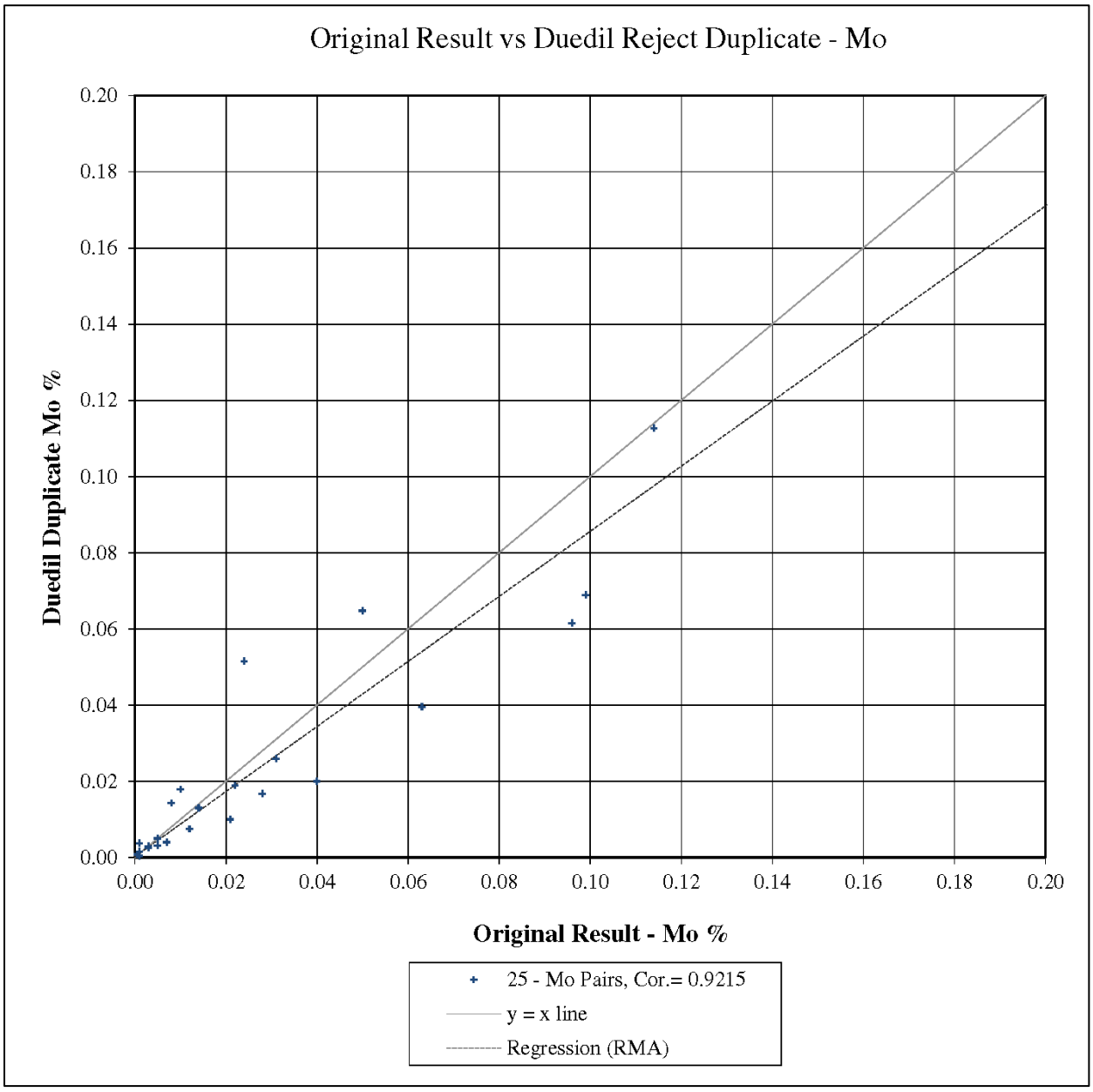
Source: RPA, 2012

Figure 11-13: Tungsten scatterplot of reject duplicates for due diligence



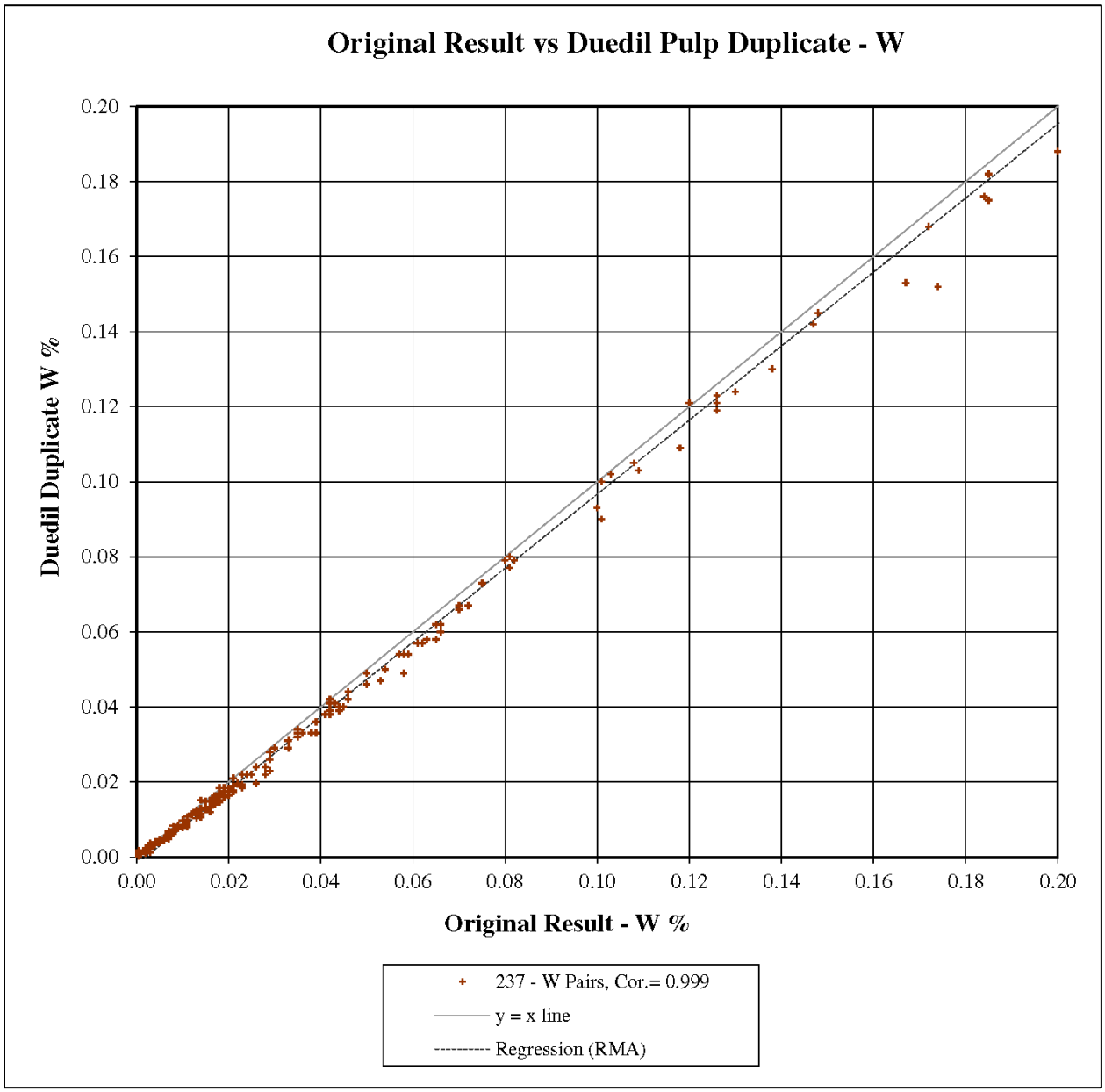
Source: RPA, 2012

Figure 11-14: Molybdenum scatterplot of reject duplicates for due diligence



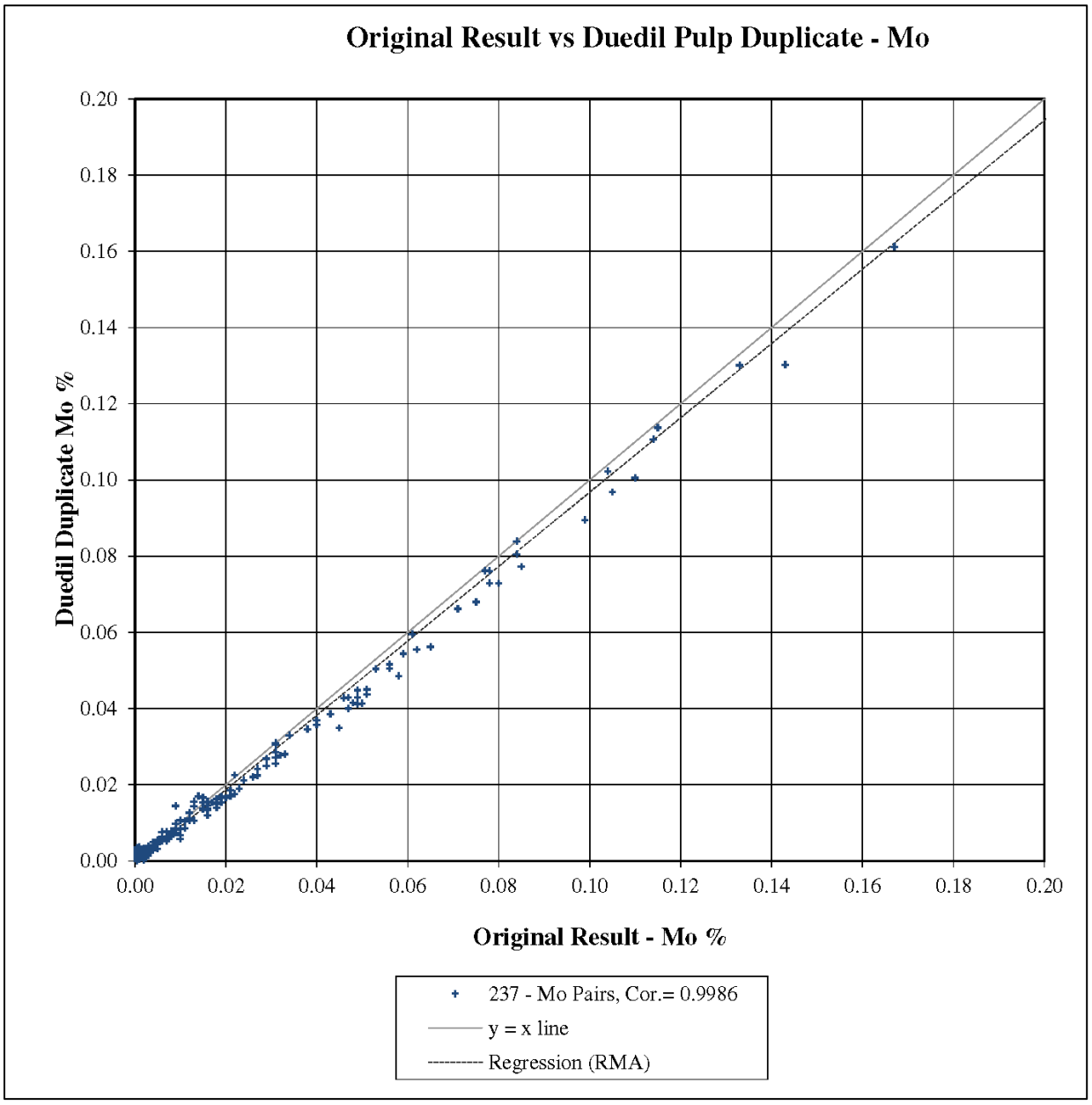
Source: RPA, 2012

Figure 11-15: Tungsten scatterplot of pulp duplicates for due diligence



Source: RPA, 2012

Figure 11-16: Molybdenum scatterplot of pulp duplicates for due diligence



Source: RPA, 2012

11.8.3 Northcliff drilling (2010)

The commercial standards used by Geodex were reportedly the best available at the time. However, the tungsten and molybdenum grades of these standards are three to ten times higher than the median values for these elements in the area of interest. In addition, the relative standard deviation (a measurement of precision) for these standards is typically between 5% and 10% (see discussion in Section 10 on 2011 Northcliff drilling). In Northcliff's opinion, these standards were suitable for exploration, but it was recognized that they needed to be replaced for the next stage of Project development. In order to implement a better QA/QC program for the ongoing drilling and engineering programs, six new matrix-

matched certified reference materials (MMCRM) were produced. These new standards were made from the coarse rejects of 2009 drill core from the Sisson property. Material was selected based on typical lithological types and tungsten grades. The six MMCRM's were prepared and certified by CDN Resource Laboratories Ltd. The ten laboratories participating in the round robin analysis are listed below:

- Activation Laboratories Ltd. (Actlabs)
- Acme Analytical Laboratories (Acme)
- ALS Minerals (ALS)
- Genalysis Laboratory Services (Genalysis)
- OMAC Laboratories Ltd. (OMAC)
- Becquerel Laboratories Inc. (Becquerel)
- SGS Canada Inc., Toronto (SGS Toronto)
- SGS Canada Inc., Vancouver (SGS Vancouver)
- AGAT Laboratories Ltd. (AGAT)
- Inspectorate Exploration and Mining Services Ltd. (Inspectorate)

Table 11-8 is a summary of the key values for the MMCRM.

Table 11-8: Summary of matrix-matched certified reference materials

Standard	W ppm	Mo ppm	Cu ppm	Original Rock Type	Source
SBRK-1	1,196	452	223	Volcanic and Sedimentary	Sisson Drill Core Rejects
SBRK-2	670	321	274	-	-
SBRK-3	310	114	295	-	-
SBRK-4	1,201	326	134	Gabbro	-
SBRK-5	802	254	151	-	-
SBRK-6	287	80	146	-	-

In the 2010 drill program, a total of 113 external QA/QC samples were applied, including 49 standards, 52 duplicates, and 12 blanks (see Table 11-7). The standards and duplicates were inserted alternately at the frequency of one in every 20 mainstream samples. Blanks were inserted at the rate of two or three per drill hole.

11.8.3.1 Standards

Northcliff implemented the new MMCRM along with the higher-grade commercial tungsten molybdenum standard W-4 in the 2010 drill program. Table 11-9 is a summary of standards used to monitor in the 2010 program.

Table 11-9: Standard materials used in 2010 drill program

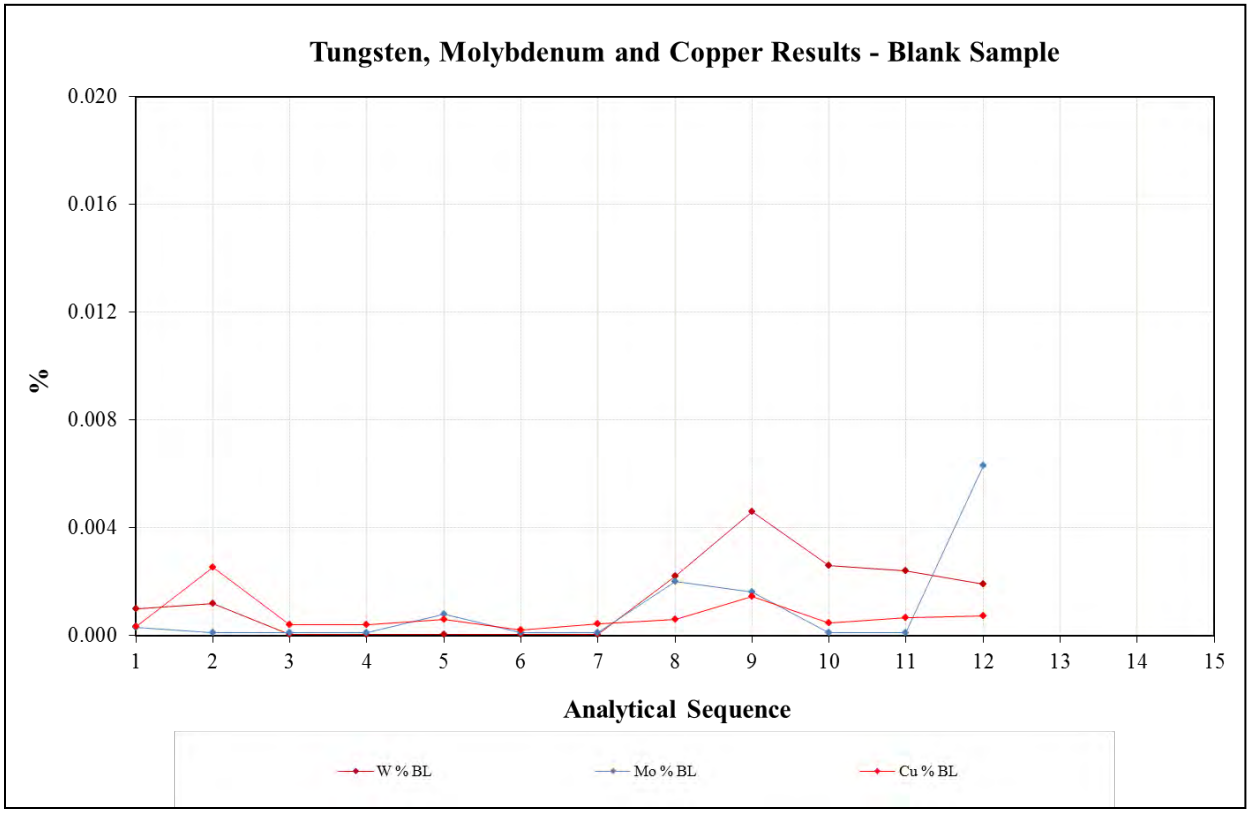
Standard	Times Used
SBRK-1	3
SBRK-2	5
SBRK-3	8
SBRK-4	2
SBRK-5	2
SBRK-6	3
W-4	26

Figure 11-21and Figure 11-22 and Figure 11-23 show the results of the standards inserted and assayed for both the 2010 and 2011 programs.

11.8.3.2 Blanks

A total of 12 barren core blanks (granite) were applied to monitor the potential contamination for the 2010 drill program. Figure 11-17 shows copper, molybdenum, and tungsten results of these blanks. No obvious contamination was indicated based on these results.

Figure 11-17: Tungsten, molybdenum and copper results of blank samples for 2010 drill program

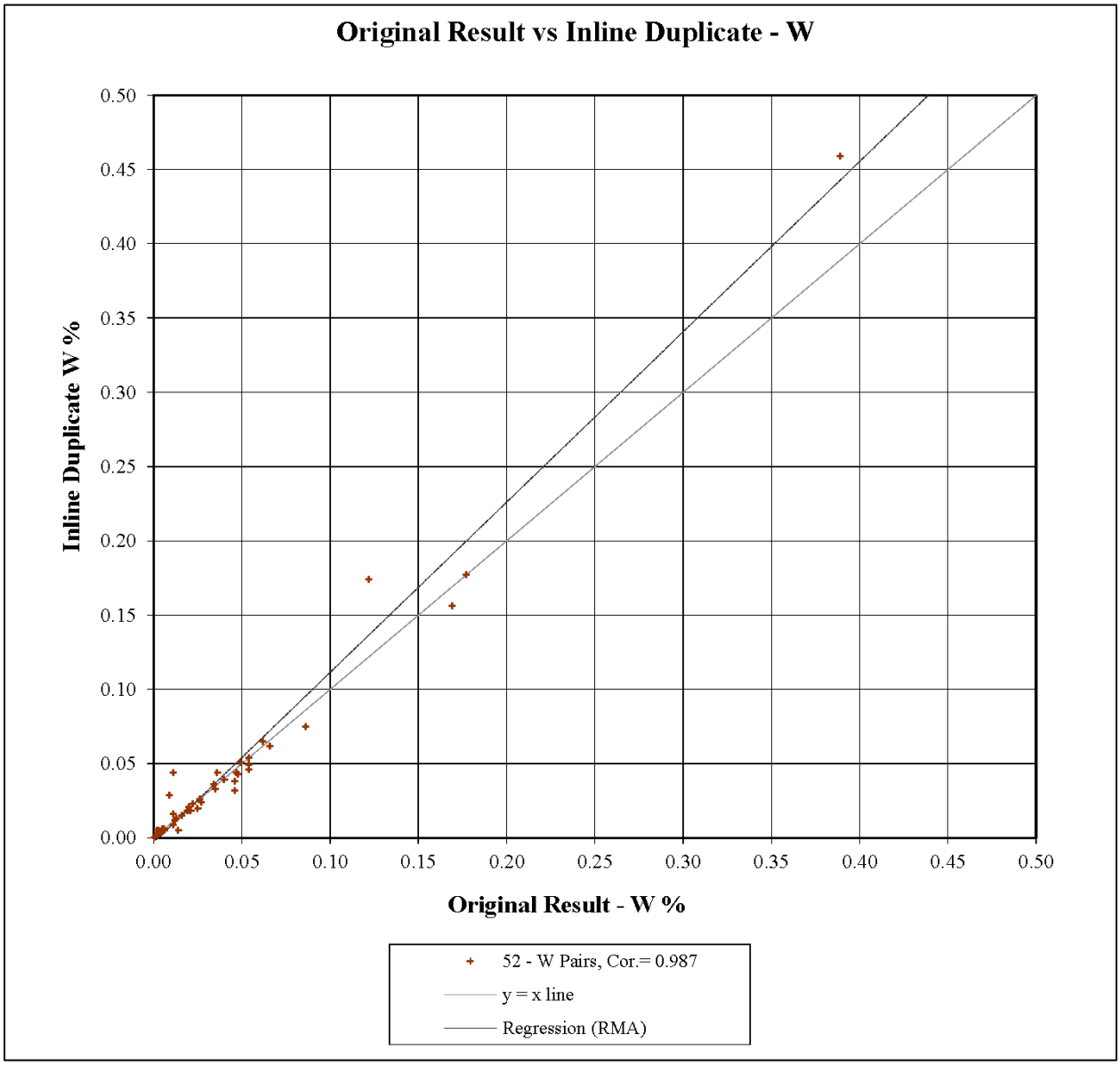


Source: RPA, 2012

11.8.3.3 Duplicates

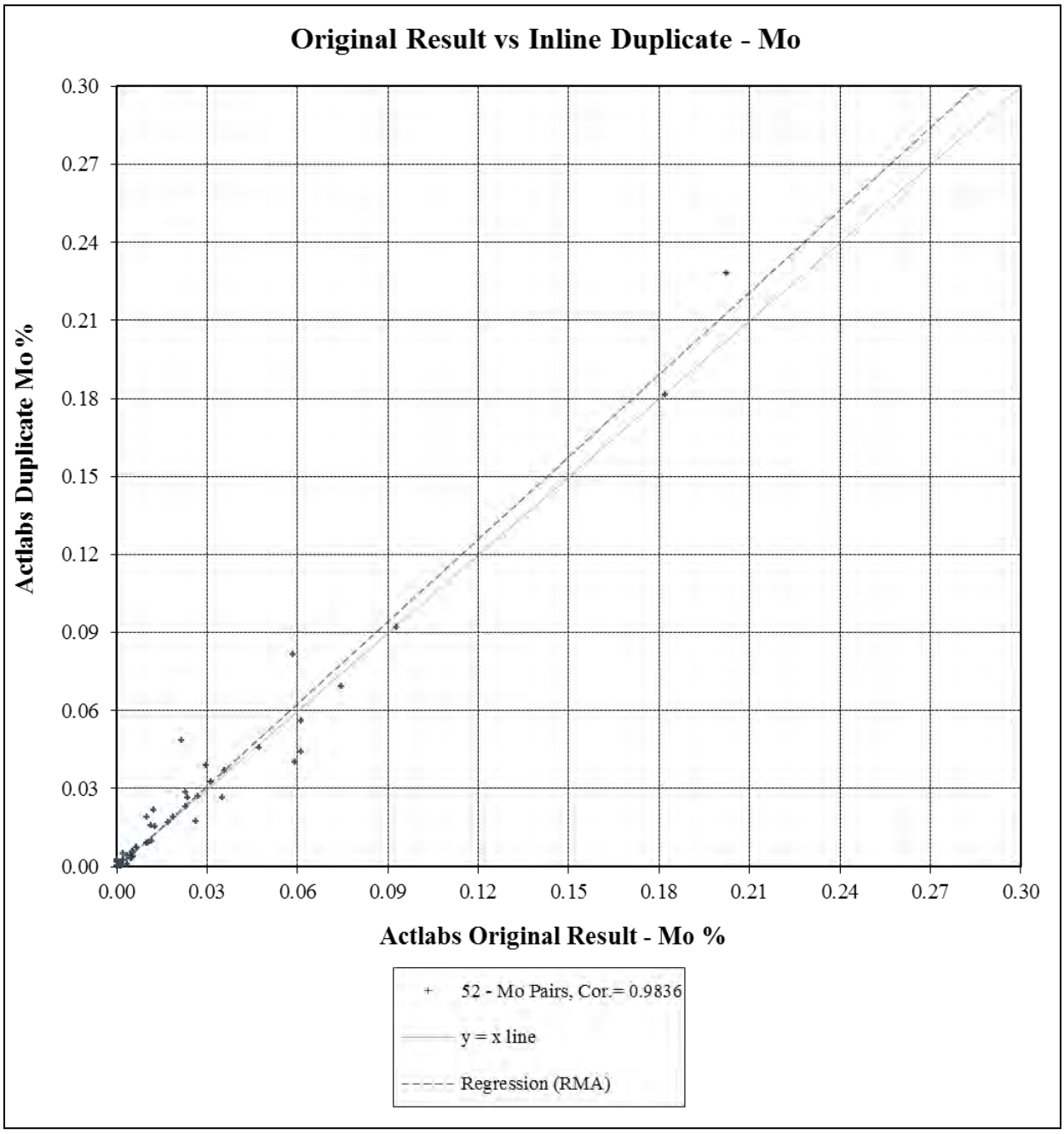
A total of 52 duplicates were applied to monitor the reproducibility of the primary assay laboratory (in-line duplicates) as well as the second check laboratory (inter-laboratory duplicates). In Northcliff's nomenclature, an in-line duplicate is a second split from the reject material, while the inter-laboratory duplicate is a split from the pulp. Figure 11-18 to Figure 11-21 are scatterplots of the duplicate pair results.

Figure 11-18: Tungsten scatterplot of in-line duplicates for 2010 drill program



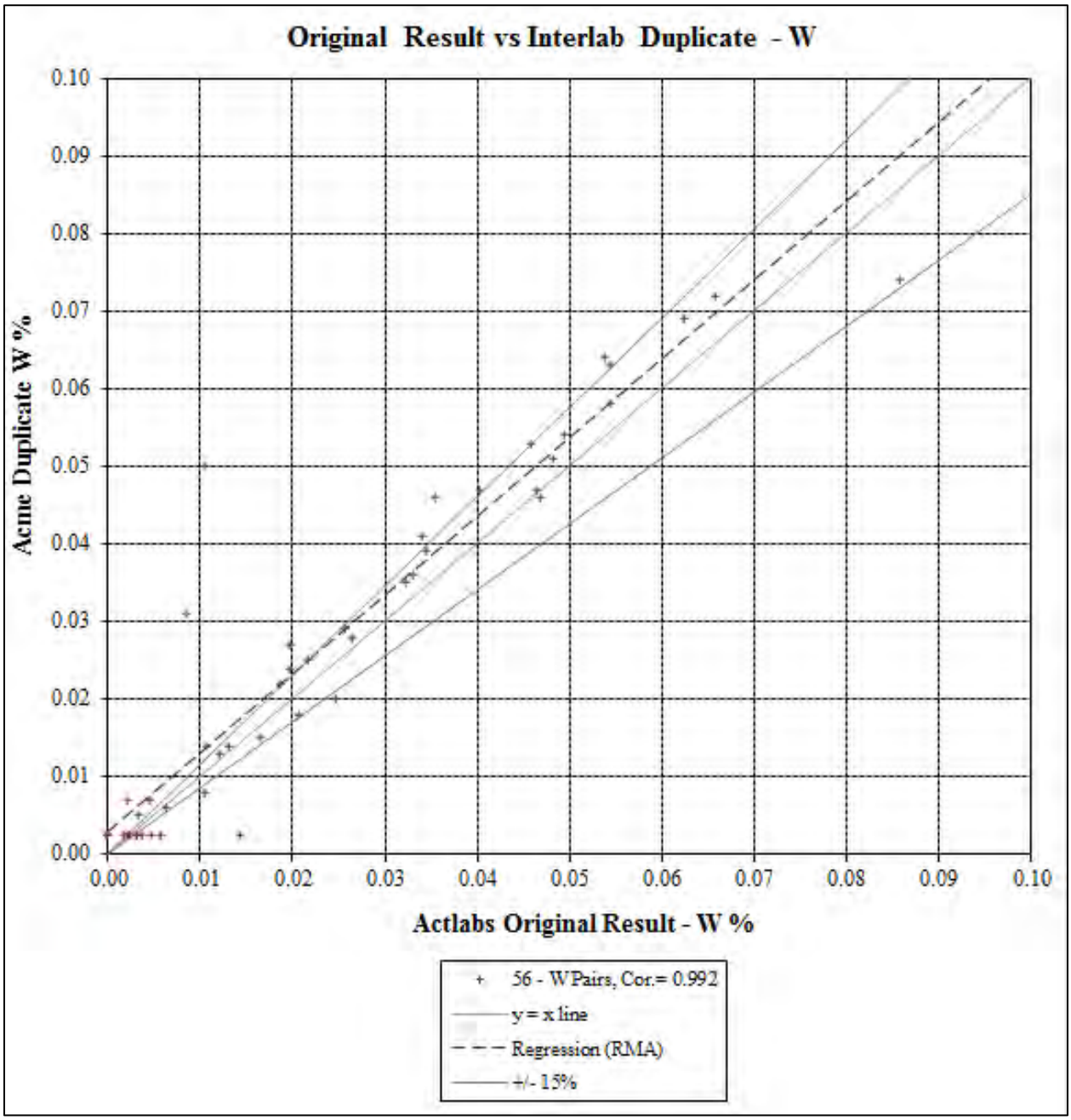
Source: RPA, 2012

Figure 11-19: Molybdenum scatterplot of in-line duplicates for 2010 drill program



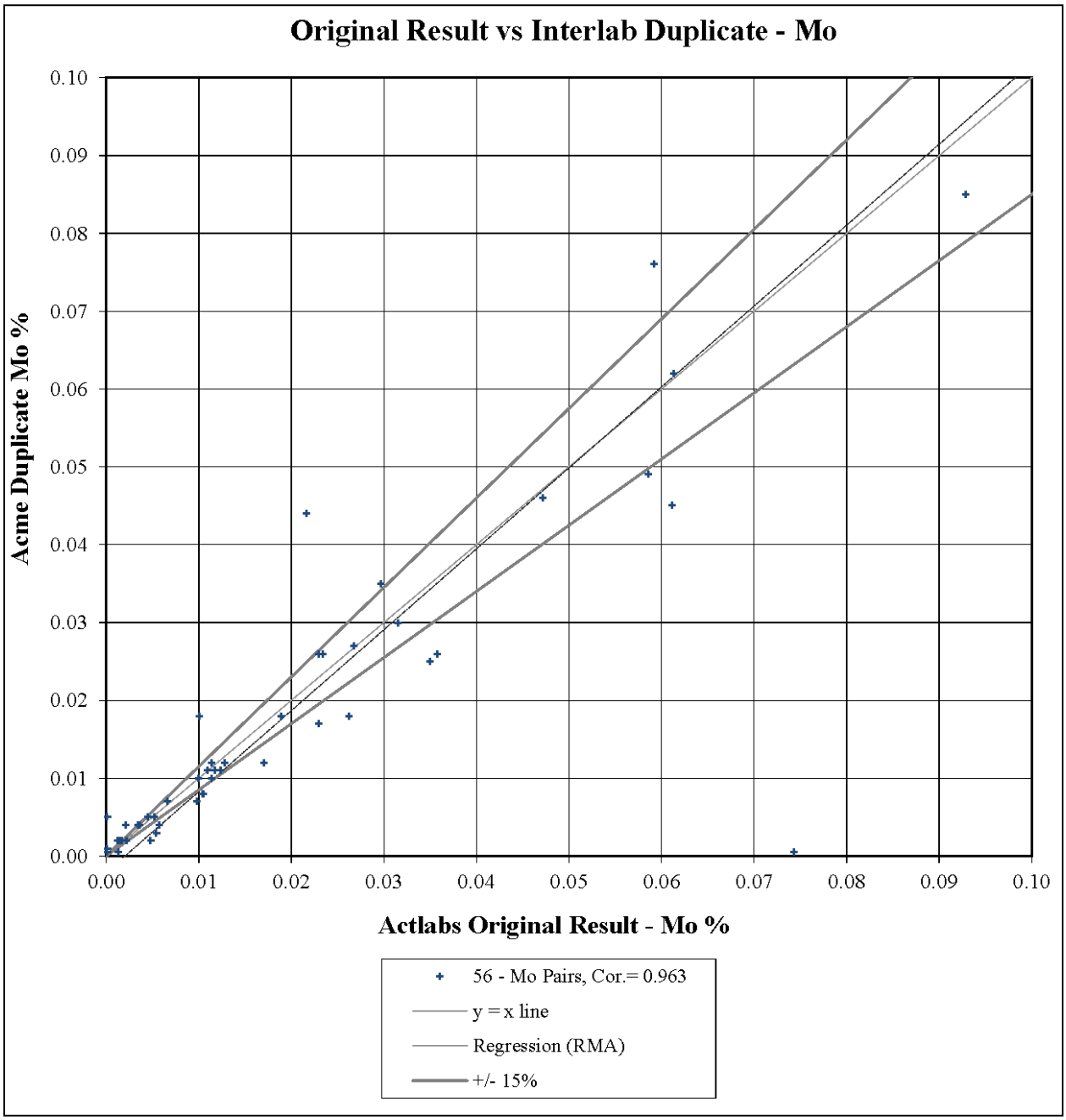
Source: RPA, 2012

Figure 11-20: Tungsten scatterplot of inter-lab duplicates for 2010 drill program



Source: RPA, 2012

Figure 11-21: Molybdenum scatterplot of inter-lab duplicates for 2010 drill program



Source: RPA, 2012

11.8.4 Northcliff drilling (2011)

In the 2011 drill program, a total of 337 external QA/QC samples were applied, including 149 standards, 144 duplicates, and 44 blanks (Table 11-10). The standards and duplicates were inserted alternately with the frequency of one in every 20 mainstream samples. Blanks were inserted at the rate of two or three for each drill hole.

Table 11-10: QA/QC summary for 2010-2011

Year	MS	DP	ST	BL	ST%
2010	929	52	49	12	5
2011	2,768	144	149	44	5
ALL	**3,697**	**196**	**198**	**56**	**5**

11.8.4.1 Standards

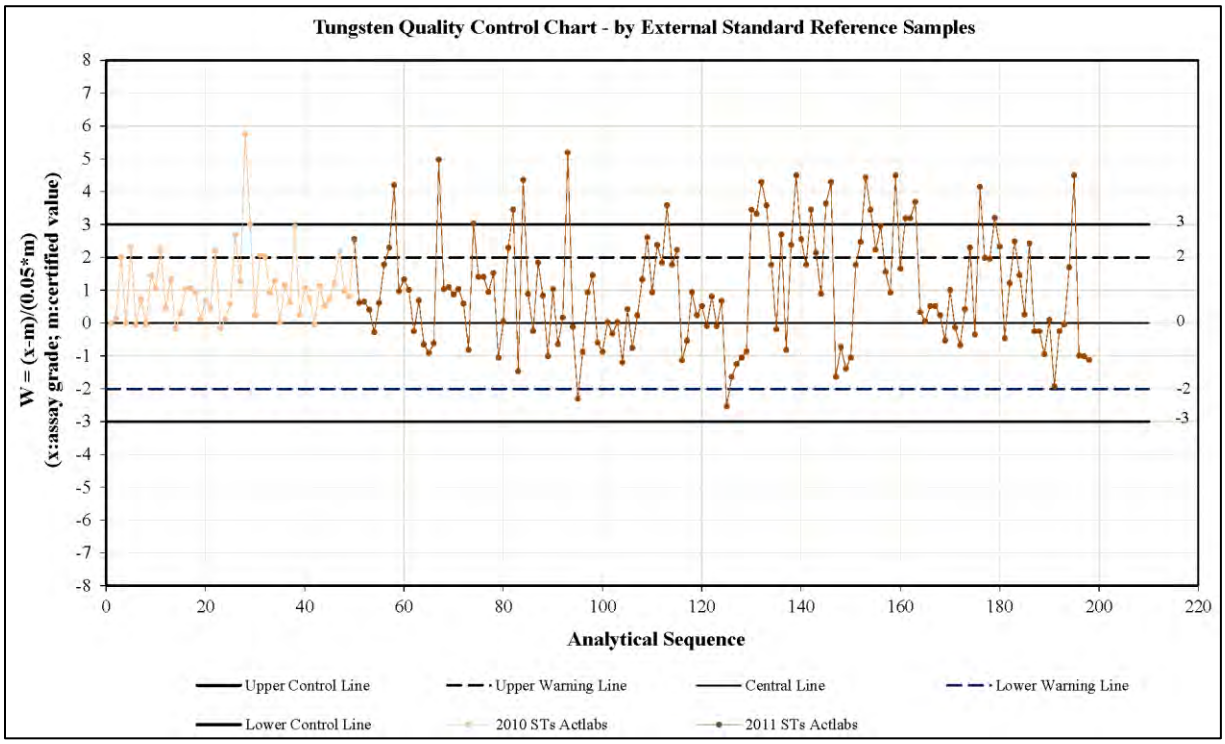
Table 11-11 is the summary of standards used to monitor the tungsten and molybdenum assays in the 2011 drill program.

Table 11-11: Standard materials used in 2011 drill program

Standard	Times Used
SBRK-1	8
SBRK-2	21
SBRK-3	49
SBRK-4	4
SBRK-5	14
SBRK-6	45
W-4	8
Overall	149

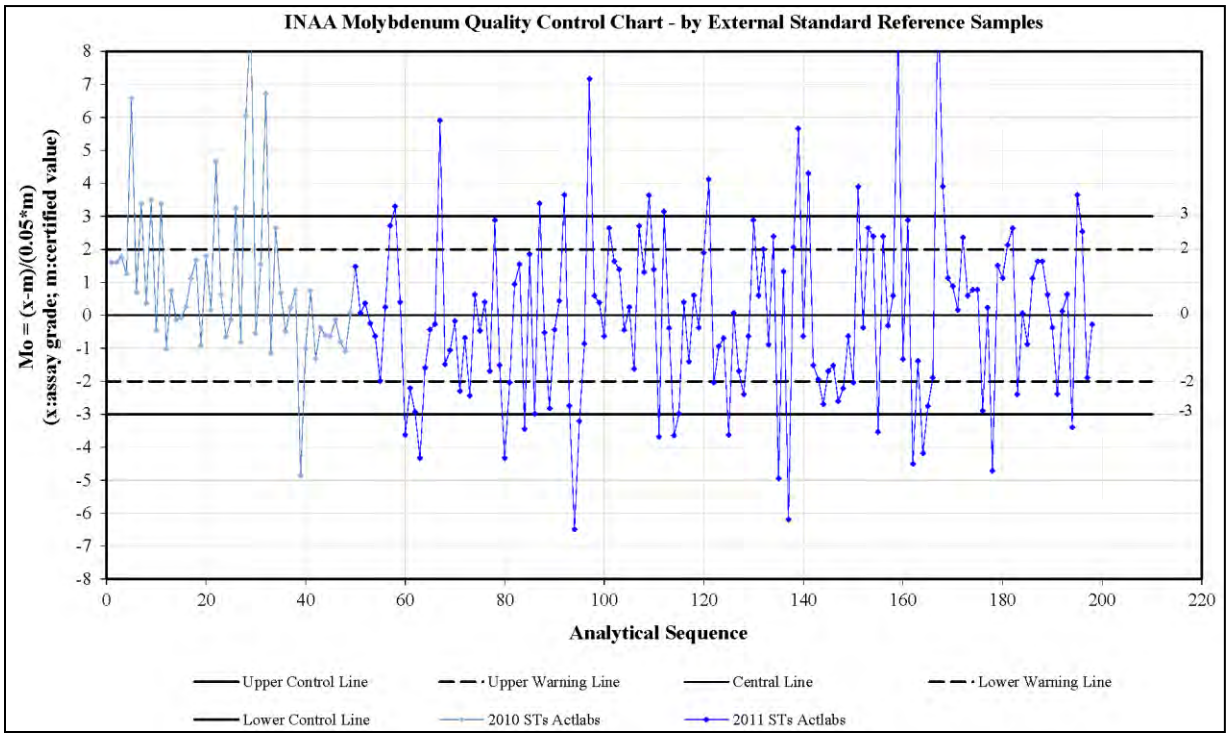
Figure 11-22 and Figure 11-23 show the results of the standards inserted and assayed along with the regular samples.

Figure 11-22: Tungsten control chart for 2010 and 2011 drill program



Source: RPA, 2012

Figure 11-23: Molybdenum control chart for 2010 and 2011 drill program



Source: RPA, 2012

The warning and failure limits for the control diagrams are based on the relative standard deviation (RSD), which is the coefficient of variation (i.e., standard deviation divided by the mean) multiplied by 100%. The RSD values for the standards, as derived from the round robin assays, are shown in Figure 11-23 and Figure 11-24. The tolerance limits (termed Recommended Control Rules by Northcliff personnel) applied to the standards assays are listed in Table 11-12.

Table 11-12: Control limits applied

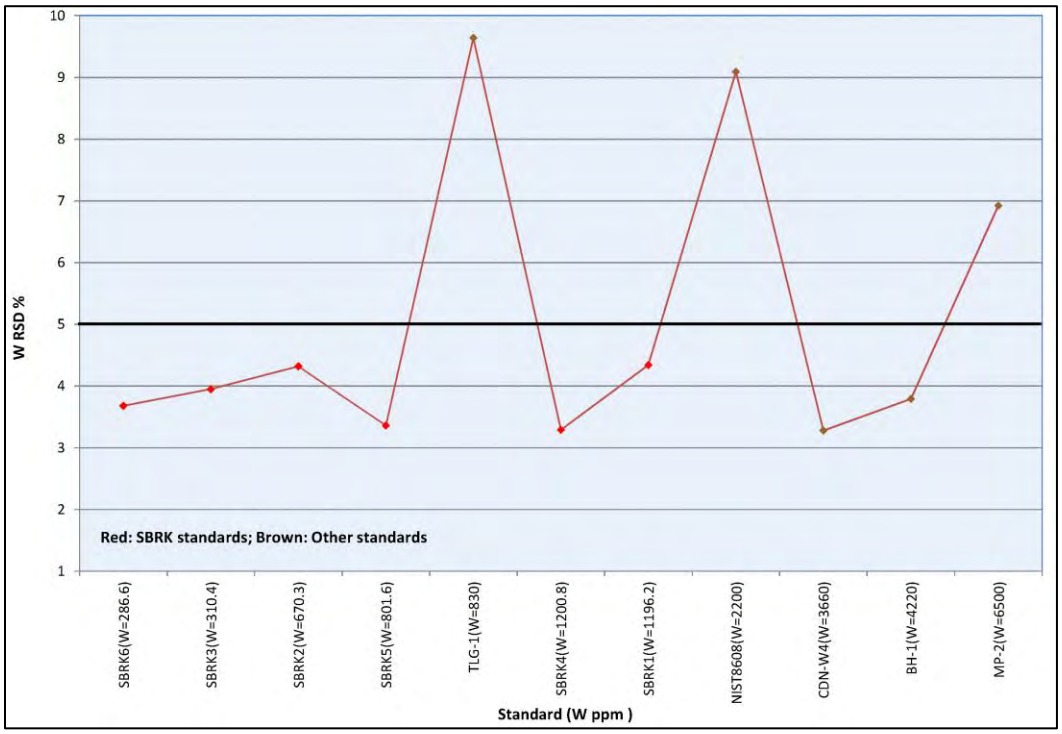
Element (commodity)	Standardized RSD	Warning limits	Control limits
Au (Pt, Pd)	5%	Mean*(1 ± 10%)	Mean*(1 ± 15%)
Mo, W	5%	Mean*(1 ± 10%)	Mean*(1 ± 15%)
Ag, Pb, Nb	4%	Mean*(1 ± 8%)	Mean*(1 ± 12%)
Cu, Zn	3%	Mean*(1 ± 6%)	Mean*(1 ± 9%)

Northcliff personnel found that the MMCRM used in the 2010 and 2011 drilling programs on the Project had generally smaller RSDs than the commercial standards, which led to a higher failure rate than usual. In Figure 11-24, RSD values for the commercial standards TLG-1, NIST8608, and MP-2 are all significantly higher than for Northcliff's internal standards. This indicates that the precision with which the assay values for these standards are known is less than for the in-house standards. The net effect was that when the conventional ±2 and ±3 SD error limits were applied, an excessive number of failures were generated for the MMCRM. This resulted in a larger number of re-assays, significant time lost in resolving apparent issues, and increased costs without an attributable improvement in the overall assay database. In response, Northcliff adopted the control limits outlined in Table 11-12 for gauging assay performance.

In the QP's opinion, the approach taken by Northcliff in developing and using the in-house standards is reasonable. The grade ranges for the samples at Sisson, including the MMCRM, are relatively low, which will result in higher variance in assay results. This tendency for assay precision to deteriorate at lower grades is quite common and well documented.

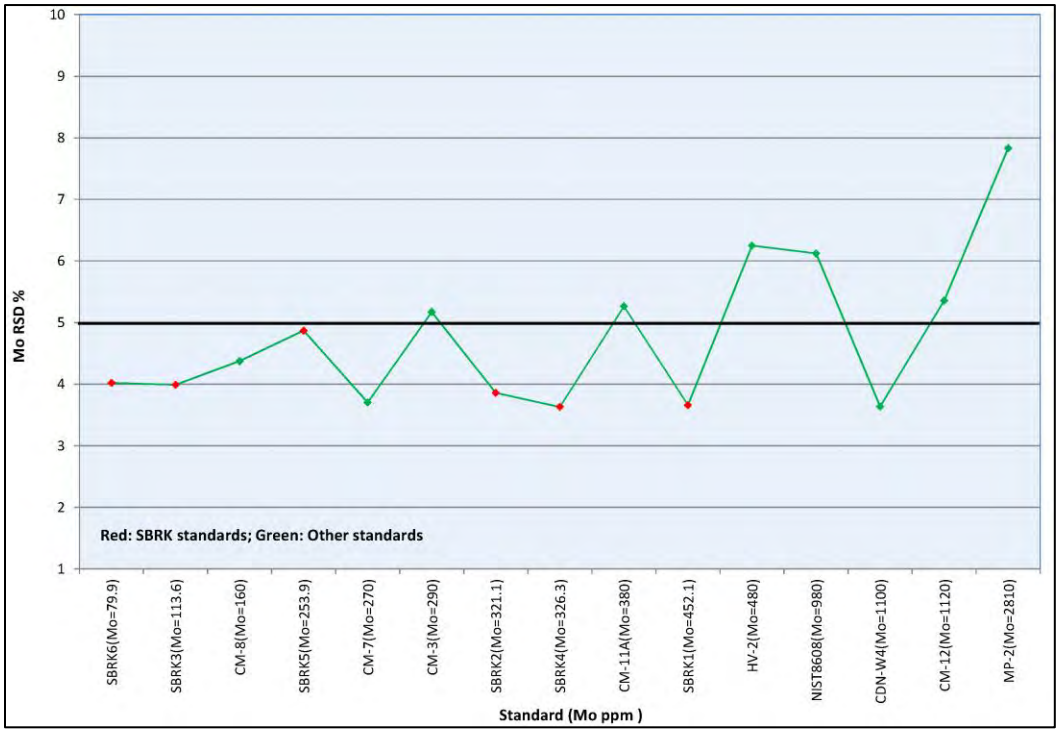
The QP notes that there is some evidence of a slight positive bias in the results for tungsten over certain periods of time, which should be investigated further. The molybdenum results demonstrate a fairly high degree of scatter, but there does not appear to be any significant bias.

Figure 11-24: Tungsten RSD (%) of certified standards



Source: RPA, 2012

Figure 11-25: Molybdenum RSD (%) of certified standards



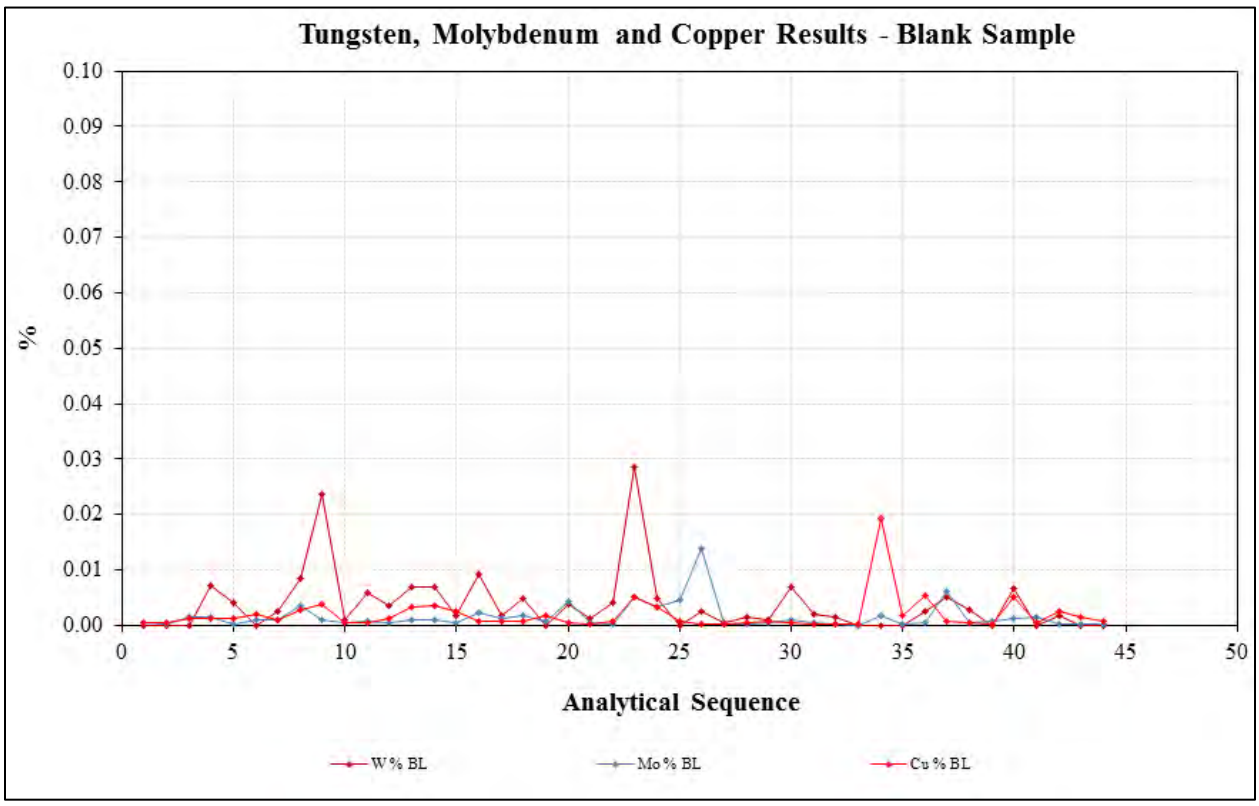
Source: RPA, 2012

11.8.4.2 Blanks

A total of 44 barren course (granite) blanks were applied to monitor the potential contamination for the 2011 drill program. Figure 11-25Figure 11-25 shows tungsten, molybdenum, and copper results of these blanks. No significant contamination was found based on these results.

In the QP's opinion, the blanks results show an acceptable failure rate.

Figure 11-26: Tungsten, molybdenum and copper results of blank samples for 2011 drilling program



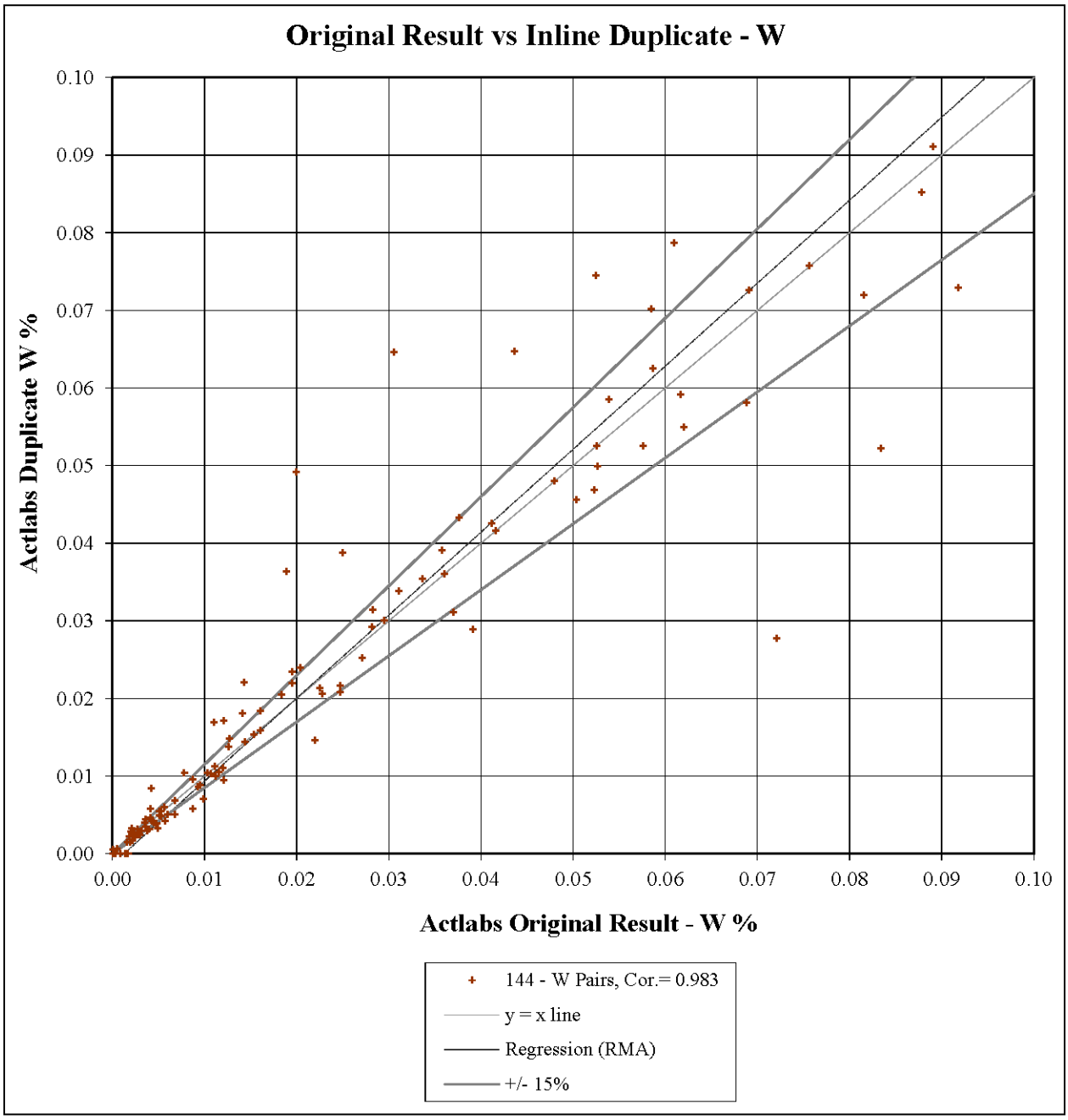
Source: RPA, 2012

11.8.4.3 Duplicates

A total of 144 duplicates were applied to monitor the repeatability of the primary assay laboratory (in-line duplicates) as well as the reproducibility of the second check laboratory (inter-laboratory duplicates). Figure 11-27 to Figure 11-30 are scatterplots of the related pairs.

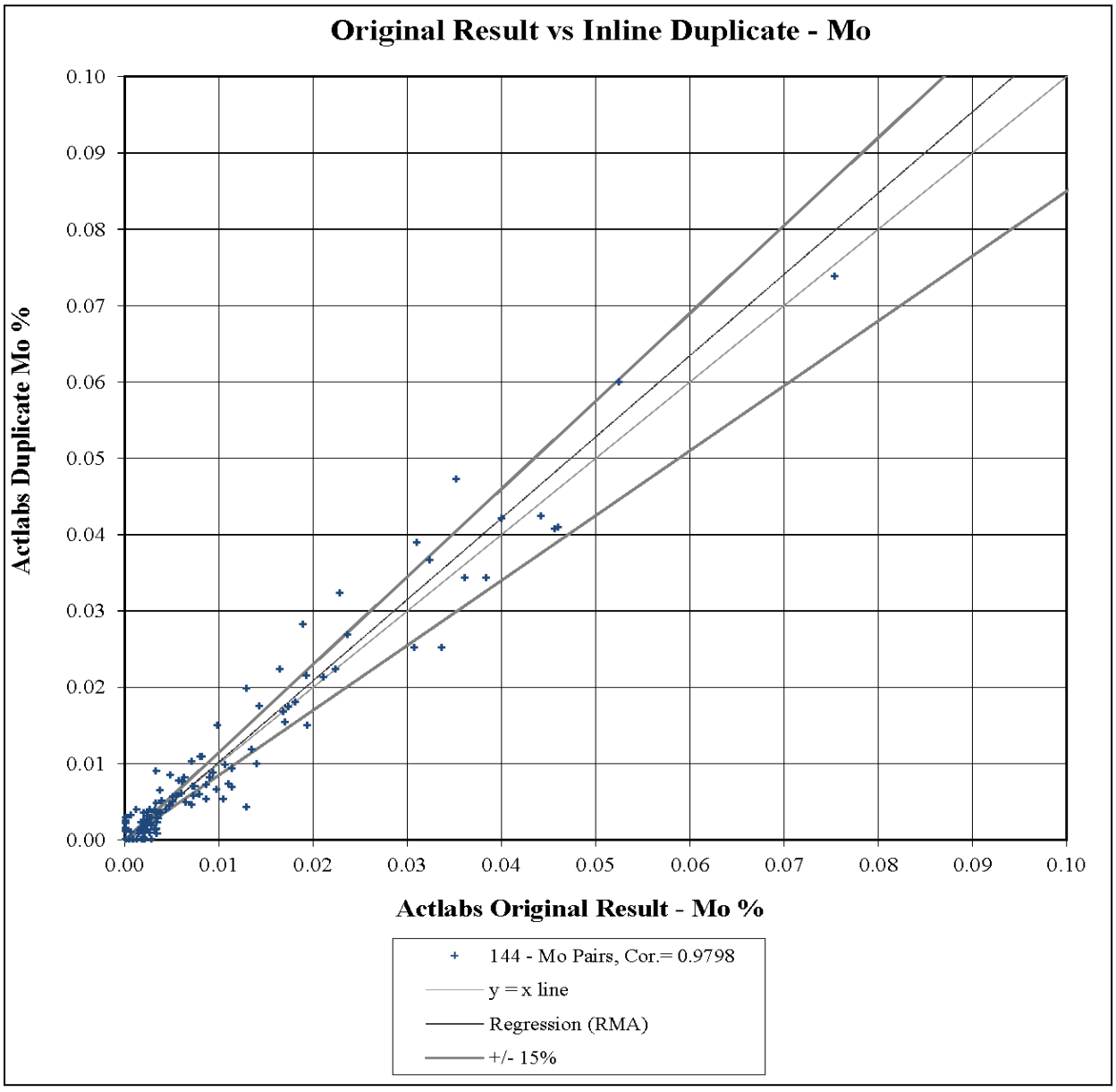
In the QP's opinion, the duplicate results demonstrate a somewhat increased level of scatter which is perhaps related to the low-grade issue. The in-line duplicates do not appear to be biased. The inter-laboratory duplicates show some evidence of bias, which, in SLR's opinion, is probably due to the use of different assay techniques.

Figure 11-27: Tungsten scatterplot of in-line duplicates for 2011 drill program



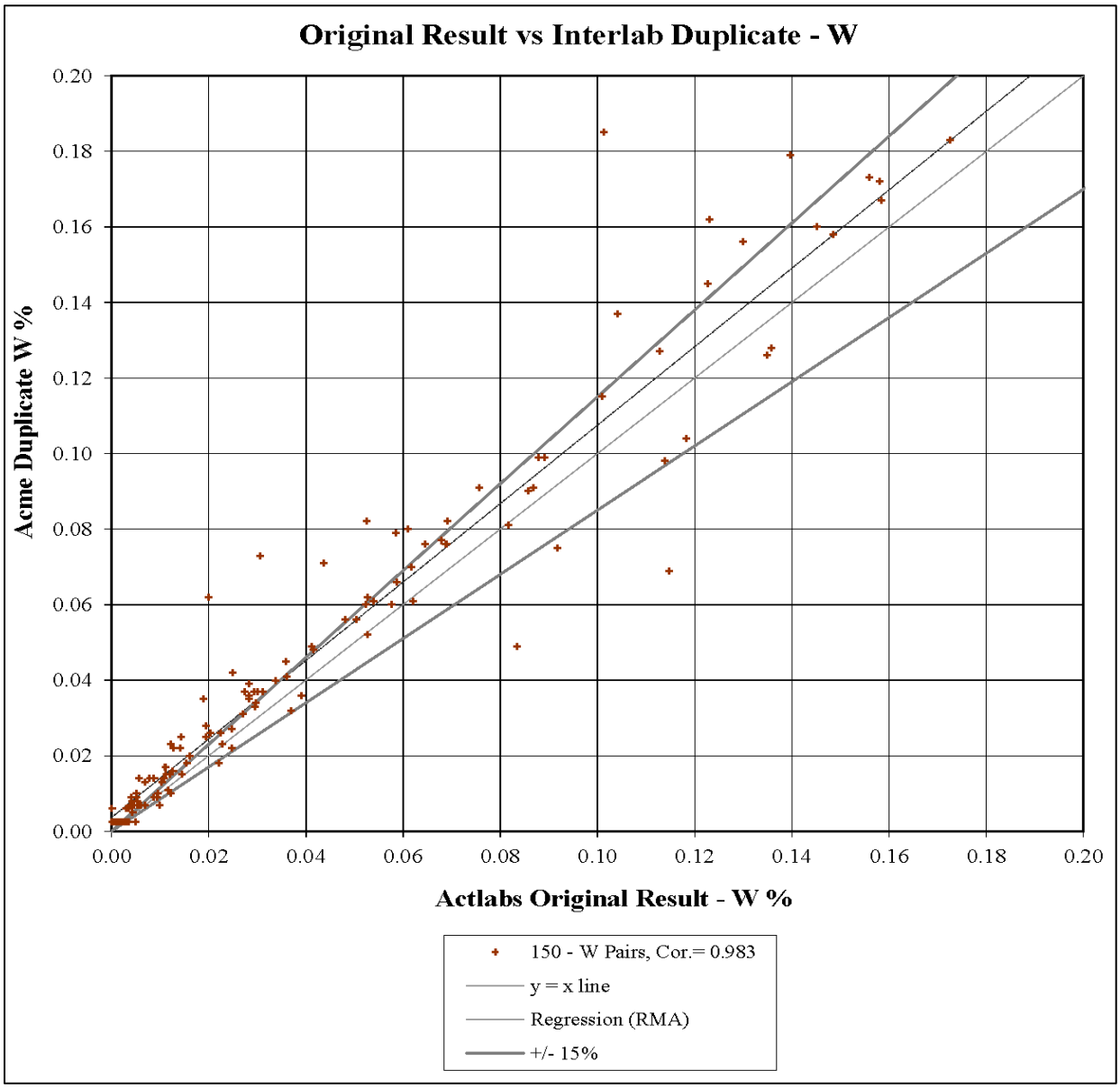
Source: RPA, 2012

Figure 11-28: Molybdenum scatterplot of in-line duplicates for 2011 drill program



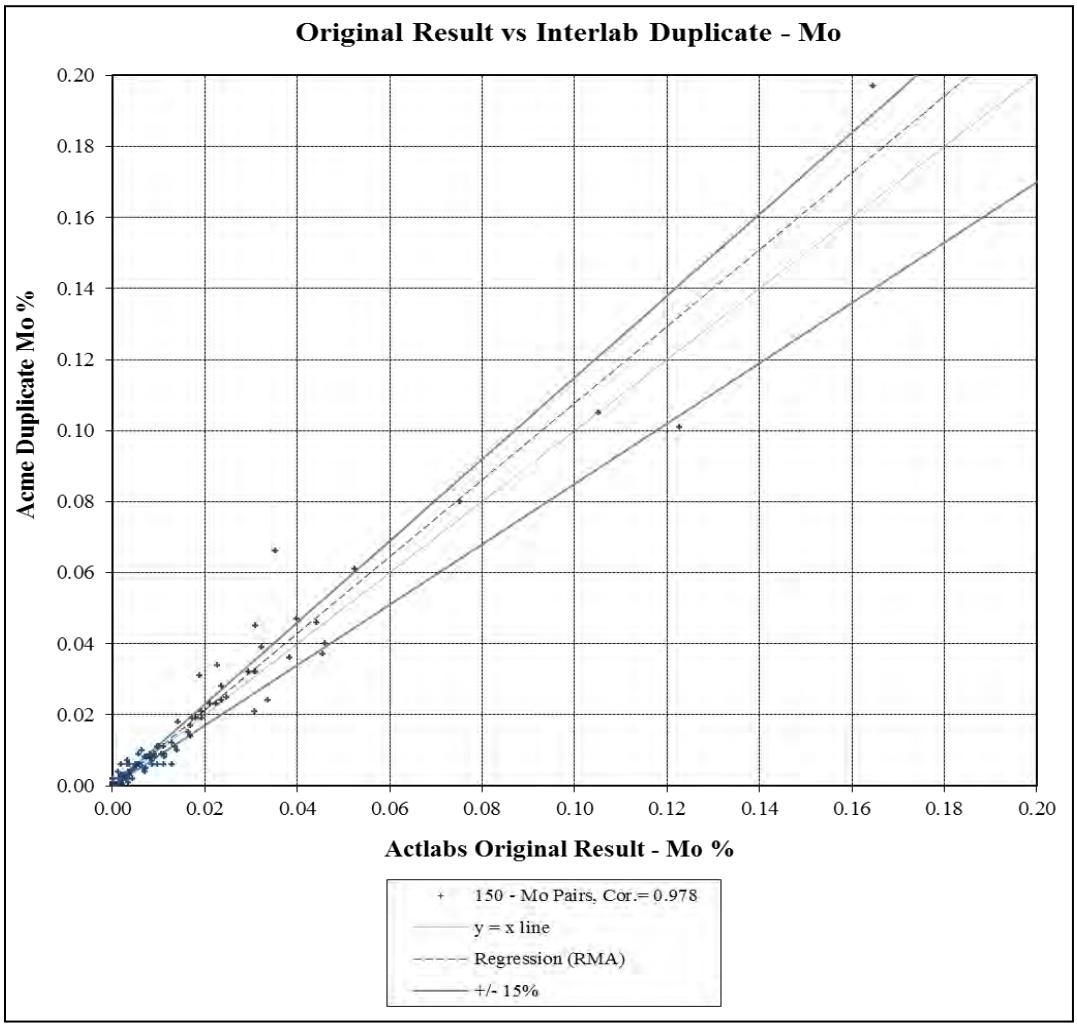
Source: RPA, 2012

Figure 11-29: Tungsten scatterplot of inter-lab duplicates for 2011 drill program



Source: RPA, 2012

Figure 11-30: Molybdenum scatterplot of inter-lab duplicates for 2011 drill program



Source: RPA, 2012

11.9 Discussion and conclusions

In the QP's opinion, the sampling and core handling have been carried out in an appropriate manner consistent with industry standard practice. The assaying has been performed at independent commercial accredited laboratories, using conventional approved methodologies. The holes have been drilled and samples taken such that they will be representative of the deposit mineralization.

The QPs note that the low-grade ranges encountered for the Project have prompted the selection of INAA as the primary analytical technique. This is not as common a technique as more typical assay methods such as ICP or AAS and is not generally offered by commercial laboratories. At present, Actlabs is one of the few in Canada that offers this service. Consequently, pulp duplicate comparisons with other laboratories, using different protocols, are likely to result in apparent biases. This could create some difficulties in evaluation of assay QA/QC results, but it is not anticipated to adversely affect the MRE.

In the QP's opinion, the sample preparation, analysis, QA/QC, and security procedures at the Project are adequate for use in the estimation of Mineral Resources.

12 Data verification

12.1 Data verification performed by Paul Chamois (SLR)

12.1.1 Verification of legacy data

For the purposes of this Technical Report, the QPs have relied on information provided by Northcliff, public sources, the technical report completed by Samuel Engineering Inc. (2013), and from other documents.

In accordance with NI 43-101 guidelines, a site visit to the Property was carried out by Paul Chamois, M.Sc.(A), P. Geo., SLR Associate Principal Geologist, on June 9, 2022. During the visit, Mr. Chamois examined drill core from historical drilling programs, confirmed the local geological setting, assessed logistical aspects related to access and conducting work in the area, and investigated factors that might affect the Project.

During the site visit, no exploration activities were ongoing on the Property.

SLR was given full access to project data, and no limitations were placed on Mr. Chamois.

Prior to the site visit, discussions were held with Louise Steward, Northcliff VP Regulatory and Government Affairs, and Gregory Davidson, Northcliff Manager of Aboriginal and Community Relations. During the site visit, Mr. Chamois was accompanied by Mr. Davidson.

SLR has relied on property title information provided by Northcliff.

12.1.2 Database verification

SLR compiled all available assay certificates, which were provided as 315 separate Microsoft (MS) Excel files from Acme, Actlabs, and ALS. These assays relate to data collected between 1979 and 2013. The certificate molybdenum and tungsten values were compared with the database molybdenum and tungsten values by matching on SAMPLEID. Only molybdenum and tungsten assays were analyzed during the data verification stage. Assay data in the provided ALS and Acme certificates relate to check assays; this data was not considered during the database validation stage.

The assay database supplied to SLR in 'SissonDrillHoleDatabase_20250818.xlsx' contained 32,918 samples. SLR compared a total of 29,179 samples, representing 89% of the assay database. Table 12-1 below contains a data verification summary.

Table 12-1: Data verification summary

Samples	Quantity
Existing in assay database	32,918
Compiled from certificates	29,915
Matched and compared	29,179 (89% of assay database)

Some certificates contain multiple molybdenum and tungsten columns denoting various detection limits. In these cases, assay values relating to the analysis method with the lowest detection limit were included. For analysis purposes, SLR converted all assays to ppm. Assays that were reported as being below the limit of detection were changed to half the detection limit. It is also noted some certificates did not assay for Mo or W and were excluded from analysis.

Many Actlabs certificates had multiple versions of the same certificate (e.g., A06-2153final.xls, A06-2153final1.xls, A06-2153final3.xls, A06-2153final4.xls, A06-2153final5.xls, A06-2153final6.xls) which were supplied to SLR. The rationale used to determine which assay values were included in the database is unclear and, at times, involved molybdenum and tungsten results for the same SAMPLEID sourced from different certificate versions.

Only one discrepancy was detected during the cross-checking of drill hole samples (Table 12-2); for sample 922170 related to drill hole SB-11-012M, there were inconsistencies between the original certificate 'A11-7340Final.xls' and the database. Two versions of the certificate were identified, with neither matching the assay values recorded in the database. The SLR QP recommends investigating this remaining discrepancy and correcting it accordingly in the assay database.

The SLR QP is of the opinion that the database is very well maintained, the database verification procedures for Sisson comply with industry standards, and the data are of sufficient quality to support the disclosure of Mineral Resources.

Table 12-2: Year by year data verification summary

Year	No. Samples	No. Samples Compared	No. Discrepancies	
			Mo	W
1979	536	220	-	-
1980	1,045	666	-	-
1981	607	-		
1982	584	-		
2005	568	516	-	-
2006	4,910	4,292	-	-
2007	12,026	11,228	-	-
2008	5,258	5,174	-	-
2009	3,018	2,988	-	-
2010	934	929	-	-
2011	2,836	2,769	1	1
2013	429	397	-	-
2014	167	-	-	-
Total	**32,918**	**29,179**	**1**	**1**

12.2 Data verification performed by Robert Gray (MMTS)

The resource model data was loaded into the mine planning software and verified by the QP through the independent recreation of the resource tonnages and grades. Topography data for the open pit and stockpile areas was verified during the QP's site visit in September 2025.

The application of geotechnical data used to inform the mine designs was independently verified through collaborative design reviews between the QP and KP engineers, who are responsible for the existing geotechnical design criteria. The application of all other mine design criteria, as well as the Mineral Reserve Estimate tonnages and grades, were verified through independent, in-house engineering peer reviews

conducted under the supervision of the QP. Furthermore, the mine design data incorporated into the general arrangement drawings was verified by the QP to ensure it remains current for this Technical Report.

The QP verified that the Mineral Reserves, mine production schedule, and estimated mining capital and operating costs are accurately represented in the project financial model. Mine capital and operating cost input data, sourced from 2025 and 2026 vendor quotations, were verified for reasonableness by the QP against competitive quotations received for similar open-pit operations in North America. Additionally, input data regarding mine operation system productivities, consumable usage, and labor requirements were verified by the QP as reasonable when compared to rates measured at comparable North American open-pit mining operations.

The QP considers the data used for mine engineering and costing to be adequate for the purposes of this Technical Report.

12.3 Data verification performed by Kevin Murray (Ausenco)

The QP reviewed the available metallurgical test work supporting the recovery assumptions and process design criteria. The metallurgical data-verification review included:

- examination of metallurgical test work reports prepared by SGS and others for test work programs completed in 2009 through to 2025
- examination of metallurgical test work results for tests completed by SGS in 2025 and 2026 but not yet assembled in final report format
- review of sample locations and lithological characteristics
- review of sample preparation, compositing, and metallurgical test protocols
- review of metallurgical balances, calculated recoveries, concentrate grades, reagent consumptions, and grind-size measurements
- discussions with representatives of Northcliff, SGS Canada, and Ausenco concerning sample origin, test conditions, metallurgical observations and the interpretation of the testwork.

The QP reviewed the spatial distribution of metallurgical samples relative to the current mineral-resource model. The samples cover the principal mineralized domains and a range of head grades expected within the proposed processing inventory.

In the opinion of the QP, the metallurgical data have been verified and are suitable for establishing the process-design criteria and metallurgical recovery assumptions used in the FS.

12.4 Data verification performed by Daniel Friedman (Knight Piésold)

The QP has verified the reasonableness and applicability of the following data sources for the purposes of the relevant sections of this technical report:

- Topographical mapping was verified during the site visit and using alternative sources of information such as publicly available imagery.
- The reasonableness of geotechnical lab testing data was verified by comparison to the expected properties based on the photos taken during the site investigations, and to the descriptions in the drillhole and test pit logs.

- The tailings properties required for design of the tailings pipelines were provided by Ausenco, and the reasonableness of the data was confirmed through discussions between KP and Ausenco and by comparison to information from other projects.
- Environmental data developed by others, including climate, hydrology, and water chemistry were obtained from published reports or public data sources.

12.5 Data verification performed by Philip Geusebroek (SLR)

QP Philip Geusebroek reviewed and verified data for the mineral resource estimate as described in Section 14 (14.1, 14.2, 14.12, 14.13.2). There was no limitation placed on access to the data needed to conduct the work.

The SLR QPs have also reviewed SLR's assay certificate verification work. The SLR QPs are of the opinion that the database is very well maintained, the database verification procedures for Sisson comply with industry standards, and the data are of sufficient quality to support the disclosure of Mineral Resources.

12.6 Data verification performed by AJ MacDonald (ISC)

The QP reviewed SRK's water treatment design basis and Knight Piesold's water balance and predictive water quality model reports and outputs, with support from Integrated Sustainability's engineering team. Verification included engineering review and selected verification of the completeness, consistency and reasonableness of the supplied treatment chemistry and flow basis. These included reviewing total and dissolved concentrations, assessing ionic balance, comparing successive design basis datasets, and evaluating implications for treatment selection, chemical consumption, solids / waste production and equipment sizing. Missing parameters, unusual concentrations and significant changes between revisions were raised with the relevant consultants for clarification, and revised information was reviewed for its implications for the WTP design.

The underlying models and source data were prepared by other consultants outside the water treatment QP's supervision. Verification focused on the supplied reports and outputs and their suitability for water treatment design; it did not include auditing the models or verifying their source data. Predicted water quality remains subject to the assumptions and limitations of those models.

The QP considers the provided data suitable for feasibility level water treatment design and cost estimates presented in this Technical Report, subject to the stated assumptions and limitations. Further confirmation of influent chemistry and treatment performance through updated modelling and treatability testing is recommended before detailed design.

12.7 Data verification performed by Jennifer Lynn McPhail (Stantec)

QP Jennifer McPhail was involved in the review of environmental, permitting, social and community impacts information for the Project as presented in Chapter 20. QP McPhail has reviewed the data and information and is of the opinion that the data used as the basis of Sections 20.1-20.4 are appropriate and adequate for the purposes of this technical report.

13 Mineral processing and metallurgical testing

13.1 Introduction

A series of metallurgical test programs have been completed on the Sisson deposit since 2006, summarized in Table 13-1.

Table 13-1: Metallurgical testwork summary

Year	Project-owner	Laboratory / Program	Samples	Program Description
2006	Geodex Minerals Ltd.	Micron Geological Ltd. Petrographic Report	5 crushed composites	Petrographic assessment on sized fractions Photomicrograhs by PL, XPL, RL and FOV SEM-EDX analysis Elemental analysis by various methods
2008	Geodex Minerals Ltd.	SGS Lakefield Research Limited Project 11710-001	Volcanics and Gabbro composites	Comminution testing QEMSCAN mineralogy, liberation analysis Mo and bulk sulphide flotation Tungsten gravity and flotation testing
2009	Geodex Minerals Ltd.	SGS Lakefield Research Limited Project 50022-001	6 drill core composites combined into a master composite	Ore sorting by Terra Vison, DE-XRT and UV sensors Heavy liquid separation, gravity release testing Gravity and flotation testing
2009	Geodex Minerals Ltd.	Inspectorate - Metallurgical Division Project 1002004	1 master composite	Centrifugal gravity concentration Heavy liquid separation Spiral gravity concentration
2010	Geodex Minerals Ltd.	Terra Vision	drill core composite	Full scale ore sorting at Commodas-Ultrasort 2 sized fractions (1.5 - 1", 1 - 0.5") Contrast filter and DE-XRT sensors applied
2012	Northcliff Resources Ltd.	SGS Canada Inc. Project 13226-001	Master composite, Gabbro sample	Comminution testing QEMSCAN mineralogy, liberation analysis Heavy liquid separation Mo, S and WO_3 flotation
2013	Northcliff Resources Ltd.	SGS Canada Inc. Project 13226-003	Domain and production year composites	HPGR and comminution testing TIMA mineralogy Heavy liquid separation, gravity concentration Mo, S and WO_3 flotation, locked cycle tests Mini-pilot and pilot flotation testing
2014	Northcliff Resources Ltd.	SGS Canada Inc. Project 13226-004	lab generated WO_3 flotation concentrate	Pressure leaching, purification, APT testing
2014-2017	Northcliff Resources Ltd.	SGS Canada Inc. Project 13226-008, 011	Year 1-10 composite, Pilot BSF Tailings, variability samples	Bond ball testing on variability samples Mo and S rougher flotation testing Ambient WO_3 flotation Hot conditioning WO_3 flotation testing

Year	Project-owner	Laboratory / Program	Samples	Program Description
2025-2026	Northcliff Resources Ltd.	Saskatchewan Research Council (SRC)	variability samples	Ore Sorting using Tomra DE-XRT
2025-2026	Northcliff Resources Ltd.	SGS Canada Inc. Project 13226-015	Master composites, variability samples	Comminution testing on variability samples Full circuit flotation testing with hot conditioning S/L separation testing on tailings

The following sections review the results of the test programs as they relate to the flowsheet selection and process design. Not all details of each test program are discussed; significant results are summarized to provide an overview of the metallurgical response of the material. Recent metallurgical testing is in progress and results available at the time of writing are included in the discussion.

13.2 Sample Origin

More than 30 tonnes of drill cores have been allocated to metallurgical testing since 2006. In some cases, the composites assembled for a given program were utilized for testing in subsequent programs. A summary of sample information and corresponding test programs is presented in Table 13-2. After the SGS 13226-011 test program completed in approximately 2016, all remaining samples and products were shipped to Northcliff's storage facility. Portions of these samples were returned to SGS in 2025 and used for testing.

Table 13-2: Metallurgical sample summary table

Year Assembled	Drill Holes	Depth Range (m)		Total Intervals Used		Litho / Domain Descriptions	Used in Program
	Drill Holes	From	To	length (m)	kg		
2006	SB-06-03	102.5	317	45	94	-	Micon Petrographic assessment
2008	SP-07-01 SBM-07-01 SB-07-21	3	348	107 178	347 1,223	Gabbro Volcanics	SGS Project 11710-001
2008	SB-06: 7 holes SB-07: 13 holes	47	360	1308	2,856	Gabbro Volcanics	SGS Project 50022-001 (initial) Inspectorate Project 1002004 SGS Project 1326-001 SGS Project 13226-015
2010	7 holes from 2006-2009	-	-	-	417	Various - Zone III	Terra Vision/Commodas-Ultrasort
2011	SB-11: 15 holes	5	252	2189	27,662	Intrusive Metavolc/FT4 Metasedimentary	SGS Project 13226-003 SGS Project 13226-008, 011
2025	SB-10: 3 holes SB-11: 7 holes	11	417	621	2,255	IGB, FTC, WKB2/3, Low Grade	SRC Phase 2 & 3 Ore Sorting SGS Project 13226-015

Year Assembled	Drill Holes	Depth Range (m)		Total Intervals Used		Litho / Domain Descriptions	Used in Program
	Drill Holes	From	To	length (m)	kg		
2025	SB-10: 3 holes SB-11: 8 holes	44	237	265	857	IQD, IGB, FTA, FTC, FT4, WKB1, WKB2/3	SGS Project 13226-015

13.3 Metallurgical Sample Composition

The following sections summarize the chemical and mineralogical data of samples tested in the programs.

13.3.1 Chemical Composition

Head assay data of metallurgical composites are presented in Table 13-3.

Table 13-3: Composite head assay data

Test Program	Comp ID	Assay %				
		WO_3	Mo	S	CaO	P_2O_5
SGS Lakefield Research Ltd Project 11710-001	Volcanics	0.090	0.036	0.54	2.5	0.12
	Gabbro	0.071	0.020	0.42	4.5	0.22
SGS Lakefield Research Ltd Project 50022-001	Master Comp	0.096	0.038	0.45	3.7	0.17
Inspectorate - Met. Division Project 1002004	Master Comp	0.106	0.032	0.56	-	-
SGS Canada Inc. Project 13226-001	Master Comp	0.100	0.480	0.55	3.9	0.16
SGS Canada Inc. Project 13226-003	Y1-10 MC	0.110	0.037	0.43	3.1	0.17
	DOM-MS	0.081	0.016	0.42	2.9	0.18
	DOM-I	0.120	0.043	0.37	7.0	0.21
	DOM-MV	0.084	0.039	0.45	2.9	0.16
	Y12 MC	0.140	0.015	0.32	3.7	0.16
	Y35 MC	0.080	0.026	0.42	3.4	0.17
	Y610 MC	0.050	0.019	0.45	3.8	0.17
	Y10 PP Feed	0.090	0.027	0.42	4.9	0.19
Saskatchewan Research Council (SRC)	IGB Feed	0.076	0.015	0.35	6.9	0.30
	FTC Feed	0.072	0.010	0.59	3.5	0.16
	WBK23 Feed	0.066	0.006	0.52	2.4	0.17
	LowA Feed	0.026	0.014	0.66	1.2	0.14
	Low B, C Feed	0.034	0.007	0.52	1.5	0.14
SGS Canada Inc. Project 13226-015	Master Comp	0.110	0.019	0.53	5.3	0.22
	V1-IQD	0.110	0.038	0.76	5.4	0.22
	V2-IGB	0.100	0.010	0.26	8.5	0.23
	V3-IGBHG	0.130	0.025	0.24	8.0	0.39

Test Program	Comp ID	Assay %				
		WO_3	Mo	S	CaO	P_2O_5
	V4-FTA	0.110	0.015	1.08	3.5	0.20
	V5-FTC	0.110	0.013	0.23	5.1	0.20
	V6-FTQ	0.040	0.012	0.61	1.9	0.11
	V7-WBK1	0.020	0.024	0.53	1.6	0.15
	V8-FT4	0.110	0.008	0.48	7.5	0.23
	V9-WBK23	0.050	0.013	0.58	1.0	0.11

The SRC head assays are averages of Phase 2 and 3 composites.

13.3.2 Mineralogical Composition

The Sisson materials are understood to have potentially economic levels of tungsten and molybdenum, primarily in the form of scheelite ($CaWO_4$) and molybdenite (MoS_2). The samples have low levels of sulphide minerals, with pyrite (FeS_2) being the dominant sulphide mineral.

Petrographic analyses by Micon identified scheelite as the dominant tungsten mineral, with few observations of wolframite ($FeWO_4$). The scheelite grain sizes varied widely, ranging from 0.01 mm to 2 mm. It was estimated that the bulk of the scheelite mass was contained within grains coarser than 0.5 mm. The observed scheelite grains were 80-90% liberated in the analyzed -0.86 +0.25 mm size fraction. There was no particular association of scheelite with other minerals, it was generally found in contact with amphibole, biotite, quartz and chlorite. Molybdenite was the main carrier of molybdenum and was observed as scattered grains ranging from 0.01 mm up to 4 mm. EDX measurements of scheelite grains indicated that low levels of molybdenum were within the grains.

The pyrite observations by Micon suggested that this mineral occurred in two forms. One appeared bright, often with crystal outlines and had few inclusions. The other had a dull pitted appearance due to numerous tiny inclusions and had generally irregular shapes that lacked crystal faces. Other sulphide minerals observed in small amounts were pyrrhotite, arsenopyrite, chalcopyrite and sphalerite.

Mineral compositions of the feed composites were measured in three SGS test programs using QEMSCAN or TIMA equipment. Mineral compositions are summarized in Table 13-4.

Table 13-4: Mineral composition of metallurgical composites

Mineral	Mineral Composition %							
	SGS 11710-001		SGS 13226-001		SGS 13226-003			
	Volcanics	Gabbro	MC	M08-G2	Y 1-10 MC	DOM-I	DOM-MV	DOM-MS
Molybdenite	0.08	0.03	0.08	0.02	0.06	0.02	0.10	0.06
Scheelite	0.27	0.27	0.22	0.09	0.12	0.38	0.19	0.22
Wolframite	0.03	0.0	-	-	-	-	-	-
Pyrite	0.75	0.88	0.55	0.45	1.13	1.22	1.43	1.51
Pyrrhotite	0.78	0.15	0.84	0.32	0.02	0.01	0.01	0.01
Other Sulphides	0.20	0.12	0.11	0.11	0.15	0.17	0.20	0.18
Quartz	35.9	20.2	27.9	9.6	28.6	13.4	29.1	29.5

Mineral	Mineral Composition %							
	SGS 11710-001		SGS 13226-001		SGS 13226-003			
	Volcanics	Gabbro	MC	M08-G2	Y 1-10 MC	DOM-I	DOM-MV	DOM-MS
Plagioclase	18.1	27.2	27.8	34.8	33.2	44.2	30.8	27.5
K-Feldspar	8.2	3.8	7.0	1.8	15.3	8.6	15.8	16.6
Chlorites	6.0	5.7	3.4	4.3	0.8	1.1	0.8	0.6
Amphibole	3.4	6.5	8.3	16.8	2.0	2.4	2.2	1.6
Muscovite	12.6	9.3	5.7	4.8	2.0	0.8	2.2	2.2
Biotite	8.1	19.2	12.7	21.0	12.6	21.6	13.2	15.5
Clays	1.26	1.05	0.46	0.42	0.04	0.01	0.04	0.05
Carbonates	1.4	1.6	0.8	0.6	0.5	1.5	2.5	3.5
Apatite	0.3	0.4	0.4	0.5				
Other	2.6	3.6	3.8	4.4	3.5	4.6	1.4	0.9

Mineral liberation values were measured on a portion of the feed composites; results are presented in Table 13-4.

Table 13-5: Mineral liberation data by QEMSCAN/TIMA

Mineral	Mineral Liberation %					
	SGS 11710-001		SGS 13226-003			
	Volcanics	Gabbro	Y 1-10 MC	DOM-I	DOM-MV	DOM-MS
Scheelite	92.3	91.7	91.4	85.6	87.9	90.0
Molybdenite	88.4	78.3	88.0	-	-	-
Grind Size Data						
100% passing µm	348	348	200	200	200	200
80% passing µm	176	205	100	100	100	100

The results indicate that the scheelite is well liberated at primary grind sizes of approximately 150 µm P_{80}. It is likely that the SGS 13226-003 composites would have continued to show high degrees of scheelite liberation at coarser grind sizes, had these been targeted for mineralogical analysis. Molybdenite was less liberated than scheelite, but still sufficiently liberated for adequate flotation performance. Typically, target mineral liberation values of 50-60% are sufficient for good rougher flotation performance, where the rougher concentrate is further reground to provide final concentrate quality liberation.

13.4 Comminution properties

Comminution tests were performed on several samples and composites over the series of test programs. A summary of Bond protocol tests results is presented in Table 13-6, a summary of drop weight test results is presented in

Table 13-7.

Table 13-6: Bond protocol comminution results

Test Program	Sample ID	Abrasion Index (g)	Crusher Wi (kWh/t)	Rod Mill Wi (kWh/t)	Ball Mill Wi (kWh/t)
SGS 11710-001	SP-G	0.07	13.7	19.2	19.1
	SP-V	0.11	10.9	17.0	15.7
SGS 13226-001	Master Comp	0.02	-	17.5	16.4
	M08-G2	0.06	-	17.6	17.2
SGS 13226-003	DOM-I Comp	0.03	-	17.2	17.5
	DOM-MS Comp	0.09	-	18.3	17.4
	DOM-MV Comp	0.10	-	17.6	16.7
	Y1-10 MC	0.29	-	17.8	17.1
	Y1-10 MC-PP	0.23	-	-	16.9
	FTQ	0.17	21.5	-	14.5
	IQD	0.12	16.9	-	15.6
	MCT	0.15	19.4	-	16.5
	IGB	0.41	17.4	-	17.9
	WKB	0.13	17.2	-	17.5
	MTF	0.26	16.2	-	14.7
	FTA	0.28	14.4	-	15.7
	IN-Y12	0.15	-	-	18.3
	IN-Y35	0.18	-	-	-
	IN-Y610	0.29	-	-	16.6
	MS-Y35	0.13	-	-	16.2
	MS-Y610	0.04	-	-	17.3
	MV-Y12	0.13	-	-	16.9
	MV-Y35	0.25	-	-	16.0
	MV-Y610	0.15	-	-	15.9
	MV-Y10	0.15	-	-	16.2
	HPGR Feed	-	-	-	16.4
Average		0.16	16.4	17.8	16.6
75th percentile		0.24	18.4	18.2	17.4

Table 13-7: Drop weight comminution results

Test Program	Sample ID	DWi (kWh/m³)	SCSE (kWh/t)	A x b
SGS 13226-015	VC-01 IQD	8.8	11.3	31.6
	VC-02 IQD	10.7	12.4	26.4
	VC-03 IGB	12.7	13.9	22.3
	VC-04 IGB	12.9	13.8	23.0
	VC-05 IGB	10.9	12.7	26.2
	VC-06 FTA	9.1	11.4	29.5
	VC-07 FTA	10.5	12.4	26.6
	VC-08 FTA	10.2	12.2	28.5
	VC-09 FTA	10.0	12.0	27.4
	VC-10 FTC	10.6	12.4	26.8
	VC-11 FTC	11.6	13.0	24.1
	VC-12 FTC	11.0	12.6	25.5
	VC-13 FTC	10.7	12.4	24.8
	VC-14 WKB1	11.3	12.7	24.7
	VC-15 FT4	10.5	12.6	27.4
	VC-16 WKB23	10.0	12.0	27.4
	VC-17 WKB23	9.5	11.7	29.4
	VC-18 FT4	8.9	12.3	30.9
Average		10.5	12.4	26.8
75th percentile		11.1	12.7	24.8

The comminution results indicate that the materials are generally very competent with respect to impact breakage. Approximately 5% of the samples in the JK Tech (SMC) test database are harder than the 75th percentile of these results. The Bond ball mill work index results indicate that the materials are moderately hard with respect to attrition grinding in a ball mill.

During the SGS 13226-003 test program, a 3,250 kg composite was assembled and provided to Koppern Machinery for high pressure grinding roll (HPGR) evaluation. The crushed feed was provided in 10 drums, of which 6 were used for open circuit tests and 4 for a closed-circuit test. The +6.35 mm product material was recycled to the next feed in closed circuit testing. The testing indicated that the material was amenable to HPGR crushing and substantial size reduction was achieved. In open circuit tests, feed moisture was varied between 1 to 4% and showed this did not have a significant effect on HPGR performance. In closed circuit testing, the -32 mm crushed feed was reduced to P80 and P50 values of 6.1 and 2.1 mm, respectively, at a specific energy consumption of 2.0 kWh/t.

13.5 Preconcentration Evaluations

The Sisson project economics could benefit from pre-concentration, as the feed material is relatively low grade and has a high comminution energy requirement. A number of methods to pre-concentrate the feed prior to the main mill process were investigated. These included particle ore sorting, SAG pebble rejection,

and heavy liquid separation. These items are discussed in the following sections but were not considered for the current flowsheet as the data and economic inputs were not sufficient to include in the current study.

13.5.1 Ore Sorting

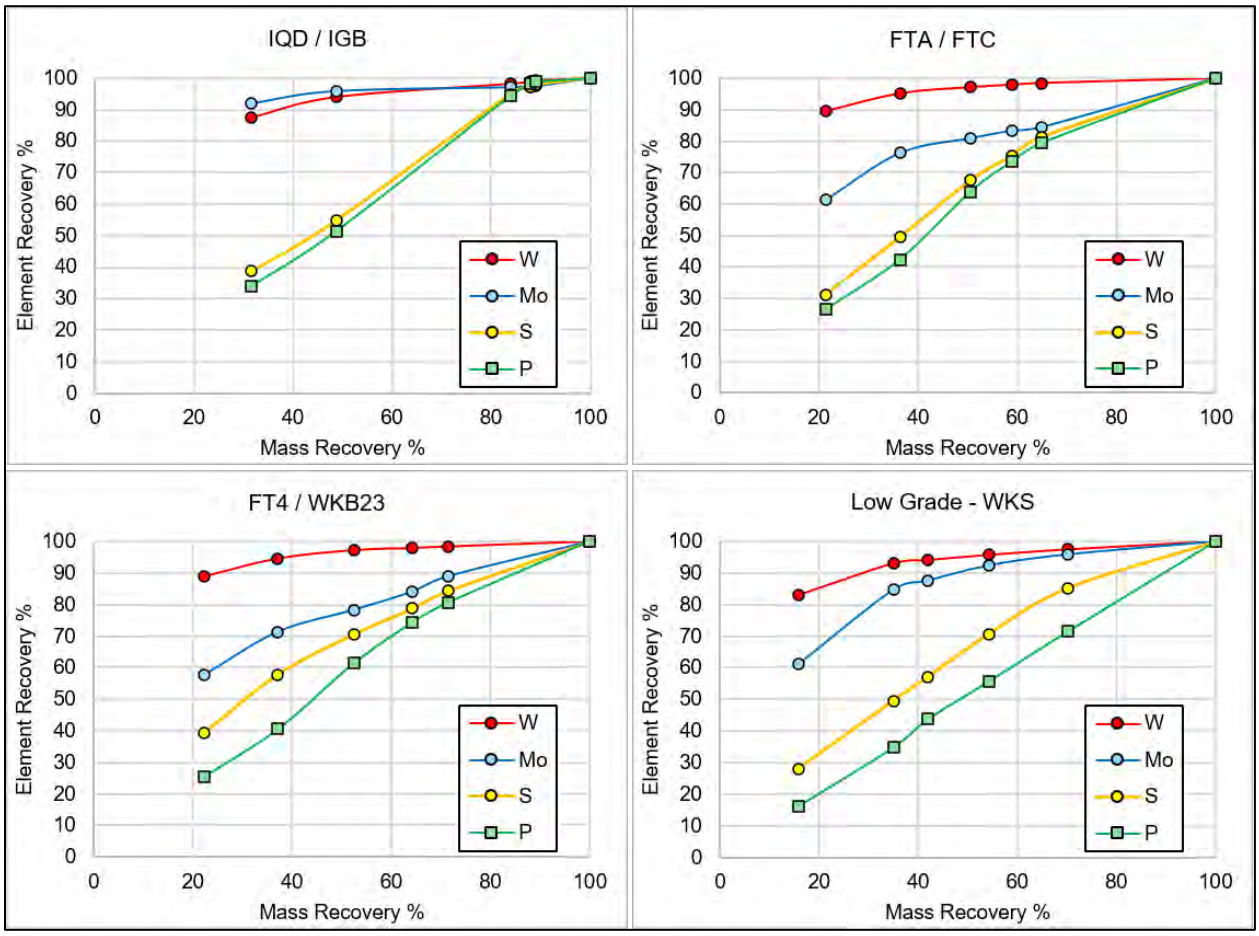
A number of particle sorting tests were conducted over the years in conjunction with metallurgical test programs. The ore sorting testing has ranged from amenability assessments to full scale sorting results on ½ drill core specimens.

The initial ore sorting evaluation was conducted by Terra Vision on -25 +12.7 mm specimens extracted from the drill core provided for the SGS 50022-001 test program. This amenability testing was conducted using Dual Energy X-Ray transmission (DE-XRT) and Ultraviolet (UV) sensors. Results suggested that tungsten could be concentrated successfully by both sensors, recovering approximately 92% of the WO_3 in about 40% of the specimen mass. Molybdenum could also be concentrated by a DE-XRT sensor, recovering 86% of the Mo in the sample set into 40% of the feed mass.

A full-scale sorter test was subsequently conducted in collaboration with Terra Vision using coarse samples assembled from selected intervals of ½ core from 7 drill holes. The material was stage crushed to -1.5 inches and screened into -1.5″+1″ and -1″+0.5″ fractions. Approximately 13% of the sample mass reported to the -0.5″ fraction and not tested. The two prepared fractions, 417 kg in total, were shipped to the Commodas-Ultrasort testing facility in Germany and processed using criteria developed on a sub-sample training set. Each size fraction was cone and quartered and three portions from each were tested using progressively less selective sensor criteria. Two sensors were applied, a Contrast Filter was used to eject a first concentrate and the intermediate waste was then reprocessed using DE-XRT to scavenge a second concentrate. The medium selectivity algorithm recovered about 88% of the WO_3 and Mo from the finer fraction feed to a total concentrate containing 48% of the mass. The same algorithm recovered about 94% of the WO_3 and 88% of the Mo to a mass of 59% of the coarser feed.

Ore sorting was investigated in 2025 in two phases of testing on five different lithological composites. The testing was conducted at the Saskatchewan Research Council on a full scale Tomra sorting machine using a DE-XRT sensor. In total, about 2.2 tonnes of ½ drill core was manually broken into suitable specimen sizes and allocated for testing. The first phase was conducted on a smaller portion of the specimens to establish sorting response from up to seven different sensor settings. These results were used to establish five sensor settings for the subsequent phase, applied to larger sample portions. The results of the optimized sorting runs on four material types are presented graphically in Figure 13-1.

Figure 13-1: Phase 3 ore sorting response – SRC



Source: Ausenco, 2026

The mass and element recovery details to the first two sensor classes are presented in Table 13-8.

Table 13-8: Phase 3 ore sorting – Class 2 DE-XRT Sensor Setting Results

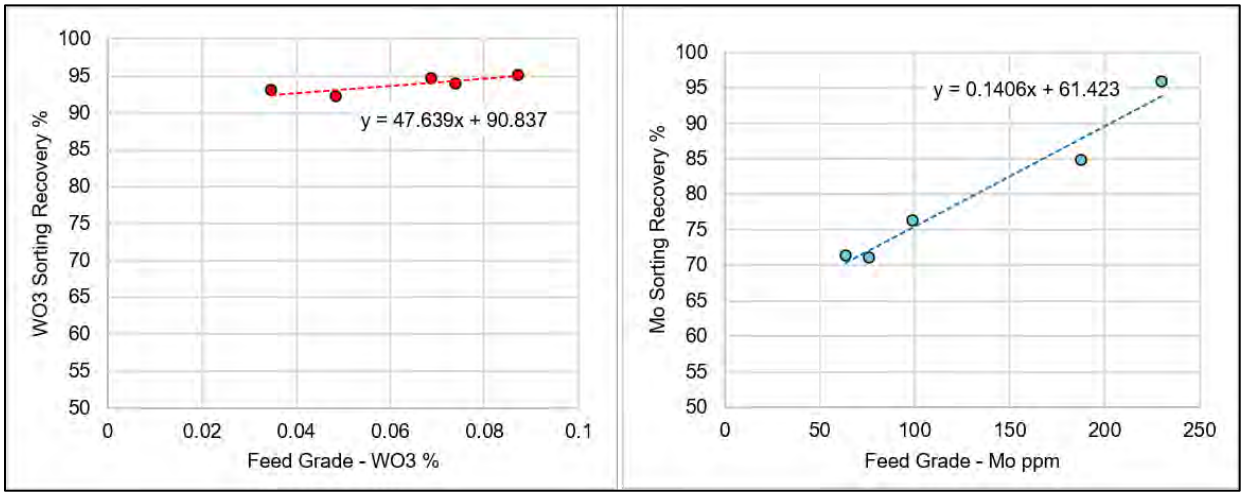
Composite	Recoveries to Class 2 Accept - %						
	mass	W	Mo	S	Ca	P	Fe
IQD / IGB	48.7	93.9	95.9	54.8	47.9	51.3	52.0
FTA / FTC	36.3	95.2	76.3	49.4	41.7	42.4	43.8
FT4 / WKB23	37.1	94.7	71.3	57.8	43.1	40.5	42.5
Low Grade - WKS	35.0	93.1	84.8	49.5	36.9	35.0	37.9
Low Grade - WKB1	34.5	92.2	71.1	46.6	41.6	36.8	36.8

The IQD/IGB material had a higher mass recovery at this setting compared to the other materials. These materials lie on the west side of the Sisson Shear and likely have a different mineralogical composition. The higher mass pull could be related to greater dissemination of the high-density mineral grains, or it could be related to a higher apparent atomic density of the host rock.

In all sorting runs, tungsten appeared to be readily recovered to the "accept" or concentrate stream. Molybdenum recovery varied and was generally lower. Diluents such as Ca and P tended to match the mass recovery, so were generally not upgraded. Sulphur would typically be expected to concentrate due to the higher density of sulphide minerals. There was some sulphur upgrading, but it was clearly less responsive than the target minerals.

The Class 2 accept recoveries for W and Mo could be related to feed grades, as presented graphically in Figure 13-2.

Figure 13-2: Ore sorting Class 2 Recoveries vs. Feed grades - SRC



Source: Ausenco 2026

A conceptual mass balance, presented in Table 13-9, can be estimated with some process assumptions:

- 25% of the secondary crusher product will be too fine for particle sorting and will advance directly to the mill feed.
- A small increase of 10% in feed grade is applied to the crushed fines and respective reduction in the coarse material reporting to the sorter.
- The mass recovery across the ore sorter is weighted as 40% from IQD/IGB material and 60% from FTA/FTC material, or 42.3%.
- Metal recoveries across the ore sorter are as per feed grade relationships in Figure 13-2.
- WO_3 and Mo recoveries from the mill feed are 78% and 82%, respectively.

The mass balance suggests that the concentrator feed could be reduced by 44% with a loss of 3.7% and 8.4% of the mined WO_3 and Mo, respectively, to the sorted rejects. No recovery increase has been attributed to the concentrator flotation process due to the increase in mill feed grade, however this may be warranted. This flowsheet would need to be evaluated using a larger feed mass and actual crushing to get more confidence in sorter performance.

Table 13-9: Conceptual mass balance with ore sorting

Stream	mass %	Assay %		Distribution %	
		WO_3	Mo	WO3	Mo
Mined Material	100	0.090	0.020	100	100
Crusher Fines	25.0	0.099	0.022	27.5	27.5
Sorter Feed	75.0	0.087	0.019	72.5	72.5
Sorter Accept	31.0	0.20	0.041	68.8	64.1
Sorter Reject	44.0	0.007	0.004	3.7	8.4
Mill Feed	56.0	0.155	0.033	96.3	91.6
Mill Tailings	55.9	0.034	0.006	21.2	16.5
WO_3 Concentrate	0.012	55.0	-	75.1	-
Mo Concentrate	0.030	-	50.0	-	75.1

13.5.2 SAG Pebble Sorting

A series of tests were conducted in the 2025 metallurgical test program at SGS to investigate grade differential and potential rejection of SAG pebbles. The tests were designed to simulate "pebble" production in a semi-autogenous grinding arrangement, as best as can be achieved using ½ drill core.

Half core material from four variability samples and a master composite were coarse crushed to approximately 80% passing 30 mm. An 18" diameter MacPherson grinding mill apparatus was loaded with 18 kg of 2.5" steel ball media and 15 kg of each crushed sample. The mill apparatus was run in a batch, dry grinding mode for 170 minutes to target a "pebble" content of 25% of the feed mass retained on 19 mm. The product was screened on 4 sized fractions. A manual sorting protocol was applied to the pebbles in the -31.5+25 mm and -25+19 mm fractions, in which pebbles with a modest amount of observable fluorescing scheelite were accepted and mostly barren pebbles were rejected. Images of unsorted pebble sets under UV light from one sample are presented in Figure 13-3.

Figure 13-3: Images of IQD Pebbles under UV Light



Source: SGS Canada

The products were then crushed and assayed. Metallurgical balances for the four variability sample tests are presented in Table 13-10 and Table 13-11. The master composite result had a poor head match, possibly due to excessive fines losses, that could not be resolved so this is not reported. Some of the molybdenum assays were below detection limits, so values of 0.005% were used.

Table 13-10: Pebble sorting metallurgical balances - Tungsten

Stream	V1-IQD			V2-IGB			V4-FTA			V5-FTC		
	mass %	WO_3 %	Dist. %	mass %	WO_3 %	Dist. %	mass %	WO_3 %	Dist. %	mass %	WO_3 %	Dist. %
-31.5+25 mm Accept	3.2	0.14	3.8	4.4	0.76	20.7	1.2	0.10	1.4	2.1	0.75	8.4
-31.5+25 mm Reject	11.5	0.02	2.0	16.2	0.02	2.0	10.2	0.02	2.4	13.3	0.02	1.4
-25+19 mm Accept	2.4	0.28	5.7	2.8	0.47	8.0	1.8	0.44	9.6	2.3	0.47	5.5
-25+19 mm Reject	13.1	0.02	2.3	14.3	0.04	3.5	11.1	0.02	2.6	14.6	0.02	1.5
-19+12.7 mm	6.4	0.08	4.4	6.8	0.10	4.2	6.5	0.02	1.6	7.6	0.04	1.6
-12.7 mm	63.4	0.15	81.8	55.6	0.18	61.7	69.2	0.10	82.4	60.0	0.26	81.5
Calculated Head	100	0.12	100	100	0.16	100	100	0.08	100	100	0.19	100
Direct Head		0.11			0.10			0.11			0.11	

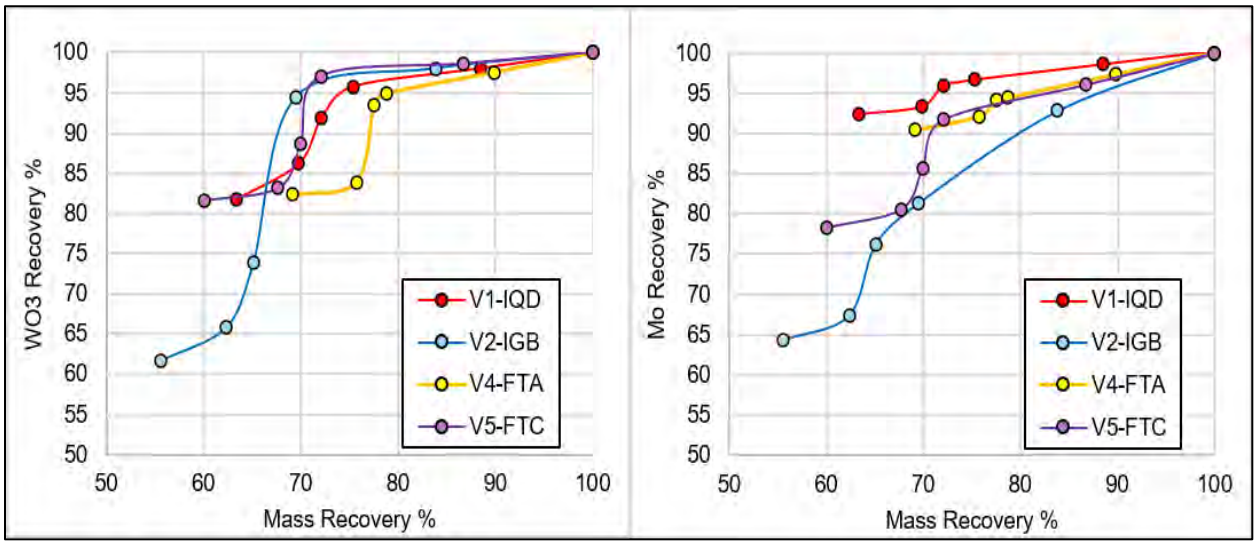
Table 13-11: Pebble sorting metallurgical balances - Molybdenum

Stream	V1-IQD			V2-IGB			V4-FTA			V5-FTC		
	mass %	Mo %	Dist. %	mass %	Mo %	Dist. %	mass %	Mo %	Dist. %	mass %	Mo %	Dist. %
-31.5+25 mm Accept	3.2	0.008	0.7	4.4	0.013	5.1	1.2	<0.005	0.3	2.1	0.048	6.1
-31.5+25 mm Reject	11.5	<0.005	1.7	16.2	<0.005	7.2	10.2	<0.005	2.7	13.3	<0.005	3.9
-25+19 mm Accept	2.4	0.034	2.3	2.8	0.036	8.9	1.8	0.021	2.0	2.3	0.038	5.1
-25+19 mm Reject	13.1	<0.005	1.9	14.3	0.009	11.5	11.1	<0.005	2.9	14.6	<0.005	4.3
-19+12.7 mm	6.4	<0.005	0.9	6.8	<0.005	3.0	6.5	<0.005	1.7	7.6	<0.005	2.3
-12.7 mm	63.4	0.050	92.4	55.6	0.013	64.4	69.2	0.025	90.4	60.0	0.022	78.3
Calculated Head	100	0.034	100	100	0.011	100	100	0.019	100	100	0.017	100
Direct Head		0.038			0.010			0.015			0.013	

The results indicate that there is some upgrading in the -19 mm material, and that the visual sorting under UV light was effective at recovering tungsten. A cumulative representation of the results is presented in Figure 13-4, in which the first 2 points are the fine fractions followed by the accept products. The results suggest that the IGB material is possibly more disseminated, as higher portions of WO_3 and Mo remained in the rejected pebble streams, particularly for molybdenum. The IGB and FTC samples had greater pebble masses, suggesting more competent material. These results agree with the SMC data.

A potential mass balance for a SAG milling circuit reflective of this data is presented in Table 13-12. The mass splits and metal distributions were averaged from the four variability sample results. Concentrator flotation recoveries estimated in the previous section are applied. In this circuit, the SAG mill would be designed to have a high pebble discharge rate of approximately 31% of the feed mass. The pebble stream would report to a sorter device and the accept stream, 5% of the total feed mass, would report to a pebble crusher and return to the SAG mill feed. The resulting ball mill feed would be approximately 74% of the feed mass. The rejected 26% of the feed mass in pebbles would need to be conveyed to a suitable disposal location. This arrangement exploits the SAG mill's potential to operate as a first stage sorting device, rejecting low grade and more competent materials. It is possible that DE-XRT sorting could provide higher metal recoveries from the pebble stream.

Figure 13-4: Pebble sorting cumulative results



Source: Ausenco 2026

Table 13-12: Conceptual mass balance with SAG pebble sorting

Stream	mass %	Assay %		Distribution %	
		WO3	Mo	WO3	Mo
Mined Material	100	0.090	0.020	100	100
SAG Trommel U/S	68.9	0.090	0.024	79.8	83.3
Pebbles (Sorter Feed)	31.1	0.058	0.011	20.2	16.7
Sorter Accept	5.0	0.275	0.029	15.4	7.4
Sorter Reject	26.1	0.017	0.007	4.8	9.3
Ball Mill Feed	73.9	0.116	0.025	95.2	90.7
Mill Tailings	73.9	0.026	0.004	20.9	16.3
WO3 Concentrate	0.012	55.0	-	74.2	-
Mo Concentrate	0.030	-	50.0	-	74.4

13.5.3 Heavy Liquid Separation

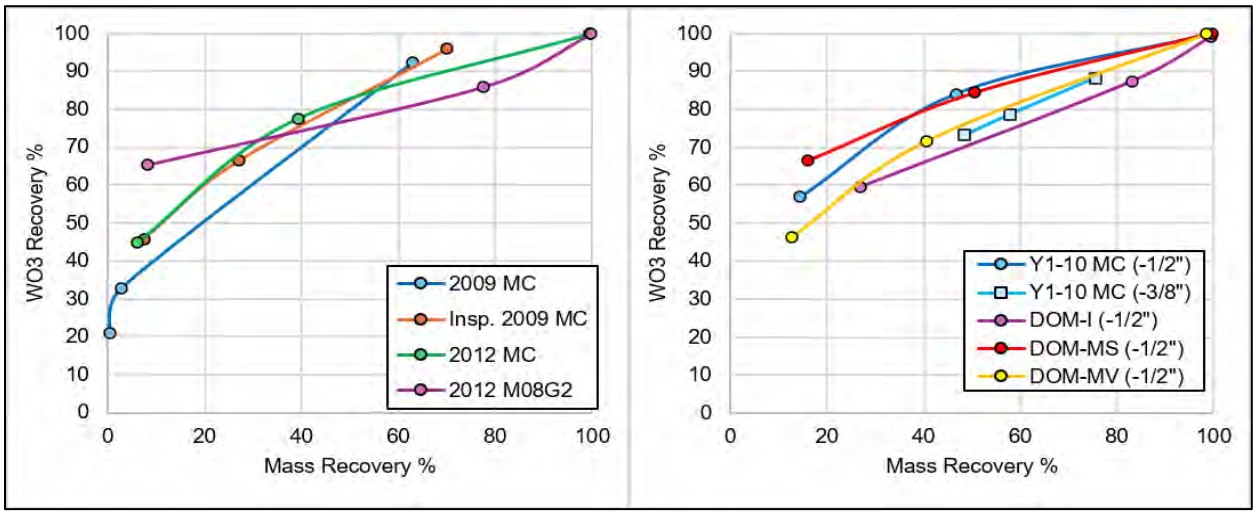
Heavy liquid separation (HLS) tests were conducted in four of the past test programs, to pre-concentrate crushed feed and understand the potential to apply dense media separation.

The tests were conducted on material crushed to either -1/2" or -3/8" (-13 or -10 mm). Material finer than a certain size was not subjected to the HLS testing as it would be difficult to separate from the dense media slurry. The size fraction of the fines excluded varied for each set of tests. The tests were conducted using liquids with a range of specific gravity values, however, these ranges varied somewhat across the tests. A summary of the HLS conditions is presented in Table 13-13. The products were analyzed on a size fraction basis, combined size fraction results for tungsten are presented in Figure 13-5.

Table 13-13: Heavy liquid separation test conditions

Program	Composite	Feed kg	Crush size (mm)	Fines			SG Points
				Size (mm)	mass %	WO_3 Dist. %	
2009 SGS	2009 MC	39	-13	-0.85	9.4	8.5	3.3, 3, 2.7
2009 Inspectorate	Insp. 2009 MC	10	-10	-0.85	5.9	7.1	2.94, 2.8, 2.6
2012 SGS	2012 MC	10	-10	-0.42	7.2	17.9	2.96, 2.8, 2.6
	M08G2	10			9.4	20.6	
2013 SGS	DOM-I	25	-13	-1.4	27.5	42.9	2.9, 2.75, 2.6
	DOM-MS	25			16.5	28.2	
	DOM-MV	25			15.9	33.1	
	Y1-10 MC	25			19.8	40.3	
	Y1-10 MC	10	-10	-0.6	14.0	27.0	2.75, 2.7, 2.65

Figure 13-5: Heavy liquid separation results – WO_3 recovery vs. mass recovery



Source: Ausenco 2026

The results on the master composites suggest that it is possible to reject approximately 40% of the crushed HLS feed to a float stream that would contain about 10% of the feed tungsten using a heavy liquid with a specific gravity value of about 2.7. The results were not always consistent. High SG minerals appeared to be more disseminated in the M08G2 gabbro composite as 78% of the HLS feed mass sank in the SG 2.8 liquid. A similar result was measured on the DOM-I composite. Both of these materials are understood to represent material from the west side of the Sisson shear. A test was repeated on the Y1-10 composite at a finer size distribution and a tighter range of heavy liquid SG values, however, the results were inferior to the initial test. Molybdenum was concentrated in the sink streams, but to a lesser extent.

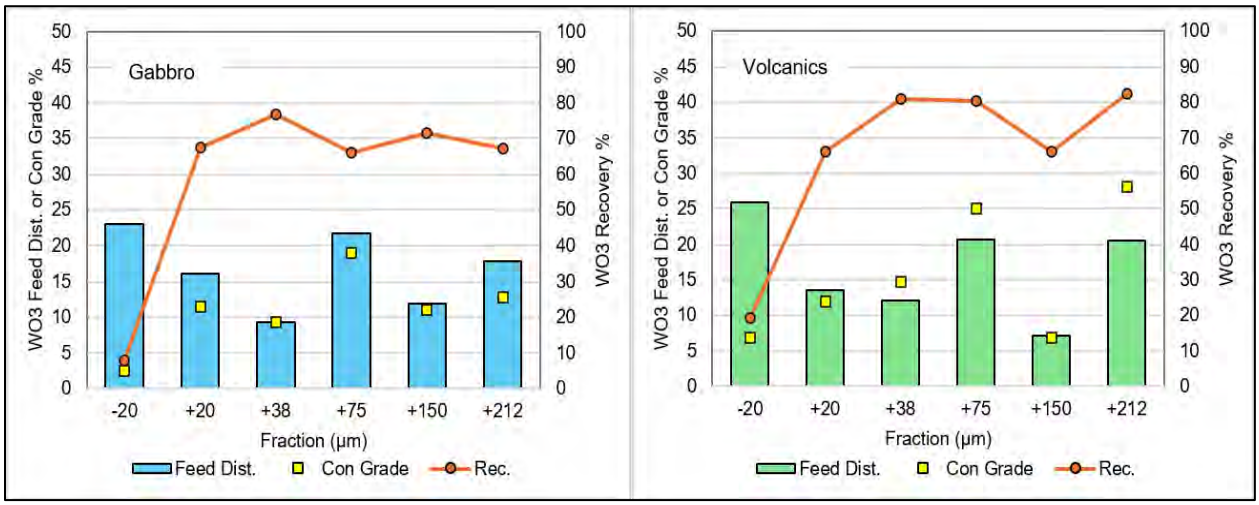
The Sisson materials are not considered to be effectively concentrated using dense media separation as the mass rejection is modest given the metal losses, and the separation is sensitive to small variations in separation liquid SG values.

13.6 Gravity Concentration

Several investigations into recovery of tungsten by gravity means have been conducted over the testing history. These included various shaking table technologies such as Wilfley, Deister and Mozley, as well as spirals and centrifugal concentrators.

The first evaluation was conducted in the 2008 SGS program with two flowsheet configurations. Gabbro and volcanic composites were stage ground to 100% passing 348 µm and flotation was applied to recover Mo and S ahead of gravity separation. The flotation concentrates contained 78% and 54% of the feed sulphur in the gabbro and volcanic composites, respectively, as well as 85-86 % of the molybdenum. The flotation tailings were separated into 6 sized fractions and a two-stage gravity concentration step, Mozley followed by super panner, was applied to each fraction. Overall, 55% and 62% of the tungsten in the gravity feeds was recovered to the gabbro and volcanic gravity concentrates, respectively. The total concentrates graded 11.1% and 15.2% WO_3, respectively. Results are presented graphically in Figure 13-6. Gravity recovery performance for the material coarser than 20 µm was considerably better than the finest fraction, averaging 69% and 77% for the gabbro and volcanic materials respectively, to concentrates grading 12.6% and 16.9% WO_3.

Figure 13-6: Fractional gravity concentration results – SGS 2008

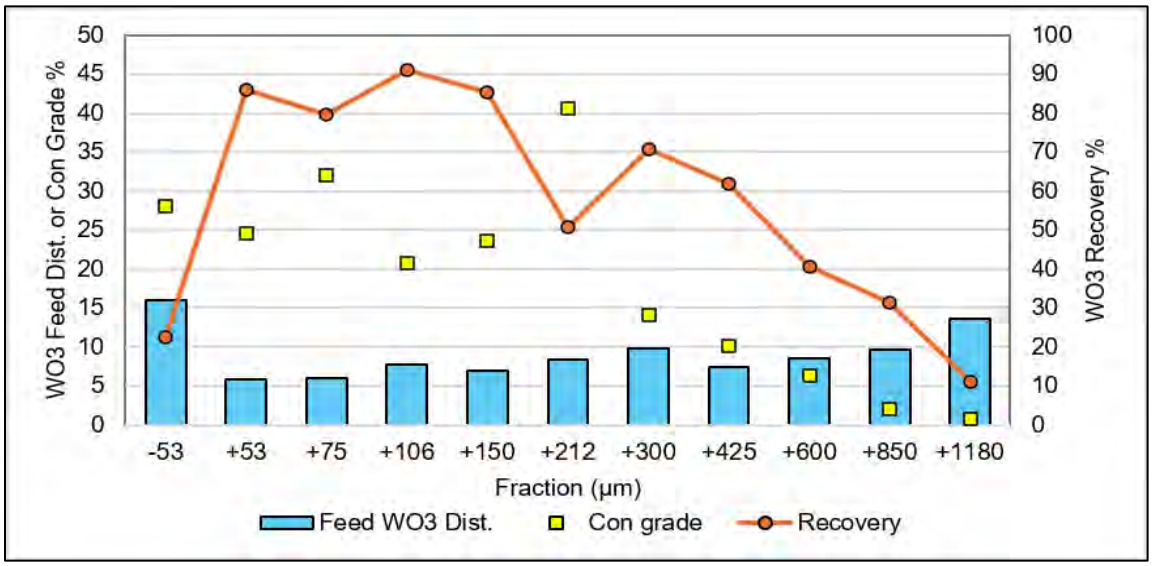


Source: Ausenco 2026

This testing was followed by a multi-stage test on the gabbro composite stage ground to 100% passing 850 µm. Molybdenite and sulphide mineral flotation was similarly applied prior to gravity concentration, however only 79% of the molybdenum and 61% of the sulphur were recovered at this coarse grind size. The flotation tailings were processed through a complex gravity flowsheet that included initial Wilfley tabling, regrinding, magnetic separation, and upgrading on Mozley equipment with additional regrinding and centrifugal scavenging. The net tungsten recovery was 52.7% to a concentrate grading 58.9% WO_3.

Another size fraction gravity assessment was conducted in the 2009 SGS program. 20 kg of the master composite at a crush size of approximately 1200 µm P_{80} was screened into 11 size fractions. Each fraction was processed by a two-stage gravity separation using a Deister table followed by Mozley upgrading. The gravity feed was not preceded however by flotation. Results are presented graphically in Figure 13-7.

Figure 13-7: Fractional gravity concentration results – SGS 2009



Source: Ausenco 2026

The results suggest that gravity concentration in the -425 +53 µm size class is worth considering, as 76% of the tungsten in that size range was recovered to a concentrate grading 22% WO_3. Approximately 13% of the sulphur in that size range was also recovered to the gravity concentrate, resulting in the concentrate grading 14.6% S. Sulphur distribution in the feed mineralogy suggests that about 49% of this gravity concentrate is sulphide minerals, with pyrrhotite accounting for about 19%. Separation of these sulphides by magnetic separation and/or froth flotation has the potential to achieve a saleable concentrate quality.

Three additional tests in the 2009 SGS program investigated gravity concentration at different points in the flowsheet. The first test applied gravity concentration on a master composite feed charge stage ground to 100% passing 850 µm, however only the +300 µm fraction was sent to gravity concentration. It is unknown how much of the feed tungsten reported to the gravity circuit. The 2 stages of gravity concentration, Deister plus Mozley upgrading, recovered 17.7% of the feed tungsten to a concentrate grading 13.5% WO_3 and 17.7% S.

A second test applied gravity concentration on a bulk sulphide flotation (BSF) tailings stream. The master composite was ground to 100% passing 212 µm and treated by flotation to recover Mo and S concentrates. The BSF tailing was deslimed and the remaining mass was processed by a similar 2 stage gravity concentration protocol. The gravity products were each analyzed at +75 and -75 µm fractions. Approximately 26% of the tungsten in the gravity feed was in the –75 µm fraction. Results indicate that the –75 µm material did not perform well across the gravity circuit as only 5% of the tungsten in this fraction was recovered. The coarse fraction performed significantly better, 77% of the tungsten in this fraction was recovered to a concentrate grading 22.2% WO_3 and 0.28% S. The +75 µm gravity concentrate contained 52.3% of the total feed tungsten. The highest-grade portion of the +75 µm Mozley concentrate assayed 54.8% WO_3 and contained 41.2% of the total tungsten feed.

A third test investigated gravity scavenging on a tungsten rougher flotation tail. The master composite was ground to 100% passing 300 µm and processed though Mo, S and WO_3 flotation in series. The tungsten flotation applied a relatively high $NaSiO_3$ dosage compared to recent testing, and as a result 60% of the feed tungsten reported to the rougher tailings. A desliming step was applied prior to gravity concentration which removed –53 µm material. This deslime stream, which contained 17% of the feed WO_3, reported to final tailings. Two methods of gravity scavenging were investigated; a Deister table and Falcon concentrator followed by Mozley table upgrading. The Deister table recovered 82% of the tungsten in the gravity circuit feed to a concentrate grading 3% WO_3. The Falcon/Mozley table process recovered 67% of the tungsten in the gravity feed to a concentrate grading 43% WO_3. The net tungsten recovery contribution from the two processes was 36 and 30%, respectively.

The second and third tests described above suggest that a circuit that includes gravity concentration on a BSF tailings stream might be a suitable process, provided it is supplemented with tungsten flotation for fines recovery.

A series of tests were conducted in the 2009 Inspectorate program using a Falcon concentrator in a series concentrating protocol, as well as spiral tests on varying size fractions. The results are not conclusive as the mass recoveries were quite high and the gravity concentrates were not marketable.

The most recent gravity concentration tests were completed in the 2013 SGS program. A 20 kg portion of the Y1-10 master composite was stage ground to 100% passing 300 µm and screened into +75 and -75 µm fractions. The -75 µm material was further deslimed with a small cyclone. 2 kg portions of the +75 µm and deslimed -75 µm fractions were processed by consecutive passes through a Knelson concentrator. The remaining mass of each fraction was processed in consecutive passes across a Wilfley table. Results for the first pass Knelson separations are presented in Table 13-14.

Table 13-14: Knelson concentrating results – SGS 2012

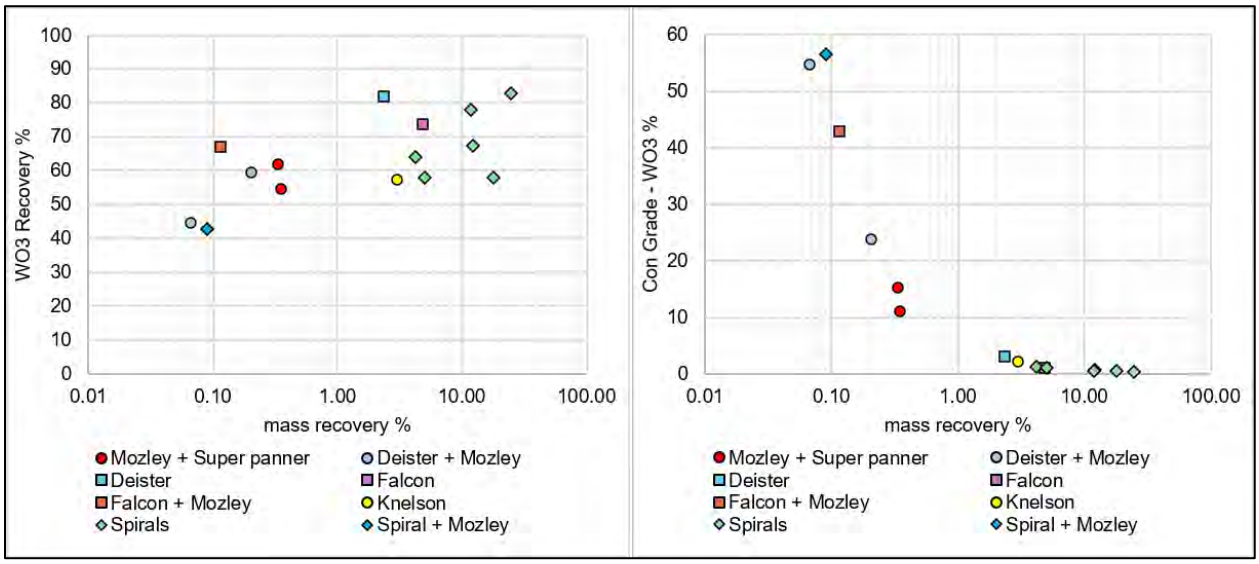
Fraction	Fraction mass %	Feed WO3		Recovery %		Con Assay %	
		Assay %	Dist. %	mass	WO3	WO3	S
-300+75 µm	49.8	0.10	49.0	2.0	41.7	2.3	5.1
-75+20 µm	36.7	0.12	43.3	1.0	15.7	1.6	3.9
-20 µm	13.5	0.06	7.7	-	-	-	-
Total	**100**	**0.10**	**100**	**3.0**	**57.5**	**2.0**	**4.7**

The results indicate that the unit tungsten recovery across the Knelson concentrator was quite high for the +75 µm fraction, measured at 85.2%. Unit tungsten recovery in the finer fraction was lower at 36%. In both fractions, sulphur recovery was about 57% of the tungsten recovery. The mass recoveries were somewhat high, which is a function of the feed mass selection. The concentration ratio would be considerably higher using a larger feed mass with likely a modest effect on tungsten recovery. The combined Wilfley table first pass recoveries were 59.5% WO_3 at a 4.5% mass recovery and 1.5% WO_3 concentrate grade.

A series of tests were conducted with spirals using BSF tailings from the mini-pilot plant, master composite material that had been ground to approximately 80% passing 150 µm via continuous closed-circuit grinding. Tungsten recovery averaged 68% at a mass recovery of 12.6%. A subsequent test using 58 kg of similar feed was processed by spirals and the spiral concentrate upgraded on a Mozley table. Tungsten recovery measured 42.7% to a concentrate grading 56.6% WO_3.

Gravity test results on ground feeds discussed above are summarized graphically in Figure 13-8 to compare methods and provide an overview of performance. Results are all relative to each gravity circuit feed, with mass recovery on a logarithmic scale. Note that gravity feeds of the square points are after some WO_3 flotation, so it is likely that a significant portion of the fine scheelite is not present in the gravity feed, which could be biasing the recoveries to appear high. Overall, the data suggests that 40 to 50% of the feed tungsten might be recoverable to a gravity concentrate that would be considered marketable. The results highlight a need to conduct these types of tests with sufficient feed mass, so that very low mass recoveries can be investigated.

Figure 13-8: Gravity results comparison – ground feed samples



Source: Ausenco 2026

Gravity concentration was excluded from the current flowsheet as it was considered that it did not improve overall tungsten recovery compared to a flotation only circuit. Gravity plus flotation testing has been limited in comparison to the amount of flotation only testing completed to date.

13.7 Molybdenum Flotation

Molybdenite in the Sisson material can be selectively recovered in the first stage of a sequential flotation circuit. The very first, unoptimized flotation tests conducted in the 2008 SGS program recovered 90 to 92% of the molybdenum to rougher concentrates containing approximately 1% of the feed mass. Many flotation tests with molybdenum flotation as the first step were performed over the testing history and provided consistent rougher flotation results. A compilation of molybdenum rougher circuit performance is presented in Table 13-15. The shaded rows represent the average of multiple tests.

The results indicate that molybdenum is amenable to recovery by rougher flotation at the primary grind sizes evaluated. The average molybdenum recovery of the values in this table is 88.9% to rougher concentrates containing 1.9% of the feed mass and grading 1.4% Mo. There did not appear to be any strong relationships between molybdenum rougher recovery and feed grade or mass pull.

The average deportment of WO_3 to the rougher concentrates is 1.7%. This recovery did have an increasing trend with mass pull, as would be expected through entrainment. The developed flowsheet has the molybdenum cleaner tailings reporting to final tailings, so scheelite in the molybdenum rougher concentrate would be a loss. Returning the molybdenum cleaner tailings to the bulk sulphide feed stream should be investigated.

Table 13-15: Molybdenum rougher concentrate results

Program	Sample	Prim. Grind	Feed	RoCon	Ro Con Grade %			Distribution %		
		P_{80} µm	Mo %	mass %	WO_3	Mo	S	WO_3	Mo	S
11710-001	Volcanics	200	0.038	1.1	0.10	3.28	9.7	1.2	92.5	16.8
	Gabbro	180	0.025	1.3	0.11	1.74	5.5	1.8	89.8	15.6
13226-001	Master Comp	170	0.042	1.9	0.16	2.14	12.2	3.2	90.3	46.5
	Gabbro	167	0.033	1.7	0.14	1.90	7.5	2.4	87.6	34.6
13226-003	Master Comp	167	0.027	2.0	0.08	1.26	11.5	1.6	89.0	52.0
	PP-02	159	0.019	2.5	0.05	0.70	4.6	2.5	90.0	30.3
	PP-03	159	0.019	2.1	0.07	0.85	4.6	1.6	88.6	25.0
	PP-04	170	0.019	2.0	0.07	0.97	5.6	1.8	88.7	27.7
	Y 1-10 MC	136	0.029	1.3	0.08	2.09	10.9	1.0	93.4	33.0
	DOM-I	161	0.024	1.4	0.09	1.55	5.7	1.1	86.8	21.4
	DOM-MS	136	0.021	1.7	0.06	1.15	6.5	1.0	92.4	26.2
	DOM-MV	136	0.027	1.5	0.07	1.70	8.2	1.1	93.2	26.9
	Y 6-10	130	0.029	3.2	0.05	0.87	9.6	2.1	95.1	57.8
	Y 3-5	130	0.022	1.5	0.12	1.34	13.7	1.6	92.0	57.2
	Y 1-2	130	0.012	2.3	0.09	0.39	9.8	2.0	76.8	68.0
	Y >10	130	0.028	2.4	0.06	0.90	13.2	1.5	76.5	67.2
Average		154	0.026	1.9	0.09	1.43	8.7	1.7	88.9	37.9

Upgrading of the molybdenum rougher concentrate was investigated in a few of the earlier test programs, but most extensively in the 2013 SGS program. In the 2013 program, molybdenum rougher concentrate generated in the mini pilot plant was combined and split into 1500 gram feed charges for batch cleaner tests. The series of tests established suitable cleaner circuit conditions which included six cleaner stages, regrinding the 1st cleaner concentrate and additions of NaCN and Na_2S as depressants for pyrite.

The developed molybdenum cleaner circuit conditions were applied to portions of rougher concentrates generated in the mini pilot plant and pilot plant. Results are presented in Table 13-16. Molybdenum cleaner circuit recovery averaged 93.4% for tests 1 and 2, M-LCT2 was a different flowsheet and not averaged.

Table 13-16: Molybdenum locked cycle test results – SGS 2013

Test #	Feed Sample	Product	mass %	Grade %	Mo Distribution %	
				Mo	Test	Overall
M-LCT1	MPP	6th Cleaner Concentrate	1.32	48.9	95.0	84.7
		1st Cleaner Tailings	98.7	0.03	5.0	4.4
		Head (Calc.)	100.0	0.68	100.0	89.1
M-LCT2	PP	6th Cleaner Concentrate	1.9	51.60	84.8	75.6
		2nd Cleaner Tailings	7.1	0.86	5.2	4.6
		1st Cleaner Tailings	91.0	0.13	10.0	8.9
		Head (Calc.)	100.0	1.17	100.0	89.1
M-LCT3	PP	6th Cleaner Concentrate	2.29	46.9	91.8	81.8
		1st Cleaner Tailings	97.7	0.10	8.2	7.3
		Head (Calc.)	100.0	1.17	100.0	89.1

13.8 Bulk Sulphide Flotation

Bulk sulphide flotation is included in the flowsheet to remove sulphur so that it does not advance into the tungsten flotation circuit. Sulphur is a penalty item in tungsten concentrates, and a level below 6% is targeted. Since the concentration ratio is so high across the tungsten flotation circuit, even a small amount of sulphur advancing can compromise the concentrate quality.

The bulk sulphide flotation circuit simply applies adequate dosages of a non-selective sulphide collector, potassium amyl xanthate (PAX) to recover sulphide minerals into a rougher concentrate. As this stream reports to tailings, it is important to limit entrainment and associated fine scheelite losses.

The 2013 production composite and lithology variability tests indicate that, on average, 2.1% of the feed mass reports to the bulk sulphide rougher concentrate. On average, this flotation stage recovers most of the remaining sulphur in the molybdenum rougher tailings, such that the combined circuits recover 87% of the feed sulphur.

13.9 Scheelite Flotation

Flotation of the tungsten bearing mineral scheelite has been investigated extensively in six metallurgical test programs. In general, this oxide mineral flotation process can be challenging, as it is reagent intensive, can be difficult to achieve selectivity against other oxide minerals, and often results in high circulating loads. As well, the low tungsten grade of the feed requires very high concentration ratios to produce marketable final concentrates. As a result, the high concentration ratios require larger test feed masses in order to generate cleaner concentrate masses that provide sufficient froth loading in small laboratory flotation cells.

Initially, this testing was focussed on what might be considered conventional oxide mineral flotation, in which a lower quality concentrate was generated that was expected to be subsequently upgraded by hydrometallurgical means. In 2014, a cleaner flotation stage that included conditioning at an elevated

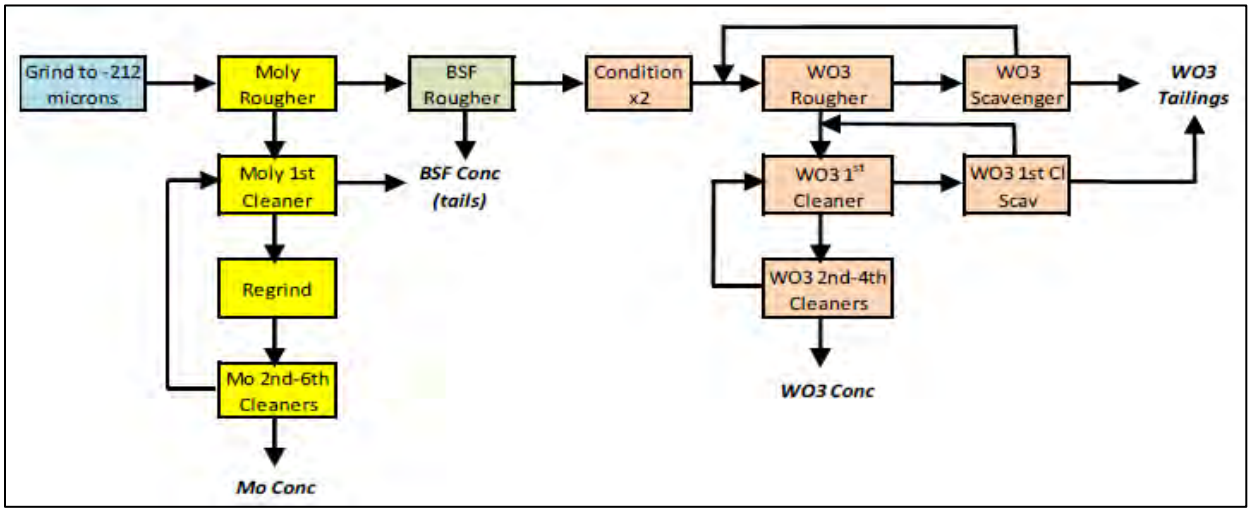
temperature was introduced, which had shown the potential to generate marketable high grade final concentrates. Both flotation strategies are discussed in the following sections.

13.9.1 Conventional Scheelite Flotation

Suitable flotation conditions for scheelite recovery post molybdenum and sulphide mineral flotation were established over the series of test programs. The process included a series of conditioning stages in which pulp chemistry was adjusted with sodium hydroxide and soda ash, addition of a Ca-mineral depressant, quebracho, followed by addition of sodium silicate as a silicate depressant, and finally the addition of a fatty acid collector. Additional dosages of these reagents were also added downstream in the rougher and cleaner flotation circuits. Many tests were conducted to evaluate the effect of varying reagent dosages, as well as collector types. The overall process flowsheet in block diagram format is presented in Figure 13-9.

This flowsheet was tested extensively in the 2013 SGS program, in various forms, using the Year 1-10 master composite. The testing included processing 3 tonnes of feed through a mini pilot plant to generate molybdenum rougher concentrate for cleaner upgrading tests and produce a bulk mass of BSF tailings for tungsten circuit process development testing. A total of 65 batch tests were conducted to evaluate the tungsten flotation process conditions. The first 18 of these tests used crushed 5 kg feed charges and so included Mo and S flotation, while the remaining 47 were conducted using BSF tailings from the mini pilot plant.

Figure 13-9: Conventional flotation flowsheet



Source: SGS 2013

Metallurgical performance conclusions from the flowsheet development testing included the following:

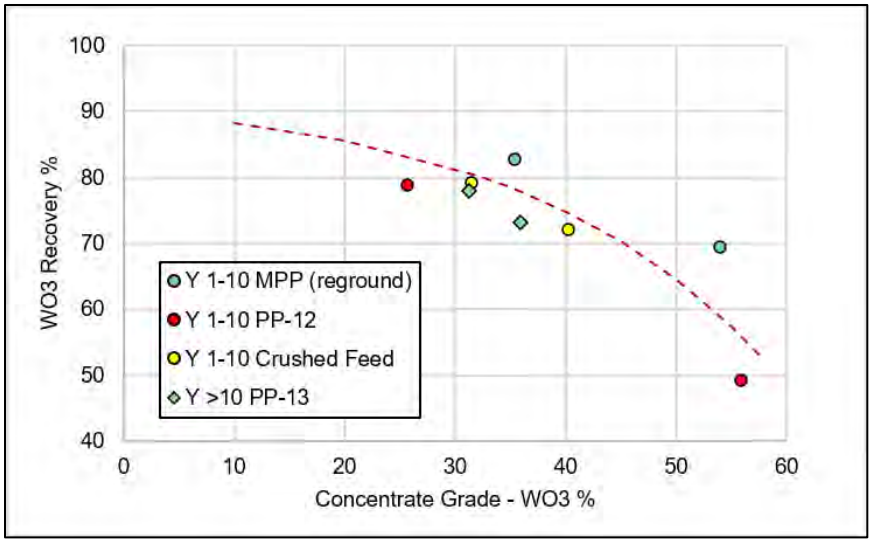
- The circuit performance was sensitive to collector dosages. Insufficient dosages improved the selectivity, but recoveries decreased.
- Losses were deemed to be greater in the coarser fractions, which required higher collector dosages. Tungsten losses due to lower collector dosages applied in conditioning could not be compensated by applying additional collector to later stages. Finer grinding to 100% passing 212 µm versus the original -300 µm target was adopted to mitigate losses of coarse scheelite.
- Rougher circuit recoveries generally varied from 75 to 88% to rougher concentrates grading 2 to 4% WO_3.

- Batch cleaner results varied considerably. A group of 4 tests measured WO_3 recoveries ranging from 44 to 53%, at concentrate grades of 40 to 57% WO_3. These appeared to be achieved using overall lower collector and depressant dosages. Another pair of tests which applied only a single cleaning stage averaged 64% recovery to an average concentrate grade of 22%. The remainder of the tests returned inferior performance.
- Regrinding was not required and generally had a negative effect on metallurgical performance.

A set of pilot plant runs were conducted at a feed rate of 150 kg/h to generate WO_3 rougher concentrate for downstream hydrometallurgical testing and tailings for characterization. The circuit included grinding in closed circuit with a 212 µm screen, molybdenum and sulphide rougher flotation, WO_3 conditioning and rougher flotation. Two bulk composites were processed: 13 tonnes of Year >10 material and 8 tonnes of Year 1-10 material. The two composites had very similar feed grades. The flotation feed sizing averaged 80% passing 112 µm over the 12 run days. Molybdenum and sulphur recoveries to their respective rougher concentrates were similar to bench scale testing. Tungsten recovery however was lower than bench scale rougher performance and averaged 68% to rougher concentrates averaging 7.2%, so the circuit was operating more selectively.

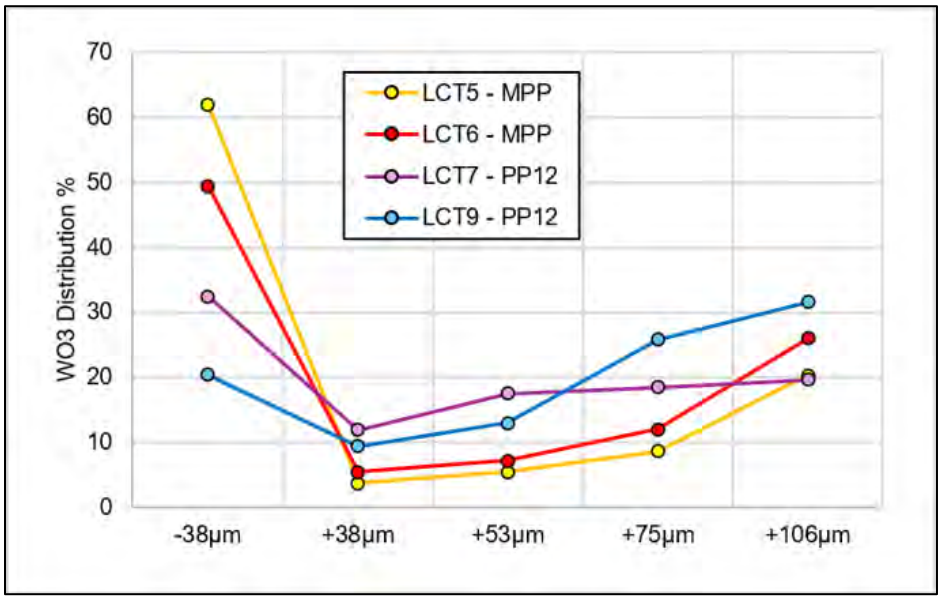
A series of locked cycle flotation tests were conducted on master composite materials to evaluate tungsten flotation performance in closed circuit. Six of the tests were completed on BSF tailings from either the mini pilot plant or the larger pilot run, two were completed on crushed feed samples for comparison. The mini pilot plant tailings were reground to 100% passing 212 µm, resulting in a P_{80} value of 120 µm. The larger pilot plant tailings were not further ground as they already had a size distribution of 80% passing 110 µm. The crushed feeds were stage ground to 100% passing 212 µm, resulting in a flotation feed sizing of approximately 120 µm P_{80}. The results are presented graphically in Figure 13-10. The results suggest that a concentrate grading 32% WO_3 could be generated at a tungsten recovery of approximately 80% using this flowsheet. On average, about 80% of the tungsten losses were in the scavenger tailings. Assays by size on the scavenger tailings from selected tests are presented graphically in Figure 13-11, suggesting that coarse losses were less of an issue at these primary grind sizes.

Figure 13-10: Locked cycle WO_3 flotation results – master composites SGS 2013



Source: Ausenco 2026

Figure 13-11: Locked cycle scavenger tailings WO_3 distribution – master composites SGS 2013



Source: Ausenco 2026

A series of rougher kinetic tests were conducted on the variability samples assembled for this program, which included three lithological composites and three production period composites. The Year 1-10 and Year >10 master composites were also tested for comparison. Each 5 kg feed charge was stage ground to 100% passing 212 µm and the developed rougher flotation conditions applied. The rougher WO_3 recoveries were lower on average than the batch Y 1-10 master composite tests, ranging from 66 to 81% and averaged 73.8%. Rougher recoveries varied with mass recovery. There was no clear relationship between feed grades or lithology type and WO_3 recovery, however the DOM-I material measured the highest WO_3 recovery of the dataset.

13.9.2 Hot Conditioning Scheelite Flotation

In 2014, an alternate processing strategy was introduced to the project, known as the Modified Petrov Method, which can improve the selectivity against calcite and other gangue minerals. The process step is referred to as "hot conditioning" and involves heating the flotation slurry to 90°C for approximately one hour with a high dosage of sodium silicate depressant. Moderate dosages of sodium hydroxide and sodium sulphide have also been added in the conditioning step. Applied to a thickened intermediate flotation concentrate, for heat and reagent efficiency due to the reduced volume, the conditioning step effectively removes the fatty acid collector from the gangue mineral surfaces. The conditioned pulp is then cooled with dilution water and advanced to cleaner flotation stages where the scheelite is recovered with significantly improved selectivity. The testing completed for the Sisson project demonstrates that this is a suitable upgrading process, but it can be sensitive to collector dosages applied to the flotation stages after hot conditioning. The testing also required large feed masses, as the concentration ratio increases significantly when targeting WO_3 concentrate grades of 50 to 60% in the final cleaner stages. Limitations in mineralized froth mass to support high SG scheelite grains can compromise performance despite having adequate chemical conditions.

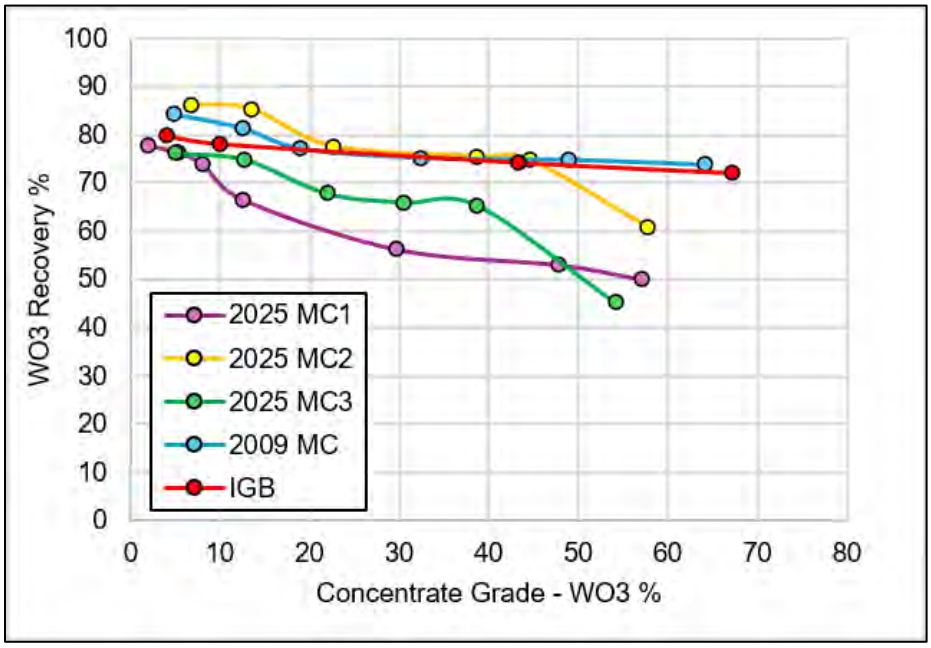
Remaining composite mass assembled in the 2013 test program were retrieved from storage to use in the 2014 program. Over the course of the test program, the remaining Y 1-10 composite was consumed and a

portion of the testing was conducted using BSF tailings from the previous pilot processing. After demonstrating improved metallurgical performance using this flowsheet in 2014 and additional testing in 2017, the project was paused due to market conditions.

In 2025, ½ drill core samples were retrieved to assemble variability composites for metallurgical testing in conjunction with an ore sorting test program. The objective of this testing was to demonstrate the updated hot conditioning flotation process flowsheet on more discrete lithological variability samples. The testing was to commence with a master composite assembled from portions of the variability samples. There were some initial challenges replicating the performance of the 2014 / 2017 testing and the master composite inventory was consumed. An alternate master composite was assembled from Phase 2 ore sorting rejects and reconstituted to represent the feed, this was also consumed during testing. Remaining crushed feed samples from earlier test programs were retrieved from storage and portions of the 2009 master composite were used for testing. Once target closed circuit metallurgical performance was achieved on this composite, the variability samples were also tested in both open and closed circuit.

Examples of the upgrading potential of the hot conditioning step are presented graphically in Figure 13-12. These results are from open circuit tests conducted in the 2025 / 2026 test program. The first three points are rougher, 1st and 2nd cleaner points in the ambient circuit, except the 2025 MC1 example that had 3 ambient cleaner stages and the last 3 points of each curve are the rougher, 1st and 2nd cleaner points after the hot conditioning step. The IGB test was also different in that it required only a single cleaner stage in both the ambient and hot conditioned circuits. Note that the ambient concentrate recovery points would be higher in closed circuit.

Figure 13-12: Open circuit hot conditioned cleaning results – SGS 2026

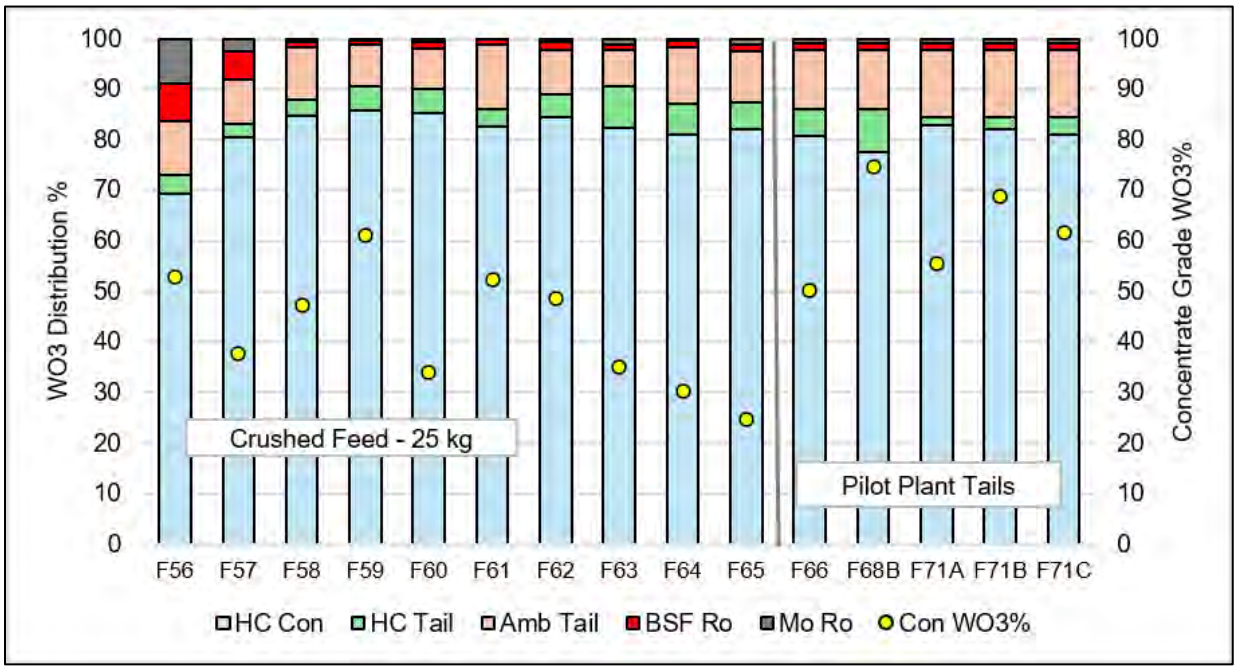


Source: Ausenco 2026

The results show that there is a significant upgrading post hot conditioning, often with high stage recoveries. The recovery decrease across the 2nd hot cleaner stage in the MC2 and MC3 tests is attributed to insufficient collector dosages as the flotation pulp becomes more dilute.

To demonstrate the full circuit in closed circuit, essentially two locked cycle tests are completed in series. The first LCT generates an ambient concentrate, that intermediate grade concentrate becomes feed to hot conditioning. The second LCT is the cleaner upgrading post hot conditioning, in which the conditioned pulp is split into a number of cycle feed charges. Results of this testing using the optimized conditions in 2014 and 2017 are presented graphically in Figure 13-13.

Figure 13-13: Hot conditioning WO_3 distribution in locked cycle tests – SGS 2014/2017

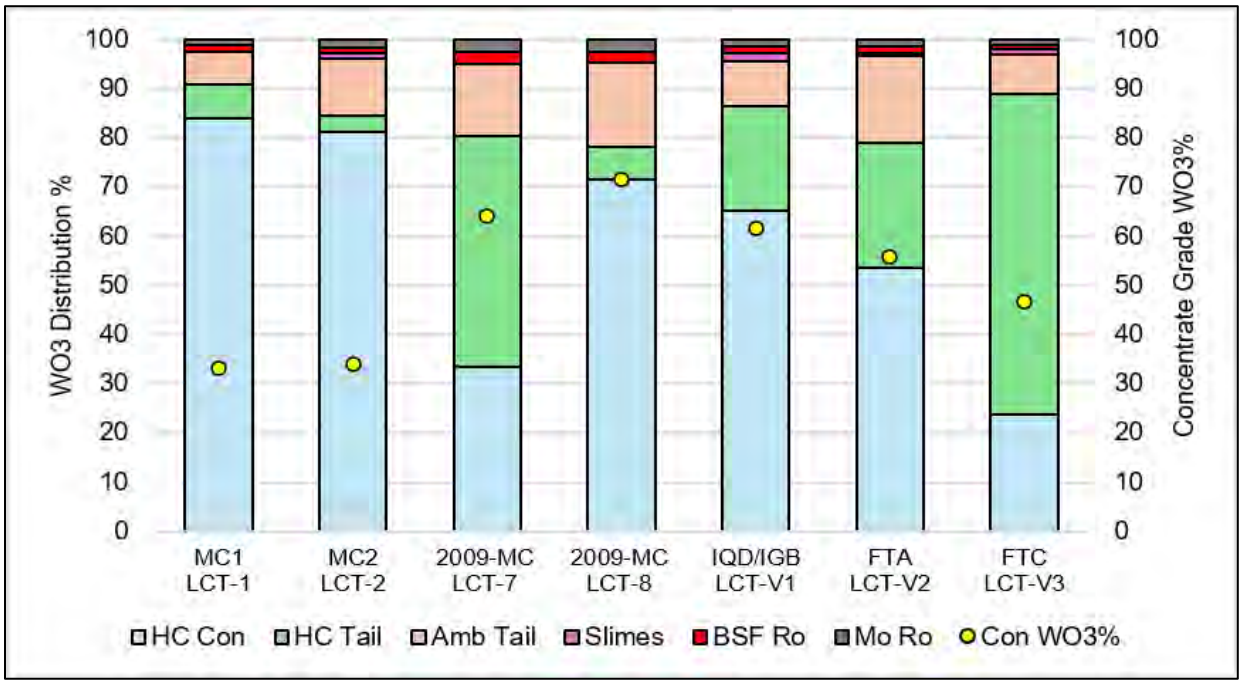


Source: Ausenco 2026

The results indicate that ambient circuit recoveries (blue plus green bars) were quite consistent, only the first two tests in this set had high losses to the Mo and BSF circuits due to excessive mass recoveries. The WO_3 recoveries across the hot conditioned cleaner circuit were also relatively consistent, averaging about 95% based on the calculated steady state metallurgical balances. The final concentrate grades varied considerably despite applying very similar conditions.

Results from locked cycle tests conducted in the 2025/2026 test program are presented in Figure 13-14. The results from both programs are tabulated in Table 13-17.

Figure 13-14: Hot conditioning WO_3 distribution in locked cycle tests – SGS 2025/2026



Source: Ausenco 2026

A range of metallurgical performance is demonstrated in the recent results and are discussed in the following comments:

- The initial two tests were conducted on similar feed material, but the composites were assembled at different points in the program from variability sample portions. Higher dosages of collector were applied in these tests which was believed to compromise selectivity against gangue minerals, resulting in low final concentrate grades.
- The initial test on the 2009-MC used a very low dosage of collector in the hot conditioned cleaner circuit. As well, the 1st cleaner tailings in this circuit was an exit stream. In the repeat test, slightly higher collector dosage was applied and the 1st cleaner tails was returned to the hot conditioned rougher feed. Both changes contributed to a significant increase in circuit recovery.
- The locked cycle tests conducted on the three lithological used lower feed masses due to sample mass constraints. The ambient circuit recoveries were consistent with other testing, however the WO_3 recoveries across the hot conditioned cleaning step were low compared to the associated open circuit tests. The three hot conditioned ambient concentrates were divided into 4 portions to carry out the post hot conditioning locked cycle test protocol. The division of ambient concentrate mass resulted in per cycle feed masses that were too low for effective froth loading, observed during the lock cycle testing, which compromised recoveries.

Table 13-17: Locked cycle test data – hot conditioned circuit

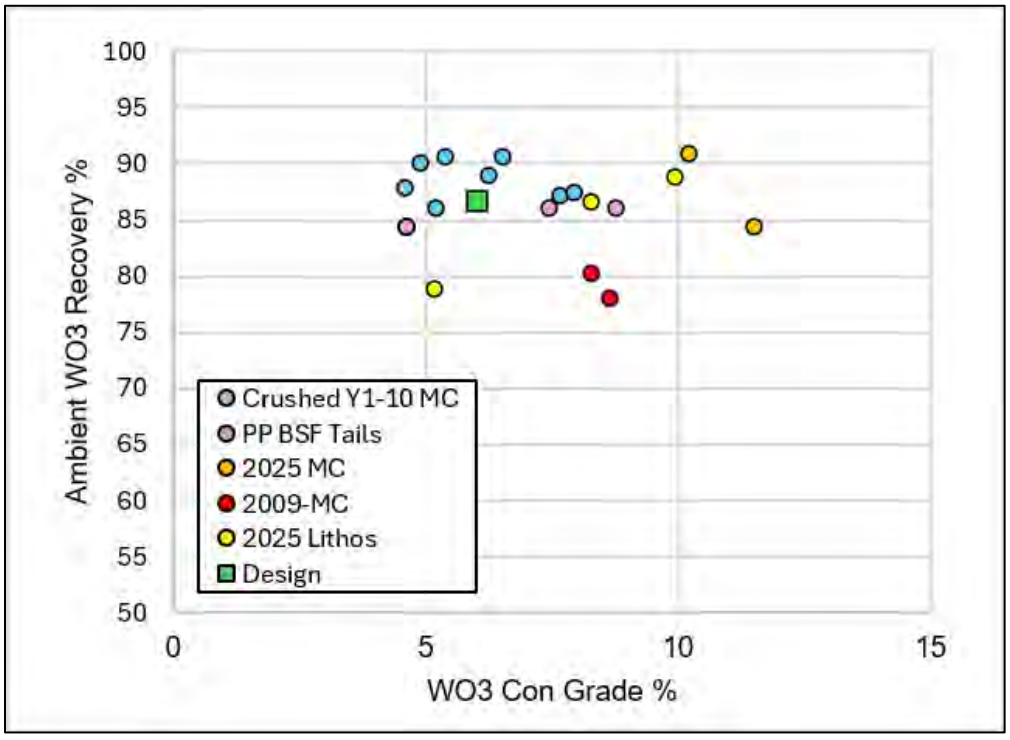
Year	Feed	Test	Feed	Prim. Grind Size		Con Grade	WO_3 Distribution - Total %					
			kg	P_{100} µm	P_{80} µm	WO_3 %	HC Con	HC Tail	Amb Tail	Slimes	BSF Ro	Mo Ro
2014	Crushed Y1-10 MC	F56	5x5	212	140	52.8	69.2	3.9	10.7	-	7.3	8.9
		F57	5x5	212	140	37.6	80.6	2.5	8.7	-	5.6	2.6
		F58	5x5	212	139	47.3	84.7	3.2	10.5	-	0.9	0.8
		F59	5x5	212	143	61.2	85.8	4.8	8.2	-	0.7	0.5
		F60	5x5	212	136	34.1	85.2	4.8	8.1	-	1.2	0.7
		F61	5x5	234	161	52.4	82.7	3.4	12.7	-	1.0	0.2
2017		F62	5x5	212	134	48.6	84.4	4.5	8.8	-	1.6	0.7
		F63	5x5	212	131	35	82.3	8.3	7.1	-	1.1	1.2
		F64	5x5	212	130	30.2	81.0	6.1	11.1	-	1.4	0.4
		F65	5x5	212	130	24.8	82.2	5.2	10.1	-	1.3	1.2
	PP BSF Tails	F66	5x12	212	119	50.2	80.6	5.4	11.7	-	1.2	1.0
		F68B	3x12	180	103	74.7	77.5	8.5	11.7	-	1.2	1.0
		F71A	6x12	212	103	55.4	82.9	1.4	13.4	-	1.2	1.0
		F71B				68.7	82.1	2.3	13.4	-	1.2	1.0
		F71C				61.6	81.1	3.3	13.4	-	1.2	1.0
2025	MC1	LCT-1	8x10	212	140	33.2	83.9	7.0	6.5	-	1.3	1.3
	MC2	LCT-2	8x10	212	140	33.8	81.2	3.2	11.6	1.1	1.3	1.6
2026	2009-MC	LCT-7	4x10	212	140	63.9	33.5	46.8	14.5	0.2	2.4	2.6
	2009-MC	LCT-8	6x10	212	140	71.5	71.5	6.6	17.1	0.1	2.1	2.6
	IQD/IGB	LCT-V1	3x10	212	140	61.5	65.1	21.5	8.9	1.9	1.3	1.4
	FTA	LCT-V2	3x10	212	140	55.8	53.6	25.3	17.8	0.5	1.4	1.5
	FTC	LCT-V3	3x10	212	140	46.7	23.7	65.1	8.1	1.0	1.0	1.1

Note that a settling step was applied in the recent testing to condition the WO_3 flotation feed at a higher density. Fine solids captured during decanting are reported as slimes.

The locked cycle results in Table 13-15 were used to estimate tungsten recoveries from this two-stage flotation process. Tests F56 and 57 were excluded due to elevated losses to the Mo and BSF circuits. The assessment is a result of two estimates that are shown in the following graphs and comments:

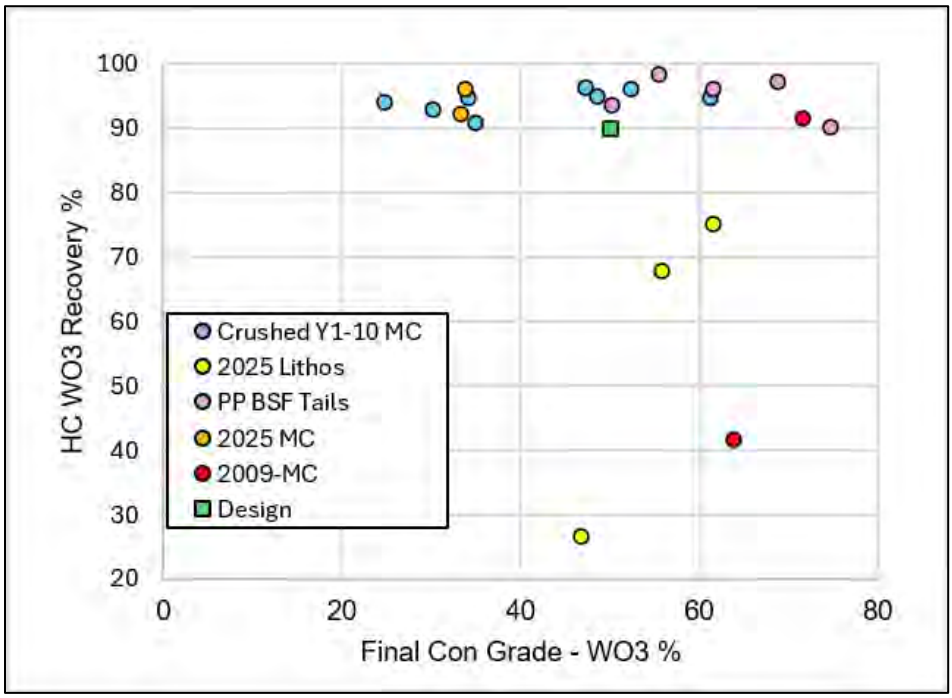
- It is estimated that 86.7% of the feed tungsten will be recovered to the ambient flotation cleaner concentrate at a grade of 6% WO_3. This circuit is expected to recirculate scavenger concentrate and cleaner tailings to the rougher circuit feed to maximize recovery and utilize one or two stages of cleaning. This grade-recovery point, shown as a green square, is compared to the locked cycle data in Figure 13-15. There was no relationship found between WO_3 feed grade and ambient circuit recovery in this data set.
- It is estimated that cleaner flotation post hot conditioning will achieve a circuit recovery of 90% and produce a concentrate grading 50% WO_3. This circuit performance, shown as a green square, is compared to locked cycle testing results in Figure 13-16.
- The resulting net tungsten recovery of 78% at a concentrate grade of 50% WO_3 is compared to locked cycle data in Figure 13-17.

Figure 13-15: Ambient circuit WO_3 recoveries in locked cycle tests



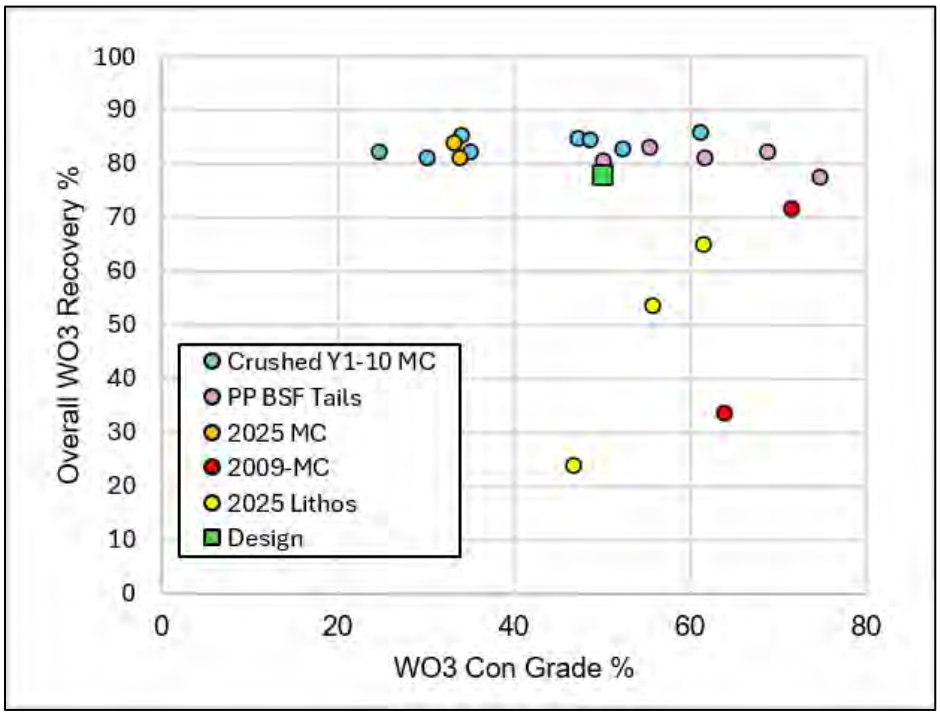
Source: Ausenco 2026

Figure 13-16: Hot conditioned cleaner circuit WO_3 stage recoveries in locked cycle tests



Source: Ausenco 2026

Figure 13-17: Total circuit WO_3 recoveries in locked cycle tests



Source: Ausenco 2026

13.10 Hydrometallurgical Testing – APT Production

A hydrometallurgical test program was completed in 2014 that evaluated the various steps involved with processing WO_3 flotation concentrates to produce commercial grade high purity ammonium paratungstate crystals (APT). Three concentrates were provided from the 2013 flotation program that varied in WO_3 content from 29.9 to 40.2%. The testing commenced with high pressure leaching in a laboratory autoclave, in which 96 to 98% of the tungsten was extracted. This was followed by impurity removal steps, solvent extraction testing, and ammonium paratungstate crystallization. The final APT product was determined by an umpire lab analysis to be of a high quality, measured at 89.1% WO_3.

13.11 Deleterious elements

There are no known levels of deleterious elements in the molybdenum final concentrates that would be of concern for marketing purposes.

Tungsten concentrates are sold in a limited market, so deleterious elements are buyer specific and are not available as might be with other mineral concentrates. Generally, a tungsten concentrate grading over 50% is WO_3 is marketable, regardless of the composition of the diluents. Sulphur is understood as being problematic for downstream hydrometallurgical processing, hence a target of lower than 6%. Sulphur values measured in hot conditioned concentrates from locked cycle tests on master composites ranged from 0.2 to 1.5%. Sulphur values on three similarly tested lithological composites measured 1.7, 5.1 and 8.5%, however the two high values were associated with unoptimized scheelite collector dosages. The expected higher recoveries of tungsten on these two materials once optimized would result in lower sulphur levels. Sulphur recovery was generally well rejected across the tungsten flotation circuits, however not always consistently. It is likely that concentrates produced from the few materials in which tarnished iron sulphides were not adequately recovered in the molybdenum and BSF circuit and advanced through the tungsten circuit could be blended to mitigate any marketing issues.

13.12 Recovery estimates

Molybdenum recovery to final molybdenum concentrate is estimated at 82%, based on the rougher circuit average recoveries and cleaner circuit recoveries from locked cycle tests. Molybdenum recovery is estimated as constant across the deposit, as the available data does not show a relationship to feed grade.

Tungsten recovery to a final flotation concentrate is estimated at 78%, as per the discussion in Section 13.9.2. Since most of the testing was completed on master composites of similar feed grade, there is not enough data at this time to develop a model that relates tungsten recovery to feed grades.

13.13 Comments on mineral processing and metallurgical testing

A considerable amount of metallurgical testing has been completed on representative samples from the Sisson project. The quantity of sample tested was in excess of 34 tonnes and has been a formidable undertaking. Despite the challenges of processing a feed material with low WO_3 contents, a number of potential processing strategies have been successfully demonstrated. The all-flotation processing strategy selected is effective at a primary grind size of 140 µm P_{80} and the hot conditioning component allows for the generation of a highly marketable tungsten concentrate. Further testing and economic evaluation should be conducted to develop a better understanding of geo-metallurgical relationships across the distinct lithologies that form the deposit as well as on alternate flowsheets that may include pre-concentration and gravity concentration.

14 Mineral Resource Estimates

14.1 Introduction

Northcliff prepared a block model (the 2012 model) and MRE for the Project in early 2012. This estimate was audited by SLR (as RPA), described in RPA (2012) and Samuel Engineering (2013), and used to prepare a MRE with an effective date of December 31, 2012 (the 2012 MRE).

The MRE disclosed in this Technical Report was prepared by SLR by running a new pit optimization on an expanded version of the 2012 model. Current metal prices, mining costs, and updated processing parameters were used as input to a new Net Smelter Return (NSR) equation and cut-off value determination.

WO_3, molybdenum (Mo), and density values from the 2012 model are unchanged in this update. In the 2012 model, values for bulk density, WO_3, and Mo were interpolated into the blocks using inverse distance squared (ID^2) weighting constrained with wireframe models of the principal geological domains.

Drill hole information collected in 2013, 2014, and 2023, after the 2012 model was prepared, was reviewed against estimated block values and found to show general accord. Most are beyond the deposit extents, and the results are not incorporated. There are only two new holes inside the Mineral Resource volume. The SLR QP checked them against the extant block grades and is of the opinion that they would not materially affect the block model grades. The QP recommends including them in the next update.

NSR calculation revisions align with the processing parameters derived during the development of this Technical Report and cost considerations derived from the 'No APT' WO_3 concentrate scenario developed by Moose Mountain Technical Services (MMTS).

The MRE was completed using Maptek Vulcan (Vulcan) software using a conventional approach including 3D solid modelling and block modelling.

This most recently revised estimate is current to December 31, 2025, and is summarized in Table 14-1. CIM Definition Standards for Mineral Resources and Mineral Reserves definitions (CIM, 2014) were used for Mineral Resource classification. Mineral resources that are not mineral reserves do not have demonstrated economic value.

Table 14-1: Summary of Mineral Resources - effective date: December 31, 2025

Category	Tonnage	Grade WO_3	Grade Mo	Contained WO_3	Contained Mo	Grade WO_3EQ	Average NSR
	(Mt)	(%)	(%)	(M MTU)	(M lb)	(%)	(C$/t)
Measured	117.9	0.068	0.022	8.0	57.2	0.098	39.13
Indicated	303.7	0.061	0.019	18.5	127.2	0.088	34.65
Total Measured and Indicated	**421.6**	**0.063**	**0.020**	**26.6**	**185.9**	**0.091**	**35.90**
Inferred	182.9	0.048	0.020	8.8	80.6	0.078	28.59

Notes:
1. CIM (2014) definitions were followed for Mineral Resources.
2. WO3Eq = WO_3% + ((Mo% x NSP_Mo x REC_Mo)/(NSP_WO_3 x REC_WO_3)).
3. Mineral Resources are estimated at an NSR cut-off grade of C$12.02/t.
4. Mineral Resources are estimated using a long-term WO3 price of US$500/MTU, and a long-term Mo price of US$23/lb.
5. Mineral Resources are constrained within a Whittle shell, generated using the NSR cut-off value.
6. Bulk density is interpolated into blocks where density sample information is present and set to 2.77 t/m^3 outside of the estimated blocks.
7. Mineral Resources are inclusive of Mineral Reserves.
8. Mineral Resources that are not Mineral Reserves do not have demonstrated economic viability.
9. To meet reasonable prospects for eventual economic extraction, Mineral Resources assume metallurgical recoveries and processing costs to be the same as for Mineral Reserves but use higher long term metal prices to calculate NSR.
10. Numbers may not add due to rounding.

The QP is not aware of any environmental, permitting, legal, title, taxation, socio-economic, marketing, political, or other relevant factors that could materially affect the MRE.

14.2 Resource database

The MRE for the Project is based primarily on information from 304 surface diamond drill holes collected in 2012 or earlier, supplemented by historical surface mapping and geophysics supporting interpretation. Most of the holes were collared at a dip shallower than -75°. Hole lengths vary widely but typically range from 100 m to 350 m. An additional two holes have been drilled within the Mineral Resource area subsequently and generally confirm block model results.

Drilling on Sisson covers an approximate area of 2,300 m (north-south) by 1,500 m (east-west), with the density of drilling decreasing toward the north and northwest. Results from 304 drill holes within the block model boundaries support the MRE. Further detail on drilling can be found in Section 10, Drilling.

SLR reviewed the drill hole data collected after the 2012 model was prepared and concluded that the results would not materially affect the Mineral Resource. The new drilling mainly occurs outside of the Mineral Resource shell volume, and so is unlikely to affect the grade, classification or density of the blocks.

14.3 Geological interpretation

14.3.1 Lithology

Drill data were partitioned into 13 separate lithology wireframes corresponding to the eight rock types and their respective sub-domains, discussed in Section 7. Descriptive statistics were generated based on these 13 lithology wireframes and the results suggest that lithology does not have strong control over the

mineralization distribution. As a result, the lithology model was not used for the purpose of this estimation. The lithology model for the deposit is shown in Figure 14-1.

14.3.2 Mineralization

After visual data inspection and statistical analysis, the grade domain approach was chosen for partitioning of WO_3 data. Minimum grade parameters for higher grade WO_3 domains were not rigidly fixed but effectively defined zones with acceptable continuity that grade above 0.035% WO_3 over ten or more metres. There were two higher grade zones modelled for Zone III and a single high-grade zone modelled for the Ellipse Zone. Figure 14-2 shows a plan map of the domains at surface.

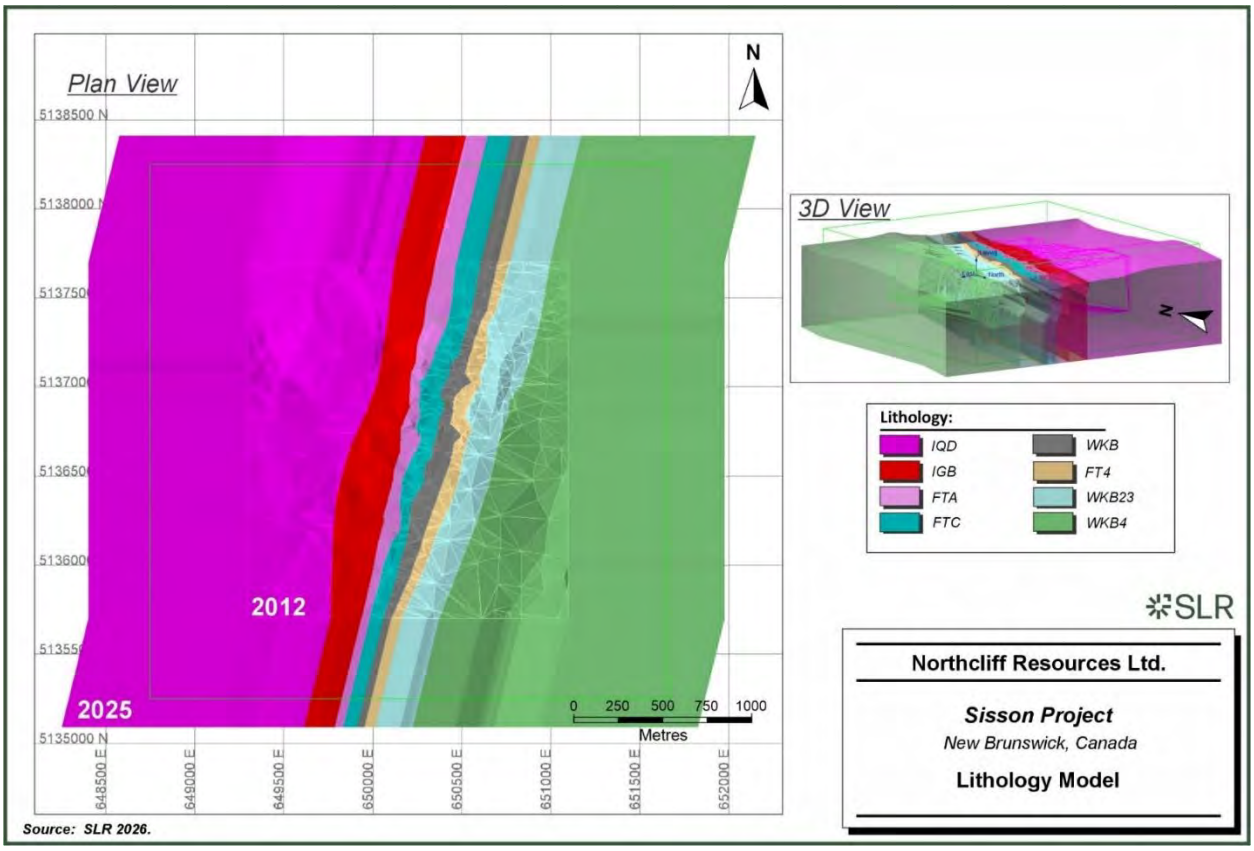
After a visual inspection of molybdenum data, it was decided to not use the grade domaining technique in this case. Molybdenum data was partitioned into the Zone III and Ellipse domains only.

SLR expanded the block model extents without changing the block grade contents, so that Whittle could run without reaching the lateral and depth limits of the 2012 block model. SLR projected the boundaries of the lithological domains laterally by approximately 600 m to 900 m and deeper by approximately 325 m. Mineralization domains were unchanged. Since the block grades themselves did not change, the purpose of this expansion was mainly to fill new blocks with density values so that they could produce dilution tonnages during pit shell optimization.

14.3.3 Topography

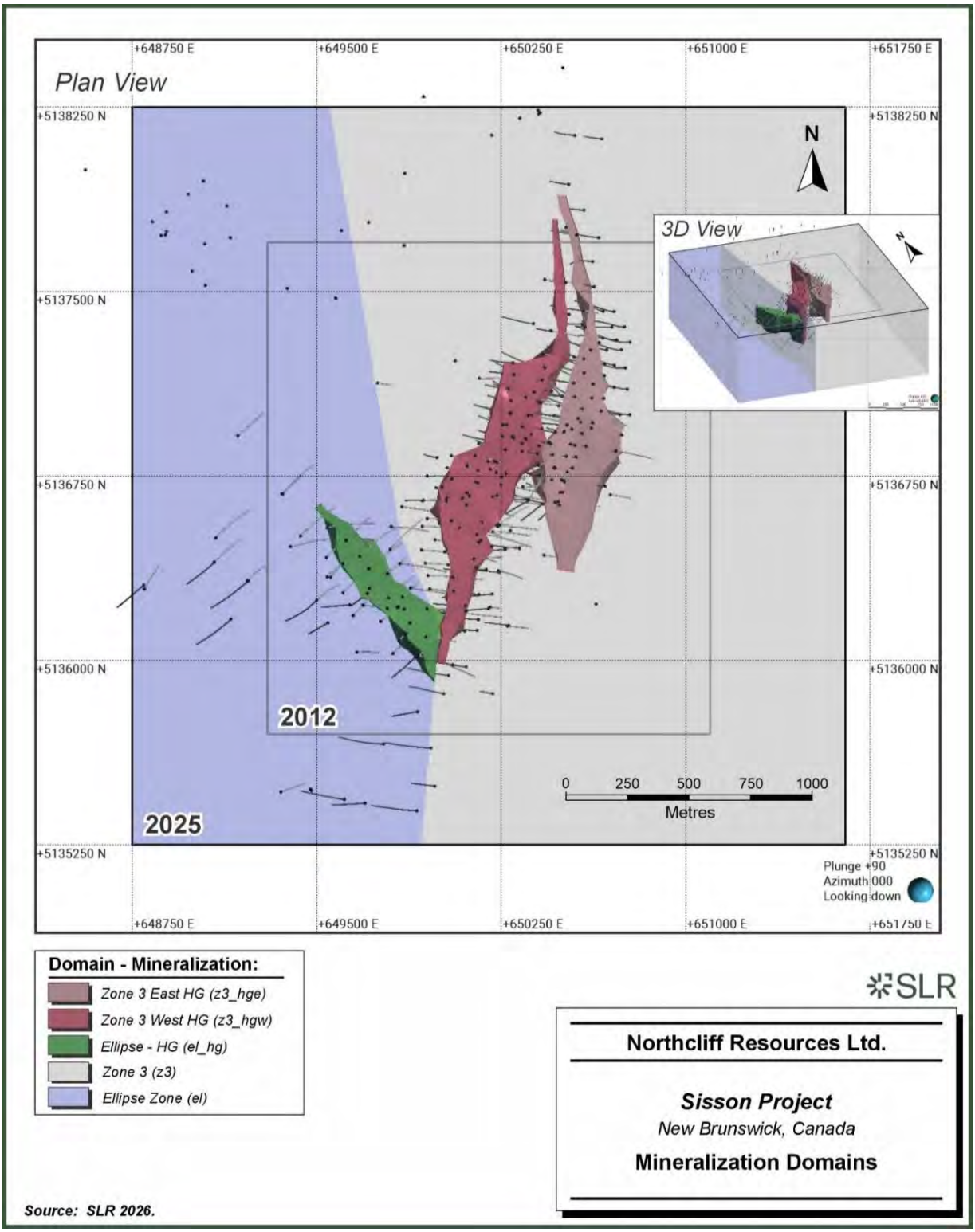
For 2025, SLR expanded the topographic surface to cover the expanded block model extents (Figure 14-1). Using QGIS and Leapfrog, SLR merged the surrounding Shuttle Radar Topography Mission (SRTM) surface data with the original Light Detection and Ranging (LiDAR) topography used in the 2012 model.

Figure 14-1: Lithology model



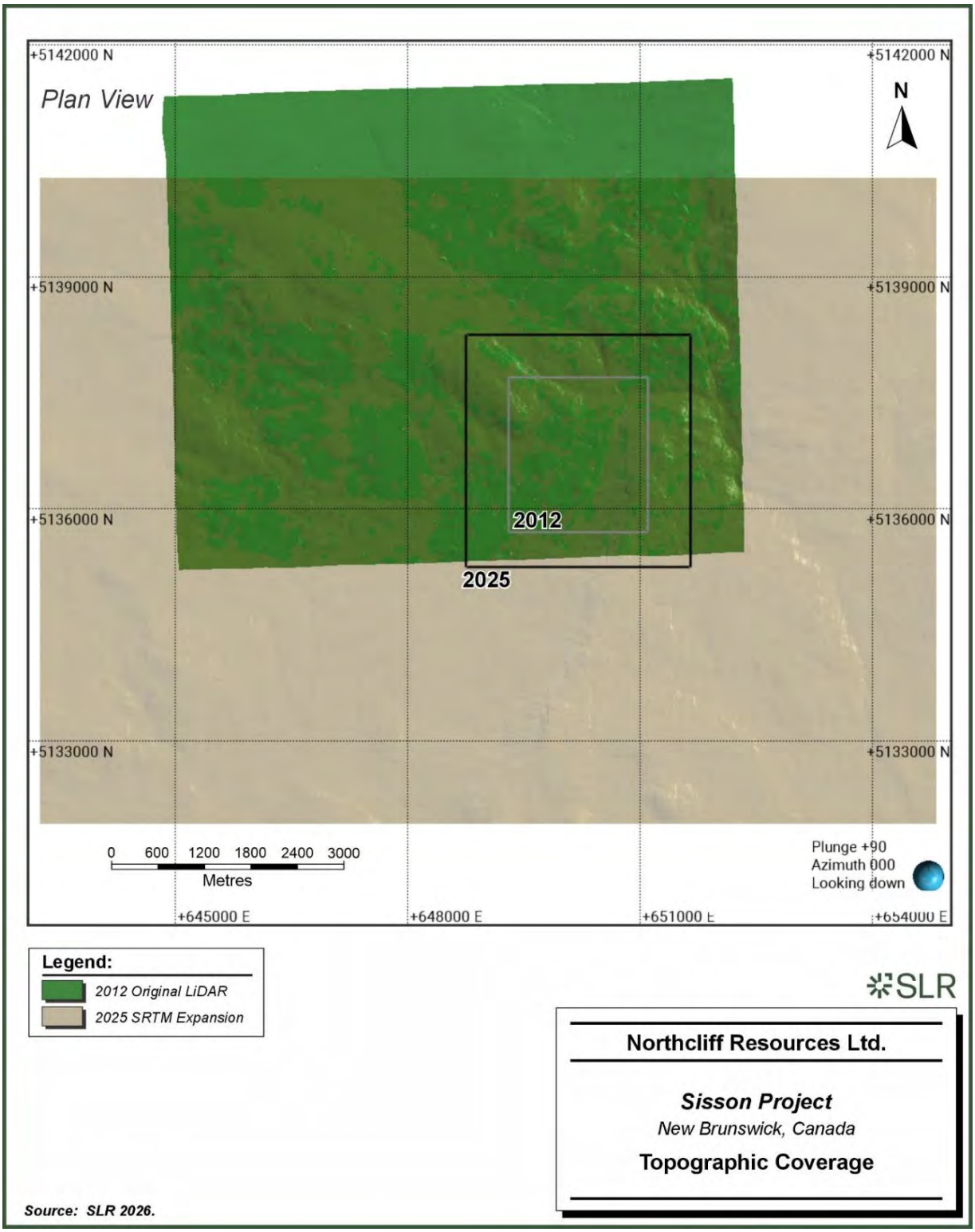
Source: SLR, 2026

Figure 14-2: Mineralization domains



Source: SLR, 2026

Figure 14-3: Topographic coverage



Source: SLR, 2026

14.4 Resource assays

The Mineral Resource assay database contains 31,357 assay intervals, and of these, 29,897 have non-zero values for WO_3 and 31,336 have non-zero values for molybdenum. Most sampled intervals are 1.5 m in length. Statistical summaries of the grade domain assays are provided in Table 14-2 and Table 14-3.

Table 14-2: WO_3 sample statistics by estimation domain

Statistic	All Domains	Ellipse HG	Ellipse LG	Z3 HG East	Z3 HG West	Z3 LG
Sample Count	31,357	2,858	2,072	5,730	11,064	9,633
Minimum	0	0	0	0.001	0	0
Maximum	4.439	2.383	0.676	3.493	3.354	4.439
Mean	0.062	0.072	0.033	0.09	0.082	0.027
Standard Deviation	0.127	0.113	0.048	0.161	0.148	0.072
CV	2.039	1.574	1.467	1.797	1.802	2.72
Variance	0.016	0.013	0.002	0.026	0.022	0.005

Table 14-3: Mo sample statistics by estimation domain

Statistic	All Domains	Ellipse Zone	Zone III
Sample Count	31,336	4,930	26,406
Minimum	0	0	0
Maximum	19.8	2.73	19.8
Mean	0.020	0.019	0.021
Standard Deviation	0.131	0.072	0.140
CV	6.456	3.767	6.791
Variance	0.017	0.005	0.020

14.5 Treatment of high-grade assays

The sample grade distributions for WO_3 and molybdenum are positively skewed, in some cases resembling log-normal distributions. For skewed distributions of this type, the highest-grade samples can have an inordinately large effect on the average grades, which can result in biased grade interpolations. In order to reduce the influence of these high samples, a cap was applied prior to compositing.

14.5.1 Capping levels

For the estimation, a series of log-normal probability plots and histograms for tungsten and molybdenum were prepared from data within the interpreted zones to examine the distribution of the assay data. Interpretation of the log-normal probability plots and histograms has yielded capping levels of 1.1% WO_3 and 0.65% Mo for both the Ellipse and Zone III domains.

In total, 66 WO_3 and 68 molybdenum assay intervals were capped. These intervals represent approximately 0.2% of the total number of assays. The net impact of the capping was to reduce the average WO_3 and molybdenum assay grades by a negligible amount. Samples were capped prior to compositing.

14.5.2 High-grade restriction

No high-grade restriction was performed during grade estimation.

14.6 Compositing

For all Sisson domains, assay intervals were composited on the basis of their respective grade domains. Samples were composited in downhole intervals of 4 m, starting at the wireframe pierce-point for each zone, continuing to the point at which the hole exited the zone. Inevitably, the final composite in each zone will be shorter than the fixed composite length unless the zone intercept is an exact multiple of the selected length.

Since the short composites were relatively few in number, their impact was considered to be insignificant, and they were left in the database. The 4 m composite length was deemed most suitable, based on the prospective equipment size for potential future mining. Grade domain solids were used to flag composites, based on their centroids (Table 14-4). A summary of composite statistics is provided in Table 14-5 and Table 14-6.

In the QP's opinion, insofar as the most common sample lengths are 1.5 m and 3.0 m, a more appropriate composite length would have been 3.0 m. This would have reduced the number of composites that straddle the assay intervals. This, however, is not considered to be a serious concern.

Table 14-4: Grade domain composite flags

Zone/Domain	Variable	Value
Zone III HG East WO_3	Bound	z3_hge
Zone III HG West WO_3	Bound	z3_hgw
Zone III LG WO_3	Bound	Z3_lg
Ellipse HG Mo	Bound	el_hg
Ellipse LG Mo	Bound	el_lg

Table 14-5: WO_3 4 m composite statistics by estimation domain

Statistic	All Domains	Ellipse HG	Ellipse LG	Z3 HG East	Z3 HG West	Z3 LG
Samples	13,356	1,274	788	2,427	4,468	4,399
Minimum	0	0.001	0.001	0.001	0	0
Maximum	0.986	0.49	0.262	0.916	0.986	0.564
Mean	0.059	0.07	0.035	0.086	0.079	0.025
Standard Deviation	0.074	0.062	0.032	0.091	0.085	0.035
CV	1.248	0.887	0.932	1.065	1.074	1.375
Variance	0.005	0.004	0.001	0.008	0.007	0.001

Table 14-6: Mo 4 m composite statistics by estimation domain

Statistic	All Domains	Ellipse Zone	Zone III
Samples	13,349	2,062	11,287
Minimum	0	0	0
Maximum	0.62	0.491	0.62
Mean	0.018	0.019	0.018
Standard deviation	0.035	0.036	0.035
CV	1.952	1.894	1.963
Variance	0.001	0.001	0.001

14.7 Trend analysis

Variograms were developed for WO_3 in each of the grade domains and for molybdenum in each of the two zones (Zone III and Ellipse). For all metals a two-structure, spherical model was fitted to the experimental variograms. This analysis was carried out by Northcliff using Snowden Supervisor software.

14.7.1 Variography

The drill spacing throughout the deposit is approximately 50 m along strike in the north-south direction by 100 m along sections in the east-west direction. This drill spacing generally yielded the best set of variograms using lag distances between 50 m and 90 m.

14.8 Search strategy and grade interpolation parameters

Grade estimation was performed using the ID^2 interpolation method. Search region geometries were based on variography analysis results (a factor of 1.5 was applied to the ranges). For WO_3 grade estimation, higher grade domain solids were employed as hard boundaries (i.e., only samples from this domain were used to estimate blocks for this domain). Lower grade domains were estimated using soft boundaries – samples from both lower grade domains were used to estimate blocks in either of these domains. Two estimation passes were performed, a minimum of three and maximum of 12 samples from a minimum of two drill holes were required before a block would be estimated in the first pass. No minimum drill hole limit was applied in the second pass. For molybdenum grade estimation, blocks in both zones were estimated using all composites (soft boundaries).

Other estimation parameters are summarized in Table 14-7.

Table 14-7: Inverse distance estimation parameters

Estimator	Zone	Estimation ID	Sample Selection	Block Selection	Major Axis	Semi Major Axis	Minor Axis	Bearing	Plunge	Dip
WO_3 (ID^2)	Zone III HG West	z3w1_1, z3w1_2	Bound=z3_hgw	wo3_zone eqs "z3_hgw" and yworld le 5137300	165	123	60	20	20	-90
	Zone III HG West (North of 5137300)	z3w1n1, z3w1n2	Bound=z3_hgw	wo3_zone eqs "z3_hgw" and yworld gt 5137300	165	123	60	350	20	-90
	Zone III HG East	z3w1_1, z3w1_2	Bound=z3_hge	wo3_zone eqs "z3_hge" and yworld le 5137300	180	144	48	20	-30	-90
	Zone III HG East (North of 5137300)	z3w1n1, z3w1n2	Bound=z3_hge	wo3_zone eqs "z3_hge" and yworld gt 5137300	180	144	48	350	-30	-90
	Zone III LG	z3w0_1, z3w0_2	Bound="z3_lg" or "el_lg"	zone eqs "z3" and wo3pct_id2 lt 0	273	255	180	20	0	-90
	Ellipse Zone HG	elw1_1, elw1_2	Bound=el_hg	zone eqs "el_hg"	225	156	138	140	0	-90
	Ellipse Zone LG	elw0_1, elw0_2	Bound="z3_lg" or "el_lg"	wo3pct_id2 lt 0	234	225	420	140	0	-90
Mo (ID^2)	Zone III	mo_z32	MoPCT > 0	zone eqs "z3"	500	280	300	35	0	-90
	Ellipse Zone	mo_el2	MoPCT > 0	zone eqs "el"	460	180	70	140	0	-90
SG	Zone III + Ellipse Zone	sg	SG > 0	material eq 1	260	300	100	15.8	39.8	-83.5

14.9 Bulk density

Northcliff interpolated density values within lithology domains. Table 14-8 presents a summary of the bulk density sample data included in the Mineral Resource model.

An average deposit density of 2.77 t/m^3 was assigned to blocks where the bulk density variable was not populated by the estimation routine.

For the peripheral blocks in the expanded block model volume created in 2025, SLR assigned the average density for each lithology and mineralized domain.

Table 14-8: Bulk density sample statistics by domain

Lithology Domain	Count	Density Min (t/m^3)	Density Mean (t/m^3)	Density Max (t/m^3)	CV
ft4	133	2.60	2.75	3.02	0.04
fta	239	2.59	2.75	3.06	0.04
ftc	409	2.58	2.74	3.51	0.04
igb	391	2.56	2.82	3.37	0.04
iqd	184	2.59	2.79	2.99	0.03
wkb1	190	2.60	2.76	3.29	0.03
wkb23	178	2.62	2.79	3.02	0.03
wkb4	17	2.73	2.78	2.87	0.01

14.10 Block model

The 2012 model was completed by Northcliff geologists in Maptek Vulcan (Vulcan) software, using a block size of 10 m x 10 m x 10 m. Variables included WO_3, Mo, and WO_3Eq, as well as specific gravity (SG), metal revenue, average distance of estimation samples, material (air/overburden/rock), and resource classification.

For 2025, SLR expanded the Vulcan block model extents laterally by approximately 550 m and deeper by approximately 270 m but carried over the original block grade estimates of the 2012 model. The key block model variables include the original lithology, domain, grade, classification, and density variables transferred to the expanded block model, and new 2025 variables made by merging the estimated values with assigned country rock values.

The block model dimensions and key variables are summarized in Table 14-9 and Table 14-10, respectively.

Table 14-9: 2025 Block model dimensions

Description	Units	X	Y	Z
Origin	m	648,750.0	5,135,250.0	-500.0
Bearing/Dip/Plunge	degrees	90.0	0.0	0.0
Offset minimum	m	0.0	0.0	0.0
Offset maximum	m	2900.0	3000.0	1040.0
Block Dimensions minimum	m	10	10	10
Block Dimensions maximum	m	10	10	10
No of blocks	count	290	300	104

Table 14-10: 2025 Key block model variables

Variables	Default	Type	Description
wo3pct_id2	-9	double	estimated WO_3 percent grade (ID^2)
mopct_id2	-9	double	estimated Mo percent grade (ID^2)
wo3eq_id2	-9	double	calculated WO_3Eq grade (wo3pct_id2 + mopct_id2*1.189)
avdist_wo3	-9	double	
avdist_mo	-9	double	
nsample_wo3	0	integer	
nsample_mo	0	integer	
nhole_wo3	0	integer	
nhole_mo	0	integer	
class	-9	integer	1 = Measured 2 = Indicated 3 - Inferred
material	1	integer	0 = Air 1 = Rock 9 = OB
zone	z3	name	zone (el; z3)
zone_wo3	z3_lg	name	zone (el_lg; el_hg; z3_lg; z3_hgw or z3_hge))
sg	-9	double	estimated sg
wo3_3hole	-9	double	
avdist_3hole	-9	double	
pass	-9	integer	pass #
geo_mar2012	unknown	name	March 2012 geology model (by J.Lang/M.Moll)
spct	-9	double	sulphur %
capct	-9	double	
pppm	-9	double	
geo_2025_eng	unknown	name	merge with 2012 values and assign country rock
sg_2025_eng	-99	double	merge with 2012 values and assign country rock
wo3pct_id2_2025_eng	-99	double	merge with 2012 values and assign country rock
mopct_id2_2025_eng	-99	double	merge with 2012 values and assign country rock
material_2025_eng	-99	integer	merge with 2012 values and assign country rock

Variables	Default	Type	Description
pppm_2025_eng	-99	double	merge with 2012 values and assign country rock
capct_2025_eng	-99	double	merge with 2012 values and assign country rock
spct_2025_eng	-99	double	merge with 2012 values and assign country rock
wo3eq_id2_2025_eng	-99	double	updated with new NSP and recovery
wo3pct_ok_2025_eng	-99	double	merge with 2012 values and assign country rock
class_2025_eng	-99	integer	merge with 2012 values and assign country rock
zone_2025_eng	unknown	name	merge with 2012 values and assign country rock
zone_wo3_2025_eng	unknown	name	merge with 2012 values and assign country rock
tonnes_block_slr	-99	double	volume * sg

14.11 Cut-off grade and Whittle parameters

Metal prices used for reserves are based on consensus, long term forecasts from banks, financial institutions, and other sources. For resources, metal prices used are slightly higher than those for reserves. The Mineral Resource uses long term price forecasts of US$500.00/MTU for WO_3 concentrate and US$23.00/lb for Mo, which results in a Net Smelter Price (NSP) for recovered metal of C$25.11/lb for WO_3 and C$27.17/lb for Mo, using a conversion ratio of 0.75 USD to 1.0 CAD. The NSR cut-off value was estimated by MMTS to be C$12.02/t.

This 2025 Mineral Resource utilizes the December 31, 2012 Mineral Resource block model and grade estimation, updating the NSR and cut-off grade parameters to reflect current market conditions and updated processing parameters. The 2025 Mineral Resource block model incorporates new cost considerations from MMTS's Mineral Reserve analyses for a 'No APT' WO_3 concentrate, with updated NSR values and new metal prices according to current market studies and metal price forecasts.

Both Mo and WO_3 are of economic interest for the Project. Northcliff has derived and applied an NSR cut-off value to include contributions from both metals. The NSR was based on metal prices, plant recovery factors, downstream treatment, insurance, transportation losses, and cost of sales parameters. The principal economic and metallurgical assumptions used in the NSR calculation are:

- WO_3 price = US$500/MTU
- Mo price = US$23/lb
- Exchange rate = USD0.75/CAD1.00
- Plant recovery for WO_3 variable (see equation below)
- Plant recovery for Mo = 82%
- APT plant recovery = 97%
- WO_3 concentrate grade = 30%
- Mo concentrate grade = 50%
- Payable Mo = 99%
- Transportation losses (Mo only) = 1.5%

Metallurgical test work conducted by Northcliff determined that there is a relationship between plant recovery for tungsten and head grade. Block tungsten recovery was estimated using a regression line formula based on the results of the metallurgical testing. This formula is shown below:

$$WO3\ recovery\ (\%)\ =\ ((-3{,}904.0\ x\ WO3\%2)\ +\ (869.6\ x\ WO3\%)\ +\ 34.4)\ x\ APT\_rec$$

Where: WO_3% = estimated block grade

APT_rec = APT plant recovery = 97%

The calculated recovery was limited to a theoretical maximum of 82% for blocks with grades greater than 0.0968% WO_3.

The SLR QP reviewed the inputs to the NSR calculation and considers them to be reasonable.

The recoverable NSR values per unit contributed by each metal were determined as Net Smelter Prices (NSP). for recovered metal. These values are multiplied by the estimated block grades to obtain an NSR value for the blocks.

These NSP values for the tungsten and molybdenum were determined to be:

NSP_ WO_3 = C$25.11/lb WO_3

NSP_Mo = C$27.17/lb Mo

The block NSR equation is as follows:

$$NSR\ =\ (WO3\%\ x\ NSP\_〚WO〛\_3\ x\ REC\_〚WO〛\_3\ x\ 2204.6\ lb/t/100\%) + (Mo\%\ x\ NSP\_Mo\ x\ REC\_Mo\ x\ 2204.6\ lb/t/100\%)$$

Where: WO_3% = estimated block grade

Mo% = estimated block grade

NSP_ WO_3 = C$25.11/lb WO_3

NSP_Mo = C$27.17/lb Mo

REC_Mo = metallurgical recovery for Mo

REC_ WO_3 = metallurgical recovery for WO_3

14.11.1 Equivalence calculation

Tungsten equivalence was calculated for each block as a metric for comparison with other Projects and previous Sission estimates. The WO_3Eq values were calculated for each block using a formula derived from the NSR calculation parameters described above. The block NSR value was then divided by the NSR multiplier for WO_3 to obtain the WO_3Eq, as follows:

$$WO3Eq\ =\ WO3\%\ +\ ((Mo\%\ x\ NSP\_Mo\ x\ REC\_Mo)/(NSP\_WO3\ x\ REC\_WO3))$$

14.11.2Whittle parameters

To comply with the CIM requirement that Mineral Resources be demonstrated to meet "reasonable prospects for eventual economic extraction", SLR prepared a preliminary Lerchs-Grossmann pit shell using the estimated costs and parameters shown in Table 14-11. Only those blocks contained within the preliminary pit shell are reported as Mineral Resources (Table 14-11) at the NSR reporting cut-off grade of C$12.02/t. SLR reviewed the inputs to the NSR calculation and considers them to be reasonable.

Table 14-11: Limiting pit shell parameters

Item	Amount
WO_3 Price	C$500/MTU
Mo Price	C$23/lb
Mining Cost	C$2.98/t
Milling + G&A Cost	C$12.02/t
Pit Slope	43°
Mo Recovery	82%
WO_3 Recovery	Recovery calculation capped at 82%

14.12 Classification

Definitions for resource categories used in this Technical Report are consistent with those defined by CIM (2014) and adopted by NI 43-101. In the CIM classification, a Mineral Resource is defined as "a concentration or occurrence of solid material of economic interest in or on the Earth's crust in such form, grade or quality and quantity that there are reasonable prospects for eventual economic extraction". Mineral Resources are classified into Measured, Indicated, and Inferred categories. A Mineral Reserve is defined as the "economically mineable part of a Measured and/or Indicated Mineral Resource" demonstrated by studies at Pre-Feasibility or Feasibility level as appropriate. Mineral Reserves are classified into Proven and Probable categories.

As previously stated in Section 14.11 of this Technical Report, only those blocks contained within the preliminary pit shell are reported as Mineral Resources at the NSR cut-off grade of C$12.02/t.

Estimated blocks were classified by geostatistical means using Vulcan scripts. The principal criterion for assignment of resource classification was the average distance to the nearest three drill holes. The classification was assigned using the following criteria:

1 Blocks with an average distance to the nearest three holes of 45 m or less were classified as Measured Resources (class = 1).

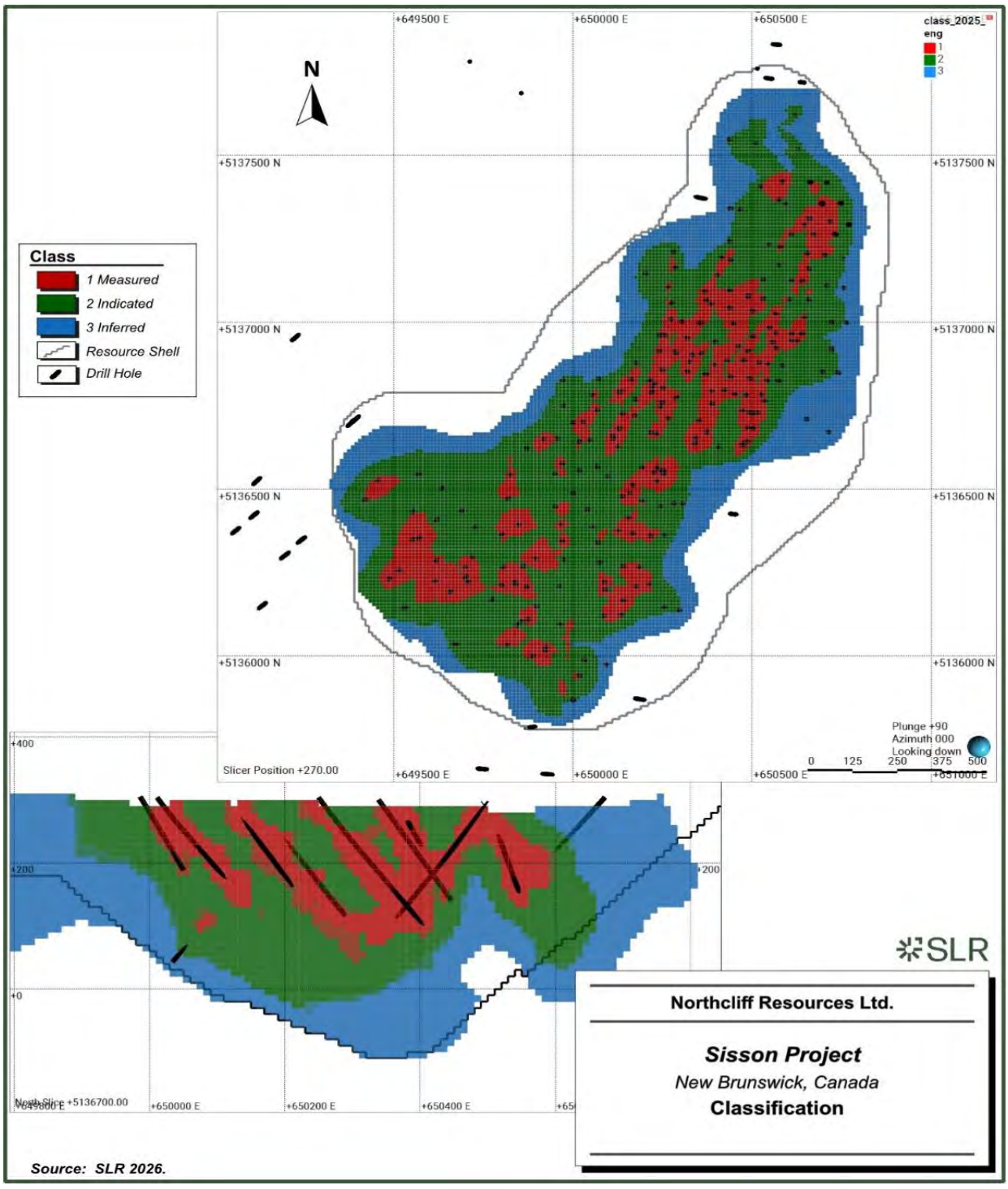
2 Blocks with an average distance to the nearest three holes between 45 m and 80 m were nominally classified as Indicated Resources (class = 2).

3 Blocks with an average distance to the nearest three holes between 80 m and 125 m were nominally assigned to the Inferred Resources (class = 3).

For the Indicated and Inferred classes, 3D category solids were created from sectional interpretations of category blocks. This allowed local smoothing of category boundaries and elimination of isolated classification artefacts. These solids were then used for repopulation of the block model variable class. A script was also created to ensure grade values were erased where no class variable had been assigned.

The QP examined the resulting classification relative to the drilling (Figure 14-4) and is of the opinion that the classification encoded in the Mineral Resource model is reasonable. Measured Resources tend to form a latticework of material where drill hole proximity permits the Measured class, and the volume of Measured and Indicated material taken together represents a coherent volume which avoids encapsulating sporadic volumes of Inferred Resources.

Figure 14-4: Block classification



Source: SLR, 2026

14.13 Block model validation

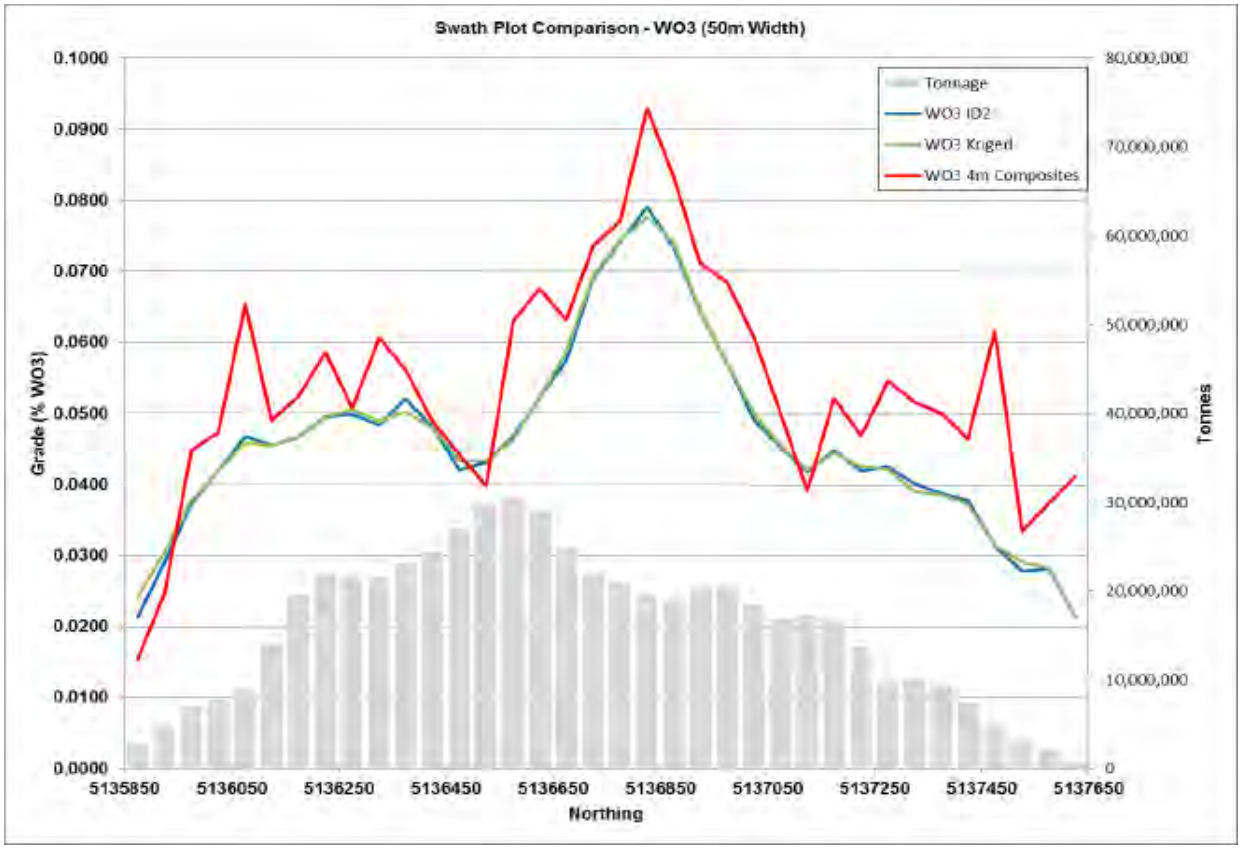
14.13.1Northcliff block validation (2012)

A visual comparison completed by Northcliff on a section-by-section basis showed acceptable degree of consistency between the block model and drill hole assay grade distribution.

Swath plots for WO_3 and molybdenum were generated to assess the model for global bias by comparing ID^2 values with ordinary kriging (OK) estimates and 4 m run length composites. These plots were compiled on 50 m vertical east-west panels through the deposit. Only blocks and composites that are within the Indicated category outline were used for swath plot generation. In the QP's opinion, the plots show a good comparison between the three methods, particularly in the main portions of the deposit. At the north end of the deposit, some deviation between WO_3 estimated grades and assay composites was noted. This deviation between estimated and composite grades may be due to the lower number of samples available to populate blocks at the edges of the model. The differences in grade between models and raw data are shown in Figure 14-5 and Figure 14-6.

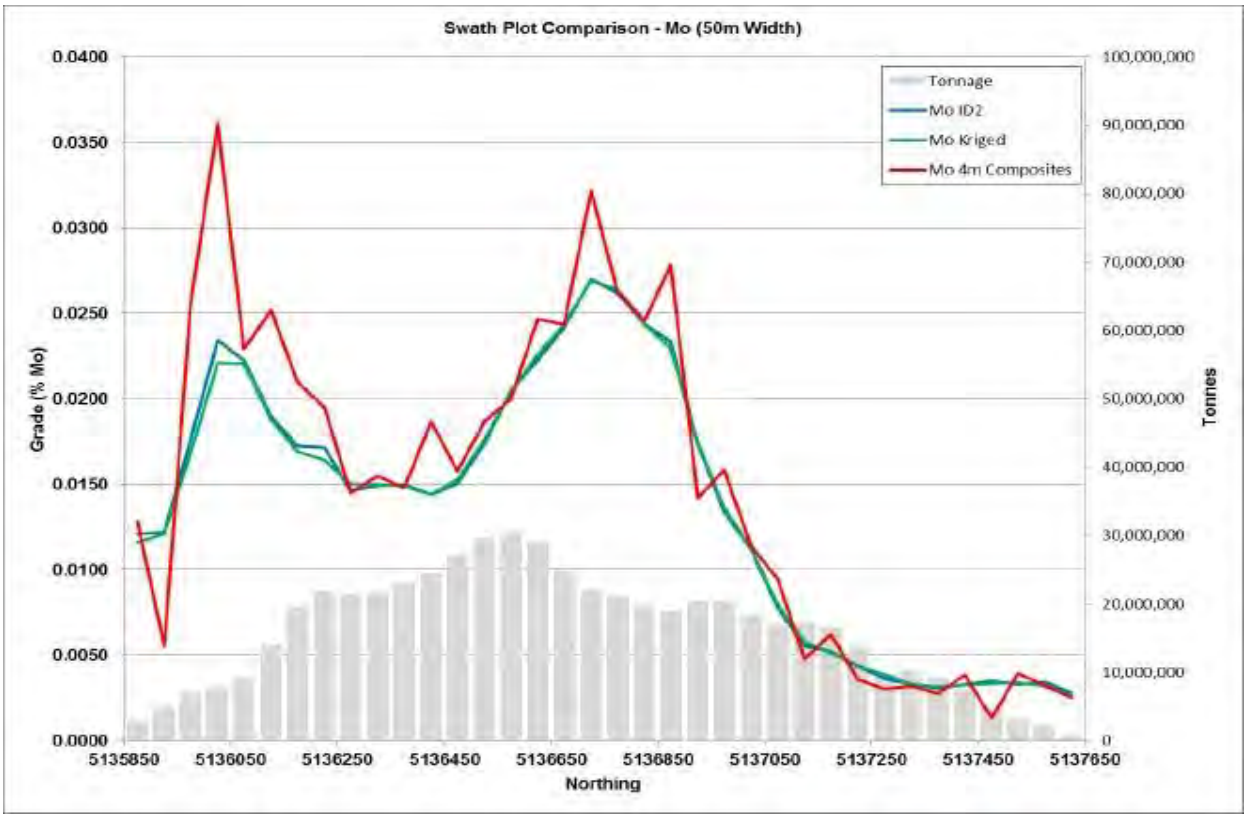
Swath plots for SG were generated to assess the model for global bias by comparing ID^2 values with drill hole SG measurements on 100 m vertical south-north panels through the deposit. Results show a good comparison between the two data sets (Figure 14-4).

Figure 14-5: Swath plot comparison for WO_3



Source: Samuel Engineering, 2013

Figure 14-6: Swath plot comparison for Mo



Source: Samuel Engineering, 2013

Figure 14-7: Swath plot comparison for SG



Source: Samuel Engineering, 2013

14.13.2 RPA validation (2012)

The following is a list of the checks performed on the drill hole database and resource models by RPA:

- Checked for duplicate drill hole collar locations and hole numbers.
- Checked collar locations for zero/extreme values.
- Checked assays in database for missing intervals, long intervals, extreme high values, blank/zero values, reasonable minimum/maximum values.
- Ran validity report to check for out-of-range values, missing intervals, overlapping intervals, etc.
- Checked for overlapping wireframes to assess potential double counting of resource volumes.
- Checked mineralized domain/wireframe extensions beyond last holes to see if they are reasonable and consistent.
- Compared basic statistics of assays within wireframes with basic statistics of composites within wireframes for both uncut and cut values.
- Checked the capping of extreme values and effect on coefficient of variation.
- Checked for reasonable compositing intervals.
- Checked that composite intervals start and stop at wireframe boundaries.
- Checked that assigned composite rock type coding is consistent with intersected wireframe coding.
- Checked if block model size and orientation is appropriate for drilling density, mineralization, and mining method.
- Checked search volume radii and orientations against available variography.
- Checked estimation parameters against available variography.
- Visually checked block resource classification coding for isolated blocks.
- Visually compared block grades to drill hole composite values on section and plan views.
- reran the block NSR calculation to confirm that it had been done correctly.
- Visually checked for grade banding, smearing of high grades, plumes of high grades, etc., on sections and plans.

While performing the above checks, RPA noted the following:

- Even after cutting, over 20% of the total molybdenum metal content resides within the top percentile of the data. This indicates that there is still a risk of overestimation of global molybdenum in the block model. This is not expected to be severe but warrants further review as the Project advances.
- The risk of overestimation of the molybdenum metal content may also be increased somewhat by the fact that the interpolation is effectively unconstrained. It is noted, however, that this risk is greatest on the periphery of model, in areas that were often not included as Mineral Resources.
- There are several areas in the model where high SG estimates have been allowed to fill blocks in an unusual manner. This occurs, again, at the edges of the model, presumably where there are few data points and not enough constraining data. Some of these blocks, however, are within the pit limit. The net impact on the global resource estimate is not anticipated to be overly severe, but it is likely that some block tonnages will be overestimated. RPA noted that, in poorly drilled portions of the deposit, it may be more appropriate to use average SGs rather than trying to interpolate them. The impact of making this change should be evaluated.

- There also appear to be differences in the statistics for the various zones, which suggest that it might be more prudent to develop independent top cuts on a zone-by-zone basis rather than a single top cut for all.

In RPA's opinion, the assumptions, parameters, and methodology used in the MRE were appropriate for the style of mineralization and proposed mining method. The resources were further constrained by an 80 m limit to the external boundary of the model. This limit was used because it represents two-thirds of the semi-major axis range of the tungsten variogram.

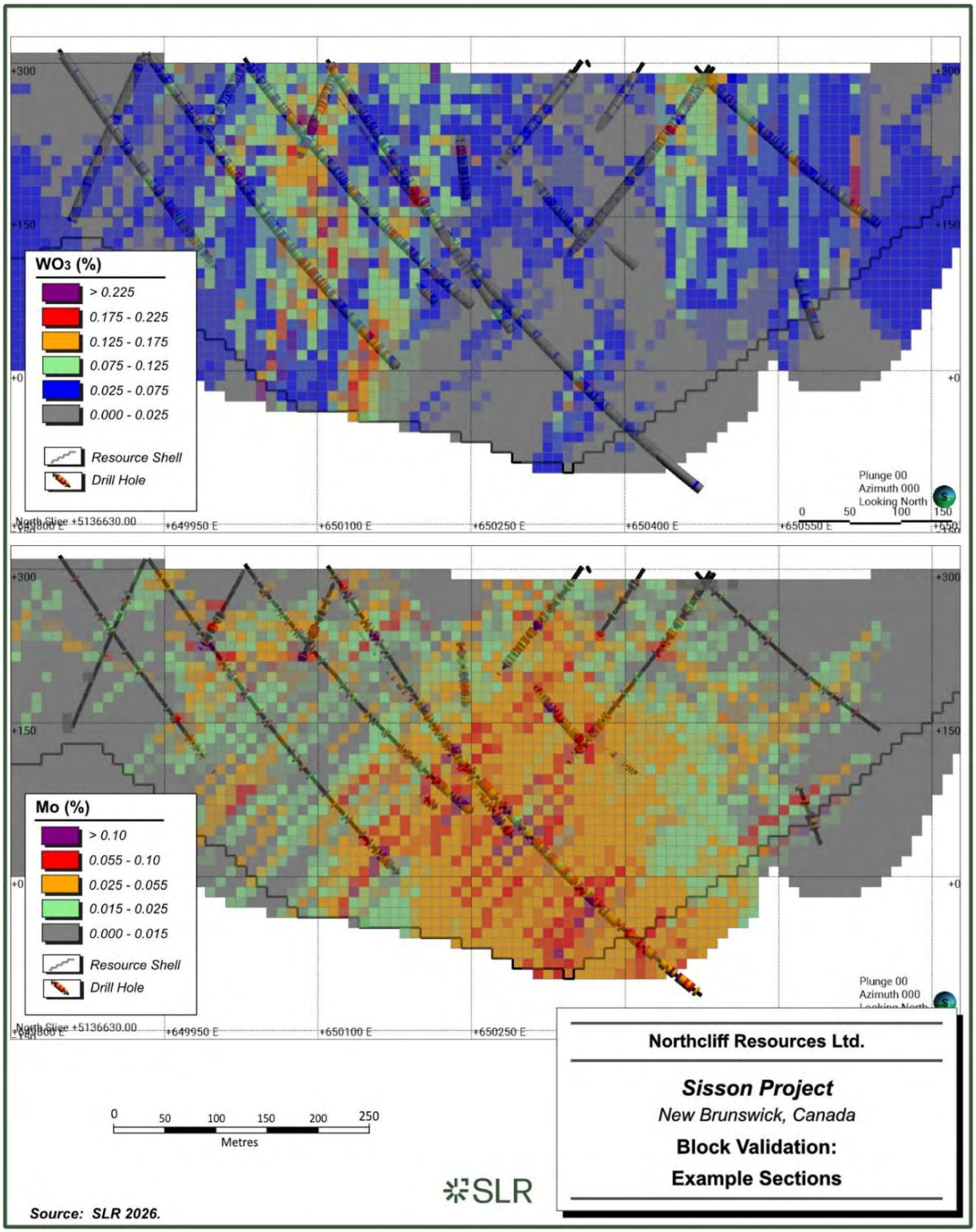
14.13.3 SLR QP validation (2025)

The following is a list of the checks performed on the drill hole database and resource model by the SLR QP:

- Checked for duplicate drill hole collar locations and hole numbers.
- Checked collar locations against expanded topographic surface.
- Checked for overlapping wireframes to assess potential double counting of resource volumes.
- Checked mineralized domain/wireframe extensions beyond last holes to see if they are reasonable and consistent.
- Checked drilled assays since 2013 to ensure no material change to the MRE is likely.
- Checked the provided block model against the previous Mineral Resource table.
- Validated block grades in the expanded model against both the previous block model provided, and the current model in use for the Mineral Reserve estimate, to ensure grades transferred correctly.
- Checked if block model size and orientation is appropriate for drilling density, mineralization, and mining method. Updated model extents for new Whittle shell.
- Visually compared block grades to drill hole values on section and plan views.
- Visually checked block resource classification coding for isolated blocks.
- Visually confirmed block lithology and mineralization domain coding in the expanded model to ensure block calculation scripts worked correctly.
- Visually checked for grade banding, smearing of high grades, plumes of high grades, etc., on sections and plans.
- Ensured that all variables sent for Whittle contained no negative values.
- Compared NSR calculations to previous and to the MMTS NSR spreadsheet. Validated calculations with comparative statistics and QQ, box, swath plots using Vulcan, Leapfrog and Python.
- Met with MMTS and Northcliff staff to confirm parameters and assumptions.
- Checked metal price and cost assumptions against SLR internal guidance and client-commissioned reports.

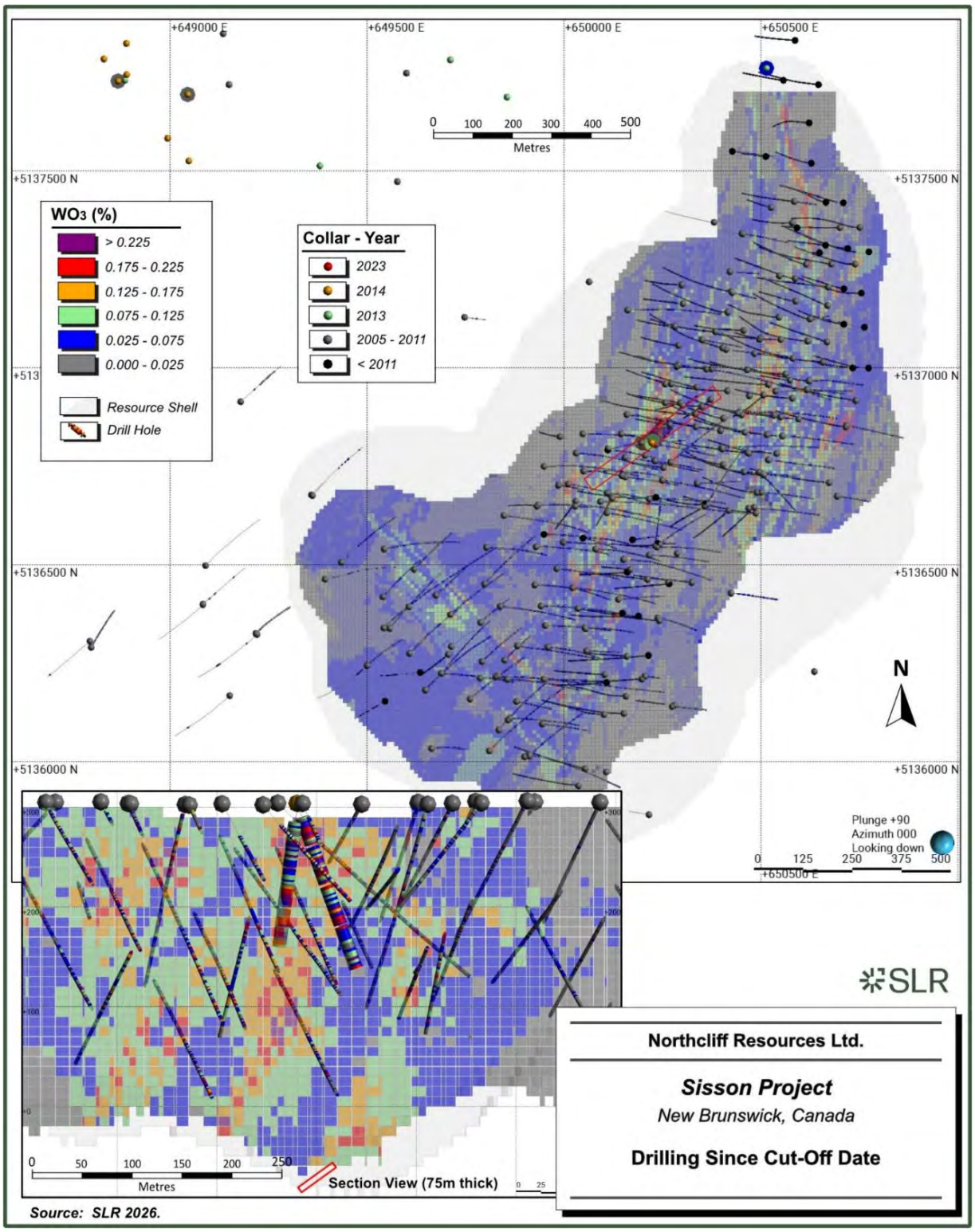
The QP concurs with RPA's findings and recommends that the next Mineral Resource model address the notes above in conjunction with a full update in Vulcan and/or Leapfrog software. At present, the block model is sufficient for the purposes of the current work, and the small amount of additional drilling after the 2012 block estimate is not likely to materially affect the grades in the current model.

Figure 14-8: Block validation: East West Section 5136630N looking due North



Source: SLR, 2026

Figure 14-9: Drilling since cut-off date



Source: SLR, 2026

14.14 Mineral Resource Estimate

The MRE was updated using Maptek Vulcan (Vulcan) software using a conventional approach including 3D solid modelling and block modelling.

This most recently revised estimate is current to December 31, 2025, and is summarized in Table 14-12. Canadian Institute of Mining, Metallurgy and Petroleum (CIM) Definition Standards for Mineral Resources and Mineral Reserves definitions (CIM, 2014) were used for Mineral Resource classification.

Table 14-12: Summary of Mineral Resources – effective date: December 31, 2025

Category	Tonnage	Grade WO_3	Grade Mo	Contained WO_3	Contained Mo	Grade WO_3EQ	Average NSR
	(Mt)	(%)	(%)	(M MTU)	(M lb)	(%)	(C$/t)
Measured	117.9	0.068	0.022	8.0	57.2	0.098	39.13
Indicated	303.7	0.061	0.019	18.5	127.2	0.088	34.65
Total Measured and Indicated	**421.6**	**0.063**	**0.020**	**26.6**	**185.9**	**0.091**	**35.90**
Inferred	182.9	0.048	0.020	8.8	80.6	0.078	28.59

Notes:

1. CIM (2014) definitions were followed for Mineral Resources.
2. $WO_3Eq = WO_3\% + ((Mo\% \times NSP\_Mo \times REC\_Mo)/(NSP\_WO_3 \times REC\_WO_3))$.
3. Mineral Resources are estimated at an NSR cut-off grade of C$12.02/t.
4. Mineral Resources are estimated using a long-term WO_3 price of US$500/MTU, and a long-term Mo price of US$23/lb.
5. Mineral Resources are constrained within a Whittle shell, generated using the NSR cut-off value.
6. Bulk density is interpolated into blocks where density sample information is present and set to 2.77 t/m^3 outside of the estimated blocks.
7. Mineral Resources are inclusive of Mineral Reserves.
8. Mineral Resources that are not Mineral Reserves do not have demonstrated economic viability.
9. To meet reasonable prospects for eventual economic extraction, Mineral Resources assume metallurgical recoveries and processing costs to be the same as for Mineral Reserves but use higher long term metal prices to calculate NSR.
10. Numbers may not add due to rounding.

The QP is not aware of any environmental, permitting, legal, title, taxation, socio-economic, marketing, political, or other relevant factors that could materially affect the MRE.

The QP is of the opinion that the Mineral Resource model could be updated with further drilling as the Project progresses.

14.14.1Comparison to previous estimate

Comparing the updated Mineral Resource to the 2012 MRE (Table 14-13) shows that overall grades have decreased by approximately five percent, with an 8.2% tonnage increase for Measured and Indicated Resources, and 2.3% decrease for Inferred Mineral Resource. The grade difference is due to lower-grade blocks reporting to each class resulting from a lower cut-off value.

Table 14-13: Mineral Resource comparison: 2025 vs. 2012

Class	Mt	WO_3 (%)	Mo (%)	WO_3 (M MTU)	Mo (Mlb)	WO_3EQ (%)	NSR (C$/t)
2012 MRE							
1 Measured	108	0.072	0.023	7.7	55.3	0.096	26.67
2 Indicated	279	0.065	0.020	18.0	122.0	0.086	23.42
1+2 Measured and Indicated	387	0.067	0.021	25.7	178.0	0.089	24.33
3 Inferred	187	0.050	0.020	9.4	82.3	0.074	18.63
2025 MRE							
1 Measured	118	0.068	0.022	8.0	57.2	0.098	39.13
2 Indicated	304	0.061	0.019	18.5	127.2	0.088	34.65
1+2 Measured and Indicated	422	0.063	0.020	26.6	185.9	0.091	35.90
3 Inferred	183	0.048	0.020	8.8	80.6	0.078	28.59
Difference (2025-2012)							
1 Measured	9.9	-0.004	-0.001	0.319	1.895	0.002	12.46
2 Indicated	24.7	-0.004	-0.001	0.525	5.210	0.002	11.23
1+2 Measured and Indicated	34.6	-0.004	-0.001	0.862	7.901	0.002	11.57
3 Inferred	-4.1	-0.002	0.000	-0.633	-1.677	0.004	9.96
Difference (%)							
1 Measured	8.4%	-5.9%	-4.5%	4.0%	3.3%	2.0%	31.8%
2 Indicated	8.1%	-6.6%	-5.3%	2.8%	4.1%	2.3%	32.4%
1+2 Measured and Indicated	8.2%	-6.3%	-5.0%	3.2%	4.2%	2.2%	32.2%
3 Inferred	-2.3%	-4.2%	0.0%	-7.2%	-2.1%	5.1%	34.8%

15 Mineral Reserve Estimates

15.1 Introduction

The Mineral Reserves for Sisson are a subset of, and physically contained within, the Measured and Indicated Mineral Resources, described in Section 14, as supported by the FSU Open-pit LOM Plan, described in Section 16.

Proven and Probable Mineral Reserves are modified from Measured and Indicated Mineral Resources. Inferred Mineral Resources are set to waste.

15.2 Mineral reserves statement

Mineral Reserves are estimated using the CIM 2019 Best Practices Guidelines (CIM, 2019) and are classified using the 2014 CIM Definition Standards (CIM, 2014).

Proven and Probable Mineral Reserves are summarized in Table 15-1.

Table 15-1: Summary of Mineral Reserves

Confidence Category	ROM (Mt)	Grade			Contained Metal	
		NSR ($/t)	WO_3 (%)	Mo (%)	WO_3 (M MTU)	Mo (M lb)
Proven	85.6	37.81	0.079	0.025	6.77	47.79
Probable	191.2	33.78	0.073	0.021	13.92	90.58
Total Proven & Probable	**276.8**	**35.03**	**0.075**	**0.023**	**20.69**	**138.37**

Notes:
1. The Mineral Reserve estimates were prepared by Robert Gray, P.Eng. (who is also the independent Qualified Person for these Mineral Reserve estimates), were reported using the 2014 CIM Definition Standards, and have an effective date of August 11, 2026.
2. Mineral Reserves utilize the current Mineral Resource Estimate and are defined by the 2026 Feasibility Study life-of-mine open pit design and production schedule.
3. Mineral Reserves are mined tonnes and grade; the reference point is the mill feed at the primary crusher and includes consideration for operational modifying factors such as loss and dilution. Estimated ROM quantities and grade are based on measurements within a 10 m x 10 m x 10 m block, with a variable application of dilution by block, averaging 2.56% dilution over the model at 0% diluting grade, and a 98.76% mining recovery.
4. Mineral Reserves are reported based on a variable (by period) NSR grade cut-off, with an effective LOM cut-off at $16.31/t.
5. NSR grade assumes: US$330/MTU WO_3 (Concentrate pricing) and US$20/lb Mo; C$0.12/lb WO_3 and C$3.39/lb Mo offsite costs (transportation, losses, insurance, treatment charges, and smelter costs); variable WO_3 metallurgical recovery of [(-3,904.0 x $WO_3\%^2$) + (869.6 x $WO_3\%$) + 34.4] with a maximum of 82% and fixed Mo metallurgical recovery of 82%
6. Numbers have been rounded as required by reporting guidelines.

Source: MMTS, 2026

15.3 Factors that may affect the mineral reserves

Mineral Reserves are based on the engineering and positive economic results described in Sections 16 to 22 of this Technical Report. Changes in the following factors and assumptions may affect the Mineral Reserve estimate:

- Metal prices
- Interpretations of mineralization geometry and continuity of mineralization zones
- Geotechnical and hydrogeological assumptions
- Ability of the mining operation to meet the annual production rate
- Changes to operating and capital cost assumptions
- Ability of the operations to achieve the estimated mining and process plant recoveries
- Ability to meet and maintain permitting and environmental license conditions and the ability to maintain the social licence to operate.

15.4 Comments on mineral reserve estimate

The Mineral Reserve estimates are based on the most current knowledge, permit status, and engineering constraints. Due to variable cut-off grades and metallurgical recoveries, the Mineral Reserves estimate in Table 15-1 can only be replicated when the reserves within the detailed pit design phases described in Section 16.4 of this Technical Report are run through an economic production schedule as described in Section 16.7 of this Technical Report.

The QP is of the opinion that the Mineral Reserves are estimated using industry-accepted practice.

16 Mining methods

16.1 Mining method selection

Mine designs and operations for the Project are typical of owner operated open pit, truck-and-shovel mining methods for year-round operations in Canadian climates.

16.2 Key design considerations

The following planning design inputs are used:

- Local topographical data in UTM Zone 19T NAD83 coordinates
- 3D block model imported by MMTS from a .csv file format received from Sisson LP on February 13, 2012, as audited by RPA, and containing the following features:
  - 10m x 10m x 10m spacing
  - In situ grades for WO_3 and Mo
  - Lithology, resource classifications and SG data
- Inferred resources are treated as barren rock with no economic values.
- Process recovery:
  - $WO3\ recovery\ (\%) = ((-3{,}904.0\ x\ WO3\%2) + (869.6\ x\ WO3\%) + 34.4)$
- Where WO3% = estimated block grade.
- The calculated recovery was limited to a theoretical maximum of 82% for blocks with grades greater than 0.0968% WO_3.
- Mo Recovery (%) = 82%.
- Owner operated fleet with the use of autonomous hauling and electrification of mining shovels.

16.2.1 Net smelter price, net smelter, and cut-off grade

NSR is defined as the dollar value in a block in $/t, available to the local operation (i.e., inside the property gates). The NSR value accounts for in situ grades, process recoveries, and the net smelter price (NSP). The NSPs are based on the market studies for WO_3 and Mo (Argus Media, 2025), with all off-site costs deducted to the Project (Table 16-1 and Table 16-2). Offsite costs include items such as transportation, losses, insurance, treatment charges, and smelter costs.

Table 16-1: Net smelter price input for WO_3

Description	Value	Unit
WO_3 (Concentrate) Price	$330	US$/MTU WO_3
WO_3 Price (in lbs)	$14.96	US$/lb WO_3
US Exchange rate	0.74	USD/CAD
Offsites	$0.12	C$/lb WO_3
Resulting NSP WO_3	$20.08	C$/lb WO_3

Table 16-2: Net smelter price input for Mo

Description	Value	Unit
Mo Price	$20.00	US$/lb
US Exchange rate	0.74	USD/CAD
Offsites	$3.39	C$/lb Mo
Resulting NSP Mo	$23.61	C$/lb Mo

Both molybdenum and WO_3 are of economic interest for the Project. The NSR of each block in the block model provides net revenue and covers the mining, processing, and any other attributed costs from the operation. The NSR in each block of the model is calculated using the following formula:

$$NSR = (WO3\% \; x \; NSP\_WO3 \; x \; Process\_Recovery\_WO3 \; x \; 2204.6 \; lb/t/100\%) + (Mo\% \; x \; NSP\_Mo \; x \; Process\_Recovery\_Mo \; x \; 2204.6 \; lb/t/100)$$

With the NSR value calculated for each mineralized block in the 3D block model, the pit benches, sub-benches or individual blocks can be examined for their contribution to positive cash flow.

Tungsten equivalence was calculated for each block as a metric for comparison with other estimates. The WO_3Eq values were calculated for each block using a formula derived from the NSR calculation parameters. The block NSR value was then divided by the NSR multiplier for WO_3 to obtain the WO_3Eq. The calculation is provided below:

$$WO3Eq = WO3\% + ((Mo\% \; x \; NSP\_Mo \; x \; Process\_Recovery\_Mo)/(NSP\_WO3 \; x \; Process\_Recovery\_WO3))$$

The cut-off grade is based on the calculated NSR. Low grade resource blocks, internal in the pit design, must be mined to expose higher grade blocks below them; therefore, they can still contribute to a positive cash flow if they have an in situ grade value that is greater than the incremental operating cost. The cost of mining blocks from the pit is required regardless of blocks being ore or barren rock, therefore the NSR value must be greater than the process and administrative costs for the block to contribute positively to the cash flow. The economic break-even cutoff grade inputs are shown in Table 16-3.

Table 16-3: Cut-off grade inputs

Description	Units	Estimate
Process Operating Costs	$/t	10.60
TSF Operating Costs	$/t	0.35
Quarry Operating Costs	$/t	0.56
G&A Costs	$/t	0.51
NSR Cutoff Grade	$/t	12.02

It is acknowledged that these estimates made during the early stages of the project have been subject to updates as other disciplines advanced their cost estimates, however the $12.02/t cut-off was retained for the entirety of the mine planning work. During production scheduling a variable cut-off grade was used each period to optimize the net present value of the schedule. Typically, the effective cut-off grade used in each period is higher than the break-even cut-off grade estimated in Table 16-3. See Section 16.7.3for details.

16.2.2 Mining loss and dilution

Mining recovery and dilution factors are applied to convert in situ resource tonnages into run-of-mine (ROM) ore. While the 3D block model already incorporates internal (whole block) dilution, contact dilution and loss must also be accounted for due to the presence of significant, distinct boundaries between the ore and barren rock.

There are three main components of contact loss and dilution considered in this study:

- Dilution of barren rock into ore where separate ore and barren rock blasts are not possible.
- Loss of ore into barren rock where separate ore and barren rock blasts are not possible.
- General mining losses and dilution due to material handling (e.g., carry-back in truck boxes, stockpile floor losses).

NSR values coded in the 3D block model define the boundaries between ore and barren rock. Contact dilution is based on the spatial distribution of these blocks and is coded directly into the model as diluted grades. For a given ore block, the magnitude of the loss and dilution increases as the number of waste contact edges increases.

For the purposes of Mineral Reserve estimation, contact loss is assumed to be volumetrically equal to contact dilution. As a result, adding 2.56% dilution over the entire model at a 0% diluting grade is considered representative of the diluted ROM material the operator will achieve when pursuing the targeted mine plan production rate. Because contact loss is offset by contact dilution, the overall mining recovery only needs to account for handling losses (1.24%), equating to a final mining recovery of 98.76%.

The estimated overall mining loss and dilution values are shown in Table 16-4.

Table 16-4: Loss and dilution % estimates

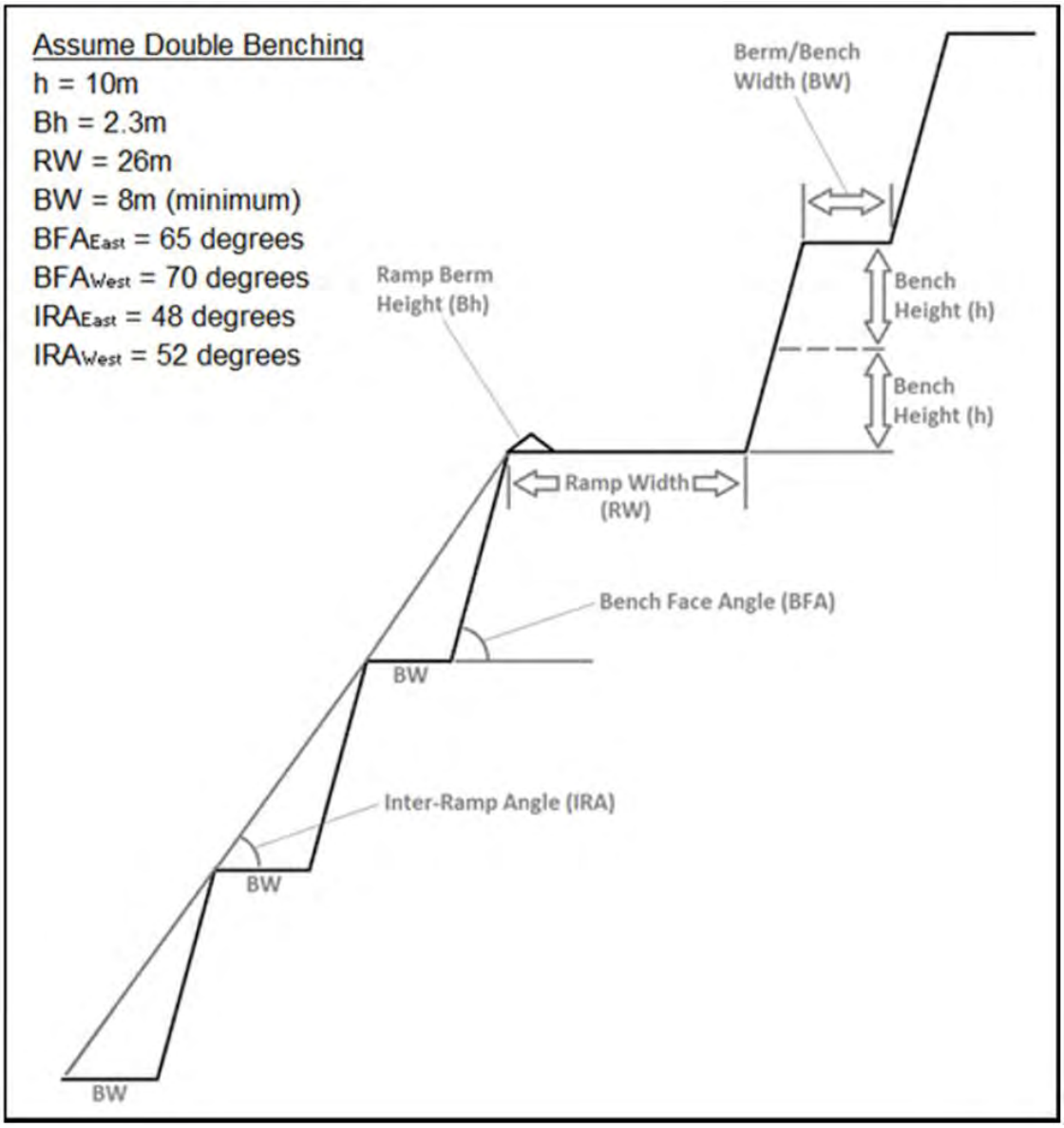
Handling Loss	Contact Block Loss	Total Loss	Dilution	Mining Recovery
1.24%	2.56%	3.80%	-2.56%	98.76%

16.2.3 Pit slopes

No new geotechnical site investigation or analysis have been completed for the open pit since the 2013 Feasibility Study. The geotechnical parameters used in the pit optimization and detailed pit design process are based on the "Feasibility Pit Slope Design" completed by Knight Piésold (2012b) and are summarized on Figure 16-1.

Double benching is recommended to achieve the steepest inter-ramp angles. A bench face angle of 70° is expected to be appropriate for the West Sector with adequate catch benches. Slightly flatter bench face angles of 65° are more appropriate for the East Sector where a relatively weaker rock mass with more adversely oriented structure is present (Knight Piésold, 2012b).

Figure 16-1: Cross-sectional schematic of pit wall with geotechnical design parameters

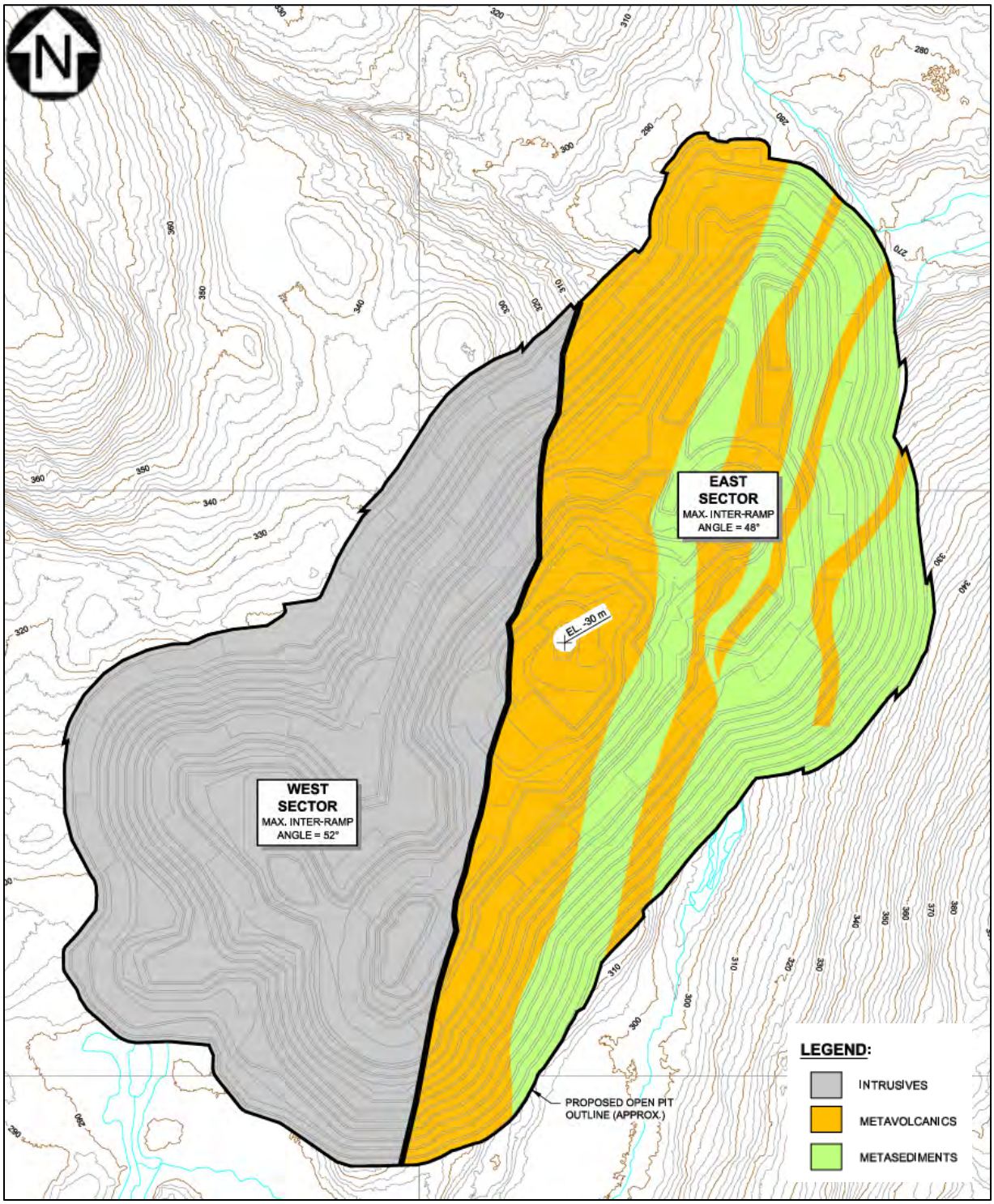


Source: MMTS, 2013

The software used for the detailed pit design (MinePlan Pit Expansion tool), was run using fixed values for bench face angles and inter-ramp angles, while berm widths were dynamically solved based on pit geometry, with a theoretical minimum of 8m as per standard safe operating practices for Canadian mines. A bench width of 8.5m was recommended by KP. Bench widths throughout the open pit are typically ~8.7m, meeting and exceeding both minimum safe operating widths and recommended geotechnical parameters.

Geotechnical parameters are divided into the East and West Domains, where the East/West domain boundary runs at approximately +10 degrees azimuth from North-South (Figure 16-2).

Figure 16-2: Pit slope design sectors



Source: Knight Piésold, 2012b

16.3 Pit optimization

The economic pit limits are determined using the Pseudoflow algorithm. This algorithm uses the NSP's, and recoveries for each block of the 3D block model and evaluates the costs and revenues of the blocks within potential pit shells. The pit optimization analysis uses a base case NSP (100% case) and varies this price by 10% down to the 50% case and up to the 140% case. This generates a series of consecutively nested Pseudoflow pit shells, using revenues to demonstrate the pit resource sensitivity. Since the cut-off grades are not changed when pit resources are calculated, this does not represent a price sensitivity analysis.

Additional cases are included in the analysis to evaluate the sensitivities of resources to, slope, permit boundaries, mill feed throughputs, mining costs and process costs.

By varying the economic parameters while keeping inputs for metallurgical recoveries and pit slopes constant, various generated pit cases are evaluated to determine where incremental pit shells produce marginal or negative economic returns. 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.

For each pit shell, an undiscounted cash flow (UCF) is generated based on the shell contents and the economic parameters listed in Table 16-1, Table 16-2 and Table 16-5. The UCFs for each case are compared to reinforce the selected point at which increased pit expansions do not materially increase value. Note that the economics are only applied for comparative purposes to assist in the selection of an optimum pit shell for further mine planning; they do not reflect the actual financial results of the mine plan.

The chosen pit shell is then used as the basis for further detailed design.

Table 16-5: Inputs into pseudoflow pit optimization

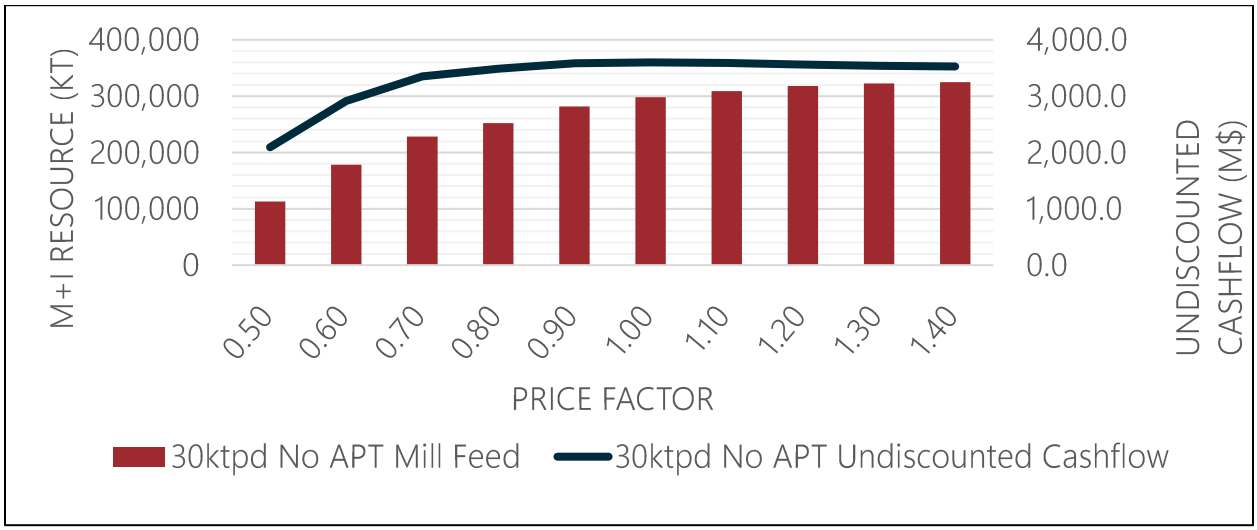
Item	Unit
Pit rim ore mining cost	$2.98/t
Pit rim barren rock mining cost	$2.98/t
Processing and G&A cost	$12.02/t
Base Slope Input	43°
Recovery WO_3	Variable
Recovery Mo	82%

16.3.1 Ultimate pit limits

Figure 16-3 shows the contents of the generated Pseudoflow pit shells. Several inflection points can be seen in the curve of cumulative resources and UCF by pit case. Discounted cashflows were produced based on these pit shells and indicate that the Case PF 0.9 to be a point at which larger pit shells will not produce material increases to project value.

The pit shell generated from Case PF 0.9 is selected as the ultimate pit limits and is used for further mine planning as a target for detailed open pit designs with berms and ramps.

Figure 16-3: Pseudoflow pit shell resource contents by case



Source: MMTS, 2026

16.4 Pit design

16.4.1 Pit phasing

Ultimate pit limits are generally split into phases or pushbacks to target higher economic margin material earlier in the mine life as well as reducing the waste stripping hurdles as the pit develops. The phase designs target producing consistently higher grades earlier in the LOM.

Minimum pushback distances of 30 m are honoured.

The Sisson pit is split into seven phases. The first phase targets higher grade, lower strip ratios area of the pit. Phases then proceed to target consistent mill feed grade and high economic margins, targeting pit shells represented by inflection points on the curve shown in Figure 16-3.

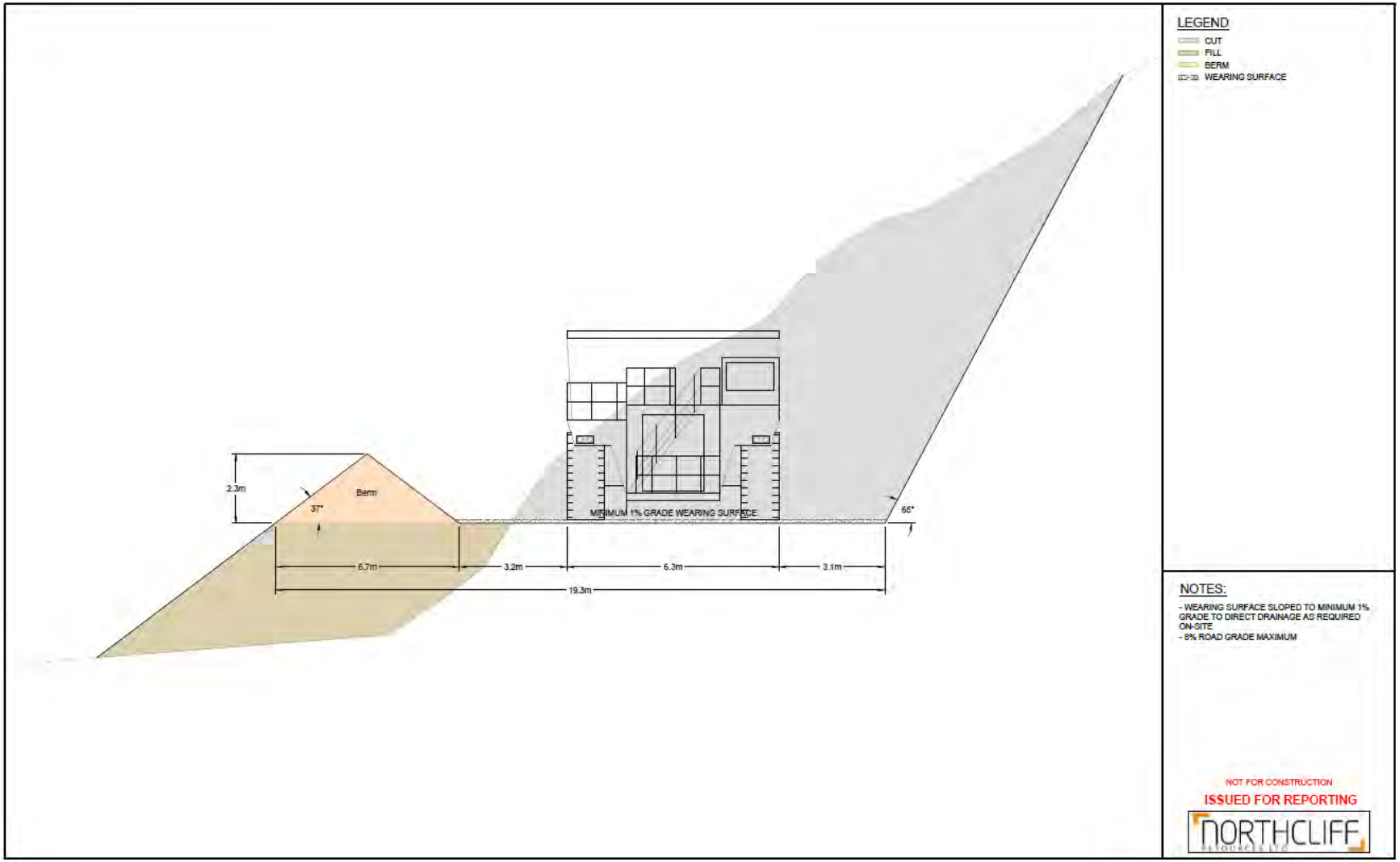
The pit design remains within the existing permitted boundary.

16.4.2 In-pit haul roads

Two-way in pit haul roads are designed at a width of 25 m to support the use of 135 t payload haul trucks (see Figure 16-5). Haul road grades are limited to a maximum of 8%. Access ramps are not designed for the last bench of the pit bottom, on the assumption that the bottom ramp segments will be removed using some form of retreat mining. The bottom two ramped benches of the pit use one-way 19m wide haul roads, at 8% grade, since bench volumes and traffic flow are reduced (see Figure 16-4).

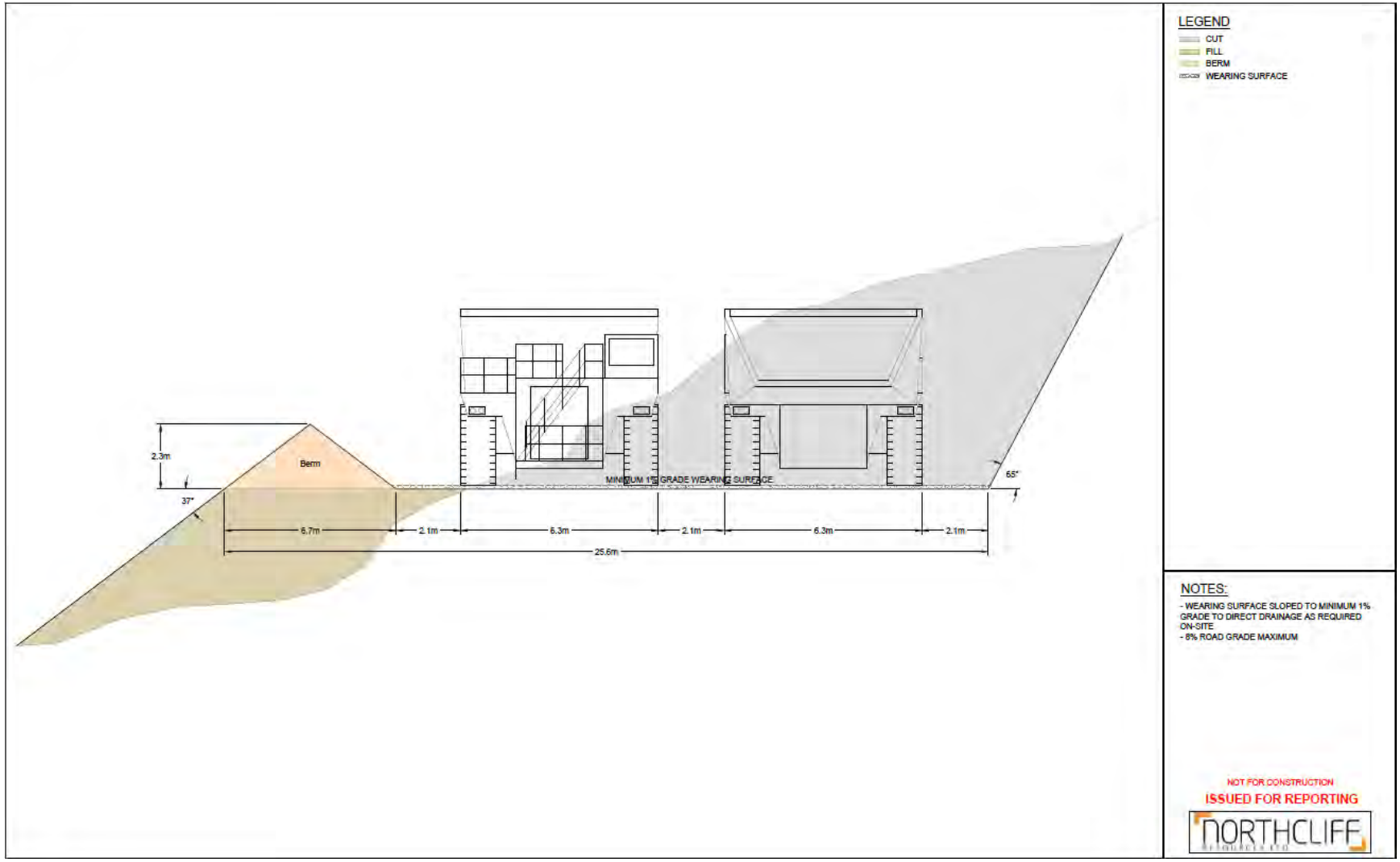
Since completing the pit design, the design and capacity of the 135-t hauler have increased across most vendors. This has changed the largest vehicle width from 6.3 m to 7.6 m, which in turn caused the width of the roads to be increased. A pit design using wider ramps to test this impact was completed, however had little impact of the contents of the pit for both ore and barren rock due to pushouts in the highwalls to target ore pockets, slightly reducing these pushouts helped mitigate the wider ramps while still capturing the ore pockets. It was determined by Northcliff that the road widths could be adjusted in the basic engineering stage of the project, without a material change to the FSU.

Figure 16-4: Typical single-lane highwall-pit haul road cross-section



Source: MMTS, 2026

Figure 16-5: Typical double-lane highwall-pit haul road cross-section



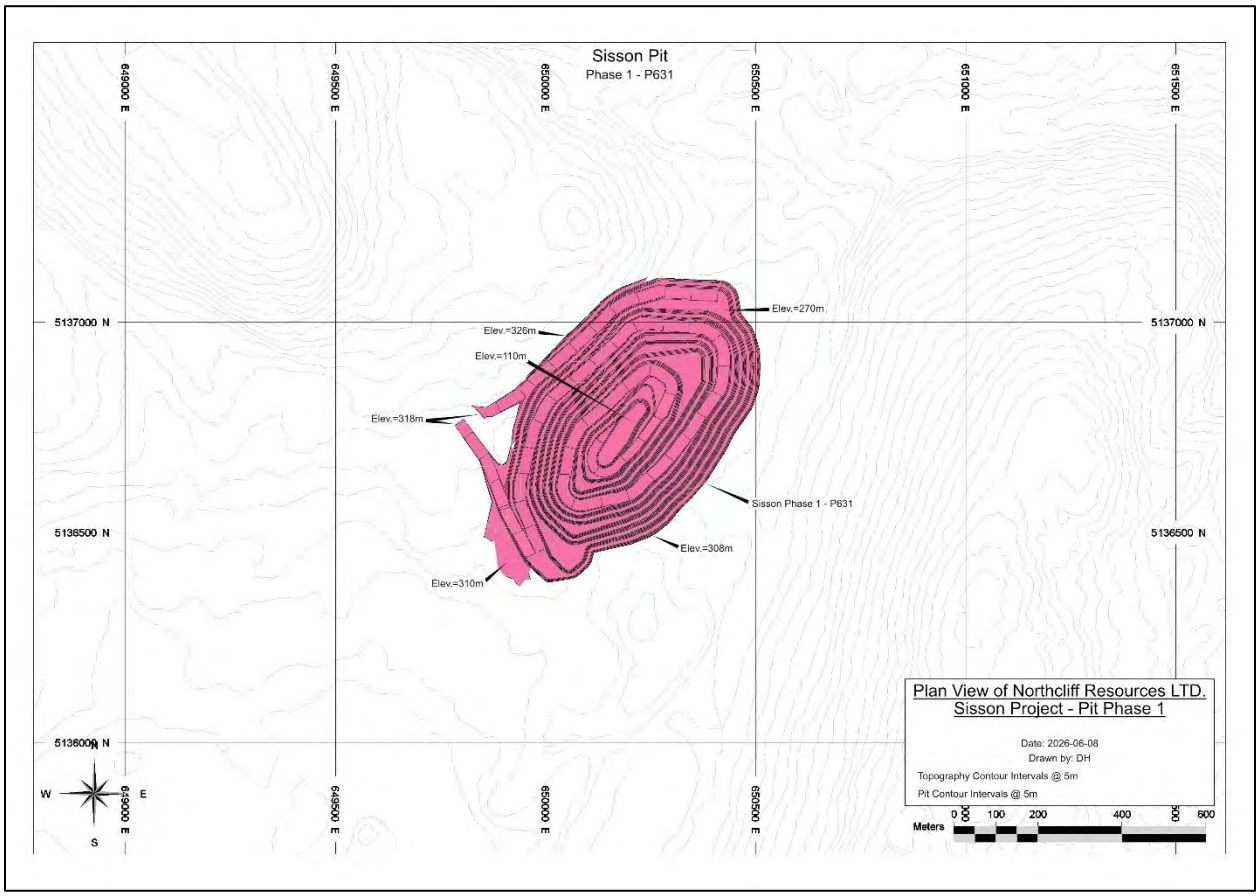
Source: MMTS, 2026

16.4.3 Results of design

The phased Sisson pit designs are shown in Figure 16-6 to Figure 16-12. Sections through the deposit showing the NSR grades are illustrated in Figure 16-13 and Figure 16-14. Where metre (m) elevations are listed below, they refer to the mine grid z-dimension elevation, which is metres above sea level.

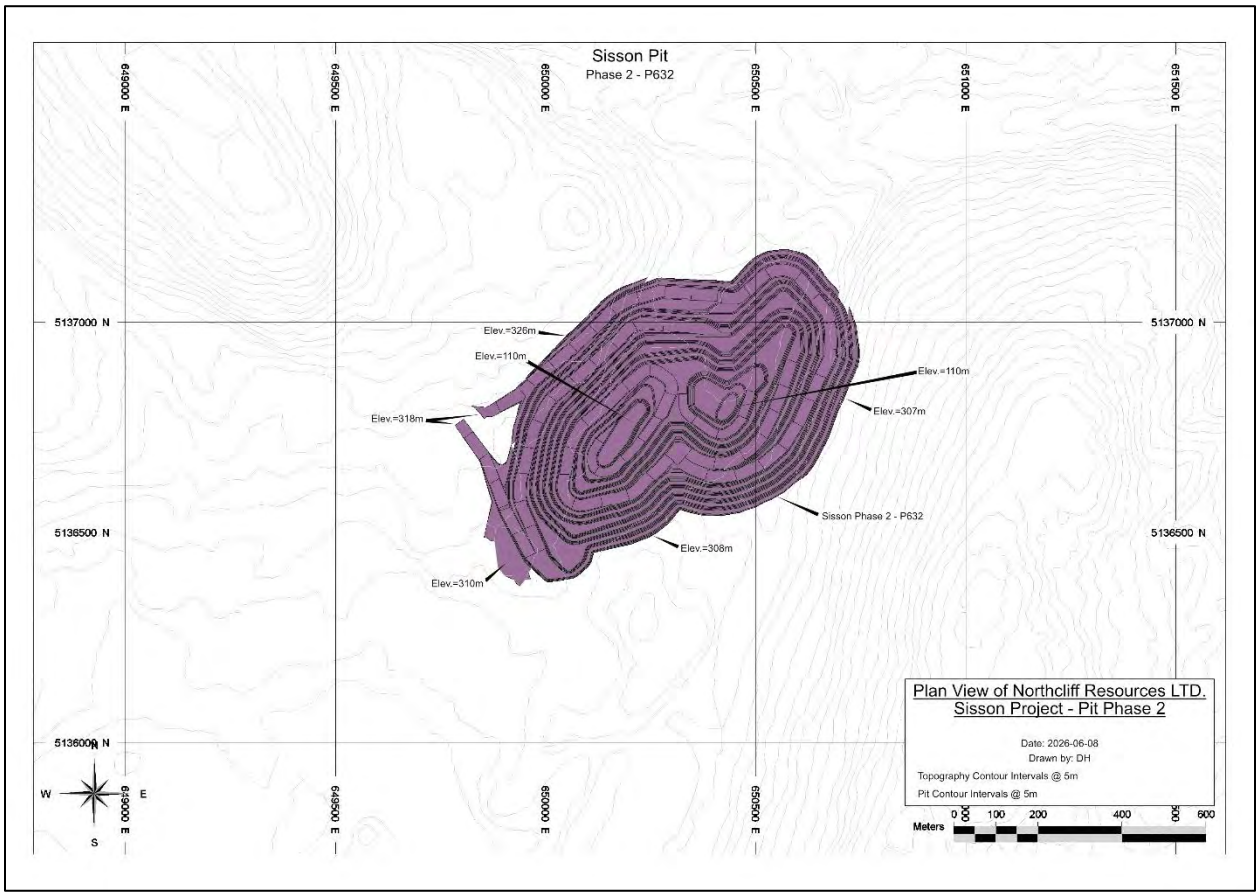
- Starter Pit, P631 (Phase 1) – This phase targets the high-grade, low-strip-ratio central portion of the deposit defined by Case PF0.5 (Figure 16-6). This phase contains about four years' worth of mill feed. The pit mines from the crest at 327 m down to the pit bottom at 110 m. An internal ramp runs counterclockwise down from the pit exit switch backing at the 290 m elevation before continuing to the second last bench of the pit. A secondary ramp in added to provide access for further pushbacks.
- Pushback, P632 (Phase 2) – This phase pushes out the starter pit to the northwest and still targets the same Case PF0.5. This phase contains about 2.5 years' worth of mill feed and mines from the pit crest at 310 m down to the pit bottom at 110 m elevation. The main ramp connects to the phase 1 ramp at the 265 m elevation, before continuing down to the 170 m elevation. This ramp also connects to the Started phase ramp at 200m where access below the 170 m bench is created.
- Pushback P633 (Phase 3) – This phase targets a low strip ratio pocket of ore directly north of Phase 2, is uses the Case PF0.7 for its limits. It is initially mined as a stand-alone pit with a ramp going clockwise from the pit crest at 297 m before switchback and continuing down to the pit bottom at 220 m elev.
- Pushback P634 (Phase 4) – This phase is another standalone pit south of Phase 1. It uses Case RF0.6 for its limits but helps even out mill grade. It mines from the pit crest at 316 m down to the pit bottom at 160 m. A ramp runs counterclockwise from the pit crest to the bottom.
- Pushback, P635 (Phase 5) – This phase is a large pushback and mines below Phase 1 and 2 and connects to Phase 4. This Phase establishes the final wall on the northwest side of the phase. The phase mines from the crest at 354 m to the bottom at 30 m. The ramp starts off clockwise before switching back at 250 m and continuing to the pit bottom.
- Pushback, P636 (Phase 6) – This phase is a large pushback to the below and north of Phase 5. It establishes final walls on the north, east and southeast sides of the pit. This phase also opens areas for backfilling of future phases. The phase mines from 340 m at the pit crest down to -30 m at the pit bottom. The ramp runs clockwise before switch backing back at the 150 m elevation. The ramp connects to various smaller segment ramps to mine different pit bottoms at -30 m, 40 m and 90 m. Phase 6 pit bottoms at -30 m and 40 m are treated as sub phases 6a and 6B respectively.
- Pushback, P637 (Phase 7) – This phase pushes out to the ultimate pit limits in the south and southwest and is the final pushback. This phases mins from the pit crest down at 322 m to two pit bottoms at 60 m and 20 m. The top half of the phase (above 150 m) uses a ramp that runs counterclockwise before switch backing at the 230 m and 200 m elevations. For benches below 150 m a secondary ramp that connects to the Phase 6 ramp at 130 m elevation is utilized.

Figure 16-6: Phase 1



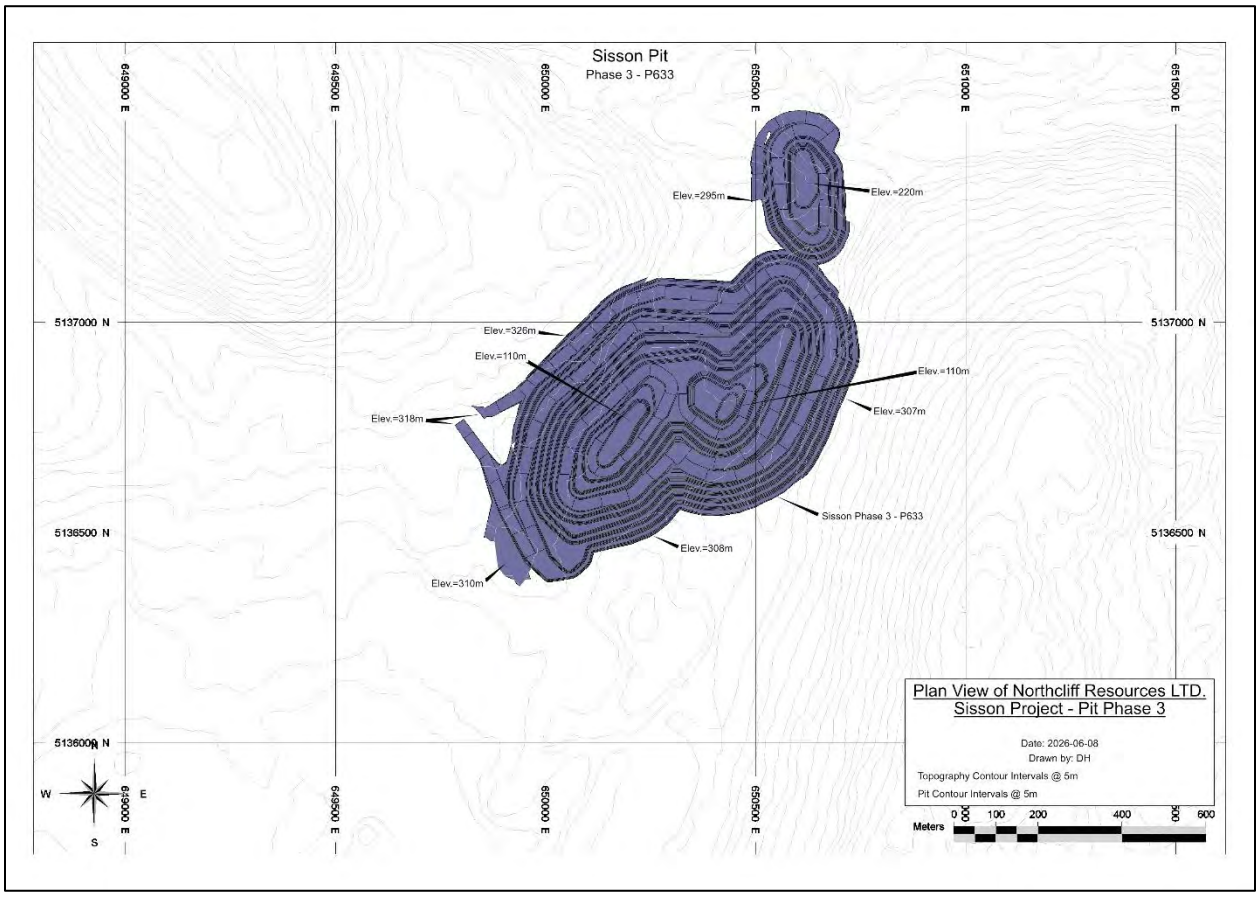
Source: MMTS, 2026

Figure 16-7: Phase 2



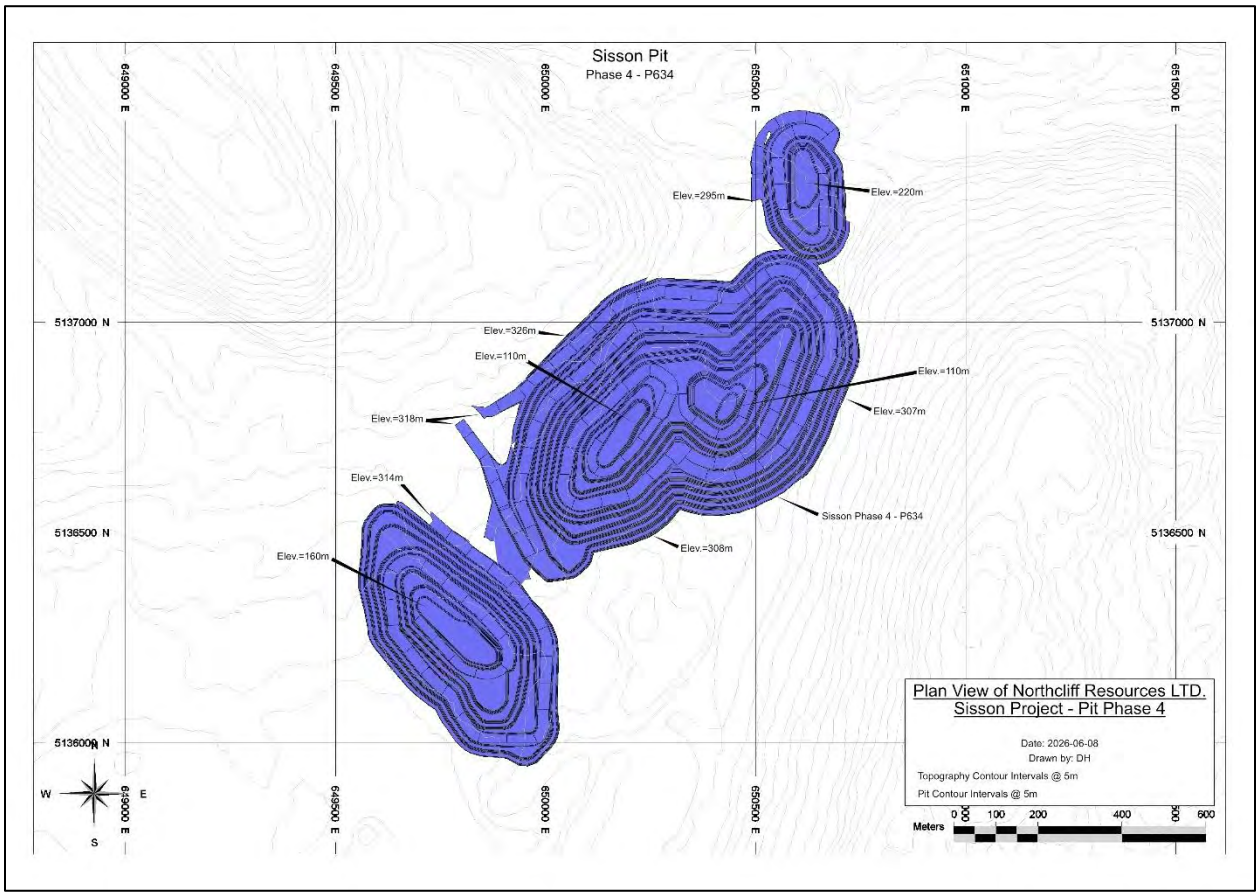
Source: MMTS, 2026

Figure 16-8: Phase 3



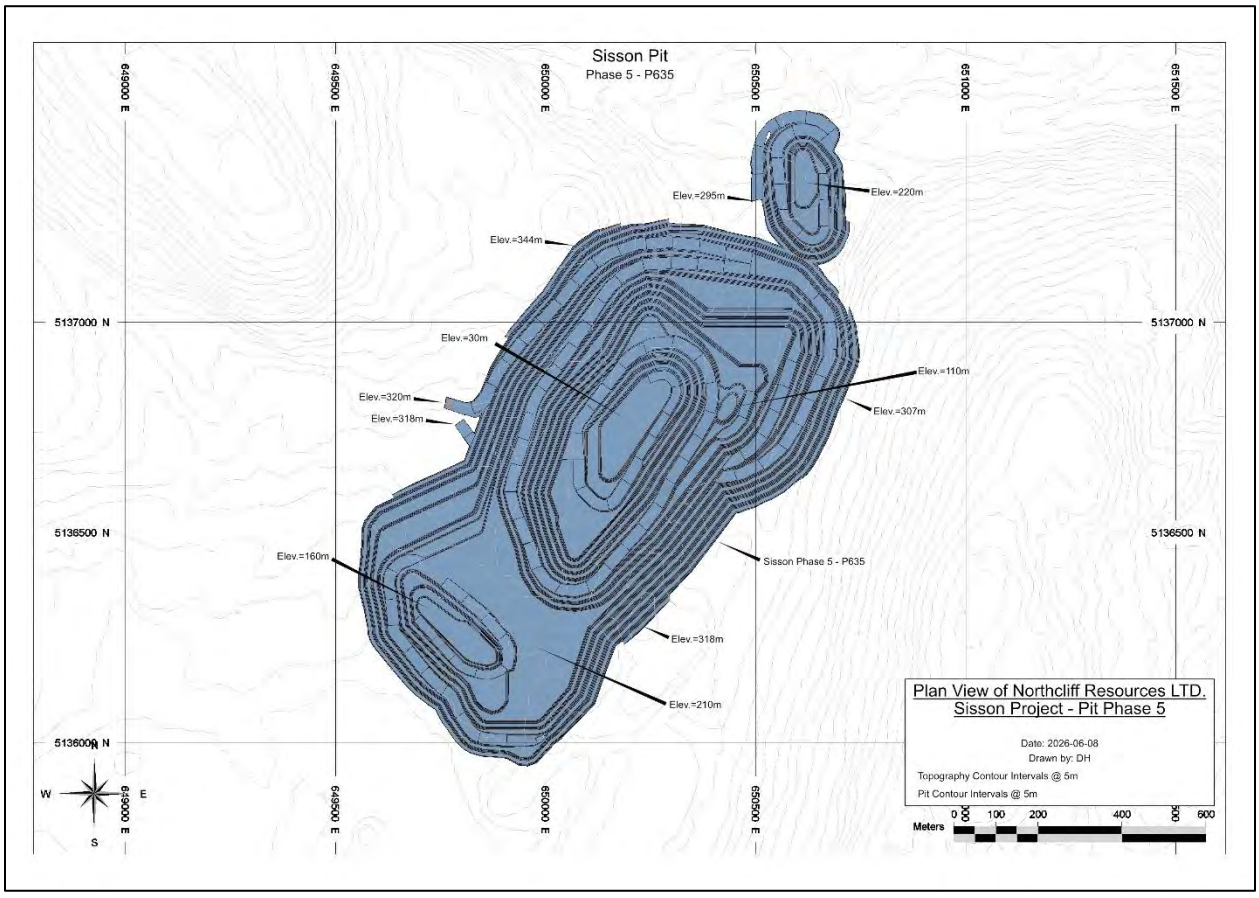
Source: MMTS, 2026

Figure 16-9: Phase 4



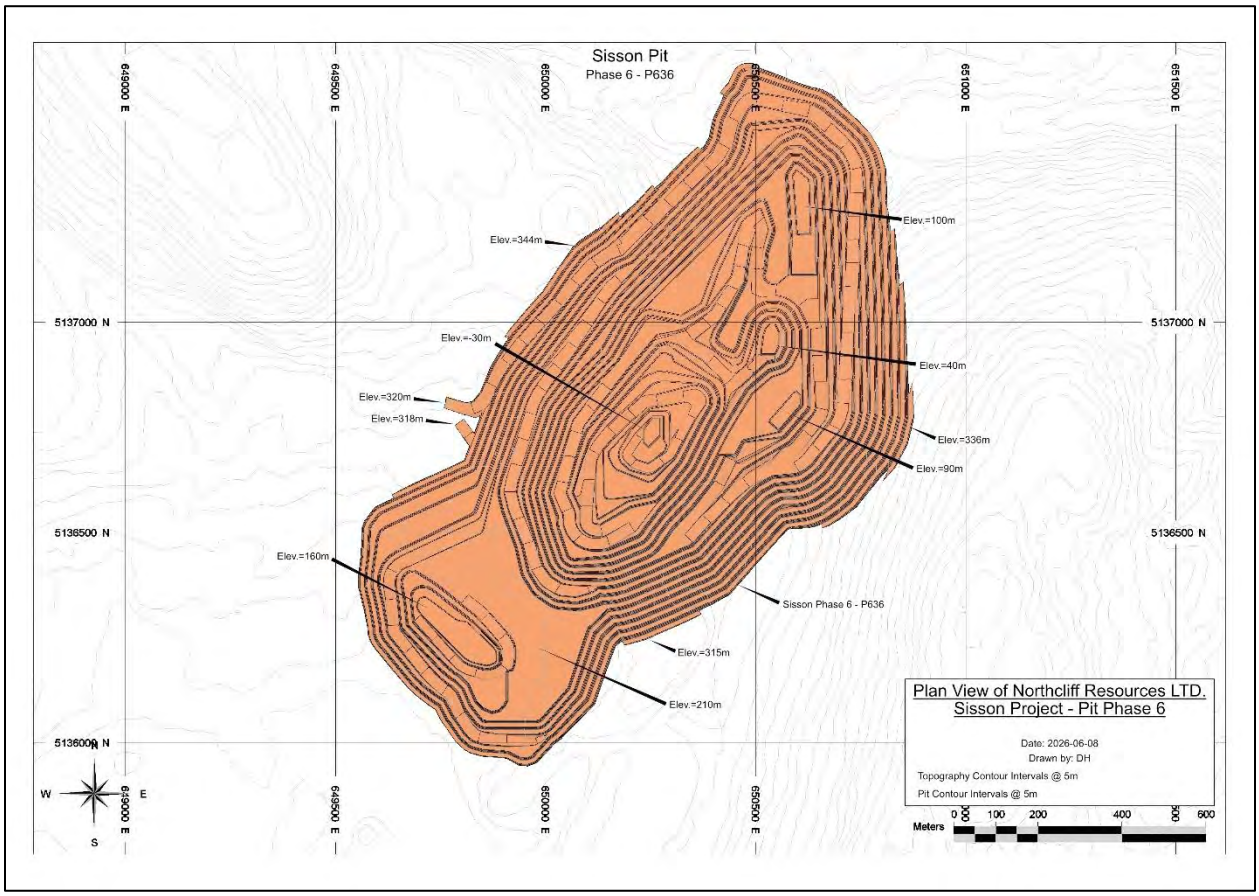
Source: MMTS, 2026

Figure 16-10: Phase 5



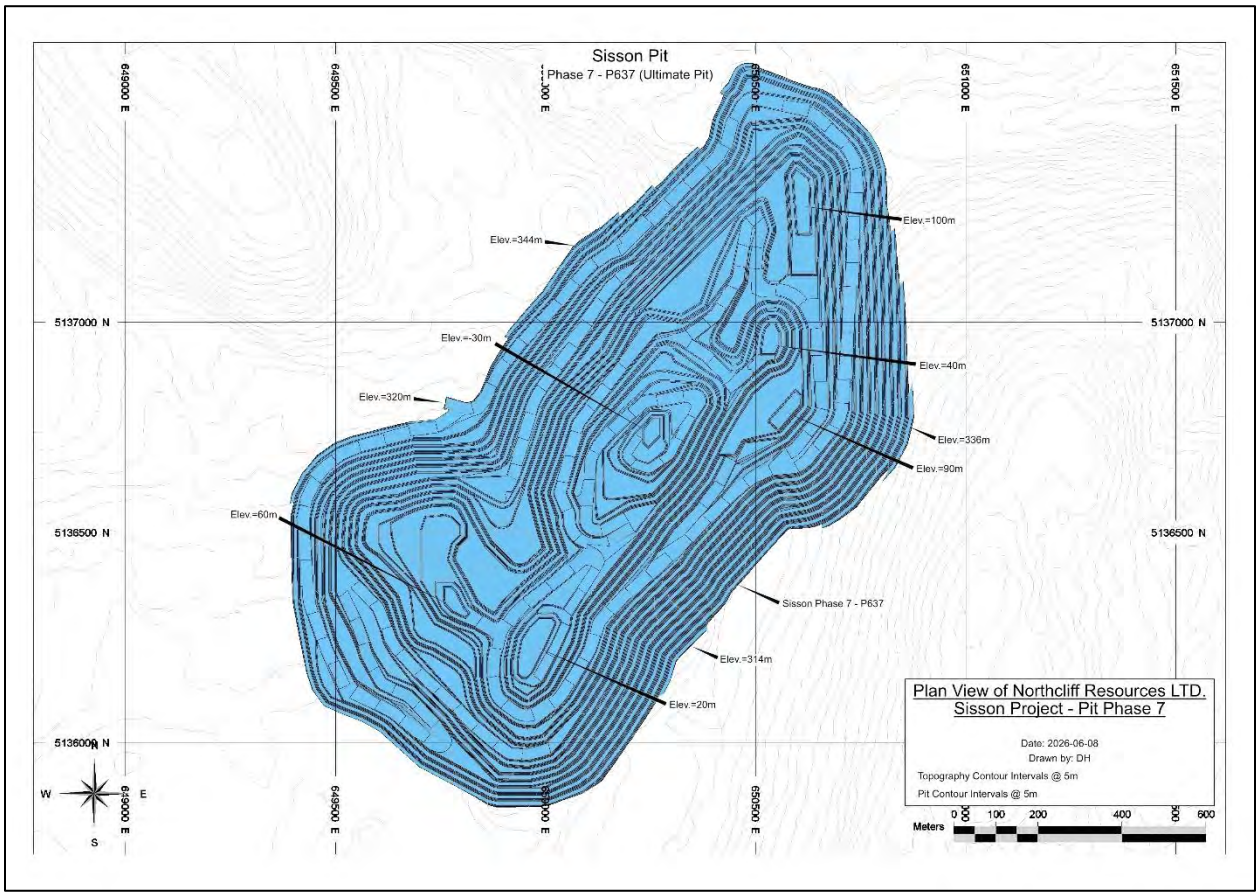
Source: MMTS, 2026

Figure 16-11: Phase 6



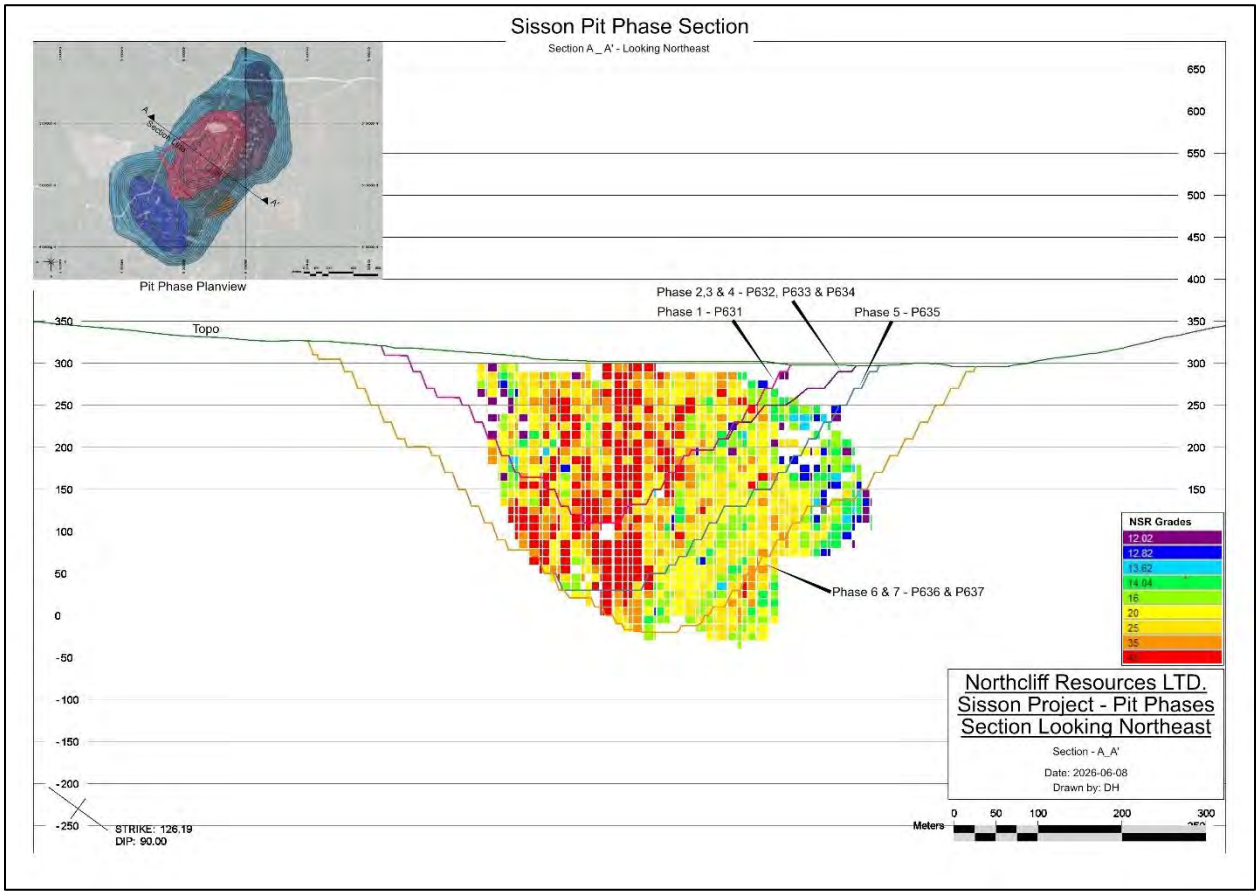
Source: MMTS, 2026

Figure 16-12: Phase 7



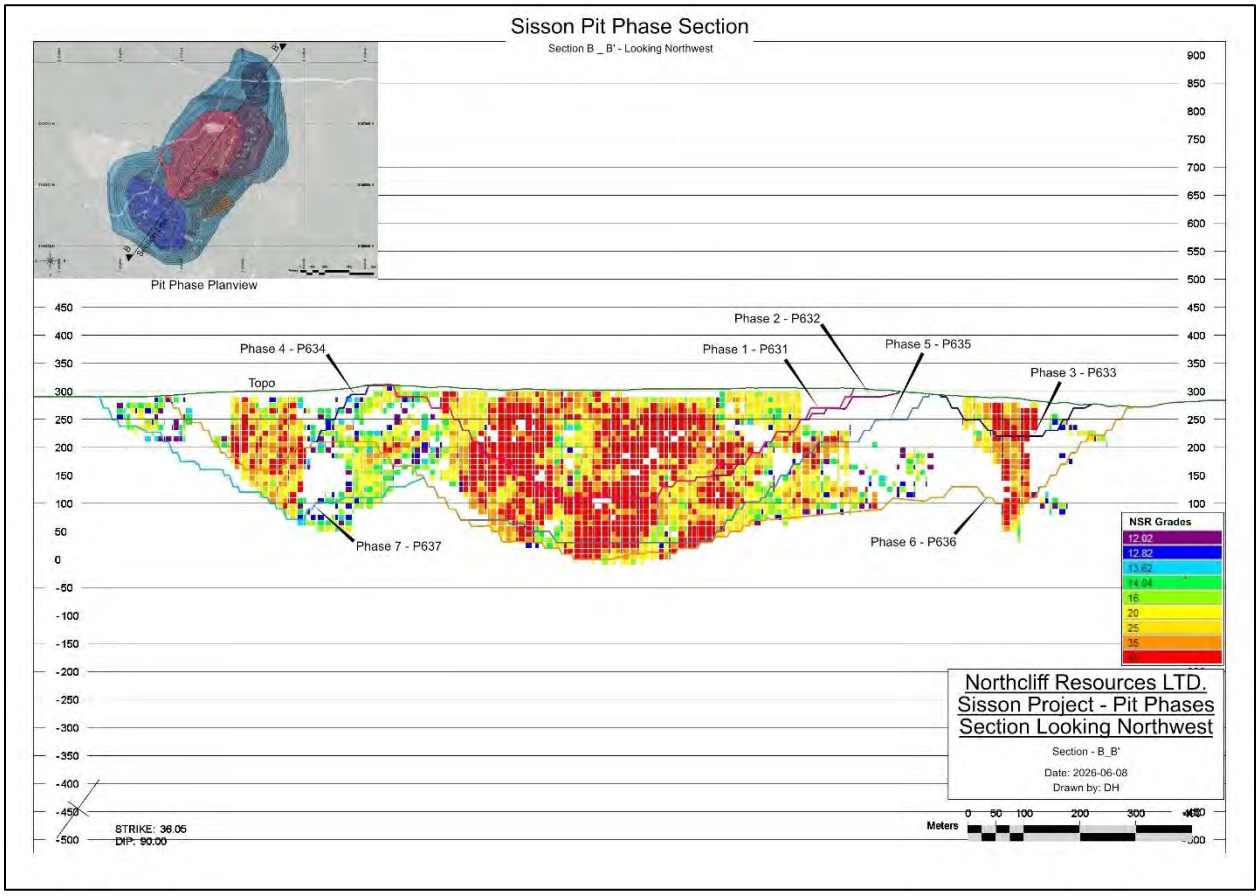
Source: MMTS, 2026

Figure 16-13: Section View 1 – Pit Phases



Source: MMTS, 2026

Figure 16-14: Section View 2 – Pit Phases



Source: MMTS, 2026

16.4.4 Pit contents by phase

The pit contents, summarized by the detailed pit design phases, are shown in Table 16-6.

Table 16-6: Pit contents by phase

Phase	Unit	PHS1	PHS2	PHS3	PHS4	PHS5	PHS6	PHS6A	PHS6B	PHS7	Total
Total Ore Mined	**kt**	**44,384**	**26,669**	**2,197**	**21,404**	**66,326**	**28,343**	**17,852**	**11,971**	**66,918**	**286,064**
NSR	$/t	40.60	41.50	42.48	36.06	34.75	32.16	29.56	36.00	28.50	34.45
WO_3%	%	0.090	0.087	0.115	0.075	0.074	0.079	0.045	0.082	0.060	0.074
Mo%	%	0.022	0.027	0.004	0.024	0.023	0.012	0.037	0.019	0.021	0.022
Mill Feed (NSR CoG varies by period)	**kt**	**39,469**	**23,868**	**1,953**	**20,195**	**66,250**	**28,343**	**17,852**	**11,971**	**66,918**	**276,820**
NSR	$/t	43.48	44.33	45.68	37.22	34.77	32.16	29.56	36.00	28.50	35.03
WO_3%	%	0.096	0.092	0.123	0.078	0.074	0.079	0.045	0.082	0.060	0.075
Mo%	%	0.023	0.028	0.004	0.025	0.023	0.012	0.037	0.019	0.021	0.023
Mid Grade Stockpile (NSR>15, NSR<MF)	**kt**	**11,435**	**5,512**	**254**	**1,232**	**75**	**-**	**-**	**-**	**-**	**18,508**
NSR	$/t	23.03	20.23	17.07	16.84	15.48	-	-	-	-	21.67
WO_3%	%	0.052	0.042	0.049	0.041	0.041	-	-	-	-	0.048
Mo%	%	0.016	0.019	0.005	0.012	0.009	-	-	-	-	0.016
Mid Grade Reclaim (NSR>15, NSR<MF)	**kt**	**6,521**	**2,711**	**10**	**22**	**-**	**-**	**-**	**-**	**-**	**9,263**
NSR	$/t	27.18	23.17	21.17	21.09	-	-	-	-	-	25.39
WO_3%	%	0.061	0.046	0.060	0.045	-	-	-	-	-	0.055
Mo%	%	0.017	0.022	0.004	0.018	-	-	-	-	-	0.018
Total Barren Rock Mined	**kt**	**22,090**	**10,828**	**3,285**	**13,041**	**68,575**	**73,826**	**6,266**	**7,442**	**72,657**	**278,010**
Strip Ratio (Barren Rock/Ore Mined)		**0.50**	**0.41**	**1.50**	**0.61**	**1.03**	**2.60**	**0.35**	**0.62**	**1.09**	**0.97**
Effective Strip Ratio ([BR+MG]/MF)		**0.85**	**0.68**	**1.81**	**0.71**	**1.04**	**2.60**	**0.35**	**0.62**	**1.09**	**1.07**
Total Pit Material Mined	**kt**	**66,473**	**37,497**	**5,482**	**34,446**	**134,901**	**102,168**	**24,118**	**19,413**	**139,575**	**564,074**

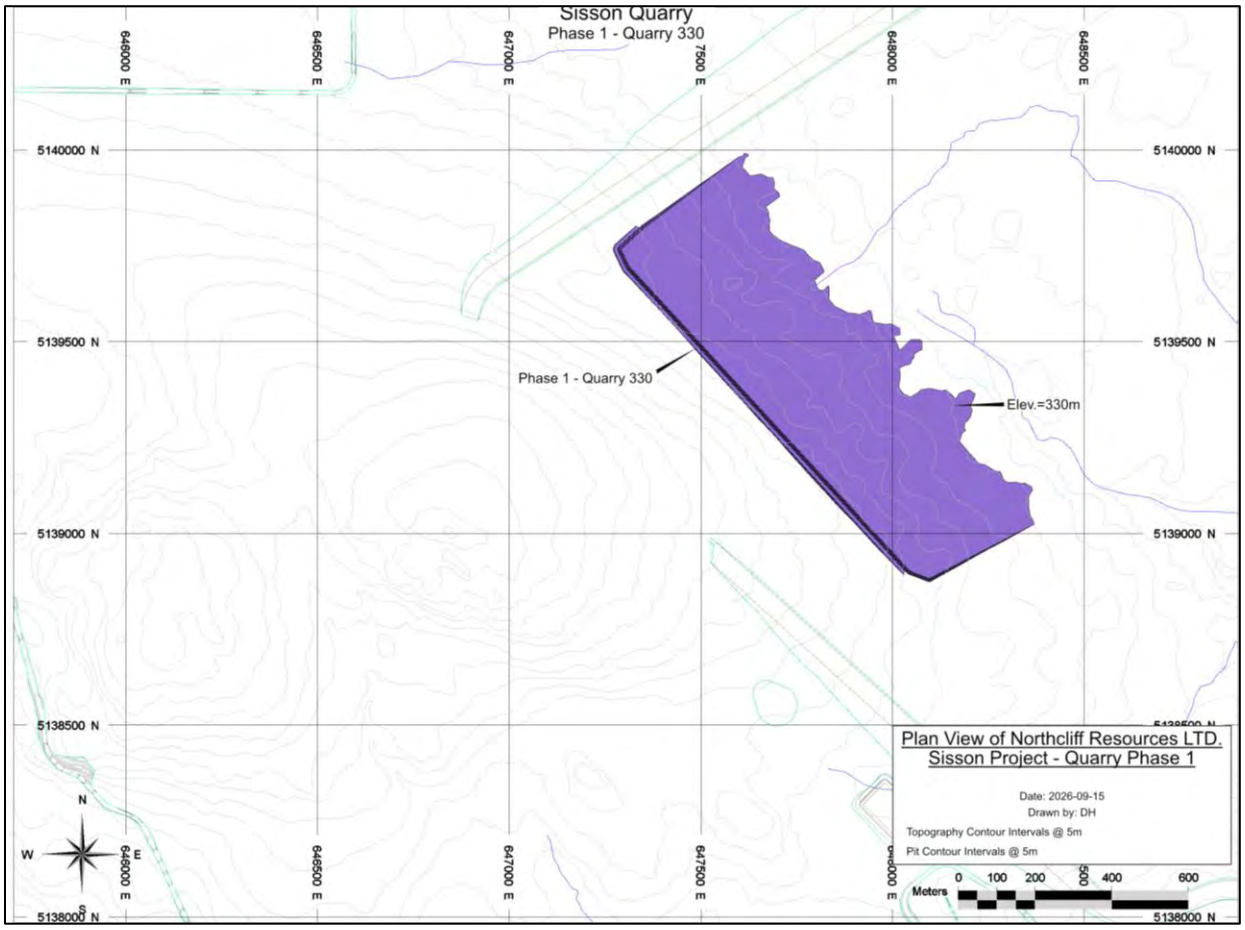
16.5 Quarry design

Due to the presence of potentially acid-generating (PAG) barren rock within the open pit, most of the TSF embankment cannot be constructed using material from the open pit. As such, material for tailings embankment construction will be sourced from a separate, non-PAG quarry site. The quarry will also be responsible for providing stemming material, rock for haul road construction and crushed rock for road and site maintenance.

The quarry will be located within the footprint of the TSF to minimize surface disturbance and is therefore designed as a tiered, 4-phase design to accommodate the tailings filling schedule. The design also allows for the future operations to direct any surface run-off into the TSF and not impact any other drainage areas. Detailed quarry designs use the same design parameters as the open pit. Phases 1 to 4 are shown below:

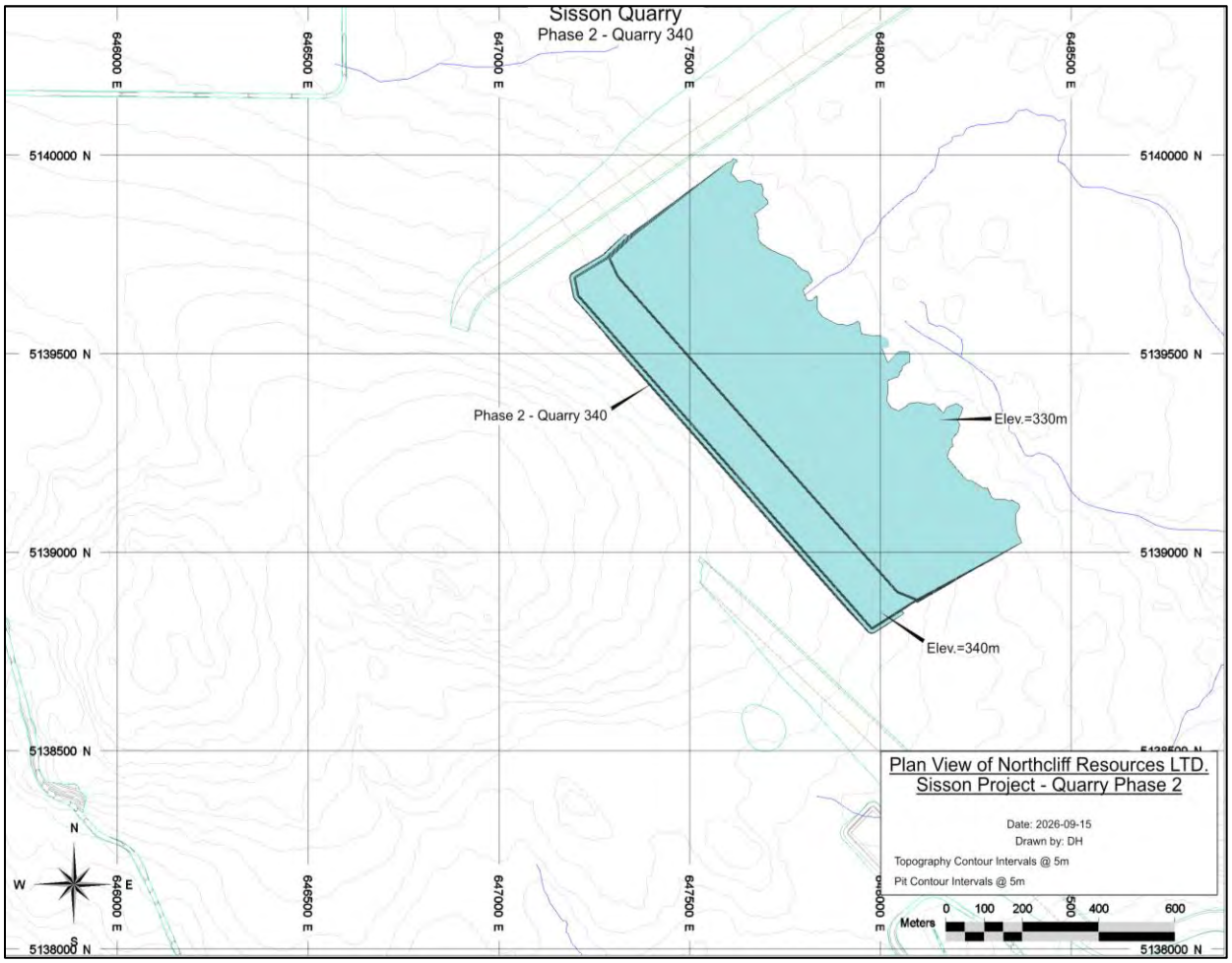
- Starter Quarry Phase 1 contains approximately 14 Mt of material and is intended to provide early quarried rock for embankment construction, mining from surface (approximately 360 m) down to 330 m. While more volume could be made available by mining to a lower elevation, this would reduce the window of available time to mine Phase 1 due to the rapid advancement of the tailings elevation during the early years of the mine life. Phase 1 will be completely filled by tailings around Year 2.
- Pushback Quarry, Phase 2 mines approximately 6 Mt from surface (approximately 360 m) down to 340 m. This provides an additional 20 m of elevation for quarry operations to remain ahead of the tailings elevation. Phase 2 will be completely filled by tailings around Year 4.
- Pushback Quarry, Phase 3 mines approximately 10 Mt from surface (approximately 385 m) down to 350 m. This provides an additional 10 m of elevation for quarry operations to remain ahead of the tailings elevation. Phase 2 will be completely filled by tailings around Year 5.
- Upper Quarry, Phase 4 mines the hilltop down from surface (approximately 410 m) to 360 m. There is approximately 66Mt of material in this phase. The footprint for this phase was designed to reduce its impact on natural drainage as much as possible and is located outside of the TSF. Mining of Phase 4 will occur when tailings have already exceeded 350 m, as such, a minimum set back of 100 m between Phase 4 and Phase 3 is observed to reduce the risk of seepage into Phase 4. At mine closure, a channel can be cut in this 100 m setback to facilitate flooding of Phase 4 of the quarry if required.

Figure 16-15: Quarry Phase 1



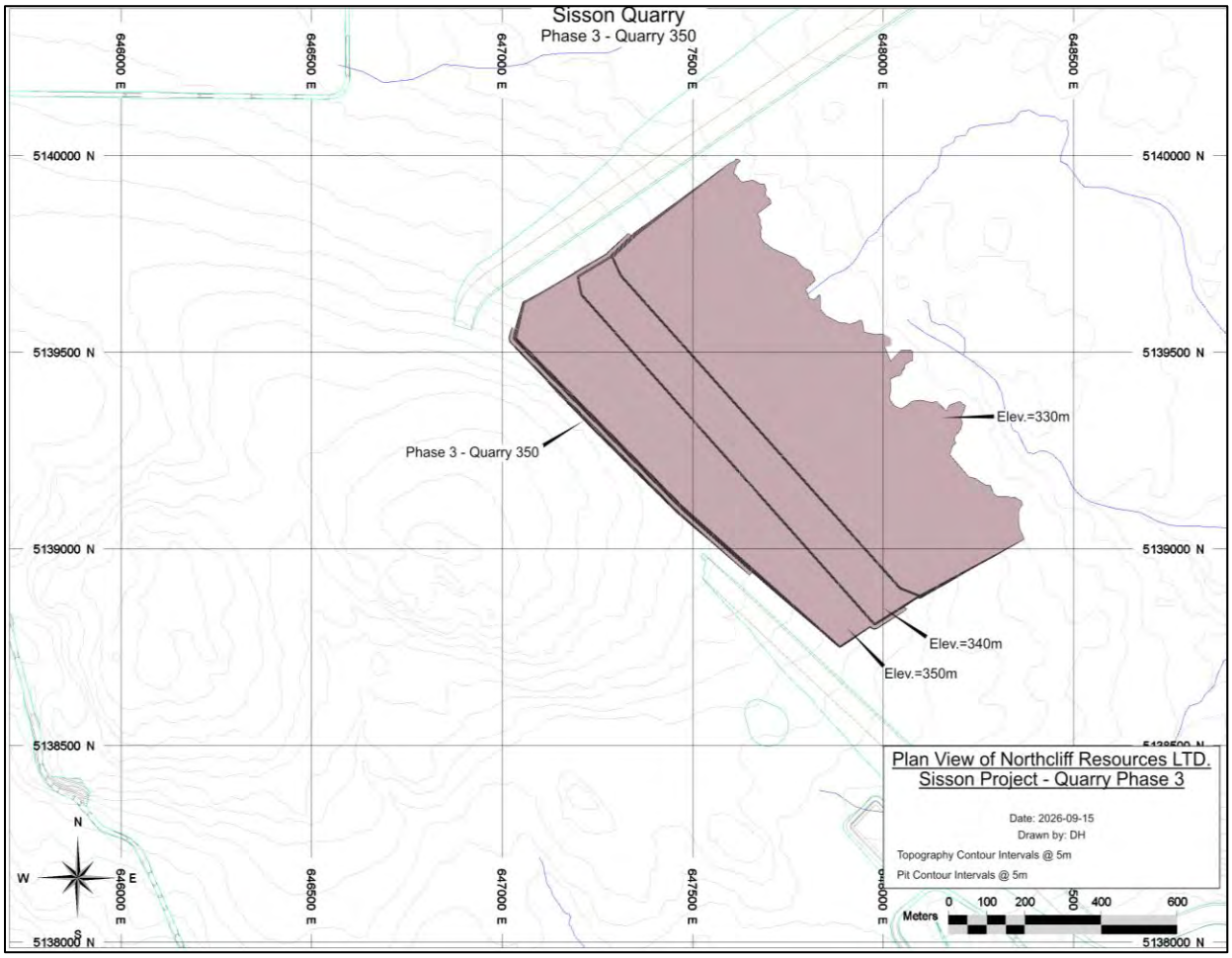
Source: MMTS, 2026

Figure 16-16: Quarry Phase 2



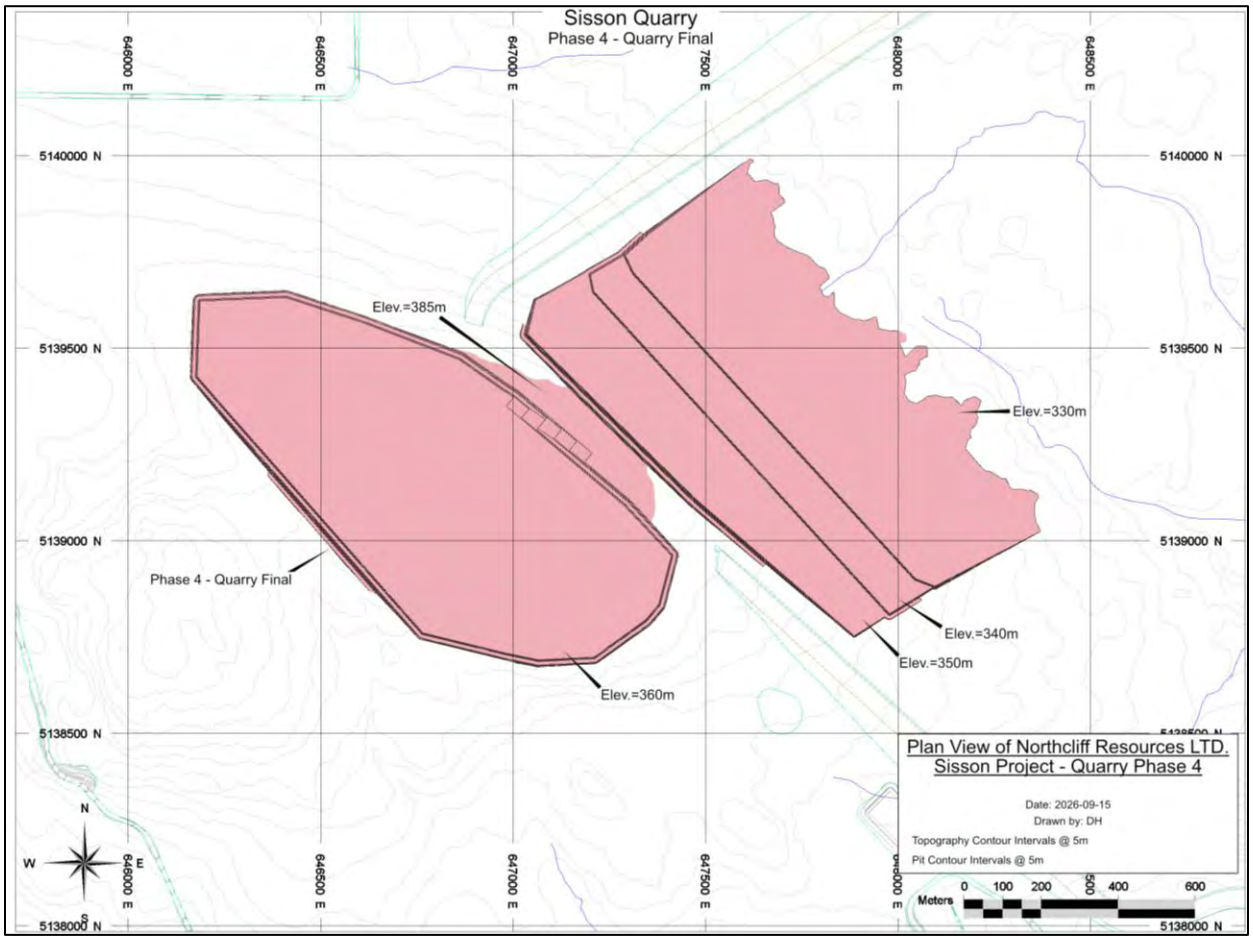
Source: MMTS, 2026

Figure 16-17: Quarry Phase 3



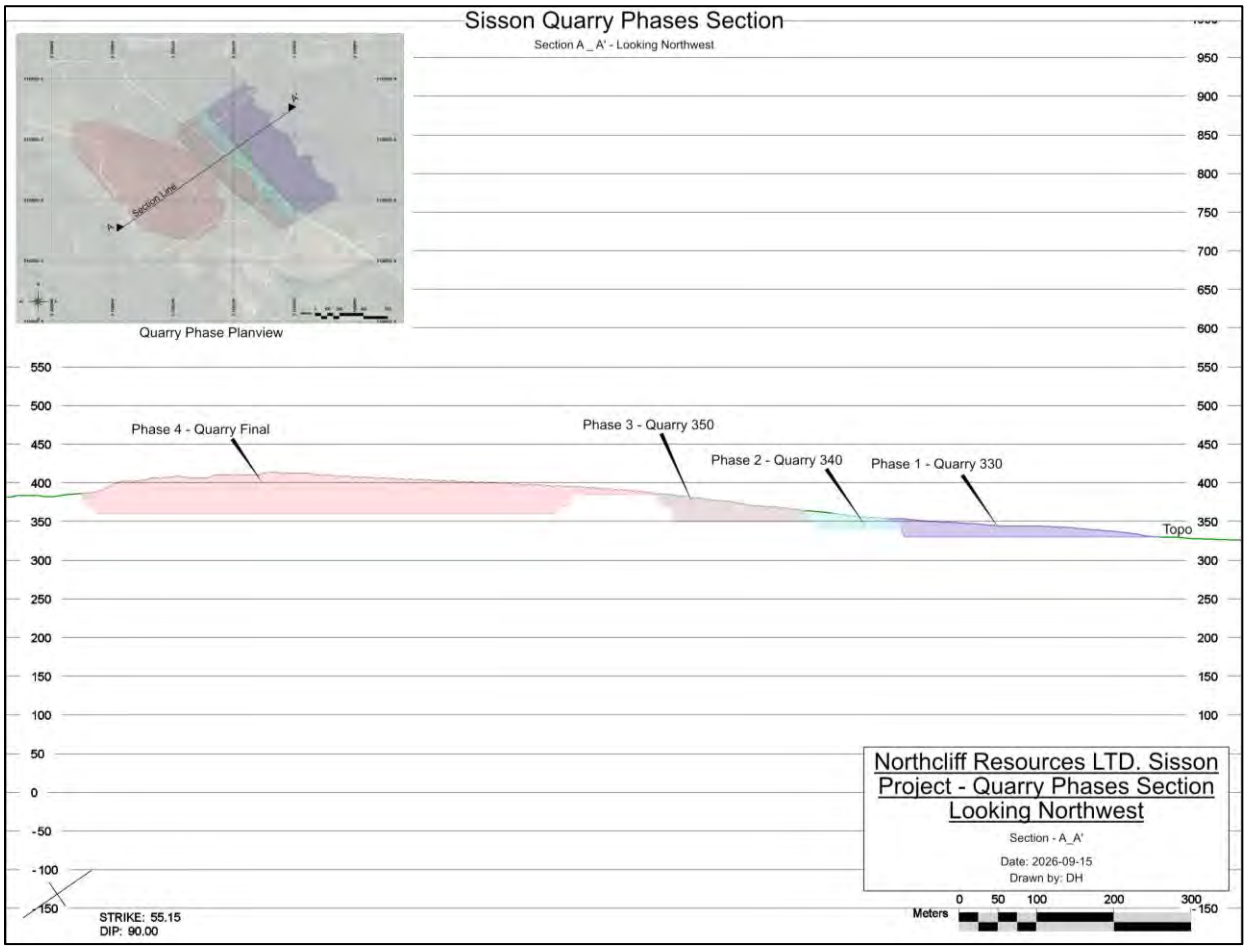
Source: MMTS, 2026

Figure 16-18: Quarry Phase 4



Source: MMTS, 2026

Figure 16-19: Section View 1 - Quarry Phases



Source: MMTS, 2026

16.6 Mine infrastructure

Major infrastructure that supports open-pit operations and that is primarily constructed by the mine fleet, is discussed below. Additional support infrastructure such as site access roads, truck shop, maintenance facilities, fuel islands, etc. are discussed in Section 18.

16.6.1 Ex-pit haul roads

Mine haul roads external to the open pits are designed to haul ore and barren rock materials from the open pits to the scheduled destinations. The mine haul roads are designed with the following key inputs:

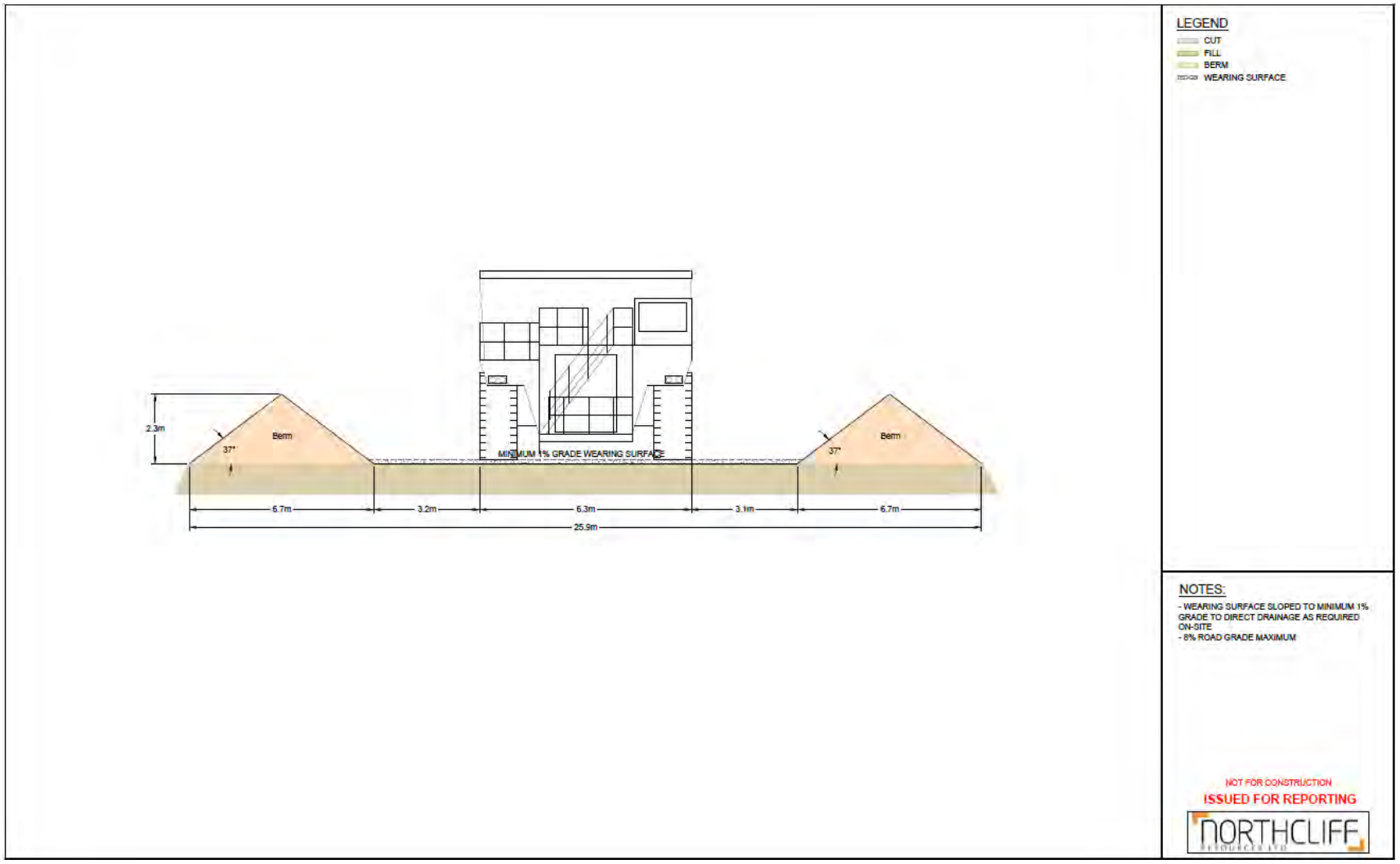
- 32 m wide ex-pit haul roads that incorporate a dual-lane running width and berms on both edges of the haul road, see Figure 16-21.
- 26 m wide ex-pit haul roads are used where single lane roads are required, see Figure 16-20.
- Sized to handle 135 t payload rigid-frame haul trucks.
- Typically, 8% maximum grade, with some short sections of 10% maximum grade.

The designed ex-pit haul roads are shown as highlighted surfaces in Figure 16-22. Timing for construction, as well as estimated cut and fill volumes are presented in Table 16-7.

Haul road routes will be initially pioneered by dozers. The roads will be expanded into full width mine haul roads by means of end-dumping with suitable quarry rock.

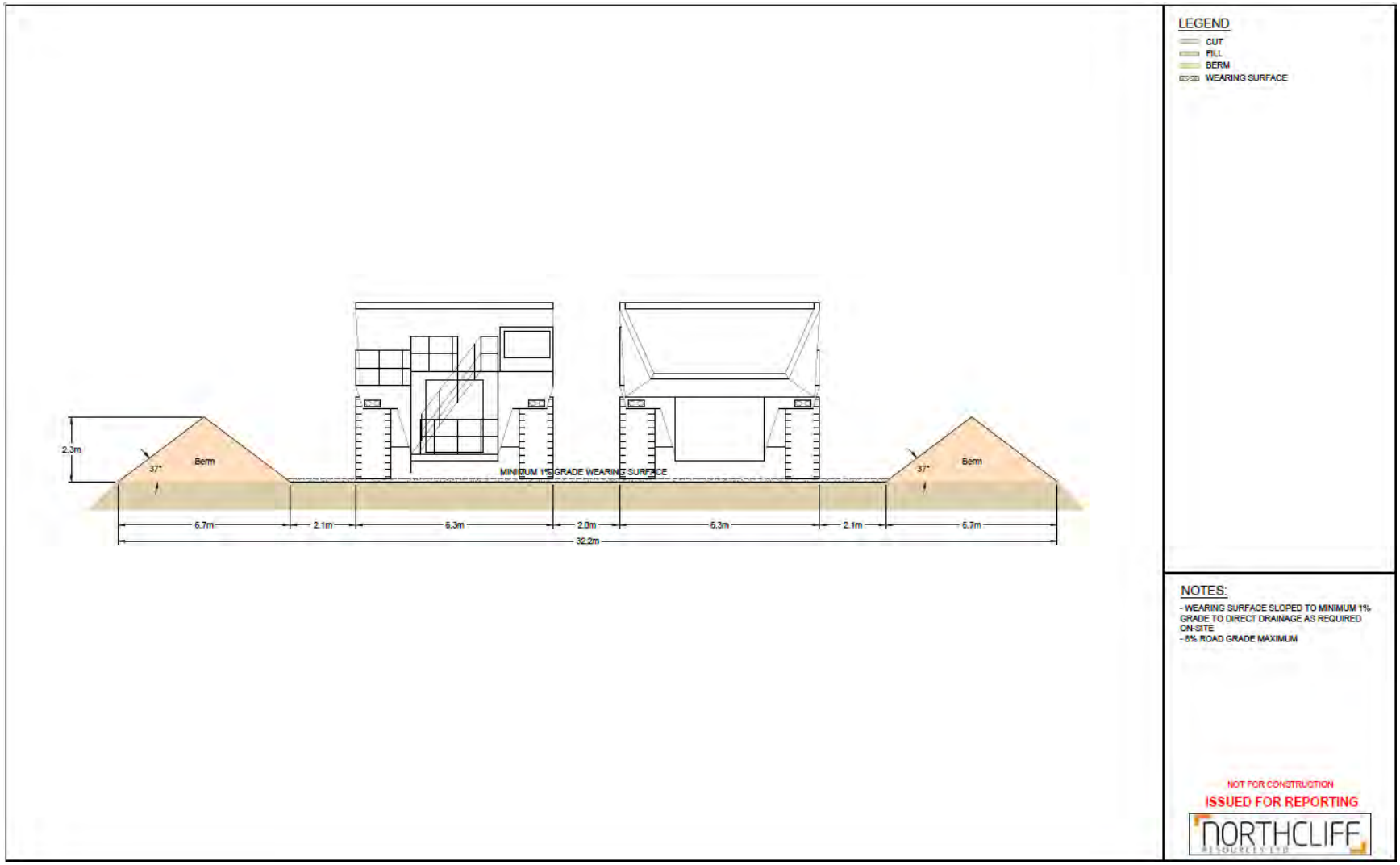
Like the in-pit haul roads, the 135 t hauler has increased width and capacity across most vendors since the external roads were designed. This has changed the largest vehicle width from 6.3 m to 7.6 m, which in turn caused the width of the roads to be increased. It was determined by Northcliff that the road widths could be adjusted in the basic engineering stage of the project, without a material change to the FSU.

Figure 16-20: Typical single-lane external-pit haul road cross-section



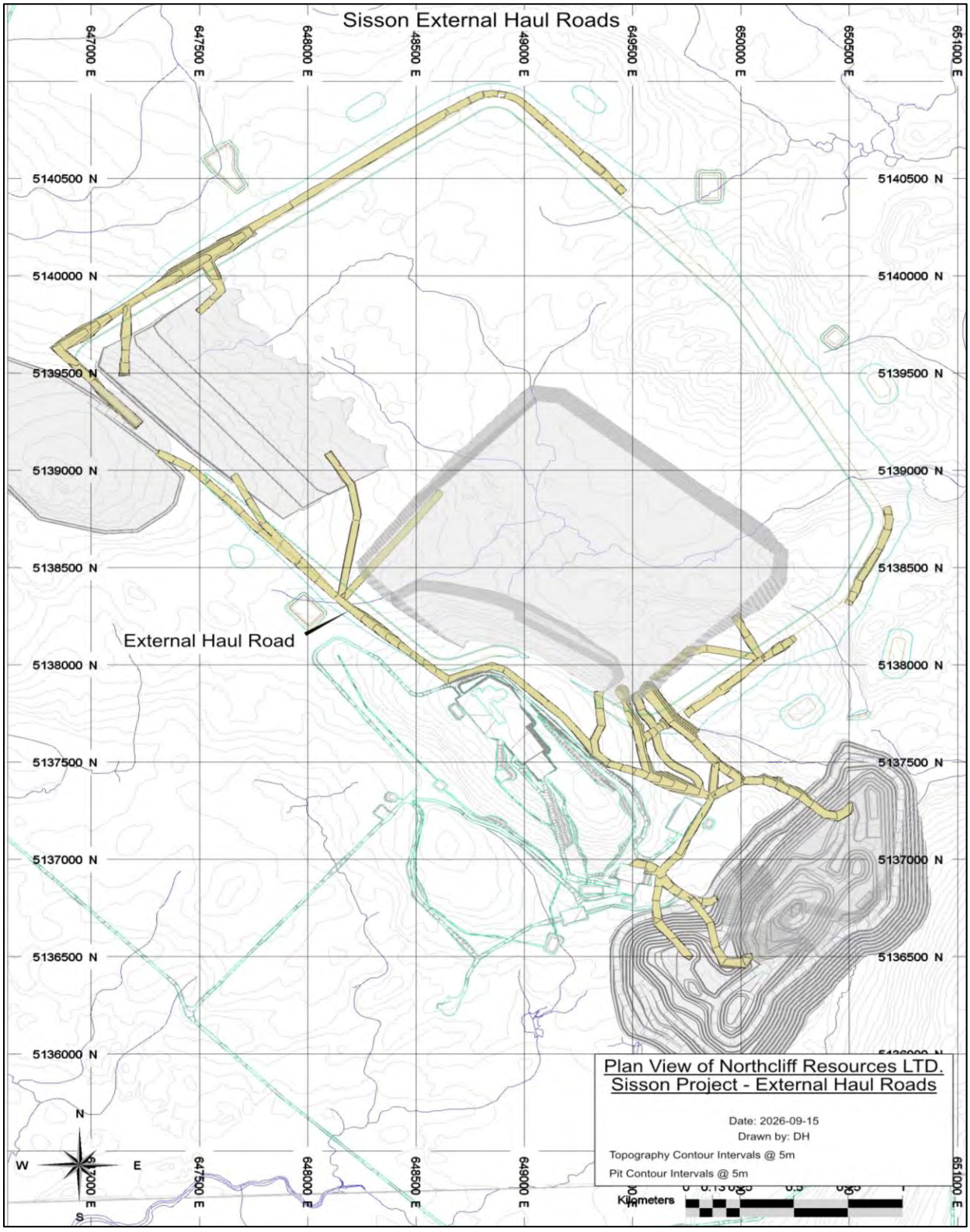
Source: MMTS, 2026

Figure 16-21: Typical double-lane external-pit haul road cross-section



Source: MMTS, 2026

Figure 16-22: Ex-pit mine haul roads



Source: MMTS, 2026

Table 16-7: Ex-pit haul road details

Year Required	Cut Volume (m^3 x 1,000)	Fill Volume (m^3 x 1,000)	Comments
Y-2	102,000	185,000	Pit to Crusher and BRSA (including Quarry road)
Y-1	25,000	132,000	Pit to BRSA/MGSP Area
Y3	7,000	50,000	Connecting road to BRSA lifts
Y7	37,000	34,000	Connecting road to Pit

16.6.2 Ore storage areas and stockpiles

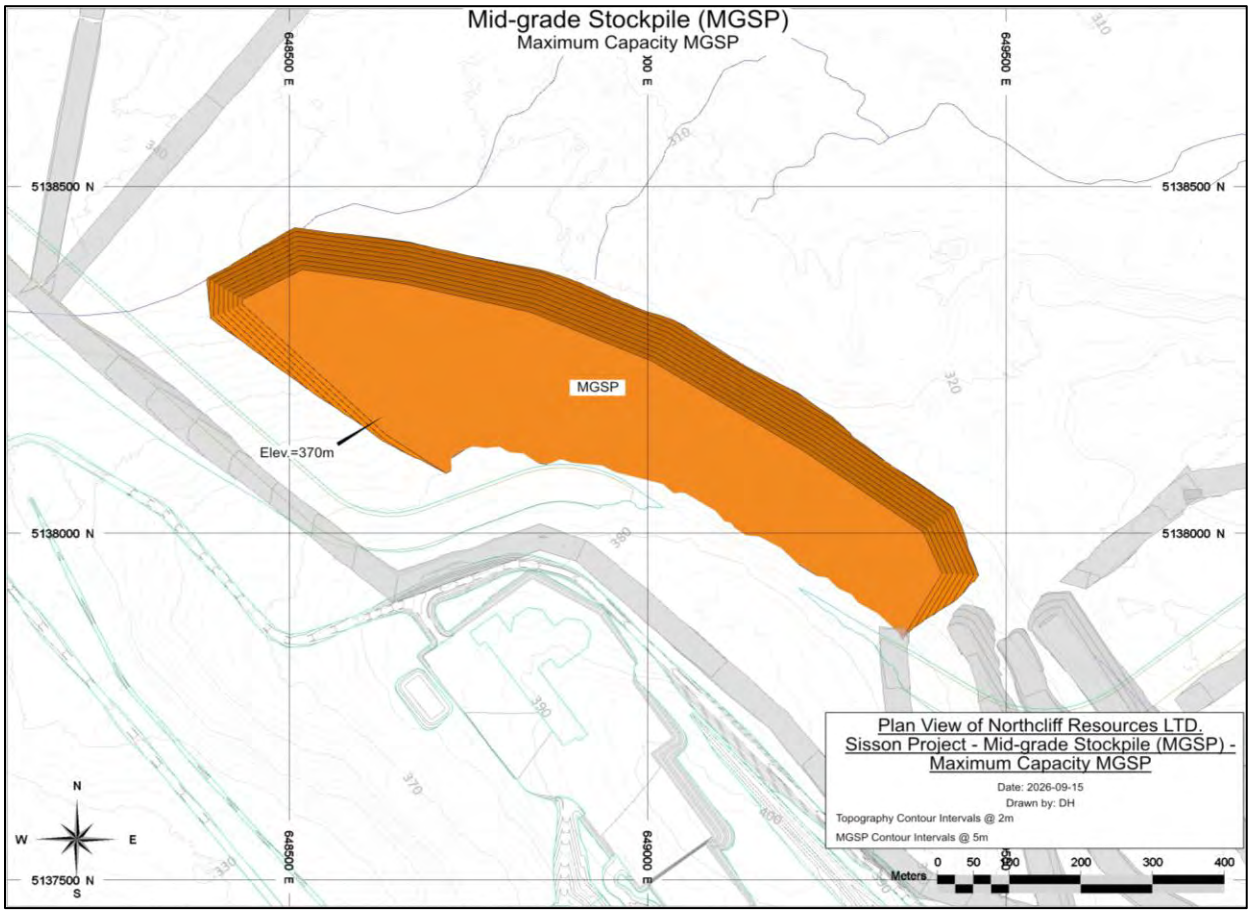
Due to the presence of PAG rock within the open pit, no external ore storage areas are considered. A mid-grade stockpile (MGSP), as shown in Figure 16-23, below, is operated within the footprint of the TSF from pre-production through to year 9 of production. After this time, the MGSP becomes challenging to operate as the rising tailings pond elevation overtakes the stockpile reclaim area (see End of Period Map, Year 7, Figure 16-40).

The maximum capacity of the MGSP is 10 Mt, which is reached in Year 3 of production. Material for the MGSP is end-dumped off the north side of the plant site hill at an elevation of 370 masl. Stockpile reclaim typically occurs at the 340 m elevation.

The average NSR grade of material in the MGSP is $17.34. By the time the rising tailings pond elevation renders the MGSP inoperable, 9.2 Mt of material becomes encapsulated beneath the tailings, with no provisions in the current mine plan for future reclaim.

The maximum capacity design MGSP is shown in Figure 16-23.

Figure 16-23: MGSP maximum capacity design



Source: MMTS, 2026

16.6.3 Barren rock storage areas

Material mined from the open pit that does not qualify as ore is designated as barren rock. Barren rock may contain tungsten and/or molybdenum mineralization; however, these grades are insufficient to meet mill feed cut-offs for a given period. Due to the presence of PAG material within the open pit, a barren rock storage area (BRSA) is designated within the footprint of the TSF. As the TSF fills with tailings over the mine life, the BRSA will be progressively encapsulated within the tailings.

The BRSA is constructed in a series of lifts, as illustrated in Figure 16-24 through Figure 16-30. This design methodology ensures that no barren rock is end-dumped directly into the tailings. End-dumping into tailings poses a slope stability risk for the BRSA crests and limits storage capacity due to the volumetric unpredictability of placing rock over tailings.

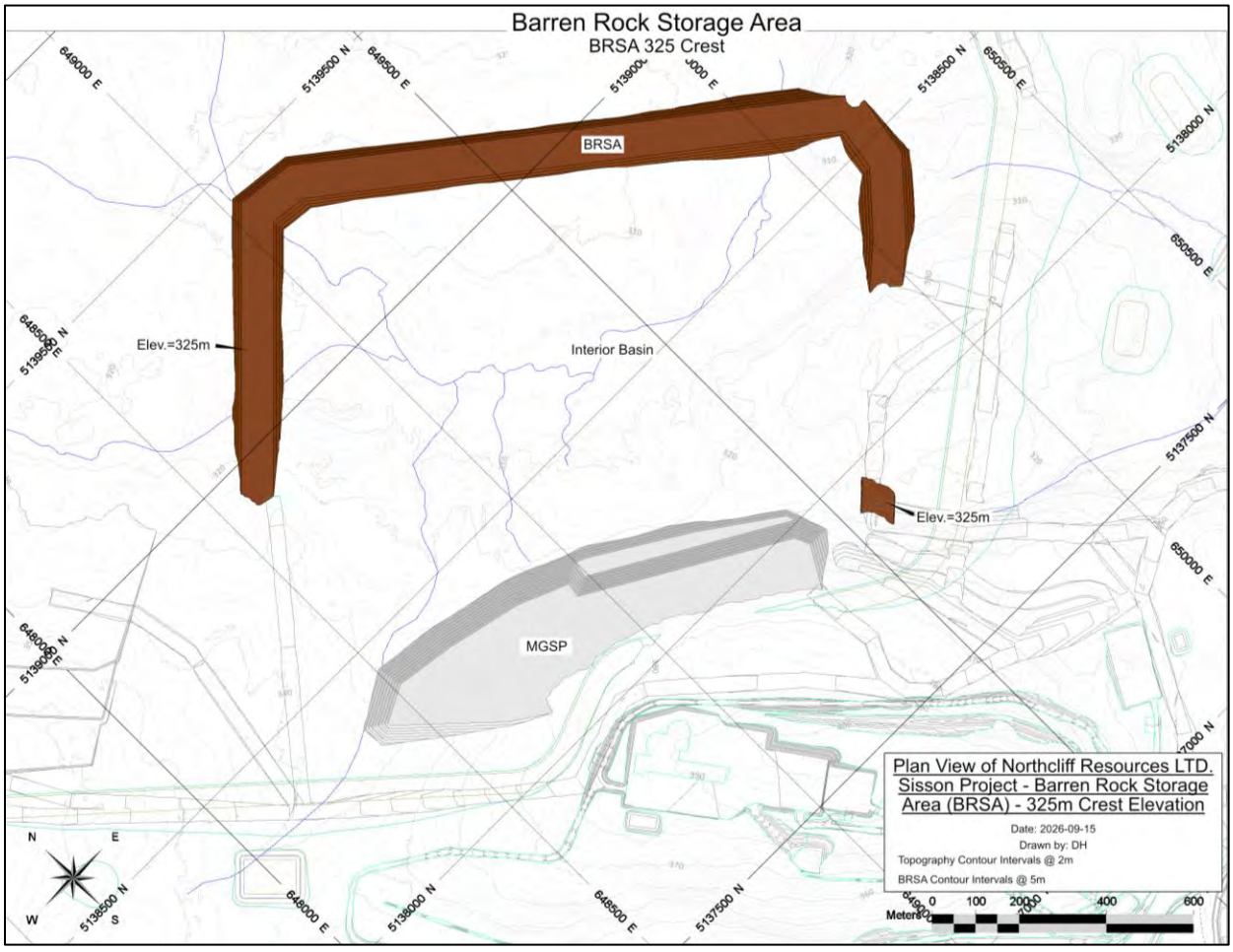
By constructing the lower lifts of the BRSA using a "causeway" design, water can filter through the rock and fill the interior basin while the tailings remain confined to the exterior side. To ensure safe operability, these causeway lifts should consistently remain 2 to 3 m above the active tailings surface. As each successive lift is constructed, the width of the causeway increases. On the exterior edge, the crest is actively managed to prevent significant end-dumping into the tailings. Conversely, end-dumping into water is permissible on the interior side. This allows the BRSA causeway to expand inward with each lift, eventually filling the interior basin completely.

Once the 360-elevation lift is reached, the final two lifts transition to conventional bottom-up construction. By Year 20, the tailings elevation becomes too high to safely operate the BRSA. However, by this stage of the mine plan, areas within the open pit will have been mined to their final floor, allowing direct backfilling to commence and eliminating the need for further BRSA activity.

Design parameters for the BRSA include:

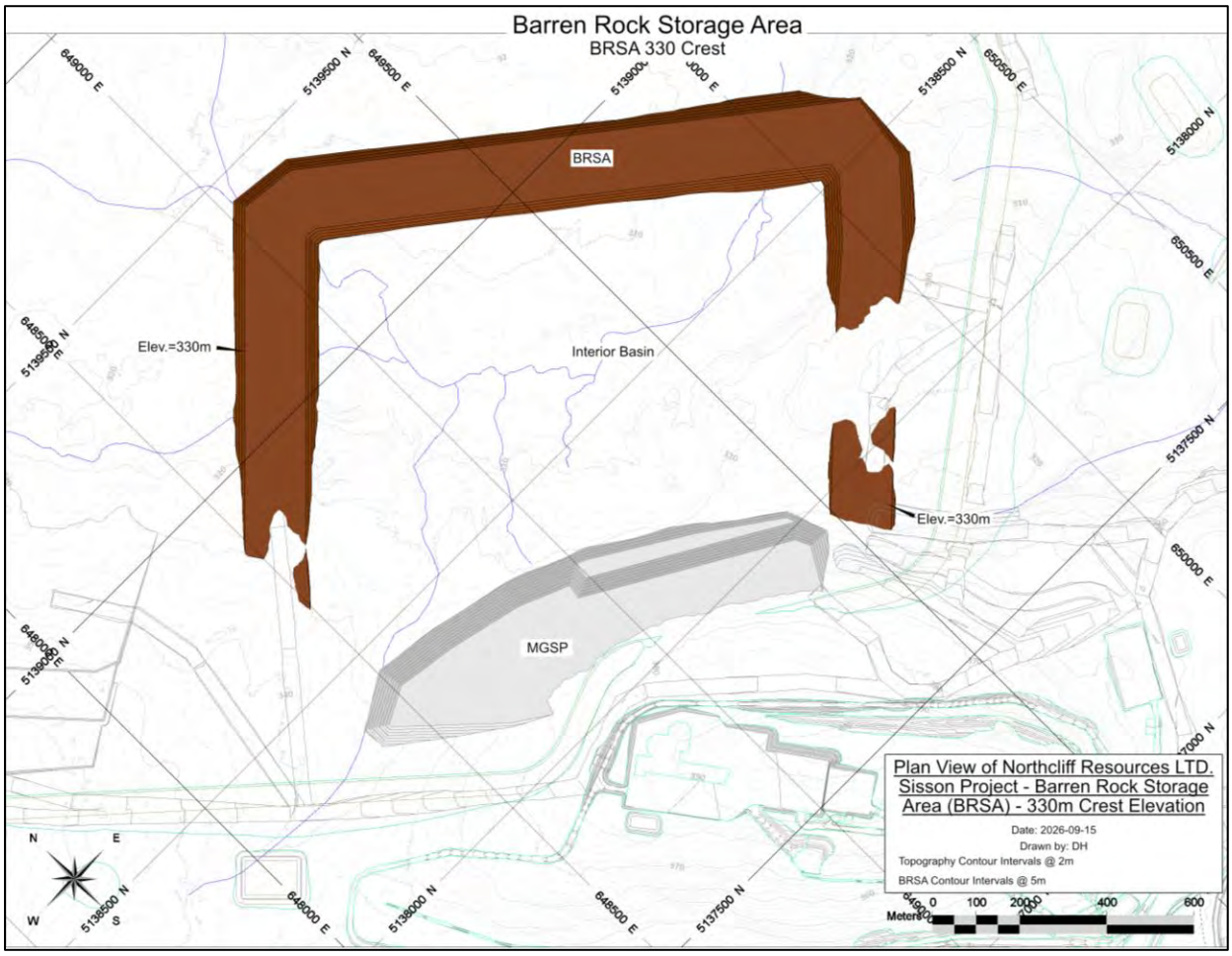
- Bottom-up lift construction exclusively on the exterior crest
- End-dump construction permitted on the interior crest
- Six 10 m lifts
- 37° overall slope angle
- 30% swell factor for dumped barren rock

Figure 16-24: BRSA Design 325 m crest elevation



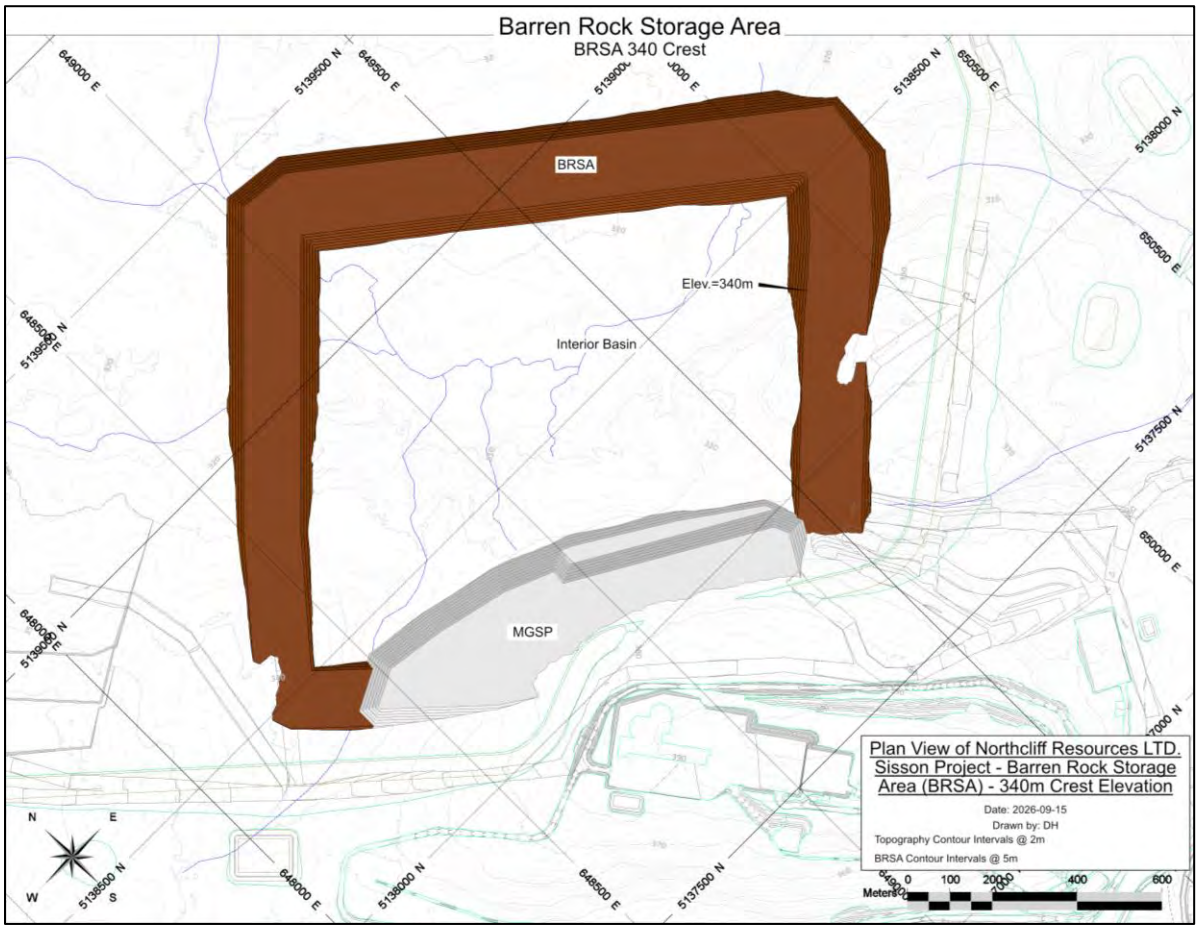
Source: MMTS, 2026

Figure 16-25: BRSA Design 330 m crest elevation



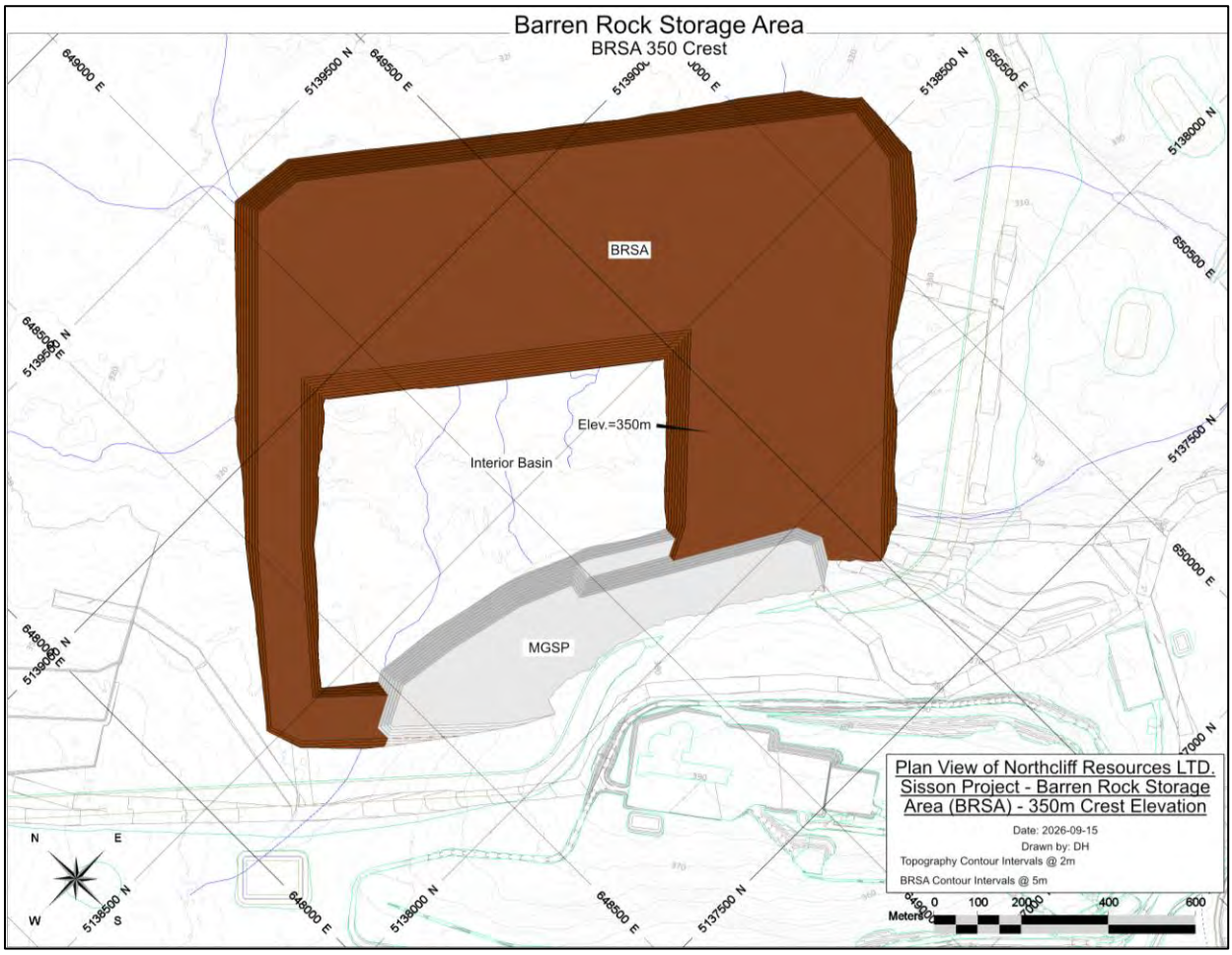
Source: MMTS, 2026

Figure 16-26: BRSA Design 340 m crest elevation



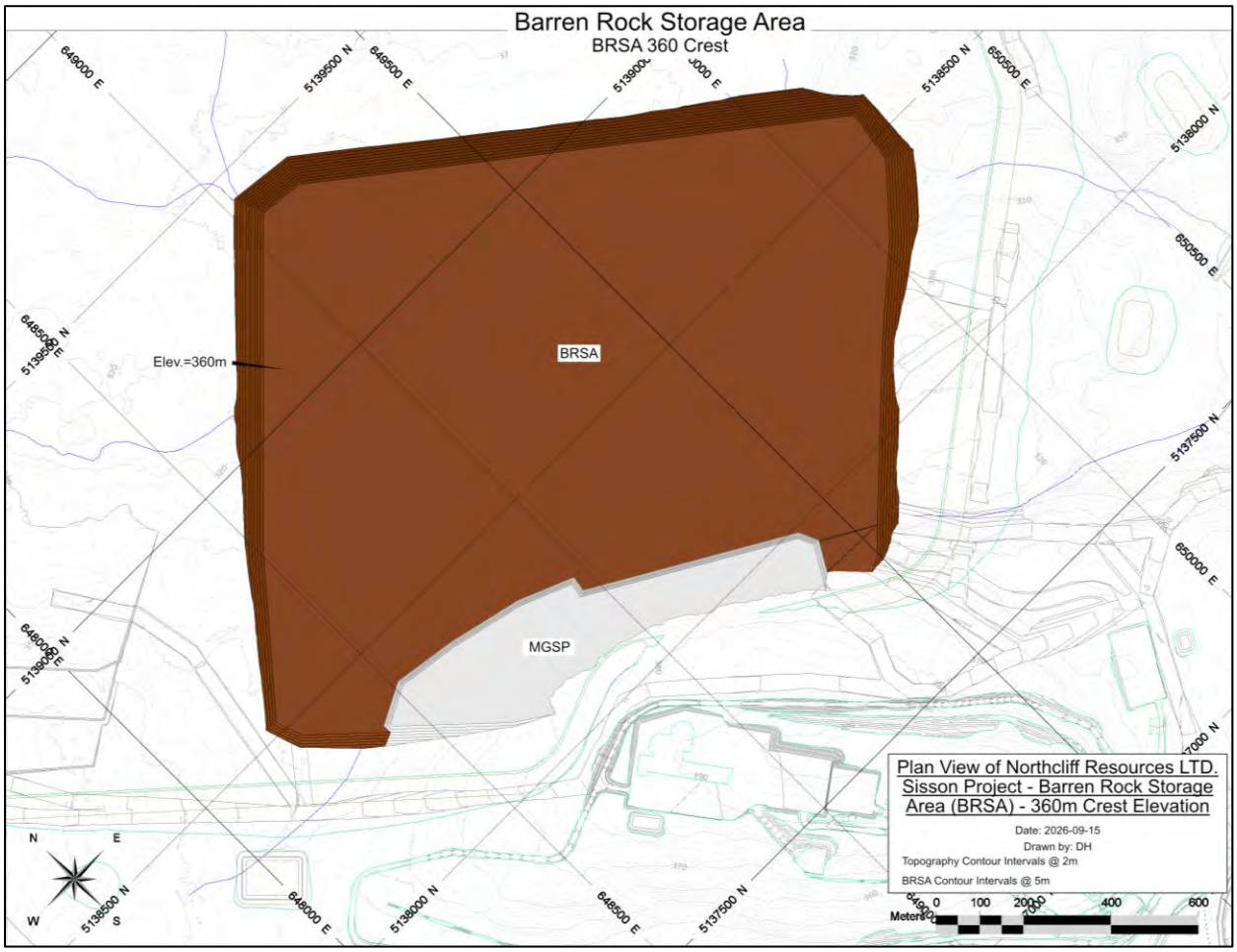
Source: MMTS, 2026

Figure 16-27: BRSA Design 350 m crest elevation



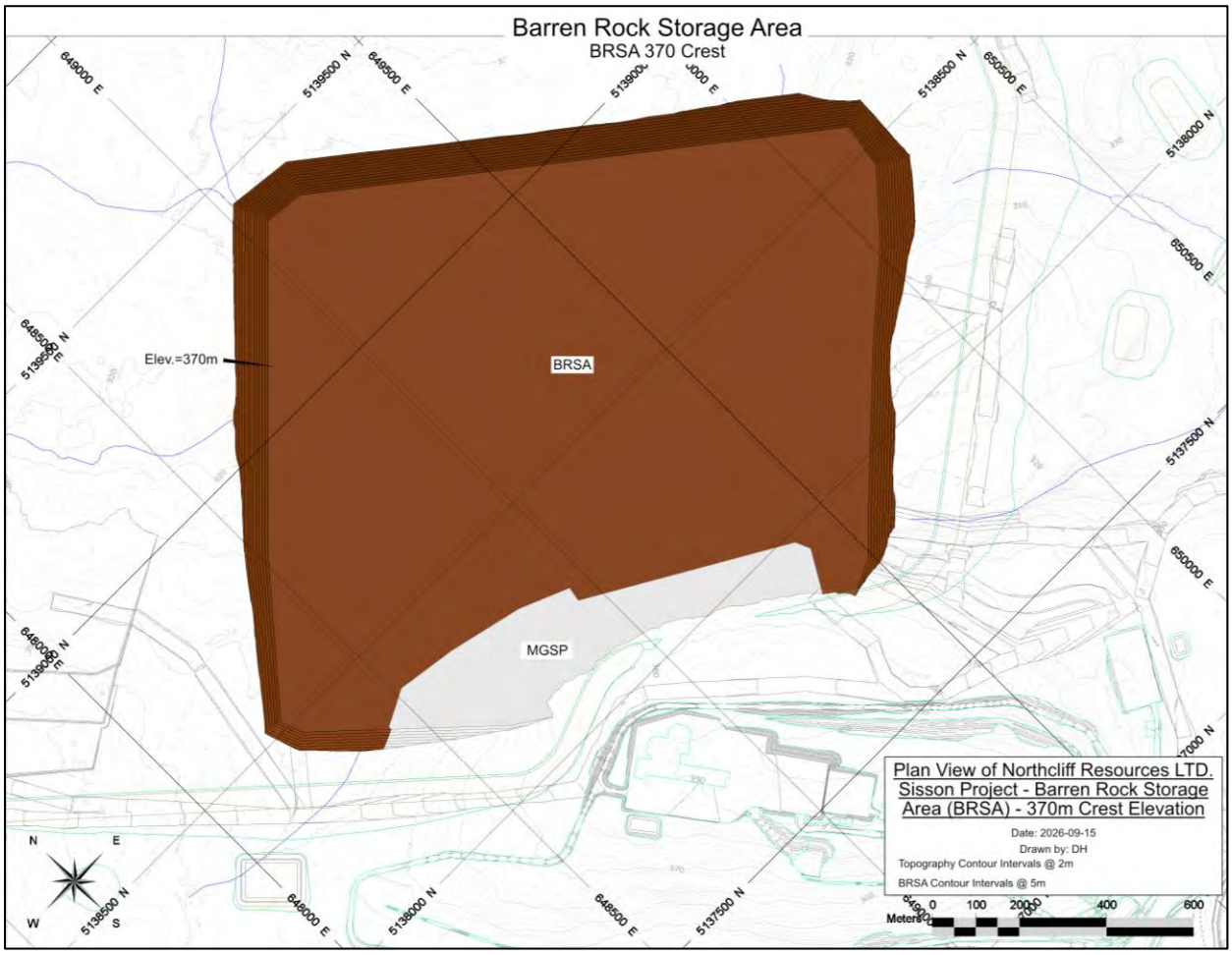
Source: MMTS, 2026

Figure 16-28: BRSA Design 360 m Crest Elevation



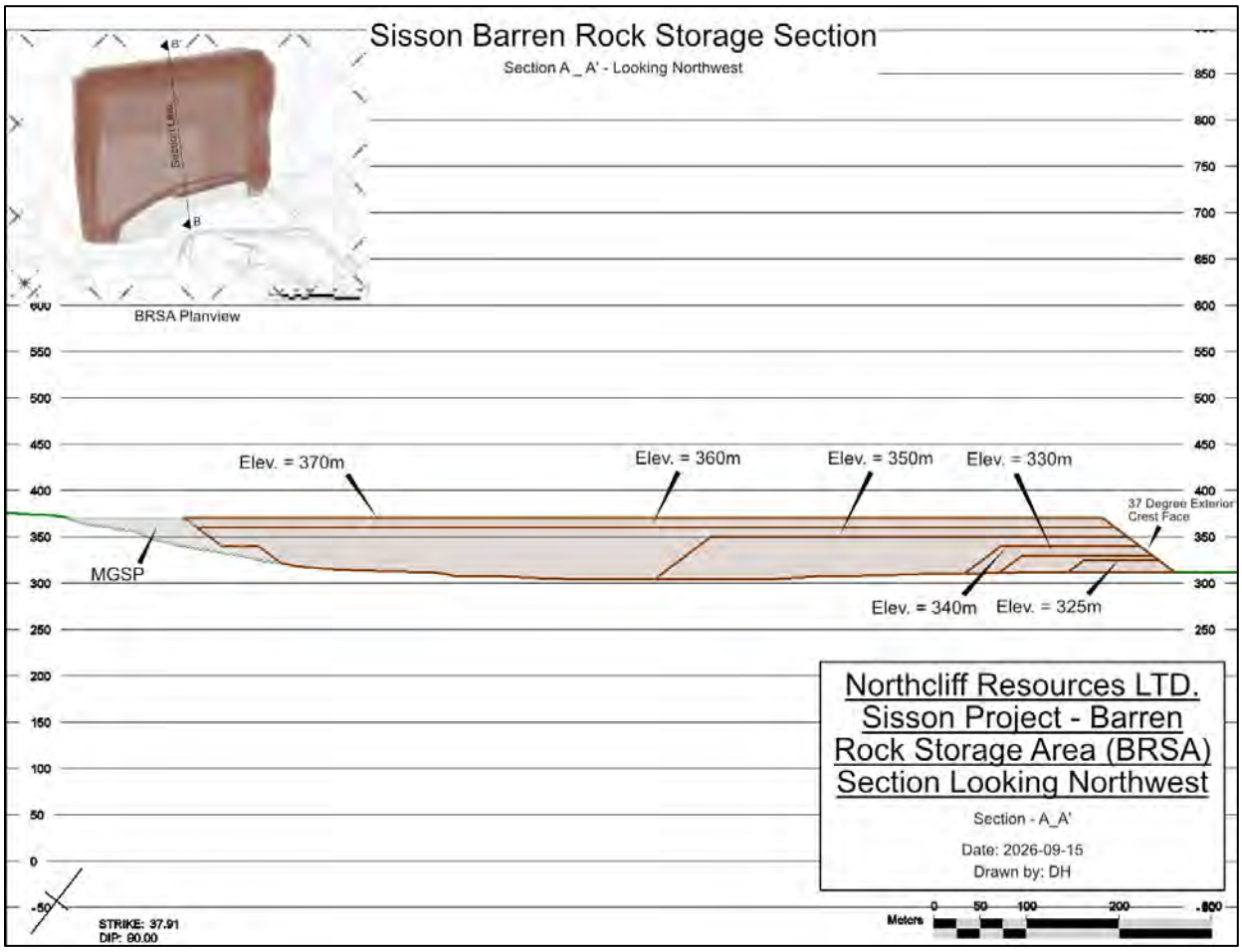
Source: MMTS, 2026

Figure 16-29: BRSA Design 370 m crest elevation



Source: MMTS, 2026

Figure 16-30: BRSA cross-section

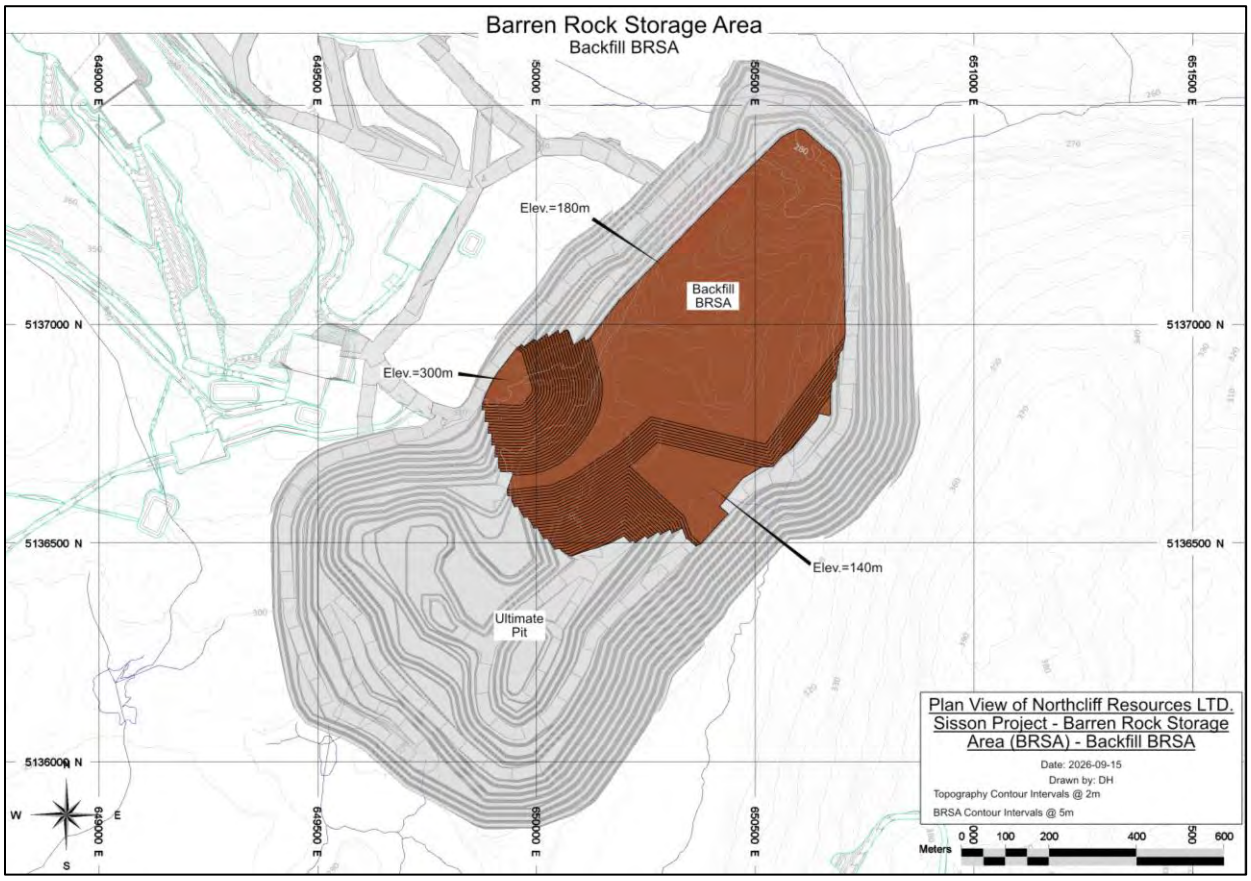


Source: MMTS, 2026

16.6.4 Backfill

Backfilling barren rock into the mined-out portion of the pit begins in Year 19, when Phase 6A is mined out, and continues through LOM. Backfilling occurs from three different elevations, beginning at the 140 m elevation, and including 300 m and 180 m elevations, as illustrated in Figure 16-31. The back fill will be placed in such a way that any active working areas below the toe of an active fill slope is protected from dangers posed by rolling rocks or debris.

Figure 16-31: Backfill designs



Source: MMTS, 2026

16.7 Production schedule

The Sisson mine production schedule was developed with MinePlan® Schedule Optimizer (MPSO), a comprehensive forward looking long-range scheduling tool for open pit mines. It is typically used to produce a LOM schedule that will maximize the NPV of a project, subject to user specified conditions and constraints. Annual production requirements, mine operating considerations, vertical advance rate, product prices, recoveries, destination capacities, equipment performance, and operating costs are used to determine the optimal production schedule.

16.7.1 Throughput rate and scheduling criteria

This mine plan provides mill feed at a nominal rate of 30,000 t/d. Assuming an allowance for down days due to weather and/or unplanned maintenance, at 350 days operating days per year this equates to 10,500 kt/a. To account for mill commissioning in the first quarter of Year 1, a derating allowance of 50% capacity is applied. As a result, the total mill feed for the first year is reduced to 9,220 kt.

Total material movement rates are constrained by maximum shovel availability in any given year. The production schedule dictates a peak material movement of 36,000 kt/a, while averaging approximately 22,000 kt/a over the LOM.

The MPSO NPV calculation utilizes estimated operating and capital costs applied against revenues generated by the tonnage and NSR values of the mill feed (where NSR is a function of process recoveries and metal prices).

It should be noted that the MPSO scope relies on simplified mining costs and does not encompass all project economics. Therefore, this internal NPV calculation is used exclusively to optimize the production schedule and is not the basis for the project's final financial model.

16.7.2 Mine load and haul fleet selection

The Project is designed as a conventional open pit mining operation with truck-and-shovel mining methods. Based on the throughput and schedule, the mining fleet has been selected and optimized to achieve the required production rates to feed the mill while maintaining flexibility for the different ore, barren rock, and quarrying requirements.

Importantly, the Project will utilize Autonomous Haul Systems (AHS) to equip the mine with driverless trucks. The objective of implementing AHS for hauling at Sisson is to improve safety, reduce mining unit costs by increased efficiency and lower personnel costs, and to adopt the latest technologies. Systems like AHS are anticipated to mitigate some recruitment and workforce challenges.

A productivity and pass-matching study was conducted to properly size the fleet and provide 30,000 t/d to the mill. Based on the material properties for the Project, loading is completed using a 200 t diesel hydraulic excavator (HEX) in backhoe (BH) configuration, 300 t electric-hydraulic excavator (E-HEX) front shovels, and a large front-end wheel loader (FEL), matched with 140 to 150 t payload autonomous off-highway trucks (OHT).

Table 16-8: Loader and hauler matching

Load and haul fleet productivities	Units	200 t BH	300 t Shovel	FEL 15 m^3
Truck fleet		147 t OHT	147 t OHT	147t OHT
Loading material situation		Ore/Barren Rock	Ore/Barren Rock	Ore/Barren Rock
Standard bucket at 1.8 t/m^3	m^3	12	16	15
Loader bucket Size @ 2.13 t./m^3	m^3	10.7	14.3	13.6
Loader bucket fill factor		90%	90%	90%
Material bank density, insitu		2.77	2.77	2.77
Material loose density, insitu	t/lcm	2.13	2.13	2.13
Material in each bucket, insitu	t	20.5	27.4	25.7
Material in each bucket, w/ moisture	t	21.5	28.8	27.0
Truck maximum payload	**t**	**147.0**	**147.0**	**147.0**
Truck load factor (carryback)		100%	100%	100%
# of Passes to fill payload	-	6.8	5.0	5.4
Number of passes	**-**	**7**	**5**	**6**
Pass time	seconds	32	32	40
Truck exchange time	minutes	1	1	1
Effective load with exchange time	minutes	5.06	3.80	5.22
Effective loader productivity	**t/h**	**1,660**	**2,142**	**1,611**
Loader annual production	**kt/a**	**8,871**	**11,444**	**8,604**

The combination of 300 t class hydraulic shovels and 150 t class haul trucks as the primary shovel and truck match results in an efficient loading arrangement, minimizing queuing and optimizing cycle times. The shovel–truck match is designed to achieve:

- Efficient pass matching (five passes per truck)
- High utilization rates for primary loading units
- Reduced idle time for both loading and hauling equipment

In addition to the loading times found in Table 16-8, within the MPSO software, equipment operating efficiency factors are applied as follows:

- 200 t HEX BH: 80.8%
- 300 t E-HEX Shovel: 80.8%
- 15 m^3 FEL: 80%
- 147 t Autonomous OHT: 90%

16.7.3 Cut-off grades optimization

To optimize the project NPV, grade bins were specified (based on NSR block values) as outlined in Table 16-9. MPSO routes grade bins based on the best available economic destination each period, resulting in a variable CoG by period. This CoG strategy increased the project NPV by stockpiling lower grade material where possible and effectively increased the revenue per tonne milled early in the schedule. It is assumed that during operations, blast hole assays will be used for ore CoGs and for a CoG strategy.

Table 16-9: Scheduling grade bins

Grade Bin		NSR ($/t) CoG Range		Destination
Bin00	Barren Rock	0	12.02	BRSA
Bin01	Low Grade	12.02[1]	12.82	BRSA or Mill
Bin02	Mid Grade	12.82	13.39	Mill or MGSP
Bin03	Mid Grade	13.39	14.00	Mill or MGSP
Bin04	Mid Grade	14.00	14.64	Mill or MGSP
Bin05	Mid Grade	14.64	15	Mill or MGSP
Bin06	Mid Grade	15.00	16	Mill or MGSP
Bin07	Mid Grade	16.00	17	Mill or MGSP
Bin08	Mid Grade	17.00	18	Mill or MGSP
Bin09	Mid Grade	18.00	19	Mill or MGSP
Bin10	Mid Grade	19.00	20	Mill or MGSP
Bin12	Mid Grade	20.00	21	Mill or MGSP
Bin13	Mid Grade	21.00	22	Mill or MGSP
Bin14	Mid Grade	22.00	23	Mill or MGSP
Bin15	Mid Grade	23.00	24	Mill or MGSP

Grade Bin		NSR ($/t) CoG Range		Destination
Bin16	Mid Grade	24.00	25	Mill or MGSP
Bin17	Mid Grade	25.00	30	Mill or MGSP
Bin18	Mid Grade	30.00	25	Mill or MGSP
Bin19	High Grade	35.00+		Mill

Note:
1. NSR $12.02/t is the Break-Even CoG, calculated from August 2025 operating cost estimates for Processing + TSF + Quarry + G&A (see Section 16.2.1)

16.7.4 Mining sequence

The mining sequence follows a standard sequential progression through Phases 1 through 7 of the open pit. Benches equal to or greater in elevation of the preceding pit phase must be minded before the same bench can be mined in a subsequent pit phase, preventing undercutting. Phase 6 splits into two independent pit bottoms, Phase 6A and 6B which can be mined in any order, though the schedule mines out Phase 6A as early as possible to allow for pit backfilling of barren rock.

BRSA and TSF Embankments are constructed as bottom-up lifts and as such the preceding lift must be 100% filled before moving onto the next lift.

16.7.5 Schedule results

Scheduling results are tabulated annually and summed for the LOM. These results include:

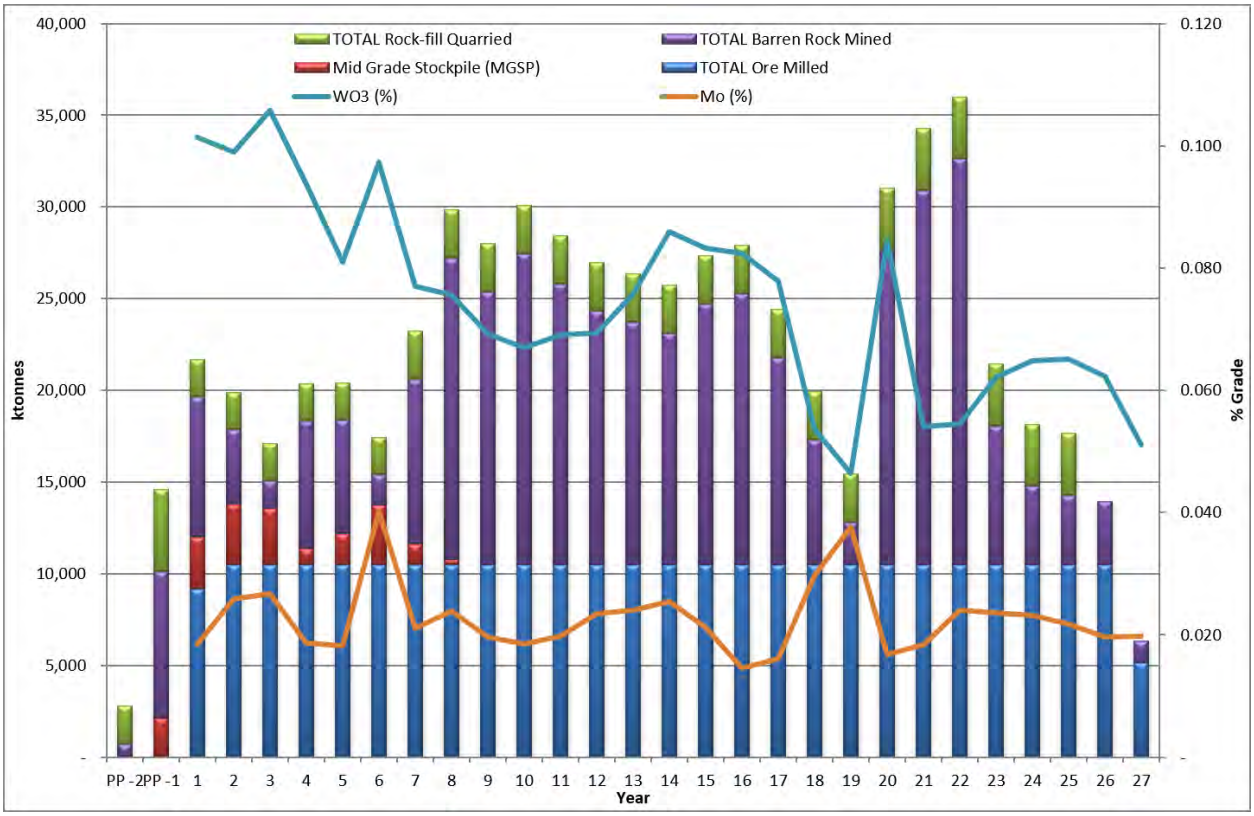
- Tonnes and grade (NSR, WO_3, Mo) mined by year and bench, broken down by material destination
- Shovel and truck requirements by period based on required operating hours
- Tonnes transported by period to different destinations (i.e. TSF embankment, mill, MGSP, and BRSA).

The annual mine production forecast is summarized in Table 16-10 and Table 16-11 and presented graphically in Figure 16-32.

Table 16-10: Summary of production Schedule 3.2.5, pre-production to Year 27

Year	Unit	PP -2	PP -1	1	2	3	4	5	6	7	8	9	10	11	12	13	14	15	16	17	18	19	20	21	22	23	24	25	26	27	LOM
Total ore mined	**kt**	**-**	**2,137**	**11,101**	**13,789**	**13,160**	**9,214**	**8,518**	**13,731**	**9,499**	**10,782**	**10,500**	**10,500**	**10,500**	**10,500**	**10,500**	**10,500**	**10,500**	**10,500**	**10,500**	**10,500**	**10,500**	**10,500**	**10,500**	**10,500**	**10,500**	**10,500**	**10,500**	**5,133**	**286,064**
NSR	$/t	-	35.44	37.04	39.76	43.20	42.17	38.39	44.39	35.49	35.44	31.59	30.09	31.50	33.32	36.01	40.68	37.78	34.59	33.46	29.89	30.61	36.28	25.02	27.55	30.33	31.24	30.76	28.72	24.25	34.45
WO_3%	%	-	0.085	0.086	0.086	0.093	0.098	0.088	0.084	0.080	0.075	0.069	0.067	0.069	0.069	0.076	0.086	0.083	0.082	0.078	0.054	0.046	0.084	0.054	0.055	0.062	0.065	0.065	0.062	0.051	0.074
Mo%	%	-	0.014	0.017	0.024	0.025	0.018	0.018	0.036	0.019	0.024	0.020	0.018	0.020	0.024	0.024	0.025	0.021	0.015	0.016	0.030	0.038	0.017	0.018	0.024	0.024	0.023	0.022	0.020	0.020	0.022
Direct mill feed	**kt**	**-**	**-**	**8,267**	**10,500**	**10,106**	**8,336**	**6,852**	**10,500**	**8,362**	**10,500**	**10,500**	**10,500**	**10,500**	**10,500**	**10,500**	**10,500**	**10,500**	**10,500**	**10,500**	**10,500**	**10,500**	**10,500**	**10,500**	**10,500**	**10,500**	**10,500**	**10,500**	**5,133**	**267,556**
NSR	$/t	-	-	43.09	45.99	49.79	44.75	43.18	51.43	38.01	35.98	31.59	30.09	31.50	33.32	36.01	40.68	37.78	34.59	33.46	29.89	30.61	36.28	25.02	27.55	30.33	31.24	30.76	28.72	24.25	35.34
WO_3%	%	-	-	0.099	0.099	0.107	0.104	0.099	0.097	0.085	0.076	0.069	0.067	0.069	0.069	0.076	0.086	0.083	0.082	0.078	0.054	0.046	0.084	0.054	0.055	0.062	0.065	0.065	0.062	0.051	0.075
Mo%	%	-	-	0.019	0.026	0.027	0.019	0.019	0.040	0.020	0.024	0.020	0.018	0.020	0.024	0.024	0.025	0.021	0.015	0.016	0.030	0.038	0.017	0.018	0.024	0.024	0.023	0.022	0.020	0.020	0.023
Mid-grade stockpile (MGSP)	**kt**	**-**	**2,137**	**2,834**	**3,289**	**3,054**	**878**	**1,666**	**3,231**	**1,137**	**282**	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	**18,508**
NSR	$/t	-	35.44	19.37	19.88	21.38	17.67	18.70	21.49	16.95	15.48	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
WO_3%	%	-	0.085	0.047	0.043	0.045	0.045	0.044	0.040	0.042	0.040	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
Mo%	%	-	0.014	0.013	0.017	0.019	0.010	0.014	0.023	0.011	0.010	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
MGSP reclaim	**kt**	-	-	**920**	-	**394**	**2,164**	**3,648**	-	**2,138**	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	**9,263**
NSR	$/t	-	-	50.04	-	30.03	25.21	21.28	-	23.72	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
WO_3%	%	-	-	0.119	-	0.069	0.055	0.048	-	0.044	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
Mo%	%	-	-	0.017	-	0.016	0.018	0.016	-	0.024	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
Mid-grade stockpile, balance	**kt**	-	**2,137**	**4,051**	**7,340**	**10,000**	**8,714**	**6,732**	**9,963**	**8,962**	**9,244**	**9,244**	**9,244**	**9,244**	**9,244**	**9,244**	**9,244**	**9,244**	**9,244**	**9,244**	**9,244**	**9,244**	**9,244**	**9,244**	**9,244**	**9,244**	**9,244**	**9,244**	**9,244**	**9,244**
NSR	$/t	-	35.44	32.24	26.70	26.13	31.77	45.75	37.88	44.26	43.38	43.38	43.38	43.38	43.38	43.38	43.38	43.38	43.38	43.38	43.38	43.38	43.38	43.38	43.38	43.38	43.38	43.38	43.38	43.38	43.38
WO_3%	%	-	0.085	0.077	0.062	0.059	0.072	0.105	0.084	0.098	0.097	0.097	0.097	0.097	0.097	0.097	0.097	0.097	0.097	0.097	0.097	0.097	0.097	0.097	0.097	0.097	0.097	0.097	0.097	0.097	0.097
Mo%	%	-	0.014	0.016	0.017	0.018	0.022	0.032	0.029	0.033	0.033	0.033	0.033	0.033	0.033	0.033	0.033	0.033	0.033	0.033	0.033	0.033	0.033	0.033	0.033	0.033	0.033	0.033	0.033	0.033	0.033
Total ore milled	**kt**	-	-	**9,187**	**10,500**	**10,500**	**10500**	**10,500**	**10,500**	**10,500**	**10,500**	**10,500**	**10,500**	**10,500**	**10,500**	**10,500**	**10,500**	**10,500**	**10,500**	**10,500**	**10,500**	**10,500**	**10,500**	**10,500**	**10,500**	**10,500**	**10,500**	**10,500**	**5,133**	**276,820**
Effective NSR Cut-off grade	**kt**	-	-	**21.00**	**23.00**	**25.00**	**17.00**	**19.00**	**25.00**	**18.00**	**16.00**	**12.82**	**15.00**	**15.00**	**15.00**	**15.00**	**15.00**	**15.00**	**15.00**	**15.00**	**14.64**	**15.00**	**15.00**	**12.02**	**14.00**	**15.00**	**15.00**	**15.00**	**14.00**	**12.82**	**16.31**
NSR	$/t	-	-	43.79	45.99	49.05	40.73	35.57	51.43	35.10	35.98	31.59	30.09	31.50	33.32	36.01	40.68	37.78	34.59	33.46	29.89	30.61	36.28	25.02	27.55	30.33	31.24	30.76	28.72	24.25	35.03
WO_3%	%	-	-	0.101	0.099	0.106	0.094	0.081	0.097	0.077	0.076	0.069	0.067	0.069	0.069	0.076	0.086	0.083	0.082	0.078	0.054	0.046	0.084	0.054	0.055	0.062	0.065	0.065	0.062	0.051	0.075
Mo%	%	-	-	0.019	0.026	0.027	0.019	0.018	0.040	0.021	0.024	0.020	0.018	0.020	0.024	0.024	0.025	0.021	0.015	0.016	0.030	0.038	0.017	0.018	0.024	0.024	0.023	0.022	0.020	0.020	0.023
Total rock-fill quarried	**kt**	**2,083**	**4,479**	**2,035**	**2,035**	**2,035**	**2,035**	**2,035**	**2,035**	**2,646**	**2,646**	**2,646**	**2,646**	**2,646**	**2,646**	**2,646**	**2,646**	**2,646**	**2,646**	**2,646**	**2,646**	**2,646**	**3,385**	**3,385**	**3,385**	**3,385**	**3,385**	**3,385**	-	-	**73,480**
Quarry Rock (ROM) to TSF Embankment	kt	1443	4,330	1,940	1,940	1,940	1,940	1,940	1,940	2,559	2,559	2,559	2,559	2,559	2,559	2,559	2,559	2,559	2,559	2,559	2,559	2,559	3,325	3,325	3,325	3,325	3,325	3,325	-	-	70,629
Quarry ROCK Stockpiled for Crushing	kt	639	149	95	95	95	95	95	95	88	88	88	88	88	88	88	88	88	88	88	88	88	59	59	59	59	59	59	-	-	2,851
Total barren Rock Mined	**kt**	**694**	**7,974**	**7,644**	**4,047**	**1,531**	**6,940**	**6,191**	**1,651**	**8,938**	**16,410**	**14,868**	**16,935**	**15,270**	**13,834**	**13,216**	**12,587**	**14,199**	**14,773**	**11,276**	**6,810**	**2,303**	**17,150**	**20,401**	**22,115**	**7,551**	**4,263**	**3,779**	**3,449**	**1,211**	**278,010**
Barren Rock (ROM) to TSF Embankment	kt	694	1,388	636	636	636	636	636	636	663	663	663	663	663	663	663	663	663	663	663	663	663	420	420	420	420	420	420	-	-	17,036
Barren Rock to BRSA	kt	-	6,586	7,008	3,412	895	6,304	5,556	1,015	8,275	15,748	14,205	16,272	14,607	13,171	12,553	11,924	13,536	14,110	10,613	6,148	1,542	-	-	-	-	-	-	-	-	183,479
Barren Rock to Backfill	kt	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	98	16,730	19,980	21,695	7,131	3,843	3,358	3,449	1,211	77,495
Strip ratio (barren rock/ore mined)	-	-	**3.7**	**0.7**	**0.3**	**0.1**	**0.8**	**0.7**	**0.1**	**0.9**	**1.5**	**1.4**	**1.6**	**1.5**	**1.3**	**1.3**	1.2	1.4	1.4	1.1	0.6	0.2	1.6	1.9	2.1	0.7	0.4	0.4	0.3	0.2	1.0
Effective strip ratio ([BR+MGSP]/MF)	-	-	-	**1.1**	**0.7**	**0.4**	**0.7**	**0.7**	**0.5**	**1.0**	**1.6**	**1.4**	**1.6**	**1.5**	**1.3**	**1.3**	1.2	1.4	1.4	1.1	0.6	0.2	1.6	1.9	2.1	0.7	0.4	0.4	0.3	0.2	1.1
Total material mined (incl Quarry))	**kt**	**2,777**	**14,591**	**20,780**	**19,871**	**16,726**	**18,189**	**16,744**	**17,416**	**21,084**	**29,839**	**28,014**	**30,081**	**28,416**	**26,980**	**26,362**	**25,733**	**27,345**	**27,919**	**24,422**	**19,957**	**15,449**	**31,035**	**34,285**	**36,000**	**21,436**	**18,148**	**17,663**	**13,949**	**6,343**	**637,554**

Figure 16-32: Summary of production Schedule 3.2.5



Source: MMTS, 2026

16.7.6 Detailed mine plan

The detailed mine plan is illustrated over the 27-year LOM in Figure 16-33 through Figure 16-46. A summary of annual activities in areas related to the mine and quarry are described in Table 16-11.

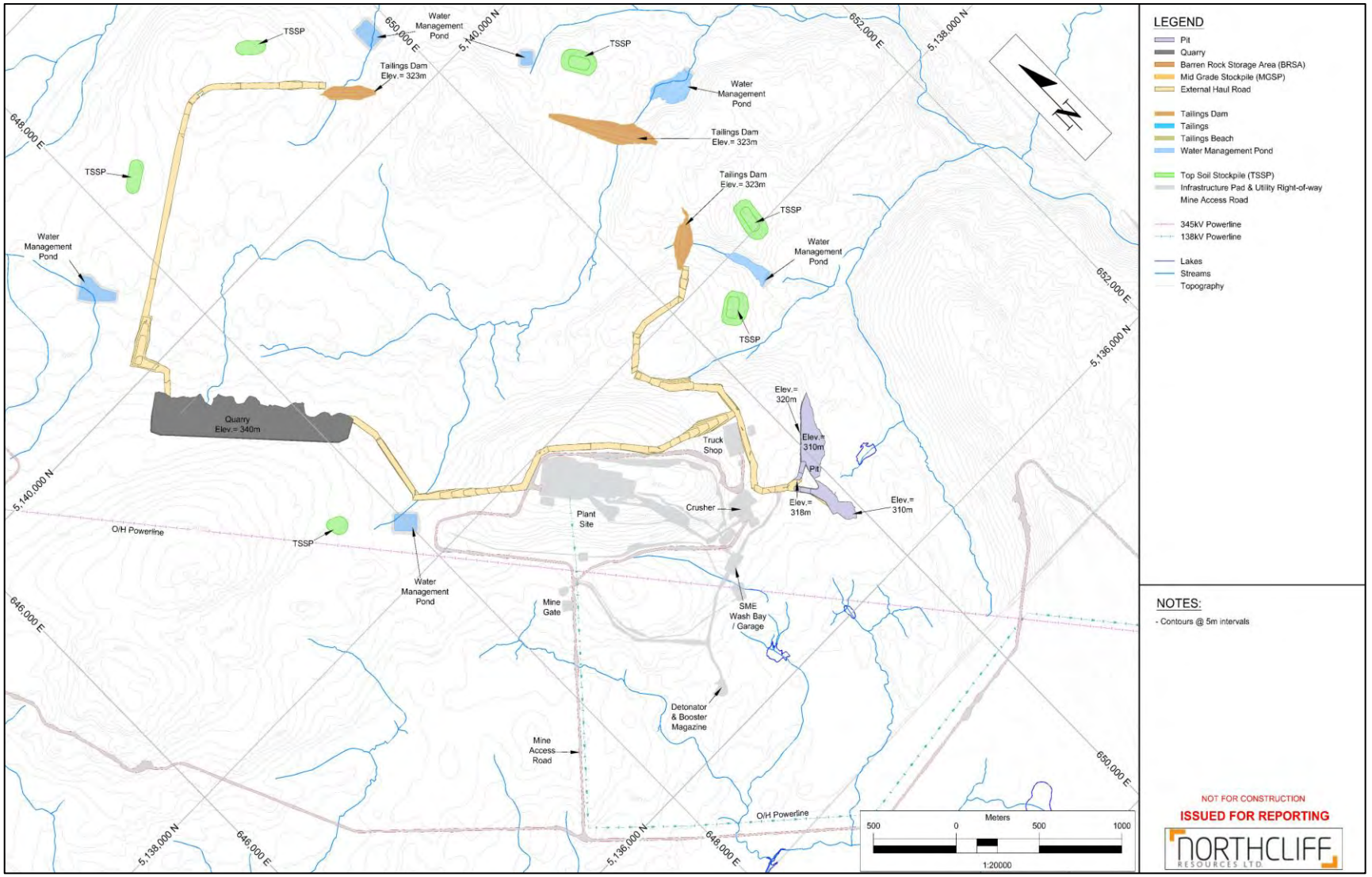
Table 16-11: Annual mine and quarry activities

Year	Activity
Y-2 and Y-1	Clearing and grubbing the phase 1 of pit and quarry areas. Clearing and grubbing of ex-pit haul road, BRSA, MGSP, and top soil stockpile footprints. Removal and stockpiling of topsoil from pit areas cleared and grubbed. Haul road construction from the pit and quarry to the TSF embankments, BRSA and MGSP Mining of the phase 1 pit down to 280 bench Mining of the phase 1 quarry pit down to 330 bench Delivery of quarry rock to the TSF embankments and various infrastructure construction areas. Placement of excess mined barren rock to the 325m elevation BRSA "causeway". Stockpiling ore in the MGSP for use in mill commissioning. See Figure 16-33 and Figure 16-34
Y1	Clearing and grubbing of pit and quarry areas (annually throughout mine life). Phase 1 pit mined down to 260 bench.

Year	Activity
	Phase 1 quarry continued mining on the 330 bench. Delivery of quarry rock and barren rock from the pit to construct TSF embankment up to 334 elev. Placement of excess mined barren rock to the 330m elevation BRSA "causeway". Stockpiling of mid-grade ore in the MGSP. See Figure 16-35
Y2 to Y5	Phase 1 pit mined out (110 bench) by end of Year 4. Phase 2 pit mined to the 230 bench. Phase 3 pit mined to the 270 bench. Phase 4 pit pre-stripped to 290 bench Haul road expansion from the pits to BRSA 340 lift. Phase 2 quarry mined out (344 bench) by end of Year 4. Phase 3 quarry mined to 360 bench. Delivery of quarry rock and barren rock from the pit to construct TSF embankment up to 341 elev. Placement of excess mined barren rock to the 340m elevation BRSA "causeway" (constructed from both sides). Stockpiling of mid-grade ore in the MGSP. MGSP stockpile reclaim from the 340m elevation See Figure 16-36, Figure 16-37, Figure 16-38, Figure 16-39
Y6 to Y7	Phase 2 pit mined out (110 bench) by end of Year 7. Phase 3 pit mined out (220 bench) by end of Year 7. Phase 4 pit mined to the 250 bench. Haul road expansion from the pits to BRSA 350 lift. Phase 3 quarry mined to 350 bench. Delivery of quarry rock and barren rock from the pit to construct TSF embankment up to 347 elev. Placement of excess mined barren rock to the 350m elevation BRSA "causeway" (constructed from both sides). Stockpiling of mid-grade ore in the MGSP. MGSP stockpile reclaim becomes impractical See Figure 16-40
Y8 to Y10	Phase 4 pit mined out (160 bench) by end of Year 9. Phase 5 pit mined to the 210 bench. Phase 6 pit early pre-stripping to the 330 bench Haul road expansion from the pits to BRSA 350 lift. Phase 3 quarry mined out (350 bench) by end of Year 8. Phase 4 quarry mined to 390 bench. Delivery of quarry rock and barren rock from the pit to construct TSF embankment up to 352 elev. Placement of excess mined barren rock to the 350m elevation BRSA, "causeway" construction completed, proceeding to fill interior basin Stockpiling of mid-grade ore in the MGSP ceases by end of Year 8. See Figure 16-41

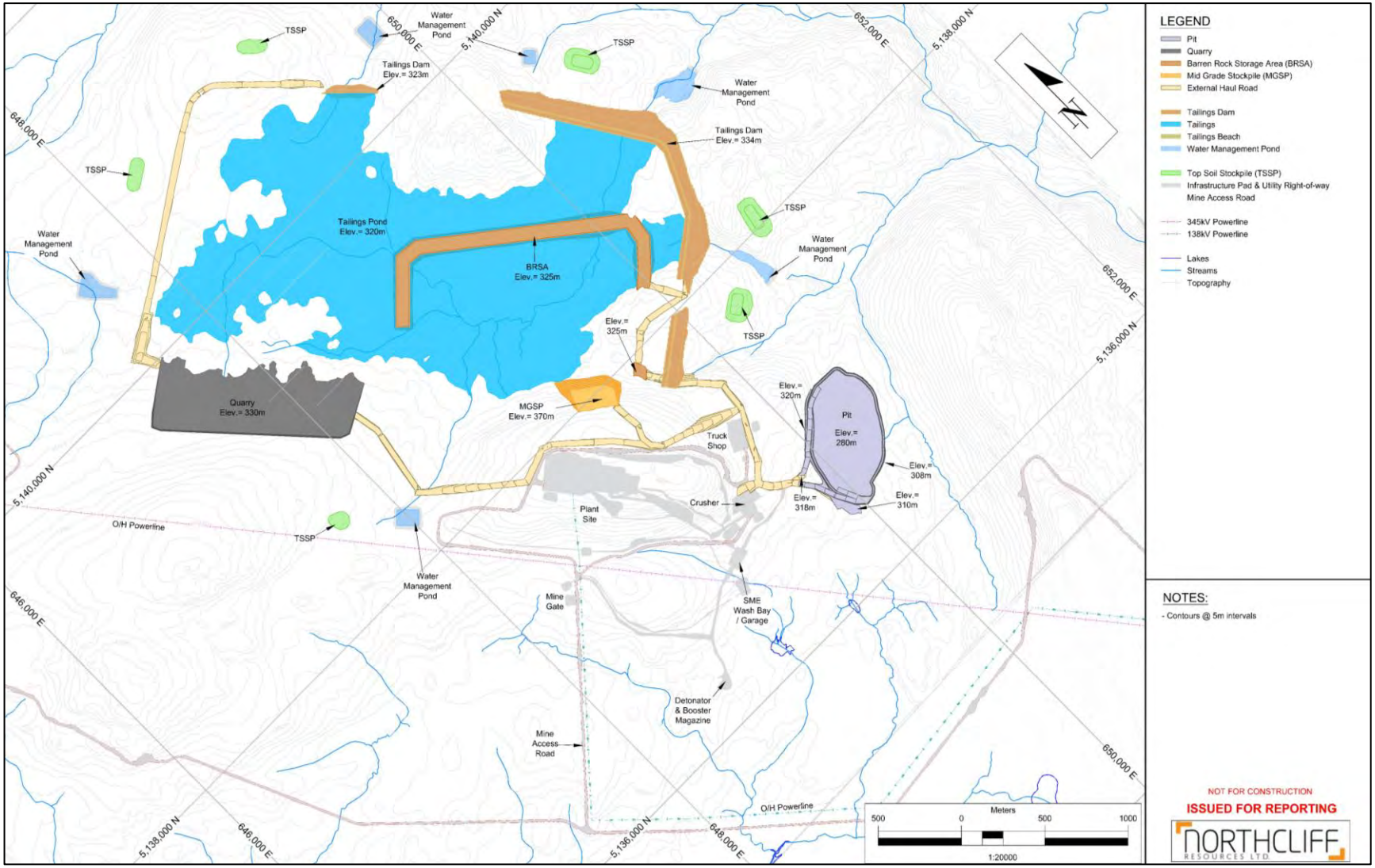
Year	Activity
Y11 to Y16	Phase 5 pit mined out (30 bench) by end of Year 16. Phase 6 pit mined to the 170 bench. Phase 4 quarry mined to 380 bench. Delivery of quarry rock and barren rock from the pit to construct TSF embankment up to 363 elev. Placement of excess mined barren rock to the 360m elevation BRSA, interior basin filled. See Figure 16-42
Y17 to Y19	Phase 6 pit mined out (140 bench) by end of Year 18. Phase 6A pit mined out (-30 bench) by end of Year 19. Phase 4 quarry mined to 370 bench. Delivery of quarry rock and barren rock from the pit to construct TSF embankment up to 363 elev. Placement of excess mined barren rock to the 370m elevation BRSA, filled to completion by end of Year 19 Backfilling of pit commences at 140m elevation by the end of Year 19. See Figure 16-43Figure 16-42
Y20 to Y23	Phase 6B pit mined out (40 bench) by end of Year 21. Phase 4 quarry continued mining on the 370 bench. Phase 7 pit mined to the 140 bench. Delivery of quarry rock and barren rock from the pit to construct TSF embankment up to 374 elev. Backfilling of excess mined barred rock, between 140m, 300m, and 180m elevations. See Figure 16-44
Y24 to Y27	Phase 7 pit mined out (20 bench) by end of Year 27. Phase 4 quarry mined out (360 bench) by end of Year 25. Delivery of quarry rock and barren rock from the pit to construct TSF embankment up to 374 elev. BRSA and MGSP are submerged beneath tailings pond by end of Year 25. Backfilling of excess mined barred rock, between 140m and 180m elevations. Closure and reclamation begins by end of Year 27 See Figure 16-45

Figure 16-33: End of period, pre-production, Year -2



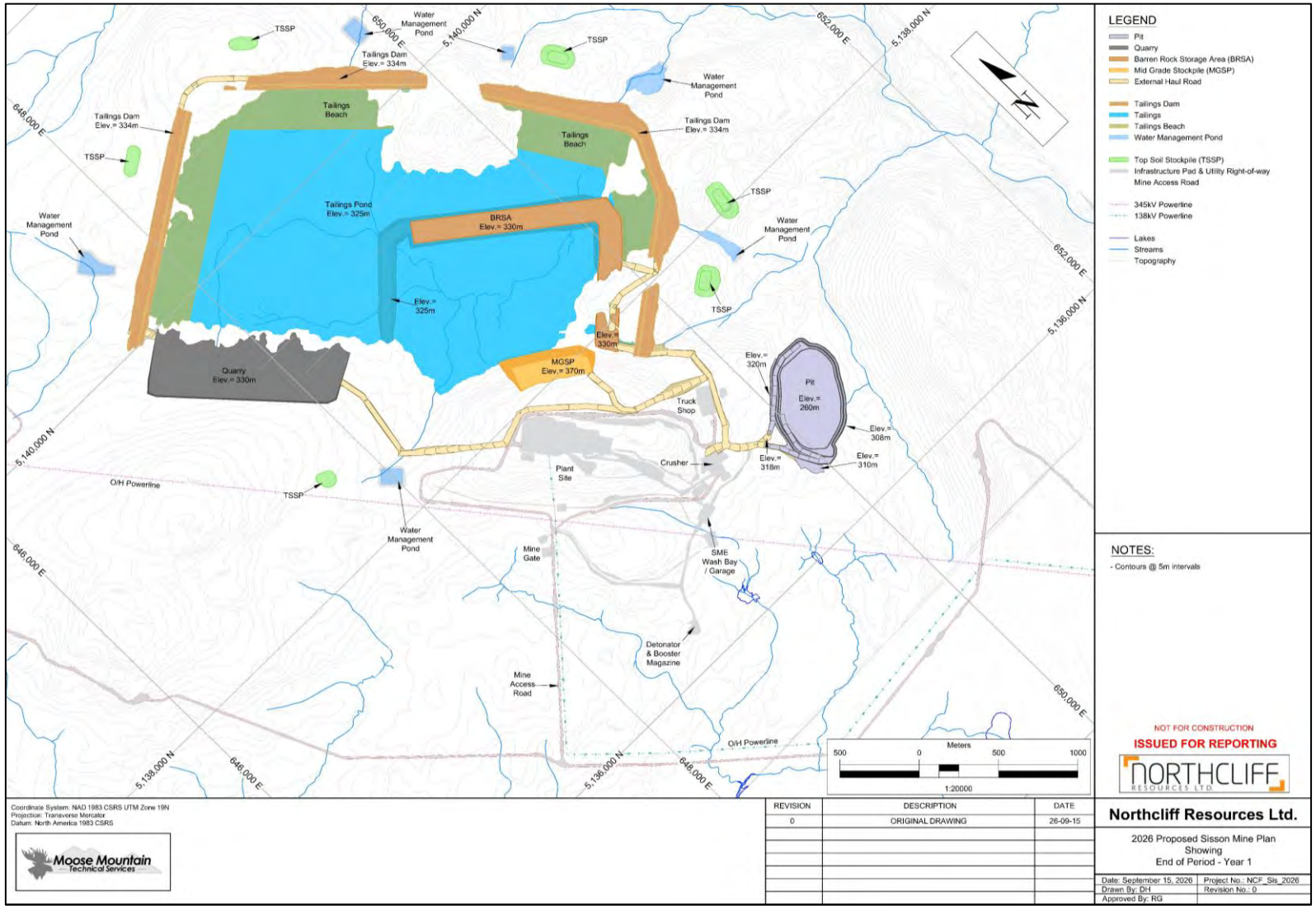
Source: MMTS, 2026

Figure 16-34: End of period, pre-production, Year -1



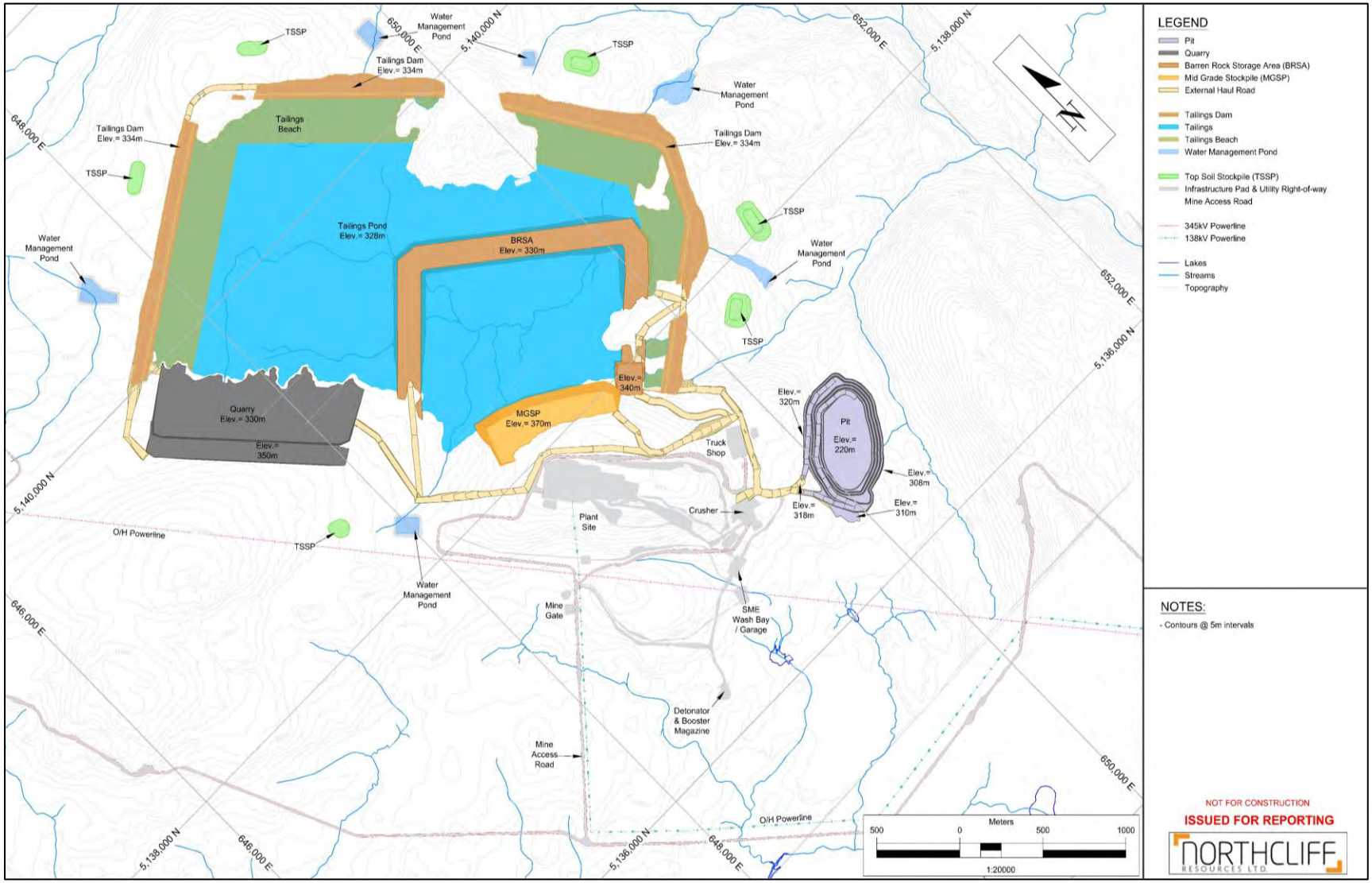
Source: MMTS, 2026

Figure 16-35: End of period, Year 1



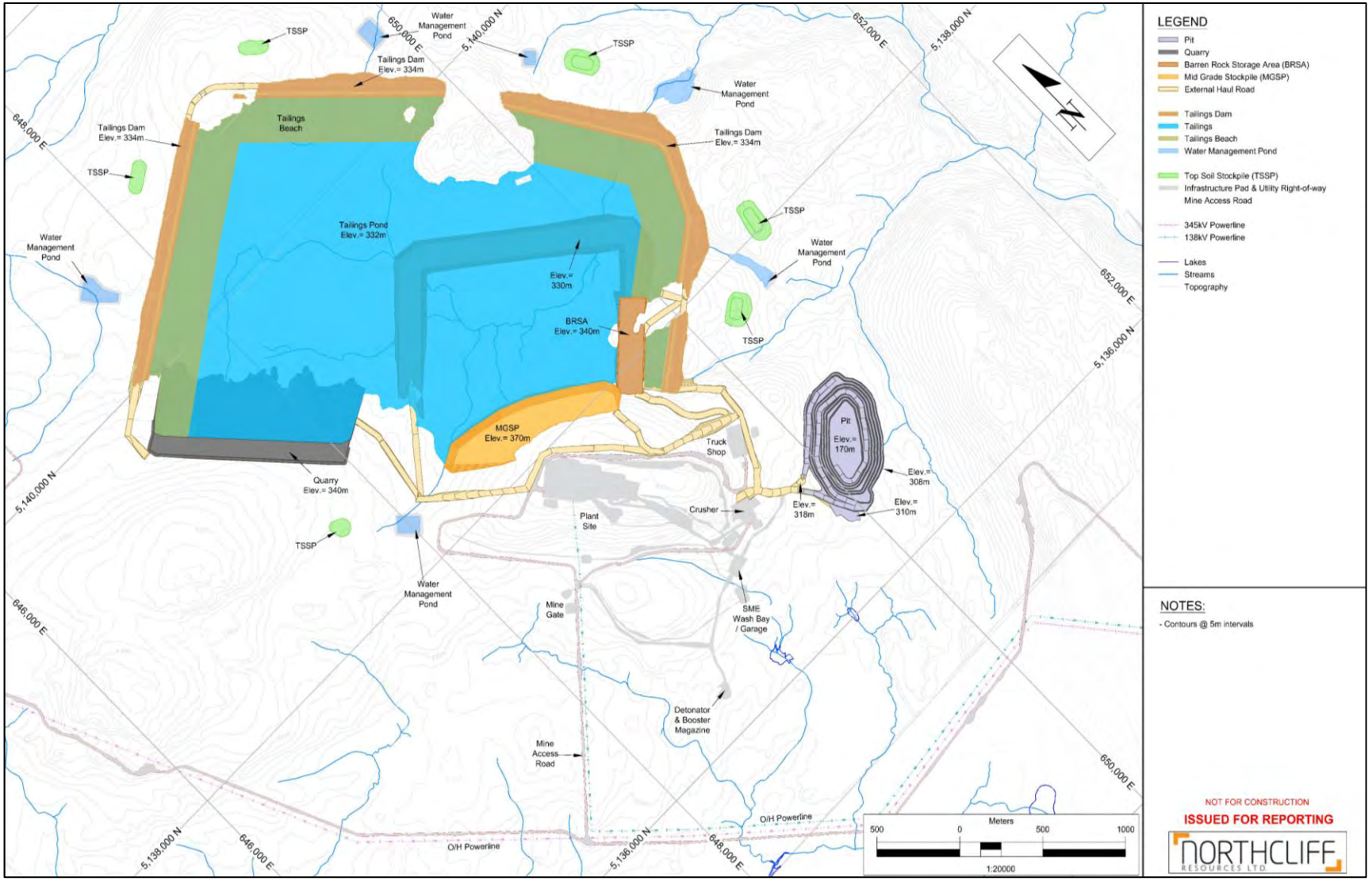
Source: MMTS, 2026

Figure 16-36: End of period, Year 2



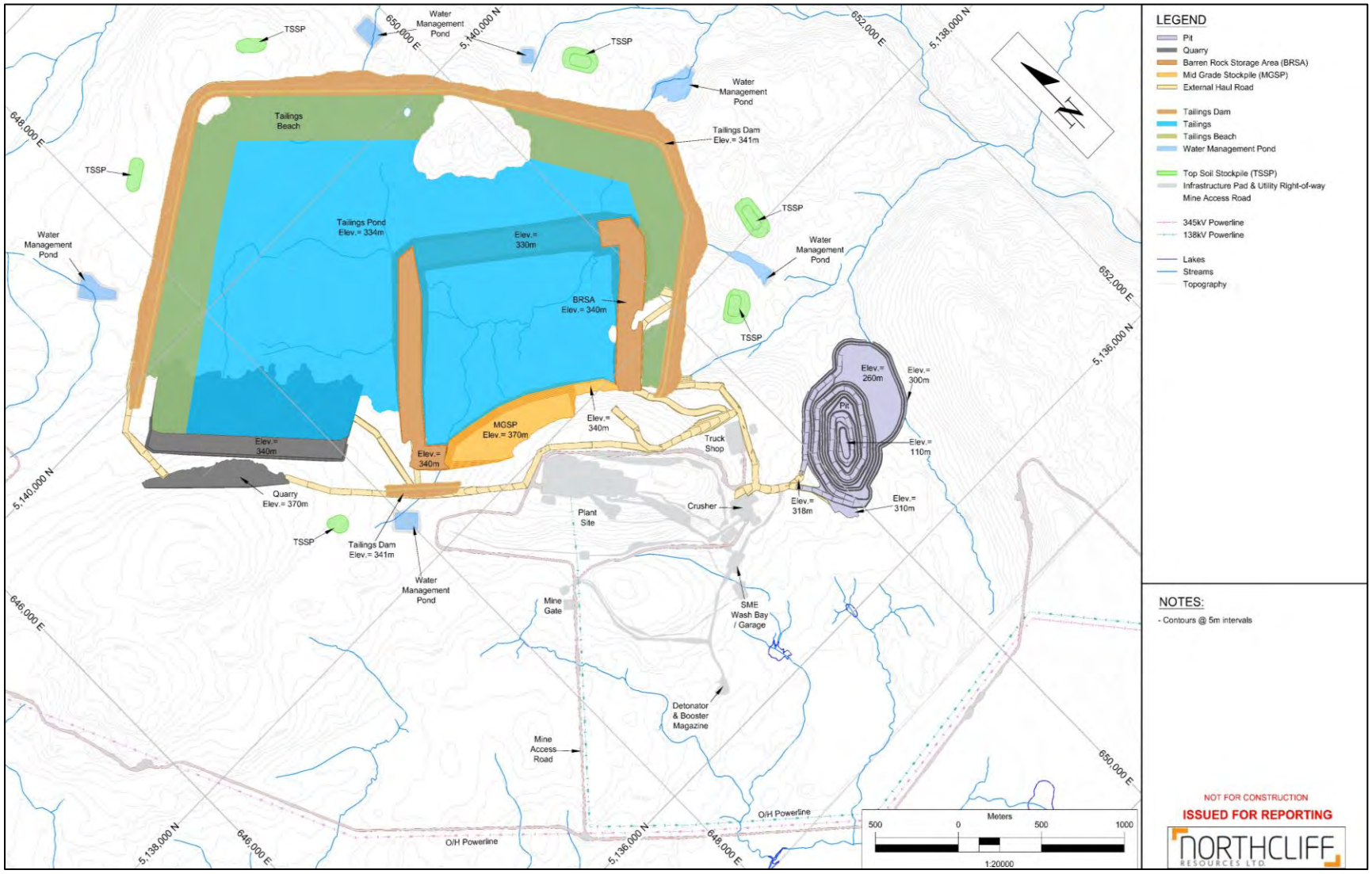
Source: MMTS, 2026

Figure 16-37: End of period, Year 3



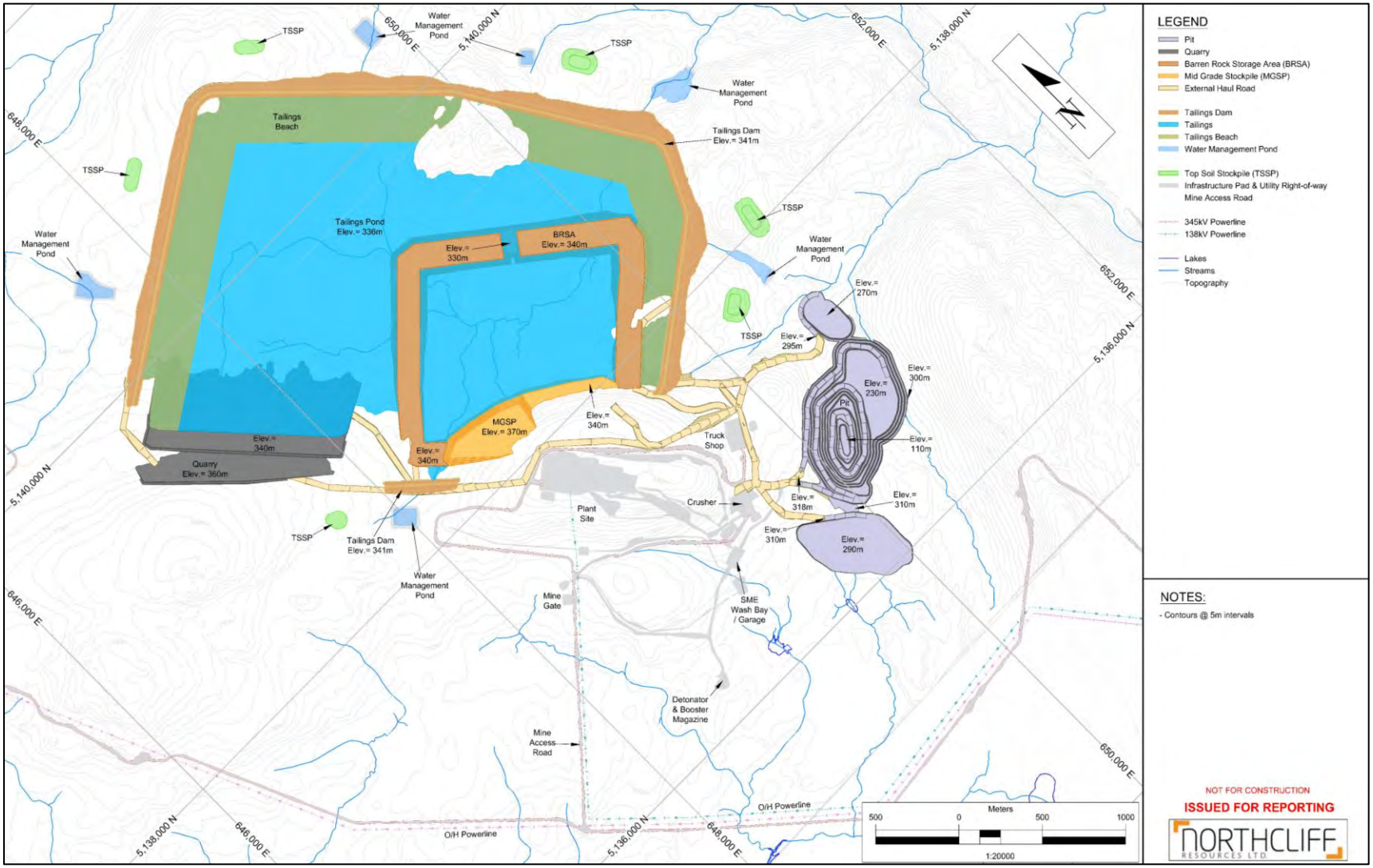
Source: MMTS, 2026

Figure 16-38: End of period, Year 4



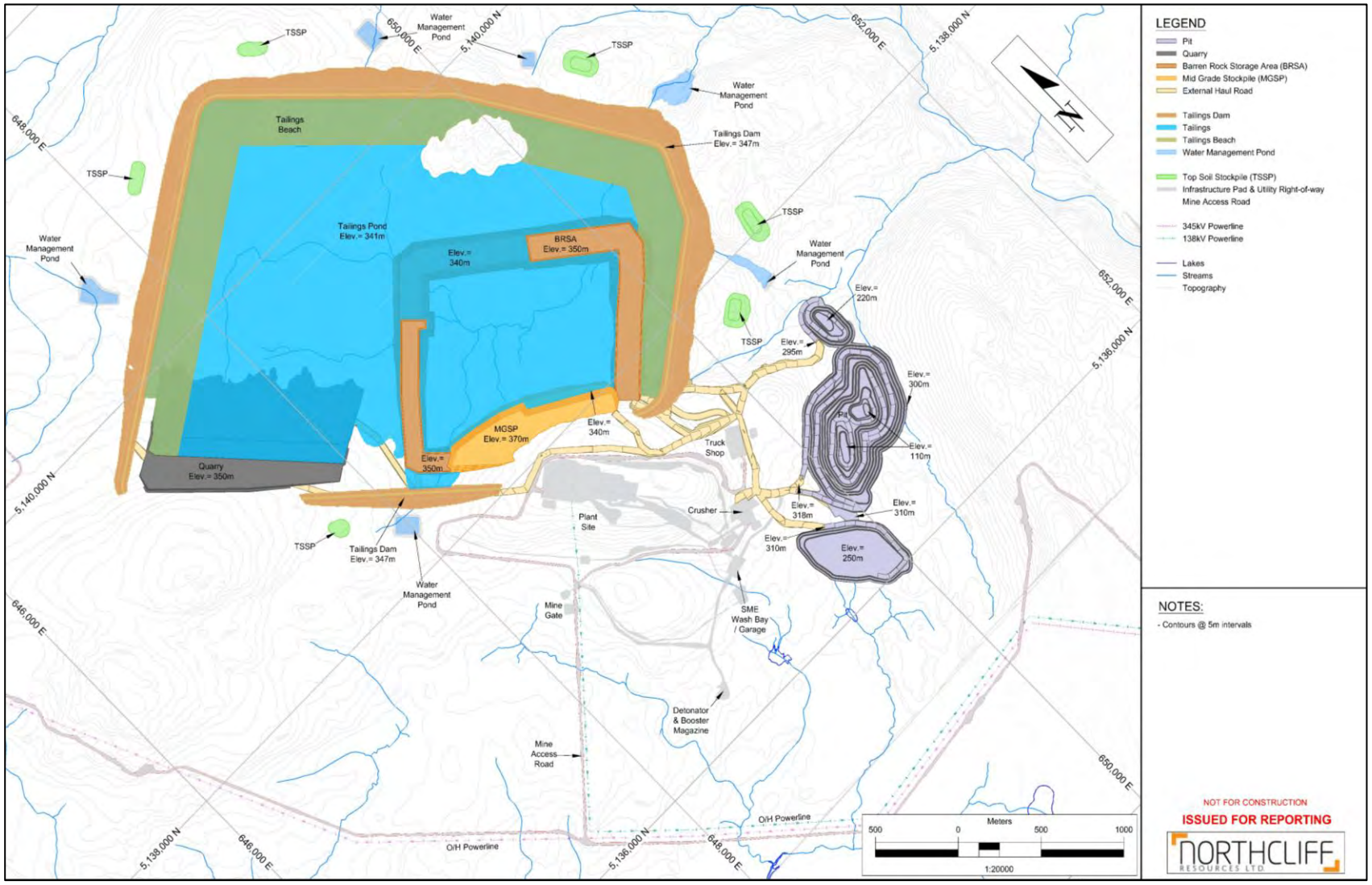
Source: MMTS, 2026

Figure 16-39: End of period, Year 5



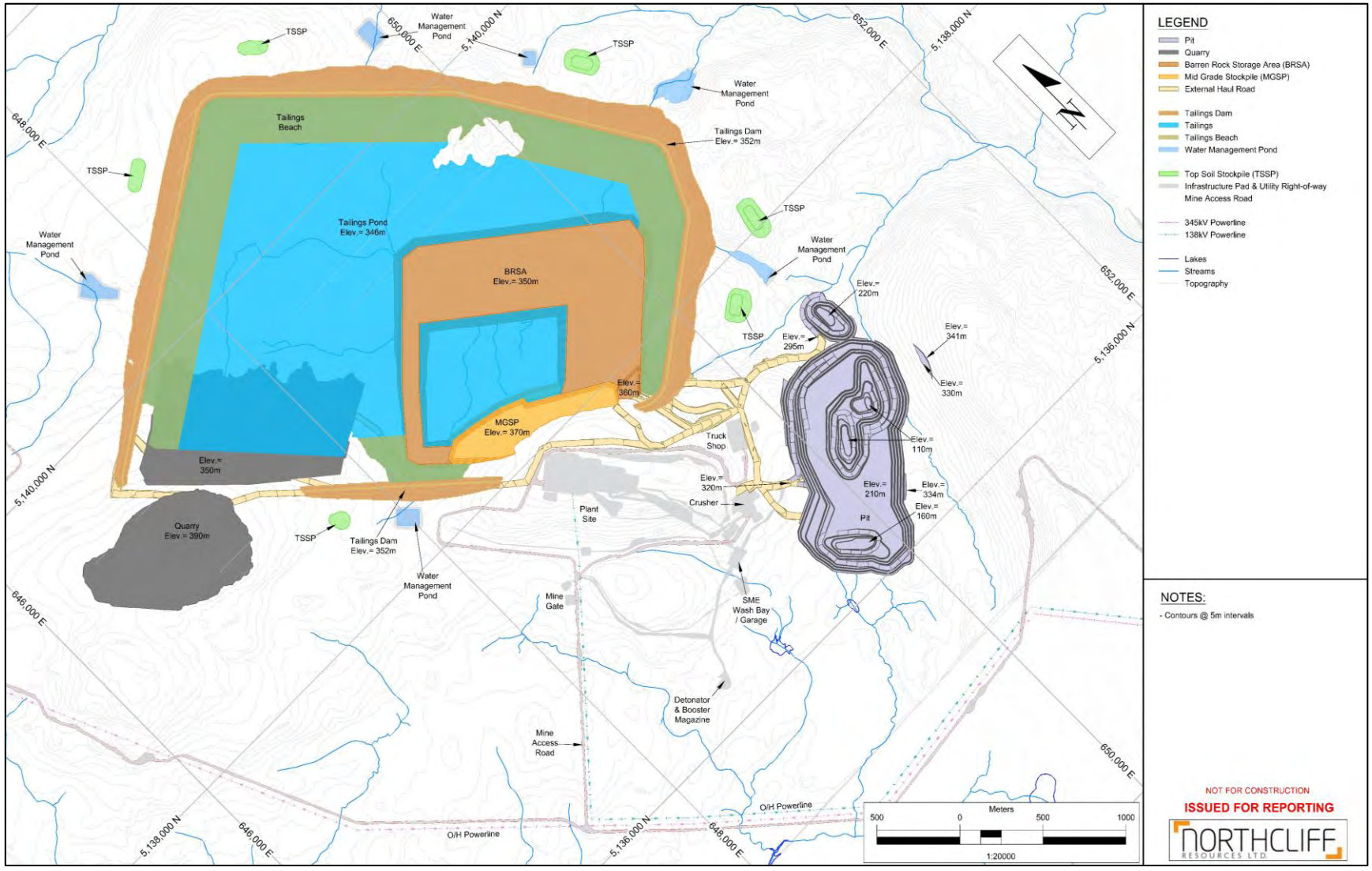
Source: MMTS, 2026

Figure 16-40: End of period, Year 7



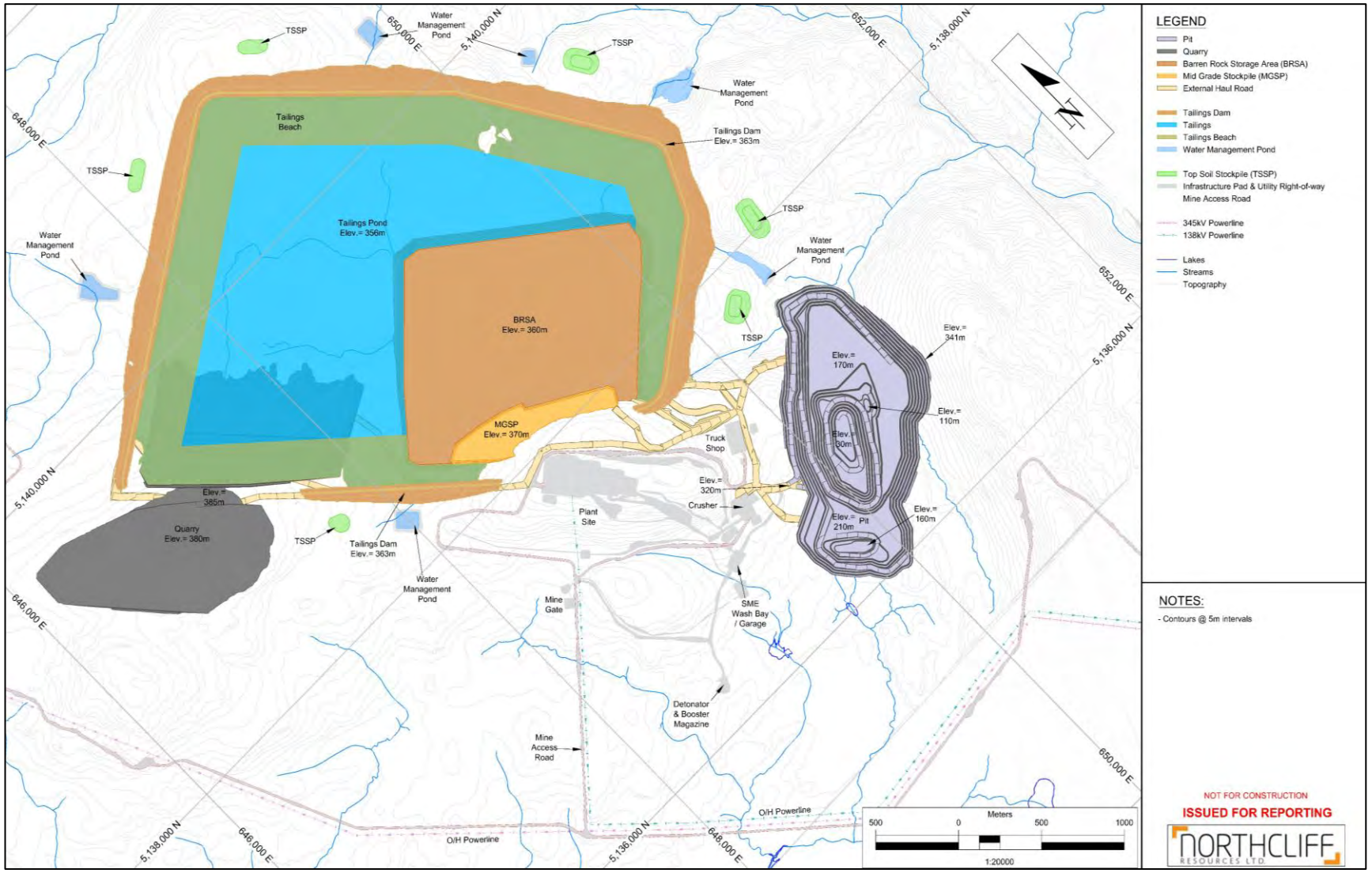
Source: MMTS, 2026

Figure 16-41: End of period, Year 10



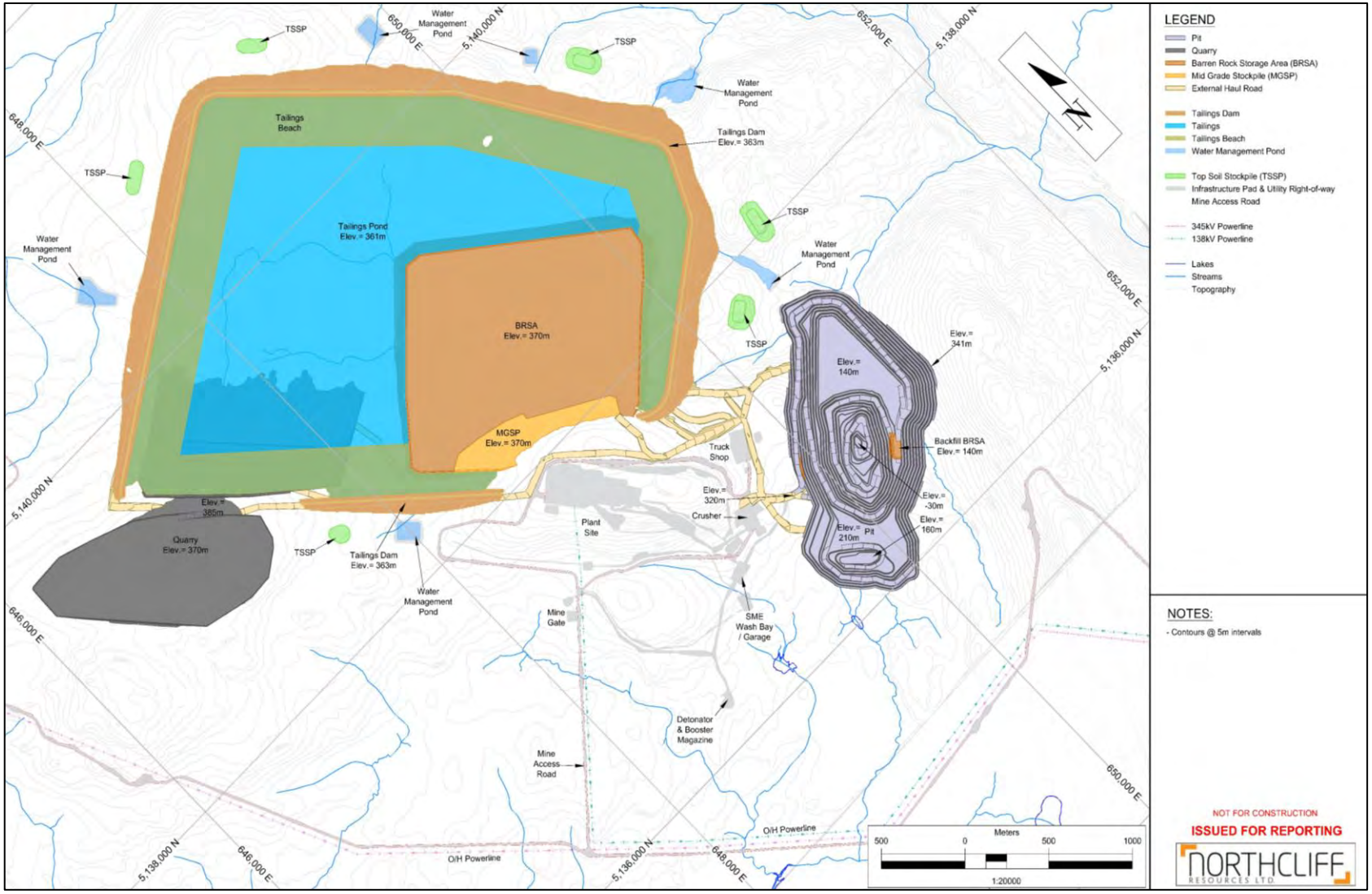
Source: MMTS, 2026

Figure 16-42: End of period, Year 16



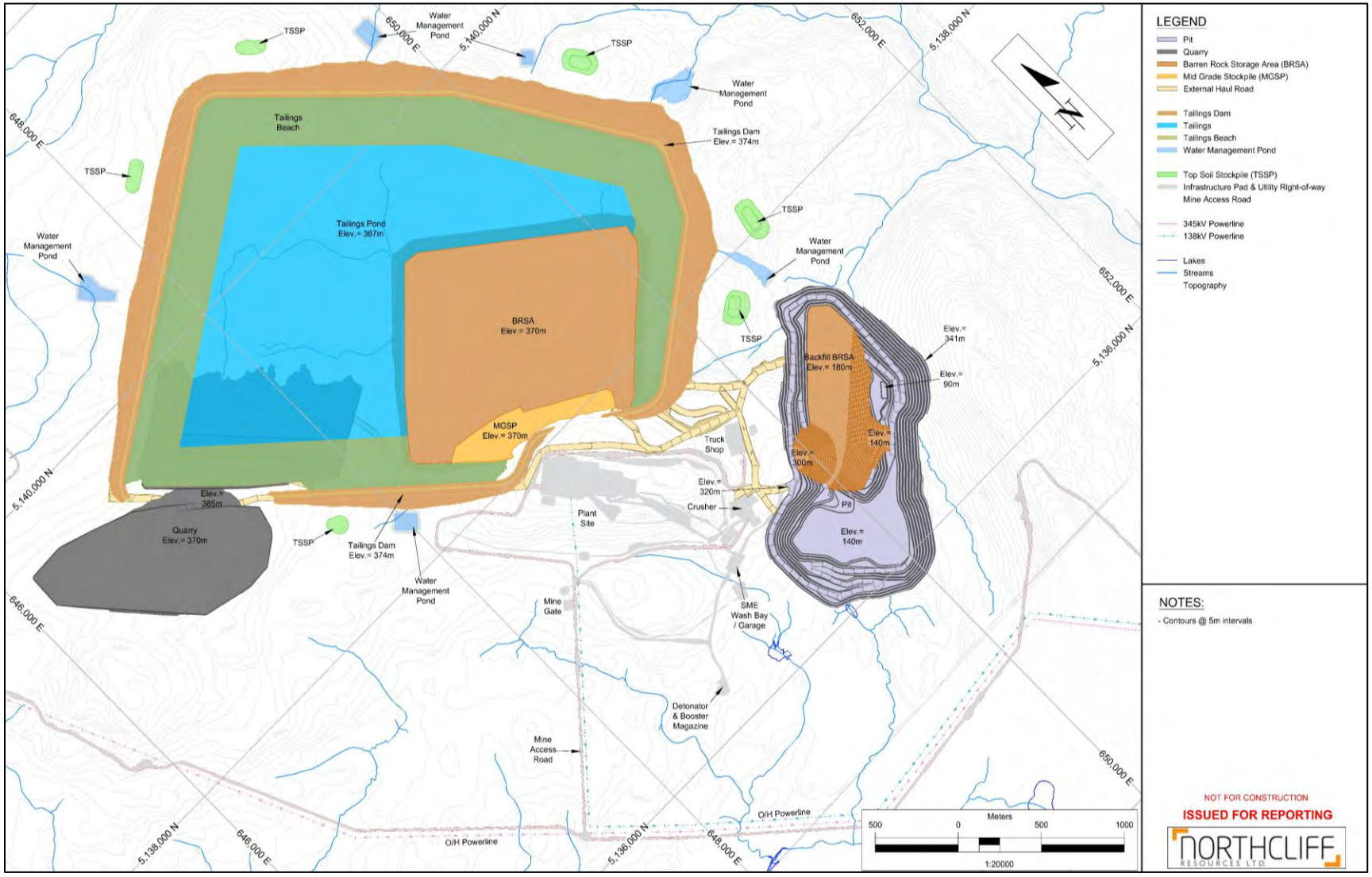
Source: MMTS, 2026

Figure 16-43: End of period, Year 19



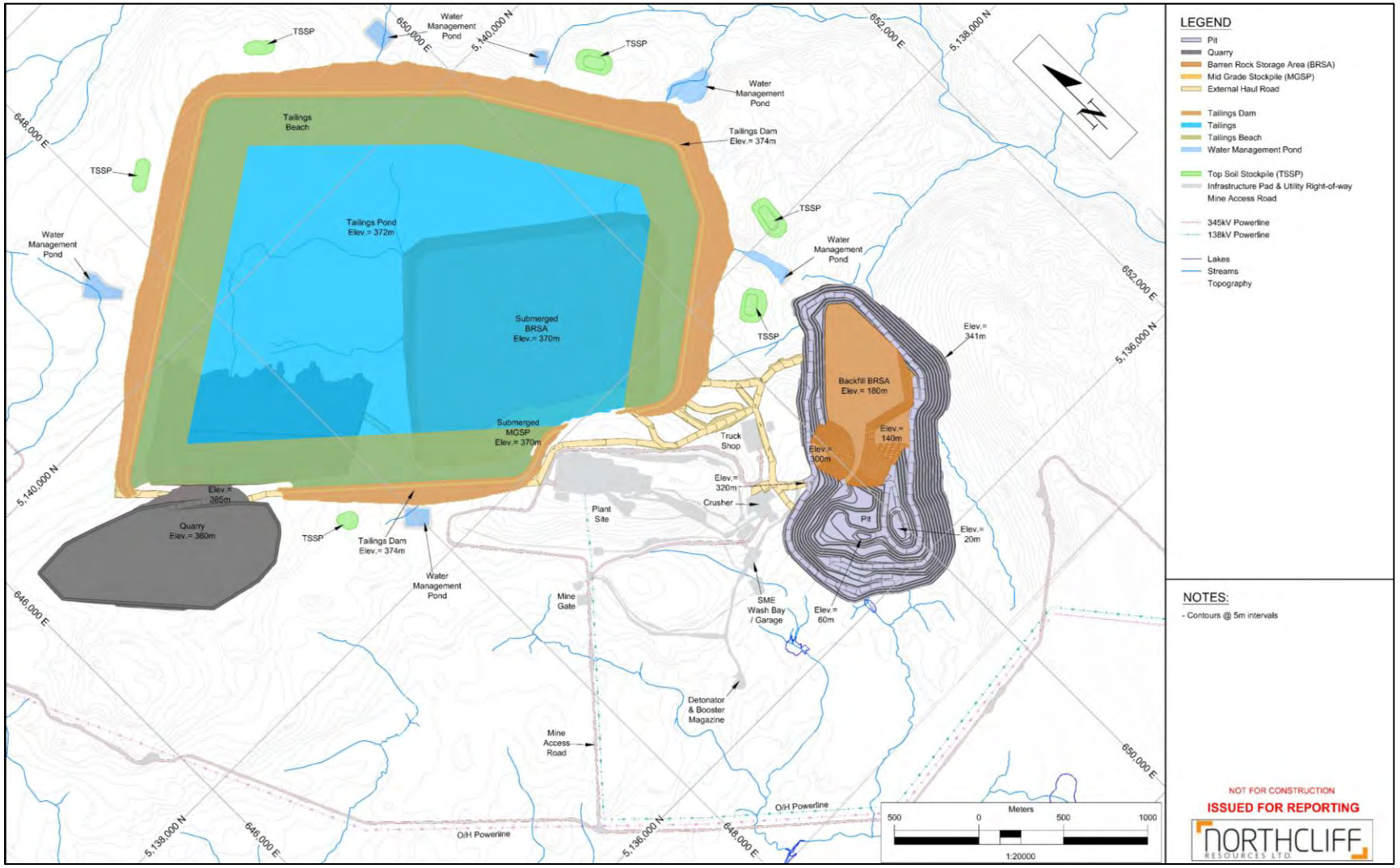
Source: MMTS, 2026

Figure 16-44: End of period, Year 23



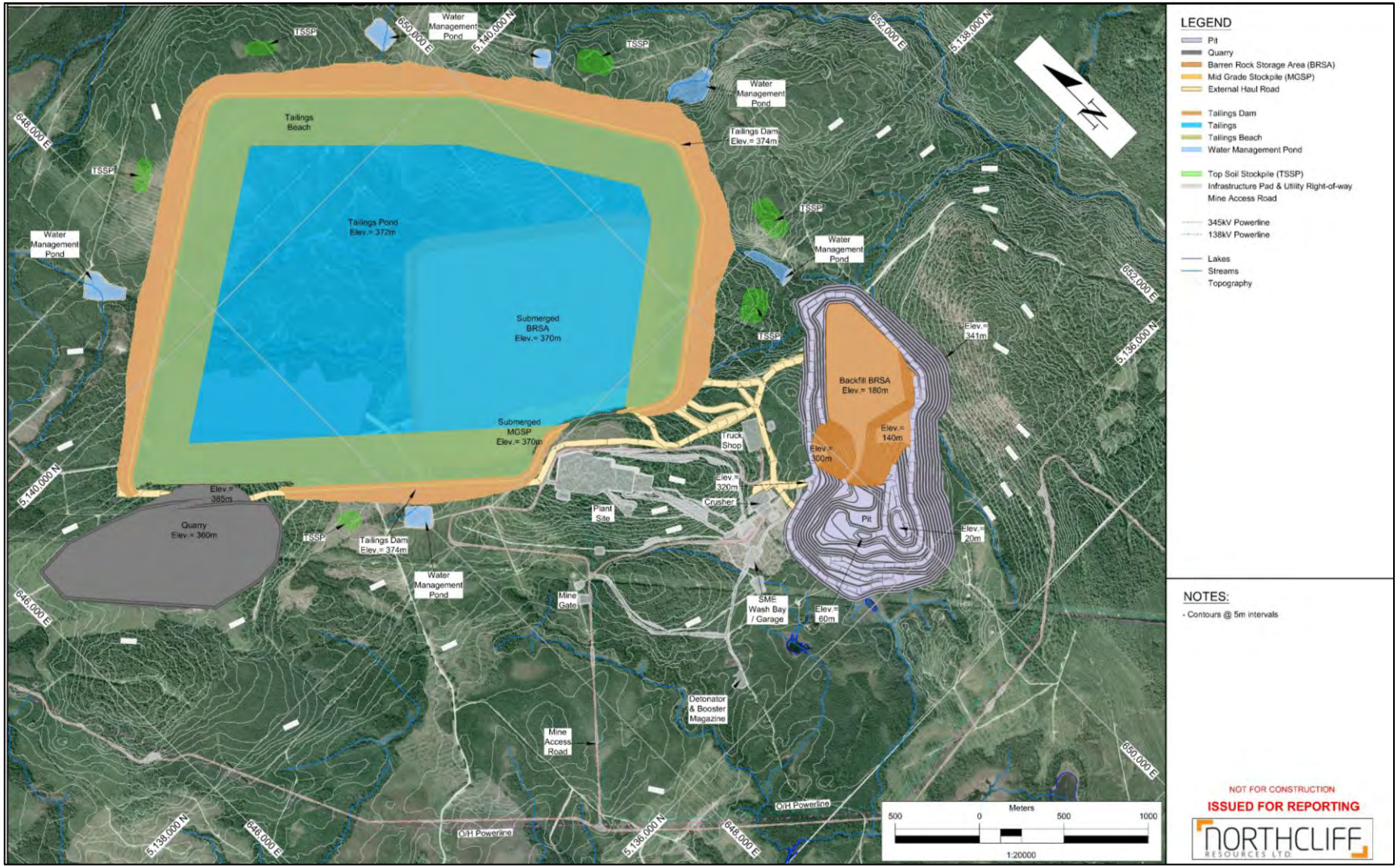
Source: MMTS, 2026

Figure 16-45: End of period, Year 27



Source: MMTS, 2026

Figure 16-46: End of period, Year 27 with photo



Source: MMTS, 2026

16.7.7 Mine closure and reclamation

Open pit mine closure and reclamation are discussed in Section 20.5

16.8 Mine operations

As a bulk, open pit mine operation, Sisson relies on a wide range of mobile equipment, technology, and support services for successful operation. The truck and shovel fleet are chosen to meet production requirements, and the support fleet is specified to the size and quantity of loading and hauling machines. The objective is to operate safely, minimize unit operating (mining) costs, ensure flexibility in operations to meet day-to-day demands.

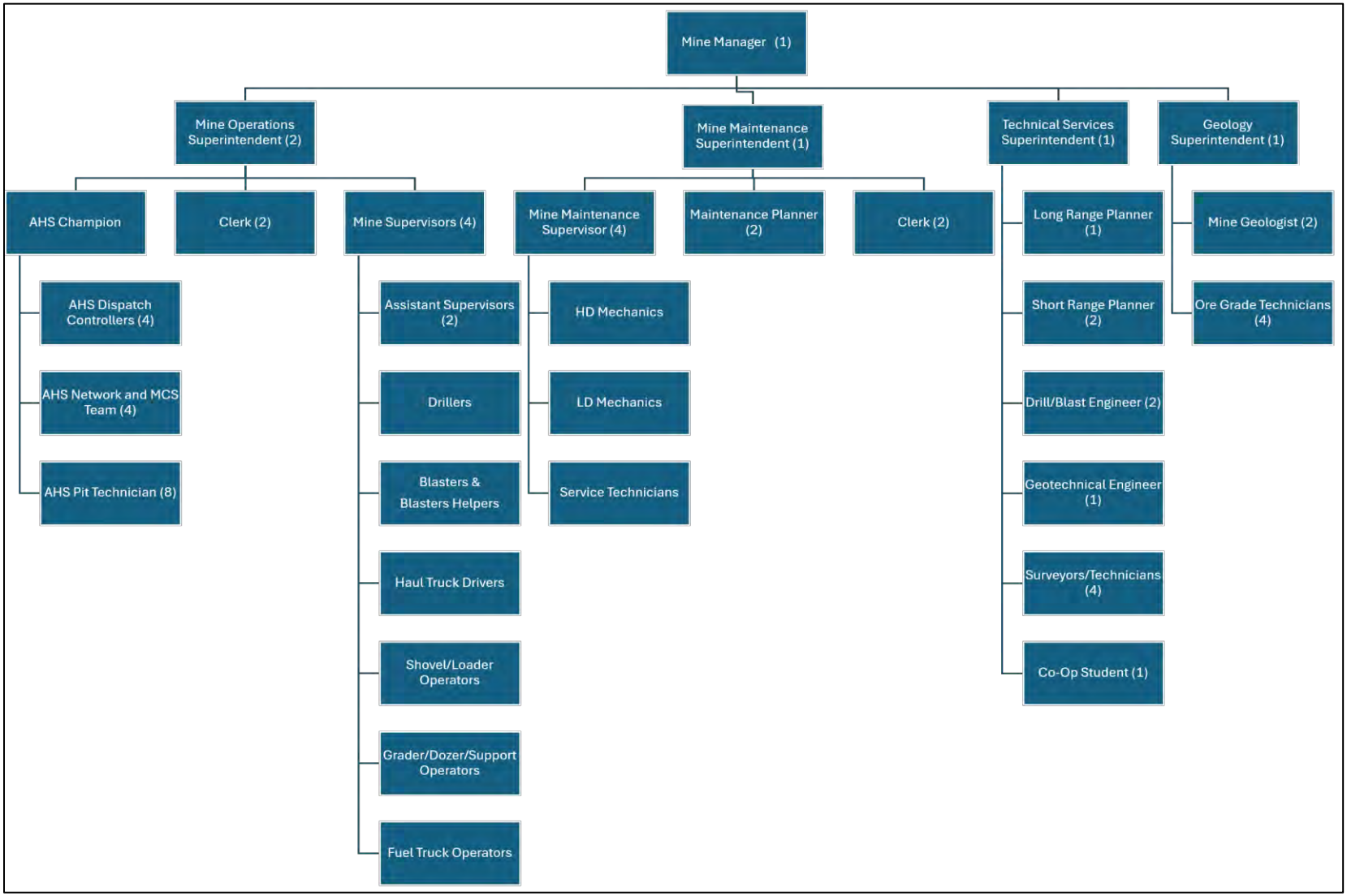
Sisson benefits by proximity to personnel and services based in and around Fredericton, which allows flexibility and competition in the range of vendors and available services on site. These services range from cleaning and uniform services, to cranes, rentals, maintenance and other contractors, and many other types of services and products.

16.8.1 Organization

In this study, direct mining and mine maintenance are planned as an owner-operated fleet with the equipment ownership and labour being directly under mine operations. Contracting out some of the direct mining activities under typical mine stripping contracts, and maintenance and repair contracts (MARC) for major fleet units has not been evaluated in this study. Mining activities are divided into three major departments:

1. Operations
2. Maintenance
3. Technical Services.

Figure 16-47: Mine operations organization chart



Source: MMTS, 2026

16.8.2 Drilling, grade control and production monitoring

Mining production drills will be fitted with High Precision GPS navigation and drill control systems to optimize drilling. Drills will be fitted with and controlled by tele-remote operating systems to improve the comfort and safety of the operating environment, improve drilling efficiency, and reduce the overall number of drill operators required.

Diesel hydraulic down-the-hole (DTH) units with 165 mm bit size are used for all drilling activities for pioneering, production, highwall, and quarry.

A reverse circulation (RC) drill is included to provide grade control drilling on a separate pattern, with 100% of ore tonnes and 25% of barren rock tonnes to have grade control drilling. An ore control system is planned to provide field control for the loading equipment to selectively mine ore-grade material separately from the barren rock.

16.8.3 Blasting and explosives

Blasting at the Project is required for all material types to provide suitable fragmentation for efficient loading and hauling of both ore and barren rock. All blasting requirements are planned as a contractor operation, where the technical design, optimization, and execution of the blast is performed by a full-service blasting company. Blasting is planned to use an emulsion product. Preliminary estimates for powder factors of 0.20 kg/t for ore and 0.17 kg/t for barren rock have been derived from a fragmentation study completed by MMTS in 2012 (MMTS D&B Study_rev2 2012 by KG.pdf, MMTS). Supporting studies completed by Orica and Dyno Nobel solidified the results of the study.

There is no planned site mixed explosives (SME) facility, storage of explosive or transfer site for the project; explosives will be transported from nearby facilities (Dyno Nobel -Fredericton plant, and Orica-Minto NB). As such no offset distances from any facilities is required.

For the Feasibility design, the explosives contractor is responsible for the following equipment and infrastructure:

- Installation of a garage and wash facility for blasting equipment, bulk trucks, and light vehicles
- Supply and operation of SME product delivery trucks for delivering product to the blastholes
- Supply and operation of blasthole stemming loader, material to be supplied by the mine.

Blasting services, 100% contractor performed, consist of:

- Full capability blasting crew, scheduled out of Fredericton for a five day per week operation, eight to nine-hour dayshift only
- Supervisor onsite
- Technical and design support
- All crew transport, safety supplies, training, and any permit requirements.

16.8.4 Loading and hauling

Production loading will be performed by 300 t electric-hydraulic shovel units. Electric-hydraulic units have been chosen due to the proximity of electric infrastructure and the efficiency of electric drive units versus diesel units. Electric drive units achieve reduced operating costs and maintenance down-time compared to the diesel units. The capital cost difference between the configurations is not significant.

Additional loading units include a 200 t diesel hydraulic backhoe. This unit provides high mobility and flexibility for effective digging in a variety of other digging conditions, such as pioneering benches, pit bottoms, and other loading scenarios.

The mine also includes a large wheel loader with the same bucket capacity as the 300 t shovel (13 to 15 m^3). The wheel loader provides high mobility to move between the pit and the quarry and serves as an effective backup unit if a 300 t shovel is down for maintenance.

16.8.5 Support and ancillary

Loading and hauling require important pit services for support. Mine pit services include:

- haul road maintenance
- pit floor and ramp maintenance
- barren rock and ore pile maintenance, including spreading and compaction of TMF destined materials
- fuel and lube services
- topsoil excavation
- secondary blasting and rock breaking
- snow removal
- lighting
- transporting personnel and operating supplies
- mine rescue.

The equipment fleet for the above functions is listed in Section 16.9 below.

Direct mining operations, mine equipment fleet ownership, mine fleet maintenance, and technical services are all planned as owner-managed functions.

Mining operations are based on 365 operating days per year with two 12-hour shifts per day. An allowance of 15 days of no production is built into the mine schedule to allow for adverse weather conditions, AHS troubleshooting and unplanned shutdowns.

The number of hourly mine operations personnel, including hourly maintenance personnel, will peak at 170 persons. Due to the shift rotation, only one-quarter of full personnel complement will be on shift at a given time. Salaried personnel of approximately 69 persons will be required for Mine Operations, Mine Maintenance, and Technical Services departments.

16.8.6 Pit dewatering

Pit dewatering is discussed in Section 18.6.6

16.9 Mining equipment

The mining equipment descriptions and specifications in this section provide general information of the size, dimension, capacity, etc. of the selected equipment. These specifications are not intended to target equipment from any specific manufacturer or vendor.

Table 16-12: Mine fleet – major equipment

	Y-2	Y-1	Y1	Y2-Y5	Y6-Y10	Y11-Y15	Y16-Y20	Y21 -Y27
Drilling								
Diesel DTH Tracked Drill 162 mm (7") holes	2	2	3	3	3	3	3	5
Diesel RC Tracked Drill 162 mm (7") holes	0	1	1	1	1	1	1	1
Loading								
300 t Electric HEX 16 m^3 bucket	0	1	1	1	2	2	2	2
200 t Diesel HEX 12 m^3 bucket	1	1	1	1	1	1	1	1
Wheel Loader 15 m^3 bucket	1	1	1	1	1	1	1	1
Hauling								
Rigid Frame Haul Truck 147 t payload	4	9	9	10	16	16	16	13
Articulated Haul Truck 40 t payload	2	2	2	2	2	2	2	2
Haul Road Maintenance/BRSA Maintenance/Construction Support								
Motor Grader 5.4 m blade	2	2	3	3	3	3	3	3
Water/Gravel Truck 40 t Articulated	2	2	2	2	2	2	2	2
Track Dozer 450 kW	2	3	3	3	4	4	4	4

Source: MMTS, 2026

Table 16-13: Mine fleet – support equipment

Unit	Function	Maximum Units
Wheel Dozer (370 kW)	Pit Maintenance, Shovel Support, Snow Clearing, Site Prep, Construction	1
Wheel Loader (6 m^3 Bucket)	Pit Maintenance, Shovel Support, Snow Clearing, Site Prep, Construction. Cable reeler on 2nd unit	2
HEX (3 m^3 Bucket)	Construction Activities, Quarry Support, Site support	2
Fuel and Lube Truck (40 t)	Fuel/lube support of excavators, wheel loader, drills and support equipment	2
Shuttle Bus	Employee Transportation	4
Pickup Trucks	Staff Transportation	8
Light Plants (6 kW)	Pit Lighting	6
On-Highway Dump Truck	Utility Material Movement	2
Flatbed Picker Truck	Material Transport, Pump Crew Support	2
Emergency Response Vehicle	First Aid and Mine Rescue	2
Maintenance Trucks	Mobile Maintenance crew and tool transport	4
Mobile Crane (90 t capacity)	Mobile Maintenance material handling	1
Float Trailer (150 t capacity)	Equipment Transport	1
Forklift (15 t capacity)	Shop Material and Tire Handling	2
Scissor Lift	Maintenance Support	1
Mobile Manlift	Mobile Maintenance Support	1

Source: MMTS, 2026

17 Recovery methods

17.1 Overall process design

The Sisson process plant features a flowsheet based on the metallurgical test work discussed in Section 13. The process plant has been designed to treat run-of-mine (ROM) ore through a crushing, grinding, and flotation flowsheet to produce separate molybdenum and tungsten concentrates. The comminution section of the selected flowsheet incorporates a three-stage crushing circuit and a single-line ball-mill grinding circuit operating in closed circuit with a cyclone cluster.

Primary cyclone overflow will feed a series of flotation circuits designed to recover molybdenum first, followed by bulk sulphides, and finally tungsten.

Molybdenum flotation will include rougher, regrind, and multi-stage cleaning stages to produce a saleable concentrate. The final molybdenum concentrate is thickened, filtered, dried and bagged for transportation to markets.

The tailings from molybdenum flotation will be directed to the adjoining Bulk Sulfide Flotation (BSF) circuit, where pyrite and other sulfide minerals are recovered to limit their potential impact on the downstream scheelite flotation process. The resulting BSF concentrate is combined with the PAG molybdenum cleaner-scavenger tailings. This combined stream is sent to the tailings storage facility (TSF) for sub-aqueous deposition to minimize oxidation of sulfide-bearing material.

The BSF tailings will proceed through the ambient tungsten rougher and scavenger flotation circuits. Rougher concentrate is cleaned in three stages, and the third-stage cleaner concentrate advances to the hot flotation circuit. After hot rougher, scavenger, and cleaner flotation, the final concentrate is thickened, filtered, dried, and packaged for transportation to markets.

Hot rougher-scavenger tailings, containing low levels of sulphides, are disposed to the TSF along with tungsten rougher-scavenger tailings stream from the ambient tungsten flotation circuit.

Key considerations for selecting the process flowsheet included the variable mineralogy, feed grades and rock hardness in the deposit. Multiple trade-off studies were conducted to select the optimal process flowsheet, including throughput assessment and comminution equipment selection. High-pressure grinding rolls (HPGR) are included to improve energy efficiency and enhance liberation ahead of tertiary screening and grinding. Minimizing scheelite slimes generation was an important criterion in selecting primary comminution equipment due to their documented impact on tungsten flotation recovery.

The overall design reflects industry-standard equipment selection, robust operability, and maintainability considerations suitable for a large-scale, long-life open-pit operation.

The process plant is designed to operate at an average nominal throughput of 30,000 t/d. A summary of the expected process operation availabilities is as follows:

- Primary crushing availability of 75%.
- Secondary and tertiary crushing, grinding, flotation availability of 90%.
- Concentrates filtration availability of 80%.

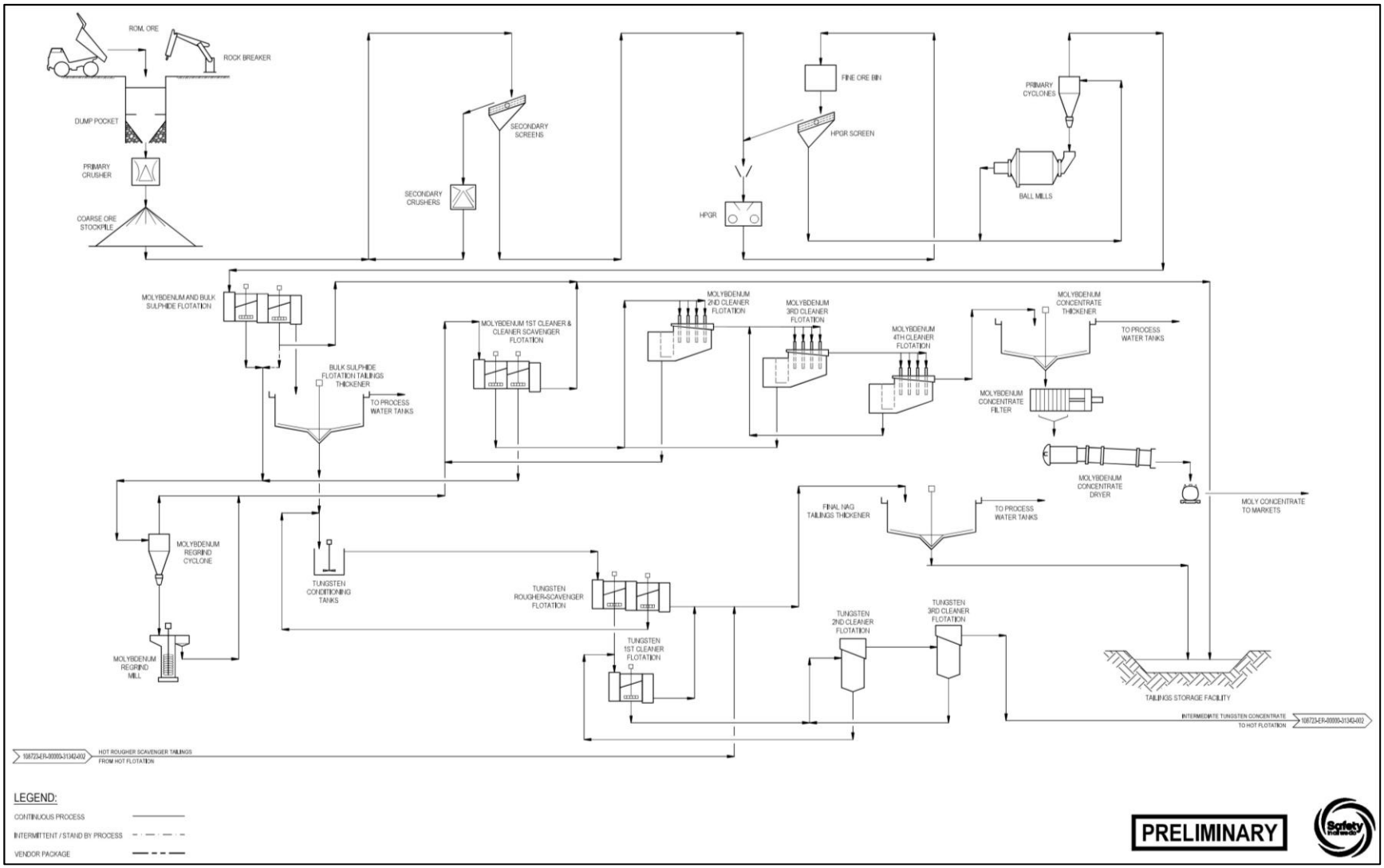
17.2 Process plant description

The process design considers the following unit operations and circuits:

- Three-stage crushing, conveying, and stockpiling of ROM ore
- Single stage, single-line grinding and hydrocyclone classification
- Molybdenum rougher, scavenger and cleaner flotation
- Molybdenum regrinding
- Molybdenum concentrate dewatering and loadout
- Bulk sulphide rougher flotation
- Tungsten rougher, rougher scavenger, and cleaner flotation
- Hot conditioning, rougher, rougher-scavenger, and cleaner flotation
- Tungsten concentrate dewatering and loadout
- Tailings thickening and transportation
- Reagent preparation and distribution
- Air services, water services and utilities.

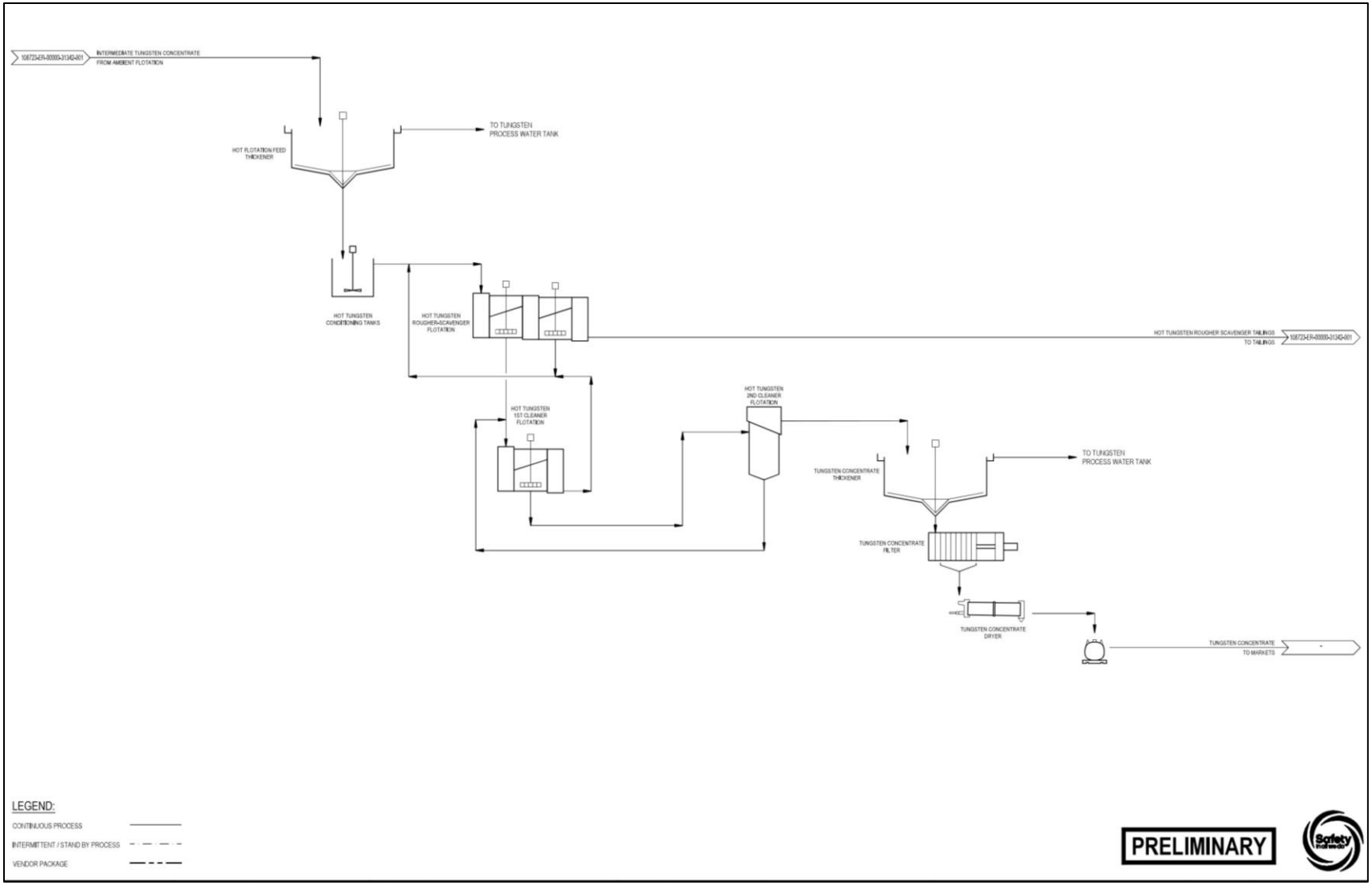
An overall process flow diagram showing the unit operations in the process plant is presented in Figure 17-1 and is described in the sections below. The proposed plant layout is presented in Figure 18-2.

Figure 17-1: Process flowsheet, ambient flotation



Source: Ausenco, 2026

Figure 17-2: Process flowsheet, hot flotation



Source: Ausenco, 2026

17.2.1 Process design criteria

Key process design criteria are listed in Table 17-1.

Table 17-1: Process design criteria

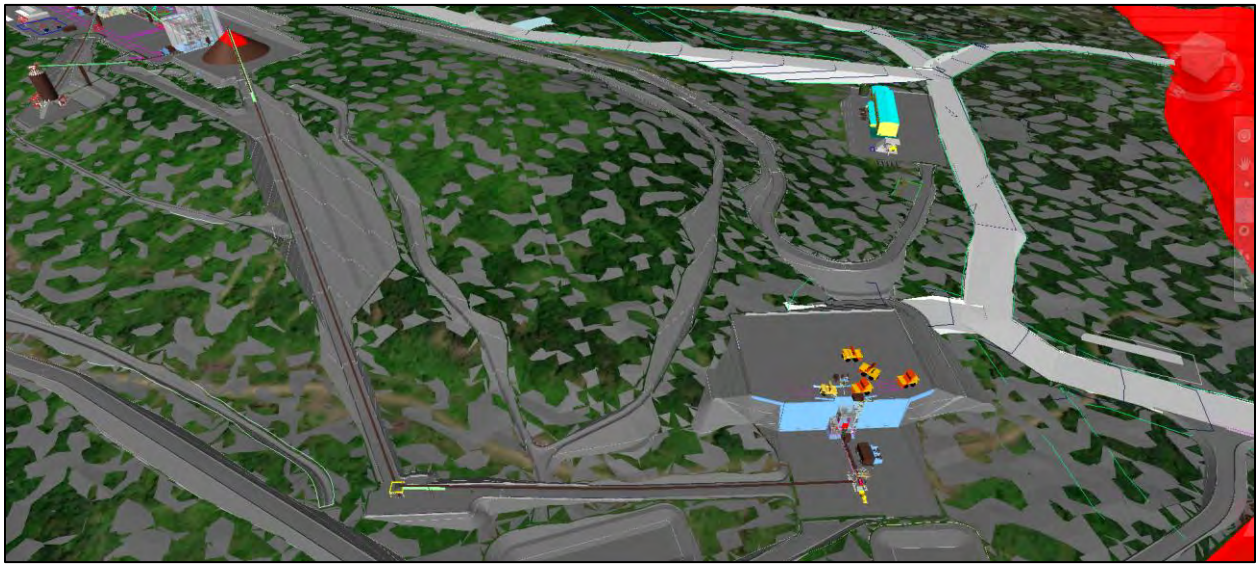
Description	Units	Value
Annual throughput, nominal	Mt/a	10.95
Operating hours per year, primary crushing	h/a	6,570
Operating hours per year, grinding and flotation	h/a	7,884
Operating hours per year, filtration	h/a	7,008
Plant feed grade, tungsten - design	% WO_3	0.110
Plant feed grade, molybdenum - design	% Mo	0.027
Plant feed grade, sulphur - design	% S_T	0.450
SMC ore competency (Axb)	-	25.0
Bond crushing work index	kWh/t	15.8
Bond rod mill work index	kWh/t	17.9
Bond ball mill work index	kWh/t	17.3
Abrasion index	g	0.502
Primary crushing feed size, 100% passing	mm	1,000
Tertiary crushing product size, 80 % passing	mm	2.6
Grinding product size, 80% passing	µm	140
Ball mill circulating load	%	350
Molybdenum rougher concentrate mass pull	% plant feed	2.7
Bulk sulphide rougher concentrate mass pull	% plant feed	2.0
Molybdenum regrind product size, 80% passing	µm	20
Tungsten rougher concentrate mass pull	% plant feed	8.10
Tungsten rougher scavenger concentrate mass pull	% plant feed	2.23
Hot rougher concentrate mass pull	% hot flotation feed	78.70
Hot rougher scavenger concentrate mass pull	% hot flotation feed	11.70
Molybdenum concentrate thickener unit area settling rate	$(t/h)/m^2$	0.10
BSF tailings thickener unit area settling rate	$(t/h)/m^2$	0.70
NAG tailings thickener unit area settling rate	$(t/h)/m^2$	0.70
Tungsten concentrate thickener unit area settling rate	$(t/h)/m^2$	0.25
Molybdenum filter, filtration rate	$(kg/h)/m^2$	330
Molybdenum concentrate filter cake moisture content	% water (w/w)	15
Molybdenum concentrate dryer product moisture content	% water (w/w)	5
Tungsten filter, filtration rate	$(kg/h)/m^2$	330
Tungsten concentrate filter cake moisture content	% water (w/w)	15
Tungsten concentrate dryer product moisture content	% water (w/w)	0.5
Molybdenum recovery to molybdenum concentrate	%	86
Molybdenum final concentrate grade	%Mo	50.0
Tungsten recovery to final tungsten concentrate	%	82
Final tungsten concentrate grade	%WO_3	50

17.3 Plant design criteria

17.3.1 Primary crushing and stockpiling

The primary crushing circuit is designed for an annual operating availability of 75% at a nominal capacity of 30,000 t/d.

Figure 17-3: Primary crushing, conveying, and coarse ore stockpiling



Source: Ausenco, 2026

ROM ore will be directly tipped into the primary crusher dump pocket which will have a capacity for 135 t. The dump pocket will be equipped with a hydraulic rock breaker to break any oversized rocks. ROM ore will flow by gravity into the primary crusher which will be choke fed to the greatest extent possible.

Any oversized ore received from the haul trucks will be broken down using the hydraulic rock breaker. The primary gyratory crusher is designed to reduce the ore to an 80% passing product size (P_{80}) of 120 mm. Crushed ore will be discharged to the primary crusher discharge vault and drawn out by the primary crusher discharge apron feeder. The feeder will discharge onto the primary crusher discharge conveyor which will be equipped with a weightometer and a self-cleaning magnet. The primary crusher discharge conveyor will move the crushed ore onto a transfer conveyor, which in turn will discharge to the coarse ore stockpile feed conveyor which will feed the coarse ore stockpile.

Dry dust collection systems will be installed at the primary crusher vault and conveyor transfer points, with collected particulates returned to the primary crusher discharge conveyor.

The area will include the following major equipment and facilities:

- primary gyratory crusher (447 kW)
- hydraulic rock breaker
- primary crusher apron feeder (1,500 mm pan width)
- primary crusher discharge conveyor (1,524 mm belt width, 40 m long, 4 m vertical lift)

- primary crusher transfer conveyor (1,524 mm belt width, 273 m long, 14 m vertical lift)
- coarse ore stockpile feed conveyor (1,524 mm belt width, 594 m long, 93 m vertical lift).

17.3.2 Coarse ore stockpiling and reclaim

Crushed ore will be reclaimed from the stockpile to feed the secondary crushing area by two parallel apron feeders. The feeders will transfer material onto the secondary screen surge bin conveyor, which will be equipped with a weightometer and self-cleaning magnet. Dry dust collection will be employed at conveyor transfer points within the stockpile and reclaim area. Particulate matter will be returned to the secondary screen surge bin conveyor.

The area will include the following major equipment and facilities:

- Two coarse ore reclaim apron feeders (1,500 mm pan width).
- Secondary screen surge bin conveyor (1,828 mm belt width, 256 m long, 28 m vertical lift).

17.3.3 Secondary crushing and screening

The secondary and tertiary crushing areas are illustrated in Figure 17-4 and described further in the following sections.

Figure 17-4: Secondary and tertiary crushing facilities



Source: Ausenco, 2026

Ore is removed from the secondary screen surge bin by a belt feeder and fed to a double-deck vibrating secondary screen. Oversize from the screen will be discharged onto the secondary screen oversize conveyor which will feed the secondary crusher surge bin feed conveyor. The secondary crusher surge bin is designed with two outlets to service the secondary crushers that are installed in duty/standby service. Ore from the surge bin will be drawn out by belt feeder and fed to the operating secondary crusher, which will discharge material onto the secondary screen surge bin conveyor for additional screening.

Secondary screen undersize will drop onto the secondary screen undersize conveyor and then be forwarded to the HPGR surge bin feed conveyor. The secondary screen undersize conveyor is equipped with a weightometer to monitor the crushing circuit throughput.

The major equipment in this area includes the following:

- two secondary crusher belt feeders (one duty and one standby, 1,524 mm belt width, 14 m long)
- two secondary cone crushers (one duty and one standby, 750 kW each)
- secondary screen surge bin feed conveyor (1,372 mm belt width, 218 m long, 25 m vertical lift)
- secondary screen belt feeder (2,134 mm belt width, 14 m long)
- secondary screen (double deck, 80 mm and 50 mm aperture sizes, 4.3 m width x 8.5 m length)
- secondary screen oversize conveyor (1,372 mm belt width, 13 m long)
- secondary screen undersize conveyor (1,219 mm belt width, 20 m long).

17.3.4 Tertiary crushing and screening

Secondary screen undersize will be transported to the HPGR surge bin by the HPGR surge bin feed conveyor equipped with a self-cleaning magnet. Ore from the HPGR surge bin will be delivered to the HPGR by a belt feeder equipped with a metal detector. If metal is detected, the metal detector will activate the diverter chute which will bypass the HPGR momentarily and direct the ore onto the HPGR discharge conveyor to clear tramp metal without interrupting the crusher operation. The HPGR conveyor equipped with a weightometer will transport crushed ore to the fine ore bin.

The fine ore reclaim feeder will transfer ore from the fine ore bin to the fine ore conveyor which will transport ore to the tertiary screen feed bin. Material will be reclaimed by two parallel belt feeders, each discharging into a feed box for water addition before feeding to a tertiary screen. Oversize from the tertiary screens will be discharged onto a common tertiary screen oversize conveyor and fed onto the HPGR surge bin feed conveyor for additional size reduction via the HPGR. Screen undersize from the two HPGR screens will directly feed the primary cyclone feed pump box.

Dry dust collection will be employed at conveyor transfer points in the secondary crushing areas. Particulate matter will be returned to the secondary screen surge bin. Wet dust suppression will be employed at conveyor transfer points and on the screens within the tertiary crushing and screening areas.

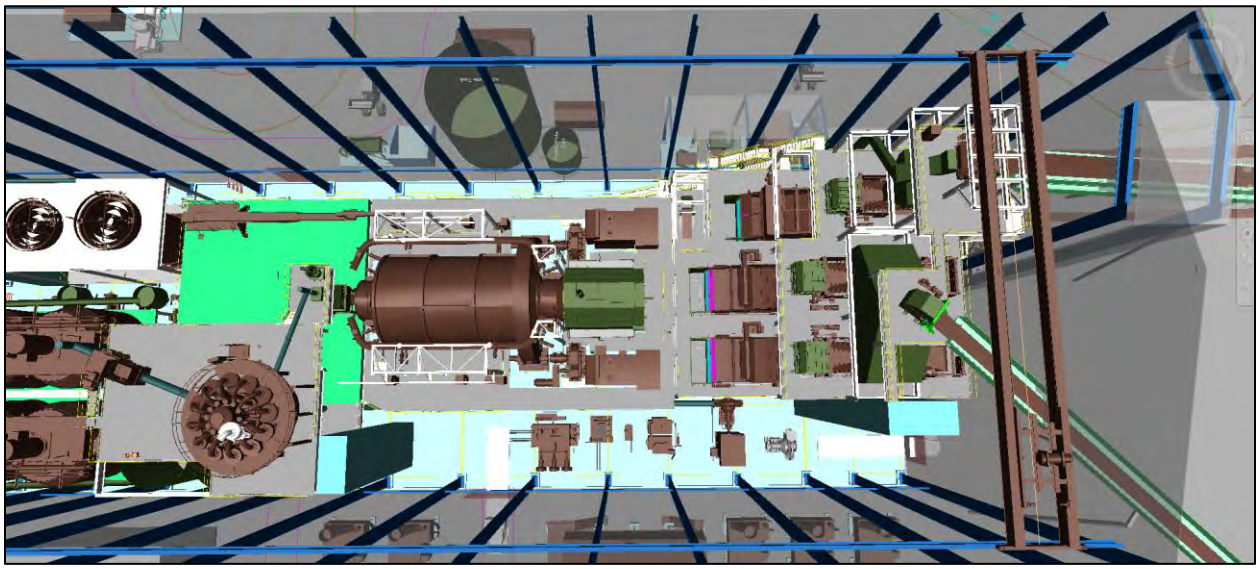
The major equipment in this area includes the following:

- HPGR surge bin feed conveyor (1,524 mm belt width, 234 m long, 39 m vertical lift)
- HPGR belt feeder (1828 mm belt width, 17 m long, 2 m vertical lift)
- HPGR (7,300 kW, 2.2 m diameter x 2.0 m width rolls)
- HPGR discharge conveyor (1,524 mm belt width, 159 m long, 26 m vertical lift)
- fine ore reclaim belt feeder (1,828 mm belt width, 17 m long, 2 m vertical lift)
- fine ore conveyor (1,372 mm belt width, 266 m long, 45 m vertical lift)
- two tertiary screen belt feeders (1,828 mm belt width, 11 m long)
- two tertiary screens (each double deck, 8 mm and 4 mm aperture sizes, 4.3 m width x 8.5 m length)
- tertiary screen oversize conveyor (1,219 mm belt width, 16 m long).

17.3.5 Grinding and classification circuit

The grinding circuit is designed to operate in a single line consisting of a ball mill and cyclone cluster in closed circuit. The circuit is sized for a throughput of 30,000 t/d.

Figure 17-5: Grinding and classification circuit



Source: Ausenco, 2026

Crushed ore from tertiary screen undersize will feed the primary cyclone pump box and be classified in the primary cyclone cluster. Cyclone overflow will feed the flotation circuit while the underflow will gravitate to the ball mill.

The circuit is sized based on a grinding circuit P_{80} of 140 µm. The ball mill will be equipped with a trommel screen to remove scats from the mill discharge. The scats will be temporarily placed in the scats tote bin and periodically cleared out by a front-end loader for disposal.

Process water is added to the primary cyclone feed pump box and mill feed inlet as required for density adjustment. Diesel (molybdenum collector) dosing will take place at the primary cyclone feed pump box to precondition the ore for flotation.

The primary cyclone overflow will be sampled for particle size analysis and metallurgical accounting.

The major equipment in this area includes the following:

- ball mill (dual pinion, 20 MW, 8.53 m dia. X 14.2 m EGL)
- primary cyclone cluster (650 mm diameter, 8 duty and 2 spares).

17.3.6 Flotation

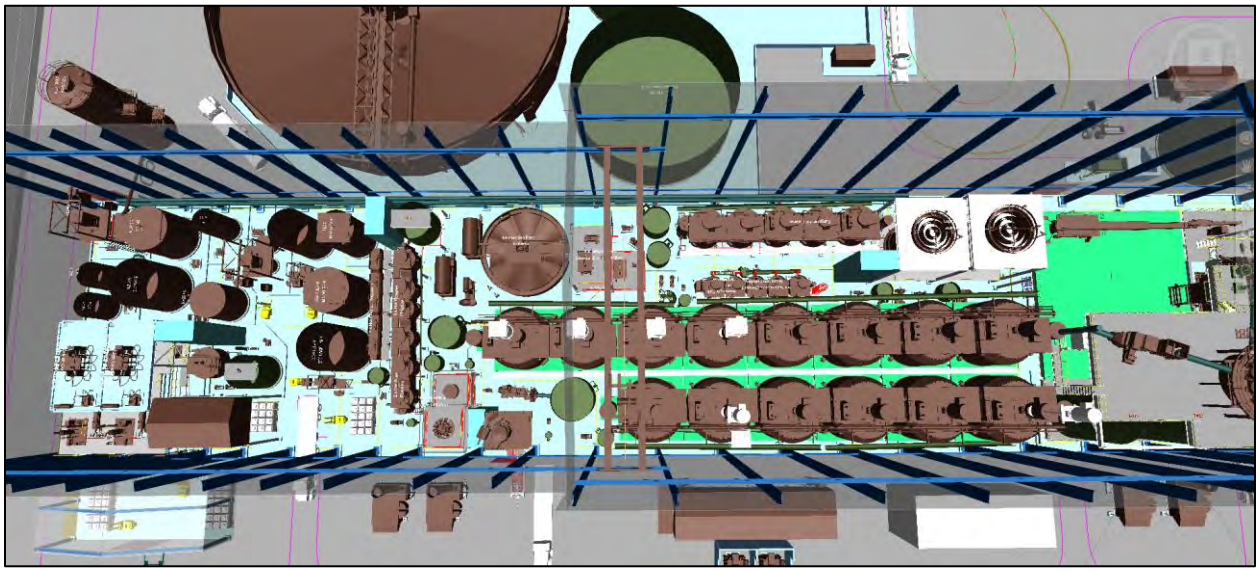
The flotation plant will include:

- molybdenum rougher and cleaner flotation
- molybdenum rougher concentrate regrinding
- bulk sulphide rougher flotation

- tungsten rougher and cleaner flotation
- hot rougher and hot cleaner flotation.

Each circuit is configured to maximize recovery and concentrate grade through staged cleaning, scavenging, and reagent conditioning. The area is illustrated below in Figure 17-6.

Figure 17-6: Flotation area



Source: Ausenco, 2026

17.3.6.1 Molybdenum and bulk sulphide rougher flotation

Primary grinding cyclone overflow will be gravity fed to the molybdenum rougher flotation circuit. The molybdenum and bulk sulphide rougher flotation cells will consist of a single bank of conventional forced air tank cells. The first five cells will float a selective molybdenum rougher concentrate and the remaining three a bulk sulphide concentrate. Valving on the sixth cell launder piping will allow flexibility for the froth to join molybdenum concentrate or BSF concentrate depending on molybdenum and sulphur grades.

Molybdenum rougher concentrate will be sent to a regrind circuit for further liberation prior to upgrading in the cleaning circuit. Tailings from the final molybdenum rougher flotation cell will flow into the first bulk sulphide flotation cell. Concentrate from the bulk sulphide flotation cells will be sent to the molybdenum cleaner scavenger tailings pumpbox prior to forwarding to the tailings area. Tailings from the bulk sulphide flotation cells will be pumped to the bulk sulphide flotation tailings thickener ahead of the tungsten flotation circuit.

Process water will be added as needed at various points across the rougher flotation cells to maintain desired densities and provide froth/launder mobility. Pine oil, diesel, and methyl isobutyl carbinol (MIBC, frother) will be dosed into the molybdenum rougher flotation cells. Potassium amyl xanthate (PAX, sulphide collector) and additional MIBC will be dosed into the bulk sulphide rougher flotation cells.

The major equipment in this area includes the following:

- Five molybdenum rougher cells (300 m^3/cell)
- Three BSF rougher cells (300 m^3/cell).

17.3.6.2 Molybdenum regrind and cleaner flotation

The rougher molybdenum concentrate will flow by gravity to the molybdenum regrind cyclone feed pump box where it will blend with the molybdenum cleaner scavenger concentrate. The combined slurry will then be pumped to a regrind cyclone for dewatering and classification. The cyclone underflow will be gravity fed to the regrind mill feed pump box and pumped to the regrind mill for size reduction. The regrind mill will discharge into the molybdenum first cleaner feed pump box where it will blend with overflow from the cyclone. The combined regrind circuit discharge targets a product size P_{80} of 20 µm.

The regrind circuit product stream will then be pumped to the molybdenum cleaner flotation area where it will be combined with molybdenum cleaner scavenger concentrate prior to entering the first stage of cleaner flotation. The molybdenum cleaner flotation circuit will consist of four stages: one stage utilizing conventional forced air tank cells that includes a scavenger stage, and three stages utilizing pneumatic cells with internal recycles. Concentrate streams from a cleaner stage will flow to the next cleaner stage, while tailings will recycle to the previous stage. Concentrate from the final cleaner stage will be pumped to the concentrate handling area. Cleaner scavenger tailings will be blended with the bulk sulphide rougher concentrate and sent to the tailings facility as final PAG tailings.

Process water will be added as needed at various points across the cleaner flotation cells to maintain desired densities, to provide froth washing, and to provide froth/launder mobility. Diesel, MIBC, sodium cyanide (NaCN, sulphide depressant), and sodium sulphide (Na_2S, sulphide depressant) will be dosed into the molybdenum cleaner flotation cells at various points.

The major equipment in this area includes the following:

- molybdenum regrind cyclone cluster (150 mm diameter, 5 duty and 2 spares)
- molybdenum regrind mill (1,800 kW)
- three molybdenum first cleaner cells (10 m/cell)
- two molybdenum cleaner scavenger cells (10 m^3/cell)
- molybdenum second cleaner cell (pneumatic, single downcomer)
- molybdenum third cleaner cell (pneumatic, single downcomer)
- molybdenum fourth cleaner cell (pneumatic, single downcomer).

17.3.6.3 Tungsten rougher flotation

Tailings from the bulk sulphide rougher flotation cells will be pumped to bulk sulphide flotation tailings thickener ahead of the tungsten flotation circuit.

Underflow from the BSF tailings thickener will be pumped to the tungsten flotation area where they will pass through a series of conditioning tanks prior to rougher flotation. Tungsten rougher and rougher scavenger flotation will consist of a single bank of conventional forced air tank cells. Rougher concentrate will be forwarded to the tungsten cleaner flotation area for further upgrading. Rougher scavenger concentrate will be returned to the conditioning tanks for additional preparation and rougher flotation.

Rougher scavenger tailings will be collected and sent to the final non-acid-generating (NAG) tailings thickener for thickening. Underflow from the thickener will be pumped the tailings storage facility as the final NAG tailings. Overflow will report to the tungsten process water tank.

Process water will be added as needed at various points across the conditioning tanks and rougher flotation cells to maintain desired densities, and to provide froth/launder mobility. Sodium carbonate (Na_2CO_3, tungsten activator), sodium hydroxide (NaOH, pH control), sodium silicate (Na_2SiO_3, sulphide depressant) and fatty acid (tungsten collector) will be dosed into the conditioning tanks, rougher flotation cells, and rougher scavenger flotation cells at various points.

The major equipment in this area includes the following:

- BSF tailings thickener (49 m diameter)
- two tungsten rougher flotation conditioning tanks (120 m^3 each)
- three tungsten rougher cells (300 m^3/cell)
- three tungsten rougher scavenger cells (300 m^3/cell)
- final NAG tailings thickener (49 m diameter).

17.3.6.4 Tungsten cleaner flotation

Tungsten rougher concentrate will be fed to the tungsten cleaner flotation area where it will be combined with tungsten second cleaner tailings prior to entering the first stage of cleaner flotation. The tungsten cleaner flotation circuit will consist of three stages: one stage utilizing conventional forced air tank cells and two stages utilizing column cells. Concentrate streams from a cleaner stage will flow to the next cleaner stage, while tailings will recycle to the previous stage. Concentrate from the final cleaner stage will be pumped downstream to the hot flotation circuit. First cleaner tailings will be sent back to the conditioning tanks for additional flotation passes.

Process water will be added as needed at various points across the cleaner flotation cells to maintain desired densities, to provide froth washing, and to provide froth/launder mobility. Na_2SiO_3 will be dosed into the cleaner flotation cells at various points.

The major equipment in this area includes the following:

- two tungsten first cleaner cells (20 m^3)
- tungsten second cleaner column cell (2 m diameter and 8 m height)
- tungsten third cleaner column cell (2 m diameter and 6 m height).

17.3.6.5 Hot rougher flotation

Concentrate from tungsten third cleaner flotation circuit will feed the hot flotation feed thickener via trash screen. Overflow will be collected in a tank before being pumped to the tungsten process water tank. Underflow from the hot flotation feed thickener will be pumped to the hot flotation area where it will pass through a hot conditioning tank prior to rougher flotation.

The hot conditioning process will be conducted at 90°C, with heat being supplied by submersible electric heaters. The conditioned slurry will then be diluted with tungsten process water and allowed to cool and forwarded to the hot rougher flotation cells.

Hot rougher and rougher scavenger flotation will consist of a single bank of conventional forced air tank cells. Hot rougher concentrate will be forwarded to the hot cleaner flotation area for further upgrading. Hot rougher scavenger concentrate will be recycled to the front of the hot rougher flotation cells. Hot rougher scavenger tailings will be collected and sent to the tungsten rougher scavenger tailings pump box before being discharged to the tailings storage facility as the final NAG tailings.

Process water will be added as needed at various points across the hot conditioning tank and hot rougher flotation cells to maintain desired densities, and to provide froth/launder mobility. NaOH, Na_2SiO_3, Na_2S, and fatty acid will be dosed into the hot conditioning tank, hot rougher flotation cells, and hot rougher scavenger flotation cells at various points.

The major equipment in this area includes the following:

- hot flotation feed thickener (6 m diameter)
- hot conditioning tank (12 m^3)
- four hot rougher cells (0.5 m^3/cell)
- three hot rougher cells (0.5 m^3/cell).

17.3.6.6 Hot cleaner flotation

Hot rougher concentrate will be fed to the hot cleaner flotation area where it will be combined with hot second cleaner tailings prior to entering the first stage of hot cleaner flotation. The hot cleaner flotation circuit will consist of two stages: one stage utilizing conventional forced air tank cells and one stage utilizing a column cell. Concentrate streams from a hot cleaner stage will flow to the next hot cleaner stage, while tailings will recycle to the previous stage. Concentrate from the final cleaner stage will be pumped downstream to the tungsten final concentrate thickener. Tailings from the hot first cleaner circuit will be sent back to the hot rougher flotation circuit for additional flotation passes.

Process water will be added as needed at various points across the hot cleaner flotation cells to maintain desired densities, to provide froth washing, and to provide froth/launder mobility.

The major equipment in this area includes the following:

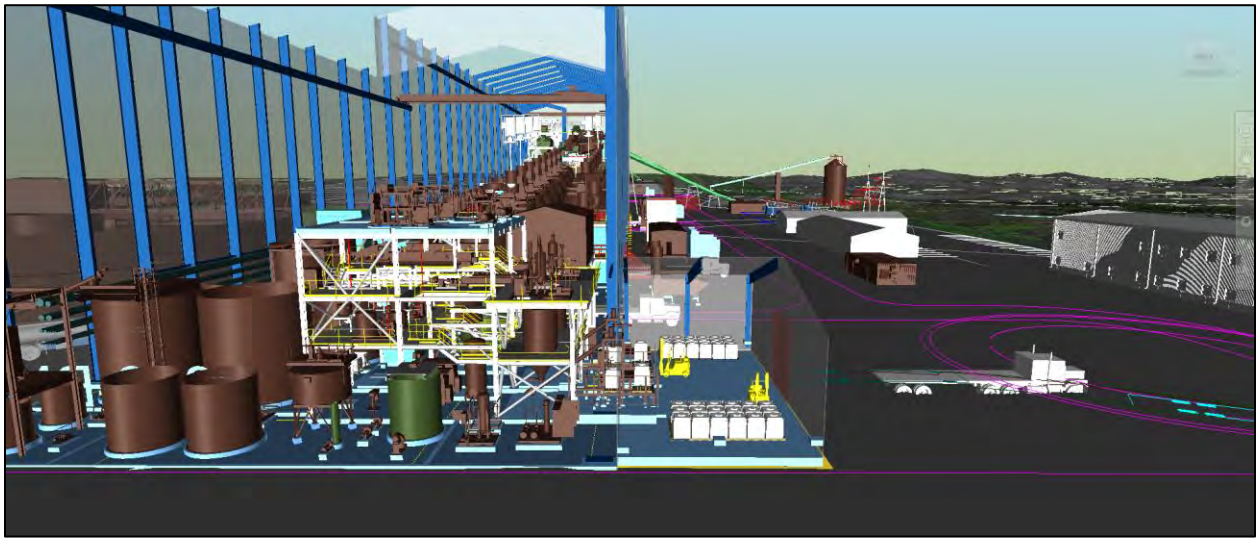
- three hot first cleaner cells (0.5 m^3/cell)
- hot second cleaner column cell (1.5 m diameter and 6 m height).

17.3.7 Concentrate dewatering and loadout

The design of each concentrate dewatering circuit consists of thickening and filtration equipment required to dewater the molybdenum and tungsten concentrates prior to loadout and shipment. Each concentrate stream will report to a dedicated high-rate thickener, where flocculant will be added to assist in the settling of the solids. The thickener overflows will be pumped to the dedicated process water tank, while the underflows will be fed to dedicated filter feed tanks each with a residence time of 12 hours.

The concentrate dewatering and loadout area is shown in Figure 17-7.

Figure 17-7: Concentrate dewatering, storage, and loadout



Source: Ausenco, 2026

17.3.7.1 Molybdenum concentrate filtration, drying and loadout

Thickened molybdenum concentrate at 60% w/w from the molybdenum concentrate thickener will be pumped to the concentrate filter area for further dewatering. Thickener underflow will enter a filter feed tank serviced by a centrifugal filter feed pump. The filter feed pump will continuously recirculate thickened concentrate back to the filter feed tank and will provide feed to the concentrate filter when required via an automated valve arrangement.

The filter will step through a filtration sequence to produce a filter cake at the target product moisture of 15%. The cake will discharge into a screw conveyor to be forwarded to a concentrate dryer. Filtrate from the concentrate filter will gravitate to the filtrate tank and pumped back to the molybdenum concentrate thickener. Cloth wash and manifold flush water will be provided for the filter on an as needed basis by dedicated pumps from a single wash water tank.

Filter cake from the molybdenum concentrate filter will discharge onto a screw feeder for transfer to the molybdenum concentrate dryer. The dryer will be an indirect heated rotary dryer that will further reduce the target moisture content to 5%. Final product will be discharged into a second screw feeder and transferred to a bagging system.

The dryer system will include an electrical heating system to provide the heat required for evaporation via a thermal oil. A wet scrubber will also be included for cleaning of off-gasses prior to discharging to atmosphere.

Final molybdenum concentrate will be discharged from the drying system into the concentrate bagging system. Material will then be bagged and labelled by operators for storage prior to transport offsite.

The major equipment in this area includes the following:

- Molybdenum concentrate thickener (3 m diameter)
- Molybdenum concentrate filter (vertical plate and frame, 5.9 m^2 filtration area)
- Molybdenum concentrate dryer
- Screw feeders.

17.3.7.2 Tungsten concentrate filtration, drying and loadout

Concentrate from hot cleaner 2 circuit will feed the tungsten concentrate via a trash screen. Overflow will be collected in a tank before being pumped to the tungsten process water tank. Thickened underflow will be pumped to the concentrate filter feed tank.

Thickened tungsten concentrate at 60% w/w from the tungsten concentrate thickener will be pumped to the tungsten concentrate filter feed tank. Thickener underflow will enter a filter feed tank serviced by a centrifugal filter feed pump. The filter feed pump will continuously recirculate thickened concentrate back to the filter feed tank and will provide feed to the tungsten concentrate filter when required via an automated valve arrangement.

The filter will step through a filtration sequence to produce a filter cake at the target product moisture of 15%. The cake will discharge into a screw conveyor to be forwarded to a concentrate dryer. Filtrate from the concentrate filter will gravitate to the filtrate tank and pumped back to the tungsten concentrate thickener. Cloth wash and manifold flush water will be provided for the filter on an as needed basis by dedicated pumps from a single wash water tank.

Filter cake from the tungsten concentrate filter will discharge onto a screw feeder for transfer to the tungsten concentrate dryer. The dryer will be an indirect heated rotary dryer that will further reduce the target moisture content to 5%. Final product will be discharged into a second screw feeder and transferred to a bagging system.

The dryer system will include an electrical heating system to provide the heat required for evaporation via a thermal oil. A wet scrubber will also be included for cleaning of off-gasses prior to discharging to atmosphere.

Final tungsten concentrate will be discharged from the drying system into the concentrate bagging system. Material will then be bagged and labelled by operators for storage prior to transport offsite.

Molybdenum tungsten concentrate handling circuits will be physically separated from one another to prevent cross-contamination.

The major equipment in this area includes the following:

- Tungsten concentrate thickener (3 m diameter)
- Tungsten concentrate filter (vertical plate and frame, 9.9 m^2 filtration area)
- Tungsten concentrate dryer
- Screw feeders.

17.3.8 Tailings disposal

The process plant will generate two distinct tailings streams: PAG tailings from the molybdenum and bulk sulphide flotation circuits, and NAG tailings from the tungsten flotation circuit.

PAG tailings, consisting primarily of molybdenum cleaner scavenger tailings blended with bulk sulphide rougher concentrate, will be pumped to the tailings storage facility for sub-aqueous deposition to minimize oxidation of sulfide-bearing material.

NAG tailings, generated from the tungsten rougher scavenger circuit, will be thickened in the final NAG tailings thickener prior to being discharged to the tailings storage facility.

Water reclaim systems will return supernatant water to the process plant for reuse, reducing freshwater demand and supporting overall site water balance.

The major equipment in this area includes the following:

- Final NAG tailings thickener (50 m diameter)

17.4 Energy, water, and process materials requirements

17.4.1 Consumption rates

Reagent consumptions are based on testwork results and standard industry practices. A summary of the nominal estimated reagent and consumable rates are presented on an annual basis in Table 17-2.

Energy consumption is based on designed equipment power demand and operating hours. Nominal energy consumption for the process plant including building heating loads and water systems is estimated at 363 GWh/a. A summary of the Project electrical demand is discussed in Section 18.

Table 17-2: Nominal annual consumption rates

Item	Delivery format	Consumption unit	Consumption rate
Diesel	bulk truck	t/a	450
Pine Oil (50%)	1000 L totes	t/a	330
MIBC	1000 L totes	t/a	110
NaCN	1 t bags	t/a	140
Na_2S	550 kg bags	t/a	490
PAX	900 kg bags	t/a	920
Sodium carbonate (Na_2CO_3)	bulk truck	t/a	11,000
NaOH (50%)	bulk truck	t/a	4,150
Sodium silicate (Na_2SiO_3)	bulk truck	t/a	60,910
Fatty acid collector (FS-2)	bulk truck	t/a	1,420
Flocculant	750 kg bags	t/a	530
Coagulant	750 kg bags	t/a	840
Antiscalant	1 t totes	t/a	22
Ball mill steel media	bulk truck	t/a	10,500
Regrind mill ceramic media	1 t bags	t/a	129
Fresh water	Wells/WTP	m^3/a	717,444

17.4.2 Reagents handling and storage

Reagents and chemicals for the process plants will be used in flotation, regrind and dewatering circuits. The reagents described in this section will be delivered in bulk or by specific container and stored onsite in separate and secure and designated areas inside or near the process plant buildings.

17.4.2.1 Diesel

Diesel will be used in molybdenum flotation to collect molybdenite mineral. Diesel will be shipped in tanker trucks and stored in a tank. Diesel will be dosed to the process by pumps as required.

17.4.2.2 Pine oil

Pine oil will be used in molybdenum flotation as activator. Pine oil will be delivered to site in 920 kg totes and dosed to the process by pumps as required.

17.4.2.3 Methyl isobutyl carbinol (MIBC)

Methyl Isobutyl Carbinol (MIBC), a conventional frother, will also be used in both molybdenum and bulk sulphide flotation. MIBC will be received on site in liquid form in 1,000 L totes. The solution will be dosed to the process by pumps as required.

17.4.2.4 Sodium cyanide (NaCN)

Sodium cyanide will be used as a sulphide depressant in molybdenum flotation. Sodium cyanide will be received on site as dry solids in 1 t bags. The bulk bag will be lifted by hoist into a bag breaker that will discharge into the sodium cyanide mixing tank. Raw water will be added and the contents agitated to make a 30% w/w solution. Sodium hydroxide will be provided as needed to maintain high pH and prohibit hydrogen cyanide (HCN) gas evolution. The solution will then be transferred to the sodium cyanide storage providing a residence time of 12 hours. The reagent will be dosed to the circuit by pumps as required.

17.4.2.5 Sodium sulphide (Na_2S)

Sodium sulphide will be used as sulphide depressant in molybdenum flotation. Sodium sulphide will be received on site as dry solids in 550 kg bags. The bulk bag will be lifted by hoist into a bag splitter that will discharge into the sodium sulphide mixing tank. Raw water will be added and the contents agitated to make a 20% w/w sodium sulphide solution. The solution will then be transferred to the sodium sulphide storage tank providing a residence time of 8 hours. The reagent will be dosed to the circuit by pumps as required.

17.4.2.6 Potassium amyl xanthate (PAX)

PAX will be used in the bulk sulphide flotation circuit to promote bulk flotation of the sulphide minerals. PAX will be received on site as dry solids in 900 kg bags. The bags will be emptied into a mixing tank and slurried to produce a solution at 15% concentration w/w. The mixed reagent will then be transferred to a storage tank providing a residence time of 12 hours. The reagent will then be dosed to the circuit by pumps as required.

17.4.2.7 Sodium silicate (Na_2SiO_3)

Sodium silicate will be used as a gangue depressant in the tungsten flotation. Sodium silicate will be delivered in solution form at 38% concentration w/w in tanker trucks and stored in a tank providing a residence time of 12 hours. Sodium silicate will be dosed to the circuit by pumps as required.

17.4.2.8 Sodium carbonate (Na_2CO_3)

Sodium carbonate will be used in tungsten flotation to enhance selective flotation of the scheelite mineral. Sodium carbonate will be received as a dry powder in bulk 25-30 t deliveries and pneumatically unloaded into the sodium carbonate storage bin. Material will be withdrawn to a mixing tank for dilution to 10% w/w solution strength. The mixed sodium carbonate will then be transferred to a storage tank providing a residence time of 12 hours. The reagent will be dosed to the circuit by pumps as required.

17.4.2.9 Sodium hydroxide (NaOH)

Sodium hydroxide will be used in tungsten flotation as pH control. NaOH (at 50% concentration) will be delivered in liquid form in tanker trucks and stored in a tank. The reagent will be dosed to the circuit by pumps as required.

17.4.2.10 Fatty acid (FS-2)

FS-2 will be used as a collector in the tungsten flotation. Fatty acid will be delivered in liquid form in tanker trucks and stored in a tank providing a residence time of 12 hours. FS-2 will be dosed to the circuit by pumps as required.

17.4.2.11 Flocculant

Flocculant will be used to aid in the settling of solids in thickening applications. Flocculant will be received on site as a dry powder in 750 kg bags. The bulk bag will be lifted by hoist and dumped into a hopper. A screw feeder will withdraw flocculant powder and forward to the mixing tank eductor and wetting head to avoid clumping. A blower will provide positive pressure for the eductor. Raw water will be added and the contents agitated to make a 0.5% w/w flocculant slurry. The slurry will then be transferred to the flocculant storage tank by gravity with a 12-hour design residence time. Flocculant will be pumped to the process as required by dosing pumps.

17.4.2.12 Coagulant

Coagulant will be used to aid in the tungsten tailings thickening application. Coagulant will be received on site as a dry powder in 750 kg bags. The powder will be mixed to a strength of 0.5% w/w. The solution will be stored in a dedicated tank with a 12-hour design residence time and pumped to the process as required by dosing pumps.

17.4.2.13 Antiscalant

Antiscalant will be used to prevent the build-up of scale in the process and tungsten process water pipes. It will be delivered to site in 1,000 kg totes.

17.4.3 Process plant water requirements

17.4.3.1 Process water

Process water will be primarily sourced from thickener overflow streams and reclaim water from the TSF. Raw water will be available for additional makeup on an as needed basis.

Horizontal centrifugal process water pumps in a duty/standby arrangement will supply process water to the various users throughout the plant site, predominantly to the grinding circuit. The process water tank will be constructed from mild steel and has a one-hour design residence time.

The area includes the following major equipment and facilities:

- process water distribution pumps (2 pumps, 600 kW/pump)
- process water tank (2,700 m^3 live volume).

17.4.3.2 Tungsten process water

Tungsten process water will be primarily sourced from thickener overflow streams and reclaim water from the TSF. Horizontal centrifugal process water pumps in a duty/standby arrangement will supply tungsten process water to the various consumers throughout the plant site, predominantly to the tungsten flotation circuit. The process water tank will be constructed from mild steel and has a 30-minute design residence time.

The area includes the following major equipment and facilities:

- tungsten process water distribution pumps (2 pumps, 450 kW/pump)
- tungsten process water tank (800 m^3 live volume)

17.4.3.3 Raw water

Raw water for the concentrator will be sourced from water treatment plant discharge and stored in the raw water tank with 8-hour design residence time.

The area includes the following major equipment and facilities:

- raw water tank (1000 m^3 live volume)

17.4.3.4 Gland seal water

Gland seal water will be sourced from the raw water tank, passed through filters to remove particulate, and pumped to various users throughout the concentrator. Gland water storage tank will have a one-hour design residence time.

The area includes the following major equipment and facilities:

- gland water distribution pumps (2 pumps, 10 kW/pump)
- two gland water filters.

17.4.3.5 Fire water

Fire water for the process plant will be sourced from 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 a fire water reticulation system that services the concentrator. The raw water tank level will maintain a minimum level of water for use by the firewater system.

17.4.3.6 Potable water

Potable water will be sourced from a dedicated potable water well, treated in a potable water system, and stored in a storage tank. The potable water tank will have a 48-hour design residence time.

The area includes the following major equipment and facilities:

- potable water treatment plant
- potable water tank (40 m^3 live volume)
- potable water distribution pumps (two pumps, 1 kW/pump)

17.4.4 Process plant air services

Plant air compressors will supply air at 750 kPa to various users throughout the concentrator, and an air dryer will provide instrument air as required.

The concentrate filters will have dedicated compressors to service the blowing, membrane squeezing, and drying requirements.

Flotation air blowers will provide lower pressure air at 74 kPa to the flotation cells.

The area includes the following major equipment and facilities:

- process plant flotation blowers (two blowers, 350 kW each)
- filtration air compressors (two compressor, 22 kW each)
- process plant air compressors (three compressors, 125 kW each).

18 Project infrastructure

18.1 Introduction

The proposed Sisson Mine is a greenfield site. The site is accessible from Highway 102/104 from the South (Primary Site Access) or from Highway 107 from the North (Secondary Site Access).

Infrastructure that will be required for the open pit mining and mineral processing operations will include the following facilities:

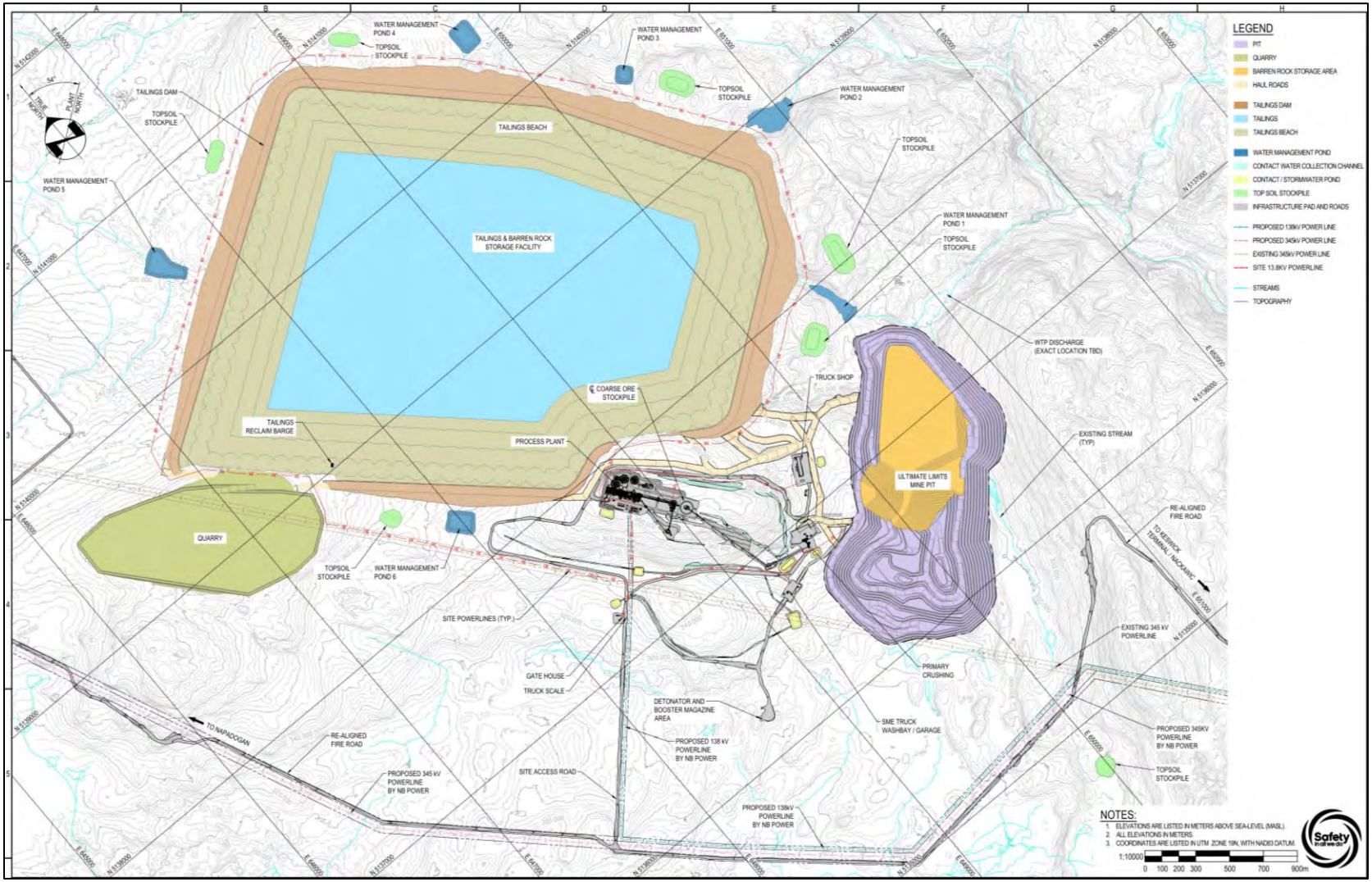
- mining facilities including truck shop, warehouse, offices, truck wash bay and garage, detonator and booster magazine, fuel storage and distribution;
- mineral processing facilities including process plant, crushing facilities, warehouse and maintenance shop, assay laboratory, freshwater infrastructure;
- quarry;
- water treatment plant;
- tailings storage facility (TSF);
- barren rock storage facility;
- general facilities including administration building, change house, potable water treatment plant, sewage treatment plant, gatehouse, weigh scale; and
- stormwater and contact water ponds, diversion ditches, and other site water management infrastructure.

The site facilities are generally arranged in accordance with the layout developed for the 2013 Feasibility Study (FS), which formed the basis of the approved Project Development Area (PDA) and Environmental Impact Assessment (EIA) approval. Consequently, the current site layout closely reflects the configuration established in the 2013 FS, with only minor refinements incorporated where required. Refer to Figure 5-1 for an outline of the PDA Boundary.

The overall site layout, Table 18-1, shows the major project facilities, including the open pit mine, tailings and barren rock storage facility, process plant, ancillary infrastructure, water management features, mine services, power distribution, and internal roads.

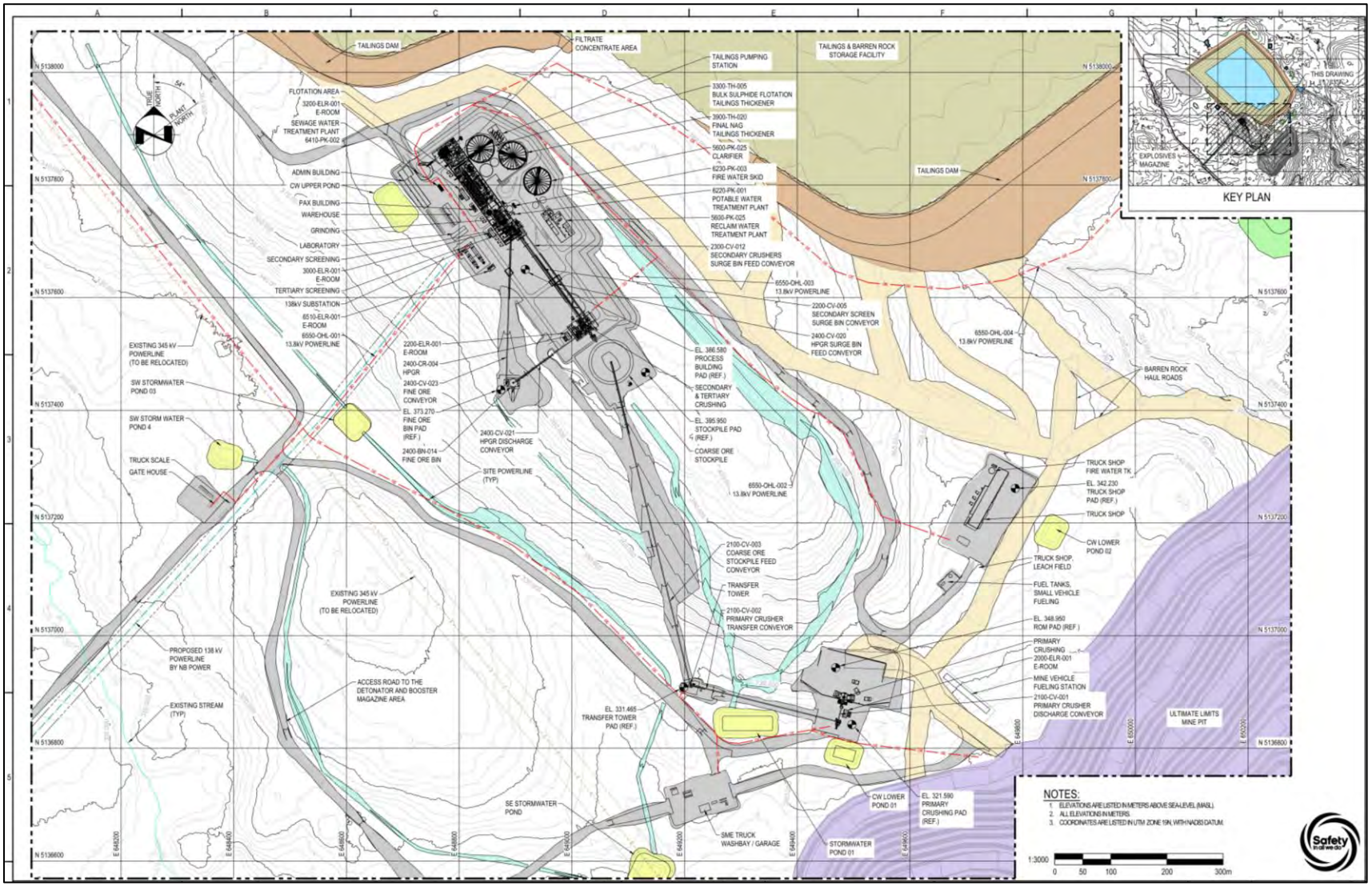
The process plant site layout, Figure 18-2, shows details of the proposed locations of the process plant, truck shop, and ancillary buildings, collectively referred to as mine site infrastructure.

Figure 18-1: Sisson Project site layout



Source: Ausenco, 2026

Figure 18-2: Sisson Project process plant layout



Source: Ausenco, 2026

18.2 Roads and logistics

18.2.1 Site access

The Project will be accessed using a combination of existing public and forestry roads and Project-specific access roads. The proposed road network includes approximately 39.2 km Primary Site Access Road (PSA), approximately 12.9 km Secondary Site Access Road (SSA), approximately 11 km Re-Aligned Fire Road, and approximately 3 km Site Access Road.

Existing road corridors will be used to the greatest practical extent to reduce new disturbance, earthworks, environmental effects, and capital cost. Upgrades will generally be concentrated at locations where the existing road geometry, pavement structure, drainage, visibility, or structural condition does not satisfy the anticipated construction and operational traffic requirements.

The proposed road upgrades may include:

- localized widening and horizontal or vertical alignment improvements;
- regrading and resurfacing;
- replacement or extension of existing culverts;
- rehabilitation of roadside ditches and drainage outlets;
- improvement of soft, saturated, or frost-susceptible subgrades;
- localized slope stabilization and erosion protection;
- provision of turnouts and passing areas; and
- installation or replacement of signage, delineators, and safety barriers.

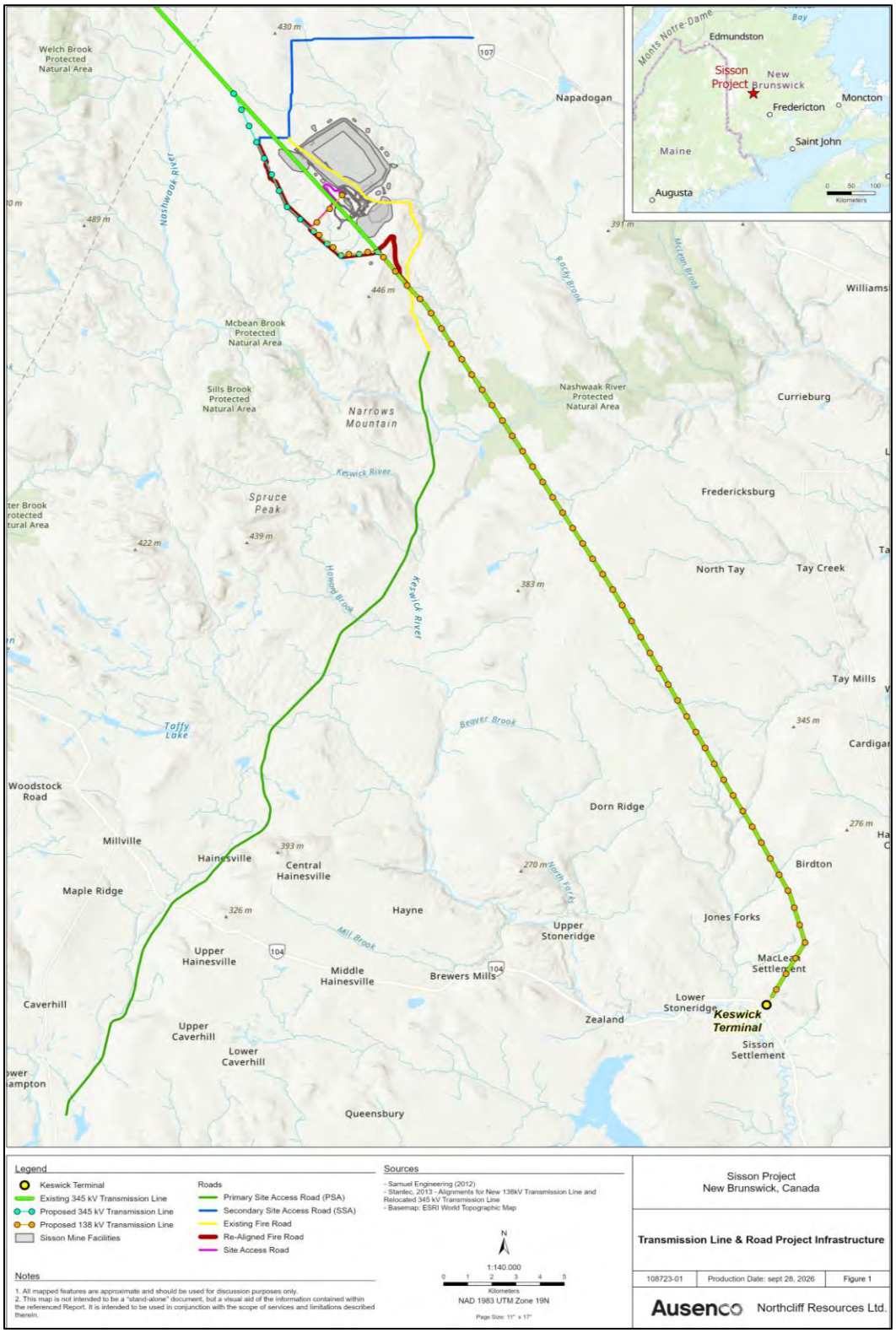
Existing bridges along the access routes were previously established for single-lane forestry traffic, including logging vehicles with a gross vehicle weight of up to approximately 62,500 kg. The condition and load capacity of these structures will be confirmed through structural assessment before they are relied upon for Project construction and operational traffic. Traffic-control measures will be implemented where continued public, forestry, or project access must be maintained during construction.

A summary of the access roads is provided in Table 18-1 and Figure 18-3 presents the general arrangement of the site access roads.

Table 18-1: Summary of the access roads

Road	Approximate Length	Status	FS Development Approach
Primary Site Access Road (PSA)	39.2 km	Road Upgrade	Existing road to be retained and upgraded through localized resurfacing, drainage improvements, widening, and geometric corrections
Secondary Site Access Road (SSA)	12.9 km	Road Upgrade	Existing and proposed route sections to be upgraded or reconstructed as required
Re-Aligned Fire Road	11 km	New Road Construction	New road with a minimum 7 m running-surface width
Site Access Road	3 km	New Road Construction	New Project access connecting the external road network to the main site infrastructure

Figure 18-3: Sisson Project - transmission line & road project infrastructure



Source: Ausenco, 2026

18.2.2 On-site roads

On-site roads will provide safe and reliable access between the main site entrance, Process Plant, ROM and Crusher facilities, Truck Shop, SME Truck Wash Bay/Garage area, water-management facilities, utility areas, and other Project infrastructure. The road network will support construction traffic, routine operations, maintenance activities, emergency response, and access to water-management and utility infrastructure.

On-site roads will have a minimum running-surface width of 8 m. Additional widening will be provided at intersections, horizontal curves, facility entrances, loading areas, and other locations where vehicle swept paths or operational requirements indicate that additional width is necessary. Two-way roads will be provided where regular opposing traffic is anticipated. Road classifications, operating speeds, grades, horizontal geometry, and intersection layouts will be selected based on the controlling design vehicles, anticipated traffic volumes, safety requirements, and site constraints.

Road alignments have been developed to balance operational safety, earthworks quantities, drainage requirements, environmental disturbance, and constructability. Steep grades, sharp curves, deep cuts, high fills, wetlands, and areas of potentially unsuitable foundation soils will be avoided where practical. Where these conditions cannot be avoided, localized widening, subgrade improvement, slope stabilization, retaining structures, safety berms, or other appropriate measures will be incorporated.

Road surfaces will be crowned or cross-sloped to promote positive drainage toward roadside ditches and prevent prolonged ponding. Culverts and cross-drainage structures will be provided where roads intersect natural drainage pathways. The road layout will also consider snow clearing and snow-storage requirements so that winter maintenance activities do not reduce the required operating width, obstruct sightlines, or block drainage features.

Final road profiles and pavement structures will be refined based on detailed topographic survey, geotechnical investigation, drainage analysis, traffic requirements, and confirmation of the final facility layout. Figure 18-1 presents the on-site access road plan.

18.2.2.1 Road Construction

Road construction will generally include clearing and grubbing, topsoil stripping, excavation, subgrade preparation, proof-rolling, placement of embankment fill, and construction of granular subbase, base, and wearing-course materials. Geotextile or geogrid reinforcement may be used where weak, wet, or variable subgrade conditions are encountered.

Granular layer thicknesses have been established based on the anticipated traffic loading, design life, subgrade strength, frost susceptibility, drainage conditions, and locally available construction materials. Potential borrow sources and excavated materials will be used as embankment fill and road-building aggregate to reduce imported-material requirements.

Road construction will also include roadside ditches, cross-drainage culverts, erosion protection, safety berms, signage, and delineators. Road surfaces will be graded to drain toward the roadside ditches and prevent prolonged ponding. Culvert inlets and outlets will be protected against erosion using riprap, geotextile, or other suitable measures.

18.2.3 Road drainage and maintenance

Roadside drainage will be integrated with the site-wide stormwater-management system. Ditches will collect runoff from the road surface and adjacent cut or fill slopes and convey it to culverts, diversion channels, stormwater ponds, or stable discharge locations. Cross-drainage culverts will be provided at

natural drainage crossings and where required to prevent water from being impounded against road embankments.

Maintenance activities will include grading, resurfacing, dust control, ditch cleaning, culvert inspection, debris and sediment removal, snow removal, and repair of erosion or frost-related damage. Access will be maintained to culverts, ponds, channels, and other water-management infrastructure for inspection and maintenance throughout Project construction and operation.

18.2.4 Concentrate transportation

Transportation planning considers average annual production of 10,000 t/a of Tungsten Concentrates, and 5,000 t/a of Molybdenum Concentrates. The targeted off taking destinations will be smelters in the US states of Ohio, Iowa and Pennsylvania. A subset of concentrate tonnages, up to 1,000 t/a of tungsten and 5,000 t/a of molybdenum, may be diverted to European facilities in Austria and Germany subject to market and smelter capacity considerations, see Table 18-2. All cargo will be pre-packaged into 2-t bulk bags at the mine site's concentrate load out facility awaiting highway haul truck load out.

Concentrate transportation is expected to be subjected to a seasonal winter shut down for six to eight weeks, where the main access road is not passable, followed by a 2-week spring thaw period with reduced road capacity. Concentrate will be stockpiled at mine and be dispatched once roads reopen.

Table 18-2: Production rates and destinations

Product	US Off-takers		European Off-takers	
	US Destinations (average t/a)	Transport Cost Estimate ($/t)	European Destinations (average t/a)[1]	Transport Cost Estimate ($/t)
Tungsten Concentrate or APT	10,000	$201 - $264	Up to 1,000	$128 - $141
Molybdenum Concentrate	5,000	$264 - $358	Up to 5,000	$128 - $141

Note:
1. Presented as diversions from US quantities.

For North American (US) off-take destinations, concentrates will be trucked using conventional, 6-axle dry-van tractor-trailers from mine directly to their destinations. Each truck will be loaded with net 20 t (10 bags) of concentrates in view of the mine access road design, Canadian and US highway regulations, to a total gross combined vehicle weight (GCVW) of 80,000 lbs. Distance to these destinations range from 1,600 to 2,500 km. Consequently, transportation costs will range from C$201.55/t to C$264.25/t to Tungsten clients, and C$264 to C$358/t to the farther Molybdenum clients. These costs include bags and pallets as consumables. Altogether transport costs can be unitized to be in the range of C$0.13-0.15/ t-km. As this is an end-to-end, direct trucking service, there are no additional terminal, transloading or ocean freight costs.

For European off-take destinations, concentrates will be loaded into sea containers directly at the mine. Each container carrying net 22 t (11 bags) of concentrates will be hauled down to Port Saint John (PSJ), New Brunswick, operated by DPWorld for ocean freight to major European container hubs such as Rotterdam, Wilhelmshaven or Hamburg. The cost for transportation of concentrates to an above European port on a CIF basis for pick up by the off-taking party range from C$128/t to C$141/t, inclusive of bags and pallets as consumables. These costs also include container rental, drayage and terminal/port fees.

Construction materials, spares and consumables sourced internationally can be moved in through PSJ. A standard sea container load can be brought to site from PSJ for C$1,038/container.

All costs presented above include fuel surcharges applicable to April-July 2026 period, at approximately +50% of base rates.

18.3 Built infrastructure

As shown in Figure 18-1 and Figure 18-2, the site area consists of several buildings. Three types of buildings have been incorporated into the project design: modular, pre-engineered, and concrete buildings. The proposed buildings for each area are listed in Table 18-3.

Table 18-3: Description of on-site buildings

Area	Name of Building/Structure	Building Construction Type	Building Geometry			
			L (m)	W (m)	H (m)	Area (m^2)
Process	Process Plant (Secondary and tertiary screening, grinding area, flotation)	Pre-engineered	137.0	36.8	39.2	5041.6
Process	Process Plant (Concentrate area)	Pre-engineered	72.5	36.8	30.1	2668.0
Process	Reclaim Tunnel	Concrete Structure	36.0	6.9	6.7	246.6
Process	HPGR Building - Secondary Crushing	Pre-engineered	27.6	30	43.8	828
Process	HPGR Building - Tertiary Crushing	Pre-engineered	18.8	30	32.7	564
Process	Tailings pump station	Pre-engineered	34.6	17.6	14.0	609.0
Process	Primary Crushing	Concrete Structure	11.2	10.2	21.1	114.2
Process	Water Treatment Plant	Pre-engineered	62.0	29.6	14.0	1835.2
Process	Concentrate Storage	Pre-engineered	10.0	21.0	6.5	210.0
Process	Concentrator Warehouse & Maintenance Shop	Pre-engineered	40.0	12.0	3.8	480
Process	Assay Laboratory	Modular	30	12	2.7	360
Ancillary	Administration building including Concentrator office	Modular	40.0	14.0	8.0	560.0
Ancillary	Offices	Modular	19.6	14.0	4.0	274.4
Ancillary	Change house	Modular	20.0	19.6	4.0	392.0
Ancillary	Gatehouse	Modular	12	2.4	2.4	28.8
Ancillary	Potable Water Treatment Plant (PWTP)	Modular	6	2.4	2.4	14.4
Ancillary	Sewage Water Treatment Plant (SWTP)	Modular	12	4.8	2.4	57.6
Mining	Truck shop (Warehouse Area)	Pre-engineered	30.3	19.0	6.0	575.7

Area	Name of Building/Structure	Building Construction Type	Building Geometry			
			L (m)	W (m)	H (m)	Area (m^2)
Mining	Truckshop	Pre-engineered	84.3	19.0	14.8	1601.7
Mining	Detonator & Booster Magazine	Modular	TBC	TBC	TBC	TBC
Mining	SME Truck Wash Bay/Garage	Modular	TBC	TBC	TBC	TBC
Electrical	E-room (Grinding)	Modular	15.0	6.0	4.0	90
Electrical	E-room (Flotation)	Modular	21.0	6.0	4.0	126
Electrical	E-room (HPGR)	Modular	11.0	6.0	4.0	66
Electrical	E-room (Primary Crushing)	Modular	12.0	6.0	4.0	72
Electrical	Control room (Substation)	Modular	13.0	8.0	4.0	104

Note: Height measured at eave.

18.3.1 Site Facilities

Site facilities include a gatehouse, administration building, laboratory building, mine truck shop, and various support buildings.

The Process Plant has been described in Section 17.

18.3.1.1 Gatehouse and security

The site will be fenced and there will be a gatehouse to deter access by unauthorized people.

The gatehouse is a modular office with a lockable gate and communications to the main site. The truck scale is located adjacent to the site access road by the gatehouse.

18.3.1.2 Main administration building

The administration building will be a steel-framed, prefabricated, slab-on-grade building. There will be both a two-storey and single story segments within the building.

The administration building will house space for site management, administration, mine management, engineering offices, conference rooms, and washrooms. Dry change, and medical and safety offices will also be located in this facility. The building will be located west of the process plant.

18.3.1.3 Laboratory building

The laboratory building will be a single-story, steel-frame, prefabricated, slab-on-grade building. This building will house an analytical lab, metallurgical lab, sample preparation area, small office area, break room, and a washroom. The building will be located west of the process plant, adjacent to the warehouse building.

18.3.1.4 Mine truck shop and support buildings

The mine truck shop will be a pre-engineered building with an overhead crane, HVAC system, and fire protection. The design includes three truck maintenance bays, one wash bay, a small welding bay and an external tire change area. An area fire/raw water tank, pumphouse, and potable water tank are located outside. Offices/dry facilities and a mining warehouse are located adjacent to the truck shop.

Mining support buildings include a Detonator & Booster Magazine and a Site Mixed Explosive (SME) Truck Washbay / Garage which will be supplied by the blasting contractor.

18.3.2 Accommodation

No camp accommodation is planned for the construction or operations phase of the project as workers will reside in, and commute from, nearby communities.

18.4 Stockpiles

Stockpiles are previously discussed in detail in Section 16.6.2

18.5 Waste storage facilities

Surplus barren rock will be stored within the Barren Rock Storage Area (BRSA) situated within and towards the south of the TSF and will be progressively encapsulated by tailings and water, or, later in the mine life, stored as pit backfill. This is previously discussed in detail in Sections 16.6.3 and 16.6.4, respectively.

18.6 Water supply and management

18.6.1 General

The water management objectives are to divert non-contact surface water back to natural drainages using diversion channels to the extent practicable, and to collect contact water to the extent practicable.

The tailings storage facility (TSF) comprises an approximately 8 km long ring-dam structure located adjacent to the process plant and open pit. The TSF is situated in the headwaters Bird Brook and other small tributaries to West Branch Napadogan Brook, and few diversion structures are required. The TSF serves as the main water management facility for the Project. Contact water is managed at the TSF and will be used to meet Project water demands, driven primarily by process plant requirements. Contact water that exceeds the Project water demand will be treated, as necessary, to meet water quality objectives and discharged.

The southern lobe of the open pit and some of the processing infrastructure are within the headwaters of the McBean Brook watershed at its boundary with the Napadogan Brook watershed. The northern part of the open pit is in the headwaters of Sisson Brook, which is a tributary to West Branch Napadogan Brook.

18.6.2 Surface water management design basis

The site-wide stormwater-management system is designed to maintain natural drainage patterns where practical, reduce flood risk to project infrastructure, control erosion and sediment transport, and separate contact water from non-contact water.

The design is based on the existing catchments and topography, the proposed grading plan, Project development footprints, soil and bedrock conditions, and the adopted climatic and hydrological criteria. Available regional meteorological records and Project-specific information will be used to establish precipitation, snowmelt, and design-storm inputs. The adopted design rainfall depths is summarized in Table 18-4.

Table 18-4: Estimated rainfall depth (mm) information for Sisson site

Duration	Return Period (Years)							
	2	5	10	25	50	100	200	500
5 min	3.7	4.8	5.7	7.0	8.1	9.5	11.0	13.4
10 min	5.3	6.8	8.1	9.9	11.5	13.4	15.6	18.9
15 min	6.4	8.4	9.9	12.1	14.1	16.4	19.1	23.2
30 min	9.1	11.8	14.0	17.2	19.9	23.2	27.0	32.8
1 h	12.9	16.7	19.8	24.3	28.2	32.9	38.2	46.3
2 h	18.2	23.7	28.0	34.4	39.8	46.5	54.0	65.5
6 h	31.5	41.0	48.5	59.5	69.0	80.5	93.5	113.5
12 h	44.5	58.0	68.6	84.1	97.6	113.8	132.2	160.5
24 h	63.0	82.0	97.0	119.0	138.0	161.0	187.0	227.0

Given the Project's location in New Brunswick, the drainage design will consider snow accumulation, spring snowmelt (freshet), freeze-thaw conditions, potential icing, and rain-on-snow events. The applicable hydrologic design criteria will be established in accordance with regulatory requirements and the Project design criteria.

In addition to rainfall-based design events, freshet and combined rainfall-snowmelt events will be considered in the overall hydrologic assessment. Freshet conditions can generate sustained high flows over extended periods due to snowmelt and coincident rainfall. Although these conditions may not govern the instantaneous peak discharges used to size smaller hydraulic structures, they may affect storage requirements, conveyance capacity, and site-wide water-management planning. Rainfall-runoff modelling and regional flood-frequency analysis will therefore be used to confirm that the proposed drainage and storage systems provide adequate performance under both event-based and seasonal conditions.

Table 18-5 presents the average monthly temperature and estimated snowmelt data for the Sisson site.

Table 18-5: Average monthly temperature and snowpack

Parameter	Month											
	Jan	Feb	Mar	Apr	May	Jun	Jul	Aug	Sep	Oct	Nov	Dec
Ave. Temp.	-11.3	-10.1	-4.1	2.9	9.5	14.5	17.3	16.2	11.9	6.0	-0.2	-7.6
Ave. Snow Water Equivalent (mm)	110	162	194	115	3	0	0	0	0	0	8	49

18.6.3 Surface water management strategy

Surface water management will be developed separately for the plant-site and mine-site areas while maintaining an integrated site-wide drainage strategy. The primary objectives are to:

- Intercept clean runoff before it enters disturbed areas;
- Convey non-contact runoff around Project infrastructure;

- Collect runoff from disturbed and potentially contaminating areas;
- Attenuate peak discharges and provide sediment removal;
- Protect roads, pads, slopes, and embankments against flooding and erosion;
- Maintain safe and controlled drainage pathways during extreme events; and
- Provide access for inspection, sediment removal, and maintenance.

Runoff will be classified according to its source and potential water quality. The preliminary management approach is summarized in Table 18-6.

Table 18-6: Run-off management approach

Runoff Source	Preliminary Classification	Management Approach
Undisturbed upstream catchments	Non-contact water	Intercepted and routed around Project infrastructure using diversion ditches, channels, and culverts
External and site access roads	Non-contact or sediment-bearing runoff	Managed using roadside ditches, cross-culverts, erosion protection, and sediment controls
Process Plant and ore-handling areas	Contact water	Collected and directed to contact-water ponds or other designated water-management facilities
Crusher, ore stockpile, and SME areas	Contact water	Collected by grading, ditches, sumps, and ponds for controlled management
Temporarily disturbed construction areas	Sediment-bearing runoff	Managed through temporary erosion- and sediment-control measures
Undisturbed downstream areas	Non-contact receiving drainage	Protected through controlled outlets and energy-dissipation measures

Figure 18-1 and Figure 18-2 present the general arrangement of the overall run-off management system.

18.6.3.1 Non-contact water management

Non-contact runoff from undisturbed catchments will be intercepted before entering the Project disturbance areas. Perimeter ditches, diversion channels, and culverts will be used to convey this runoff around roads, pads, ore-handling areas, and other Project facilities. The collected runoff will be directed to stormwater ponds for temporary storage and attenuation before being released to the environment through controlled outlet structures.

The stormwater ponds will attenuate peak flows, promote the settlement of suspended sediment, and reduce the potential for erosion at downstream discharge locations. Pond outlets will be configured to limit discharge rates and velocities to acceptable levels. Emergency overflow pathways will also be provided to safely convey flows exceeding the pond's normal operating or design capacity.

Diversion channels will generally follow natural drainage patterns where practical. Channel slopes and cross-sections will be selected to provide adequate hydraulic capacity while limiting flow velocities and erosion potential. Riprap, geotextile, or equivalent armouring will be provided where permissible velocities may be exceeded or erodible soils are present.

Pond and channel outlets will be configured to prevent concentrated discharges from causing erosion, slope instability, or adverse effects on downstream drainage features. Energy-dissipation and erosion-protection measures will be incorporated where significant elevation changes or high discharge velocities are anticipated.

18.6.3.2 Contact-water management

Runoff from the process plant, crusher, ore stockpiles, ore-handling areas, SME facilities, and other areas where stormwater may contact potentially contaminating materials will be treated as contact water unless demonstrated otherwise through water-quality assessment.

Contact water will be collected using facility grading, perimeter berms, ditches, sumps, and stormwater or contact-water ponds. Gravity conveyance will be used where practical. Pumping will be provided where topography does not permit gravity drainage or where water must be transferred to another storage, treatment, or process-water facility.

Contact-water-management infrastructure will be arranged to minimize the potential for uncontrolled releases and mixing with non-contact runoff. The need for pond or channel liners will be determined based on anticipated water quality, foundation permeability, hydrogeological conditions, and applicable regulatory requirements.

18.6.3.3 Stormwater and contact water ponds

Stormwater ponds will be provided to attenuate peak runoff, promote sediment settlement, and control discharges from developed catchments. Contact-water ponds will collect and provide operational storage for runoff from designated contact-water areas before the water is reused, treated, transferred to another storage facility, or discharged in accordance with applicable water-quality requirements.

Contact-water ponds will be lined with a high-density polyethylene (HDPE) geomembrane to minimize seepage and protect the underlying soils and groundwater. The liner system will be placed over a smooth, prepared subgrade that is free of sharp objects and other materials that could damage the geomembrane. Geotextile cushioning or other protective layers will be provided where required. The liner will be secured within perimeter anchor trenches, and pipe penetrations and other connections will be appropriately sealed. Liner installation, seam welding, and testing will be completed under an approved construction quality assurance and quality control program.

The stormwater and contact-water ponds will be sized to store runoff generated by a 1-in-100-year, 24-hour storm event. The required pond storage will include allowances for operational water, design-storm runoff, and sediment accumulation. Appropriate freeboard will be provided above the maximum design water level. Pond embankments, foundations, slopes, spillways, and outlet structures will be designed based on the applicable geotechnical, hydrological, environmental, and consequence-classification requirements.

Where controlled discharge is permitted, outlet structures will be configured to limit downstream peak flows and prevent erosion. Where direct discharge is not permitted, collected water will be pumped to an appropriate process-water, treatment, or storage facility. Safe overflow or exceedance pathways will be established for stormwater ponds. Exceedance management for contact-water ponds will be configured to retain excess water or direct it to the TSF, where practicable, thereby minimizing the potential for an uncontrolled release to the environment.

A summary of the ponds, including their contributing catchments, functions, storage volumes, design criteria, liner requirements, and outlet arrangements, is provided in Table 18-7.

Table 18-7: Contact water (CW) and stormwater ponds information

Pond	Pond Vol. (m^3)	Water Depth (m)	Freeboard (m)	Water-Surface Area (m^2)
CW Upper Pond	4,150	2.4	0.6	2,422
CW Lower Pond 1	2,660	2.4	0.6	1,631
CW Lower Pond 2	2,600	2.4	0.6	1,596
Stormwater Pond 1	5,900	2.4	0.6	3,351
Stormwater Pond 2	12,140	3.4	0.6	4,731
Stormwater Pond 3	3,625	2.9	0.6	2,117
Stormwater Pond 4	1,370	2.9	0.6	1,383

18.6.3.4 Ditches, channels, and culverts

Stormwater ditches and channels will be provided along access roads, around facility pads, and between developed drainage catchments. Culverts will be installed at road crossings and natural drainage pathways to maintain cross-drainage and prevent water from being impounded against road embankments.

Ditches, channels, and culverts will be sized for the applicable design events established in the Project design criteria. Hydraulic design will consider peak discharge, flow depth, velocity, inlet and outlet control, debris blockage, sediment transport, icing, and potential effects on upstream and downstream infrastructure.

Culvert inlets and outlets will be stabilized using riprap aprons, geotextile, headwalls, or other erosion-protection measures where required. Minimum culvert diameters will consider both hydraulic requirements and practical maintenance needs. Culvert locations and elevations will be coordinated with the road profiles and natural drainage system.

18.6.3.5 Erosion and sediment control

Temporary erosion and sediment control measures will be implemented during clearing, earthworks, and construction. These measures may include diversion berms, temporary ditches, sediment traps, check dams, silt fencing, stabilized construction entrances, erosion-control blankets, and progressive revegetation.

Permanent erosion protection will be provided at channel bends, steep sections, culvert outlets, spillways, and other locations subject to concentrated flow. Disturbed areas will be stabilized and revegetated as soon as practical following completion of grading.

The erosion and sediment control system will be inspected during construction and after significant precipitation or snowmelt events. Accumulated sediment will be removed when it reduces the required capacity or affects the performance of drainage infrastructure.

18.6.4 TSF construction water management

Construction water management at the TSF will commence during site establishment approximately three years prior to mill start-up. Water management during this phase will consist of establishing collection ponds, coffer dams, pumping systems, runoff collection ditches, and diversion channels. Some of the

temporary works such as cofferdams and bypass diversion channels will be removed once the initial starter embankments have been constructed and commissioned for water collection.

18.6.5 TSF operational water management

The operational water management plan for the Project includes:

- The TSF operating pond consists of supernatant water from the non-diverted runoff, direct precipitation, and tailings supernatant water. The pond will be maintained through the reclaim and water treatment plant feed systems such that sufficient capacity exists for operational flexibility and storm inflow storage.
- The TSF operating pond will be maintained remote from the embankments using rotational discharge from the TSF embankment crest. Staged operational spillways will be installed to maintain the maximum flood elevation at least 100 m from the embankments.
- Tailings will be selectively deposited from the crest of the embankments to develop tailings beaches which function as an extensive low permeability zone to mitigate seepage through the embankments.
- Water management ponds at low points around the TSF perimeter will collect seepage and runoff from the TSF embankments.
- Groundwater monitoring wells will be located downstream of the water management ponds. Water interception wells will be installed and operated if excessive seepage is detected.

18.6.6 Open pit water management

Open pit development will have a significant impact on the local hydrogeological regime, as the open pit will become a groundwater discharge area. The existing groundwater table is at or near surface, and progressive development of the open pit will result in a gradual lowering of the groundwater table in the vicinity of the excavation.

An average combined groundwater and direct precipitation inflow of about 160 L/s was estimated for the life-of-mine pit design based on the long-term average annual precipitation for the site. Approximately 363,000 m^3 of water will flow into the pit during the 1 in 100-year 24-hour design storm event.

The pit water management systems will include:

- Diversion ditches to collect surface runoff, snowmelt, and seepage along the pit crest
- Pumps and pipeline systems that can remove the direct precipitation and groundwater infiltration from the open pit during mining to maintain a trafficable pit floor for mine operations. The pumps and pipelines will operate intermittently at a constant flowrate of 720 m^3/h to progressively remove the inflow volume resulting from the 1 in 10-year design flood within approximately ten days and the 1 in 100-year flood within about three weeks.
- A water management pond at the north of the open pit, to allow for sediment settling before the water from the open pit being used for mill process water or released to environment.
- Contingency pit slope depressurization measures that may include perimeter pumping wells and/or in-pit pump wells to reduce the pore water pressures in the pit walls

18.6.7 Water balance

A mine site water balance model was developed in GoldSim® to evaluate the performance of the proposed water management facilities under a range of climatic conditions. The model applies a 74-year synthetic daily climate time series for precipitation, temperature, and potential evapotranspiration, which preserves observed climate variability, including prolonged wet and dry periods, and generates 74 realizations that are cycled over the modelled period. Long-term daily temperature and precipitation series were developed for the Project using by developing correlations between concurrent data for the site (Sisson-Brook) and the Aroostook climate station, which were then applied to the long-term non-continuous precipitation and temperature records to generate corresponding synthetic long-term non-continuous climate series for the site. The non-continuous synthetic datasets were subsequently infilled using downscaled ERA5 Land reanalysis data and data from other regional stations to develop fully continuous datasets that were used as inputs for water balance modelling.

These simulations produce a range of potential flows and storage volumes for the Project facilities and were used to develop the operating criteria for the water management facilities and to estimate inflows to the water treatment plant (WTP).

The water balance was simulated from pre-production through post-closure using GoldSim® on a daily time step, with results summarized on a monthly or annual basis. The model timeframe includes:

- Pre-production: before commercial production
- Operations: during operation of the process plant
- Closure: after commercial production ceases and before the open pit fills with water)
- Post-closure: after the open pit fills

The base case monthly operational water balance model was developed using average estimated values for runoff and precipitation. The mean annual precipitation for the Sisson Project area was estimated to be approximately 1192 mm with 69% of the annual precipitation falling as rain and 31% falling as snow. The annual average potential evapotranspiration for the Sisson Project site was estimated to be 525 mm using the Thornthwaite approach.

The TSF Pond is predicted to be in a surplus condition for the entire operating life of the mine, indicating that the system (including the TSF and contributing catchments) can supply sufficient water to meet the process plant water requirements, even under dry conditions. The magnitude of the water surplus increases from approximately 400 m^3/h in the early years of operations to a maximum of approximately 1250 m^3/h as the project footprint increases.

18.6.8 Water supply

Process water will be primarily sourced from thickener overflow streams and reclaim water from the TSF. The process plant has assumed all freshwater make-up will be sourced from the WTP discharge at 53 m^3/h (15 L/s).

Potable water is to be supplied by wells. The site potable water tank has a storage volume of approximately 39 m^3. A potable water treatment system is incorporated in the project infrastructure. The potable water requirement has been estimated at 17 m^3/d (0.2 L/s) for the project based on 120 people onsite per day with no camp. Based on previous well tests, one well is assumed to be adequate to service the flow rate, a second may be required as a backup or water could be trucked in, when necessary.

18.6.9 Water treatment

The proposed Water Treatment Plant (WTP) and associated infrastructure will be located on the north side of the process plant, close to the tailings pond. The water from the tailings storage facility will be conveyed to the WTP via a designated pipeline for treatment. The WTP design is based on water quality modelling results (KP, 2026) and resulting design criteria as developed by SRK (SRK, 2026). Design criteria include influent flowrate, influent water chemistry, and effluent water quality objectives. The maximum monthly influent flowrate during operations is 2083 m^3/h. The WTP will target total suspended solids (TSS), bicarbonate, cyanide, thiosalts, and dissolved metals. The WTP is designed to treat these constituents to meet applicable Schedule 4 discharge limits under the Canadian Metal and Diamond Mine Effluent Regulations (MDMER) and acute lethality requirements.

The WTP will support mining efforts during the operations and post-closure active care phases of the Project. Treatment will be based on a High-Density Sludge (HDS) process that supports treatment for metals, bicarbonates and solids. Oxidation of thiosalts will be conducted in a dedicated reactor via peroxide addition. After metal precipitation, pH neutralization using sulphuric acid will be required prior to discharge. Treated water will be discharged to Sisson Brook with a small portion recycled for use in the process plant. Waste sludge will be collected from the bottom of the clarifier. This sludge waste stream will be pumped and fed to the tailings thickener to be co-deposited with tailings.

HDS is a proven method for metals removal from mine-impacted water. Recirculation of generated sludge in the HDS process allows for a denser sludge, which enhances the removal of contaminants. Hydroxide and iron co-precipitation is achieved via dosing reagents such as lime and ferric sulphate. Operation at elevated pH also promotes calcite precipitation to remove excess bicarbonate. Ferric addition is included to support oxidation reactions or for TSS removal when bicarbonate concentrations are low. Precipitated solids are flocculated using a standard anionic polymer and settled as sludge in a conventional clarifier.

In addition to ferric sulphate and lime, hydrogen peroxide is also added to the lime reactor tank to oxidize thiosalts. Thiosalts are expected to be oxidized to sulfate, supported by dissolved iron present in the system. The effectiveness of hydrogen peroxide oxidation is dependent on the overall water chemistry.

The final neutralization stage is included to decrease the treated water pH using sulphuric acid to meet effluent requirements prior to discharge to the environment, to prevent calcite scaling downstream for recycle water, and reduce the fraction of ammonia present as un-ionized ammonia through conversion to ammonium. The chemical reagents used in the water treatment process and their on-site storage are summarized in Table 18-8.

The WTP will be housed within a centralized WTP building with the majority of equipment located inside. Large tanks and clarifiers will be located outside and designed for the climate conditions.

Table 18-8: Sisson WTP – Reagents storage volume

Reagent	On Site Storage
Quick lime silo	342 t
Hydrogen peroxide	43 m^3
Ferric sulphate	54 m^3
Granular flocculant	Minimum three, 750-kg bags
Flocculant dosing tank	17 m^3
Sulfuric acid	35 m^3

During the next phase of the project, potential improvements may include optimization of equipment selection, reagent consumption, and process operating conditions. More detailed process modelling and confirmation of the influent water chemistry will be required to assess treatment performance and refine chemical requirements. Additional water chemistry and flow data will also be required to confirm the WTP capacity required for the closure/post-closure period.

18.7 Tailings storage facilities

The TSF is designed for secure and permanent storage of approximately 268 Mt of tailings, 9.2 Mt of mid-grade ore, and 201 Mt of barren rock from the proposed mining operations over a 27-year mine life. Approximately 184 Mt of barren rock will be placed in the BRSA and 17 Mt will be placed in the upstream shell zone of the TSF embankment; the barren rock in both locations will be progressively encapsulated with tailings. General arrangements of the TSF are presented in Figure 16-33 through Figure 16-45.

18.7.1 TSF foundation conditions

The site geotechnical conditions are characterized by the following overburden and bedrock units:

- **Unconsolidated sediments –** The estimated overburden thickness varies between approximately 1.5 m and 17 m and is generally thicker in the valley bottoms and becomes thinner on the hill slopes. The overburden typically comprises of a thin veneer of top-soil and organic materials, underlain predominantly by lodgement and ablation glacial till materials. Glaciofluvial and alluvial materials are present along historical and current drainages. No evidence of glaciolacustrine deposits was identified in the site investigations to date, which include test pits and sonic boreholes.
- **Bedrock** – The majority of the bedrock in the TSF basin is composed of granites. The bedrock at the southeast corner of the project site consists of faulted and deformed metamorphosed rock including granitoids/diorites/gabbros, metavolcanics, and quartz and biotite rich metasedimentary units. In situ hydraulic permeability testing indicates that the rock mass has a hydraulic conductivity in the range of 10^{-8} m/s. Weathered bedrock and fractured zones occur across the project site with an average hydraulic conductivity in the range of 10^{-6} m/s.

Groundwater levels are generally shallow, occurring near the ground surface, except in hilltop areas, where deeper groundwater levels of up to 20 m below ground surface are observed. Groundwater flow is expected to be mostly dominated by recharge occurring on the relatively higher ground, and discharge focused in the adjacent topographic lows such as creek bottoms, wetlands and lowlands.

18.7.2 Seismicity

The design earthquakes and seismic parameters for the project were determined based on a review of regional seismicity and a site-specific seismic hazard assessment conducted for the Project (KP, 2012). A design earthquake magnitude of 7.0 has been adopted for all events with return periods of 500 years or greater, based on the regional tectonics, historical seismicity, and probabilistic seismic hazard deaggregation.

18.7.3 Tailings characteristics

Thickened tailings from the process plant will be discharged to the TSF at an average slurry solids content of 55% by weight, at a nominal throughput of 30,000 t/d, or approximately 10,500 kt/a.

Laboratory testing of representative tailings samples was completed during the 2013 Feasibility Study (KP, 2013). Bench-scale tests were carried out to estimate the physical properties of the tailings, including classification tests, slurry settling tests, air drying tests, and consolidations and permeability tests.

The average settled dry densities of the NAP and PAG tailings were estimated to be approximately 1.47 t/m^3 and 1.0 t/m^3, respectively (KP, 2013). It is anticipated that tailings will consolidate to their average settled dry densities throughout life of mine. Actual field densities will depend on the rate of rise, grind size and reagent addition, thickener performance, degree of consolidation, overall depth of deposited tailings, and tailings segregation during deposition into the impoundment.

18.7.4 Hazard classification

A preliminary dam classification was conducted based on the criteria provided by the Canadian Dam Associations (CDA) Dam Safety Guidelines (2013, 2019) to define the design earthquake and flood events for the TSF.

The classification of a TSF is typically based on the potential incremental consequences of a hypothetical dam failure. Incremental consequences are defined as the total damage from an extreme event (e.g., probable maximum flood) with dam failure minus the damage that would have resulted from the same event had the dam not failed. The incremental losses consider loss of life, environmental and cultural values, and infrastructure and economic impacts.

The Sisson Project is situated in a relatively remote area with no major development, infrastructure, and public access only via logging roads, indicating that there is low risk of third-party economic loss. The potential for loss of life is likely minor following a dam failure but cannot be discounted, particularly during operation of the project. Archaeological studies completed in the project area identified some potential sites that could fall under Heritage Concern. Catastrophic release of tailings and/or process water is deemed to potentially have a substantial impact on downstream watercourses and habitat and that restoration or compensation in kind would not be practicable.

Based on these considerations, the TSF embankments are currently assigned a hazard classification of 'Extreme' during operations and closure due to the potential environmental consequences.

18.7.5 TSF alternatives assessment

A tailings management alternatives assessment (KP, 2015) was completed in 2017 as part of the Schedule 2 Amendment process to authorize disposal of mine waste into fish-bearing waters, in accordance with the Metal and Diamond Mining Effluent Regulations.

18.7.5.1 Site location alternatives

Four potential TSF sites in the Sisson Project area were identified during the scoping stage. The current site was selected as the preferred location on the basis of environmental, logistical and economic factors. This site is located closest to the open pit and results in a minimized overall project footprint. Three TSF configurations were further assessed at the preferred site, from which the current project configuration was chosen.

18.7.5.2 Tailings technology alternatives

A trade-off study (KP, 2015) was completed to evaluate the following tailings technologies:

- Slurry tailings,

- Paste tailings, and
- Filtered dry stack tailings.

The outcome of the alternatives assessment verified that slurry tailings is the preferred technology due to the site climate and the benefits of using slurry tailings to manage the ML/ARD potential of the barren rock.

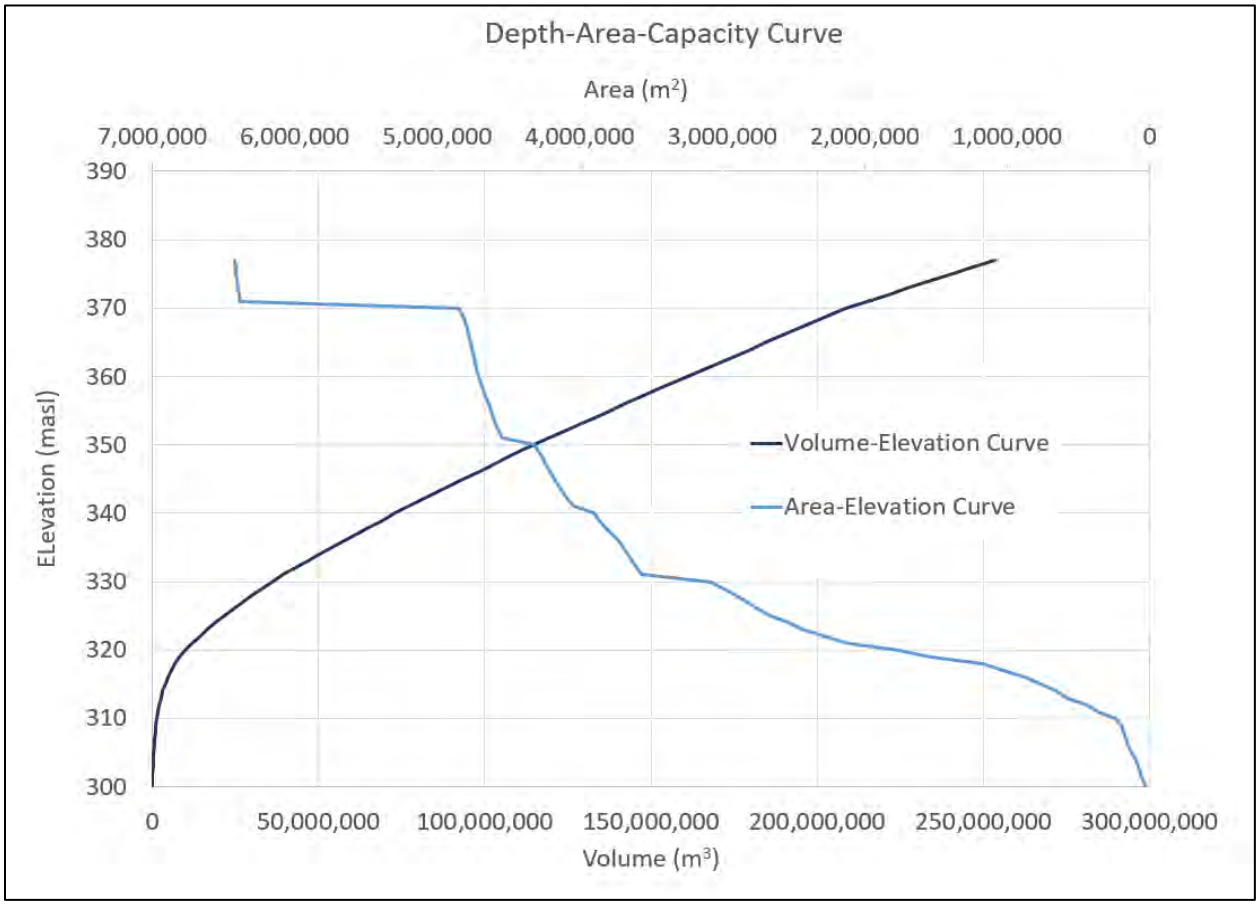
The TSF concept presented herein is consistent with the conclusions of the alternatives assessment.

18.7.6 TSF filling schedule

The TSF will be constructed to store tailings and barren rock produced over the mine life. The TSF will also manage the site water inventory and store contact water prior to treatment and discharge.

The TSF embankments will be constructed in stages as the elevation of the stored tailings and ponded water increases over the mine life as shown on Figure 16-33 through Figure 16-45.

Figure 18-4: TSF depth-area-capacity relationship



Source: Knight Piésold, 2026d

Tailings will be spigotted from the embankment crest around the TSF. The coarse fraction of the tailings will settle more rapidly than the fines and will accumulate closer to the discharge points forming a gentle beach with a slope of about one percent. Finer tailings particles will travel further and settle at a flatter slope adjacent to and beneath the supernatant pond.

Barren rock and mid-grade ore generated from open pit mining will be placed in the TSF. A portion of the barren rock will be used as Zone C2 material in the construction of the upstream shell zone of the dam. The remainder of barren rock will be placed in the TSF by the mine trucks, and the active lift will remain above the TSF supernatant pond to provide a working platform. The barren rock will be encapsulated by deposited tailings solids to the extent practicable.

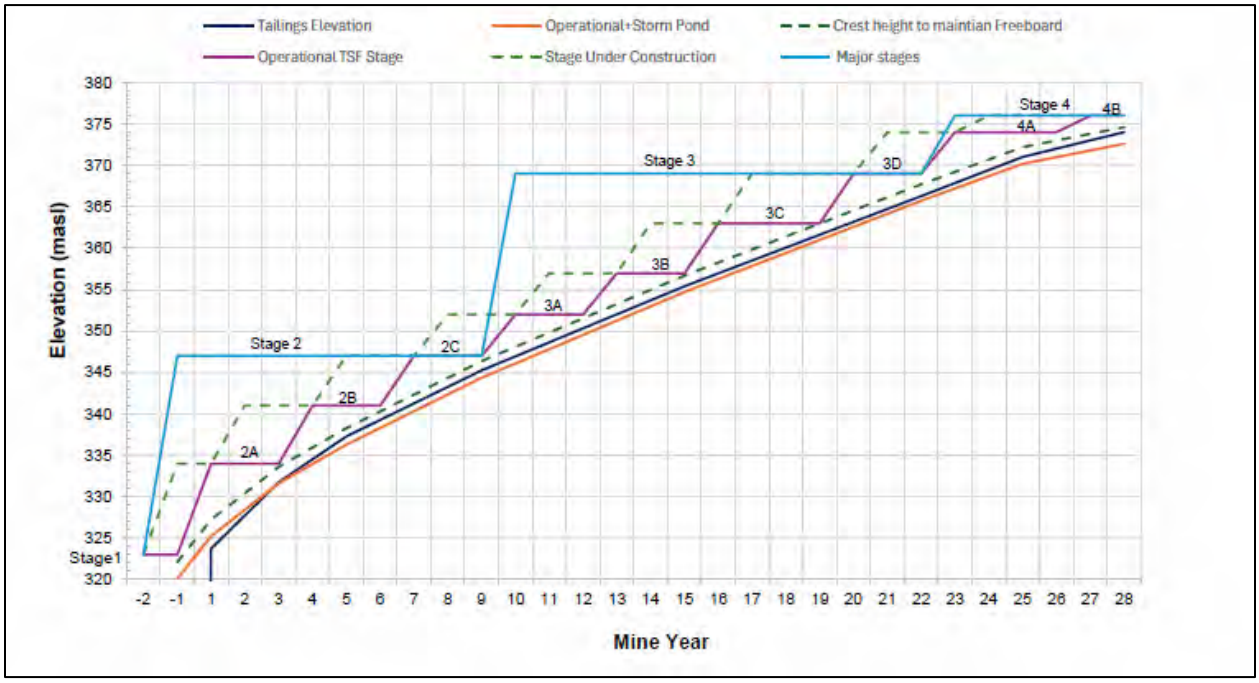
Embankment construction materials including Zone C1 rockfill, Zone T and Zone F will be sourced from a quarry located at the northwest of the TSF. The TSF will progressively flood portions of the quarry as it is mined out.

18.7.7 Embankment staging & design

18.7.7.1 Embankment staging

The embankments will be constructed in stages as zoned earthfill and rockfill structures. Stage 1 includes the initial starter embankment that will be constructed prior to mill start-up; Stages 2 through 4 will be constructed throughout the mine life to meet tailings and water storage requirements. The final embankment will have an elevation of 376 masl.

Figure 18-5: TSF filling schedule



Source: Knight Piésold, 2026d

18.7.7.2 Foundation preparation

The in situ glacial tills are generally suitable for the embankment foundation and will not be removed unless necessary in localized areas. Loose or unsuitable foundation materials including soft, saturated, and other deleterious material will be removed, re-graded, or replaced with approved general fill material. Rock outcrops and rock protrusions that affect embankment construction will be removed where needed. Once the unsuitable materials have been removed, the foundation surface will be shaped with local cut and fill to

facilitate placement and compaction of the embankment materials. Irregularities, ruts, scouring, and washouts will be removed to create a uniform foundation surface. Soils at the contact between the foundation and the fill will be compacted to a density compatible with overlying fill material. Where bedrock is exposed, the surface will be cleaned of loose or weathered material prior to fill placement.

Estimated excavation depths generally range between approximately 1 m and 5 m and include removal of topsoil and locally unsuitable materials only; competent overburden is expected to remain in place. These estimates are preliminary and further investigation is required before final design. The excavation depths represent anticipated removal of unsuitable materials to expose competent foundation conditions and may be refined during construction based on observed subsurface conditions.

Pressure grouting through a concrete plinth installed along the upstream toe of the starter embankments and the centreline of the subsequent raises can be implemented as a contingency measure in zones where measured bedrock permeability does not meet the threshold criterion.

18.7.7.3 Construction materials

Primary embankment material zones include:

- Zone R – Starter Embankment: Select overburden (till) meeting specified gradation requirements, sourced within the TSF basin footprint.
- Zone C1 – Downstream Rockfill Shell: Hard, durable rockfill from non-PAG quarried material.
- Zone C2 – Barren Waste Rock: Rockfill composed of barren waste rock from ongoing mining operations.
- Zone S – Seal (Core): Low-permeability fine-grained materials sourced from local till borrow areas.

Filter and drainage components are incorporated to manage seepage and mitigate internal erosion:

- Filter Zone (F): Processed sand placed on the upstream face of the starter dam as liner bedding, and downstream of the core to prevent migration of fines into the downstream shell.
- Drainage Zone (T): Gravel placed downstream of the filter zone within the embankment, providing drainage and effective seepage control while maintaining a compatible filter relationship with Zone F.

18.7.7.4 Starter embankments (Stage 1)

Three starter embankments will be constructed at the low points in the TSF impoundment using select overburden from local borrow sources near the embankment sites. The embankments will have a geosynthetic liner on the upstream face to contain the start-up water pond. The liner will be anchored into a trench keyed into lower permeability bedrock along the upstream toe of the embankment or tied into a concrete plinth. Selective foundation grouting can be used as a contingency measure along the upstream toe to reduce hydraulic conductivity in highly weathered or fractured bedrock.

18.7.7.5 Ongoing embankment raises (Stages 2 to 4)

The TSF embankments will be progressively raised using the centreline construction method, incorporating a lower permeability central zone (Zone S) along with quarried rockfill for the downstream shell (Zone C1). Transition and filter zones will be incorporated for compatibility and internal stability of the embankment fill materials. A rockfill zone (Zone C2) will be progressively constructed upstream of the embankment centreline to support the central zone and allow raises to be constructed proud of the beach elevation when needed. The upstream zone will also keep the tailings beach and pond further from the central zones of the embankment and will provide erosion protection from tailings discharge and wave runup.

18.7.7.6 Seepage management

Selective tailings deposition will be used to maintain wide tailings beaches and keep the supernatant pond remote from the embankments and reduce seepage. The normal operating pond will be stored in the depression formed by spigotted deposition from the TSF embankments. Embankment seepage will be collected in the water management ponds downstream of the main embankments.

18.7.7.7 Contact water management

Surface water runoff from the embankment faces or other impacted areas in the vicinity of the TSF will also be collected in the water management ponds located along the downstream toe of the embankments. Water collected in the water management ponds will be continuously monitored and pumped back into the TSF depending on water quality. Groundwater monitoring wells will be installed around the TSF to monitor seepage and water quality. If necessary, water interception wells would be installed where required to recover groundwater seepage and to pump it back to the water management ponds.

18.7.8 TSF stability & seepage modelling

18.7.8.1 Seepage analysis

Steady state seepage analyses were completed to estimate the phreatic surface for stability analyses and to estimate seepage flows for design of the seepage collection system. Seepage rates were estimated for unique sections of the embankments for start-up (containing only water at normal pond levels) and for the final TSF configurations (including water and tailings). These analyses were carried out to support the design of seepage management features in the TSF.

18.7.8.2 Stability analysis

Two-dimensional limit-equilibrium embankment stability analyses were carried out for both static and seismic conditions under the following cases:

- Static conditions during operations and post-closure
- Earthquake loading from the Operating Basis Earthquake (OBE) and the Maximum Design Earthquake (MDE)
- Post-earthquake conditions using residual (post-liquefaction) tailings strengths.

The results of the stability analyses satisfy the minimum factor of safety requirements and indicate that the proposed design is acceptable to maintain both short term (operational) and long term (post-closure) stability. The seismic analyses confirm that the estimated embankment deformations during earthquake loading from the OBE or MDE would be minor and would not have a significant impact on embankment freeboard or result in loss of embankment integrity. The results also confirm that the embankments are not dependent on tailings strength to maintain overall slope stability.

18.7.9 Construction methodology

The construction of the TSF is divided into the four main stages as shown in Table 18-9.

Table 18-9: TSF staging

TSF Stage	Embankment Crest Elevation	End Year	Primary Construction By
1	323 masl	-1	Contractor
2	347 masl	10	Mine fleet
3	369 masl	23	Mine fleet
4	376 masl	27	Mine fleet

18.7.9.1 Construction considerations

- Climate
- The primary scheduling constraints are due to seasonal variations and the resulting changes in temperature, snowpack, precipitation, and streamflow.

18.7.9.2 Environmental

- Water quality downstream of the embankments (silt and sediment control from impacted areas)
- In-stream flow requirements downstream of the work areas to maintain aquatic habitat
- Restrictions on timber clearing and other disturbance due to bird nesting.

18.7.9.3 Precipitation, runoff and temperature

- Access restrictions due to snowfall in winter months
- Lower construction productivities due to cold weather (e.g. ice hazards on access roads, material cohesion)
- Cold weather limitations on placing and compacting cohesive materials and welding geosynthetic liner
- Snow clearing
- Spring freshet, storm events, and flood flows.

18.7.10 Tailings pipeline and pump systems

The tailings pump and pipeline systems for the two tailings streams include:

- Tungsten (NPAG) Tailings Distribution System
- Molybdenum (PAG) Tailings Distribution System.

The NPAG tailings distribution system was designed to convey thickened slurry at 55% solids (by weight) from the process plant along the TSF embankments to the deposition spigots. The NPAG tailings distribution system will operate at a nominal slurry flowrate of approximately 1,750 m^3/h with surge capacity up to approximately 1,875 m^3/h. The NPAG distribution system will vary over the mine life between gravity flow and pressurized flow depending on the active spigot location and the elevation of the TSF embankment crest. Tailings will be discharged from a series of spigots along the TSF embankment crest, spaced at

approximately 500 m intervals. Rubber-lined steel pipes are required in some sections of the distribution system to manage pressures and abrasion associated with the thickened tailings flow. High density polyethylene (HDPE) pipe will be used where possible. The total tailings discharge pipe length is approximately 8 km.

The PAG tailings distribution system will convey molybdenum cleaner scavenger flotation tailings from the process plant to the TSF via subaqueous deposition at a nominal flowrate of approximately 240 m^3/h, with a surge capacity up to approximately 310 m^3/h. PAG tailings will be sub-aqueously discharged in a single HDPE pipeline directly into the TSF supernatant pond.

18.7.11 Water reclaim system

The TSF reclaim water system will pump water from the TSF supernatant pond to the mill head tank at the process plant for reuse. The pumping requirements and power load requirements are greatest in the early years and will decrease over the mine life as the elevation of the supernatant pond increases. The reclaim water system is sized to provide the maximum reclaim water demand required at the mill, and the maximum WTP demand concurrently. The water reclaim system will consist of the TSF reclaim barge and an 800 mm diameter HDPE pipe running approximately 2,500 m from the reclaim barge position to the mill head tank. The reclaim water system will operate at a maximum flow rate of approximately 3,190 m^3/h.

18.7.12 Instrumentation and monitoring

Quantifiable Performance Objectives (QPOs) and associated Trigger Action Response Plans (TARPs) will be developed for the TSF and water management ponds to provide a framework for performance monitoring. QPOs are defined as "measurable monitoring parameters that are required to be maintained within limits" and serve to provide a framework for performance monitoring (EMLI, 2024).

Geotechnical instrumentation comprising vibrating wire piezometers, slope inclinometers and movement monuments will be installed within the TSF embankments and foundations during initial construction and ongoing dam raises. Measurements will be collected and analyzed regularly during and after construction to monitor the response of the foundations and embankments to fill loading, tailings deposition, supernatant pond water level changes. Remote sensing techniques including drone-based photogrammetry, Light Detection and Ranging (LiDAR), and Interferometric Synthetic Aperture Radar (InSAR) may also be used to track site development and detect surface changes (e.g., embankment movement).

The instrumentation and surveillance data will be monitored against the QPOs during construction and operation of the TSF to assess facility performance and to identify conditions different to those assumed during design. Amendments to the ongoing designs and/or remediation work will be implemented to respond to changing conditions, should the need arise.

Flow and level monitoring instrumentation will be installed in the embankment foundation drainage system and seepage collection ponds. Groundwater monitoring wells will be installed at suitable locations downstream of the embankments to monitor water levels and water quality.

In addition to performance monitoring and routine inspections carried out by mine personnel on a shift/daily/weekly and monthly basis, an annual facility performance review (AFPR) will be completed by a suitably qualified professional engineer to confirm that the TSF is operating in a safe and efficient manner. A formal dam safety review will be conducted every five years during operation by an independent qualified geotechnical engineer.

18.8 Power and electrical

18.8.1 Facility power supply

Northcliff has negotiated power supply with the local utility, New Brunswick Power Corporation (NB Power). To supply approximately 58 MW of secure and reliable power to the Sisson Mine Project, NB Power requires the following:

- Construction of a new 42-km, 138-kV wood pole line from Keswick Terminal to Sisson Mine. The line will run parallel to the existing 345 kV line 3011 and requires an additional 25 m easement adjacent to the existing 50m right of way.
- Develop a new transmission line bay at Keswick Terminal which will include a new 138 kV breaker, associated disconnect switches, grounding switches, instrument transformers, and protection panels.
- Northcliff will be required to contribute to capital as defined by the NBSO OATT rules, Transmission Expansion Policy, Attachment K, for the supply of the 58 MW load (NB Power, 2026).

Separate from the power supply, NB Power is required to relocate approximately 9.5 km of 345 kV Line 3011 away from the proposed open pit mine, plant, and rock quarry. The replacement line will be approximately 10.5 km considering the proposed route.

Construction of the 138 kV line and Keswick Terminal modifications are required to commence mine operations; however, the 345 kV relocation is not required until approximately Year 8 within the current mine plan.

NB Power will own the line and the Keswick switchgear. Power costs are estimated at $0.075 per kWh in the Feasibility Study.

The transmission line infrastructure is shown on Figure 18-3.

18.8.2 Power demand

The maximum demand for the Sisson Project is estimated at 59.2 MW. The power demand for each phase is summarized in Table 18-10. The maximum total connected load for the Project will be 81 MW which includes all duty, stand-by and intermittent loads with a normal operating load of 59MW. With operational restrictions applied, the maximum load will be maintained within the 58 MW power supply by NB Power. To achieve this, the NAG tailings pumping system will be restricted to a single duty pump servicing only spigots in close proximity to the plant site during maximum winter HVAC loads.

Table 18-10: Sisson project electrical demand

Phase*	Maximum Demand (kW)
1 – Year 0-6	58,661
2 – Year 7-16	59,216
3 – Year 17+	59,216

*Phases 1-3 are defined by tailings pumping system expansions required due to TSF lifts.

Due to similar power requirement for all three phases, the outdoor substation is sized for the Phase 3 total load demand. Two 50/60/72 MVA, 138 kV/13.8 kV oil-filled power transformers will be installed, each capable of supplying the plant's maximum demand. The transformers will be connected to a 13.8 kV switchgear with a normally open bus tie. When one transformer is out of service, the power system configuration will allow the other to support the total process load, thus enhancing system reliability.

18.8.3 Site power reticulation

The site power reticulation system will distribute electrical power through a network of 13.8 kV overhead lines originating from the process plant's 13.8 kV switchgear housed within the primary electrical rooms at the outdoor substation.

Overhead distribution lines will be constructed using aluminum conductor steel-reinforced cable (ACSR) and supported by wooden poles.

The overhead powerlines will provide power from the 13.8 kV switchgear to the following facilities:

- Shovels
- SMW Garage
- Primary Crusher
- Secondary Crusher
- Process Plant
- Reclaim
- TSF Pond
- Pit Dewatering
- Truck Shop.

18.8.4 Power plant distribution

The largest electrical loads at the process plant are the two ball mills. The drive systems for ball mills include motors, and variable frequency drives (VFDs). The ball mill drive systems will be supplied via cable circuits from the plant's primary 13.8 kV switchgear. All other process and non-Process Plant loads will be powered via 4160 V and 600 V motor control centres (MCCs) housed within electrical rooms strategically located throughout the plant area.

Power to the electrical rooms will be supplied by resistance-grounded, secondary substation-type, oil-filled distribution transformers located adjacent to the respective electrical rooms. All electrical rooms will be adequately rated for the environment and outfitted with lighting and small power transformers, distribution boards, uninterrupted power supply (UPS) systems, fire alarm and detection, and HVAC systems.

To reduce installation time, the electrical rooms will be prefabricated modular buildings installed on structural framework above ground level for bottom entry of cables. Additionally, electrical rooms will be located as close as practical to the electrical loads to optimize conductor sizes and minimize cable lengths. Grounded pad-mounted and pole-mounted transformers will be used to step down the voltages at the SME truck wash, pit dewatering, shovels, reclaim, TSF pond, truck shop, and gate house, Power will terminate at the local 600 V distribution boards.

18.8.5 Emergency power supply

Emergency power is required across the site for critical loads, including remote areas such as the primary crusher, truck shop, and the TSF. Due to the widespread distribution, the emergency power for the entire site will be supplied by a single 5 MW, 13.8 kV, diesel-powered generator. The emergency generator will be connected to the 13.8 kV switchgear housed within the primary electrical rooms at the substation. Emergency power will be distributed throughout the site via the existing 13.8 kV overhead distribution line.

18.9 Fuel supply, storage, and distribution

Diesel will be stored near the mine truck shop in double-walled steel tanks for use in the mine fleet vehicles with a capacity of 266,000 L. The volume stored will represent 6 days of storage.

Allowance for light-vehicle fueling has been incorporated at a separate fueling bay adjacent to the truck shop.

Fuel will be delivered to the mine site via tanker trucks.

18.10 Fire protection

The firewater protection system for the overall project site will be serviced by two separate networks, each supplied by a combined fire / raw water storage tank and associated pumphouse.

At the upper mine site area, the tank and pumphouse will be located beside the main Process Plant building which includes an indoor ring main with sprinklers, hoses and wall hydrants. Underground branches with hydrants supply the Secondary Crushing Building and other nearby administration buildings. Conveyors local to the area between Secondary Crushing, the Process Plant, and the Fine Ore Bin will be protected by fire sprinklers along the galleries.

The firewater tank and pumphouse for the lower mine site area will be located beside the Truck Shop, which will include an indoor ring main supplying wall hydrants, sprinklers and fire hoses. An underground firewater branch will lead from the Truck Shop to the Primary Crushing area to supply hydrants and sprinklers.

Due to their length, the conveyors 2100-CV-002/003 between the upper and lower mine site areas will not have integral sprinkler systems and will instead be protected by fire truck.

19 Market studies and contracts

19.1 Introduction

This section summarizes the market study completed by Argus Media, as commissioned by Northcliff Resources Ltd. The QP has reviewed the market reports and the results support the assumptions used in this study

19.2 Tungsten market overview

Tungsten is a critical industrial and strategic metal primarily consumed in tooling, mining equipment, industrial machinery, automotive manufacturing, aerospace applications, and defense systems. Tungsten demand is closely correlated with global economic activity and manufacturing output.

Argus Media forecasts global primary tungsten demand to increase from approximately 90,000 t tungsten in 2026 to 129,000 t tungsten by 2040. Demand growth is expected to be driven primarily by industrial activity, defense spending, mining equipment consumption, and emerging applications associated with electrification and advanced technologies.

The defense sector is expected to remain the fastest-growing end-use market, supported by global rearmament, higher military spending, and increasing demand for armor-piercing ammunition, aerospace systems, missile technologies and shielding applications. Tungsten demand is forecast to rise from approximately 87,000 t in 2025 to about 128,900 t by 2040 (2.6% compound annual growth rate (CAGR)). While automotive, mining and industrial applications remain important demand pillars, defense-related consumption is expected to be a key growth driver through the forecast period.

19.3 Tungsten supply

The global tungsten market remains highly concentrated, although China is increasingly transitioning from a dominant exporter of tungsten products to a major importer of concentrate feedstock. Chinese mine production declined in 2025 because of ore grade depletion, mining quota restrictions and rising costs. Export controls introduced in February 2025 and additional restrictions on dual-use products have tightened global supply chains and elevated the strategic importance of non-Chinese supply sources.

Argus forecasts global primary tungsten supply increasing from approximately 79,500 t in 2025 to 123,250 t by 2040 in the base case. Key near-term developments include the Sangdong Mine in South Korea, the Boguta project in Kazakhstan and the Hemerdon restart in the United Kingdom. Despite these additions, forecast supply growth remains uncertain and project execution risk is expected to support prices over the medium to long term.

19.4 Tungsten trade and strategic importance

Tungsten has been designated as a critical mineral by numerous jurisdictions, including the United States, Canada, and the European Union. Concerns regarding supply-chain security and reliance on Chinese production have increased significantly in recent years.

China implemented export controls on several tungsten products in February 2025, including ammonium paratungstate (APT), tungsten oxides, carbides, and alloys. Concurrently, the United States and European Union have enacted policies aimed at diversifying supply chains and encouraging development of alternative sources.

The *European Union's Critical Raw Materials Act* (CRMA) and U.S. critical mineral strategies are expected to provide long-term support for new tungsten projects located in stable jurisdictions. Market participants increasingly recognize the strategic value of non-Chinese tungsten supply.

19.5 Tungsten pricing

Tungsten prices rose sharply during 2025 and early 2026 following Chinese export controls, concentrate shortages, growing defense-related demand and widening price divergence between Chinese domestic and ex-China markets. Argus considers tungsten to be enduring a structural market shift underpinned by supply security concerns and constrained mine supply.

Table 19-1: Tungsten concentrate price forecast to 2040

Year	Base (US$/MTU WO_3)
2026	2,215
2027	1,925
2028	1,385
2029	1,475
2030	1,520
2031	1,580
2032	1,660
2033	1,690
2034	1,720
2035	1,765
2036	1,775
2037	1,785
2038	1,840
2039	1,865
2040	1,880

Source: Argus, 2026

The forecast reflects persistent supply tightness, Chinese export controls, accelerating defense demand and a market that approaches balance around the end of the decade before potentially returning to deficit in the late 2030s if new mine supply underperforms expectations.

For this study, prices beyond 2040 use the 2040 price until end of production.

19.6 Tungsten off-site charges

For this study, a tungsten concentrate transport cost of $264/t concentrate was calculated based on shipping concentrate to four domestic facilities by road. See Table 19-2 for smelters considered and costs to calculate the average.

Table 19-2: Tungsten concentrate transport costs

Facility Name	Location	Travel Distance (km)	Cost (C$/t)
Elmet Technologies	Euclid, Ohio, United States	1,666	264.76
Kennametal	Latrobe, Pennsylvania, United States	1,530	245.50
Global Tungsten & Powders	Towanda, Pennsylvania, United States	1,236	201.55
Hyperion Materials & Technologies	Columbus, Ohio, United States	1,888	291.22

There is also a 0.15% deduction for tungsten losses, insurance and representation applied to tungsten gross revenue.

19.7 Molybdenum market overview

Molybdenum is primarily consumed in stainless steel, alloy steel, energy infrastructure, petrochemical facilities, transportation equipment, and defense applications.

Global molybdenum demand is forecast to increase from approximately 305,000 t in 2025 to approximately 432,000 t by 2040 under the Argus base case. Growth is expected to be supported by transportation, chemical and petrochemical industries, power generation, defense applications and energy-transition technologies.

19.8 Molybdenum supply and pricing

Molybdenum production is dominated by China, Peru, Chile, and the United States. Much of global supply is recovered as a by-product from copper operations.

Argus forecasts global molybdenum supply increasing from approximately 267,000 t in 2026 to 325,000 t by 2040 in the base case. Supply remains constrained by the by-product nature of much molybdenum production and declining ore grades at several major operations.

Table 19-3: China molybdenum concentrate (45% Mo) price forecast

Year	Base (US$/MTU)	Conversion (US$/lb)
2026	744	33.75
2027	788	35.74
2028	732	33.20
2029	696	31.57
2030	652	29.57
2031	638	28.94
2032	620	28.12
2033	613	27.81
2034	620	28.12
2035	624	28.30
2036	628	28.49
2037	629	28.53
2038	630	28.58
2039	629	28.53
2040	627	28.44

Source: Argus Media, 2026

For this study, prices beyond 2040 use the 2040 price until end of production.

19.9 Molybdenum off-site charges

Molybdenum treatment and refining charges and other offsite costs were determined using 12% of molybdenum in concentrate value. This assumption was carried over from the 2013 Feasibility Study and determined to be a reasonable estimate for this study.

19.10 Marketability of project products

The proposed tungsten concentrate and molybdenum concentrate products are expected to meet established industry specifications and are considered readily marketable. Potential customers include:

- tungsten processors and APT producers in Europe, North America, and Asia;
- carbide manufacturers and specialty metal producers;
- molybdenum roasters and alloy steel producers; and
- strategic consumers seeking non-Chinese supply sources.

Argus identifies Europe and the United States as particularly attractive markets due to supply diversification initiatives and dependence on imported tungsten units. China also remains a significant potential market, reflecting its growing concentrate import requirements.

19.11 Contracts

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

19.12 Conclusions

Independent market analysis indicates favorable long-term fundamentals for both tungsten and molybdenum. Demand growth is expected to exceed historic levels due to increasing defense expenditures, critical mineral supply-chain initiatives, industrial growth, and energy transition. The market remains highly dependent on Chinese supply, creating opportunities for new producers located in stable jurisdictions. Based on the Argus market study, there is reasonable evidence that future tungsten and molybdenum concentrate production from the Project can be sold into existing global markets under commercially achievable terms.

20 Environmental studies, permitting and social and community impact

20.1 Environmental considerations

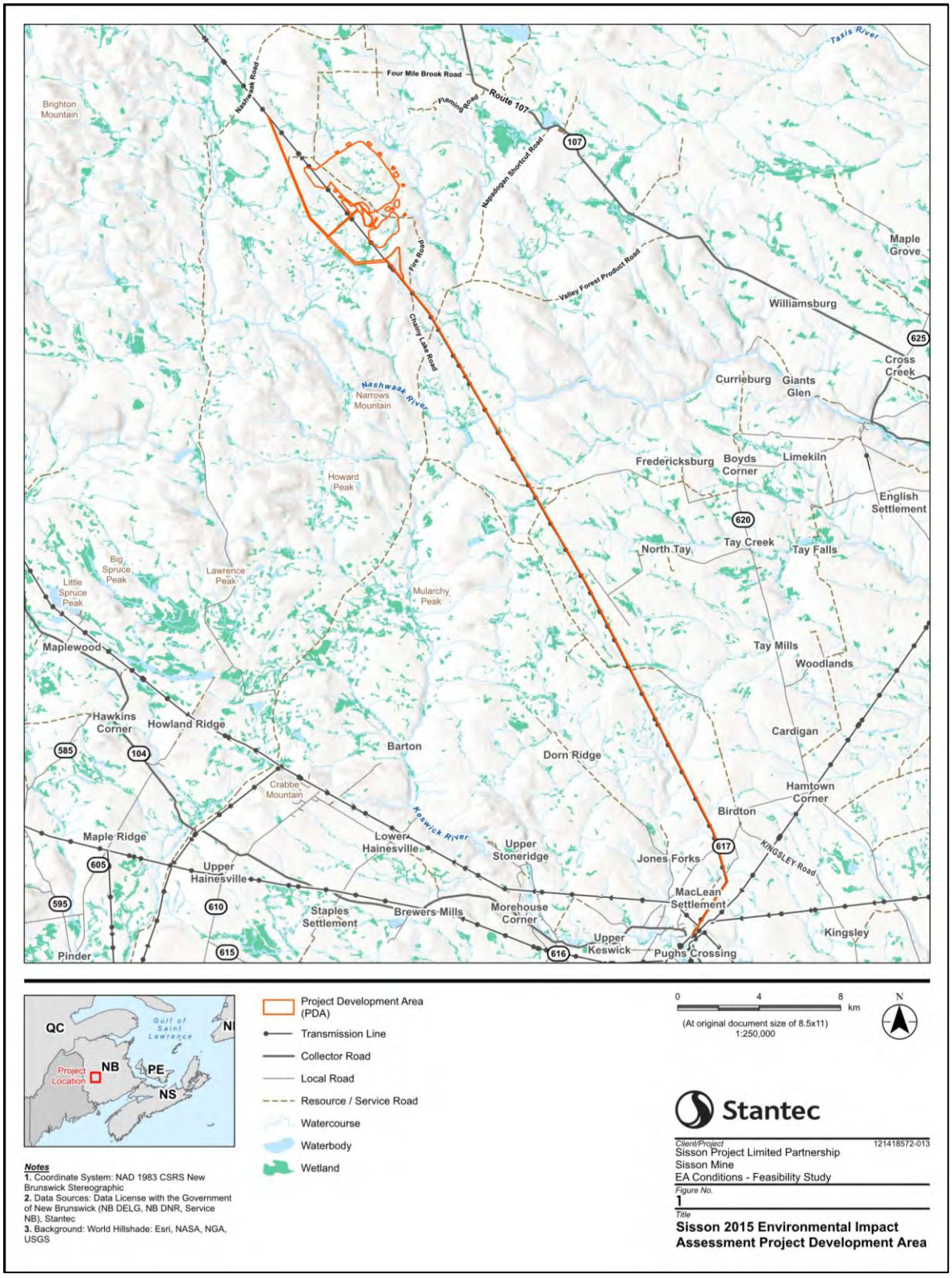
The Information presented in this section is based on publicly available information, including the Environmental Impact Assessment (EIA) Report (Stantec, 2015) and associated baseline studies for the Project. The environmental baseline studies were conducted in the Project Development Area (PDA) and surrounding vicinity. The PDA is defined as the area of physical disturbance associated with the Project, as per the EIA Report. The PDA is shown in Figure 20-1. The information included herein may require review and updates, should changes to the scope, area, or design of the Project occur as project planning and design progress.

The Project site is on provincial Crown land approximately 10 km southwest of the community of Napadogan, New Brunswick, and approximately 60 km directly northwest of the city of Fredericton. The Project also lies within the traditional territory of the Wolastoqiyik. The Napadogan area is a sparsely populated and relatively undeveloped area of Central New Brunswick. The area has a long history of natural resource development and use, particularly in support of extensive forest resource harvesting activities which have been central to the New Brunswick economy for over a century and are dominant at the landscape level. Mining will be a new natural resource-based development in the Napadogan area but is well-known elsewhere in the province. While the area is important for resource-based economic activities, it is also important for hunting, fishing, and outdoor recreation undertaken by the people of New Brunswick.

The Project straddles a topographical divide that separates the headwaters of the McBean Brook and Napadogan Brook watersheds. Both brooks drain to the Nashwaak River, which enters the St. John River at Fredericton, New Brunswick. Overall, water quality within the Nashwaak River watershed, as recorded by the Nashwaak Watershed Association (NWAI, 2024), is considered to be high, as would be expected for this relatively rural and undeveloped area in central New Brunswick. Key indicators of water quality including dissolved oxygen, E. coli, nitrate, and pH seldom exceed the Canadian Council of Ministers of the Environment (CCME) Freshwater Aquatic Life Guidelines, indicating that surface waters are, in general, suitable for supporting a variety of fish populations (NWAI, 2024). These observations are consistent with data collected by Sisson and Stantec in various water courses surrounding the Project location.

The Project lies within the Northern Appalachians seismic zone. Eastern Canada, including the Northern Appalachians seismic zone, is part of a stable continental region with a relatively low rate of earthquake activity.

Figure 20-1: Sisson Project EIA Project Development Area



Note: Stantec, 2026

20.1.1 Baseline and supporting studies

Table 20-1 lists the environmental baseline studies completed in support of the Project, between 2011 and 2013. These environmental studies were submitted as part of the federal Environmental Assessment (EA) and provincial EIA processes (Stantec, 2013; Stantec, 2015). Stantec conducted the baseline studies listed in Table 20-1, except where noted.

Table 20-1: Baseline studies

Attachment Name	Release Date	Summary of Findings
Baseline Ambient Air Quality Technical Report	June 1, 2012	The air contaminant concentrations measured at Napadogan were well below the applicable ambient air quality objectives and standards. No objectionable odours were detected during visits to Napadogan.
Baseline Sound Quality Technical Report	June 1, 2012	Residents along Highway 107 in Napadogan experience frequent elevated sound pressure levels due to truck traffic, including exceedances of the night-time noise guideline level. Sound pressure levels at the camps were generally low, with sporadic periods of elevated sound pressure levels.
Baseline Aquatic Environment Technical Report	June 1, 2012	Surveys found that aquatic habitats were supportive of a wide variety of freshwater fish species. Water quality and sediment quality in the area are generally good to very good and typical for a rural and undeveloped area in central New Brunswick. Typical fish species were found in the watershed. Atlantic salmon and American eel were found in the study area; no other aquatic species at risk (SAR) or species of conservation concern (SOCC) species were observed.
Baseline Aquatic Field Surveys of the Corridor for the Relocation of the Existing 345 kV Transmission Line and Fire Road	February 15, 2013	The water quality, fish habitat, and species observed in the linear facilities corridor are characteristic of areas that have been previously surveyed. No disturbance or direct loss of fish habitat is anticipated from the relocation of the transmission line.
Baseline Wildlife and Wildlife Habitat Technical Report	June 1, 2012	Considering conservation rankings current at the time of the baseline report, nine bird SAR or SOCC were observed in the survey area. Two mammal SAR were previously recorded in the survey area but were not observed during the field surveys. When updated to current conservation rankings, the number of SAR are 13 (8 bird, 4 mammal, 1 herptile) and SOCC 8 (bird).
Baseline Vegetated and Wetland Environments Technical Report	June 1, 2012	The area is typical of most regions in New Brunswick in that most wetlands are forested, although there is a notable absence of eutrophic wetlands compared to other ecodistricts in the Central Uplands Ecoregion. The forested wetlands are generally poor in nutrients, low in plant diversity and largely dominated by black spruce with ericaceous shrub understory. There is some peat formation in these wetlands. The scarcity of bogs in the Study Area is typical to the Central Uplands Ecoregion. Evidence of beaver activity of varying ages indicates that such activity has shaped the hydrology and vegetation communities of the wetlands in the Study Area.

Attachment Name	Release Date	Summary of Findings
Baseline Metal Concentrations in Soil and Biota Technical Report	June 1, 2012	Samples of soil, grass, poplar tips, small mammals, slugs, earthworms, wild berries, and fish were made to determine trace metal concentrations.
Baseline Socioeconomic Technical Report	June 1, 2012	The New Brunswick economy has traditionally been based on natural resource development, with tourism and communication technology industries also making substantial contributions. Land use in and around the Project site includes forestry operations and a considerable amount of recreational activity.
Baseline Heritage Resources Technical Report	June 1, 2012	Outside of watercourse shorelines, the Project site generally has a low potential for archaeological resources. Archaeological investigations have continued at the Project site; additional information is provided below.
Baseline Hydrogeology Report (by Knight Piésold)	January 2, 2013	Groundwater recharge will occur on higher ground and discharge primarily in the valley bottoms. Regional discharge is likely east toward West Branch Napadogan Brook. Groundwater levels are expected to be a subdued reflection of the ground surface with local groundwater flow toward streams and wetlands.
Baseline Water Quality Report (by Knight Piésold)	December 4, 2014	Surface waters at the site are very soft, poorly buffered, and slightly acidic to slightly basic, with metal concentrations driven primarily by groundwater recharge and peaking during low-flow periods. Few health-based Guidelines for Canadian Drinking Water Quality (GCDWQ) exceedances occurred, including arsenic and lead. Aluminum, cadmium, fluoride, pH, and iron most frequently exceeded the more stringent Canadian Environmental Quality Guidelines for the Protection of Aquatic Life (Freshwater) (CCME-FAL) across all catchments, and mercury exceeded the CCME-FAL in 11% of samples. Groundwater is calcium-bicarbonate type with pH and hardness increasing with depth and toward the open pit area. Health-based GCDWQ exceedances were limited to fluoride and arsenic. Iron and manganese frequently exceeded GCDWQ aesthetic limits and arsenic, fluoride, iron, pH, aluminum, and cadmium exceeded the CCME-FAL in both the TSF and open pit areas.

The results of the baseline studies were incorporated into the draft and final EIA Reports (Stantec, 2013; Stantec, 2015). A summary of archaeological investigations that have continued at the Project site is provided below. This summary was provided by Cultural Resource Management Group Limited, which undertook the work.

Evaluation of archaeological potential within the overall Sisson Project footprint began with modelling of archaeological potential for the area undertaken by the Archaeology and Heritage Branch (AHB) of the Department of Tourism, Heritage and Culture (DTHC) and was refined by numerous Archaeological Impact Assessments (AIA) undertaken by archaeological consulting firms between 2008 and 2015.

Archaeological assessment for the Project culminated in 2015 when Cultural Resource Management Group Limited was retained by AHB in partnership with the six Wolastoqey Nations to complete the remaining AIA requirements for an overall EIA. This archaeological project involved the training and employment of more than 100 Wolastogiyik individuals in the largest field project of its kind undertaken in the Maritimes.

Completing archaeological testing requirements across numerous areas of interest, the archaeological project involved the excavation and recording of 5,015 shovel tests over the span of 49 field days, as part of the Field Evaluation stage of an AIA. The formal training included a week of archaeological, first aid, and workplace safety training to precede the beginning of field work, and a 14-week college level Archaeological Technician course to be offered to project staff who complete the 12-week field program. On-the-job training throughout the field program was also provided.

The archaeological project concluded that archaeological concern within the proposed development footprint for the Project was restricted to two areas. These areas were identified as the locations of Pre-contact Period artifacts recovered during standard and intensified archaeological testing. The two areas were registered as archaeological sites CcDs-2 and CcDs-6. The specific age and nature of both sites remain undetermined.

Due to the proximity of these archaeological sites to the natural geological resources being sought for mining, and due to other environmental constraints in their surroundings, ground impacts at the sites cannot be avoided by enacting changes to project design. Likewise, site preservation cannot be achieved through the application of protective measures. For both sites, mitigation must consist of full documentation of all existing archaeological resources as they are exposed and removed during comprehensive archaeological excavation under the supervision of a permitted professional archaeologist prior to any construction. This mitigative excavation will be undertaken in partnership with the Wolastoqey Nation in New Brunswick (WNNB) and DTHC. The result will be a detailed and permanent record of the archaeological features and strata, as well as the recovery of artifacts. The work will be undertaken as outlined in a Heritage Mitigation Plan that is currently being finalized with review by AHB involving engagement with WNNB.

20.1.2 Environmental monitoring

A framework for the development of an Environmental and Social Management System (ESMS) was provided in the EIA Report (Stantec, 2015). A part of the ESMS is the Environmental Management Plan (EMP); a summary of the EMP is provided in this subsection. The EMP is planned to be completed in 2027.

The main purpose and objectives of the EMP are to:

- implement the Project in conformance with Sisson Project Limited Partnership's Responsible Mineral Development Policy;
- identify the general management strategies and plans required to implement key environmental commitments, and mitigation and management measures, for issues and risks identified during the EIA for the Project; and
- confirm the Project complies with the applicable laws and regulations of New Brunswick and Canada.

A Follow-up and Environmental Monitoring Plan will be prepared and implemented by Northcliff for the Project, building on the elements outlined and recommended in Chapter 9 of the EIA Report (Stantec, 2015), and as they are defined during regulatory review, approval, and permitting processes. It will consolidate the monitoring requirements outlined in the various component environmental management plans and will include, as they are specified, monitoring to comply with various permit requirements. In general, the plan will include necessary monitoring to meet the following requirements:

- Regulatory permits
- EIA follow-up programs under the *Canadian Environmental Assessment Act* as recommended in Chapter 9 of the EIA Report (Stantec,2015)

- Environmental effects monitoring under the federal Metal and Diamond Mining Effluent Regulations.

An Environmental Protection Plan (EPP) for construction will be developed to guide the construction of the Project in accordance with applicable federal and provincial environmental protection legislation and regulations. The EPP will be developed prior to the commencement of construction and will be submitted to the appropriate regulatory agencies for review and approval. In addition to referencing key aspects of design mitigation, the EPP will outline all environmental protection, mitigation, and response measures to be employed during construction. The EPP will be the key EMP document for construction.

The EPP will be designed to be a site-specific, field-usable document that will contain the following information:

- Responsibilities of Northcliff, contractors, and all site personnel.
- Purpose, organization and maintenance of the EPP including contractors and subcontractors.
- Specific mitigation measures to be implemented during routine and non-routine (e.g., fires) construction activities.
- Contingency plans to be followed in the case of an accidental event during construction (e.g., spill response, erosion and sedimentation control failure).
- A list of permits, approvals, authorizations, and key personnel to be contacted in case of an emergency.

The EPP will contain the applicable elements of the Emergency Preparedness and Response Plan for construction.

20.2 Water and waste management

The Project is situated in the headwaters Bird Brook, Sisson Brook, West Branch Napadogan Brook, and McBean Brook. There are few natural watershed areas upstream of the Project and clean surface water will be diverted around the Project facilities to the extent practicable. Surplus water from the Project site will be treated and discharged to Sisson Brook. The site water management considerations are presented in Section 18.6.

Contact water is managed at the TSF and will be used to meet Project water demands, driven primarily by process plant requirements. Contact water that exceeds the Project water demand will be treated, as necessary, to meet water quality objectives, including those within the federal *Fisheries Act* and the New Brunswick *Clean Water Act,* and discharged to Sisson Brook downstream of the open pit.

Tailings and Barren Rock will be stored in the TSF as described in Section 18.7.

20.3 Permitting considerations

An EIA of the Project was required under the *Canadian Environmental Assessment Act* (CEAA) as well as under the *New Brunswick Environmental Impact Assessment Regulation–Clean Environment Act* (the "EIA Regulation"). Though the former CEAA was repealed in 2012 and replaced with the new *Canadian Environmental Assessment Act*, 2012 ("CEAA 2012"), which itself was replaced by the *Impact Assessment Act* (IAA) in 2019, any environmental assessments at the comprehensive study level that had begun under the former Act (like the Project) would continue under the former Act; thus, all references to the CEAA in this EIA Report are to the Act as it existed before the passage of CEAA 2012.

The Project EIA Report (Stantec, 2013) was submitted to the federal and provincial governments for review in July 2013. Comments and information requests (IRs) on the July 2013 report were received from the provincial and federal governments, the public, and First Nations, and Northcliff responded to those IRs between October 2013 and November 2014. A Final EIA Report was submitted on February 2015 to meet the requirements of the provincial EIA Regulation, incorporating, where applicable and appropriate, the responses to IRs submitted by Northcliff to governments (Stantec, 2015). Thus, for the purpose of the provincial EIA process, the February 2015 EIA Report supersedes the July 2013 EIA Report submitted to both the provincial and federal governments for review. The July 2013 version remains the official version of the EIA Report for the purpose of the federal environmental assessment under CEAA.

The New Brunswick Department of Environment and Local Government (NBDELG) issued an approval for the Project in 2015 (NBDELG, 2015). Attached to the provincial EIA approval granted for the Project were 40 conditions that were to be fulfilled as the Project progresses to the start of construction, during construction, and operations. A Comprehensive Study Report was issued by the Canadian Environmental Assessment Agency in April 2016, which summarizes the Project and its environmental assessment (CEAA, 2016). The Project received EIA approval from the federal government in 2017 (Government of Canada, 2017).

Northcliff is currently working towards the fulfillment of the conditions associated with its in-hand provincial approval for the EIA. Of the 40 conditions of approval, there are 28 that must be completed before the start of construction, including revised water quality modelling results interpretation, water monitoring/management plan and water supply source assessment, wetland compensation plan, approval of the tailings dam, baseline environmental surveys, financial security plan, fish rescue plan, an environmental and socioeconomic management system and environmental management plan, and work related to the tailing storage facility. These also include obtaining applicable permits and approvals that are required, including obtaining watercourse and wetland alteration permit(s), an approval to construct, mining lease, and securement of appropriate land tenure, as well, ongoing compliance and reporting that is required regardless of the Project's development stage. The remaining 12 conditions of approval are tied to operations, and include continued compliance and reporting updates, permits and approvals for operation of the Project, training opportunities/programs, and the development of a reclamation plan. Because the Project could not substantively start prior to December 2020, Northcliff applied for an extension of the EIA approval in 2020. As part of the application, Sisson LP confirmed that there were no changes to the Project scope, location, or activity of the EIA approval issued December 3, 2015. The NBDELG reviewed the application and extended the Project commencement date by two years to December 3, 2022. Additional requirements were requested at this time by NBDELG, including a species at risk status update and an updated greenhouse gas inventory and related work. A second extension request was made on May 20, 2022, which again confirmed that there were no changes to the Project or the terms and conditions of the EIA approval issued December 3, 2025. The NBDELG reviewed the application and extended the Project commencement date by three years, to December 3, 2025. An additional request to extend the EIA approval was granted by NBDELG in 2025, extending the EIA approval to December 3, 2030. This latest extension also included additional requirements to which Sisson must update financial projections, continue to progress First Nations engagement, conduct further study into the proposed Tailings Storage Facility (TSF), and complete the Heritage Resource Protection Protocol. The additional requirements also included updated studies and field surveys, some of which are substantial undertakings. These include wetland delineations and functional assessments, ambient air quality modelling, updated migratory bird surveys and the completion of cavity nest surveys for pileated woodpeckers. These additional requirements are to be completed alongside the fulfillment of the EIA conditions for the Project to progress towards construction and operations.

A summary of residual effects for each valued component (VC) as per the EIA Report is provided in Table 20-3. The EIA Report should be consulted for a full description of predicted residual effects of the Project (Stantec, 2015).

Table 20-2: Summary of residual effects for valued components assessed in the EIA Report

Valued Component	Potential Environmental Effects	Summary of Residual Effects Predictions	Select Key Mitigation
Atmospheric Environment	Change in Atmospheric Environment	The dispersion modelling carried out for the Project predicted some limited exceedances of the ambient air quality objectives for particulate matter for both the construction and operation phases of the Project, as well as infrequent localized hydrogen sulphide (H_2S) levels above the odour threshold within the Project Development Area. However, these occurrences are expected to be infrequent, localized near the site and access roads, and of short duration. Annual direct GHG emissions from operation were estimated to be 47,691 t per carbon dioxide equivalent per year (including sources such as personnel, equipment, and material transportation), and less than one third that amount during construction. The residual environmental effects of the Project on a Change in Atmospheric Environment during all phases of the Project are rated not significant.	Implementation of equipment and vehicle maintenance program to improve operational efficiency and reduce emissions. Application of water sprays on the site access road connecting the Project site to the realigned Fire Road as well as on-site roads within the PDA as required to reduce dust generation. Seeding and re-vegetating topsoil and overburden storage piles as soon as possible after disturbance. Use of dust collection systems on the primary crusher and within the processing plant, and partial covering of conveyors. Use of H_2S and ammonia (NH_3) scrubbers on APT plant.
Acoustic Environment	Change in the Acoustic Environment	Residual project-related effects to sound pressure levels during the construction and operation phases are predicted to not be distinguishable from background levels at the nearest receptor to the Project, except for blasting events. Sound from blasting events will be infrequent and intermittent in nature. Sound and vibration effects will be managed with design selection and mitigation measures. The residual environmental effects of the Project on a Change in Acoustic Environment during all phases of the Project are rated not significant.	Limit construction activity to daytime hours where feasible. Limit blasting activity to daytime hours only, where feasible, and reduce the frequency of blasts. Notify nearby residents and camp owners of the blasting schedule. Routine trucking during daytime time hours only. Carry out preventative maintenance on equipment. Processing equipment enclosed in buildings.
Water Resources	Change in Water Resources	Residual Project-related effects to users of water resources during the construction and operation phases are related to downstream water quality and watercourse alteration. Although the modelling predicted that sodium concentrations in downstream watercourses will be elevated, this effect does not exceed the health-based guideline. Watercourse alterations will be authorized through permits. The residual environmental effects of the Project on a Change in Water Resources during all phases of the Project are not rated significant.	Document the pre-construction status and condition of water supplies at recreational campsites (nearest receptors). Maintain existing drainage patterns to the extent possible. Implement erosion and sedimentation control during construction and operation and document measures taken as prescribed in the EPP. Site potential freshwater wells for the Project outside the zone of influence of the TSF so that Project water quantity and quality requirements are met. Design water management structures to reduce erosion and assure adequate water conveyance in extreme events. Recycle water from the TSF for use in the processing to reduce Project demands on the environment for water, and to reduce the production of contact water. Install and operate groundwater pump-back wells below the northwestern TSF embankment to collect some groundwater seepage for return to the TSF if required. Flood the open pit during Closure to reduce the potential for metal leaching and acid rock drainage (ML/ARD) from remaining pit walls. Maintain ponded water over PAG tailings and waste rock within the TSF after closure to effectively mitigate the potential for ML/ARD.
Aquatic Environment	Change in the Aquatic Environment	Residual Project-related effects during construction will cause the indirect loss of fish habitat. A Fish Habitat Offsetting Plan has been developed for the Project. Effects on water and sediment quality will occur during each phase. Water releases will occur starting as early as year 1 and continuing through 27, and modelling predicts intermittent exceedances of applicable guidelines. Effects on the benthic macroinvertebrate communities during operation are predicted to occur until surplus water is released. A negligible effect on fish habitat productivity due to reduced stream flow is predicted. Effects on fish passage, fish health, and fish population as a result of the Project are related to changes in stream flow and water quality. Project activities are not anticipated to result in a notable reduction in the capacity of fish passage, a decline in fish health, or a notable reduction in spawning habitat. The residual environmental effects of the Project on a Change in Aquatic Environment during all phases of the Project are rated not significant.	Relocation of fish from watercourses within the TSF and open pit to nearby watercourses with suitable habitat. Siting of Project facilities to reduce disturbance of watersheds and watercourses. Design water management structures to reduce erosion and assure adequate water conveyance in extreme events. Treat (as required) surplus mine contact water before discharge to the environment. Construct engineered drainage collection channels to collect TSF embankment run-off and seepage and associated collection in lined WMPs which are pumped back to the TSF. Construct engineered drainage and diversion channels to divert non-contact water around the Project facilities wherever possible.

Valued Component	Potential Environmental Effects	Summary of Residual Effects Predictions	Select Key Mitigation
			Construct and operate a water treatment facility to treat surplus water from the Project before discharge, as required. Develop site-specific water quality objectives in watercourses downstream of the Project, consistent with CCME guidance, for implementation as part of the Approval to Operate. Maintain pit lake level to maintain it as a groundwater sink until water quality meets discharge conditions of the Approval to Operate.
Terrestrial Environment	Change in Wildlife Populations	Habitat loss is a residual Project-related effect during the construction and operation phases. However, no habitat will be lost that is unique to the region or critical to the survival of a wildlife SAR or SOCC population. Some habitat will be restored during Decommissioning, Reclamation, and Closure. Managed conservation areas will not be substantively affected by the Project. Secure species, as well as SAR, are not likely to be substantively affected by the Project activities. The residual environmental effects of the Project on a Change in Wildlife Populations during all phases of the Project are rated not significant.	Avoidance of, to the extent feasible, known locations of wildlife SAR and SOCC. Reduce the loss or fragmentation of mature forest habitat and interior forest. Co-location of linear facilities, where possible, to other linear disturbances to reduce the environmental effects of fragmentation. Reduce the size of temporary workspaces. Limiting clearing and grubbing to infrastructure footprint to that which is necessary. Maintenance of natural buffers around wetlands and riparian zones. Use of down-lighting, so as not to attract migrating birds An Avifauna Management Plan (AMP) to address incidental take. Establishment of buffers and protection of active migratory bird nests until fledging, upon their discovery in work areas. Scheduling of clearing activities outside the breeding season of migratory birds (when possible). Environmentally sensitive areas identified during clearing and construction will be flagged and avoided until an assessment has been completed.
Vegetated Environment	Change in the Vegetated Environment	Vegetation communities will be lost or altered as a result of the Project, with the extent of the loss limited to the PDA and adjacent areas that may be indirectly influenced by Project activities. This includes forested Crown land, including areas that are harvested and would be part of forest harvesting plans for close to an entire harvesting cycle (60 to 80 years). The Project will also cause the loss of Old Forest Community, which will be temporary until replacement stands are located and designated. No plant SARs were recorded during field surveys and the potential for encountering plant SAR was determined to be low. One plant SOCC was observed and mitigation measures have been identified to avoid disturbing the plant and adjacent area. Dust deposition on vegetation and soil is expected to negligibly affect soil quality and plant health. The residual environmental effects of the Project on a Change in the Vegetated Environment during all phases of the Project are rated not significant.	Clean, coarse fill material will be used for grading, to reduce the risk of introducing or spreading exotic and/or invasive vascular plant species. Construction machinery will be cleaned prior to entering and leaving wetlands to reduce the risk of introducing or spreading exotic and/or invasive species from one wetland to another. Any vascular plant SAR or SOCC within or adjacent to the PDA will be flagged and/or fenced off, and construction activities will be reduced in areas adjacent to SAR or SOCC, whenever possible. Construction activities will be reduced in wetland areas to reduce the potential environmental effects of disturbance, such as erosion and sedimentation. Conservation Vegetation Communities within the PDA will be replaced within the ecoregion and license block whenever stands meeting the criteria are available.
Wetland Environment	Change in Wetland Environment	Though the Project will result in the direct and indirect loss of area and function of some wetlands, the direct loss of GeoNB-mapped wetlands will be compensated. The extent of indirect loss of GeoNB-mapped and un-mapped wetland in areas outside of the PDA will be evaluated through a follow-up program, the results of which will be used to determine the requirements for adaptive management. Overall, it is not expected that these losses will be substantive. The residual environmental effects of the Project on a Change in Wetland Environment during all phases of the Project are rated not significant.	Clearing activities will be restricted to necessary portions of the PDA, and not beyond, to reduce the amount of habitat lost or altered through direct disturbance, or adjacent edge effects. Standard erosion and sedimentation control measures will be employed. Development of a Wetland Compensation Plan that meets the applicable provincial and federal requirements. Ongoing compensation for net loss of wetland function in accordance with the applicable provincial and federal wetland conservation policies. Quarried, crushed material will be used for road building in and near wetlands, to reduce the risk of introducing or spreading exotic and/or invasive vascular plant species.
Public Health and Safety	Change in Public Health	Health risk estimates for short-term exposures at the recreational campsites, nearest residences in Napadogan, and Human Health and Ecological Risk Assessment (HHERA) receptor locations meet the appropriate benchmarks. The Project is not expected to affect the health risks for long-term inhalation exposures, exposure to soil, or ingestion of water. Project-related activities have the potential to affect the health risks for consumption of fish for some metals; however, a long-term change in health status is not expected.	All mitigation measures identified for Atmospheric Environment, Water Resources, Aquatic Environment.

Valued Component	Potential Environmental Effects	Summary of Residual Effects Predictions	Select Key Mitigation
		The residual environmental effects of the Project on a Change in Public Health during all phases of the Project are rated not significant.	
	Change in Public Safety	Public access to the site will be limited by an entrance gate and the presence of security personnel. The safety risks to workers will be mitigated by complying with the requirements of various governing standards and regulations. The Project will not directly or substantially endanger the safety of the public and/or Project employees. The residual environmental effects of the Project on a Change in Public Safety during all phases of the Project are rated not significant.	Compliance with health and safety legislation, use of best practices for the operation of a mine (e.g., to reduce fly rock and other hazards), and safety by design. Mitigation measures identified for Transportation, and Accidents, Malfunctions and Unplanned Events.
Labour and Economy	Change in Labour	The Project's labour requirements will increase the demand on the labour market during all Project phases. However, these demands will be largely within the available supply of workers and tradespeople in New Brunswick. This represents an important contribution to employment in Central New Brunswick, where unemployment rates tend to be high with reduced economic activity. Availability issues in the province have not been forecast for the majority of construction trades and occupations. After 27 years of operation, Northcliff will work with these displaced workers to transition to retirement or retraining to re-enter the workforce as necessary. The residual environmental effects of the Project on a Change in Labour during all phases of the Project are rated not significant.	Northcliff will work with local education and training institutions and communicate work requirements to improve the availability of appropriate programs, allowing local people opportunities to gain qualifications for employment.
	Change in Economy	The Project will generate considerable opportunities for employment, income, taxation, royalties, and other direct, indirect, and induced economic benefits, particularly during construction and operation. Based on economic modelling, the Project will create considerable direct employment. The Project will be an important contributor to the Canadian GDP. Tax revenues will contribute significantly to provincial and federal government revenues. The Project will also contribute substantive indirect and induced employment and GDP from the supply of goods and services to the Project and from general improvement to the local, regional, and provincial economies. By Closure, the Project will have contributed substantially to the advancement of the local and regional economies such that any local goods or services added to the area to service the Project would be available for other industrial projects nearby or elsewhere. Similarly, the training and skills development of the labour force associated with the Project will be transferrable to other projects nearby or elsewhere and these skilled tradespeople would be available to service other projects upon closure. The residual positive environmental effects of the Project on a Change in Economy are expected to be significant for construction and operation and not significant during Decommissioning, Reclamation, and Closure.	Qualified local workers will be given priority consideration for Project employment. Local companies will be given preference for site contract work where qualified companies and suppliers can be identified. Northcliff will work with the local business community to communicate requirements and expectations for contracting opportunities and to identify new Project-related business opportunities for local companies. Northcliff will continue to engage with the public throughout operation, allowing sufficient time to plan for and mitigate any adverse environmental effects on Economy that may occur during Decommissioning, Reclamation and Closure. Northcliff will continue to engage with the public throughout its planning procedures for Decommissioning, Reclamation and Closure, allowing optimal time to plan for and mitigate environmental effects.
Community Services and Infrastructure	Change in Community Services and Infrastructure	There is sufficient capacity in temporary and permanent accommodations and housing to meet the demand of Project employees. The area will continue to be policed by the Royal Canadian Mounted Police (RCMP), the Fredericton Police Force, and the Woodstock Police Force throughout the Project life. Project employee policies and programs, such as drug and alcohol policies and employee assistance programs, and Project security considerations will help to reduce additional demands on police. The Project will have a fire truck and mine rescue vehicle on site and will be able to respond to any emergencies that occur at the Project site without having to place demands on existing fire and ambulance systems. Although hospitals near the Project are likely near capacity, this is a similar situation to conditions that are occurring throughout New Brunswick, and the Project is not expected to substantively affect the demand on hospitals. Existing recreational and entertainment facilities have sufficient capacity to meet the increase in demand that the Project will place on them, and the Project will create business opportunities for new facilities of this nature, should there be an increased demand for them. The residual environmental effects of the Project on a Change in Community Services and Infrastructure during all phases of the Project are rated not significant.	Human resource policies and practices. Employee Assistance Program. Environmental and Social Management System (ESMS). On-site security. Liaison with agencies and local authorities, and regular updates on Project activities and plans. Mine rescue and paramedics.

Valued Component	Potential Environmental Effects	Summary of Residual Effects Predictions	Select Key Mitigation
Current Use of Land and Resources for Traditional Purposes by Aboriginal Persons	Change in Land and Resource Use	The Project will change the primary land use within the PDA from forestry to industrial mining, but relatively few changes to the immediate area around the Project will occur as a result of the Project. Most of the PDA, including the open pit and TSF, is located on Crown land, and the Project will result in substantive economic benefits to the residents of New Brunswick; this change in the use of Crown land is consistent with the Crown Land Management Strategy. The Project will prevent access to the PDA by the public. However, there is more than adequate land within the region for recreational land and resource users to carry out activities at current levels. Portions of the open pit and TSF will be visible from areas accessible to the public. However, given the local landscape, they will not be visible from the nearby recreational campsites or other recreational areas in the area. The residual environmental effects of the Project on a Change in Land and Resource Use during all phases of the Project are rated not significant.	Forestry management plans will be revised by Crown licensees to incorporate the harvesting of forestry resources in the PDA as part of Site Preparation. Northcliff will provide information to licensees well in advance of construction to facilitate planning in collaboration with the New Brunswick Department of Natural Resources (NBDNR.). Where possible in accessible areas (i.e., along cleared right-of-ways), trees and other vegetation will be left in place or encouraged to grow to obstruct the view of Project facilities, reducing the change in the nature of the viewshed and muffling nuisance noise. No trespassing signs will be posted along the perimeter of the Project site to alert local land users of the presence of the Project and its facilities.
Heritage Resources	Change in Heritage Resources	With the implementation of the proposed mitigation, the Project will not result in the unauthorized permanent disturbance to, or destruction of a heritage resource considered by the provincial heritage regulators to be of major importance that is not mitigated. Archaeological shovel testing and excavation is being and will be carried out under provincial legislation and authorization and with full knowledge of First Nations. The residual environmental effects of the Project on a Change in Heritage Resources during all phases of the Project are rated not significant.	The footprint of the PDA was modified to eliminate some areas of elevated archaeological potential. Shovel testing was performed during assessment, and further archaeological test pitting will be undertaken prior to construction. A Heritage Mitigation Plan has been developed.
Transportation	Change in Transportation	Although the Project will contribute to an increase in traffic within the region during construction and operation, the Project contribution is small when compared to existing traffic volumes on the existing provincial highway network that has been designed to service the needs of industry, forestry operations, and residents of the area. There may be short-term, intermittent and temporary disruptions in traffic flow on the forest roads during construction of refurbishments to the existing forest road infrastructure and the transportation of materials, supplies, and equipment to the Project site. Through careful Project planning, Northcliff has considered the contribution of Project-related traffic on the forest roads within the area and is committed to working with the Crown Timber Licence Holders and NBDNR to refurbish and maintain the existing forest road network to facilitate the safe movement of road traffic within the area. The residual environmental effects of the Project on a Change in Transportation during all phases of the Project are rated not significant.	During construction, bussing of personnel to/from the Project location from off-site parking lots in Nackawic and Napadogan, and potentially from other towns. The designation of principal truck routes to the Project site to limit truck traffic to primary site access (PSA) and secondary site access (SSA) routes during all phases. Adherence to current design standards and best practices for forest road construction. In consultation with NBDNR and the Crown Timber Licence Holders, maintenance of the roadway and roadside warning signs to reduce traffic safety risks along the forest roads that are part of the PSA and SSA routes during all phases. In consultation with the New Brunswick Department of Transportation and Infrastructure (NBDTI), NBDNR, and the Crown Timber Licence Holders, the clearing of bushes along roadsides to improve sight distance at the approaches of the intersections of the PSA and SSA routes at provincial highways during operation. Compliance with the existing forest roads best practices that require use of CB radio systems (in Northcliff -controlled vehicles like heavy trucks and buses) for communicating the location of heavy or vehicles among drivers will reduce traffic safety risks along the PSA and SSA routes during all phases. Traffic Plan will be developed to guide Project employees and delivery vehicles that specifically identifies roadway hazards along the PSA and SSA routes.

Northcliff continues to progress its workstream to fulfill the 40 conditions associated with its provincial EIA approval. The fulfilment process requires Northcliff to come to an agreement with the provincial regulator on the scope of work for the respective EIA conditions, to which a dynamic and iterative review process is executed involving Northcliff and its consultants, and the province's Technical Review Committee (TRC), which includes certain entities within the Canadian federal government. Once the scope of work is agreed to, Northcliff will proceed in planning and executing the work program (e.g., obtaining necessary field and work permits). A report is then prepared by Northcliff describing the results of the work program for the respective EIA condition(s) and is sent to province. Once again, a dynamic and iterative process is executed to which review comments are provided by the TRC. The respective EIA condition is fulfilled once Northcliff receives confirmation from the province. All 40 EIA conditions of approval must be fulfilled for operations, of which a subset of 28 EIA conditions must be fulfilled for construction. The additional requirements associated with the extensions received in 2022 and 2025, along with any other requirements issued by the province since the EIA approvals must also be fulfilled alongside the EIA conditions.

20.3.1 Environmental permits

Following approval under the provincial and federal EA processes, numerous approvals, authorizations, and permits are required prior to initiating construction. Each of these permits or authorizations is applied for separately with relevant information included in the applications. Compliance with terms and conditions of approvals, standards contained in federal and provincial legislation and regulations, and commitments made during the EA/EIA processes (including application of mitigation measures and monitoring and follow-up requirements), will need to be assured throughout all project phases.

Table 20-3 and Table 20-4 provide a list of key approvals, authorizations, and permits that may be required from provincial and federal agencies and departments. The status of each approval, authorization, or permit is also identified.

Table 20-3: List of approvals, authorizations, and permits - federal

Environmental Permit, Approval, or Authorization Activity	Legislation	Issuing/Approval Agency	Status of Application
Permit, Licence or Certificate for the Manufacturing, Testing, Sale, Storage, Transportation and Importation of Explosives	*Explosives Act R.S.C.*, 1985, c. E-17 (Section 7) and attendant regulations	Natural Resources Canada	To be initiated during construction
Permit to Import, Export, or Transport Hazardous Waste or Hazardous Recyclable Materials	*Canadian Environmental Protection Act,* 1999 (S.C. 1999, c. 33) (Section 185(1)(b)) and Export and Import of Hazardous Waste and Hazardous Recyclable Materials Regulation	Environment and Climate Change Canada	To be initiated prior to operation
Authorization for Harmful Alteration, Disruption or Destruction (HADD) of Fish Habitat of the Destruction of Fish by Means Other than Fishing	*Fisheries Act,* c. F-14, Sections 34.4(2)(b) and 35(2)(b)	Fisheries and Oceans Canada	Completed

Environmental Permit, Approval, or Authorization Activity	Legislation	Issuing/Approval Agency	Status of Application
Scientific Collection Permit	Fisheries (General) Regulations 93-53 under the *Fisheries Act* (Section 52)	Fisheries and Oceans Canada	To be completed annually
Designation of a Waterbody as a Tailings Impoundment	Metal Mining Effluent Regulations 2002-222 (MMER) under the *Fisheries Act*	Environment and Climate Change Canada	Completed
Introductions and Transfers Permit	Fisheries (General) Regulations 93-53 under the *Fisheries Act (Section 56)*	Fisheries and Oceans Canada	To be completed as required for fish rescues

Table 20-4: List of approvals, authorizations, and permits - provincial

Environmental Permit, Approval, or Authorization Activity*	Legislation	Issuing/Approval Agency	Status of Application
Approval to Construct, Modify, or Operate a Source	Air Quality Regulation 97-133 – *Clean Air Act*	Environment and Local Government	To be initiated prior to construction and/or operation
Open Burning Permit	Air Quality Regulation 97-133 – *Clean Air Act*	Environment and Local Government	To be initiated prior to construction, if needed
Approval to Construct, Modify, or Operate a Source	Water Quality Regulation 82-126 – *Clean Environment Act*	Environment and Local Government	To be initiated prior to construction and/or operation
Approval of the Discharge Point	Water Quality Regulation 82-126 – *Clean Environment Act*	Environment and Local Government	To be initiated prior to operation
Approval of any Source, Wastewater Work, or Waterworks	Water Quality Regulation 82-126 – *Clean Environment Act*	Environment and Local Government	To be initiated prior to operation
Written permission to cease operation of a wastewater work or waterworks	Water Quality Regulation 82-126 – *Clean Environment Act*	Environment and Local Government	To be initiated prior to closure
Site Approval	Petroleum Product Storage and Handling Regulation 87-97 – *Clean Environment Act*	Environment and Local Government	To be initiated prior to construction
Environmental Approval	Petroleum Product Storage and Handling Regulation 87-97 – *Clean Environment Act*	Environment and Local Government	To be initiated prior to operation

Environmental Permit, Approval, or Authorization Activity*	Legislation	Issuing/Approval Agency	Status of Application
Storage license	Petroleum Product Storage and Handling Regulation 87-97 – *Clean Environment Act*	Environment and Local Government	To be initiated prior to operation
Watercourse and Wetland Alteration (WAWA) Permit	Wetland and Watercourse Alteration Regulation 90-80	Environment and Local Government/Health	Several WAWA permits have been received to allow work to proceed on the site. A WAWA permit will be applied for before the start of Construction.
Well Driller's Permit	Water Well Regulation 90-79 - *Clean Water Act*	Environment and Local Government/Health	To be initiated prior to construction
Work Permit to Conduct an Industrial Operation Upon Forest Land	*Forest Fires Act F-20*	Natural Resources	To be initiated prior to construction
Burn Permit	*Forest Fires Act* General Regulation 84-204	Natural Resources	To be initiated prior to construction, if needed
Special Permit – restricted highway access by weight	Highway Act H-5	Transportation and Infrastructure	To be initiated prior to construction
Mining Lease	*Mining Act* M-14., Section 67	Energy and Mines	To be initiated prior to construction
Quarry Lease	*Quarriable Substances Act* Q-1.1	Energy and Mines, Natural Resources, Finance	To be initiated prior to construction
Permit to Engage in Activities	*Species at Risk Act 2012* C.6, Section 35(1)	Natural Resources	To be initiated prior to construction
Transportation of Dangerous Goods Permit	*Transportation of Dangerous Goods Act 2011*, C.232 Regulation 89-67	Public Safety	To be initiated prior to construction
License of Occupation	Crown Lands and Forests Act C-38.1 (Section 26)	Natural Resources	To be initiated prior to construction
Topsoil Removal Permit	*Topsoil Preservation Act 2011*, c.230	Environment and Local Government	To be initiated prior to construction
Operating Permit	*Elevators and Lifts Act E-6*	Public Safety	To be initiated prior to operation

Environmental Permit, Approval, or Authorization Activity*	Legislation	Issuing/Approval Agency	Status of Application
Development and Building Permits	*Community Planning Act C-12*	Environment and Local Government/Rural District Planning Commission	To be initiated prior to construction
Boiler and Pressure Vessel Certificate	*Boiler and Pressure Vessel Act* c.122 (Section 13(1))	Training and Employment Development/Public Safety	To be initiated prior to construction

20.3.2 Additional permits and authorizations

Additional permits and authorizations beyond those listed in Table 20-3 and Table 20-4 above are not anticipated.

20.4 Social and community considerations

Northcliff is committed to engaging with Indigenous communities, the public, and stakeholders in an open, transparent, and responsive manner on the Project.

Northcliff established a public, stakeholder and Indigenous engagement program as part of its project development activities since 2010 and implemented this engagement program in advance of the EIA Report process to develop positive, long-term relationships and to facilitate the success of the Project. The objectives of this effort are to:

- Provide project information and seek input from members of the general public, Indigenous peoples, stakeholders, and interested parties.
- Identify, document, monitor and consider issues and concerns arising from the engagement process.
- Request information on the current use of lands and resources for traditional purposes by Indigenous persons and land uses by non-Indigenous persons and how those activities might be affected by the Project.
- Provide early notification of Sisson Project field activities, and public consultation opportunities associated with the EIA process.
- Identify the need for planning, design, and management measures that will avoid, mitigate, or resolve the issues raised through the public, stakeholder, and Indigenous engagement process.
- The engagement program continues to active with meaningful dialogue and project information sharing with the public, stakeholders and indigenous groups.

20.4.1 Indigenous engagement and agreements

Northcliff has engaged with local First Nations communities since it became involved with the Project. The Company established a strong record of consultation during the EIA process.

Northcliff sought engagement with First Nations communities in New Brunswick via letters, phone calls, and meetings. Communications and project information sharing with First Nations took place at meetings, open

houses and site visits, as well, project information was provided in newsletters, flyers and factsheets, as well as letters that notified First Nations of key regulatory milestones and open house events. The Assembly of First Nations Chiefs of New Brunswick (the Assembly) was the coordinating entity for providing Project information to other First Nations, except for the St. Mary's and Woodstock First Nations, that represented themselves. Northcliff met with representatives from St. Mary's, Woodstock, Tobique, Madawaska, and Kingsclear First Nations, as well as Mi'kmaq chiefs through the Assembly.

The New Brunswick government and the Wolastoqey Nation concluded the Sisson Mine Accommodation Agreement, an agreement concerning the development of the Sisson Mine Project in January 2017. The Wolastoqey Nation includes six communities Matawaskiyak (Madawaska), Neqotkuk (Tobique), Wotstak (Woodstock), Bilijk (Kingsclear), Sitansisk (St. Mary's) and Welamukotuk (Oromocto).

Since receiving the EIA approvals, Northcliff has continued to engage with the Wolastoqiyik, in particular, Wotstak (Woodstock First Nation) with which Northcliff has had a Co-Operation agreement since 2017. Also, Northcliff continues to collaborate with the WNNB, by sharing information relating to the Project. Northcliff signed a Capacity Funding Agreement with the WNNB in 2024. In December 2025, the Province of New Brunswick provided communication protocols to Northcliff which expanded its current engagement scope with the Wolastoqey communities to include the consultative bodies for the Mi'gmawe'l Tplu'taqnn Incorporated (MTI), Kopit Lodge, Peskotomuhkati Nation and reaching out to all First Nations communities in New Brunswick to ensure a transparent, open and inclusive engagement process.

20.4.2 Stakeholder engagement

Public engagement and public participation activities undertaken by Sisson have involved a wide range of stakeholders, including communities, fish and wildlife organizations, environmental non-governmental organizations, local business owners, and members of the public. Key community and stakeholder engagement activities have included:

- information sharing through the Project's website, newsletters, emails, information office;
- open houses staffed by Project team members, including consultants and regulatory bodies such as NBDELG;
- working groups that include community members, stakeholder groups, and the Project team;
- presentations, meetings, and workshops with stakeholder groups; and
- career information sessions about possible employment opportunities and availability of education and training programs offered locally.

The key topics raised during stakeholder engagement included:

- TSF design for heavy rain events or earthquakes;
- water quality of water from processing, the TSF, pit, and stormwater;
- management of acid rock drainage;
- effects on salmon spawning habitat and salmon *Species at Risk Act* (SARA) status;
- amount of fish habitat loss;
- wildlife impacts, including waterfowl landing in the TSF pond;
- effects of mine dust on mine workers, plants, and wildlife;
- emergency response services availability and communications with local fire departments;
- potential for increased truck traffic in neighbouring communities; and

- measures to protect recreational users around the Project.

Northcliff communicated responses to these topics and, where relevant, directed stakeholders to information in the EIA that addressed the topic. Additional details on stakeholder engagement can be found in Section 4.3 of the EIA.

Community outreach is ongoing regarding the status and activities on the project, to engage local contractors with the requisite capabilities and skills in support of the field programs, and to discuss longer term employment, procurement (supply chain) and other opportunities (capabilities and skillset development) of the project. Northcliff has engaged and is collaborating with New Brunswick based firms Ignite, Opportunities New Brunswick (ONB) and CC Goodwin Consulting to assist with communications designed to meaningfully share project information with First Nations and also other stakeholders and communities in the province. In 2026 Northcliff has:

- with certain of its consultants, held virtual meetings in March, April, July and August with the mayors and councillors of the local communities impacted by the Project;
- had multiple meetings with Opportunities ONB to understand the skills available in the province to support the development of the mine, as well, understand where the capabilities are and where to build the skills;
- attended and sponsored industry conferences in New Brunswick such as Exploration, Mining & Petroleum Conference held in Fredericton (October of every year) and Canadian Institute of Mining Conference, Bathurst New Brunswick (April 2026), and provided project updates through presentations and discussions at booth at conference booth;
- CEO met with certain service industry stakeholders in the Miramichi, Bathurst, Moncton and Fredericton (cities/urban centres in New Brunswick) in April; and
- plans to hold in person meetings in the fall of 2026 to have more detailed discussions about the results in the FSU and opportunities associated with development of the proposed project.

Additionally, Northcliff has a local hire mandate for ongoing field work. It has engaged local contractors for geotechnical drilling, site support and other activities associated with field programs.

20.5 Closure and reclamation planning

Following the completion of mining, decommissioning, reclamation and closure of the Project will be undertaken to establish physical, chemical and biological stability of the site to the extent practicable, and to meet desired end land uses, all as required under provincial and federal legislation and regulations. All facilities, buildings and other infrastructures will be removed and the sites reclaimed except for those that will be used for ongoing care and maintenance of the site. The water management system will be reconfigured as needed to support the long-term stability of the site. The TSF embankments and beaches will be capped and re-vegetated, and a spillway will run off direct to the open pit. Once the pit fills, water will be treated if necessary to meet regulatory requirements and discharged to Sisson Brook.

Financial security is required by the province for reclamation and closure of the Project, including ongoing activities in post-closure. The amount of the required security will grow over the life of the Project to an estimated value of $210 million at closure in Year 27. The estimated closure costs are based on a conceptual level closure and reclamation plan developed for the Environmental Assessment (EvEco, 2013). The closure concept is based on assumptions pertaining to water treatment requirements, soil replacement and revegetation details, and desired land end-uses.

20.6 Closure cost estimates

Estimated closure costs for the Sisson Project are $123 million for reclamation, closure, and post-closure water treatment excluding water treatment in perpetuity. The reclamation, closure and post-closure water treatment costs are used to determine the reclamation bond that is funded during the construction and production periods.

A total reclamation bond of $210 million would be accumulated over the mine life comprising $114 million in contributions and $96 million in accrued interest. For the economic analysis, these reclamation bond contributions as well as additional financial surety costs comprise the $147 million in total reclamation and closure funding over the life of mine.

Post closure water treatment costs comprise the majority of closure costs funded by the reclamation bond. The post-closure water treatment cost assumptions used for the economic analysis are set out below:

- Total annual costs for the WTP are estimated at $3.28 million for the first 11 years of closure, including annual sustaining capital costs of $0.74 million based on a 2.0% allowance and annual operating cost of $2.54 million based on the minimum flow for the modeled flow percentile of 95%
- Total annual costs for the WTP are estimated to be $5.92 million thereafter in perpetuity, including annual sustaining capital costs of $0.92 million based on a 2.5% allowance and annual operating cost of $5.00 million based on the average flow for the modeled flow percentile of 95%.
- Refer to Section 21.2.10 for additional details.

21 Capital and operating costs

21.1 Introduction

The capital and operating cost estimates presented in this FSU provide substantiated costs that can be used to assess the economics of the Sisson Project. The estimates are based on an open pit mining operation; the construction of a process plant; associated tailings storage facility and water management, utilities, and on-site and off-site infrastructure; as well as Owner's costs and contingency.

The estimates conform to Class 3 guidelines for an FS-level estimate with a ±15% accuracy according to the Association for the Advancement of Cost Engineering International (AACE International).

All costs are expressed in Canadian dollars, unless otherwise noted.

21.2 Capital costs

21.2.1 Overview

The capital cost estimate was developed in Q2-2026 CAD dollars based on budgetary quotations for equipment and construction contracts, as well as Ausenco's in-house database of projects and studies including experience from similar operations.

The estimate includes mining, crushing, processing, on-site infrastructure, tailings storage facility and water management, offsite infrastructure, project indirect costs, project delivery, owners' costs, and contingency.

The following parameters and qualifications were considered:

- No allowance has been made for exchange rate fluctuations.
- There is no forward escalation added to the estimate.
- A growth allowance has been included.

Data for the estimates have been obtained from numerous sources, including:

- mine schedules;
- engineering design and cost estimates by Ausenco, MMTS, KP, and ISC;
- Facility Study cost estimate for the power transmission lines by NB Power;
- topographical information obtained from the site survey;
- geotechnical investigations;
- budgetary equipment quotes from suppliers based in North America;
- budgetary unit costs from several local contractors for civil, concrete, steel, electrical, piping, and mechanical works; and
- data from similar recently completed studies and projects.

Major cost categories (permanent equipment, material purchase, installation, subcontracts, indirect costs, and Owner's costs) were identified and analyzed. A contingency was applied in the cost estimate and was based on ranging the accuracy of the data by discipline and WBS level 3 and applying a probabilistic method (Monte Carlo Simulation). An overall contingency amount was derived in this fashion.

The capital cost summary is presented in Table 21-1 by WBS and Table 21-2 by discipline. The initial capital cost for the Sisson Project is $1,528 million; and sustaining capital costs is $571 million; total costs are $2,099 million.

Table 21-1: Capital cost estimate summary by WBS Level 1

WBS LVL 1 Description	WBS LVL 1	Initial Cost (C$M)	Sustaining Cost (C$M)	Total Cost (C$M)
Mining	1000	193.3	215.4	408.7
Crushing	2000	210.6	-	210.6
Process Plant	3000	248.6	1.7	250.3
Tailings Storage Facility & Water Management	5000	118.3	256.0	374.3
Utilities	6000	71.7	-	71.7
On-Site Infrastructure	7000	114.2	-	114.2
Off-Site Infrastructure	8000	125.3	-	125.3
Total Direct Costs		**1,082.0**	**473.2**	**1,555.2**
Indirect Costs & Contingency	9000	445.6	97.9	543.6
Total Indirect Costs		**445.6**	**97.9**	**543.6**
Project Total		**1,527.7**	**571.1**	**2,098.8**

Note: Values may not sum due to rounding. Total direct and indirect costs may differ between presentations by WBS and by Discipline due to differences in cost allocation methodology. Aggregate values are consistent with those disclosed in the August 31, 2026 press release.

Table 21-2: Capital cost summary by discipline

Disc.	Discipline Description	Initial Cost (C$M)	Sustaining Cost (C$M)	Total Cost (C$M)
B	Earthworks	46.5	-	46.5
C	Concrete	67.5	-	67.5
S	Structural Steelwork	41.2	-	41.2
A	Architectural	65.3	-	65.3
F	Platework	44.9	-	44.9
M	Mechanical Equipment	235.7	1.7	237.4
P	Piping	58.5	-	58.5
E	Electrical Equipment	72.1	0.06	72.1
L	Electrical Bulks	40.0	-	40.0
I	Instrumentation	16.6	-	16.6
R	Third-party Packages	408.1	477.3	885.4
	Total Direct Costs	**1,096.4**	**479.0**	**1,575.5**
U	Field Indirects	53.5	44.9	98.3
O	Owner's Costs	31.5	-	31.5

Disc.	Discipline Description	Initial Cost (C$M)	Sustaining Cost (C$M)	Total Cost (C$M)
V	Other (Spares, Fills, Vendors)	33.6	-	33.6
T	Project Delivery	112.7	5.1	117.8
Y	Contingency	200.1	42.1	242.1
Total Indirect Costs		**431.3**	**92.1**	**523.4**
Project Total		**1,527.7**	**571.1**	**2,098.8**

Note: Values may not sum due to rounding. Total direct and indirect costs may differ between presentations by WBS and by Discipline due to differences in cost allocation methodology. Aggregate values are consistent with those disclosed in the August 31, 2026 press release.

21.2.2 Estimate exchange rates

Vendors and contractors were requested to price in native currency. The estimate is prepared in the base currency of Canadian Dollars (CAD). Pricing has been converted to CAD using the exchange rates in Table 21-3.

Table 21-3: Estimate exchange rates

Currency Abbreviation	Symbol	Currency	Exchange Rate
CAD	C$	Canadian Dollar	1.000
AUD	AU$	Australian Dollar	0.9901
EUR	€	Euro	1.6067
USD	US$	United States Dollar	1.3806

21.2.3 Direct costs – mine capital costs

Mine capital costs have been developed from detailed mine designs, production schedule and operational data for mining equipment. Mine development is to be completed by the Owner's fleet, made up of major mining production machines and smaller support equipment.

21.2.3.1 Equipment costs

Vendor quotations from 2025 Q4 are used for mining equipment, services and infrastructure costs. Mining equipment will be purchased new, and in the period it is required. Equipment productivity and availability are built up from first principals and supporting data from vendors.

Blasting quotes have been provided by three Canadian blasting contractors. Mining equipment quotes have been provided by five separate mining vendors and who have a presence in eastern Canada. Costs for tires, fuel and software have been sourced from local vendors.

A study on the viability of AHS specific to the Sisson project has been completed by equipment vendors. This study formed the basis of equipment cost inputs, infrastructure cost and labour factors and positions. Tele remote drill costs, infrastructure and labour factors and positions are provided by equipment vendors.

21.2.3.2 Mine development

The Owner's fleet will be responsible for pre-stripping areas, hauling construction rock and placement of rock as required. Material movement quantities have been developed based off 3D CAD designs and equipment productivities estimated based off vendor data (i.e. MinePlan haulage and CAT DOZSIM).

Productivities have been reduced for the preproduction period of the project to account for inefficiencies that are common with startup projects.

21.2.3.3 Mine labour

Hourly labour rates are calculated using two 12-hour shifts. Time for schedule delays and non-schedule delays are included. Operator efficiencies have been estimated per specific equipment based off operational data and vendor quotes. Salary positions include both shift-based schedules and Monday-Friday work weeks for various positions.

21.2.3.4 Project costs

Vendor quotes are used for the following other mining cost items:

- Site GPS and wireless data networks
- Geology, grade control and mine planning software licenses
- Mine communications systems (handheld and machine onboard)
- Tele-remote and Semi- Auto drill Server and Infrastructure
- Autonomous Hauling System Implementation
- Diesel fuel costs
- Power costs.

Operational data or allowance estimates are used for the following cost items:

- Maintenance tooling and supplies
- Mine rescue gear
- Mine survey gear and supplies
- Pit Electrification system (mobile substation, transformers, cables, connectors), see Table 21-5
- Mine fleet spare parts
- Crushing costs for rock for construction.

21.2.3.5 Mining costs

Mine capital costs are derived from vendor quotations and based on other projects' operational data.

Pre-production mine operating costs have been capitalized and are included in the capital cost estimate. These costs include drill and blast, load and haul, grade control, support and General Mine Expense (GME) costs.

The mine equipment fleet cost is shown in the year that equipment is required.

The costs of expanding the mine equipment fleet after mill start up, as well as replacement purchases, for units retired after their useful life, are treated as sustaining capital.

Mine Operations site development costs have been capitalized including:

- Clearing and grubbing

- Topsoil removal and stockpiling
- Crushed rock production
- Haul road construction.

The following Mine Operations Infrastructure items have also been capitalized:

- Site GPS and wireless data networks
- Mine survey gear and supplies
- Geology, grade control and mine planning software licenses
- Maintenance tooling and supplies
- Mine rescue gear
- Mine communications systems (handheld and machine onboard)
- Pit geotechnical instrumentation
- Mine fleet spare parts.

The mining cost estimate is presented in Table 21-4.

Table 21-4: Mine capital cost estimate

WBS LVL 2	WBS Description	Responsible	Initial Cost (C$M)	Sustaining Cost (C$M)	Total Costs (C$M)
1100	Mine Preproduction Development	MMTS	55.6	0.0	55.6
1200	Mine Development: Total New Capital	MMTS	88.9	52.0	140.9
1200	Mine Development: Total Replacement Capital	MMTS	0.0	150.2	150.2
1200	Mine Development: Other Capital Items	MMTS	10.6	2.5	13.1
1300	Electrical Equipment and Bulks for Mine Equipment	Ausenco	0.4	0.0	0.4
1400	Electrical Equipment and Bulks for Mine Infrastructure	Ausenco	0.7	0.0	0.7
1500	Mine Buildings and Infrastructure	Ausenco	4.0	0.0	4.0
1600	Mine Pit Dewatering	KP	3.5	10.8	14.3
1800	Quarry Development	MMTS	29.6	0.0	29.6
Mining Cost Total			**193.3**	**215.5**	**408.7**

Note: Values may not sum due to rounding.

21.2.4 Direct costs – crushing, process plant, tailings storage facility & water management, utilities, and on-site infrastructure capital costs

A summary of the crushing, process plant, tailings storage facility and water management, utilities, and on-site infrastructure capital cost estimate is presented in Table 21-5.

Table 21-5: Capital cost estimate summary – crushing, process plant, tailings storage facility & water management, utilities, and on-site infrastructure

WBS LVL 2	WBS LVL 2 Description	Initial Cost (C$M)	Sustaining Cost (C$M)	Total Cost (C$M)
	Crushing			
2100	Primary Crushing	66.5	-	66.5
2200	Coarse Ore Storage & Reclaim	25.3	-	25.3
2300	Secondary Crushing	41.1	-	41.1
2400	Tertiary Crushing	77.8	-	77.8
Crushing Total		**210.6**	**-**	**210.6**
	Process Plant			
3100	Grinding	89.7	-	89.7
3200	Molybdenum Flotation	34.9	-	34.9
3300	Bulk Sulphide Flotation	17.9	-	17.9
3400	Tungsten Flotation	38.8	-	38.8
3500	Molybdenum Concentrate Handling	12.3	-	12.3
3600	Concentrator Reagent Preparation	19.6	-	19.6
3700	Hot Tungsten Flotation	13.3	-	13.3
3800	Tungsten Concentrate Handling	9.1	-	9.1
3900	Tungsten Tailings and Thickening	13.0	1.7	14.7
Process Plant Total		**248.6**	**1.7**	**250.3**
	Tailings Storage Facility & Water Management			
5100	Tailings Disposal System	4.2	27.9	32.0
5200	Tailings Facility	42.6	215.5	258.1
5300	Reclaim Water System Subtotal	6.2	8.3	14.4
5400	Water Management Systems Subtotal	19.8	4.4	24.1
5600	Water Treatment Plant Subtotal	44.1	-	44.1
5800	Open Pit Water Management System Subtotal	1.5	-	1.5
Tailings Storage Facility & Water Management Total		**118.3**	**256.0**	**374.3**
	Utilities			
6100	Air Systems	6.1	-	6.1
6200	Water Systems	29.8	-	29.8
6400	Sewage & Wastewater Systems	1.0	-	1.0
6500	Electrical Services	32.3	-	32.3
6500	Electrical Services (Pit Electrification)	2.7	-	2.7
6500	**Electrical Services Subtotal**	**34.9**	**-**	**34.9**

WBS LVL 2	WBS LVL 2 Description	Initial Cost (C$M)	Sustaining Cost (C$M)	Total Cost (C$M)
Utilities Total		**71.7**	**-**	**71.7**
	On-Site Infrastructure			
7400	Administration/Office Facility	0.8	-	0.8
7500	Laboratory	3.5	-	3.5
7600	Truck shop	22.7	-	22.7
7700	Process Service Facilities	86.6	-	86.6
7800	Ancillary Facilities	0.6	-	0.6
On-Site Infrastructure Total		**114.2**	**-**	**114.2**
Total		**763.4**	**257.7**	**1,021.2**

Notes: Values may not sum due to rounding.
Ausenco prepared the capital cost estimate for WBS 2000–7000 using cost inputs from KP for the TSF, tailings disposal system, reclaim water system, water management systems, and open pit water management system, from ISC for the WTP, and from MMTS for the pit electrification, as detailed in Section 21.2.4.1, Section 21.2.4.2, and Section 21.2.3.

The definition of process equipment requirements was based on process flowsheets and process design criteria, as defined in Section 17. All major equipment was sized based on the process design criteria to derive a mechanical equipment list. Mechanical scopes of work were developed and sent for budgetary pricing to equipment suppliers (see Table 21-6 and Table 21-7). For mechanical equipment costs, 96% of the value was sourced from budgetary quotes; the remainder was sourced by benchmarking against North American concentrator mining projects and studies.

Table 21-6: Mechanical equipment price basis

Source	Total (C$M)	%
Budgetary Quotes	211.4	96.0
Estimated (database)	8.4	4.0
Total	**219.8**	**100.0**

Note: Costs exclude freight and growth.

Table 21-7: Mechanical equipment & packages

Package No.	Equipment
P0001	Crushers
P0002	Apron Feeders
P0003	Belt Conveyors and Belt Feeders
P0004	HPGR
P0005	Vibrating Screens
P0006	Overflow Ball Mill
P0007	Flotation Cells
P0008	Flotation Columns
P0009	Shop Fabricated Tanks

Package No.	Equipment
P0010	Filter Presses
P0011	Samples, Multiplexers, and Analyzers
P0014	Liner Handlers
P0015	Rock Breaker
P0016	Overhead Bridge Cranes
P0017	Dust Control System
P0018	Hydrocyclones
P0019	Agitators
P0020	Thickeners
P0021	Slurry Pumps
P0024	Rotary Screw Air Compressors
P0025	Potable Water Treatment Plant
P0026	Sewage Water Treatment Plant
P0027	Fire Protection System
P0029	Scrubbers
P0031	Centrifugal Solution Water Pumps
P0037	Regrind mill
P0038	Pneumatic Flotation Cells
P0039	Reagent Systems
P0045	Off-Plot Piping and Valves
P0046	Other Pumps
P0047	Fans, Blowers, Air Receiver, Air Compressor, Air Dryer
P0048	Systems, Packages, Buildings, Plants
P0049	Heaters
P0050	Cranes & Lifts
P0051	Pressure Washer, Shower and Eyewash Station
P0052	Mixers, Scale, Eductor, Screw Feeder, Filters

Similarly, the major electrical equipment was sized based on the project's equipment list. Scopes of work were developed to receive budgetary pricing from equipment suppliers (see Table 21-8 and Table 21-9). For the electrical equipment, 93% of the value was sourced from budgetary quotations. The remainder was sourced by benchmarking against other recent North American flotation concentrator mining projects and studies.

Table 21-8: Electrical equipment supply price basis

Source	Total ($/M)	%
Budgetary Quotes	62.4	93.4
Estimated (factored)	4.4	6.6
Total	**66.8**	**100.0**

Table 21-9: Electrical equipment & packages

Package No.	Equipment
P2005	Process Control System (PCS) & RIO Cabinets
P0040	Electrical Rooms/Control Rooms & Associated Equipment
P0042	Transformers/NGR/Surging Arrestors
P0043	Diesel Generators
P0044	Substations/Surging Arrestors/Circuit Breakers/Disconnect Switch

In support of the major installation construction contracts, engineering for the process plant and infrastructure was completed to a FS level of definition, allowing for bulk material quantities (earthworks, concrete, structural steel, architectural, platework, piping, electrical and instrumentation bulks) to be derived from major commodities, as outlined in Table 21-10.

Table 21-10: Total project costs summary by major commodity

Code	Commodity Description	Total (C$M)
A	Architectural	65.3
B	Earthworks	46.5
C	Concrete	67.5
E	Electrical equipment	72.1
F	Platework	44.9
I	Instrumentation	16.6
L	Electrical bulks	40.0
M	Mechanical equipment	237.4
O	Owner's costs	31.5
P	Piping	58.5
R	Third party estimates	885.4
S	Structural steelwork	41.2
T	Project delivery	117.8
U	Field indirects	98.3
V	Other (spares, fills, vendors)	33.6
Y	Contingency	242.1
Project Total		**2,098.8**

Note: Values may not sum due to rounding.

After the derivation of all the bulk material quantities (earthworks, concrete, steel, piping, cables, etc.), for the process plant, tailings storage facility and surface infrastructure areas, major construction contracts were formed and submitted to the market for budgetary pricing, as per Table 21-11.

Table 21-11: Construction contract packages

Package No.	Equipment
C5001	Earthworks
C5002	Concrete
C5003	Pre-Engineered Buildings
C5004	Modular Buildings
C5005	SMP (Structural, Mechanical, Piping)
C5007	Overhead Power Lines
C5008	Electrical Equipment Install & Electrical Bulks
P0101	Platework Supply
P0102	Structural Steel Supply
P0103	In-Plant Piping Supply
P0104	Off-Plant Piping Supply
P0105	Valves Supply
PFB01	Piping Bulk Factors
EFB01	Electrical Bulk Factors
IFB01	Instrumentation Bulk Factors

21.2.4.1 Tailings Storage Facility (TSF) and Water Management

The initial and sustaining capital, and operating expenditures for the TSF and water management scope include:

- Staged construction of the Tailings Storage Facility (TSF) embankments
- Construction of the water management ponds, collection channels, and diversion channels
- Installation of pump and pipeline mechanical systems for tailings distribution, reclaim water, open pit dewatering, and Water Management Pond reclaim/seepage collection
- Local electrical, instrumentation, and control systems for in-scope pump systems
- Planned replacement of in-scope mechanical systems throughout the 27-year mine life

Major items excluded from the TSF estimate and covered by Others include quarrying and hauling of rockfill (Zones C1/C2); construction and operation of water treatment facilities; electrical power supply to pump stations; and access roads.

The estimate was developed using the following methodology:

- Key activities required to achieve the scope of work were identified
- Unit rates were developed from first principles by estimating required production rates, assigning corresponding equipment fleets and labour, and iterating to match equipment to material quantities and cycle times, considering material densities, efficiencies, and haul distances
- Neat-line quantities for key activities were estimated from the design drawings and 3-D modelling
- Design growth and waste factors

- Annual costs were estimated by applying total quantities and related unit rates to the construction schedule
- Lump sum allowances and provisional costs were included where sufficient detail was not available to develop unit rate estimates.

Early site works (access, site establishment, impoundment basin preparation, cofferdams, starter embankments, water management earthworks) and specialty work (synthetic liners, filter/transition zones) will be constructed by Contractors. Rock quarrying, waste rock hauling, and rockfill hauling and placement will be carried out by the Mine. All mechanical and electrical work will be carried out by Contractors.

Table 21-12: Sisson TSF – capital cost estimate

Tailings Storage Facility & Water Management		Initial Cost (C$M)	Sustaining Cost (C$M)	Total Cost (C$M)
5100	Tailings Disposal System	4.2	27.9	32.0
5200	Tailings Facility	42.6	215.5	258.1
5300	Reclaim Water System	5.4	8.3	13.6
5400	Water Management Systems	4.4	4.4	8.7
5800	Open Pit Water Management System	0.7	-	0.7
Tailings Storage Facility & Water Management Total		**118.3**	**256.0**	**374.3**

21.2.4.2 Water Treatment Plant (WTP)

Capital costs associated with the WTP were developed by ISC. These estimates were prepared using first-principles engineering methods and factored equipment costs. Values are expressed in Q2-2026 Canadian dollars and reflect an expected confidence interval of -15% to +35%. The Sisson WTP total direct and indirect costs are estimated at approximately C$44.1 million.

Table 21-13: Sisson WTP – capital cost estimate

WBS LVL 2	WBS Description	Responsible	Initial Cost (C$M)	Sustaining Cost (C$M)	Total Costs (C$M)
5600	Water Treatment Plant				
5600	Direct Costs - WTP	ISC	20.4	0	20.4
5600	Indirect Costs - WTP	ISC/Ausenco	10.5	0	10.5
5600	Civil & Concrete Quantities	ISC/Ausenco	4.3	0	4.3
5600	Concrete & Electrical Quantities (separate from ISC Scope)	Ausenco	6.1	0	6.1
5600	Growth Allowance	Ausenco	2.8	0	2.8
Water Treatment Plant Cost Total			**44.1**	**0**	**44.1**

Notes: Values may not sum due to rounding. The table represents the WTP estimate by ISC for direct and indirect costs. ISC provided Ausenco quantities for detailed civil works and concrete to price using Ausenco's contractor rates and include the costs in the consolidated Capex. Ausenco applied a growth allowance on the full WTP scope based on the maturity of their design and costs.

21.2.5 Direct costs – off-site infrastructure capital costs

A summary of the off-site infrastructure cost estimate is presented in Table 21-14.

Table 21-14: Off-site infrastructure capital cost summary

WBS LVL 2	WBS LVL 2 Description	Initial Cost (C$M)	Sustaining Cost (C$M)	Total Cost (C$M)
8100	Power Transmission Lines	112.4	-	112.4
8200	Site Access Roads	12.9	-	12.9
Off-Site Infrastructure Total		**125.3**	**-**	**125.3**

Note: Values may not sum due to rounding.

21.2.5.1 Power supply

The cost estimate for power transmission facilities and modifications are based on the 2026 Facilities Study for Interconnection of a Mining Project at Sisson Mine, NB prepared by NB Power (NB Power, 2026). The cost estimate incorporates an accuracy range of -25% to 35%. It is assumed that the project will be largely executed through multiple contractors while under the supervision of NB Power.

The power transmission facilities required for interconnection of the Sisson Mine project includes 42 km of new 138 kV line from Keswick Terminal to the Sisson Mine site, Keswick Terminal modifications, as well as relocation of 9.5 km of the existing 345 kV Line 3011.

21.2.5.2 Off-Site roads

The Site Access Roads (off-site infrastructure) are described in Section 18.2 of this Technical Report.

The estimate allows for diversions and upgrades to the existing unpaved site access roads. Existing roads that constitute the primary site access and secondary site access, totaling 52.1 km, will be retained and upgraded through localized widening, geometric improvements, resurfacing, drainage enhancements, culvert replacements, and safety upgrades.

The 11 km Re-Aligned Fire Road and 3 km Site Access Road will be constructed as new roads to provide the required access to the Project site. A material take-off was completed based on a Civil3D model design of the upgraded road and was applied against the contractor installation rates quotes for typical activities such as clearing, topsoil removal, excavation, and backfilling with granular material.

21.2.6 Growth allowance

A growth allowance has then been allocated to each line item in the capital cost estimate to reflect the level of definition of design and pricing strategy, of which is a provision for additional costs that will be recognized in future project phases as engineering is advanced.

Estimate growth is:

- intended to account for items that cannot be quantified based on current engineering status but empirically known to appear;
- accuracy of quantity take-offs and engineering lists based on the level of engineering and design undertaken at Feasibility Study level; and

- pricing growth for the likely increase in cost due to development and refinement of specifications as well as re-pricing after initial budget quotations and after finalization of commercial terms and conditions to be used on the project.

Growth has been calculated on a line-item level by evaluating the status of the engineering scope definition and maturity and the ratio of the various pricing sources for equipment and materials used to compile the estimate. The growth rate applied was based on guidance aligning to a Class 3 AACE estimate, and the level of definition of the project scope. The capital cost growth allowance is presented in Table 21-15.

Table 21-15: Growth cost summary

Discipline Description	Total Growth (C$M)	Total Growth %
Earthworks	4.6	9.9%
Concrete	5.0	7.4%
Structural Steelwork	3.1	7.4%
Architectural	4.8	7.4%
Platework	3.3	7.4%
Mechanical Equipment	17.6	7.4%
Piping	4.8	8.3%
Electrical Equipment	5.3	7.4%
Electrical Bulks	3.3	8.3%
Instrumentation	1.4	8.3%
Third Party Packages	23.8	2.7%
Total Directs	**77.0**	**4.9%**
Field Indirects	8.9	9.1%
Owner's Costs	-	-
Other (Spares, Fills, Vendors)	3.1	9.1%
Project Delivery	-	-
Contingency	-	-
Total Indirects	**12.0**	**2.3%**
Total Growth	**89.0**	**4.2%**

Note: Values may not sum due to rounding.

21.2.6.1 Exclusions

The following costs and scope will be excluded from the capital cost estimate:

- Operating costs (except capitalized mining costs and training of new operations employees during capital period)
- Taxes and duties
- Future exploration costs
- Environmental approvals
- Special incentives (schedule, safety or others)

- No allowance has been made for loss of productivity and/or disruption due to religious, union, social and/or cultural activities
- Escalation beyond the base date Q2-2026
- Environmental impact assessment/permitting
- Future scope changes
- Demolition and salvage of any existing on-site structures
- Lost time due to weather, labour availability, and disruption or force majeure events
- Training of operations personnel
- Sunk costs including land acquisition
- Costs associated with geotechnical field investigations
- Final closure plan and estimate (included in financial model)
- Management reserve
- Financing costs.

21.2.7 Indirect costs

Indirect costs (Area 9000) include all costs that are necessary for project completion but not related to the direct construction cost, including project indirect costs, Owner's costs, and provision costs, as outlined below.

Project indirect costs include the following:

- temporary construction facilities, utilities and construction services;
- project delivery (EPCM) costs;
- vendor representative costs during commissioning and construction;
- spares for operating, commissioning and capital;
- first fills and initial charges; and
- contractor commissioning assistance. Owner's costs include the following:
- owner's execution team and staffing;
- office administration equipment and software
- Insurance
- Community development
- Legal, permits, and fees
- royalties
- operational readiness; and
- closure.

Provision costs include the following:

- contingency.

Indirect costs are summarized in Table 21-16 based on presentation by WBS and are described in the following sections.

Table 21-16: Indirect costs

WBS LVL 2	WBS LVL 1 Description	Initial Cost (C$M)	Sustaining Cost (C$M)	Total Cost (C$M)
9100/9200	Project Preliminaries	214.1	55.8	270.0
9500/9600	Owner's Cost	31.5	-	31.5
9900	Contingency	200.1	42.1	242.1
Total Indirect Capital Cost		**445.6**	**97.9**	**543.6**

Note: Values may not sum due to rounding.

21.2.7.1 Project indirects

21.2.7.1.1 Temporary services, maintenance of temporary facilities, temporary buildings, temporary site facilities, and temporary utilities

Project preliminaries (construction field indirects) are items or services which are not directly attributable to the construction of specific physical facilities of plant or associated infrastructure but are required to be provided as support during the construction period.

These costs may include:

- Temporary construction facilities - site offices, induction centre, first aid facilities, admin, portable toilets, temporary fencing, temporary roads and parking.
- Temporary utilities and services – power supply, temporary grounding and generators, construction lighting, and water supply, snow removal.
- Construction support – site clean-up and waste disposal, material handling, maintenance of buildings and roads, testing and training, service labour, site transport, site surveys, QA/QC, and security.
- Construction equipment, tools and supplies purchased by the Owner or EPCM contractor – heavy equipment and cranes, large tools, consumables, scaffolding and purchased utilities.
- Material Transportation & Storage incurred by the Owner or EPCM contractor – All types of freight, agents, staging and marshalling.
- Site office – Local services and expenses, communications and office furniture.

KP indirects were calculated using KP's benchmarks of ratios over total direct costs.

Construction field indirect costs were allocated based on estimates and allowances. The total estimated costs for temporary construction facilities, utilities, and construction services is $53.5 million.

21.2.7.1.2 Construction camp, accommodation, and messing/living out allowance

No onsite camp was allowed for in the estimate, all labour can be sourced from within the region and can utilize the nearby towns for accommodation. It is the responsibility of the contractors to provide their own meals during the working day. An allowance of $10-$12/h, depending on the rate considered by key contracts, has been included in the direct costs under the contractor labour rates for each major contract as a living-out-allowance including local accommodations and subsistence.

21.2.7.1.3 Project delivery (EPCM)

Engineering, Procurement and Construction Management (EPCM) services costs cover such items as engineering and procurement services (home office based), construction management services (site based),

project office facilities, IT, staff transfer expenses, field inspection and expediting, commissioning, corporate overhead, fees and profit.

The EPCM (engineering, procurement, and construction management) estimate has been developed from a deliverables list and by identification of resources over a defined schedule. A detailed assessment of consultants and project general expenses and commissioning services are also included in the EPCM costs.

The total estimated EPCM cost for the project is C$113.3 million.

21.2.7.1.4 Vendor representatives and assistance

Vendor representative costs during commissioning and construction includes vendor representative support during the installation of the purchased equipment and during commissioning of the equipment.

Vendor representative costs have been estimated based on previously completed North American concentrator mining projects.

The total vendor representatives and assistance cost is estimated to be $6.3 million.

21.2.7.1.5 Spares

Commissioning spare quantities were recommended and priced by equipment suppliers. Where equipment pricing was not solicited from vendors, factors were applied based on standard estimating practices.

Capital spares prices for mechanical equipment are based on the prices provided by equipment vendors during the enquiry process. If vendors did not provide a cost for capital spares, a factored allowance was included based on the supply price and benchmarked against Ausenco's in-house database of projects. Benchmark factors were based on a six-month period of capital spares.

The total cost of spares is estimated to be C$16.9 million.

21.2.7.1.6 First fills and initial charges

First fills include the costs for the initial construction, first fills for installed equipment and commissioning first fills which consist of chemicals, fuels, lubricants, etc. and is an allowance based on historical data.

Commissioning First Fills costs were developed and included in the OPEX, the cost of the initial fills for process and construction is included in the Capex. The total first fills cost is estimated to be to be C$5.4 million.

21.2.7.1.7 Contractor commissioning assistance

Contractor commissioning assistance during commissioning has been allowed for in the cost estimate. The costs provided are for construction contractors to assist the commissioning team with routine tasks during commissioning.

The total contractor commissioning assistance cost is estimated to be C$5.0 million.

21.2.7.2 Owner capital costs

Owner costs of $31.5 million have been provided by Northcliff and include the following:

- Owner's project team and expenses
- Owner's project management
- community development
- administration, finance, insurance, legal, and permit fees

- communications system, office equipment, technology, and software

21.2.7.3 Contingency

Contingency accounts for the difference in costs between the estimated and actual costs of materials and equipment. The level of contingency varies depending on the nature of the contract and the client's requirements. Due to uncertainties at the time the capital cost estimate was developed (in terms of the level of engineering definition, basis of the estimate, schedule development, etc.), it is essential that the estimate include a provision to cover the risk from these uncertainties.

For the Ausenco, KP, ISC, NB Power, and Owner's scope, a Probabilistic Contingency analysis was performed which consisted of a contingency ranging workshop taking place internally and evaluated the major cost components in terms of confidence of pricing and quantity basis and provided input ranges for potential underrun/overrun. The ranging inputs were applied as percentages to the base estimate and then ran in a Monte Carlo model using the @Risk program. The @Risk simulation completed for the above noted scopes, determined the contingency to be 17.9% of base accumulative costs at P_{50} confidence level.

The contingency percentages applied for the mining scope were provided by MMTS.

The following contingency percentages were applied:

- Process Plant and Site Infrastructure – 17.9% (P_{50} calculation);
- Mining – 2.4% (Allowance from MMTS);
- Tailings Management – 17.9% (P_{50} calculation);
- Off-site Power Supply – 17.9% (P_{50} calculation);
- Water Treatment Plant & Pipeline to Site – 17.9% (P_{50} calculation);
- Owners – 17.9% (P_{50} calculation).

A summary of the contingency cost is shown Table 21-17.

Table 21-17: Estimate contingency

Party	Contingency %	Initial Contingency (C$M)	Contingency %	Sustaining Contingency (C$M)
Ausenco	17.9%	151.8	10.0%	0.2
Knight Piésold Consulting	17.9%	10.9	10.0%	31.7
Moose Mountain Technical Service	1 Lot	4.8	5.0%	10.2
Integrated Sustainability Consultants	17.9%	6.8	-	-
NB Power	17.9%	20.2	-	-
Northcliff Resources Ltd.	17.9%	5.6	-	-
Total Contingency	**15.1%**	**200.1**	**7.5%**	**42.1**

Note: Values may not sum due to rounding.

21.2.8 Salvage

Salvaging costs have been excluded from the Project.

21.2.9 Sustaining capital

The total sustaining costs for the Sisson project are C$571 million. The values presented in Table 21-18 show total sustaining costs. No salvage credit has been applied as outlined in Section 21.2.8.

An allowance of 10% contingency has been applied for each year of sustaining capital.

Table 21-18: Total Initial & sustaining costs (C$M)

WBS LVL 1	WBS LVL 1 Description	Initial	Y+1	Y+2	Y+3	Y+4	Y+5	Y+6	Y+7	Y+8	Y+9	Y+10	Y+11	Y+12	Y+13	Y+14
1000	Mining	193.3	14.9	0.1	0.1	6.1	0.2	0.5	7.5	29.1	2.5	19.4	7.3	19.6	32.3	7.5
2000	Crushing	210.6	-	-	-	-	-	-		-	-	-	-	-	-	-
3000	Process Plant	248.6	0.9	-	-	-	-	-	0.9	-	-	-	-	-	-	-
5000	Tailings Storage Facility & Water Management	118.3	42.5	12.0	12.5	11.6	7.9	6.9	11.0	14.7	14.5	11.5	6.8	5.9	6.0	5.4
6000	Utilities	71.7	-	-	-	-	-	-	-	-	-	-	-	-	-	-
7000	On-Site Infrastructure	114.2	-	-	-	-	-	-	-	-	-	-	-	-	-	-
8000	Off-Site Infrastructure	125.3	-	-	-	-	-	-	-	-	-	-	-	-	-	-
9000	Indirect Costs	445.6	18.1	5.2	5.3	5.7	2.1	2.1	3.1	5.5	2.9	3.9	2.5	3.4	4.4	2.4
Total		**1,527.7**	**76.3**	**17.4**	**17.9**	**23.4**	**10.3**	**9.5**	**22.4**	**49.3**	**19.9**	**34.9**	**16.6**	**29.0**	**42.7**	**15.3**

WBS LVL 1	WBS LVL 1 Description	Y+15	Y+16	Y+17	Y+18	Y+19	Y+20	Y+21	Y+22	Y+23	Y+24	Y+25	Y+26	Y+27	Total
1000	Mining	12.7	7.2	4.5	0.2	4.2	12.7	25.7	-	0.9	-	-	-	-	408.7
2000	Crushing	-	-	-	-	-	-	-	-	-	-	-	-	-	210.6
3000	Process Plant	-	-	-	-	-	-	-	-	-	-	-	-	-	250.3
5000	Tailings Storage Facility & Water Management	6.2	6.2	6.2	6.8	6.8	54.6	-	-	-	-	-	-	-	374.3
6000	Utilities	-	-	-	-	-	-	-	-	-	-	-	-	-	71.7
7000	On-Site Infrastructure	-	-	-	-	-	-	-	-	-	-	-	-	-	114.2
8000	Off-Site Infrastructure	-	-	-	-	-	-	-	-	-	-	-	-	-	125.3
9000	Indirect Costs	2.8	2.9	2.3	2.0	2.3	7.7	3.2	1.3	1.4	1.3	1.3	1.3	1.3	543.6
Total		**21.7**	**16.3**	**13.0**	**9.0**	**13.4**	**75.0**	**29.0**	**1.3**	**2.2**	**1.3**	**1.3**	**1.3**	**1.3**	**2,098.8**

Note: Values may not sum due to rounding.
Sustaining capital costs for Mining (Area 1000) were provided by MMTS and for TSF & Water Management (Area 5000) were provided by KP. Ausenco has integrated all sustaining capital costs into the overall capital cost estimate.

21.2.10 Closure costs

Closure costs include:

- Direct and indirect costs for physical closure and reclamation of the site including the TSF, open pit, mill facilities and supporting infrastructure. The direct costs include decommissioning, removal, resurfacing, topsoil placement, reseeding, TSF spillway construction, and related indirect costs for closure and reclamation as estimated by KP (Knight Piésold, 2026c) during each year of construction and operations. Indirect costs include contractor mobilization and demobilization, construction management, materials and services, and engineering and specialist input.
- Closure and reclamation costs are estimated to increase from $37.5 million after the completion of construction and commencement of operations to $87.3 million in Year 27, the final year of production.
- Post closure water treatment costs, excluding costs in perpetuity, are estimated to be $36 million over the quiescent period after the completion of operations.
- Post closure water treatment costs in perpetuity, once the open pit fills, are estimated to be $5.9 million per year.
- Post closure water treatment annual sustaining capital costs are estimated at $0.74 million with a 2.0 % allowance, $0.92 million with a 2.5 % allowance, and $ 1.11 million with a 3.0% allowance, assuming that the full-scale system will continue to operate during post closure at a reduced capacity due to lower expected flows.
- Post closure water treatment annual OPEX is estimated for the following flowrates cases:
    - For the minimum modeled flow at the 50% percentile (170 m^3/h), OPEX is estimated to be $1.84 million
    - For the average modeled flow at the 50% percentile (940 m^3/h), OPEX is estimated to be $3.61 million
    - For the maximum modeled flow at the 50% percentile (978 m^3/h), OPEX is estimated to be $3.70 million
    - For the minimum modeled flow at the 95% percentile (473 m^3/h), OPEX is estimated to be $2.54 million
    - For the average modeled flow at the 95% percentile (1,545 m^3/h), OPEX is estimated to be $5.00 million
    - For the maximum modeled flow at the 95% percentile (1,592 m^3/h), OPEX is estimated to be $5.11 million
- For the average flow cases in OPEX, periods with a flowrate of 0 m3/h are excluded from the average flowrate calculation. The water chemistry is based on the closure year with a flowrate closest to the calculated average flowrate.
- Post closure water treatment OPEX estimates reflect reduced reagent consumption and assume no changes in labour, power, maintenance/lab requirements.

21.3 Operating costs

21.3.1 Overview

Operating costs include the ongoing cost of operations related to mining, processing, tailings disposal and general and administration activities. The operating cost estimates were developed in Q2-2026 CAD dollars, unless otherwise specified. Table 21-19 provides a summary of the tonnage weighted average operating costs over the life of mine, on a $/t milled basis.

Table 21-19: Operating cost summary

Description	C$M/a (LOM Average)*	C$/t milled (LOM Average)
Mining	77.8	7.50
Process	131.7	12.72
Quarry	8.7	0.83
G&A	12.0	1.17
Tailings	0.7	0.07
WTP	8.1	0.79
Total	**239**	**23.09**

Note: Values may not sum due to rounding.
* Production-weighted average operating cost over life of mine.

21.3.2 Mine operating costs

The mine operating cost estimates were developed in Q4-2025 CAD dollars and are based on Owner-managed operations, maintenance and technical services departments, and have been built up from first principles. Mine operating cost estimates consist of the following components:

- Equipment operating cost – the activities of drilling, blasting, loading, hauling, mining support, and equipment maintenance. Equipment operating costs are based on the total annual operating hours calculated from the assumed equipment productivities and the cost per service meter unit (SMU) hour to operate the equipment; which includes quoted cost and consumption rates for such inputs as fuel, lubes, explosives, tires, undercarriage, ground engaging tools, drill bits, rods, and strings, machine parts, machine major components, and operating and maintenance labour ratios.
- Salary and hourly personnel – mine department salary staff and general mining labour; maintenance and operator labour costs are included as part of the equipment operating costs, so distributed into the direct mining functions (drill, load, haul, etc.). Salaried labour costs are distributed under GME. During peak production, the mine is expected to employ 69 salaried personnel and 97 hourly personnel. Estimated labour rates for planned hourly and salaried personnel, including burdens, are based off existing Atlantic Canada operations.
- General mine expense (GME) costs – tools, equipment, and services necessary to support mine operations. Includes items such as mine planning software, grade control overheads, geotechnical instrumentation for pit and dumps, technical consultants, maintenance outside services, and office costs and overheads for mine operations, mine maintenance and technical services departments.

Table 21-20: Mine operating cost estimate

Mine Operating Cost Summary	$/t mined	C$/t milled	LOM (C$M)
Drilling	0.38	0.76	209.8
Blasting	0.48	0.95	264.0
Loading	0.27	0.54	149.7
Hauling	1.31	2.62	725.1
Support	0.64	1.28	355.7
Unallocated Labour	0.07	0.15	40.2
Subtotal- Direct Costs	**3.15**	**6.30**	**1,744.4**
Mine Operations GME	0.31	0.63	174.2
Mine Maintenance GME	0.11	0.21	58.9
Mine Engineering GME	0.18	0.35	98.1
Subtotals - GME Costs	**0.60**	**1.20**	**331.1**
Total Mine Operating Costs	**3.75**	**7.50**	**2,075.5**

Note: Values may not sum due to rounding.

21.3.3 Quarry operating costs

Quarry operating costs are derived the same manner as mine operating costs. The mining fleet will be operating in both the pit and the quarry. Costs to drill, blast, load, and haul quarry material to either the crusher for processing into specific material used for the TSF embankment or directly to the TSF embankment. Spread and compacting of material is included under KP scope. Costs for site support and GME are split between the open pit and the quarry based on a ratio of the material moved.

Table 21-21: Quarry operating cost estimate

Mine Operating Cost Summary	$/t mined	$/t milled	Total (C$M)
Drilling	0.38	0.09	25.4
Blasting	0.43	0.10	28.9
Loading	0.27	0.07	18.3
Hauling	1.24	0.30	83.3
Support	0.42	0.10	28.3
Unallocated Labour	0.07	0.02	5.0
Subtotal- Direct Costs	**2.83**	**0.68**	**189.1**
Mine Operations GME	0.32	0.08	21.5
Mine Maintenance GME	0.11	0.03	7.3
Mine Engineering GME	0.18	0.04	12.1
Subtotals - GME Costs	**0.61**	**0.15**	**40.9**
Total Quarry Operating Costs	**3.44**	**0.83**	**230.0**

Note: Values may not sum due to rounding.

21.3.4 Process operating costs

21.3.4.1 Overview

Unless stated otherwise, all costs presented in this section are in Q2-2026 Canadian dollars (CAD, C$). The estimate is based on 30,000 t/d operations consisting of crushing, grinding, molybdenum and tungsten flotation, and concentrate dewatering.

Table 21-22 summarizes the LOM average operating costs for the process plant.

Table 21-22: Overall operating costs for process plant

Description	LOM average cost (C$M/a)	C$/t	% of Total
Power	27	2.59	20
Reagents	62	5.98	47
Consumables	20	1.95	15
Maintenance	8	0.78	6
Process + Maintenance Labour	13	1.22	10
Mobile Equipment	1	0.13	1
Lab Services	1	0.07	1
Total	**132**	**12.72**	**100**

Note: Values may not sum due to rounding.

21.3.4.2 Basis of estimate

The following was used to determine the Project's LOM process operating costs in agreement with the cost definition and estimate methodologies:

- Processing unit operations were developed from first principles and benchmarked against similar or comparable processing plants to ensure their relative accuracy.
- Process equipment and materials will be purchased as new.
- Grinding media consumption rates have been estimated based on the measured ore characteristics and regrind mill media consumption rates were estimated from benchmark data and vendor supplied information.
- Reagent consumption rates have been estimated based on metallurgical testwork and standard operating practices on a nominal basis.
- Process mobile equipment costs include fuel, maintenance, and the lease price for the mobile equipment.
- Concentrate transportation, treatment, refining, and other related costs are not included in this estimate

21.3.4.3 Electrical power

Power costs of the process plant and ancillary facilities were calculated from the average power utilization in the electrical load list. A power cost of $0.075/kWh was used as provided by New Brunswick (NB) Power.

Annual power costs for the processing plant are estimated at C$27 M/a or C$2.59/t of mill feed.

21.3.4.4 Reagents, wear items and grinding media

Reagent consumption rates are derived from the testwork outlined in Section 13. Reagent pricing was derived from recent vendor quotations from international and local suppliers, with freight included for delivery to the project site.

Mill media consumption is based on the abrasion properties of the mill feed and the mill throughput. Consumption rates for maintenance consumables such as liners are based on benchmarks for replacement rates for each equipment type. Maintenance consumable unit pricing was obtained from vendor quotations and Ausenco's internal database of benchmark costs.

Annual reagent costs are expected to be C$62 million per year, or C$5.98/t of mill feed. Annual consumable costs are expected to be C$20 million per year, or C$1.95 of mill feed.

21.3.4.5 Maintenance parts and supplies

The process plant annual maintenance cost was derived from the supplied mechanical equipment cost for each phase based on the mechanical equipment list using an average factor of 3.8%, which varies depending on the process area. The factors were determined from benchmark maintenance costs derived from several recent Canadian projects.

Annual maintenance activities are anticipated to cost C$8 million per year, or C$0.78/t of mill feed.

21.3.4.6 Labour

Staffing requirements were estimated by benchmarking against similar projects and include management, metallurgy, operations, maintenance, and assay laboratory operation. The total operational labour averages 128 employees. Labour rates were estimated using recent data from an operating mine in Atlantic Canada and burden calculations to consider payroll deductions for benefits and government programs.

Labour costs for the process plant are estimated at C$13 million per year or C$1.22/t of mill feed.

21.3.4.7 Mobile equipment

Vehicle costs are based on a scheduled number of light vehicles and mobile equipment. The costs include fuel, maintenance, spares and tires, annual registration and insurance fees, and equipment leasing costs.

For plant operations and maintenance, the estimated annual cost of mobile equipment is C$2 million per year during the initial six years of leasing, decreasing to C$1 million per year following completion of the lease term. This averages to C$0.13/t of mill feed over the life of the Project.

21.3.4.8 On-site laboratory services

The operating cost estimate for the laboratory was developed based on a detailed sampling plan for operations. Unit costs for sample preparation and assaying were applied to each sample based on recent service provided quotations.

Annual lab costs are estimated at C$1 million per year or C$0.07/t of mill feed. These values do not include mine grade control sampling, which is captured elsewhere.

21.3.4.9 Water

Operating costs for water services include site water management equipment, potable and sewage treatment plants, and the site effluent water treatment plant. The cost estimate incorporates power consumption, reagent usage, and routine maintenance requirements for these facilities.

Annual operating cost for water services are estimated at C$9 million per year or C$0.91/t of mill feed.

21.3.5 Infrastructure operating costs

21.3.5.1 Tailings and water management

The TSF and water management operating costs summarized in Table 21-23 include electrical power, snow clearing, and scheduled replacement of mechanical equipment associated with waste and water management during operations.

Table 21-23: Tailings and water management– annual operating cost summary

Operating Cost Summary	$/t milled	Total (C$M)
Tailings Delivery	0.016	4.60
Water Reclaim	0.025	7.20
Seepage and Sediment Control	0.005	1.32
Open Pit Dewatering	0.012	3.43
Snow Clearing	0.007	1.91
Total TSF Operating Costs	**0.07**	**18.5**

Note: Values may not sum due to rounding.

Operating costs for the tailings mechanical systems include electrical power, maintenance, and replacement. Annual renewal/replacement allowances: 10%/a for HDPE tailings pipe, 2.5%/a for rubber-lined steel tailings pipe, 5%/a for HDPE water pipe.

21.3.5.2 Water Treatment Plant (WTP)

Annual operating costs for the Sisson WTP were estimated by ISC using a first-principles approach. These values reflect Q2 2026 Canadian dollars and exclude contingency, escalation, and Owner's indirect costs.

The estimates consider full design throughput, year-round operation, and include chemical consumption, labour, utilities, maintenance, and insurance. Unit pricing was developed from vendor quotes, industry benchmarks, and internal cost libraries.

The Sisson WTP is expected to operate during the early development period and remain active through operations and closure. As flow rates and water chemistry are anticipated to vary significantly throughout these project phases, operating costs are also expected to vary, primarily due to chemical consumption. The estimated average annual operating cost for the Sisson WTP is approximately $8.25 million, with low and high annual operating costs of approximately $2.24 M and $11.65 million, respectively. Annual operating cost are generally lower during Years 1 to 5 (below 70 per cent of average), remain within ±30 per cent of the average annual cost during Years 6 to 19, and are generally higher (above 130 per cent of average) during Years 20 to 27 before returning to average levels in Year 28. Seven full-time employees are estimated and included in the WTP operating cost estimate.

Table 21-24: Sisson WTP – annual operating cost summary

Cost Category	Low (Year 1 to 5)		Average (Year 6 to 19, 28)		High (Year 20-27)	
	Annual Cost (C$M)	% of Total	Annual Cost ($M)	% of Total	Annual Cost (C$M)	% of Total
Chemical Consumption	0.79	3527	6.80	82	10.19	88
Operating Labour	0.88	390	0.88	11	0.88	8
Utilities	0.32	141	0.32	4	0.32	3
Maintenance and Repairs, Operating Supplies & Laboratory Charges	0.26	129	0.26	3	0.26	2
Total	**2.2495**	**100**	**8.25**	**100**	**11.65**	**100**

Note: Values may not sum due to rounding.

21.3.6 General and administrative operating costs

General and administrative (G&A) costs are expenses not directly related to production and include expenses not included in mining, processing, external refining, and transportation costs. These costs were developed using Ausenco's in-house data on Canadian projects and existing operations, and specific inputs from Northcliff. The G&A costs are divided into the following areas:

- G&A maintenance, including access road maintenance
- G&A personnel, including 25 salaried and 22 hourly personnel for a total of 47 personnel
- G&A mobile equipment
- human resources, including training, recruiting, First Nations engagement and community relations
- site administration, maintenance, and security, including subscriptions, memberships, advertisement, office supplies and garbage disposal
- health and safety, including personal protective equipment, mine rescue training and equipment, hospital service cost, and first aid
- environmental, including water sampling, air monitoring, acid rock drainage monitoring, and TSF monitoring costs
- IT and telecommunications, including hardware and support services
- contract services, including insurance, sanitation and cleaning, license fees, and legal fees.

G&A costs are detailed in Table 21-25.

The peak operational workforce is estimated at 352 employees, including personnel associated with general and administrative (G&A), mining, process plant, and water treatment plant (WTP) functions.

Project personnel will consist of a locally sourced labour force. The workers are assumed to reside in, and commute from, nearby communities. The Sisson workforce will primarily travel to site from neighboring communities to assembly points and then by company-provided buses to site. No onsite housing is required.

Table 21-25: G&A cost summary

Department	LOM average (C$M/a)
G&A maintenance	0.6
G&A personnel	4.9
G&A mobile equipment	0.2
Human resources	0.3
Contract services	2.6
Health and safety	0.4
Environmental	0.9
IT & telecommunications	0.6
Administration, sanitation and cleaning	0.3
G&A Power	1.4
Total	**12.2**

Note: Values may not sum due to rounding.

22 Economic analysis

22.1 Introduction

The following section relies on financial modelling conducted by Northcliff and audited by Ausenco. It has been reviewed by the QP as acceptable for use.

22.2 Forward-looking information

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

- mineral reserve and resource estimates;
- assumed commodity prices and exchange rates;
- the proposed mine production plan;
- projected mining and process recovery rates;
- assumptions as to mining dilution and ability to mine in areas previously exploited using mining methods as envisaged;
- the timing and amount of estimated future production;
- sustaining costs and proposed operating costs;
- assumptions as to closure costs and closure requirements; and
- assumptions as to environmental, permitting, and social risks.

Additional risks to the forward-looking information include:

- changes to costs of production from what is assumed;
- unrecognized environmental risks;
- unanticipated reclamation expenses;
- unexpected variations in quantity of mineralized material, grade, or recovery rates;
- accidents, labour disputes and other risks of the mining industry;
- geotechnical or hydrogeological considerations during mining being different from what was assumed;
- failure of mining methods to operate as anticipated;
- failure of plant, equipment, or processes to operate as anticipated;
- changes to assumptions as to the availability of electrical power, and the power rates used in the operating cost estimates and financial analysis;
- ability to maintain the social licence to operate;
- changes to interest rates; and
- changes to tax rates.

22.3 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, head grade, recovery, total operating cost, exchange rate, sustaining capital costs and initial capital costs.

The capital and operating cost estimates developed specifically for this project are presented in Section 21 of this Technical Report in Q2 2026 Canadian dollars. The economic analysis has been run on a constant dollar basis with no inflation.

22.4 Financial model parameters

22.4.1 Assumptions

The economic analysis was performed using the commodity prices detailed in Table 19-1 and Table 19-3 from Argus Media. Commodity prices can be volatile, and there is the potential for deviation from the forecast.

The economic analysis used the following assumptions:

- Construction period of two years.
- Total mine operations life of 27 years.
- Cost estimates in constant Q2 2026 Canadian dollars with no inflation or escalation factors considered.
- Results based on 100% ownership.
- Capital cost funded with 100% equity (no financing cost assumed except where stated in mining equipment).
- All cash flows discounted to start of construction period using middle of period discounting convention.
- All metal products are sold in the same year they are produced.
- Project revenue is derived from the sale of tungsten concentrate and molybdenum concentrate.
- No contractual arrangements for refining currently exist.

22.4.2 Taxes

The Project has been evaluated on a post-tax basis to provide an approximate value of the potential economics. The tax model was compiled by Northcliff and reviewed by Mark Peters, a tax expert working for HDI. The calculations are based on the tax regime as of the date of the 2013 FS Technical Report and updated for 2026 regulations. At the effective date of this Technical Report, the Project is assumed to be subject to the Income Tax and Mineral Tax applicable in New Brunswick, Canada, resulting in estimated total payments of $14,499 million over the life of mine.

22.4.3 Working capital

An estimate of working capital has been incorporated into the economic analysis based on the following assumptions.

Table 22-1: Estimate of working capital

Item	Units	Value
Accounts Receivable - Tungsten	month(s)	1
Tungsten Inventory in Transit	month(s)	1
Accounts Receivable - Moly	month(s)	1
Molybdenum Inventory in Transit	month(s)	1
Spare Parts Inventory	C$ M	15
Accounts Payable	month(s)	1
Consignment Inventory %	%	25

22.4.4 Reclamation bond

A reclamation bond schedule was calculated using the following parameters.

Table 22-2: Reclamation bond inputs

Item	Units	Value
Reclamation Bond Interest Rate	%	4.05
"BBB" Letter of Credit Costs	%	1
Production period, years	year(s)	27
Bond contribution period, years	year(s)	29
Reclamation bond balance at start of construction	C$ M	–
Post-closure water treatment costs, in perpetuity	C$ M/a	5.9
Start of post-closure water treatment in perpetuity	Years from start of production	39
Quiescent period, end of production	Years to fill final pit	12

The schedule estimates a total LOM cost of C$147 million.

22.4.5 Offsite costs

The model uses the following parameters which are detailed in Section 19.

Table 22-3: Offsite cost inputs

Item	Units	Value
Tungsten Freight	C$/t conc	264
Tungsten losses, insurance and representation	%	0.15
Tungsten concentrate moisture content	%	0.5
Molybdenum total offsite charges (incl. freight, insurance etc.)	% Mo value in concentrate	12
Molybdenum concentrate moisture content	%	5

22.5 Economic analysis

The economic analysis was performed, assuming an 8% discount rate. On a pre-tax basis the net present value (NPV) discounted at 8% is C$12,290 million; the internal rate of return (IRR) is 68.3%, and payback period is 1.3 years. On a post-tax basis, the NPV discounted at 8% is C$6,915 million; the IRR is 49.8%, and the payback period is 1.6 years. The analysis was done on an annual cashflow basis; the cashflow output is shown graphically in Figure 22-1 and summarized in Table 22-4.

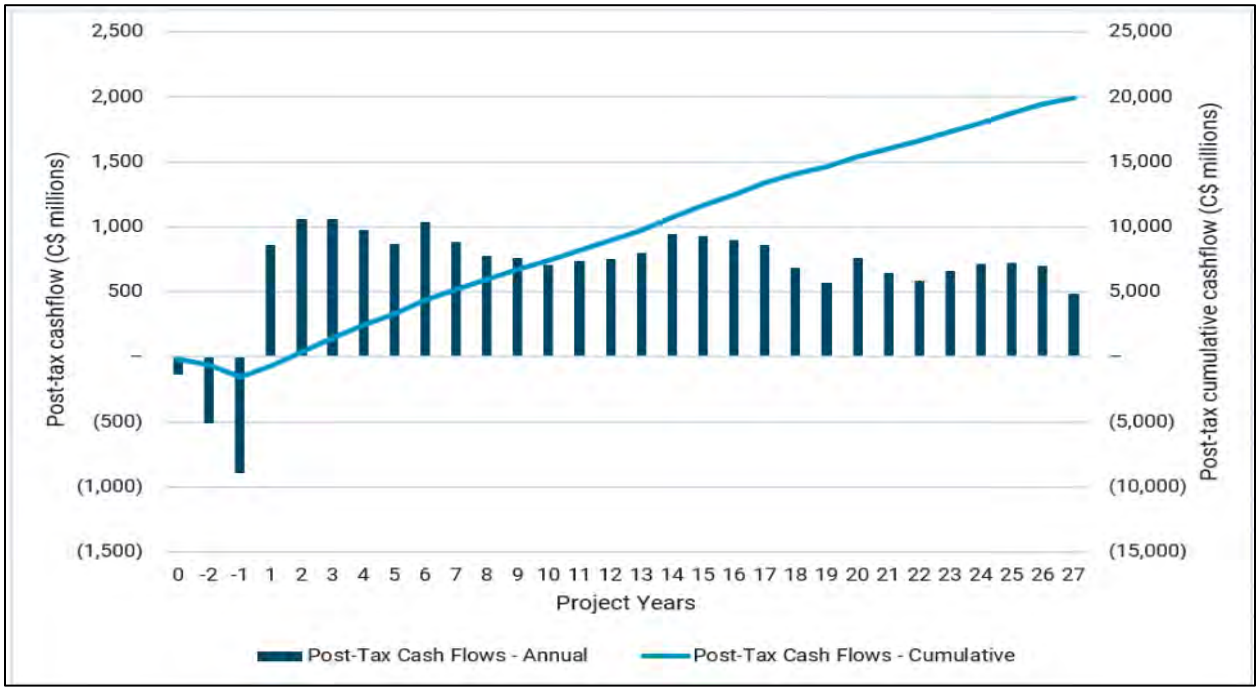
Table 22-4: Economic analysis summary

Economic Summary	Units	LOM	Annual Average	Years 1 to 5 Average
Metal Price - Tungsten (Concentrate)	US$/MTU WO_3*	1,821		1,634
Metal Price - Molybdenum	US$/lb	28.46		28.51
Foreign Exchange Rate	C$:US$	1.35		1.35
Production				
Mining Volume	Mt	638	22	18.5
Milling Volume	Mt	277	10.3	10.2
Strip Ratio	waste:ore	0.97		0.47
Tungsten Grade	%	0.0747%		0.096%
Molybdenum Grade	%	0.0227%		0.0217%
Tungsten (WO_3) Recovery	%	78.0%		78.0%
Molybdenum Recovery	%	82.0%		82.0%
Tungsten (WO_3) Production	MMTU WO_3*	16.1	0.598	0.767
Molybdenum Production	Mlbs	113.5	4.2	4
Gross Revenue	$ M	43,699	1,618	1,845
Operating Costs				
Total Operating Costs	$ M	6,391	239	221
Operating Costs - Mining	$/t milled	7.50		6.52
Operating Costs - Milling	$/t milled	12.72		12.84

Economic Summary	Units	LOM	Annual Average	Years 1 to 5 Average
Operating Costs - Waste Management	$/t milled	1.69		1.04
Operating Costs - G&A	$/t milled	1.17		1.21
Total operating costs to concentrate	$/t milled	23.09		21.62
Operating Costs - Less: Molybdenum by-product credit	$/t milled	(13.85)		(13.26)
Total operating cash costs to concentrate, net of by-product credits[(1)]	$/t milled	9.23		8.35
Concentrate All-in Sustaining Costs, net of by-product credits[(2)]	$/t milled	14.24		14.21
Total operating costs to concentrate	$/MTU WO_3*	396		289
Operating Costs - Less: Molybdenum by-product credit	$/MTU WO_3*	(238)		(177)
Total operating cash costs to concentrate, net of by-product credits[(1)]	$/MTU WO_3*	158		112
Concentrate All-in Sustaining Costs, net of by-product credits[(2)]	$/MTU WO_3*	244		190
Total operating costs to concentrate	US$/MTU WO_3*	293		215
Operating Costs - Less: Molybdenum by-product credit	US$/MTU WO_3*	(176)		(130)
Total operating cash costs to concentrate, net of by-product credits[(1)]	US$/MTU WO_3*	117		85
Concentrate All-in Sustaining Costs, net of by-product credits[(2)]	US$/MTU WO_3*	181		143
Capital and Other Costs				
Initial Capital Costs	C$M	1,528	na	na
Sustaining Capital Costs	C$M	571	21	29
Reclamation Funding	C$M	147	5	6
Taxes and Royalties	C$M	14,499	537	545
Pre Tax Economics				
NPV 8, pre-tax (calculated at mid-year)	C$M	12,290	-	-
IRR, pre-tax (calculated at mid-year)	%	68.3%	-	-
Payback, pre-tax (years of production)	year(s)	1.3	-	-
Post Tax Economics				
Post-Tax Cash Flows	C$M	19,896	686	965
NPV 8, post-tax (calculated at mid-year)	C$M	6,915	-	-
IRR, post-tax (calculated at mid-year)	%	49.8%	-	-
Payback, post-tax (years of production)	year(s)	1.6	-	-

Notes:
* MTU = metric tonne unit = 1% of a metric tonne = 10kg
** reclamation funding over construction and operations period of 29 years combined.
(1) Metal prices for the economic analysis were applied based on Argus' independent market study forecasts on an annual basis until 2040 and thereafter the 2040 forecast prices were applied for the remainder of the project life.
(2) Operating cash costs includes mine, mill, waste management and G&A costs less by-product credits .
(3) All-in sustaining cost includes operating cash cost, offsite costs, reclamation costs and sustaining capital costs .
Values may not sum due to rounding.
Source: Northcliff, 2026

Figure 22-1: LOM post-tax cumulative free cash flow



Notes: Year 0 denotes long lead items requiring payment before model start date.
Source: Northcliff, 2026

Table 22-5: Cashflow statement on an annualized basis

Discounted Cash Flow Summary	Units	Total LOM	-2	-1	1	2	3	4	5	6	7	8	9	10	11	12	13	14	15	16	17	18	19	20	21	22	23	24	25	26	27
expressed in Canadian dollars unless otherwise stated																															
INPUT ASSUMPTIONS																															
Mining Volume	Mmt	638	2.8	14.6	20.8	19.9	16.7	18.2	16.7	17.4	21.1	29.8	28.0	30.1	28.4	27.0	26.4	25.7	27.3	27.9	24.4	20.0	15.4	31.0	34.3	36.0	21.4	18.1	17.7	13.9	6.3
Milling Volume	Mmt	277	-	-	9.2	10.5	10.5	10.5	10.5	10.5	10.5	10.5	10.5	10.5	10.5	10.5	10.5	10.5	10.5	10.5	10.5	10.5	10.5	10.5	10.5	10.5	10.5	10.5	10.5	10.5	5.1
Strip Ratio	waste:ore	0.97	na	3.73	0.69	0.29	0.12	0.75	0.73	0.12	0.94	1.52	1.42	1.61	1.45	1.32	1.26	1.20	1.35	1.41	1.07	0.65	0.22	1.63	1.94	2.11	0.72	0.41	0.36	0.33	0.24
Tungsten Grade	%	0.075	-	-	0.101	0.099	0.106	0.094	0.081	0.097	0.077	0.076	0.069	0.067	0.069	0.069	0.076	0.086	0.083	0.082	0.078	0.054	0.046	0.084	0.054	0.055	0.062	0.065	0.065	0.062	0.051
Tungsten Recovery	% to concentrate	78.0	-	-	78.0	78.0	78.0	78.0	78.0	78.0	78.0	78.0	78.0	78.0	78.0	78.0	78.0	78.0	78.0	78.0	78.0	78.0	78.0	78.0	78.0	78.0	78.0	78.0	78.0	78.0	78.0
Metal Price - Tungsten (Concentrate)	US$/MTU ($WO_3$)		1,385	1,475	1,520	1,580	1,660	1,690	1,720	1,765	1,775	1,785	1,840	1,865	1,880	1,880	1,880	1,880	1,880	1,880	1,880	1,880	1,880	1,880	1,880	1,880	1,880	1,880	1,880	1,880	1,880
Metal Price - Molybdenum	US$/lb		33.20	31.57	29.57	28.94	28.12	27.81	28.12	28.30	28.49	28.53	28.58	28.53	28.44	28.44	28.44	28.44	28.44	28.44	28.44	28.44	28.44	28.44	28.44	28.44	28.44	28.44	28.44	28.44	28.44
Foreign Exchange Rate	C$/US$1		1.35	1.35	1.35	1.35	1.35	1.35	1.35	1.35	1.35	1.35	1.35	1.35	1.35	1.35	1.35	1.35	1.35	1.35	1.35	1.35	1.35	1.35	1.35	1.35	1.35	1.35	1.35	1.35	1.35
CASH FLOWS																															
Gross Revenue	**C$M**	**43,699**	**-**	**-**	**1,613**	**1,921**	**2,134**	**1,884**	**1,672**	**2,190**	**1,664**	**1,667**	**1,553**	**1,517**	**1,579**	**1,614**	**1,749**	**1,971**	**1,885**	**1,819**	**1,736**	**1,331**	**1,239**	**1,877**	**1,259**	**1,308**	**1,462**	**1,516**	**1,511**	**1,438**	**590**
Tungsten	C$M	39,341	-	-	1,490	1,729	1,942	1,751	1,541	1,899	1,510	1,493	1,409	1,382	1,435	1,443	1,574	1,785	1,731	1,713	1,618	1,115	964	1,754	1,125	1,133	1,290	1,347	1,352	1,295	519
Molybdenum	C$M	4,358	-	-	123	192	192	133	131	291	153	174	144	135	144	171	175	186	154	106	117	217	275	123	134	175	172	169	159	143	71
Realization Charges	**C$M**	**668**	**-**	**-**	**21**	**30**	**31**	**23**	**22**	**42**	**24**	**26**	**22**	**21**	**22**	**26**	**27**	**29**	**25**	**19**	**20**	**30**	**36**	**21**	**20**	**25**	**25**	**25**	**24**	**22**	**10**
Tungsten Offsite Costs	C$M	145	-	-	6	7	8	7	6	7	6	6	5	5	5	5	6	6	6	6	6	4	3	6	4	4	5	5	5	5	2
Molybdenum TC/RC + Other Offsite Costs	C$M	523	-	-	15	23	23	16	16	35	18	21	17	16	17	21	21	22	18	13	14	26	33	15	16	21	21	20	19	17	8
Net Smelter Return	**C$M**	**43,032**	**-**	**-**	**1,592**	**1,891**	**2,103**	**1,862**	**1,651**	**2,148**	**1,640**	**1,641**	**1,531**	**1,496**	**1,556**	**1,589**	**1,722**	**1,942**	**1,860**	**1,800**	**1,716**	**1,301**	**1,202**	**1,856**	**1,239**	**1,283**	**1,437**	**1,491**	**1,487**	**1,416**	**579**
Tungsten	C$M	39,197	-	-	1,484	1,723	1,934	1,744	1,535	1,892	1,505	1,487	1,404	1,377	1,430	1,438	1,569	1,779	1,725	1,707	1,612	1,111	960	1,748	1,121	1,129	1,286	1,342	1,347	1,290	517
Molybdenum	C$M	3,835	-	-	108	169	169	117	115	256	135	153	127	119	127	151	154	163	136	93	103	191	242	108	118	154	151	149	140	126	62
Operating Costs	**C$M**	**6,391**	**-**	**-**	**211**	**224**	**226**	**224**	**221**	**227**	**242**	**248**	**247**	**256**	**264**	**249**	**254**	**244**	**264**	**254**	**242**	**234**	**239**	**252**	**257**	**258**	**247**	**229**	**231**	**222**	**123**
Mining	C$M	2,076	-	-	68	67	68	67	64	64	78	87	84	94	101	87	91	82	101	91	80	71	74	85	90	92	77	60	61	64	30
Milling	C$M	3,522	-	-	121	134	134	134	134	134	133	133	133	133	133	133	133	133	133	133	133	133	133	133	133	133	133	133	133	133	68
APT Plant	C$M	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
Waste Management	C$M	468	-	-	10	10	12	11	11	17	19	17	17	17	18	17	18	17	19	17	17	18	21	22	22	22	25	24	25	12	12
G&A	C$M	324	-	-	13	12	12	12	12	12	12	12	12	12	12	12	12	12	12	12	12	12	12	12	12	12	12	12	12	12	12
Operating Cash Flow	**C$M**	**36,641**	**-**	**-**	**1,381**	**1,667**	**1,877**	**1,638**	**1,429**	**1,921**	**1,397**	**1,393**	**1,284**	**1,239**	**1,292**	**1,340**	**1,469**	**1,698**	**1,596**	**1,547**	**1,474**	**1,067**	**963**	**1,604**	**982**	**1,025**	**1,190**	**1,262**	**1,256**	**1,194**	**456**

Discounted Cash Flow Summary	Units	Total LOM	-2	-1	1	2	3	4	5	6	7	8	9	10	11	12	13	14	15	16	17	18	19	20	21	22	23	24	25	26	27
expressed in Canadian dollars unless otherwise stated																															
Initial Capital Costs	**C$M**	**1,528**	**637**	**891**	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
Mine	C$M	193	77	116	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
Crushing	C$M	211	84	126	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
Process Plant	C$M	249	99	149	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
APT Plant	C$M	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
TSF & Water Management	C$M	118	47	71	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
Utilities	C$M	72	29	43	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
On-site Infrastructure	C$M	114	46	69	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
Off-site Infrastructure	C$M	125	50	75	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
Indirects	C$M	206	83	124	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
Owner's Cost	C$M	37	37	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
Freight	C$M	2	1	1	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
Contingency	C$M	200	83	117	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
Sustaining Capital Costs	**C$M**	**571**	**-**	**-**	**76**	**17**	**18**	**23**	**10**	**10**	**22**	**49**	**20**	**35**	**17**	**29**	**43**	**15**	**22**	**16**	**13**	**9**	**13**	**75**	**29**	**1**	**2**	**1**	**1**	**1**	**1**
Mine	C$M	215	-	-	15	0	0	6	0	0	7	29	2	19	7	20	32	7	13	7	5	0	4	13	26	-	1	-	-	-	-
Crushing	C$M	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
Process Plant	C$M	2	-	-	1	-	-	-	-	-	1	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
APT Plant	C$M	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
TSF & Water Management	C$M	256	-	-	42	12	13	12	8	7	11	15	14	11	7	6	6	5	6	6	6	7	7	55	-	-	-	-	-	-	-
Utilities	C$M	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
On-site Infrastructure	C$M	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
Off-site Infrastructure	C$M	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
Indirects	C$M	98	-	-	18	5	5	6	2	2	3	5	3	4	2	3	4	2	3	3	2	2	2	8	3	1	1	1	1	1	1
Owner's Cost	C$M	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
Freight	C$M	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
Contingency	C$M	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
Working Capital	C$M	-	-	-	259	49	35	(40)	(35)	82	(86)	(0)	(18)	(7)	9	7	22	37	(15)	(9)	(13)	(68)	(17)	108	(103)	7	27	10	(1)	(11)	(229)
Reclamation Funding	C$M	147	5	6	6	6	6	6	6	6	6	6	6	5	5	5	5	5	5	5	5	5	5	5	4	4	4	4	4	4	4
Pre-Tax Cash Flows	**C$M**	**34,527**	**(642)**	**(897)**	**1,040**	**1,595**	**1,818**	**1,649**	**1,448**	**1,824**	**1,455**	**1,338**	**1,277**	**1,206**	**1,261**	**1,299**	**1,399**	**1,640**	**1,584**	**1,535**	**1,469**	**1,122**	**962**	**1,417**	**1,052**	**1,012**	**1,157**	**1,246**	**1,251**	**1,200**	**680**
Income Taxes and Royalties	C$M	14,499	-	-	179	536	760	670	582	786	574	565	519	500	522	543	596	694	656	637	607	439	393	654	408	424	495	528	528	504	198
Post-Tax Cash Flows	**C$M**	**19,896**	**(642)**	**(897)**	**860**	**1,060**	**1,058**	**979**	**866**	**1,038**	**882**	**773**	**758**	**706**	**739**	**756**	**803**	**946**	**928**	**898**	**862**	**682**	**568**	**762**	**644**	**588**	**662**	**718**	**723**	**696**	**482**

Source: Northcliff, 2026

22.6 Sensitivity analysis

A sensitivity analysis was conducted on the base case pre-tax and post-tax NPV and internal rate of return of the Project, using the following variables: commodity prices, discount rates, operating costs, initial capital costs, sustaining capital costs, metal recovery, head grade and exchange rate. Table 22-6 and Table 22-7 present the Project's sensitivities to the variables on ±25% basis.

As shown in Figure 22-2 and Figure 22-3, the sensitivity analysis revealed that the Project is most sensitive to changes in foreign exchange rate, commodity price, head grade and recovery; it is less sensitive to total operating cost, sustaining capital cost, and initial capital cost.

Table 22-6: Pre-tax sensitivity summary

NPV	75%	80%	85%	90%	95%	100%	105%	110%	115%	120%	125%
Grade	8,214	9,030	9,845	10,660	11,475	12,290	13,105	13,921	14,736	15,551	16,366
Recovery	8,214	9,030	9,845	10,660	11,475	12,290	13,105	13,921	14,736	15,551	16,324
Initial CAPEX	12,643	12,573	12,502	12,431	12,361	12,290	12,220	12,149	12,079	12,008	11,937
Sustaining CAPEX	12,352	12,340	12,328	12,315	12,303	12,290	12,278	12,265	12,253	12,241	12,228
On-site OPEX	12,861	12,747	12,633	12,519	12,404	12,290	12,176	12,062	11,948	11,834	11,720
FX Rate	8,206	9,023	9,840	10,657	11,473	12,290	13,107	13,924	14,741	15,558	16,375
Metal Prices	8,206	9,023	9,840	10,657	11,473	12,290	13,107	13,924	14,741	15,558	16,375

IRR	75%	80%	85%	90%	95%	100%	105%	110%	115%	120%	125%
Grade	52%	56%	59%	62%	65%	68%	71%	74%	77%	80%	83%
Recovery	52%	56%	59%	62%	65%	68%	71%	74%	77%	80%	83%
Initial CAPEX	84%	80%	77%	74%	71%	68%	66%	64%	62%	60%	58%
Sustaining CAPEX	69%	69%	69%	68%	68%	68%	68%	68%	68%	68%	68%
On-site OPEX	70%	70%	69%	69%	69%	68%	68%	68%	67%	67%	66%
FX Rate	52%	56%	59%	62%	65%	68%	71%	74%	77%	80%	83%
Metal Prices	52%	56%	59%	62%	65%	68%	71%	74%	77%	80%	83%

	Pre-Tax NPV Sensitivity to Discount Rate					
	Commodity Price					
Discount Rate		75.0%	90.0%	100.0%	110.0%	125.0%
	3.0%	15,269	19,561	22,423	25,284	29,576
	5.0%	11,742	15,115	17,363	19,612	22,985
	8.0%	8,206	10,657	12,290	13,924	16,375
	10.0%	6,593	8,623	9,975	11,328	13,358
	12.0%	5,368	7,077	8,216	9,355	11,064

	Pre-Tax IRR Sensitivity to Discount Rate					
	Commodity Price					
Discount Rate		75.0%	90.0%	100.0%	110.0%	125.0%
	3.0%	52%	62%	68%	74%	83%
	5.0%	52%	62%	68%	74%	83%
	8.0%	52%	62%	68%	74%	83%
	10.0%	52%	62%	68%	74%	83%
	12.0%	52%	62%	68%	74%	83%

Source: Northcliff, 2026

Table 22-7: Post-tax sensitivity summary

NPV	75%	80%	85%	90%	95%	100%	105%	110%	115%	120%	125%
Grade	4,585	5,051	5,517	5,982	6,448	6,915	7,381	7,847	8,313	8,779	9,245
Recovery	4,585	5,051	5,517	5,982	6,448	6,915	7,381	7,847	8,313	8,779	9,221
Initial Capex	7,134	7,091	7,047	7,003	6,959	6,915	6,871	6,826	6,782	6,738	6,694
Sustaining Capex	6,955	6,947	6,939	6,931	6,923	6,915	6,907	6,899	6,891	6,883	6,875
On-site Opex	7,251	7,184	7,117	7,049	6,982	6,915	6,847	6,780	6,713	6,645	6,578
FX Rate	4,580	5,047	5,514	5,981	6,447	6,915	7,382	7,849	8,316	8,783	9,250
Metal Prices	4,580	5,047	5,514	5,981	6,447	6,915	7,382	7,849	8,316	8,783	9,250

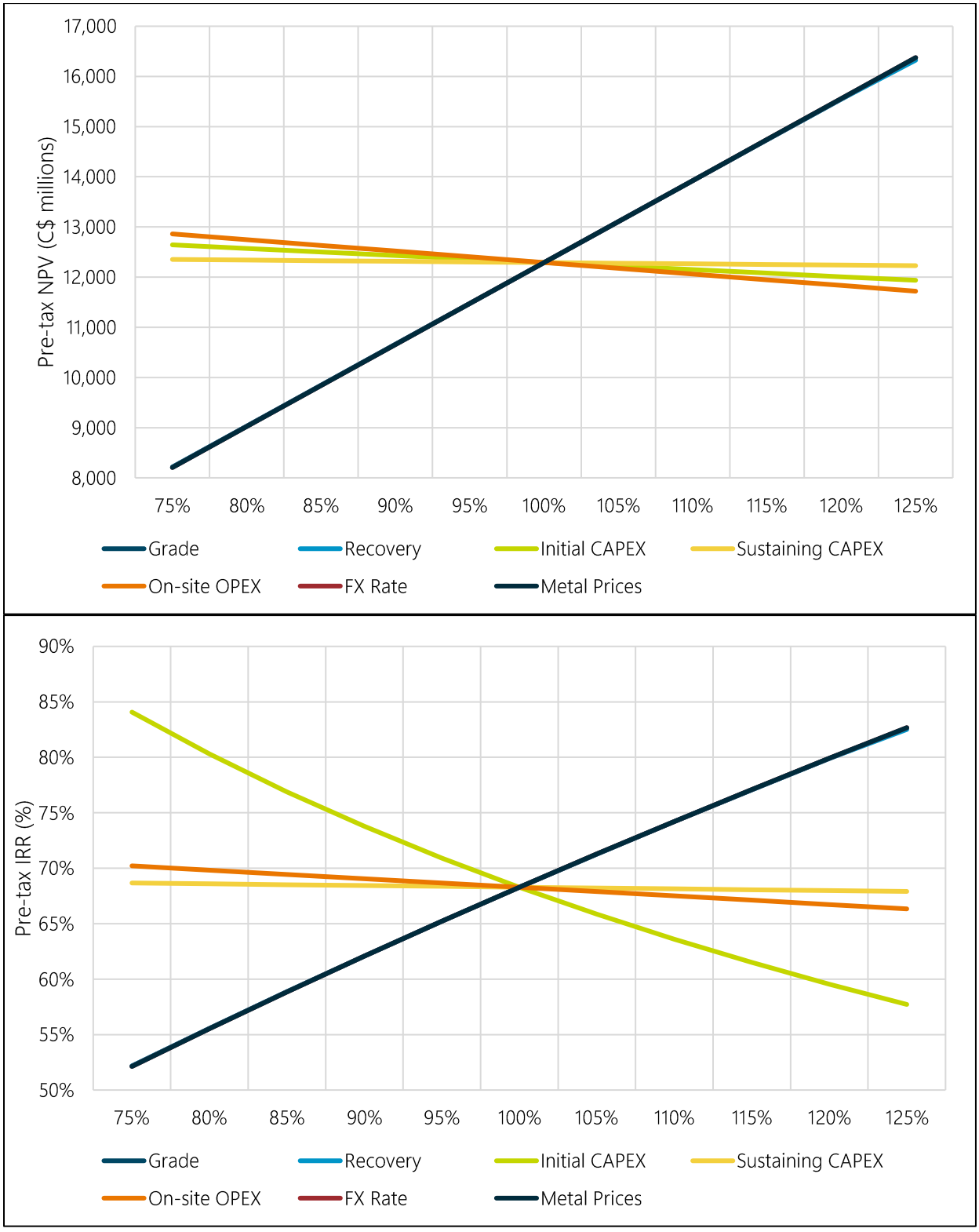
IRR	75%	80%	85%	90%	95%	100%	105%	110%	115%	120%	125%
Grade	38%	41%	43%	45%	48%	50%	52%	54%	56%	58%	60%
Recovery	38%	41%	43%	45%	48%	50%	52%	54%	56%	58%	60%
Initial Capex	61%	59%	56%	54%	52%	50%	48%	46%	45%	44%	42%
Sustaining Capex	50%	50%	50%	50%	50%	50%	50%	50%	50%	50%	50%
On-site Opex	51%	51%	51%	50%	50%	50%	50%	49%	49%	49%	48%
FX Rate	38%	41%	43%	45%	48%	50%	52%	54%	56%	58%	60%
Metal Prices	38%	41%	43%	45%	48%	50%	52%	54%	56%	58%	60%

	Post-Tax NPV Sensitivity to Discount Rate					
	Commodity Price					
Discount Rate		75.0%	90.0%	100.0%	110.0%	125.0%
	3.0%	8,768	11,226	12,864	14,503	16,961
	5.0%	6,678	8,607	9,894	11,180	13,110
	8.0%	4,580	5,981	6,915	7,849	9,250
	10.0%	3,622	4,781	5,555	6,328	7,488
	12.0%	2,893	3,869	4,521	5,172	6,149

	Post-Tax IRR Sensitivity to Discount Rate					
	Commodity Price					
Discount Rate		75.0%	90.0%	100.0%	110.0%	125.0%
	3.0%	38%	45%	50%	54%	60%
	5.0%	38%	45%	50%	54%	60%
	8.0%	38%	45%	50%	54%	60%
	10.0%	38%	45%	50%	54%	60%
	12.0%	38%	45%	50%	54%	60%

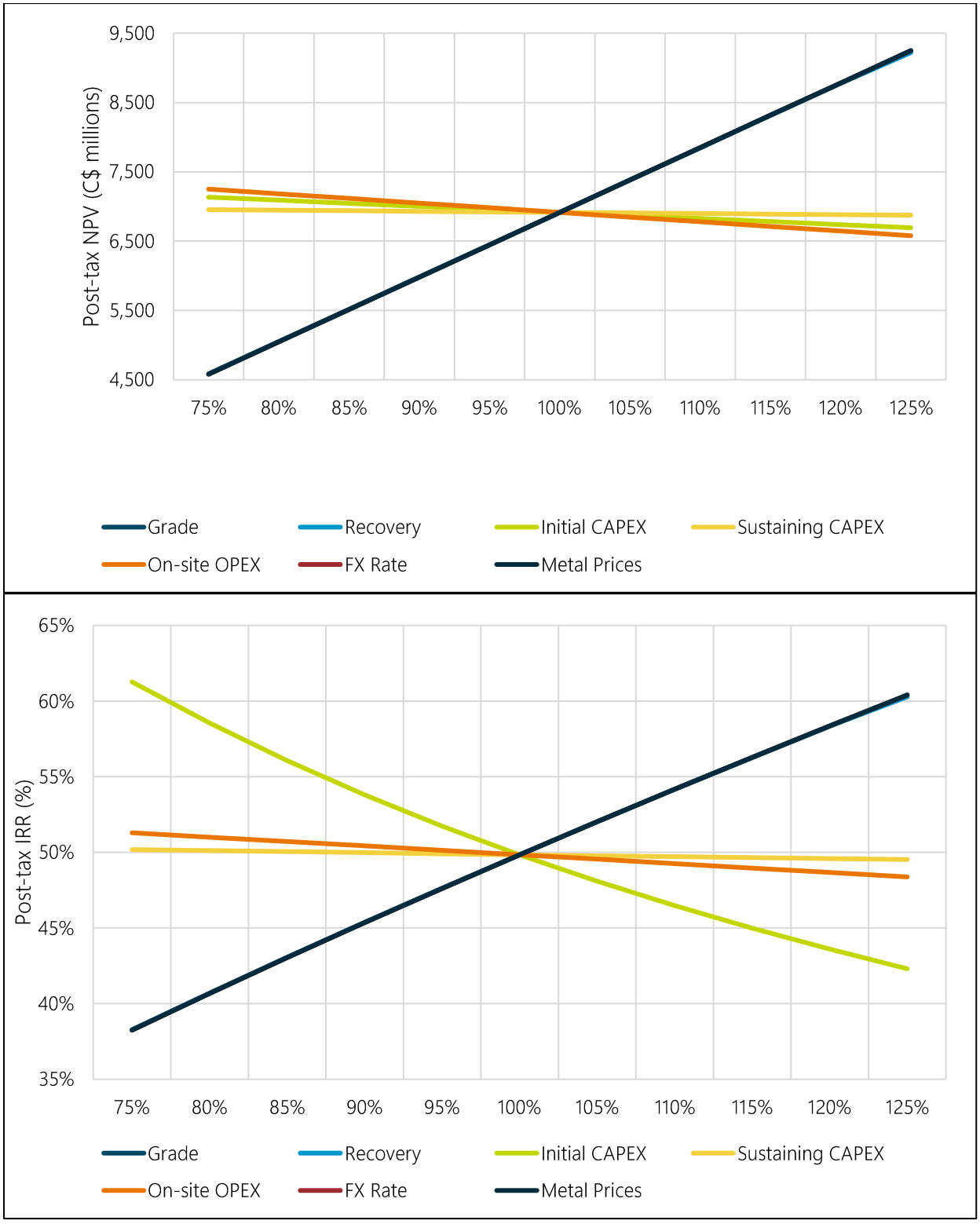
Source: Northcliff, 2026

Figure 22-2: Pre-tax sensitivity summary



Note: Recovery, head grade, metal price and FX rate overlap. Source: Northcliff, 2026
Source: Northcliff, 2026

Figure 22-3: Post-tax sensitivity summary



Note: Recovery, head grade, metal price and FX rate overlap.
Source: Northcliff, 2026

23 Adjacent properties

Sisson is the most advanced and significant mineral project in the area. There are no adjacent properties with relevant disclosure on them.

24Other relevant data and information

24.1 Sisson project execution plan

This Project Execution Plan (PEP) establishes the FS-level execution basis, delivery strategy, and governance framework for advancing the Sisson Project from the Feasibility Study (FS) stage through subsequent engineering and execution phases. The PEP covers the plan for engineering, procurement, construction, and commissioning of the Project and identifies key interfaces, constraints, and execution priorities to support effective project delivery. The Project is currently planned to be implemented using an engineering, procurement, construction management (EPCM) delivery model, subject to subsequent Project authorization and commercial arrangements.

A preliminary EPCM schedule has been developed and will be updated during the next project phase. A peak construction workforce of approximately 500 direct workers has been estimated based on benchmark data from comparable projects and Ausenco's experience in delivering similar projects. Construction methodologies, workforce requirements, and execution planning will be further developed during the subsequent phase of the Project.

24.2Execution strategy

The Sisson execution strategy is based on progressing the selected FS configuration into Basic Engineering while maintaining alignment between the site investigation programs, engineering, procurement, construction planning, commissioning and the integrated execution schedule.

Construction and procurement packaging will be aligned with the execution sequence and work front priorities.

Key execution priorities are:

- advance contracting and construction-management planning to support the proposed execution sequence, procurement packages and construction work fronts;
- progress engineering required to support Early Works and long-lead procurement;
- establish the primary and secondary site access corridors early to open independent construction work fronts;
- advance quarry development, initial site earthworks, TSF starter works and other enabling activities in parallel where practical;
- maintain alignment between construction activities, applicable permitting and approval requirements, environmental commitments and site-access constraints throughout project execution;
- prioritize long-lead equipment and vendor data required to protect engineering and construction milestones;
- coordinate the NB Power transmission, interconnection and site substation interfaces with the permanent power and commissioning sequence;
- sequence major earthworks, concrete and building enclosure to reduce exposure to New Brunswick winter conditions and enable continued SMP and E&I work indoors; and
- manage interfaces between mine development, process facilities, TSF/WRF, utilities and off-site infrastructure through the project WBS, responsibility matrix and integrated execution schedule.

Construction and procurement packaging will be aligned with the execution sequence and work front priorities.

The execution strategy is intended to progress the FS design basis into Basic Engineering while protecting the principal schedule drivers, including Early Works readiness, site access, long-lead equipment, permanent power availability, TSF and water-management readiness, and construction work front release.

24.3 Engineering management plan

Engineering execution will be coordinated through the project WBS, defined responsibilities, battery limits, and technical interfaces. An overall engineering integrator will coordinate inputs from specialist consultants, vendors, and major service providers to maintain alignment with the project design basis, execution schedule, procurement priorities, construction sequence, and commissioning requirements. Engineering will progress in accordance with applicable Project governance, safety-in-design, and stage-gate requirements.

The engineering program will progress from the Feasibility Study and Basic Engineering phases into Detailed Engineering, where the selected project layout and design basis will be developed into construction-ready documentation, procurement packages, and vendor data. The FS establishes the selected Project configuration and design basis at feasibility-study level. Basic Engineering and Detailed Engineering will progressively develop this basis into procurement- and construction-ready documentation, incorporating vendor information and the additional engineering definition required for execution. As the Project progresses beyond the FS, outstanding design development, interface definition, vendor engineering and execution-phase planning will be completed I accordance with the applicable stage-gate requirements. Material design alternatives affecting the selected FS configuration are expected to be resolved before progression into Detailed Engineering. Throughout execution, engineering deliverables will be prioritized to support early works, long-lead procurement, and construction milestones, while ongoing value engineering will focus on opportunities to improve cost, constructability, operability, and schedule performance.

Specialist engineering services, including geotechnical, hydrology, and other discipline-specific studies, will be delivered by qualified consultants under the coordination of the Engineering Manager. Engineering will also provide technical support for permitting activities, early works development, and construction planning. Regular constructability reviews at key design maturity stages will ensure that designs are optimized for safe, efficient, and practical construction, with lessons and recommendations incorporated into the detailed design. This integrated and disciplined approach is intended to minimize project risk, maintain alignment between engineering, procurement, and construction activities, and support successful project execution.

24.4 Procurement and contracts strategy and management plan

The procurement and contracting strategy is structured to support the overall project execution schedule, engineering maturity, and construction sequencing. Northcliff will retain contractual authority for major supply and construction contracts, while the EPCM Contractor will manage procurement and contracting activities within its authorized scope, including package development, tendering, bid evaluations, expediting, and contract administration. Procurement priorities will focus on Early Works activities, long-lead equipment, permanent power infrastructure, and critical construction packages required to maintain the integrated project schedule. A detailed Procurement and Contracting Management Plan will be

developed during the next project phase to define tendering, commercial management, logistics, vendor quality assurance, expediting, and contract administration requirements.

Procurement activities will be integrated with engineering and construction planning so that equipment, materials and vendor information are available in accordance with the required engineering and construction dates. Early procurement attention will focus on long-lead equipment, Early Works requirements, permanent power infrastructure and other packages capable of constraining construction, energization or commissioning. Vendor information from critical equipment packages will be incorporated into the engineering schedule and design-development process.

Major supply and construction packages will be competitively tendered in accordance with the Project procurement strategy. Package boundaries and procurement timing will continue to be refined during subsequent engineering as design maturity, required-on-site dates and construction work front requirements are confirmed.

24.5 Project controls and reporting plan

Project Controls will provide integrated framework for managing scope, schedule, cost, progress, change, and risk throughout project development and execution. The project work breakdown structure (WBS) and integrated execution schedule provide the common basis for aligning engineering, procurement, construction, commissioning, and cost and schedule information. The execution schedule identifies the principal Project milestones key interfaces, required-on-site dates, and external dependencies. Cost and schedule controls will be progressively developed as the Project advances beyond the FS while maintaining alignment with the FS estimate, execution schedule and approved Project scope.

Formal change-management processes will be used during subsequent Project phases to control changes to the approved design basis, scope, cost, schedule, and execution strategy. Project controls and risk management will be integrated so that material execution risks and mitigation actions are reflected in schedule development, cost forecasting and execution planning., Key execution risks identified during the FS include Early Works and site-access readiness, environmental and seasonal constraints, winter productivity, long-lead procurement and vendor-data timing, contractor availability and mobilization, NB power and permanent-power dependencies, TSF and water- management interfaces, and coordination of concurrent mine, process plant and infrastructure work fronts. These risks will continue to be monitored and developed during subsequent engineering and execution planning.

24.6 Construction execution plan and commissioning

Construction is currently planned to be executed under an EPCM delivery model, with Northcliff retaining Project ownership and approval authority and the EPCM Contractor providing overall construction management, integration, and coordination. The project will be developed through a phased greenfield construction strategy beginning with Early Works, including site access roads, quarry development, clearing, drainage controls, and enabling infrastructure, followed by bulk earthworks, TSF construction, mine development, process plant construction, utilities, and permanent infrastructure. Construction activities will be organized through defined contract packages and progressive work front releases, allowing civil, concrete, structural, mechanical, piping, and electrical works to proceed concurrently where practical. The execution strategy emphasizes early establishment of access, logistics, quarry material supply, and critical infrastructure, while managing interfaces between the mine, process plant, TSF, water management systems, and power infrastructure. Seasonal constraints, environmental work windows, temporary services,

and constructability considerations will be incorporated into detailed planning to support safe and efficient execution. Temporary power and construction services will support Early Works, construction and selected pre-commissioning activities until permanent Project services become available. Permanent Project power will be coordinated with the NB Power transmission, and interconnection works, Project substation and site distribution system, with staged energization aligned to construction completion and commissioning requirements.

Commissioning will follow a progressive, system-based approach beginning with pre-commissioning and advancing through dry commissioning, wet commissioning, start-up, ramp-up, and final operational handover. The strategy is based on phased turnover of completed systems rather than waiting for full project completion, enabling commissioning activities to commence while construction continues in other areas. Successful commissioning will depend on readiness of power systems, water management infrastructure, TSF facilities, mine ore supply, vendor support, and operational personnel. Under the planned EPCM delivery model, the EPCM Contractor is expected to lead commissioning planning, systemization, and turnover coordination, with Northcliff retaining responsibility for operational acceptance and final handover. The overall approach is intended to reduce schedule risk, support progressive system completion, and enable a controlled transition from construction to sustainable operations.

Project execution will be supported by established HSE, quality and risk-management systems appropriate to the applicable engineering and execution phase. These systems will address safety-in-design, construction HSE requirements, quality assurance and control, vendor and contractor quality, non-conformance management, and the identification and mitigation of material Project risks. Environmental and permitting requirements relevant to construction and operations are addressed elsewhere in this Technical Report.

24.7 Execution schedule

A preliminary integrated Project execution schedule has been developed as part of the FSU and is presented in Figure 24-1, with milestones detailed in Table 24-1. The schedule integrates engineering, procurement, Early Works, mine development, TSF and water-management works, process plant and infrastructure construction, permanent power, commissioning and ramp-up. The schedule basis reflects the planned execution sequence, key construction work fronts, major procurement requirements, seasonal constraints, and principal external interfaces.

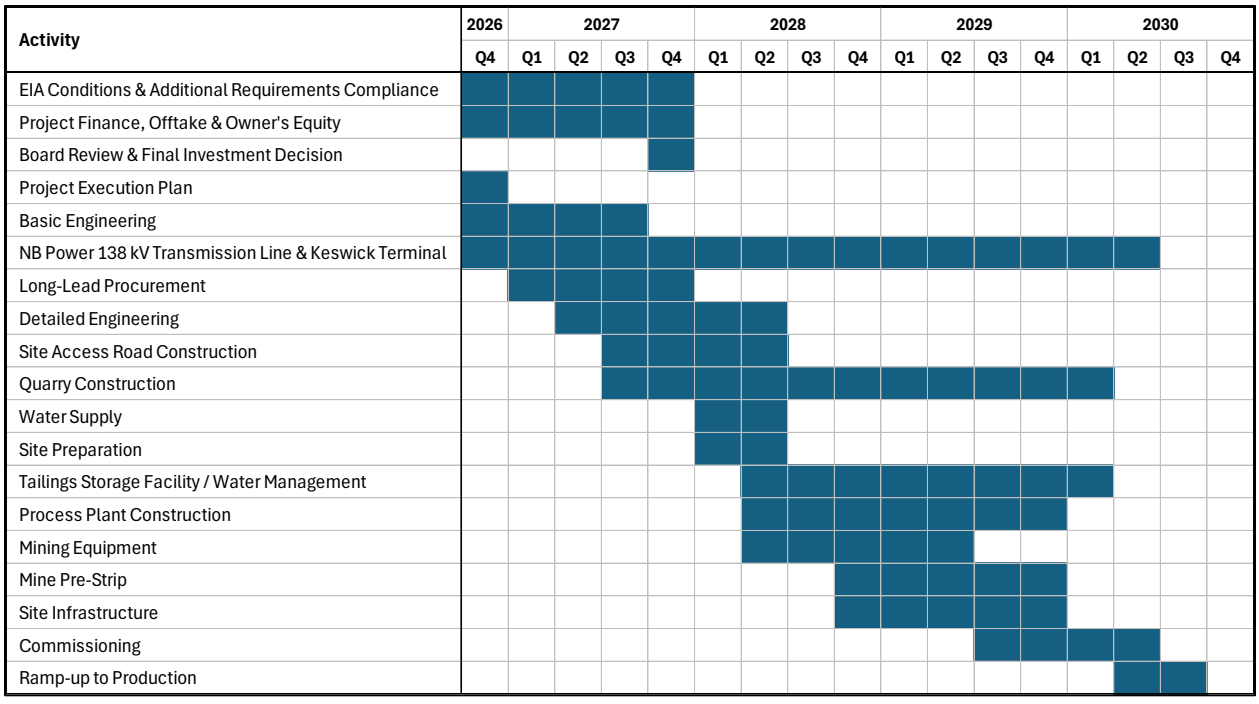
Key schedule drivers include timely completion of site access, availability of quarry and construction materials, release and delivery of long-lead equipment, NB Power transmission and interconnection activities, availability of permanent power, TSF and water-system readiness, mine ore availability, and progressive mechanical completion and turnover of systems for commissioning.

Independent of the initial Project construction and commissioning schedule, approximately 9.5 km of NB Power's 345 kV Line 3011 must be relocated to maintain separation from the planned open pit, process plant, and rock quarry infrastructure. Based on the current mine plan, this relocation is not anticipated to be required until approximately Year 8 and is therefore not a predecessor to initial Project commissioning or production. Therefore, it has not been incorporated in the execution schedule presented in Figure 24-. NB Power's Facility Study has outlined that the 345 kV Transmission Line relocation works, inclusive of engineering, procurement, and construction would take approximately four years (NB Power, 2026).

Table 24-1: Sisson Project execution milestones

Milestone	Target Date
Basic Engineering commencement	Q4 2026
Early works commencement	Q3 2027
Main construction commencement	Q1 2028
Mechanical completion	Q1 2030
Permanent power available	Q2 2030
Dry commissioning complete	Q2 2030
Wet Commissioning complete	Q2 2030
First production / start-up	Q2 2030
Commercial Production	Q3 2030

Figure 24-1: Sisson Project execution schedule



Source: Ausenco, 2026

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 Technical Report.

25.2 Mineral tenure, surface rights, royalties and agreements

- Sisson Project comprises 707 mineral claim units covering approximately 14,140 ha in three contiguous claim groups.
- All claims are active, in good standing, and held by Sisson Limited Partnership.
- Northcliff Resources owns 88.5% of Sisson LP.
- Todd Minerals Ltd. owns 11.5% of Sisson LP.
- Todd also controls 81.9% of Northcliff Resources.
- Northcliff does not own surface rights over the project area.
- Property is free and clear of mortgages, liens, pledges, and other material third-party encumbrances.
- No royalties or back-in rights apply.
- No known title issues affecting access or development.
- No known environmental liabilities.
- All permits required to conduct the drilling and site work currently underway on the property have been obtained. All necessary permits to carry out the recommended programs will be routinely applied for and obtained prior to commencing those activities.

25.3 Geology, mineralization, and Mineral Resource Estimates

- Collar and downhole surveys have been conducted using conventional and appropriate methods and equipment.
- The database has been compiled and maintained in a competent manner, using a reasonable level of verification and validation.
- Verification of Mineral Resource assays against original certificates resulted in almost no discrepancies.
- The geology of the deposit, mineralization styles, and controls to mineralization are well understood and have been applied appropriately in the MRE.
- The assumptions, parameters, and methodology used in the MRE are appropriate for the style of mineralization and proposed mining method.
- The MRE was prepared by running a new pit optimization on a spatially expanded version of the block model (the 2012 model) which supported the 2012 MRE (December 31, 2012). Current metal prices, mining costs, and updated processing parameters were used as input to a new NSR equation and cut-off value determination. WO_3 and Mo values from the 2012 model were unchanged.
- The Mineral Resource classification follows the CIM (2014) definitions, as incorporated into NI 43-101.

- The MRE shows an increase in tonnes and metal content, and decrease in grades, over the 2012 MRE. The changes are due to inclusion of lower grade blocks in a similarly sized optimization shell due to higher metal prices and an associated lower NSR cut-off value.
- As of December 31, 2025, the total Measured and Indicated Mineral Resources at the Project are estimated to total 421.6 Mt grading 0.063% WO_3 and 0.020% Mo, containing 26.6 million MTU of WO_3 and 185.9 Mlb of Mo. In addition, Inferred Mineral Resources are estimated to total 182.9 Mt grading 0.048% WO_3 and 0.020% Mo, containing 8.8 M MTU WO_3 and 80.6 Mlb Mo. Mineral Resources that are not Mineral Reserves do not have demonstrated economic viability.

25.4 Exploration and drilling

- The drilling database contains approximately 69.3 km of drilling for 407 drill holes spanning 47 years.
- Core recoveries have been generally excellent with minimal sample loss.
- Most drill core handling, logging, and sampling protocols have been well documented and carried out in a manner consistent with industry-standard practices.
- The analytical methods have been well documented. While methodologies have changed somewhat over the history of the Project, the various operators have employed conventional protocols and employed independent commercial laboratories accredited for the methods used.
- Reasonable and appropriate analytical QA/QC and sample security protocols have been observed for all programs since Geodex commenced exploration in 2004. For all recent operators, QA/QC protocols have been well documented, and results have been acted upon in a timely and acceptable fashion.
- Diamond, RC, and sonic drilling have been completed across the project area to acquire information on the mineralised system, for including metallurgical, geotechnical, and hydrogeological purposes.
- The Sisson Deposit is currently well defined by drilling to the northeast, northwest, and south. The exploration potential in the immediate vicinity of the deposit is limited, and there are no plans for additional exploration work.

25.5 Metallurgical testwork

A considerable amount of metallurgical testing has been completed on composites from the Sisson project. The composites were assembled from drill core samples that provide a reasonable spatial coverage across the deposit. The material has quite competent comminution properties and requires considerable crushing and grinding energy to achieve the designed primary grind size of 140 µm P_{80}. The testing indicates that there may be opportunities to pre-concentrate the feed, however these processes have not yet been sufficiently demonstrated to include in the designed flowsheet. Mineralogical assessments indicate that the feed material has sufficient liberation values at primary grind sizes of –300 µm in order to effectively recover molybdenum and tungsten from the flotation feed. The selected primary size is finer than required for target mineral liberation, however it has been selected to mitigate coarse scheelite losses from grains that are challenging to recover in the tungsten flotation froth.

Molybdenum flotation is effective and can achieve saleable concentrate quality with a modest amount of regrinding on the 1st cleaner concentrate and sufficient cleaner stages.

Tungsten flotation across the ambient circuit has consistently achieved recoveries in the 80-90% range in closed circuit, generating a concentrate of suitable quality for subsequent upgrading after hot conditioning.

The hot conditioning process followed by additional cleaner flotation has demonstrated successful upgrading, however this circuit appears to be sensitive to collector dosages and addition points. Challenges have been encountered in this testing due to the low mass of flotation feed material by this point in the circuit. Further testing on larger feed sample masses is recommended.

Despite the low feed grade of the material, testing has demonstrated that a significant portion of the scheelite can be recovered by gravity means. Further evaluation should be conducted to investigate inclusion of gravity concentration as a means to increase overall recovery and possibly provide a synergetic benefit with the flotation circuit by removing coarse scheelite.

25.6 Mineral reserve estimates

Proven and Probable Mineral Reserves have been modified from Measured and Indicated Mineral Resources. Inferred Mineral Resources have been set to barren rock. The Mineral Reserves are supported by the 2026 Feasibility Study Mine Plan and classified in accordance with the 2014 CIM Definition Standards.

Sisson Proven and Probable Mineral Reserves total 276.8 Mt at an average NSR of $35.03/t and 0.075% WO_3 and 0.023% Mo.

Changes in the following factors and assumptions may affect the Mineral Reserve estimate: metal price, interpretations of mineralization geometry and continuity of mineralization zones, geotechnical and hydrogeological assumptions, ability of the mining operation to meet the annual production rate, operating cost assumptions, process plant and mining recoveries, the ability to meet and maintain permitting and environmental license conditions, and the ability to maintain the social licence to operate.

25.7 Mining methods

The mining operations will use conventional drill, blast, load, and haul open pit mining methods and equipment, with autonomous haul trucks. A reasonable open pit mine plan, mine production schedule and mine capital and operating costs have been developed for Mineral Reserves at the Sisson project.

Mining operations will start approximately 18 months prior to mill operations and will run for ~16.5 years after mill start-up. An annual mill feed rate of 10.5 Mt is targeted, with an allowance for ramp up in the first year of mill operations targeting 9.2 Mt. Mid-grade ore is stockpiled in the MGSP and reclaimed where possible, up until Year 7, after which the MGSP is submerged beneath tailings. There is no plan within this feasibility study to recover the remaining stockpiled material.

Pit layouts and planned mine operations are typical of other open pit mine operations in Eastern Canada, and the unit operations within the developed mine operating plan are proven to be effective for these other operations.

The mine plan supports the cash flow model and financials developed for the Feasibility Study.

25.8 Recovery methods

The recovery methods align with conventional practices in the industry for comminution and flotation of industrial metals, and consider proven processes used in the industry for similar projects such as hot conditioning ahead of tungsten flotation. Previous studies, coupled with new testwork results, were used to develop the resulting flowsheet.

25.9 Infrastructure

25.9.1 Site infrastructure

The planned mine is a greenfield site and requires mine and process-related infrastructure. Access roads to and around the project site will be required. No camp accommodation is planned for the construction or operations phase of the Project as workers will reside in, and commute from, nearby communities.

Site access assumes that the 11 km Re-aligned Fire Road and 3 km Site Access Road will be constructed. The 39 km Primary Site Access and 13 km Secondary Site Access roads will be upgraded to support construction and operational traffic.

Power will be supplied by NB Power. A proposed 138 kVA transmission line from Keswick Terminal to the project site.

Process water will be supplied from the TSF reclaim and WTP discharge. Stormwater will be managed within the Sisson Project boundaries by capturing and conveying contact water for treatment and diverting non-contact water away from developed areas. Well water will be used for potable purposes.

25.9.2 Tailings storage facilities

The Tailings Storage Facility (TSF) will contain approximately 268 Mt of tailings, 9.2 Mt of mid-grade ore, and 201 Mt of barren rock from the proposed mining operations over a 27-year mine life. The TSF embankment will comprise an approximately 8 km long ring-dam structure that will be constructed with earthfill and quarried rock.

The TSF is located adjacent to the process plant and open pit and contributes to a compact site layout that is situated in the headwaters of Napadogan Brook and McBean Brook and therefore does not require extensive surface water diversions. The TSF has high storage efficiency resulting from the relatively low and long perimeter embankment. The foundation conditions typically comprise dense glacial deposits and weathered bedrock overlying competent bedrock.

Thickened tailings will be discharged from spigots along the embankment crest to form long, sloping beaches towards the centre of the TSF. Barren rock will be placed within the TSF concurrently with tailings deposition such that the rock is progressively encapsulated by the tailings and ponded water. Water from the small supernatant pond in the TSF will be reclaimed to the process plant with barge-mounted pumps.

The proposed TSF concept is typical for a hard rock mining project of this scale located in similar physiographic regions. The design is consistent with international good practice and conforms to the guidance provided by the Canadian Dam Association and the Mining Association of Canada. The design concept is suitably flexible to accommodate the expected variability in climate, ground conditions, and mine and mill operation.

25.9.3 Water treatment facilities

Water treatment infrastructure is a critical component of the Project, supporting environmental compliance for discharge, operational water reuse, and closure planning.

The WTP will provide long-term treatment for the Project. The current design basis assumes a maximum treatment capacity of 2,083 m^3/h, based on the updated water balance model. The WTP design includes HDS precipitation using lime for metals, bicarbonates and solids removal, peroxide oxidation for thiosalts removal. After metal precipitation, pH neutralization using sulphuric acid will be required prior to discharge.

Treated water will be discharged to Sisson Brook with a small portion recycled for use in the Mineral Processing facility. The sludge waste stream will be pumped and fed to the tailings thickener to be co-deposited with tailings. The WTP facility is designed to remain operational through active closure and into post-closure care.

The system incorporates features to handle seasonally variable flow, cold temperatures, and complex influent chemistry. Redundancy, heat tracing, and modularity are integrated to ensure resiliency and compliance under all operating scenarios. The water treatment strategy has been developed to support the Project's overall water management plan and supports key environmental performance objectives and meets permit requirements.

25.10 Environmental, permitting and social considerations

The provincial EIA included 40 conditions of approval, and Northcliff is working to fulfill these. Numerous approvals, authorizations, and permits are required prior to construction of the Project. Compliance with terms and conditions of approvals, standards contained in federal and provincial legislation and regulations, and commitments made during the EA/EIA processes will need to be assured throughout all project phases.

Northcliff is committed to engaging with Indigenous communities, the public, and stakeholders in an open, transparent, and responsive manner on the Project. Northcliff has engaged with local First Nations communities since it became involved in the Project, and public/community outreach is ongoing. This collaboration will continue as the Project advances.

25.11 Capital cost estimate

The capital cost estimate was developed in Q2 2026 CAD dollars based on budgetary quotations for equipment and construction contracts, as well as Ausenco's in-house database of projects and studies including experience from similar type operations.

The estimate conforms to the AACE Class 3 guidelines for an FS-level estimate with a ±15% accuracy range.

The estimate includes mining, processing, onsite infrastructure, tailings storage and water management facilities, offsite infrastructure, project indirect costs, project delivery, owner's costs, and contingency.

The total initial capital cost for the Sisson project is $1,528 million and LOM sustaining capital cost is $571 million; resulting in a total capital cost of $2,099 million.

The closure costs are estimated at $123 million for reclamation, closure, and post-closure water treatment (excluding water treatment in perpetuity). $5.9 million is estimated per year for post-closure water treatment in perpetuity.

25.12 Operating cost estimate

The operating cost estimates are reflective of Q2 2026 pricing, unless otherwise specified, noted using Canadian dollars as the base currency. Allowance for inflation has not been included. Operating costs include the ongoing cost of operations related to mining, processing, quarry development, tailings disposal, water treatment, and general and administration activities.

The process operating and general and administrative operating cost estimates were developed from budgetary quotations and Ausenco's in-house data on Canadian projects and existing operations. The mine

operating and quarry operating cost estimates are based on Owner-managed operations, maintenance and technical services departments, and have been built up from first principles.

An average operating cost was estimated for the Sisson Project based on the proposed mining schedule. The average LOM unit operating cost for the Sisson Project is estimated to be C$23.09/t milled. For more details, refer to Section 21.3.

25.13 Economic analysis

The economic analysis was performed, assuming an 8% discount rate. On a pre-tax basis the NPV discounted at 8% is C$12,290 million; the IRR is 68.3%, and payback period is 1.3 years. On a post-tax basis, the NPV discounted at 8% is C$6,915 million; the IRR is 49.8%, and the payback period is 1.6 years.

25.14 Risks and opportunities

25.14.1 Risks

25.14.1.1 Exploration and drilling

No exploration drilling is planned on the Project.

25.14.1.2 Metallurgical testing

WO_3 cleaner flotation post hot conditioning has not consistently achieved target metallurgical performance and there is a risk tungsten recovery and/ or concentrate grades may not be as high as estimated. This has been attributed to low feed sample masses and unoptimized collector dosages. This risk can be mitigated with further testing to ensure that the process variables can be effectively controlled and that the process equipment selection provides for sufficient dosage points and control to provide a robust design.

The majority of the metallurgical testing has been completed on master composites, and metallurgical performance on a variability basis is not yet well established. There may be some materials that return metallurgical performance that is below the estimated recoveries and concentrate grades. This risk can be mitigated with further variability testwork.

Full circuit testing using recycle process water in the tungsten flotation circuit has not been completed and may have a negative impact on the flotation process and recovery and/ or concentrate grade. This risk can be mitigated with further testwork to demonstrate if there are any effects and how to minimize any negative effects.

25.14.1.3 Mineral Resource Estimate

If additional drill data becomes available as a result of other programs such as metallurgical or geotechnical, these data should be incorporated into an updated geological model, if warranted. An updated geological model would enable a full update to the MRE. This would not likely impact the current MRE significantly, as the classified mineralization is covered by the current MRE drilling. Grades and tonnages may undergo local refinement, but global MRE figures would likely remain similar to the present estimate.

25.14.1.4 Mineral Reserve Estimate

While the Project's financial model is exposed to market volatility, the physical Mineral Reserve estimates are heavily insulated against both moderate metal price drops and operating cost escalations. The open-pit limits were established using conservative Q4 2025 metal prices, meaning market prices would have to

fall below that initial baseline to negatively impact reserve quantities. Furthermore, rather than relying on a static break-even cut-off grade, the mine plan employs a variable cut-off strategy alongside the MGSP. Because 9.2 Mt of the MGSP is ultimately encapsulated beneath the tailings and left unrecovered, the effective LOM cut-off grade ($16.31/t NSR) is considerably higher than the Project's break-even threshold, buffering the reserves against moderate increases in operating costs.

Given these substantial built-in economic margins, the primary risk to the Mineral Reserves lies in geological and grade variance. As with all mineral projects, there is a risk that the actual ROM grades and tonnages will deviate from block model predictions. If operational contact dilution or handling losses exceed the modeled 2.56% and 1.24% respectively, the resulting mill feed grade will be negatively impacted, subsequently affecting overall project cash flows.

25.14.1.5 Mining methods

The following risks exist within the mine plan and should be reviewed in future stages of project development:

- The mine plan relies on an AHS to achieve the targeted unit costs and productivity. Risks include potential schedule delays during the initial site integration and network connectivity or latency issues (Wi-Fi/LTE) in the pit.
- The current mine plan relies on the MGSP operating within the TSF footprint until Year 9. If mine production outpaces milling capacity, or if TSF water and tailings levels rise faster than modeled, there is a risk that the stockpile reclaim area could be overtaken earlier than anticipated, resulting in the loss of more than the currently estimated 9.2 Mt of mid-grade ore.
- The strict requirement to encapsulate all potentially acid-generating (PAG) barren rock within the BRSA causeway design introduces some risk due to sequencing errors. Delays in TSF embankment construction or volumetric swell prediction errors could temporarily constrain pit waste stripping, impacting the ore delivery schedule.

25.14.1.6 Recovery methods

The following risks exist within the process plant design and should be reviewed in future stages of project development:

- Molybdenum flotation equipment sizing was conducted using limited locked cycle test results, potentially resulting in bottlenecks should high variability be seen in pockets of the mine plan. This risk is likely minimal and can be further reduced via additional flowsheet validation testwork.
- Dilution flotation testwork was not available at the time of design when sizing the Jameson cells and column flotation cells. Dilution testwork and pilot plant testing, where applicable, should be conducted as part of the next phase of engineering to validate the selected equipment sizing.
- No signature plot data was available at the time of design for the molybdenum regrind mill, which may become limited if actual energy requirements are above design. This risk can be mitigated by conducting signature plot testing with the selected vendor during the next phase of engineering.
- Concentrate dewatering data, including static settling, dynamic settling, and filtration, were not available for the molybdenum and tungsten concentrate areas. This may lead to additional capacity being required if actual solid-liquid separation requirements are beyond the current design. This risk can be mitigated by conducting the necessary testwork with the selected vendor during the next phase of engineering.

- Tungsten flotation requires a significant amount of reagent material, particularly sodium silicate and sodium carbonate. The quantities outlined in Section 17 may be limited by local production and supply, leading to increased costs if alternative sources are required. This risk can be mitigated by conducting a logistics study during the next phase of engineering to ensure adequate materials can be sourced for the Project.

25.14.1.7 Infrastructure

25.14.1.7.1 Process plant and site access roads

The geotechnical investigations conducted for the Sisson Project in 2011 and 2014 did not encompass the entire development footprint defined in the current feasibility study, resulting in gaps in representative subsurface information for certain proposed plant and laydown areas, building foundations, retaining walls, ponds, and associated civil infrastructure.

The available geotechnical information for the access roads and structural condition and load-capacity assessments of existing bridges should also be confirmed and supplemented, as required, to support the current Project design.

Additional geotechnical investigations and structural assessments are required to support Detailed Engineering and address remaining gaps in site coverage and infrastructure condition information. The absence of adequate geotechnical interpretation, representative subsurface data, and confirmed structural capacity of existing bridges introduces uncertainty into the engineering design. Advancing engineering without this information increases reliance on assumptions and may result in overdesign, design revisions, capital cost increases, and schedule delays.

25.14.1.7.2 Tailings storage facilities

The following risks were identified for the TSF:

- The geological conditions within the foundation differ substantially from the current characterization leading to:
- Greater foundation excavation depths and volumes
- Flatter embankment slopes
- Realignment of the embankment centreline
- More complex seepage cut-off and recovery systems such as synthetic liners and foundation grouting
- The physical properties of the tailings are substantially different or more variable than expected leading to:
    - Greater or lesser rates of consolidation, leading to variable water release, unpredictability in the water balance, and faster or slower rates of rise
    - Increased seepage due to higher hydraulic conductivity
- The quantity and suitability of locally sourced fine-grained materials are substantially less than anticipated leading to:
    - Requirement to import material from off site
    - Modifications to the TSF design to incorporate alternative materials.
- The geochemical source terms used in the predictive water quality modelling for tailings, barren rock, construction materials are based on a limited set of historical samples and tests that may not fully represent the variability of future full-scale conditions.

- The geochemical source term for process water discharged to the TSF with the tailings that is used in the predictive water quality modelling is based on locked cycle tests and may not fully represent full-scale conditions.

25.14.1.7.3 Quarry

Geochemical characterization of the proposed quarry rock for TSF dam embankment construction is based on a limited dataset. Kinetic test work on September 2011 drill core forms the basis of the current source terms used in the water quality model for embankment fill and exposed quarry faces. Subsequent analysis of samples from November 2013 drilling shows elevated arsenic in approximately 25% of the samples, with variable concentrations, which is consistent with regional granite mineralization. Existing kinetic test work used only low-arsenic intervals and does not provide leachability data for potentially higher-arsenic material. The potential for adverse geochemical properties is considered possible but unconfirmed. The following risks are identified:

- The current embankment fill and quarry face source terms may not represent the quarried rock, and arsenic leachability may be greater than predicted.
- Unsuitable quarry rock would be disposed of in the TSF, increasing stored volumes in the TSF and potentially requiring larger areas or depths for the TSF and quarry. The waste rock source term needs to be updated.
- Exposed quarry high walls with elevated arsenic could generate poorer-quality contact water post-closure when the quarry and pit walls become the primary arsenic sources. The geochemical source terms may require revision.
- If the current quarry is unsuitable, alternative sources may need to be identified and tested, potentially outside the currently assessed areas, with implications for environmental assessment and permitting.
- Future geochemical testing and updated modelling may require adjustments to the WTP arsenic removal process.

25.14.1.7.4 Water treatment facilities

The preliminary risk assessment conducted for the water treatment facilities identified the following key risks and their potential impacts:

- The current WTP design basis is derived primarily from predictive water quality and water balance modelling. Uncertainty remains regarding design flows, influent chemistry, operating scenarios, and long-term closure conditions. Changes to the design basis could affect process selection, treatment capacity, capital cost, operating cost, and project schedule.
- The WTP design flow is based on modeled 95th percentile monthly flow projections. If hydrologic, hydrogeologic, storage, or operational assumptions prove conservative, treatment infrastructure may be oversized. Conversely, changes in site water management or climate conditions could increase treatment requirements.
- Water treatment performance assumptions, including reagent demand, thiosalt oxidation, solids separation, sludge production, and treated water quality, have not been fully validated under representative operating conditions. Actual treatment performance may differ from design assumptions, affecting compliance, operability, and operating cost.
- The current basis focuses on MDMER Schedule 4 effluent criteria and acute lethality requirements. There is risk that provincial requirements, receiving-environment objectives, CCME aquatic life guidelines,

mixing zone limitations, or project-specific approval conditions could require additional treatment, monitoring, discharge management, or mitigation measures.

- Long-term post-closure water quality predictions are based on modelling and assumptions regarding pit lake development, geochemistry, and site hydrology. Actual closure water quality may differ from predictions, affecting long-term treatment requirements and closure liabilities.

25.14.1.8 Environmental, permitting, social and community

The Project has received approvals under the federal and provincial environmental assessment legislation and under the federal Fisheries Act. As such, regulatory risk is reduced. However, given the passage of time, there is some risk that regulators may impose more stringent environmental requirements that were contemplated as part of the approvals processes.

As the Project will result in the loss of wetland, wetland compensation is required. Given the unprecedented size of the compensation that is required, there is some risk regarding the development of a feasible compensation project or projects.

25.14.1.9 Commodity Prices

The ability of mining companies to fund the advancement of their projects through exploration and development is influenced by commodity prices. Variations in the commodity prices may lead to reduced or elevated revenues compared to those projected in this study.

25.14.2 Opportunities

25.14.2.1 Exploration and drilling

No exploration drilling is planned on the Project.

25.14.2.2 Metallurgical testing

There are opportunities to optimize the process flowsheet what could provide economic benefits.

- The recent testwork program identified several parameters that require increased control and there is an opportunity to improve tungsten recovery and/ or concentrate grade with increased parameter control and could be realized with further testwork.
- Ore sorting could provide economic benefits by reducing capital and/or operating costs. This opportunity could be realized with comprehensive economic analysis of further testwork that confirms WO_3 upgrading in the fines portion and sorter response on crushed variability samples.
- The economics of SAG milling and pebble sorting could provide economic benefits by reducing capital and/or operating costs. This opportunity could be realized with further sorting testwork.
- Inclusion of gravity concentration could provide economic benefits by reducing capital and/or operating costs. Recovery of coarse liberated scheelite by gravity means could allow the primary grind size to increase, while mitigating challenges of coarse scheelite losses in the tungsten flotation circuit. This opportunity could be realized with further testwork.
- Returning Mo 1st cleaner tailings to the bulk sulphide circuit and adding a cleaning stage on the bulk sulphide rougher concentrate could provide economic benefit. Further testwork could determine the potential value of reducing the WO_3 that is currently lost to these rougher concentrates.
- There is an opportunity to increase the recovery of fine scheelite grains using different flotation technologies. This could be evaluated with further testwork investigating flotation equipment known to improve fine particle recovery.

25.14.2.3 Mineral Resource Estimate

If additional drill data becomes available as a result of other programs such as metallurgical or geotechnical, any refinements to the geological model should be included and the additional analytical data be incorporated into a new MRE. These data may enhance local grade estimates but they are unlikely to materially effect the global resource.

25.14.2.4 Mineral Reserve Estimate

Opportunities exist to incrementally optimize the Mineral Reserves and project economics based on recent metal market improvements. The current open-pit limits and production schedule were established using conservative Q4 2025 price assumptions. While major pit expansion is naturally constrained by the existing permit boundaries and the fact that the current design already captures the vast majority of the M&I resources, re-evaluating the pit limits at the higher 2026 tungsten prices may identify localized areas where marginal resources could be converted into mineable reserves.

Additionally, higher metal pricing may improve the economic case for the 9.2 Mt of mid-grade material currently slated for encapsulation beneath the tailings by Year 9. Future engineering phases could evaluate PAG rock mitigation, alternative stockpile locations, MGSP elevations, or TSF deposition strategies to determine if recovering this remaining material justifies the associated re-handling and infrastructure trade-offs.

25.14.2.5 Mining methods

The following opportunities may exist to add additional value to the mine plan and should be reviewed in future stages of project development:

- The current pit designs rely on historical geotechnical parameters. An updated geotechnical and hydrogeological drilling program targeting the ultimate pit walls could yield data supporting steeper inter-ramp and overall pit slope angles. Steeper walls would reduce the volume of barren rock that must be stripped and handled, directly improving the LOM strip ratio and reducing mining costs, and indirectly reducing the volume of the TSF.
- Incorporating trolley assist technology on the main, long-haul in-pit ramps could significantly reduce diesel consumption for the 147t autonomous trucks. This would lower operating costs, decrease greenhouse gas emissions, and increase uphill haul speeds, effectively improving the productivity of the haul fleet.
- Further geochemical sampling and kinetic testing could demonstrate that a larger portion of the barren rock within the pit is non-PAG. Reclassifying this material would allow it to be used in the construction of the downstream TSF embankments, drastically reducing the volume of material that needs to be drilled, blasted, and hauled from the separate surface quarry.
- Conducting a trade-off study to replace the primary 147t AHS trucks in the quarry with a dedicated, smaller-capacity (e.g., 90t) manual or autonomous fleet could yield cost savings. A smaller fleet would allow for narrower quarry ramps, a reduced environmental footprint, and better pass-matching with the smaller loading equipment required for selective quarrying.

25.14.2.6 Recovery methods

Further opportunities exist to optimize the current process flowsheet with respect to capital and operating costs. The following opportunities have been identified:

- The final plant design proceeded in parallel to the feasibility study testwork program and selected a conservative design point to progress engineering and cost estimation efforts. Under the final recovery models optimization of the equipment sizing may be possible, particularly in the tungsten flotation areas, should a higher-grade intermediate tungsten product be achievable.
- Additional testwork can be conducted to optimize reagent dosages in the tungsten flotation areas, reducing operating costs.
- Additional testwork can be conducted to investigate removal of the coagulant system by identifying conditions for settling the NAG tailings with flocculant alone.

The following additional opportunities have been identified for the Project outside of the current plant design:

- Ore sorting was explored in parallel to the feasibility study, with initial results appearing positive. Incorporation into the flowsheet could bring financial benefits to the Project if further testwork and economic evaluation supports inclusion. Given the initial results achieved in these initial tests, additional metallurgical and economic evaluations are warranted to further assess the applicability, operational integration, and potential value of ore sorting within the Sisson Project development strategy.
- Prior project assessments explored producing an ammonium-paratungstate (APT) hydrometallurgical product instead of a tungsten concentrate. This product typically provides a premium compared to concentrate but the Project lacked sufficient testwork to assess this flowsheet as part of the current feasibility study. If the requisite testwork can be conducted to validate a design and recovery model, it may be possible to collect this premium in the future.

25.14.2.7 Water treatment facilities

The opportunities to mitigate the risks identified in the preliminary risk assessment are presented below:

- Continue refinement of the water and load balance model, confirm design assumptions, and update the WTP design basis as additional project information becomes available.
- During the next phase of the Project, more detailed process modelling and confirmation of influent water chemistry during the operation and closure/post-closure phases will be required to refine equipment selection, reagent consumption, and process operating conditions. This work will support further optimization of the WTP design and identify opportunities to optimize capital and operating costs.
- Continue optimization of water management infrastructure, storage volumes, and operating strategy. Reconcile treatment requirements with water balance modelling as project design advances.
- Conduct additional bench-scale and/or pilot-scale testing to confirm treatment performance, reagent consumption, sludge production, and treated water quality.
- Confirm applicable federal and provincial compliance requirements during permitting and environmental review. Assess receiving-water impacts and verify compliance assumptions with regulators and environmental specialists.
- Review the studies and/or increase pond volume to buffer high-precipitation events.
- Continue geochemical characterization and update predictive models as additional site data become available. Incorporate adaptive management into closure planning.

25.14.2.8 Environmental, permitting, social and community

Potential environmental, permitting and social opportunities within this Project include the following:

- Collaboration with Indigenous communities, the public, and stakeholders to advance the Project.
- Partnering with local education institutions to provide training for project employees.
- Working with local businesses on contracting opportunities to provide local benefits.
- Assessment of energy efficiency measures throughout the life of the Project to reduce greenhouse gas emissions.

26 Recommendations

26.1 Introduction

The Project shows positive economics, as demonstrated by the results presented in this Technical Report. It is recommended that the Project be advanced through to the basic engineering phase. The following sections detail the recommended future work for the Project. The estimated costs are summarized in Table 26-1.

Table 26-1: Recommendations cost summary

Program component	Estimated total cost (C$M)
Geology and Mineral Resource	
• Geology and mineralization	0.05
• Mineral Resource	0.15
Mine planning	
• Barren Rock and PAG Characterization	0.3
• Mine geotechnical drilling	0.0*
• Mine planning and trade-off studies	0.4
• Operational readiness and cost validation	0.35
Metallurgical testing and process recovery	
• Metallurgical testing and process recovery	0.95
• Particle Ore Sorting	0.2
Infrastructure	
• Site access, logistics, and targeted geotechnical investigation	1.3
• Site optimization and value engineering	0.35
• Fresh water and potable water supply	0.45
• Power supply and electrical infrastructure development	0.75
• Tailing management facility	2.9
• Water treatment facilities	0.8
Project execution and procurement	
• Procurement, contracting, and long-lead equipment strategy	0.4
• Early works definition and execution readiness	0.55
Environmental studies	1.0
Total	**10.9**

*No cost is shown because the geotechnical program is currently underway.

26.2 Geology and mineralization

Should additional drill data become available as a result of other programs such as metallurgical or geotechnical, the potential for enhancing the understanding of the metallogeny of the deposit and refinement of the geological model should be investigated.

If necessary, this geologic update is estimated to cost approximately C$0.05 million.

26.3 Exploration and Mineral Resource

26.3.1 Exploration

The Sisson deposit has essentially been closed off by drilling to the northeast, northwest, and south. At the present time, Northcliff considers the exploration potential in the immediate vicinity to be limited and has no plans for additional exploration work.

26.3.2 Mineral Resource

If additional drill data becomes available as a result of other programs such as metallurgical or geotechnical, any refinements to the geological model should be included and the additional analytical data be incorporated into a new MRE.

Any MRE update is estimated to cost approximately C$0.15 million.

26.4 Mine planning

26.4.1 Barren rock and PAG characterization

Conduct further geochemical sampling and kinetic testing of the barren rock within the open pit limits. Currently, due to the classification of potentially acid-generating (PAG) material, all barren rock within the pit must be placed in the BRSA within the TSF footprint, requiring a separate quarry to supply material for the downstream TSF embankments. A detailed sampling program to better delineate non-PAG waste rock could allow a larger portion of barren rock to be used in TSF construction, significantly reducing quarrying requirements and overall mining costs. In parallel with geochemical testing, the volumetric swell factor of the barren rock should be validated alongside a detailed, coupled schedule review of the pit stripping sequence versus the TSF embankment construction requirements to mitigate sequencing risks. (C$0.3 million)

26.4.2 Mine geotechnical drilling

The current pit slopes rely on geotechnical parameters established in previous studies. It is recommended to execute an updated geotechnical and hydrogeological drilling program focused on the ultimate pit wall limits. The objective of this program is to obtain additional oriented core data and structural information to optimize and potentially steepen the inter-ramp and overall pit wall angles, which would favorably reduce the life-of-mine strip ratio.

It is understood that Northcliff has engaged a consultant, and that this work currently underway, therefore no additional cost estimate is required.

26.4.3 Mine planning and trade-off studies

During the basic engineering phase, several targeted mine planning optimizations and trade-off studies should be completed to refine the capital and operating cost models:

- **Mine Plan Value Optimization:** The current open-pit limits and production schedule were optimized using conservative Q4 2025 metal price assumptions, whereas the final project financial model

incorporates the higher tungsten prices observed in 2026. It is recommended to conduct a trade-off study to re-optimize the pit limits and schedule using the updated, higher metal prices, which may identify additional economic reserves. Furthermore, these higher prices significantly increase the value of the Mid-Grade Stockpile (MGSP). Under the current plan, 9.2 Mt of mid-grade material becomes permanently encapsulated beneath the tailings. This study should evaluate alternative stockpile locations, elevations, or tailings management strategies, including a dynamic review of the TSF water balance, to allow for the full recovery and processing of this 9.2 Mt of valuable mill feed. (C$0.08 million)

- **Pit Design Updates:** Revise the detailed pit phase designs to incorporate the wider haul roads required by the updated 147 t autonomous truck dimensions. Concurrently with this redesign, evaluate the geometric and economic feasibility of integrating trolley assist infrastructure on the main in-pit ramps to reduce fuel consumption and increase uphill haul speeds. (C$0.12 million)
- **External Haul Road Optimization:** Conduct a detailed alignment optimization of the external haul road network, specifically targeting the primary routes between the pit, the BRSA, and the TSF embankments, to minimize earthwork cut/fill volumes and optimize autonomous cycle times. (C$0.05 million)
- **Quarry Fleet Trade-off:** Complete a trade-off study comparing the base case use of the primary 147 t autonomous haul fleet in the quarry against the use of a dedicated, smaller-capacity (~90 t), manually operated or autonomous truck fleet. A smaller fleet may allow for narrower quarry ramps, reduced footprint and improved shovel/truck matching for the smaller quarry loading units. (C$0.05 million)
- **AHS Site Integration:** Conduct an Autonomous Haulage System site integration study in collaboration with selected vendors. This study will finalize the communication network layout, identify required Wi-Fi/LTE trailer locations, and define the specific protocols for the Autonomous Operating Zones (AOZ) where light vehicles interact with the autonomous fleet. (C$0.1 million)

The combined mine planning and trade-off studies are estimated to cost approximately C$0.4 million.

26.4.4 Operational readiness and cost validation

To ensure a smooth transition from basic engineering into production, the following operational and financial validation programs are recommended:

- **Grade Control and Reconciliation:** Develop a comprehensive grade-control and operational reconciliation program prior to production. Given the mine plan's reliance on early high-grade targeting and a variable cut-off strategy, establishing precise spatial resolution for ore/barren-rock delineation, blast design, and stockpile management is critical to validating the economic model and minimizing dilution. (C$0.1 million)
- **AHS Operational Readiness and Mixed-Fleet Procedures:** Develop detailed standard operating procedures (SOPs) and mine-control systems specifically tailored to AHS. This includes establishing dispatch protocols, equipment utilization metrics, and strict safety procedures for interactions between the autonomous 147t haul fleet, manually operated loading units, and light vehicles within the AOZ. (C$0.15 million)
- **Equipment Productivity and Cost Validation:** Validate the mine cost model and equipment productivity assumptions using final OEM specifications. Because the Project is introducing AHS, it is essential during detailed engineering to secure updated supplier quotations and validate assumptions regarding autonomous cycle times, fuel consumption, tire wear, and maintenance component lifecycles to further de-risk the sustaining capital and operating cost models. (C$0.1 million)

These operational readiness and cost validation studies are estimated to cost approximately C$0.35 million.

26.5 Metallurgical testing and process recovery

26.5.1 Metallurgical testing and process recovery

The following activities are recommended to support the design of the processing plant beyond the feasibility study:

- Additional review of equipment sizing and selection based on updated mine schedules and geometallurgy outcomes.

Additional metallurgical testing is recommended to support the process plant design and investigate processing opportunities as follows:

- Obtain at least 10 tonnes of drill core dedicated to metallurgical testing.
- Allocate 4 tonnes of bulk sample to follow up ore sorting testing, in which the material is stage crushed to a size distribution that is reasonably representative of a secondary crushing circuit. Conduct full scale sorter tests on the coarse fractions. Utilize the fines and sorter accepts for downstream metallurgical testing.
- Conduct a semi-autogenous grinding test designed to generate a pebble stream hosting 25-30% of the feed mass. Conduct DE-XRT sorting tests on the pebbles and subsequent metallurgical testing on the combined fines and sorter accepts.
- Investigate inclusion of gravity concentration in conjunction with tungsten flotation.
- Conduct pilot scale testing to generate bulk quantities of Mo rougher concentrate and sulphide tailings for downstream testing.
- Conduct a regrinding energy assessment on a suitable mass of molybdenum 1st cleaner concentrate.
- Conduct a variability test program using suitable sample masses to confirm the developed flowsheet and develop a recovery model that relates metallurgical performance to feed characteristics.
- Conduct mineralogical assessments on variability feed samples as well as rougher tailings and intermediate concentrates to better understand the effect of mineral assemblage on metallurgical performance.
- Conduct testwork with flotation technologies specifically designed for fine particle recovery.

The testing and engineering activities described above are estimated to cost C$0.95 million.

26.5.2 Particle ore sorting

Further evaluation of particle ore sorting is recommended as a potential future opportunity for the Sisson Project. Additional representative bulk-scale testwork and engineering studies should be undertaken to confirm sorting performance, optimize operating parameters, assess sorter rejects and develop the required material handling and sorting circuit. The technical and economic evaluation should consider potential impacts on mill feed grade, mine cut-off grade, mining and processing rates, plant capacity, capital and operating costs, energy consumption, and overall project economics. The potential use of sorter rejects as non-potentially acid generating (non-PAG) material for tailings co-disposal or aggregate should also be investigated.

The recommended ore sorting evaluation is estimated to cost approximately C$0.2 million.

26.6 Infrastructure

26.6.1 Site access, logistics and geotechnical investigation

During the Basic Engineering phase, complete a coordinated review and validation of the existing project access, logistics and supporting geotechnical information, supplemented by targeted additional investigation where required, to confirm the infrastructure and logistics basis required to support construction and future mine operations.

The review should build on the assessment and investigation work completed for the 2015 Sisson EIA and subsequent project studies, with particular emphasis on confirming that the previous work remains applicable to the current project configuration, construction methodology and execution strategy.

Recommended work includes:

- Review and validate the Primary Site Access Road (PSA) and Secondary Site Access Road (SSA), building on the assessment work completed for the 2015 Sisson EIA. The review should confirm the current alignment, road geometry, travel times, seasonal constraints, drainage requirements, required upgrades and suitability for construction traffic and long-term operations.
- Complete targeted condition and capacity assessments of bridges and major culverts along the proposed access routes, as required, including confirmation of their suitability for heavy-haul transportation and major equipment deliveries.
- Complete targeted additional geotechnical investigations along the site access roads and within the process plant and major infrastructure areas, as required, to address gaps identified through review of the previous investigations and reduce remaining uncertainty associated with foundations, earthworks and road construction.
- Complete a project logistics assessment addressing oversized and heavy-haul transportation, bridge capacities, seasonal access constraints, delivery routes, receiving and offloading areas, laydown requirements and construction traffic management, building on previous assessment work where applicable.
- Integrate the findings into the Early Works scope, site development plan, logistics planning, and integrated project execution schedule.

The recommended site access, logistics, and targeted additional geotechnical investigation work is estimated to cost approximately C$1.3 million.

26.6.2 Site optimization and value engineering

During early Basic Engineering, complete a multidisciplinary site optimization and value engineering review to identify opportunities to reduce capital cost, improve constructability and protect the Project schedule.

The review should consider:

- overall process plant and infrastructure arrangement;
- major earthworks and cut/fill balance;
- internal haul and travel distances;
- construction access, laydown, crane access and material handling;
- utility routing and major service corridors;

- permanent and temporary facilities;
- interfaces between the process plant, mine infrastructure, TSF, water management and off-site infrastructure;
- winter construction requirements and opportunities to improve seasonal productivity;
- opportunities for prefabrication, modularization or pre-assembly where economically justified; and
- sequencing opportunities that could reduce schedule duration or construction congestion.

The review should be completed early enough in basic engineering so that identified opportunities can be incorporated before layouts and major design interfaces are frozen.

The recommended site optimization and value engineering is estimated to cost approximately C$0.35 million.

26.6.3 Fresh water and potable water supply

Complete additional investigation and testing to confirm the permanent fresh water and potable water supply basis for the Project.

Recommended work includes:

- complete additional well testing and hydrogeological assessment, as required, to confirm sustainable yield;
- confirm raw water quality and seasonal variability;
- confirm potable water treatment requirements and design criteria;
- confirm the preferred source, routing and infrastructure required to supply the process plant and site facilities; and
- integrate the confirmed supply basis with the site-wide water balance, utility design and permitting requirements.

The recommended freshwater and potable water supply is estimated to cost approximately $0.45 million.

26.6.4 Power supply and electrical infrastructure development

Advance the electrical infrastructure and NB Power interfaces during Basic Engineering to confirm the design and execution requirements for permanent project power.

Recommended work includes:

- confirm the project electrical load basis and required capacity of the 138 kV supply system;
- advance coordination with NB Power for the transmission line, Keswick Terminal interface and project connection requirements;
- advance the main substation and site distribution design sufficiently to confirm equipment ratings, interfaces, civil requirements and energization sequence;
- confirm the relationship between temporary construction power and permanent project power;
- identify critical electrical equipment and interfaces that could constrain construction, pre-commissioning or commissioning; and
- align required engineering and procurement dates with the integrated project execution schedule.

The recommended power supply and electrical infrastructure development is estimated to cost approximately C$0.75 million.

26.6.5 Tailings storage facility

The following work is recommended for completion during the basic engineering phase and before detailed design to address the risks associated with the TSF, and to update the water quality predictive model:

- Carry out the next phase of geotechnical and hydrogeological site investigation at the TSF, including:
  - additional boreholes in the overburden and bedrock along the embankment alignment to increase the resolution within foundations; and
  - laboratory testing of soil and rock samples to extend the database of strength and permeability data.
- Carry out additional geotechnical and geochemical investigations and laboratory testing of potential borrow sources for embankment construction, focusing on fine-grained materials for the low permeability zones within the TSF embankments.
- Carry out additional geotechnical, geochemical, and mineralogical testing on representative tailings samples generated from future metallurgical testing. Include testing of a representative range of tailings types resulting from the expected variability in ore type, grind size, etc., that may be produced during the operating life of the Project.
- Extend the geochemical source term database for the tailings solids and associated process water to better represent the potential range of conditions that may be encountered during the operating life of the Project.
- Review and update the seismic hazard assessment study for the project site.
- Carry out a dam breach inundation study for the TSF.
- Update the reclamation and closure plan for the Project.

The following work is recommended for completion during the basic engineering phase to address the risks associated with quarry material geochemical characterization uncertainty, it is recommended that Northcliff engage a qualified geochemical consultant to address the risk associated with the limited quarry geochemical data set. It is understood that Northcliff has engaged a consultant and planning is advancing to complete this work.

- Evaluate water quality model sensitivity to variable embankment fill, quarry highwall, and waste rock source terms for higher-arsenic material to bound potential outcomes.
- Define a quarry drilling and sampling program that is sufficient to define arsenic spatial distribution within the proposed quarry footprint, and in potential expansion areas if the initial results show significant variability.
- Select representative higher-arsenic intervals from the drilling and sampling program for kinetic (humidity cell) testing to generate source term leachability data.
- Update the quarry rock, dam fill, and potential quarry waste rock source terms using new test work results; re-run the predictive water quality model to evaluate impacts on water chemistry, WTP design, and environmental effects. The current model results should not be used to evaluate environmental effects until source terms are updated.

- Establish a management plan, including protocols for screening extracted material and criteria for acceptance/rejection to confirm that geochemically acceptable rock is used for dam construction, with unsuitable material diverted to a suitable waste rock storage area, e.g., the BRSA within the TSF.
- If testing confirms significant unsuitable material, identify, sample, and test alternative quarry sources. Assess implications for the existing environmental assessment and future permitting.
- Assess opportunities to beneficially use potentially elevated-arsenic quarry rock in the upstream zones of the TSF embankment where barren rock is currently placed (i.e., Zone C2) and evaluate the effects using an updated predictive water quality model.

Table 26-2 presents the estimated costs for the recommended work to be completed during the basic engineering phase to address risks associated with the TSF and the quarry.

Table 26-2: Costs for recommended tailings storage facility work program

Tailings storage facility	Estimated Total Cost (C$M)
Geotechnical and hydrogeological site characterization	1.0
Tailings characterization	0.15
Water balance and water quality	0.30
Seismic hazard assessment	0.10
Dam breach inundation study	0.50
Updated reclamation closure plan	0.35
Quarry characterization and management plan	0.5
Total	**2.9**

26.6.6 Water treatment facilities

Recommendations related to the next stages of the water treatment design are as follows:

- Incorporate expected WTP flows (min, nominal, and max) and full suite water quality predictions (P50 and P95) for the closure/post-closure phases based on updated closure plan, hydrology and hydrogeology, water balance, storage, routing and operating assumptions into the WTP design criteria.
- Incorporate detailed receiver water quality assumptions for all phases into the WTP design basis including applicable discharge requirements, including MDMER Schedule 4 criteria, acute lethality requirements, New Brunswick provincial requirements, CCME aquatic life guidelines, mixing zone assumptions, and any project-specific approval conditions (as applicable).
- Confirm source terms and conditions for the operational and closure/post-closure phases.
- Conduct bench or pilot testing on raw water to:
  - validate the design assumptions;
  - confirm cyanide and thiosalt oxidation performance;
  - confirm metals, TSS and calcite solids removal performance;
  - confirm treated effluent quality, including parameters relevant to acute lethality;
  - validate engineering criteria such as reagent demand, reaction time, solids settling, sludge generation, and sludge handling assumptions;
  - refine capital and operating cost inputs; and

- complete WTP engineering definition with diffident detail support permitting and implementation planning.

The above recommendations are estimated to cost approximately C$0.8 million.

26.7 Project execution and procurement

26.7.1 Procurement, contracting, and long-lead equipment strategy

Develop a Procurement and Contracting Management Plan early in Basic Engineering to define the procurement and contracting approach required to support project execution.

The plan should address:

- construction and supply package strategy;
- proposed commercial approach by package type;
- tendering and award sequence;
- local and regional contractor and supplier capabilities;
- procurement governance and approval requirements;
- vendor data requirements;
- expediting, inspection and logistics requirements; and
- identification and management of long-lead equipment and materials.

For selected long-lead equipment packages, evaluate the benefit of making limited early commitments for vendor engineering and certified design information ahead of full equipment purchase. Where justified, this could allow foundation, structural, electrical, controls and layout engineering to advance while limiting Northcliff's near-term capital commitment.

This approach should be focused on equipment where the design basis is sufficiently mature and vendor information is required to protect downstream engineering or construction dates.

The recommended procurement, contracting, and long-lead equipment strategy development is estimated to cost approximately C$0.4 million.

26.7.2 Early works definition and execution readiness

During Basic Engineering, further define the Early Works scope and authorization strategy required to protect the overall project execution schedule.

The Early Works review should consider:

- PSA, SSA, Realigned Fire Road and internal site-access development;
- contractor quarry development and availability of construction materials;
- initial site clearing and preparation;
- temporary construction services and power;
- receiving, laydown and construction logistics areas;
- early water management and erosion/sediment control works;
- enabling works required for TSF and process plant construction; and

- engineering, permitting, procurement and authorization dates required to release the work.
- The recommended Early Works scope should be integrated into the project execution schedule and clearly identify activities that may require authorization ahead of the main construction decision.
- The recommended early works definition and execution readiness is estimated to cost approximately C$0.55 million.

26.8 Environmental studies

Recommendations related to environmental, permitting and social considerations include the following:

- Continue with Indigenous, public, and stakeholder engagement programs that have been ongoing since 2010.
- Continue environmental monitoring baseline data collection and reporting, as required by the Project's provincial EIA approval, and additional requirements issued by NBDLEG upon extension of the Project (see Section 2.3 for additional detail). These include, but are not limited to:
- Spawner surveys of Atlantic Salmon
- Wetland compensation planning
- Water source supply assessment
- Baseline surveys of traditional country foods
- Dust deposition modelling
- Updated greenhouse gas inventory and ambient air dispersion modelling
- Updated breeding bird and wetland field surveys
- Develop partnerships with local education institutions
- Implement contracting and hiring policies that prioritize local benefits.

An estimated budget of approximately C$1.0 million is recommended for environmental studies.

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