New Polaris Project
NI 43-101 Technical Report & Feasibility Study

British Columbia, Canada

Effective Date: July 21, 2025
Report Date: September 2, 2025

Prepared for:

Canagold Resources Ltd
625 Howe St, Suite 1250
Vancouver, BC, Canada, V6C 2T6

Prepared by:

Ausenco Engineering Canada Inc.
1050 West Pender Street, Suite 1200
Vancouver, BC, Canada, V6E 3S7

List of Qualified Persons:

Tommaso Roberto Raponi, P Eng., Ausenco Engineering Canada ULC
Kevin Murray, P. Eng., Ausenco Engineering Canada ULC
Jonathan Cooper, P. Eng., Ausenco Engineering Canada ULC
James Millard, P Geo., Ausenco Sustainability ULC
Mohammad Ali Hooshiar Fard, P. Eng., Ausenco Engineering Canada ULC
Sue Bird, P.Eng., Moose Mountain Technical Services Ltd
Dino Pilotto, P Eng., JDS Energy & Mining Inc
Michael Levy, P Eng., JDS Energy & Mining Inc

CERTIFICATE OF QUALIFIED PERSON
Tommaso Roberto Raponi, P. Eng.

I Tommaso Roberto Raponi, P.Eng., certify that:

1. I am employed as a Principal Metallurgist with Ausenco Engineering Canada ULC, (Ausenco), with an office address of Suite 1550 - 11 King St West, Toronto, ON M5H 4C7.
2. This certificate applies to the technical report titled "New Polaris Gold Project, NI 43-101 Technical Report & Feasibility Study, British Columbia, Canada" (the "Technical Report") prepared for Canagold Resources Ltd (the "Company"), with a Report Date of September 2, 2025, and an effective date of July 21, 2025 (the Effective Date").
3. I graduated from the University of Toronto with a Bachelor of Applied Science degree in Geological Engineering with specialization in Mineral Processing in 1984.
4. I am a Professional Engineer registered with the Professional Engineers of Ontario (license No. 90225970), the Engineers and Geoscientists of British Columbia (license No. 23536), and the Northwest Territories and Nunavut Association of Professional Engineers and Geoscientists (license No. L4508).
5. I have practiced my profession continuously for over 40 years with experience in the development, design, operation and commissioning of mineral processing plants, focusing on gold projects, both domestic and internationally. My project design and development experience include the generation of capital and operating costs for mineral processing plants and associated infrastructure and financial modeling of project economics.
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 those sections of the Technical Report that I am responsible for preparing.
7. I have not visited the project site.
8. I am responsible for Sections 1.8, 12.2, 13, 25.4, 25.14.1.1, 25.14.2.1, 26.2 and 27 of the Technical Report.
9. I am independent of Canagold Resources Ltd, as independence is defined in Section 1.5 of NI 43-101.
10. I have not been previously involved with the New Polaris 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: September 2, 2025

"Signed and sealed"

Tommaso Roberto Raponi, P.Eng.
Ausenco Engineering Canada ULC Permit to Practice 1001905, Engineers and Geoscientists British Columbia

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, V6E 3S7.
2. This certificate applies to the technical report titled "New Polaris Gold Project, NI 43-101 Technical Report & Feasibility Study, British Columbia, Canada" (the "Technical Report") prepared for Canagold Resources Ltd (the "Company"), with a Report Date of September 2, 2025, and an effective date of July 21, 2025 (the Effective Date").
3. I graduated from the 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 gold flotation concentrate production studies including Skeena's Eskay Creek and Seabridge Gold's Courageous Lake projects.
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 those sections of the Technical Report that I am responsible for preparing.
7. I have not visited the project site.
8. I am responsible for Sections 1.1, 1.12, 1.13.1, 1.14, 1.16-1.19, 2.1-2.3, 2.5-2.8, 3.1, 3.4-3.5, 17, 18.1.1, 18.2-18.7, 19, 21.1, 21.2.1-21.2.2, 21.2.4-21.2.10, 21.3.1-21.3.2, 21.3.4-21.3.5, 22, 24, 25.1, 25.8-9, 25.11-13, 25.14.1.4.1, 25.14.1.6-25.14.1.7, 25.14.2.3, 25.14.2.4.2, 26.1, 26.5.2 and 27 of the Technical Report.
9. I am independent of Canagold Resources Ltd, as independence is defined in Section 1.5 of NI 43-101.
10. I have not been previously involved with the New Polaris 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: September 2, 2025

"Signed and sealed"

Kevin Murray, P.Eng.

Ausenco Engineering Canada ULC Permit to Practice 1001905, Engineers and Geoscientists British Columbia

CERTIFICATE OF QUALIFIED PERSON
Jonathan Cooper, P. Eng.

I, Jonathan Cooper, P. Eng., certify that:

1. I am employed as a Water Resources Engineer with Ausenco Sustainability ULC ("Ausenco"), with an office address of 11 King Street West, Suite 1500, Toronto, Ontario M5H 4C7.
2. This certificate applies to the technical report titled "New Polaris Gold Project, NI 43-101 Technical Report & Feasibility Study, British Columbia, Canada" (the "Technical Report") prepared for Canagold Resources Ltd (the "Company"), with a Report Date of September 2, 2025, and an effective date of July 21, 2025 (the Effective Date").
3. I graduated from the University of Western Ontario with a Bachelor of Engineering Science in Civil Engineering in 2008, and University of Edinburgh with a Master of Environmental Management in 2010.
4. I am a Professional Engineer registered and in good standing Professional Engineers Ontario (registration #100191626), Engineers and Geoscientists British Columbia (registration #37864) and Northwest Territories and Nunavut Association of Professional Engineers and Geoscientists (registration # L4227).
5. I have practiced my profession continuously for over 15 years with experience in the development, design, operation, and commissioning of surface water infrastructure. Previous projects that I have worked on that have similar features to the North Island Project are the Kwanika-Stardust for NorthWest Copper located in British Columbia; Colomac Gold Project located in the Northwest Territories and the Crawford Project located in Ontario.
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 those sections of the Technical Report that I am responsible for preparing.
7. I visited the project site from September 17 to 24, 2023.
8. I am responsible for Sections 1.13.2, 2.4.1, 18.9, and 27 of the Technical Report.
9. I am independent of Canagold Resources Ltd, as independence is defined in Section 1.5 of NI 43-101.
10. I have been previously involved with the New Polaris Project; I was a co-contributor on the ongoing Environmental Assessment.
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: September 2, 2025

"Signed and sealed"

Jonathan Cooper, P. Eng.

Ausenco Sustainability ULC Permit to Practice 1003471, Engineers and Geoscientists British Columbia

CERTIFICATE OF QUALIFIED PERSON
James Millard, P. Geo.

I, James Millard, P. Geo., certify that:

1. I am employed as a Director, Strategic Projects with Ausenco Sustainability Inc ("Ausenco"), with an office address of Suite 100, 2 Ralston Avenue, Darthmouth, NS B3B 1H7, Canada.
2. This certificate applies to the technical report titled "New Polaris Gold Project, NI 43-101 Technical Report & Feasibility Study, British Columbia, Canada" (the "Technical Report") prepared for Canagold Resources Ltd (the "Company"), with a Report Date of September 2, 2025, and an effective date of July 21, 2025 (the Effective Date").
3. I graduated from Brock University in St. Catharines, Ontario in 1986 with a Bachelor of Science in Geological Sciences, and from Queen's University in Kingston, Ontario in 1995 with a Master of Science in Environmental Engineering.
4. I am a member (P. Geo.) of the Association of Professional Geoscientists of Nova Scotia, Membership No. 021 and the Association of Professional Engineers, Geologists and Geophysicists of the Northwest Territories and Nunavut, Membership No. 1624.
5. I have practiced my profession for over 30 years. I have worked for mid- and large-size mining companies where I have acted in senior technical and management roles, in senior environmental consulting roles, and provided advise and/or expertise. These key areas include feasibility-level study reviews; NI 43-101 report writing and review; due diligence review of environmental, social, and governance areas for proposed mining operations and acquisitions, and directing environmental impact assessments and permitting applications to support construction, operations, and closure of mining projects. In addition to the above, I have been responsible for conducting baseline data assessments, surface and groundwater quantity and quality studies, mine rock geochemistry and water quality predictions, mine reclamation and closure plan development, and community stakeholder and Indigenous peoples' engagement initiatives. Recently, I acted as Qualified Person for environmental/sustainability sections in the following project reports: "Volcan Project, NI 43-101 Technical Report on Preliminary Economic Assessment, Tierra Amarilla, Atacama Region, Chile"; "Colomac Gold Project, NI 43-101 Technical Report and Preliminary Economic Assessment, Northwest Territories, Canada"; "Santo Tomás Copper Project, NI 43-101 Technical Report and Preliminary Economic Assessment, Northern Sinaloa State, Mexico"; "Lemhi Gold Project, NI 43-101 Technical Report and Preliminary Economic Assessment, Idaho, USA"; "Tolillar Project NI 43-101 Technical Report on Preliminary Economic Assessment, Salta Argentina"; "Santo Domingo Project NI43-101 Technical Report on Feasibility Study Update, Atacama Region, Chile"; "Cerro Las Minitas Project NI 43-101 Technical Report Preliminary Economic Assessment, Durango State, Mexico"; and, "Panuco Project NI 43-101 Technical Report and Preliminary Economic Assessment, Sinaloa State, Mexico."
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 those sections of the Technical Report that I am responsible for preparing.
7. I have not visited the project site.
8. I am responsible for Sections 1.15, 3.3, 20, 25.10, 25.14.1.5, 25.14.2.5, 26.6, and 27 of the Technical Report.
9. I am independent of Canagold Resources Ltd, as independence is defined in Section 1.5 of NI 43-101.

10. I have not been previously involved with the New Polaris 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: September 2, 2025

"Signed and sealed"

James Millard, P. Geo.
Ausenco Sustainability ULC Permit to Practice 1003471, Engineers and Geoscientists British Columbia

CERTIFICATE OF QUALIFIED PERSON
Mohammad Ali Hooshiar Fard, P. Eng.

I, Mohammad Ali Hooshiar Fard, P. Eng., certify that:

1. I am employed as a Geotechnical Engineer with Ausenco Engineering Canada ULC (Ausenco), with an office at 1050 W Pender St, Vancouver, BC V6E 3S7.
2. This certificate applies to the technical report titled "New Polaris Gold Project, NI 43-101 Technical Report & Feasibility Study, British Columbia, Canada" (the "Technical Report") prepared for Canagold Resources Ltd (the "Company"), with a Report Date of September 2, 2025, and an effective date of July 21, 2025 (the Effective Date").
3. I graduated from Sharif University of Technology with a BSc and an MSc in Materials Science and Engineering in 2003 and 2006, respectively. In 2011, I graduated from the University of Alberta with a PhD in Materials Engineering.
4. I am a Professional Engineer registered with the Engineers and Geoscientists British Columbia (No. 40965) and Engineers Yukon (3343).
5. I have practiced my profession for 21 years. My relevant experience includes designing tailings and waste rock storage facilities as well as managing geotechnical field investigation and lab testing programs for mining projects across the globe. I have acted as the Qualified Person for similar projects, such as:
   a. Berg Project Preliminary Economic Assessment
   b. Marban Project Prefeasibility Study
   c. Val-d'Or East Project Preliminary Economic Assessment

   A summary of the more recent portion of my professional career is as follows:

a. Geotechnical Mining Engineer, Ausenco, Canada	2018–present
b. Geotechnical Mining Engineer, AECOM, Canada	2013–2017
c. Senior Geotechnical Consultant, SRK Consulting Inc., Canada	2011–2013

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 those sections of the Technical Report that I am responsible for preparing.
7. I have not visited the project site.
8. I am responsible for Sections 1.13.3, 18.1.2, 18.8, 25.14.1.4.2, 25.14.2.4.1, 26.5.1, and 27 of the Technical Report.
9. I am independent of Canagold Resources Ltd, as independence is defined in Section 1.5 of NI 43-101.
10. I have no previous involvement with the New Polaris 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: September 2, 2025

"Signed and sealed"

Mohammad Ali Hooshiar Fard, P.Eng.
Ausenco Engineering Canada ULC Permit to Practice 1001905, Engineers and Geoscientists British Columbia

CERTIFICATE OF QUALIFIED PERSON
Sue Bird, P.Eng.

I, Sue Bird, P.Eng., certify that:

1. I am a Geological and Mining Engineer with Moose Mountain Technical Services, with a business address of #210 1510 2nd St North Cranbrook BC, V1C 3L2.
2. This certificate applies to the technical report titled "New Polaris Gold Project, NI 43-101 Technical Report & Feasibility Study, British Columbia, Canada" (the "Technical Report") prepared for Canagold Resources Ltd (the "Company"), with a Report Date of September 2, 2025, and an effective date of July 21, 2025 (the Effective Date").
3. I graduated with a Geologic Engineering degree (B.Sc.) from the Queen's University in 1989 and a M.Sc. in Mining from Queen's University in 1993.
4. I am a Professional Engineer in the Province of British Columbia. (#25007).
5. I have worked as an engineering geologist for over 30 years since my graduation from university. I have worked on precious metals, base metals and coal mining projects, including mine operations, mineral resource estimates and due diligence evaluations.
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 those sections of the Technical Report that I am responsible for preparing.
7. I visited the New Polaris Project on August 25, 2022, for one day.
8. I am responsible for Sections 1.2-1.7, 1.9, 2.4.2, 3.2, 4-11, 12.1, 14, 23, 25.2-25.3, 25.5, 26.3, and 27 of the Technical Report.
9. I am independent of Canagold as independence is described by Section 1.5 of NI 43–101.
10. I previously co-authored as QP for Section 14 the 2019 report entitled: "The New Polaris Gold Project, British Columbia, Canada 2019 Preliminary Economic Assessment", the 2023 report entitled "The New Polaris Gold Project, British Columbia, Canada, 2023 Resource Estimate Update." and the 2025 report entitled "The New Polaris Gold Project, British Columbia, Canada 2025 Resource Estimate Update" with an effective date of January 22, 2025.
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.
12. 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: September 2, 2025

"Signed and sealed"

Sue Bird, P.Eng.
Moose Mountain Technical Services Permit to Practice 1003309, Engineers and Geoscientists British Columbia

CERTIFICATE OF QUALIFIED PERSON
Dino Pilotto, P.Eng.

I, Dino Pilotto, P.Eng., certify that:

1. I am employed as a General Manger of Technical Services with JDS Energy & Mining Inc., (JDS), with an office address of 900 - 999 West Hastings St., Vancouver, BC V6C 2W2.
2. This certificate applies to the technical report titled "New Polaris Gold Project, NI 43-101 Technical Report & Feasibility Study, British Columbia, Canada" (the "Technical Report") prepared for Canagold Resources Ltd. (the "Company"), with a report date of September 2, 2025, and an effective date of July 21, 2025 (the Effective Date").
3. I graduated from University of British Columbia with a Bachelor of Applied Science in Mine and Mineral Process Engineering in May 1987.
4. I am a professional engineer registered with the Professional Engineers and Geoscientists British Columbia (No. 198167) and Yukon Professional Engineers and Geoscientists (No. 2527).
5. I have practiced my profession continuously since June 1987. I have been involved with mining operations, mine engineering and consulting covering a variety of commodities at locations in North America, South America, Africa, and Eastern Europe.
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 those sections of the Technical Report that I am responsible for preparing.
7. I have visited the project site on June 3, 2025, for 1 day.
8. I am responsible for Sections 1.10, 1.11, 2.4.3, 15, 16.1, 16.3–16.13, 21.2.3, 21.3.3, 25.6-25.7, 25.14.1.2, 25.14.2.2, 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 not been previously involved with the 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: September 2, 2025

"Signed and sealed"

Dino Pilotto, P.Eng.
JDS Energy & Mining Inc, Permit to Practice 1001427, Engineers and Geoscientists British Columbia

CERTIFICATE OF QUALIFIED PERSON
Michael Levy, P.Eng.

I, Michael E. Levy, P.E., certify that:

1. I am employed as Geotechnical Manager with JDS Energy & Mining Inc. with an office at Suite 900 - 999 West Hastings St, Vancouver, BC V6C 2W2.
2. This certificate applies to the technical report titled "New Polaris Gold Project, NI 43-101 Technical Report & Feasibility Study, British Columbia, Canada" (the "Technical Report") prepared for Canagold Resources Ltd. (the "Company"), with a report date of September 2, 2025, and an effective date of July 21, 2025 (the Effective Date").
3. I hold a bachelor's degree (B.Sc.) in Geology from the University of Iowa in 1998 and a Master of Science degree (M.Sc.) in Civil-Geotechnical Engineering from the University of Colorado in 2004.
4. I am a registered Professional Engineer (P.E.) in the states of Colorado (#40268) and registered Professional Engineer (P.Eng.) in the province of British Columbia (#216542). I am a current member of the Society for Mining, Metallurgy & Exploration (SME) and the American Society of Civil Engineers (ASCE).
5. I have practiced my profession continuously for 26 years (since 1999) and have been involved in numerous mining geotechnical projects across the Americas.
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 those sections of the Technical Report that I am responsible for preparing.
7. I personally visited the property that is the subject to the Technical Report on June 3, 2025, for 1 day.
8. I am responsible for Sections 2.4.4, and 16.2 of the Technical Report.
9. I am independent of Canagold Resources, Ltd., as independence is defined in Section 1.5 of NI 43-101.
10. I have had no prior involvement with the property that is the subject of the Technical Report.
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: September 2, 2025

"Signed and sealed"

Michael Levy, P. Eng.
JDS Energy & Mining Inc, Permit to Practice 1001427, Engineers and Geoscientists British Columbia

Important Notice

This report was prepared as National Instrument 43-101 101 Technical Report for Canagold Resources Ltd (Canagold) by Ausenco Engineering Canada ULC and Ausenco Sustainability ULC (Ausenco), Moose Mountain Technical Services Ltd, and JDS Energy & Mining Inc., collectively the Report Authors. The quality of information, conclusions, and estimates contained herein is consistent with the level of effort involved in the Report Authors' services, based on i) information available at the time of preparation, ii) data supplied by outside sources, and iii) the assumptions, conditions, and qualifications set forth in this report. This report is intended for use by Canagold, 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 are at that party's sole risk.

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1 SUMMARY

1.1 Introduction

Canagold Resources Ltd (Canagold) commissioned Ausenco Engineering Canada ULC ("Ausenco Engineering") to compile a Feasibility Study (FS) of the New Polaris Project (the Project). The FS was prepared in accordance with the Canadian disclosure requirements of National Instrument 43-101 (NI 43-101) and in accordance with the requirements of Form 43-101 F1.

The responsibilities of the engineering companies who were contracted by Canagold to prepare this report are as follows:

- Ausenco Engineering managed and coordinated the work related to the report and developed FS-level design and cost estimate for the process plant, general site infrastructure, and economic analysis.
- Ausenco Engineering completed geotechnical studies, site-wide water balancing, and developed the FS-level design and cost estimate of the co-storage facility for tailings and waste rock.
- Ausenco Sustainability ULC (together with Ausenco Engineering, "Ausenco") carried out an assessment of the site-wide water management, including sizing and designing water management related structures for the waste rock stockpiles and process area.
- JDS Energy & Mining Inc. (JDS) designed the underground mine, mine production schedule, and mine capital and operating costs.
- Moose Mountain Technical Services (MMTS) completed the work related to property description, accessibility, local resources, geological setting, deposit type, exploration work, drilling, mineral resource estimate and QA/QC.
- Tanka Engineers LLP (Tanka) provided a third party review and optimization of the civil design, access road alignment and process equipment pricing.

The proposed project is an underground gold mine with an anticipated production rate of approximately 1,000 tonnes per day (t/d). The proposed mine will operate year-round, processing, on average, approximately 365,000 tonnes per year (t/a) of ore. The mine life will be 8.1 years.

A Mineral Resource Estimate (MRE) and a Preliminary Economic Assessment (PEA) on the Project were completed in 2007. The PEA was updated in 2009 and again in 2011, using the same Mineral Resource data from 2007, but incorporating revised gold prices and updated capital and operating cost estimates. An updated MRE and PEA were completed in 2019. Another MRE update followed in 2023. The most recent MRE update, completed in 2025 and presented herein, builds on the 2023 MRE by incorporating results for antimony, arsenic and sulphur.

1.2 Project Description and Location

The New Polaris Project (formerly Polaris-Taku Mine) is an early tertiary mesothermal gold mineralized body located in northwestern British Columbia approximately 100 kilometers (km) south of Atlin, BC, and 60 km northeast of Juneau, Alaska. The nearest roads in the area end 20 km south of Atlin, and approximately 100 km from the Project. Currently, access is by aircraft, with a small airstrip for light aircraft located on the property. In the past, shallow draft barges were used to access the site via the Taku River, allowing the transport of bulk supplies and heavy equipment to the property as well as shipment of flotation concentrate from the site.

The project area lies on the eastern flank of the steep, rugged, Coast Range Mountains, with elevations ranging from sea level to 2,600 meters above sea level (m asl). The climate is characterized by heavy rainfall during the late summer and fall months, and comparatively heavy snowfall during the winter, often interspersed with periods of rain.

1.3 Mineral Tenure, Royalties, and Agreements

The property consists of 61 contiguous Crown granted mineral claims and one modified grid claim covering 1,108 hectares (ha). All claims are 100% owned and held by New Polaris Gold Mines Ltd, a wholly owned subsidiary of Canagold Resources Ltd (Canagold).

Canagold is challenging the validity of the historic 15% net profit interest (that can be reduced to 10% with the issuance of 150,000 Canagold shares) on the project.

1.4 History

The deposit was mined by underground methods from 1938 to 1942, and from 1946 to early 1951, producing a total of 740,000 tonnes (t) of ore at an average grade of 10.3 grams per tonne (g/t) gold (Au). Recent exploration work, since 1988, has been directed at gaining knowledge about the geology of the area and expanding the resource base of the mineralized zones. Geological mapping, geochemical surveys, geophysical techniques, and drilling have expanded the resources at the Project.

1.5 Geology and Mineralization

The deposit is composed of three sets of veins (quartz-carbonate stringers in altered rock), the “A-B” veins are northwest striking and southwest dipping, the “Y” veins are north striking and dipping steeply east and finally the “C” veins are east-west striking and dipping to the south to southeast at 65º to vertical. The “C” veins appear to hook around to the north and south into the other two sets of veins so that their junctions form an arc. Gold is refractory and occurs primarily as finely disseminated grains with arsenopyrite that mineralizes the altered wallrock. The next most abundant sulphide mineral is pyrite, followed by minor stibnite and trace amounts of sphalerite. Mineralized zones range from 15 to 250 m in length and from 0.3 to 14 m in width.

1.6 Exploration

The New Polaris property represents an advanced exploration project on a former gold producer. Surface mapping, geochemistry and geophysics over the "mine wedge" were completed by Orequest in 1988 and further surface mapping and geochemistry, on the "north grid" were completed in 1993. The early exploration work in the area located gold mineralization on surface and subsequent exploration led to mining of approximately 689,500 t of material grading 10 g/t of gold. More recent exploration work, since 1988, has been directed at gaining knowledge about the geology of the area and expanding the resource base of the mineralized zones.

1.7 Drilling and Sampling

Canarc (now Canagold) explored the "C" vein system between 1988 and 1997 and carried out infill drilling from 2003 through 2006, as well as in 2021 and 2022, to better define the continuity and grade of the vein systems. Drilling done in 2023-2024 was completed outside the current resource area and does not affect the 2023 estimate. Vein sampling was performed using wireline diamond drilling with NQ-size core. True widths of the mineralized zones vary from 70% to 100% of the drill core intercept angle.

The Quality Assurance and Quality Control (QA/QC) program was consistent across the above-mentioned campaigns. Samples were collected on site by Canagold personnel and shipped to the ALS Chemex laboratory in Vancouver. As part of QA/QC procedures, core samples were regularly accompanied by blanks, duplicates, and certified reference standards. Field procedures were carried out efficiently and in accordance with accepted engineering standards.

Sample preparation, analysis, and security procedures undertaken by Canagold generally follow industry standards and accepted exploration best practices. Sufficient verification checks have been completed on the databases to provide confidence that they are reasonably free of errors and may be used to support Mineral Resource estimation.

1.8 Mineral Processing and Metallurgical Testwork

The New Polaris Project has a long history of metallurgical testing. The flotation flowsheet is based on the flowsheet used during historical production in the 1940's. In 2022, a flotation pilot plant was used to generate flotation concentrate (using this flowsheet) for BIOX™ pilot testing. Preliminary testing completed in 2019 showed the BIOX™ process was successful in liberating gold from refractory concentrate for recovery with conventional cyanidation, with gold extraction of 96%. The pilot plant operation showed that higher antimony concentrations required longer oxidation retention time at very dilute slurry densities. The arsenopyrite content of the concentrate also requires external acid addition. At full scale, these conditions would make the process uneconomic, and the testing program was terminated. In 2024, flotation testing with this flowsheet (rougher flotation with two stages of cleaning) was used to generate recovery data for the production of flotation concentrates for sale to third parties. The concentrate will contain arsenic at deleterious concentrations that may factor into its marketability. Life-of-mine average gold recovery is 88.9% at average concentrate grade of 128 g/t gold and 12.3% arsenic.

1.9 Mineral Resource Estimate

An updated MRE has been prepared in 2025. The updated resource estimate uses all available drillhole data within the previously modeled resource estimate shapes, with historical data compared to and validated with recent drilling. The Au resource is based on 1,692 assay intercepts from 234 drillholes which intersect the veins within the data set. The subset of antimony resource updated in this report is based on 6,660 assay intercepts from 215 drillholes. Inverse distance squared (ID2) has been used to interpolate the gold and antimony grade of the veins which were modeled by MMTS using Implicit modeling.

The effective date is this Resource Estimate is January 22, 2025, which includes the antimony in addition to the base case gold resource. Mineral resources that are not mineral reserves do not have demonstrated economic viability.

Table 1-1: Updated Mineral Resource Estimate and Comparison to the 2019 Resource

Class	Cut-off Au (g/t)	2025 Resource			2019 Resource			Difference as a Percent (2025-2019)/2019		
		Tonnage (kt)	Au (g/t)	Au (koz)	Tonnage (kt)	Au (g/t)	Au (koz)	Tonnage	Au Grade	Au Metal
Indicated	**3**	3,118	11.21	1,124	1,798	10.40	601	73	8	87
	4	2,965	11.61	1,107	1,687	10.80	586	76	8	89
	5	2,769	12.11	1,078	1,556	11.30	565	78	7	91
	6	2,525	12.75	1,035	1,403	12.00	541	80	6	91
	7	2,270	13.45	981	1,260	12.60	510	80	7	92
	8	2,049	14.09	928	1,105	13.30	473	85	6	96
	9	1,814	14.81	864	947	14.10	429	92	5	101
	10	1,594	15.55	797	1,639	9.50	501	-3	64	59
Inferred	**3**	1,061	8.24	281	1,582	9.80	498	-33	-16	-44
	4	926	8.93	266	1,483	10.20	486	-38	-12	-45
	5	817	9.52	250	1,351	10.70	465	-40	-11	-46
	6	706	10.16	231	1,223	11.20	440	-42	-9	-48
	7	603	10.78	209	942	12.50	379	-36	-14	-45
	8	491	11.52	182	753	13.80	334	-35	-17	-46
	9	371	12.51	149	653	14.60	307	-43	-14	-51
	10	291	13.33	125	0	0.00	0			

Table 1-2: Mineral Resource Estimate for Antimony (Sb) within the Base Case Au Resource

Class	Tonnage	Sb (%)	Sb Metal (lb)	Sb Metal (t)
Indicated	859,989	0.6547	12,412,778	5,630.2
Inferred	99,581	1.2004	2,635,337	1,195.3

Notes for Mineral Resource Estimate:

1. Mineral resources are not mineral reserves and do not have demonstrated economic viability.
2. There is no certainty that all or any part of the mineral resources will be converted into mineral reserves.
3. Resources are reported using the 2014 CIM Definition Standards and were estimated using the 2019 CIM Best Practices Guidelines.
4. The base case Mineral Resource has been confined by "reasonable prospects of eventual economic extraction" shape using the following assumptions:
   a. Metal prices of US$1,750/oz Au and Forex of US$0.75=C$1.00.
   b. Payable metal of 99% Au.
   c. Off-site costs (refining, transport and insurance) of US$7.00/oz.
   d. Mining cost of C$82.78/t.
   e. Processing costs of C$105.00/t, and G&A and site costs of C$66.00/t.
   f. Metallurgical Au recovery of 90.5%.
5. The resulting Net Smelter Return equation is: NSR (C$/t)=Au*90.5%*US$74.72 g/t.
6. The specific gravity is 2.81 for the entire deposit.
7. The antimony Resource is reported as a subset of the total Mineral Resource within the 4.0 g/t Au underground mining shape.
8. Sb is a by-product of Au processing; therefore, it is reported using the same classification as the Au Resource within the 4.0 g/t Au shape.
9. Numbers may not add due to rounding.

The geologic continuity of the “C” vein system has been well established through historic mining and diamond drilling. Grade continuity has been quantified using semi-variograms, which are used to determine the distances (ranges) and directions of maximum continuity in the three principal directions. The ranges are used for Classification.

The classification to Indicated or Inferred required that the true thickness of the vein is at least 2 m. Classification is based primarily on anisotropic distances to drillholes with 50-m grid drill spacing being targeted. However, additional adjustments have been made to ensure a cohesive shape of Indicated material is produced, as summarized in Section 14.

1.10 Mineral Reserve Estimate

Mineral Reserves are reported in accordance with National Instrument (NI 43-101) “Standards of Disclosure for Mineral Projects.” The Mineral Reserve estimate follows the Canadian Institute of Mining, Metallurgy and Petroleum (CIM) 2014 “CIM Definition Standards for Mineral Resources & Mineral Reserves” (CIM, 2014) and the “CIM Estimation of Mineral Resources & Mineral Reserves Best Practice Guidelines” (CIM, 2019). The independent QP who takes responsibility for the Mineral Reserves estimate is Dino Pilotto, P.Eng. of JDS Energy & Mining Inc. (JDS).

Modifying factors were applied in the process of converting Measured and Indicated Mineral Resources to Mineral Reserves. The Mineral Reserves were estimated using a gold price of US$2245/oz. The Mineral Reserves by mining area and in total are summarized in Table 1-3.

Table 1-3: Mineral Reserves in Total and by Mining Area as of July 10, 2025

Classification	Mine Area	Tonnage (kt)	Gold Grade (g/t)	Gold Metal (koz)
Proven	Zone C	-	-	-
	Zone Y	-	-	-
	Total	**-**	**-**	**-**
Probable	Zone C	2,525	9.98	810
	Zone Y	306	9.63	95
	Total	**2,830**	**9.94**	**904**
Proven and Probable	Zone C	2,525	9.98	810
	Zone Y	306	9.63	95
	Total	**2,830**	**9.94**	**904**

Mineral Reserve Estimate Notes:

1. Mineral Reserves are derived from Measured and Indicated Mineral Resources.
2. Figures in the table may not compute exactly due to rounding.
3. Mineral Reserves are based on a cut-off values of:
   a. 6.0 g/t Au for drift and fill stopes.
   b. 5.5 g/t Au for longhole stopes.
   c. 4.2 g/t Au marginal cut-off value for development ore.
   d. Cut-off grades are based on a gold price of US$2,245/oz.
4. Assumed exchange rate is C$1.39 = US$1.00.
5. Metallurgical recovery assumed to be 89.75%; payable assumed to be 90%.
6. Mineral Reserves are reported on a 100% ownership basis; Canagold owns 100% of the New Polaris property.

1.11 Mining Methods

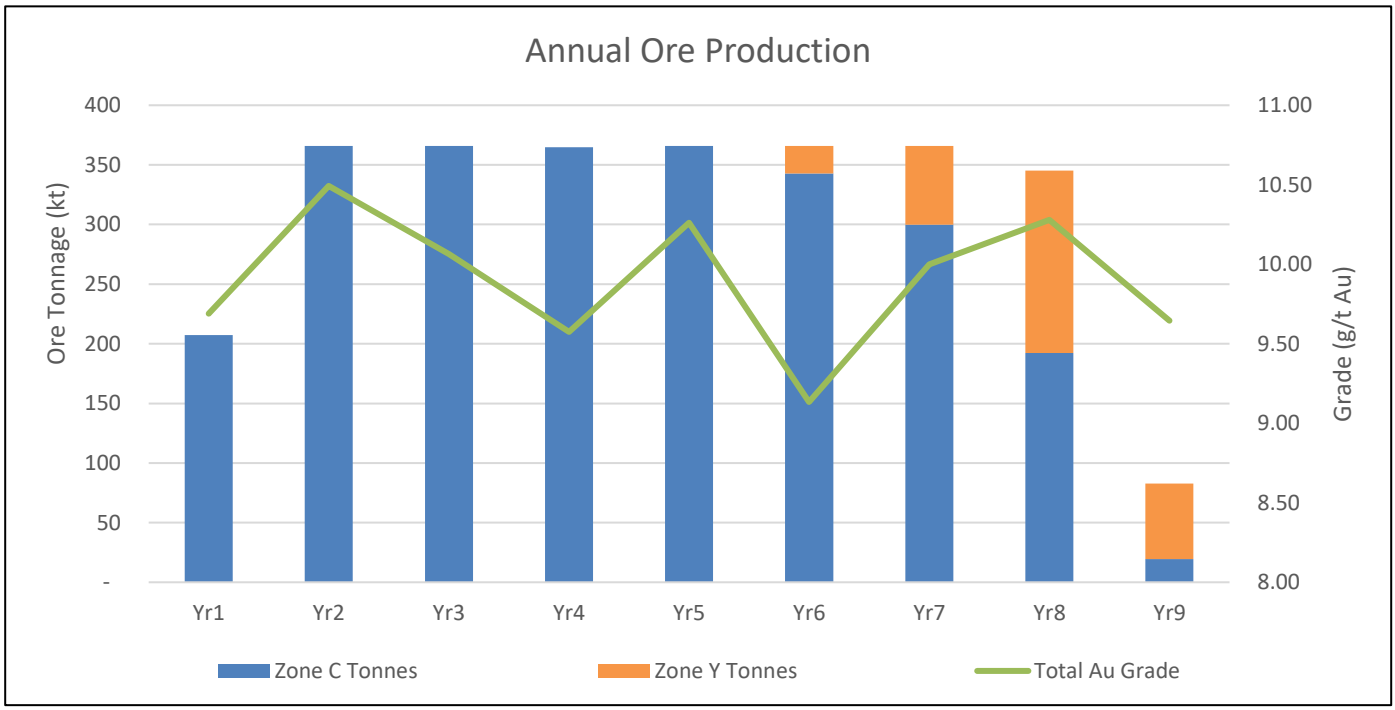
The mine plan targets a nominal ore production rate of 1,000 tonnes per day (t/d) over a period of just over eight years (see Figure 1-1). Mining is planned to occur in both Zone C and Zone Y, and within multiple veins within each zone. The rock mass is typically classified as “Good” with no noticeable difference in rock quality between the hanging and foot walls in both zones.

In consideration of mining selectivity, minimizing tailings generation, and logistical challenges in the transport of cementitious backfill binder to site for use in paste backfill, a combination of Longhole Stoping (LHS) and drift and fill (DAF) was selected for ore extraction. Relative to the assessed ground conditions, the selected mining methods are deemed appropriate. Figure 1-1 illustrates the planned underground annual ore production.

Drift and fill mining is the primary source of mill feed accounting for 60% of the tonnes. The remaining portion (40%) of the Mineral Reserves would be mined through LHS methods. LHS is applied to all of Y Zone and select areas of C Zone where geotechnical and deposit characteristics are suitable. Ore would be extracted using drill jumbos, longhole drills and a fleet of Load-Haul-Dump (LHD) machines loading into haul trucks that would haul ore to the surface via a ramp system. On the surface, ore would be stockpiled for feeding into the process plant.

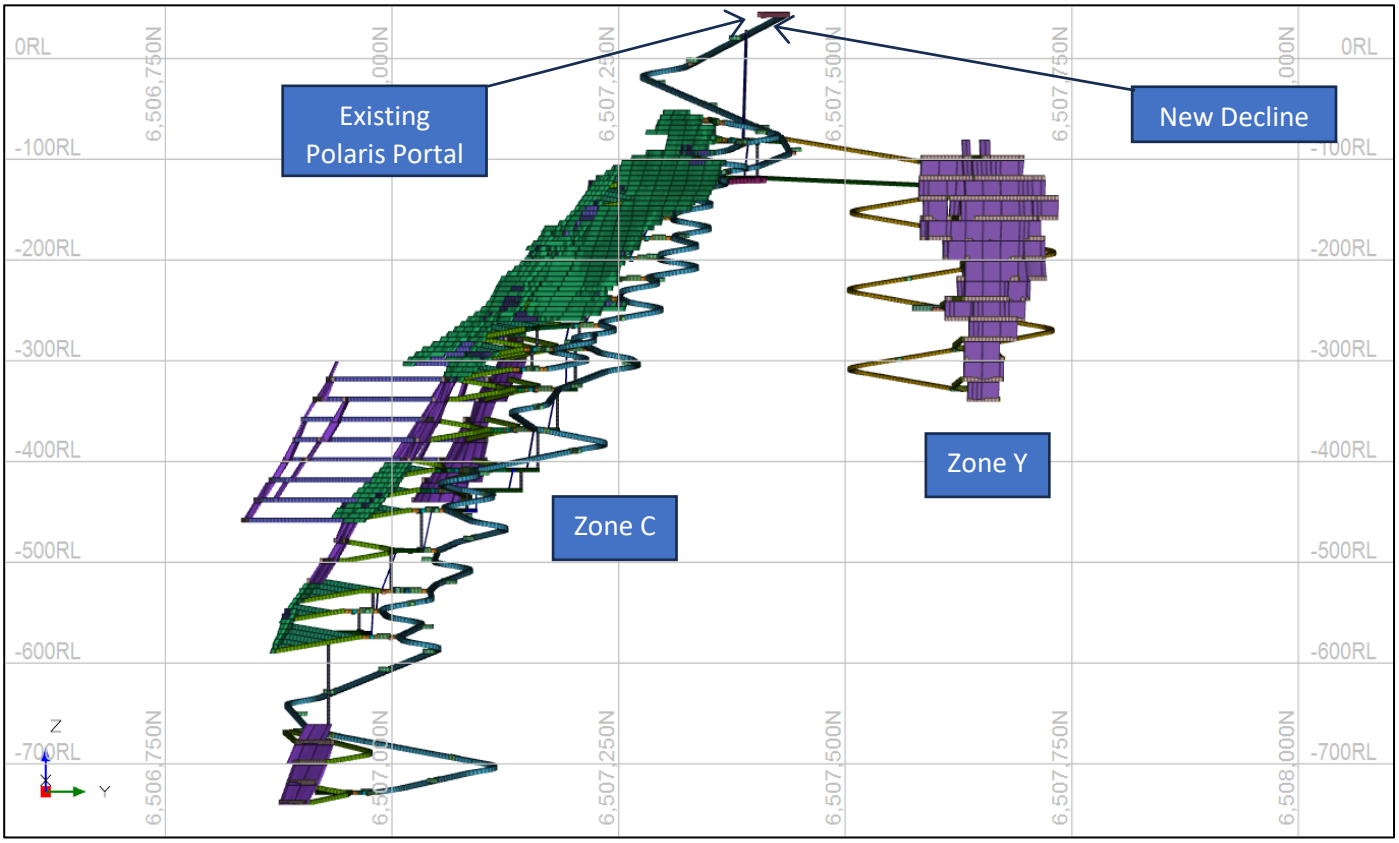
Initial development of the mine is planned to be performed by a mining contractor, with the transition to full owner-operator occurring in the second year of the production period.

Figure 1-1: Annual Ore Production Total by Zone



Source: JDS, 2025.

Figure 1-2: Long Section View of the New Polaris Underground Mine (Looking West)



Source: JDS, 2025.

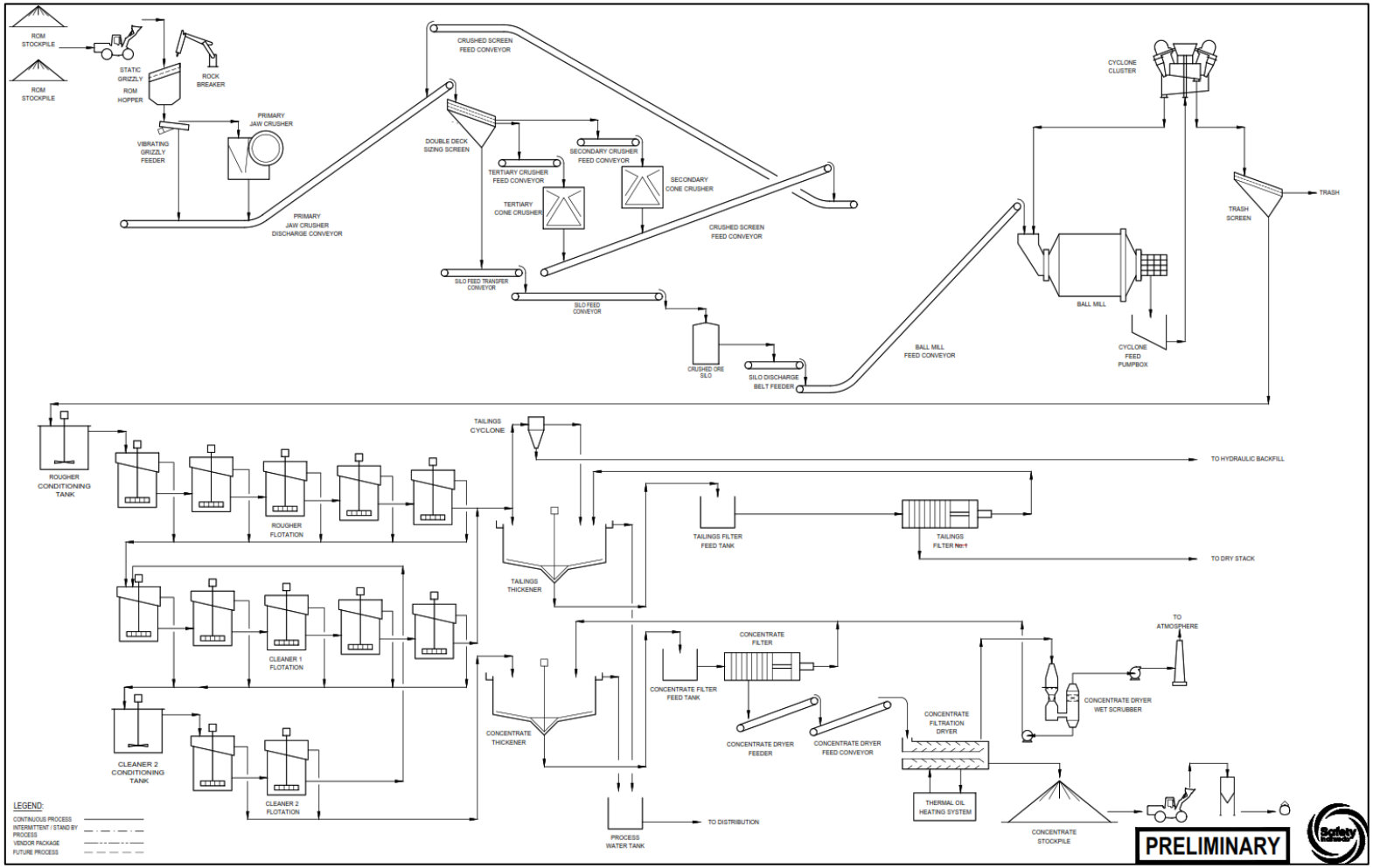
1.12 Recovery Methods

The proposed process plant (Figure 1-3) will have a throughput of 1,000 t/d and produce a gold concentrate from material sourced from the underground mine. The design utilizes three stages of crushing, a ball mill and cyclone in closed circuit, flotation, and concentrate dewatering. The mine will operate for a total of 8.1 years.

The crushing circuit consists of a primary jaw crusher followed by a secondary and a tertiary cone crusher. Crushed ore will be stored in a silo.

The flotation circuit will include rougher flotation and two stages of cleaning. Tailings from the rougher flotation cells will be sent to the tailings area and the concentrate will be refined in the cleaners. Tailings from the first cleaner will combine with the rougher tailings and sent to the tailings area. Concentrate from the first cleaner will be further cleaned in the second cleaner. Tailings from the second cleaner will be sent back to the first cleaner for another pass.

Figure 1-3: Process Flowsheet



Source: Ausenco, 2025.

Concentrate from the second cleaner will be dewatered using a thickener followed by a horizontal filter press. The filter cake will then be dried in a dryer to bring down the moisture content to a suitable level for bagging and air freight.

The tailings area will consist of a tailings thickener and filter press and tailings cyclone. Tailings from flotation will report to either the tailings thickener or tailings cyclone depending on whether the backfill plant is running. If the backfill plant is active, the tailings cyclone will send coarse underflow to the plant and the overflow fines will report to the tailings thickener. When the backfill plant is not active the tailings will directly go to the thickener. Thickened underflow tailings will be further dewatered in the tailings filter press prior to dry stacking.

The summary of the design process operating availabilities are as follows:

- Primary, secondary, and tertiary crushing availability of 66%
- Grinding, flotation, and thickening availability of 92%
- Filtration availability of 85%

1.13 Project Infrastructure

1.13.1 Surface Infrastructure

The New Polaris Project is located near the confluence of the Taku and Tulsequah Rivers, in a remote area of northwest British Columbia, Canada. The area is not accessible by road and has no nearby grid power to support the mine's operation. The site will include the following facilities and infrastructure:

- Mining facilities including mine administration offices, mine dry, truck workshop, explosives storage, fuel storage and distribution, ore stockpiles, and waste stockpiles.
- Process facilities including the process plant, crushing facilities, backfill plant, process plant workshop, assay laboratory, reagent and concentrate stroage.
- Co-storage facility (CSF) for waste rock and tailings.
- Catchments, ponds, and other site water management and treatment infrastructure.
- New airstrip.
- New barge landing facility.
- Access roads from the airstrip, barge landing facility and explosives facility to the mine site area.
- Haul roads from the mine site area to the CSF.

An overall site layout is provided in Figure 1-4.

The most significant road building and upgrading activity is to re-establish the main access road between the mine site and the proposed new airstrip and barge landing. This facilitates transport of material goods and consumables into the

site from the new airstrip. All roads will be 5 m in width for single lane traffic. Pull outs will be placed at intervals to allow passing of two-way traffic appropriate to the level of service of the road. Radio control of the road will also be required. The typical method of clearing, topsoil removal, and excavation will be employed, incorporating drains, safety bunds and backfilling with granular material and aggregates for road structure.

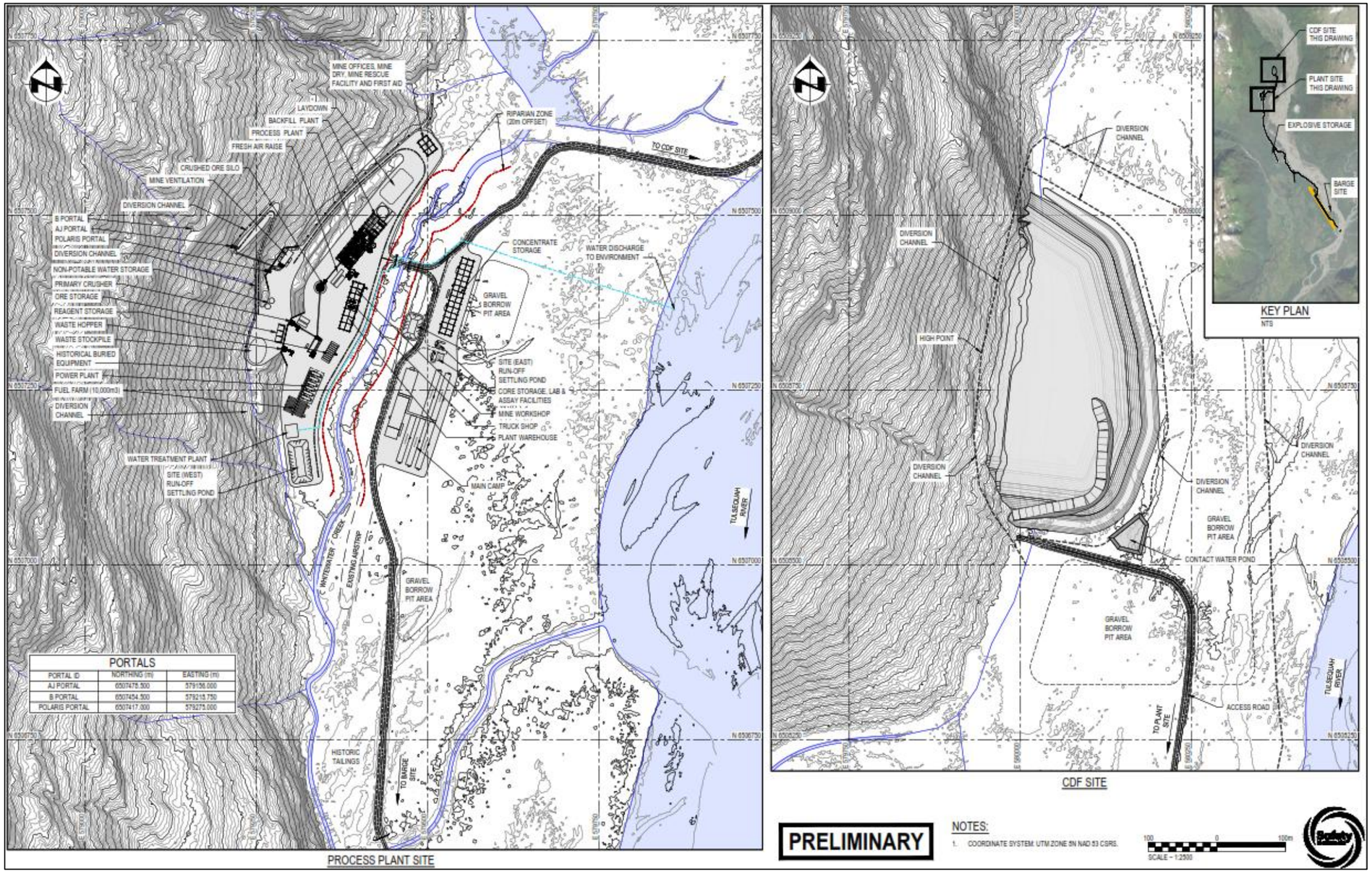
The existing New Polaris airstrip is too small to accommodate aircraft (ATR 72) appropriately sized for the mine during full operations; therefore, a new 1700-m airstrip located approximately 8 km south of the mine site, near the Tulsequah and Taku River confluence, has been included in this study.

Near the airstrip a barge landing site will be constructed as construction equipment and materials are proposed to be transported up the Taku River.

Two locations for a barge landing site are being evaluated. One on the north side of the Taku River, and one on the west side of the Tulsequah River, near the confluence of the Taku and Tulsequah River, approximately 10 km south of the mine site. The plant site consists of the necessary infrastructure to support the processing operations. All infrastructure buildings and structures will be built and constructed to all applicable codes and regulations. The project site will include an underground mine, a process plant, administration buildings, plant maintenance shop and warehouse, accommodations and other buildings.

Due to the site's remote location, there is no existing power supply to connect to the Project. It will include five 2-MW diesel generators as the main power source for the site, with one additional 2-MW diesel generator for backup and maintenance. The generators will be placed adjacent to one another with day tanks servicing each unit, all housed within a power generating facility.

Figure 1-4: New Polaris Site Layout



Source: Ausenco, 2025.

1.13.2 Site-Wide Water Management

The objective of surface water management is to protect groundwater and surface water resources. The infrastructure and catchments for the project's feasibility study were delineated based on topography data and footprints of facilities.

Contact and non-contact water are managed separately for the Project. Diversion channels, incorporated into the portal access roads and grading of the process plant pad area will divert non-contact hillside runoff away from mining operations. It is envisioned that the camp site and process area will be graded towards stormwater ponds to remove sediment prior to discharge to the environment (if effluent meets discharge requirements) or sent to treatment.

The underground mine workings at the site are currently flooded and will require dewatering prior to the excavation of a new ramp to provide access to the ore. Mine dewatering will require active treatment and will be pumped directly from the mine workings to a water treatment plant.

1.13.3 Co-Storage Facility (CSF)

Waste management involves the containment and long-term handling of waste materials generated during life-of-mine (LOM) operations, including waste rock and tailings. Based on waste rock availability, a decision was made to co-dispose of filtered tailings with waste rock in a lined CSF. This approach offers several advantages, such as eliminating the need for two separate facilities, resulting in a smaller environmental footprint, improved stability, reduced erodibility, and more efficient closure. Waste rock will be used to construct a containment shell around the filtered tailings. The CSF will accommodate approximately 2.32 Mt of filtered tailings and 0.31 Mt of mine waste rock, which will be transported to the facility by haul trucks and co-disposed throughout the LOM.

CSF water management system includes structures for surface and seepage water management. The diversion channels will collect non-contact surface runoff from the natural areas surrounding the CSF and convey it to drainage areas outside the CSF footprint. The CSF surface water collection system is designed to collect contact surface runoff and direct it to a collection pond. Contact water, expected to contain only sediment from the facility, will be routed to the collection pond before being discharged to the environment. Water quality monitoring will be conducted regularly to ensure that discharged water meets applicable standards. If the water does not meet these standards, it will be treated prior to discharge.

1.14 Market Studies and Contracts

Project economics were estimated based on long-term metal prices of US$2,500/oz Au. These prices are in accordance with consensus market forecasts from various financial institutions and are consistent with historic prices for these commodities.

There are several potential customers for New Polaris concentrate, including gold roasters in China, traders, copper and lead smelters, and pressure oxidation (POX) plants. The base case for this study is selling to metal traders for blending, with delivery to China. The concentrate is expected to be flown from the mine to Juneau, Alaska, then transported by barge to Tacoma, Washington, for ocean shipment to the final destination.

No transportation or off-take contracts for the concentrates are currently in place; however, any future agreements are expected to align with industry norms. Similarly, there are no contracts currently in place for the supply of reagents, utilities, or other bulk commodities required for the construction and operation of the Project.

1.15 Environmental, Permitting and Social Considerations

The New Polaris project involves the re-development of the Polaris-Taku Mine located approximately 100 km south of Atlin, BC on the west bank of the Tulsequah River in British Columbia, Canada. New Polaris is located within the territory of the Taku River Tlingit (TRT). The Taku River/T'akú Téix' Conservancy lies approximately 10 km south of the project site and encompasses the British Columbia portion of the Taku River main stem from the Alaska border, to the confluence of the Nakina and Inklin Rivers. There is no proposed infrastructure or mining activities planned within this Conservancy; however, the proposed site access is by barge from Alaska via the Taku River which passes through the conservancy.

1.15.1 Environmental Considerations

The Tulsequah River flows south to a confluence with the Taku River, approximately 10 km downstream of the existing mine site. The Taku River is a transboundary river, flowing west into Alaska, and into the Pacific Ocean, 25 km downstream of the Canada-United States border. The Tulsequah River has a catchment area of approximately 2,300 km2 and is fed by glaciers and snowmelt. The river flows through a steep and rugged terrain, which results in rapid changes in water levels during periods of heavy rain or snowmelt. Peak flows typically occur from June to August and low flows primarily occur in January and February.

There exists multi-year and seasonal baseline and supporting studies focused on the following subject areas:

- Air quality and noise
- Fish and aquatic resources
- Vegetation
- Ecosystems and wetlands
- Species at risk
- Terrestrial wildlife and wildlife habitat
- Geochemistry
- Surface water hydrology and water quality
- Groundwater quantity and quality
- Terrain stability mapping

A mine water balance and hydrogeological model have been developed for the mine site. These models will require further refinement based on additional drilling, monitoring and testing.

The key environmental considerations of mineral processing and tailings management for the site include the production of a bulk sulphide concentrate which will be shipped off-site for sale. The benefits of shipping concentrate for off-site refining include eliminating the use of cyanide and acid on site relative to conventional cyanide leach and doré production. In addition, tailings are planned to be deposited underground as thickened tailings and in a surface co-storage facility (CSF) as filtered semi-dry tailings with waste rock. The CDF facility will be fully lined with a geomembrane to capture any seepage and runoff with the excess water recycled as process water or treated and released to the environment.

To protect environmentally sensitive water bodies and wetlands, the proposed project design includes diversion structures such as berms with drainage ditches, diversion channels, and sediment ponds. This proposed design would capture surface drainage from the site facilities and divert it away for use in the facility process or allow water to settle out any solids prior to any discharge. On-site water that does not meet discharge criteria will be directed to a water treatment plant before discharge to the environment. Mine dewatering will require active treatment and underground mine water will be pumped directly from the underground workings to the water treatment plant.

A comprehensive environmental monitoring plan, in accordance with future permits and authorizations is anticipated throughout the construction, operations, and closure stages of the mine. In addition, over 20 socio-environmental mitigation and management plans are anticipated to be required for the project covering all most environmental and social/cultural aspects of the Project.

1.15.2 Permitting Considerations

Canagold has received a mineral exploration approval in the form of an Exploration Permit issued in 2020, under the BC Mines Act to complete exploration activities and geotechnical drilling to define and understand the potential Mineral Resource associated with the Project.

The proposed Project does not require a review under the Impact Assessment Act of Canada as it would have a daily production capacity of 1,000 t/d, which falls below the 5,000 t/d limit outlined in the federal Physical Activities Regulation.

The provincial EA process was initiated with the submission of the Project Description to the BC EAO, under the 2002 Environmental Assessment Act (EAA), in 2023. The Project constitutes a reviewable project pursuant to Part 3 of the Reviewable Projects Regulation (BC Reg. 607/19) of the Act, as a new project with a production capacity greater than or equal to 75,000 t/a of mineral ore. The BC EAO issued a Readiness Decision Report in 2024 recommending that New Polaris proceed to an EA and on that basis, Canagold and their permitting team have been progressing an Environmental Assessment for the Project.

Key permits and authorizations that are anticipated for the Project have been identified including up to a total of ten provincial BC and eight federal permits/authorizations. Due to the seasonal nature of the project and its access by barge an Early Works Application package is being advanced to expedite preliminary construction preparation activities.

The proposed site access is by barge from Alaska via the Taku River, therefore there are permitting considerations related to several United States regulatory requirements.

1.15.3 Social Considerations

The Initial Project Description and Engagement Plan for New Polaris Gold Mine were approved by the EAO in March 2023, and subsequently the EAO issued a notice of the Readiness Decision for New Polaris to proceed to an environmental assessment under the provincial Environmental Assessment Act. The two participating Indigenous nations for the environmental assessment are Taku River Tlingit First Nation and Teslin Tlingit Council (TTC). Tlingit First Nation territory overlaps the New Polaris Gold Mine. In 2007 the TRTFN, produced a comprehensive Taku River mining policy, the Taku River Tlingit First Nation Mining Policy (Tak River Tlingit First Nation, 2019). The TRTFN Mining Policy is the first of its kind in Canada. The policy outlines a clear set of rules for engagement, principles, procedures, and applications for mining related activities with the TRTFN within their territory, The goal of the policy is to provide a transparent and open guide for all potential and future partners.

As part of its own evaluation of the New Polaris Gold Mine, Taku River Tlingit First Nation established a Technical Working Group. This Technical Working Group will also engage bilaterally with the EAO and provide technical advice to the EAO on topics relating to Taku River Tlingit First Nation's rights and interests.

The New Polaris Project area has been and continues to be used by the TRT for cultural activities, and the Taku River is home to productive salmon fisheries, which are critically important to TRT community members. The New Polaris site is located within a management and protection area proposed by the TRTFN in 2023, known as the T'akú Tlatsini Indigenous Protected and Conserved Area (Takú IPCA). The Taku River/T'akú Téix' Conservancy also lies immediately south of the airstrip, which is part of the project site, and encompasses the British Columbia portion of the Taku River mainstem to the Alaska border. Both the Takú IPCA and Taku River/T'akú Téix' Conservancy aim to protect critical watersheds and ecosystems while allowing for economic diversification by establishing protected areas in tandem with areas where responsible development can occur.

The Project has the potential to affect various aspects of the TRTFN livelihood and traditional resource use. The potential effects on Indigenous Interests will be assessed and mitigation developed through a process of continuous and meaningful engagement and consultation.

Teslin Tlingit Council (TTC) is a self-governing First Nation based in Teslin in Southern Yukon Territory. The Village of Teslin is set on the confluence of Nisutlin Bay and Teslin Lake, located 183 kilometers east of Whitehorse on the Alaska Highway. BC EAO has identified the TCC as a participating nation for the purposes of the Environmental Assessment process for the Project.

It is the EAO's practice to notify Alaska Tribes who are nearby to projects undergoing a BC EA in the transboundary region.

During the public comment period for Early Engagement, the Alaska Tribes raised concerns related to the project's rationale and viability, regulatory engagement, requirements and oversight, barging, cumulative effects, and the potential impacts of New Polaris on culturally significant resources such as salmon and wild medicines.

Indigenous engagement has been taking place in scale relative to the level of exploration and clean up activity at the site. From 2006 to 2021 little activity took place at the site therefore engagement with Affected Parties was also limited. With the resumption of exploration drilling in 2021, engagement has become more frequent, focused, and directed.

More recent engagement activities have included forming of a joint Technical Working Group for sharing of project updates, plans, permit applications, meetings, and on-site visits along with monitoring, and field studies.

1.15.4 Closure and Reclamation Considerations

Under the BC Mines Act, anyone who engages in mining exploration work or mining operations determined by regulation must submit a reclamation plan. A conceptual reclamation and closure plan and a closure security estimate will need to be developed to support the submission of an EA report to the province or to the federal agency. The reclamation security will need to be posted to BC government prior to the commencement of construction the construction phase.

1.16 Capital and Operating Cost

1.16.1 Capital Cost Estimate

The capital cost estimate was developed in Q1 2025 Canadian Dollars (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 conforms 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).

The estimate includes mining, processing, on-site infrastructure, tailings and waste rock facilities, 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 escalation added to the estimate.
- A growth allowance was included.

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

- Mine schedules.
- FS-level engineering design by Ausenco, JDS, Tanka.
- Topographical information obtained from the site survey.
- Geotechnical investigations.
- Budgetary equipment quotes from suppliers based primarily in China.
- Budgetary unit costs from several contractors for civil, concrete, steel, electrical, piping, and mechanical works.

- Data from similar recently completed studies and projects.

Major cost categories (underground mine development, 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 1-4. The total initial capital cost for the New Polaris Project is C$250.4M, the LOM sustaining costs are C$225.0M. Closure costs are estimated at C$20.5M.

Table 1-4: Capital Cost Estimate

Description	Initial Capital Cost (C$M)	Sustaining Cost (C$M)	Closure Cost (C$M)	Total Cost (C$M)
Mining	63.3	196.1	-	259.4
Process Plant	43.0	-	-	43.0
Tailings Facilities	7.4	4.7	-	12.1
On-Site Infrastructure	38.5	-	-	38.5
Off-Site Infrastructure	9.4	-	-	9.4
Total Direct Costs	**161.6**	**200.8**	**-**	**362.4**
Project Indirects	42.3	-	-	42.3
EPCM Services	9.8	-	-	9.8
Owners Costs	7.8	-	-	7.8
Contingency	28.8	24.2	-	53.0
Total Indirect Costs	**88.8**	**24.2**	**-**	**113.0**
Project Total	**250.4**	**225.0**	**20.5**	**495.9**

Note: Values may not sum due to rounding.

1.16.2 Operating Cost Estimate

The operating cost estimate was developed in Q1 2025 Canadian dollars based on a blend of budgetary quotes for equipment, consumables, and reagents, diesel cost for power, local labor rates, and Ausenco's in-house database of projects and studies from similar projects.

Considerations have been made to account for the reduced power consumption in the initial two years of operation when the mine plan is ramping up, resulting in a lower mill throughput and lower annual operating costs.

Life-of-mine operating cost estimates for the Project are categorized into mining, processing, and general and administrative (G&A) expenses, as shown in Table 1-5. The annual operating cost presented reflects Year 4, which is reflective of a typical operating year at full capacity.

Table 1-5: Operating Expenditure Summary

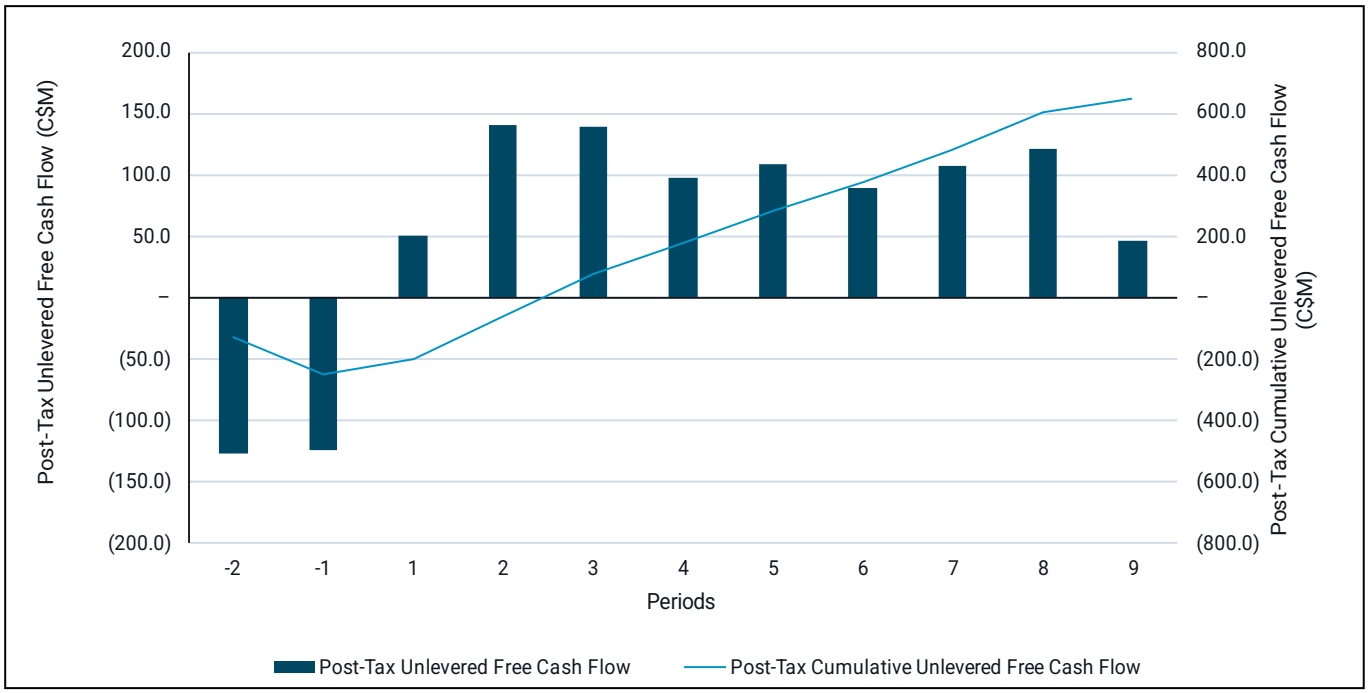
Cost Area	Total (C$M)	C$/t Milled	% of Total
Mining	383.3	135.45	50.6
Process	181.9	64.28	24.0
Barging	0.4	0.16	0.1
G&A	191.2	67.58	25.3
Total	**757.0**	**267.47**	**100.0**

Note: Closure reclamation bond (G&A) costs are included. Values may not sum due to rounding.

1.17 Economic Analysis

The economic analysis was performed assuming a 5% discount rate with no inflation (constant dollar basis). On a pre-tax basis, the net present value (NPV) discounted at 5% is C$666M; the internal rate of return (IRR) is 38.4%, and payback period is 2.3 years. On a post-tax basis, the NPV discounted at 5% is C$425M; the IRR is 30.9%, and the payback period is 2.4 years. Using a spot price of US$3,300/oz Au, on a pre-tax basis, NPV discounted at 5% is C$1,246M; the IRR is 58.7%, and the payback period is 1.6 years. On a post-tax basis, the NPV discounted at 5% is C$793M; the IRR is 47.2%, and the payback period is 1.7 years. It must be noted, however, that tax estimates involve many complex variables that can only be accurately calculated during operations and, as such, the post-tax results are only approximations. The analysis was performed on an annual cashflow basis; the cashflow output is shown graphically in Figure 1-5 and summarized in Table 1-6.

Figure 1-5: Projected LOM Post-Tax-Free Unlevered Cash Flow



Source: Ausenco, 2025.

Table 1-6: Economic Analysis Summary

Item	Units	Base	Spot
Gold Price	US$/oz Au	2,500	3,300
Foreign Exchange Rate	USD/CAD	0.72	0.72
Discount Rate	%	5.0%	5.0%
Production			
Mine life	years	8.3	8.3
Total resource mined	kt	2,830	2,830
LOM Au grade	g/t	9.94	9.94
Average annual mined resource	kt/a	342	342
Total recovered Au	koz	806	806
Average annual recovered Au	koz/a	97	97
Total payable Au	koz	708	708
Average annual payable Au	koz/a	86	86
Revenue/Costs			
Total revenue	C$M	2,444	3,226
Average annual revenue	C$M/a	296	390
EBITDA	C$M	1,469	2,251
Average annual EBITDA	C$M/a	178	272
Total on site operating costs (mining, process, G&A)	C$M	757	757
Mining unit cost	C$/t mined	85.93	85.93
Mining unit cost	C$/t milled	135.45	135.45
Process unit cost	C$/t milled	64.44	64.44
G&A unit cost	C$/t milled	67.58	67.58
Total off site operating costs (transport, treatment & refining)	C$M	218	218
Cash Costs			
Total cash cost*	US$/oz Au	998	998
All-in sustaining cost**	US$/oz Au	1,248	1,248
Capital Costs			
Initial CAPEX	C$M	250	250
Sustaining CAPEX	C$M	225	225
Closure CAPEX	C$M	21	21
Salvage value	C$M	19	19
Pre-Tax Economics			
NPV (5%)	C$M	666	1,246
IRR	%	38.42%	58.73%
Payback	year	2.3	1.6
NPV/initial capital	-	2.7	5.0
Post-Tax Economics			
NPV (5%)	C$M	425	793
IRR	%	30.90%	47.24%
Payback	year	2.4	1.7
NPV/initial capital	-	1.7	3.2

* Total cash cost consist of mining costs, processing costs, mine-level G&A, off-site charges and royalties.

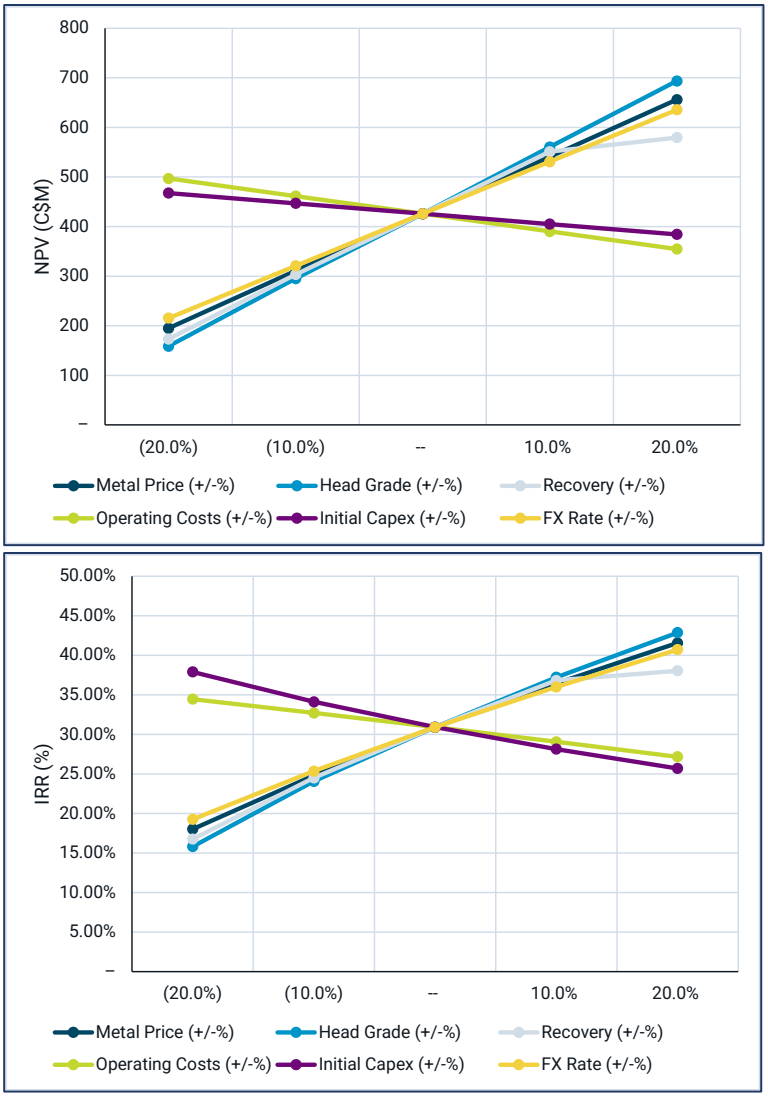
** AISC includes total cash cost, sustaining capex and closure capex.

1.17.1 Sensitivity Analysis

A sensitivity analysis was conducted on the base case pre-tax and post-tax NPV and internal rate of return (IRR) of the Project, using the following variables: metal prices, discount rate, operating costs, initial capex, metal recovery, head grade and foreign exchange (FX) rate.

A summary of post-tax economic sensitivities to commodity prices, operating costs, initial capex, metal recovery, head grade and FX rate is shown in Figure 1-6. The project is most sensitive to head grade and metal price with exchange rate and recovery to a lesser extent.

Figure 1-6: Post-Tax Sensitivity Summary



Source: Ausenco, 2025.

1.18 Interpretations and Conclusions

The total probable Mineral Reserves for the New Polaris Project are estimated at 2,830 Mt grading 9.94 g/t Au.

Based on the assumptions and parameters presented in this report, the Feasibility Study demonstrates positive economics, with a post-tax NPV5% of C$425M and a post-tax IRR of 30.90%. The results of the Feasibility Study support advancing the Project to the next stage of development.

1.19 Recommendations

The following recommendations are suggested to further derisk the Project by progressing key areas such as exploration, mining, hydrogeology, infrastructure and environmental and permitting considerations. Canagold intends to make a decision on FEED design following completion of the Feasibility Study, in line with permitting timeline. This phase will further define the Project, confirm costing, and serve as the starting point for the Engineering, Procurement and Construction Management (EPCM) process. EPCM activities will begin when the Company makes a construction decision, which will follow receipt of permits and completion of the financing process. Table 1-7 below summarizes the recommended work to be completed after the Feasibility Study and prior to a construction decision.

Complete a flotation test program focused on producing a separate antimony concentrate to determine if saleable antimony concentrates can be produced. Producing a separate antimony concentrate will add significant value to the project and eliminate any antimony penalties from the processing of the gold sulphide concentrate.

Initiate field programs and engineering evaluation for micro-hydro power to supplement or replace diesel-generated electricity at the site, providing a reliable and renewable energy source.

Table 1-7: New Polaris Recommended Work Program

Program Component	Estimated Total Cost (C$M)
Metallurgical Testwork	0.8
Mine Engineering	0.2
Backfill	0.9
CSF	0.8
Hydro Power Feasibility & Field Investigations	6.7
Environmental Studies, Permitting and Social Considerations	0.4
Total	**9.8**

2 INTRODUCTION

2.1 Introduction

Canagold Resources Ltd (Canagold) commissioned Ausenco to compile a Feasibility Study (FS) of the New Polaris Project (the Project). The FS was prepared in accordance with the Canadian disclosure requirements of National Instrument 43-101 (NI 43-101) and in accordance with the requirements of Form 43-101 F1.

The responsibilities of the engineering companies who were contracted by Canagold to prepare this report are as follows:

- Ausenco managed and coordinated the work related to the report and developed FS-level design and cost estimate for the process plant, general site infrastructure, and economic analysis.
- Ausenco completed geotechnical studies, site-wide water balancing, and developed the FS-level design and cost estimate of the co-storage facility for tailings and waste rock.
- Ausenco Sustainability ULC carried out an assessment of the site-wide water management, including sizing and designing water management related structures for the waste rock stockpiles and process area.
- JDS Energy & Mining Inc. (JDS) designed the underground mine, mine production schedule, and mine capital and operating costs.
- Moose Mountain Technical Services (MMTS) completed the work related to property description, accessibility, local resources, geological setting, deposit type, exploration work, drilling, mineral resource estimate and QA/QC.

The proposed project is an underground gold mine with an anticipated production rate of approximately 1,000 tonnes per day (t/d). The proposed mine will operate year-round, processing, on average, approximately 365,000 tonnes per year (t/a) of ore. The mine life will be 8.1 years.

A Mineral Resource Estimate (MRE) and a Preliminary Economic Assessment (PEA) on the Project were completed in 2007. The PEA was updated in 2009 and again in 2011, using the same Mineral Resource data from 2007, but incorporating revised gold prices and updated capital and operating cost estimates. An updated MRE and PEA were completed in 2019. Another MRE update followed in 2023. The most recent MRE update, completed in 2025 and presented herein, builds on the 2023 MRE by incorporating results for antimony, arsenic and sulphur.

2.2 Terms of Reference

The report supports disclosures by Canagold in a news release dated July 21, 2025, entitled, “Canagold Announces Positive Feasibility Study Results for the New Polaris Project.”

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, the individuals presented in Table 2-1 are each considered to be a "qualified person" as defined by NI 43-101. Report sections for which each QP is responsible are also listed in Table 2-1.

Table 2-1: Report Contributors

Qualified Person	Professional Designation	Position	Employer	Independent of Canagold	Site Inspection Complete
Tommaso Roberto Raponi	P. Eng.	Senior Mineral Processing Specialist	Ausenco Engineering Canada ULC	Yes	No
Kevin Murray	P.Eng.	Principal Process Engineer	Ausenco Engineering Canada ULC	Yes	No
Jonathan Cooper	P. Eng.	Senior Water Resources Engineer	Ausenco Sustainability ULC	Yes	Yes
James Millard	P. Geo.	Director, Strategic Projects	Ausenco Sustainability ULC	Yes	No
Mohammad Ali Hooshiar Fard	P.Eng.	Lead Geotechnical Engineer	Ausenco Engineering Canada ULC	Yes	No
Sue Bird	P. Eng.	Principal and VP, Resources	Moose Mountain Technical Services Ltd	Yes	Yes
Dino Pilotto	P. Eng.	GM, Technical Services	JDS Energy & Mining Inc	Yes	Yes
Michael Levy	P. Eng.	Geotechnical Manager	JDS Energy & Mining Inc	Yes	Yes

2.4 Site Visits and Scope of Personal Inspection

A summary of the site visits completed by the QPs is presented in Table 2-2.

Table 2-2: Site Visits

Qualified Person	Date of Site Visit(s)
Jonathan Cooper	September 17–24, 2023
Sue Bird	August 25, 2022
Dino Pilotto	June 3, 2025
Michael Levy	June 3, 2025

2.4.1 Site Inspection by Jonathan Cooper, P Eng.

Jonathan Cooper visited the site from September 17 to 24, 2023. Activities during the site visit included the following:

- Conducted discharge measurements along Whitewater Creek.
- Conducted water quality sampling along Whitewater Creek, Tulsequah River and various groundwater monitoring wells on site.
- Site reconnaissance of potential hydroelectric station locations.
- Attended a meeting with the TRTN working group.

2.4.2 Site Inspection by Sue Bird, P. Eng.

Sue Bird visited the site on August 25, 2022. Activities during the site visit included the following:

- Examined core logging procedures.
- Examined drill cores from the 2022 program.
- Conducted a general overview of the property including selected drill sites, historic core, an underground tour and the condition of existing project infrastructure.

2.4.3 Site Inspection by Dino Pilotto, P. Eng.

Dino Pilotto (JDS) visited the New Polaris site on June 3, 2025. Activities during the site visit included the following:

- The purpose of this visit was to review the underground portal areas and underground workings with the Canagold team.
- The underground mine was visited by way of the AJ Portal where some of the old workings and part of the No. 1 Main Shaft were observed.

2.4.4 Site Inspection by Michael Levy, P. Eng.

Michael Levy (JDS) visited the New Polaris site on June 3, 2025. Activities during the site visit included the following:

- The purpose of this visit was to review the underground portal areas and underground workings with the Canagold team.
- The underground mine was visited by way of the AJ Portal where some of the old workings and part of the No. 1 Main Shaft were observed.

2.5 Effective Dates

This Technical Report has significant dates as follows:

- New Polaris Mineral Resource Estimate: January 22, 2025
- New Polaris Mineral Reserve estimate: July 10, 2023
- Financial analysis: July 14, 2025

The effective date of this report is based on the date of the press release, which is July 21, 2025.

2.6 Information Sources and References

This Technical Report is based on internal company reports, maps, published government reports, and public information as listed in Section 27. It is also based on information cited in Section 3.

The authors are not experts with respect to legal, socio-economic, land title, or political issues, and are therefore not qualified to comment on issues related to the status of permitting, legal agreements, and royalties. Information related to these matters has been provided directly by Canagold and include, without limitation, validity of mineral tenure, status of environmental and other liabilities, and permitting to allow completion of environmental assessment work. These matters were not independently verified by the QPs but appear to be reasonable representations that are suitable for inclusion in Section 4 of this Report.

2.7 Previous Technical Reports

The New Polaris Project has been the subject of previous technical reports, as summarized in Table 2-3. This FS report supersedes past reports stated below.

Table 2-3: Previous Technical Reports

Reference	Company	Name
Moose Mountain Technical Services, 2007	Canarc (now Canagold)	The New Polaris Gold Project, British Columbia, Canada – 2007 Preliminary Economic Assessment
Moose Mountain Technical Services, 2009	Canarc (now Canagold)	The New Polaris Gold Project, British Columbia, Canada – 2009 Preliminary Economic Assessment
Moose Mountain Technical Services, 2011	Canarc (now Canagold)	The New Polaris Gold Project, British Columbia, Canada – 2011 Preliminary Economic Assessment
Moose Mountain Technical Services, 2019	Canarc (now Canagold)	The New Polaris Gold Project, British Columbia, Canada – 2019 Preliminary Economic Assessment
Moose Mountain Technical Services, 2023	Canagold	The New Polaris Gold Project, British Columbia, Canada – 2023 Resource Estimate Update
Moose Mountain Technical Services, 2025	Canagold	The New Polaris Gold Project, British Columbia, Canada – 2025 Resource Estimate Update

2.8 Currency, Units, Abbreviations and Definitions

All units of measurement in this report are metric and all currencies are expressed in Canadian dollars (symbol: C$ or currency: CAD) unless otherwise stated. Contained gold metal is expressed as troy ounces (oz), where 1 oz = 31.1035 g. All material tonnes are expressed as dry tonnes (t) unless stated otherwise. A list of abbreviations and acronyms is provided in Table 2-4, and units of measurement are listed in Table 2-5.

Table 2-4: Abbreviations and Acronyms

Abbreviation	Description
AACE	Association for the Advancement of Cost Engineering International
AAS	atomic absorption spectroscopy
ABA	acid base accounting
AMC	Australian Mining Consultants (Canada) Ltd
ANFO	Ammonium nitrate and fuel oil
ANP	Ammonium nitrate prill
ARD	Acid rock drainage
Au	gold
Az	azimuth
BEV	battery electric vehicle
BIF	banded iron formation
BTS	Brazilian tensile strength
BWi	bond ball mill work index
CAD:USD	Canadian-American exchange rate
CCME-FAL	Canadian Council of Ministers of the Environment – Freshwater Aquatic Life
CHF	Cemented hydraulic fill
CIM	Canadian Institute of Mining, Metallurgy and Petroleum
CIM Definition Standards	CIM Definition Standards for Mineral Resources and Mineral Reserves 2014
CIP	carbon in pulp
COG	cut-off grade
CRM	certified reference material
CWi	Bond crusher work index
DAF	Drift and fill
DCIP	direct current resistivity and induced polarization
DD	Dip direction
DDH	diamond drill hole
EGBC	Engineers and geoscientists British Columbia
E-GRG	extended gravity recoverable gold
ELOS	Equivalent linear overbreak/sloughing
EM	electromagnetic
FA	fire assay
FAR	Fresh air raise
FEED	Front-end Engineering Design

Abbreviation	Description
FET	federal excise tax
FIFO	Fly-in fly-out
FOS	Factor of Safety
FS	feasibility study
FW	footwall
FX	foreign exchange
G&A	general and administration
GHG	Greenhouse gas emissions
GPR	gross production royalty
GQCV	greenstone-hosted quartz-carbonate vein deposits
GRAV	gravimetric finish method
HDPE	High-density Polyethylene
HF	Hydraulic fill
HW	Hanging wall
IBC	intermediate bulk container
ICP	inductively coupled plasma
ICP-OES	inductively coupled plasma - optical emission spectrometry
ID2	inverse distance squared
ID3	inverse distance cubed
IOCG	iron oxide copper gold
IP	induced polarization
IPCA	Indigenous Protected and Conserved Area
IRGS	intrusion-related gold system
ISO	International Organization for Standardization
JDS	JDS Energy & Mining Inc
LHD	load-haul-dump
LIDAR	light detection and ranging
LOM	Life of mine
LUP	land use permit
MCF	mechanized cut and fill
MRE	mineral resource estimate
MSO	Mineable shape optimizer
NaCN	Sodium cyanide
NAD 83	North American Datum of 1983
NI 43-101	National Instrument 43-101 (Regulation 43-101 in Quebec)
NN	nearest neighbour
NSR	net smelter return
NTS	national topographic system
OK	ordinary kriging
PEA	preliminary economic assessment
PEB	Pre-Engineered Building

Abbreviation	Description
PFS	prefeasibility study
PGA	Peak ground acceleration
PGE	platinum group elements
QA/QC	quality assurance/quality control
QP	qualified person (as defined in National Instrument 43-101)
RAA	Regional Assessment Area
RAR	return air raise
RBF	Radial basis function
RL	Relative level
RMR	Rock mass rating
ROM	run of mine
RQD	rock quality designation
RWi	Bond rod mill work index
SAG	semi-autogenous grinding
SCC	Standards Council of Canada
SD	standard deviation
S_d-BWI	micro hardness or bond ball mill work index on SAG ground material
SEDEX	sedimentary exhalative deposits
SG	specific gravity
SLD	Single line diagram
STD	Standard deviation
TCS	Triaxial compressive strength
TKM	Tonne kilometer
TMF	tailings management facility
UCS	Uniaxial compressive strength
UG	underground
UTM	Universal Transverse Mercator coordinate system
UV	ultraviolet
VLF-EM	very low frequency electromagnetic
VMS	volcanogenic massive sulphide

Table 2-5: Units of Measurement

Abbreviation	Description
%	percent
% solids	percent solids by weight
CAD	Canadian dollar (currency)
C$	Canadian dollar (as symbol)
$/t	dollars per metric ton
°	angular degree
°C	degree Celsius
€	Euro (as symbol)
µm	micron (micrometer)
cm	centimeter
cm^3	cubic centimeter
EGL	effective grinding length
EUR	Euro (currency)
ft	foot (12 inches)
g	gram
g/cm^3	gram per cubic centimeter
g/L	gram per liter
g/t	gram per metric ton (tonne)
GPa	gigapascal
h	hour (60 minutes)
ha	hectare
hp	horsepower
Hz	hertz
k	kilo (thousand)
kg	kilogram
kg/t	kilograms per metric tonne
km	kilometer
km^2	square kilometer
kPa	kilopascal
kt	kilotonne
kV	kilovolt
kVA	Kilovolt-amperes
kW	kilowatt
kWh/t	kilowatt-hour per tonne
L	liter
lb	pound
L/s	Liters per second

Abbreviation	Description
m, m^2, m^3	meter, square meter, cubic meter
M	million
Ma	million years (annum)
m/mo	meters per month
m/s	meters per second
m^3/h	Cubic meters per hour
m^3/s	cubic meters per second
m asl	meters above mean sea level
mbgs	meters below ground surface
mm	millimeter
Moz	million (troy) ounces
MPa	Megapascal
Mt	million tonnes
MW	megawatt
oz	troy ounce
oz/t	ounce (troy) per tonne
oz/ton	ounce (troy) per short ton (2,000 lbs)
ppb	parts per billion
ppm	parts per million
t	metric tonne (1,000 kg)
tkm	tonne kilometer
ton	short ton (2,000 lbs)
t/h	tonnes per hour
t/d	tonnes per day
USD	US dollars (currency)
US$	US dollar (as symbol)
V	volt

3 RELIANCE ON OTHER EXPERTS

3.1 Introduction

The QPs have relied on other expert reports which provided information regarding permitting, social and community impacts, taxation, and marketing for sections of this report.

3.2 Property Agreements, Mineral Tenure, Surface Rights and Royalties

The QP has not reviewed the mineral tenure, nor independently verified the legal status, ownership of the Project area or underlying property agreements. The QP has fully relied upon the information supplied by Canagold Corp. and experts retained for Canagold for this report.

This information was provided by Canagold on April 1, 2025, in the following documents:

- Dentons Canada LLP. (2025). *CCM Mineral Title Opinion Dentons 20221213*, (Internal document). Updated by Canagold April 1, 2025.
- Dentons Canada LLP. (2022). *CCM Mineral Title Opinion Dentons 20221213*, (Internal document). Dated December 13, 2022.
- Canagold, 2025. Email from Canagold on August 22, 2025 regarding a potential Net Profit Interest of 15% to Rembrandt from a 1994 agreement which Canagold is in the process of disputing due to lack of documentation.

This information is used in support of Section 1.2, 1.3, 1.9, 4, 14, 25.2 and 25.5 of the Report. The QP has assumed that the information in this letter is accurate and understands that the information in such letter may not be relied upon by any other party without the consent of Dentons Canada LLP.

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

The QP has not reviewed the archaeological evidence, wildlife monitoring or ecological data of the Project area, nor independently verified the geochemical sampling and laboratory results. The QP has fully relied upon the information supplied by experts retained for Canagold for this report.

This information was provided by Canagold on April 1, 2025, in the following documents:

- Ecofor. (2023, February 24). 2022-0022-002, AIA of Canagold Resources Ltd. New Polaris Gold Mine . *Prepared by Ecofor Consulting Ltd for Canagold Resources Ltd.*
- Ecofor. (2021, July 19). Archaeological Overview Assessment Report: Proposed Developments for the New Polaris Project. Prepared by Ecofor Consulting Ltd. for Hemmera.
- Ecofor. (2022, January 21). 2022-0022-001, PFR of Canagold Resources Ltd. New Polaris Gold Mine . Prepared by Ecofor Consulting Ltd for Canagold Resources Ltd.

- EcoLogic Consultants Ltd. (2024, August 6). New Polaris - Summary of Bat Investigations. *Memorandum prepared by EcoLogic Consultants Ltd. for Canagold Resources Ltd.*
- EDI. (2021). New Polaris Project – Barge Route Wildlife Gap Analysis. Prepared by Environmental Dynamics Inc. (EDI) for Hemmera Envirochem Inc.
- Madrone Environmental Services Ltd. (2024, July 29). Terrain Staibliity Mapping Report. *Prepared by Madrone Environmental Services Ltd for Ausenco Engineering Canada Inc.*
- URS. (2007, November). Draft report on: Phase 2 Geochemical Assessment, Vein Sample Static Tests, New Polaris Project, Atlin Mining Division, BC. *URS File # 39548826*.

This information is used to support Section 1.15, 20, and 25.9.

3.4 Taxation

The QPs have not independently reviewed the project taxation position. The QPs have fully relied upon the taxation information derived from PwC who was retained for this information. Project taxation information has been provided through email communications as documented in the following communication:

- Catalin Kilofliski, Canagold CEO. "Tax model completed by pwc attached" Internal document prepared for Canagold and emailed to Ausenco on June 26, 2025.

This information is used in support of Section 1.17, 22, and 25.12.

3.5 Markets

The QPs has not independently reviewed the marketing, smelter terms, or metal price forecast information. The QP has fully relied upon the information derived from experts retained by Canagold. The market study was documented in the following communication:

- Catalin Kilofliski, Canagold CEO. "New Polaris Marketing Study" Internal document prepared for Canagold by DenoAdvisory and emailed to Ausenco on June 24, 2025.

This information is used in Section 19, and in support of the financial analysis in Section 22.

Metals marketing, global concentrate market terms and conditions, and metals forecasting are specialized businesses requiring knowledge of supply and demand, economic activity and other factors that are highly specialized and requires an extensive database that is outside of the purview of a QP.

The QP considers it reasonable to rely on such information as the consultants are specialists in commodities trading. Detailed information outlining all payables, penalties, deductions, and charges, was provided to arrive at an estimated value of concentrate. It also provides information for sellers, buyers, and finance providers to understand the value of each material based on the current market terms.

4 PROPERTY DESCRIPTION AND LOCATION

4.1 Introduction

The New Polaris property consists of 61 contiguous Crown grants, and one modified grid claim totaling 1,108 ha, located 100 km south of Atlin, BC, and 60 km northeast of Juneau, Alaska. Located at approximately 58º42'N Latitude and 133º37'W Longitude, the deposit lies on the western flank of the Tulsequah River Valley, as illustrated in Figure 4-1

The claims are 100% owned and held by New Polaris Gold Mines Ltd., a wholly owned subsidiary of Canagold Resources Ltd (Canagold). Table 4-1 and Table 4-2 summarize the claims, and their locations are shown on Figure 4-1. Apart from the W.W.1 claim, which is a modified grid claim, all other claims are Crown granted as of the effective date of the report, all annual tax payments have been made and the crown claims are in good standing. The W.W.1 claim has sufficient work filed to keep it in good standing until February 4, 2026. The Crown granted claims were legally surveyed in 1937. The mineralized areas are shown on Figure 7-2, which shows the geology of the property and the locations of the mineral showings.

The Polaris No. 1, Silver King No. 1, Silver King No. 5, Black Diamond, Lloyd and Ant Fraction Crown grants include surface rights. Surface rights for the remainder of the property lie with the Crown.

Mining of the AB vein system, and to a lesser extent the Y and C veins, was carried out from the 1930s to early 1950s. Much of the former infrastructure has been reclaimed. A $300,500 reclamation bond is in place, and it is the QP's opinion that this adequately covers the cost of reclaiming the original mill site and infrastructure. Currently, there is no legal or regulatory requirement to remove or treat the tailings on the property; however, as part of the environmental permitting process, the best long-term treatment option for the secure storage of tailings is under review.

Canagold holds a Notice of Work permit (MX-1-208), issued by the BC Ministry of Energy, Mines and Low Carbon Innovation, authorizing the activities described in the permit through to its expiry on March 31, 2028.

Water rights will need to be acquired from the Province of British Columbia to permit the use of water for mining, processing, and potable water during operations. As the area has a positive water balance and the project plans to maximize water recycling to reduce freshwater demand, the approvals are generally granted, subject to acceptable conditions.

Canarc merged with Suntac in 1992 and acquired a 100% interest from Rembrandt in 1994, subject to a 15% net profit interest, which Canagold can reduce to 10%. The economic model excludes the 15% Net Profit Interest currently as Canagold is disputing it.

To the extent known, there are no other significant factors or risks that may affect access, title, rights, or the ability to perform the proposed work on the Project.

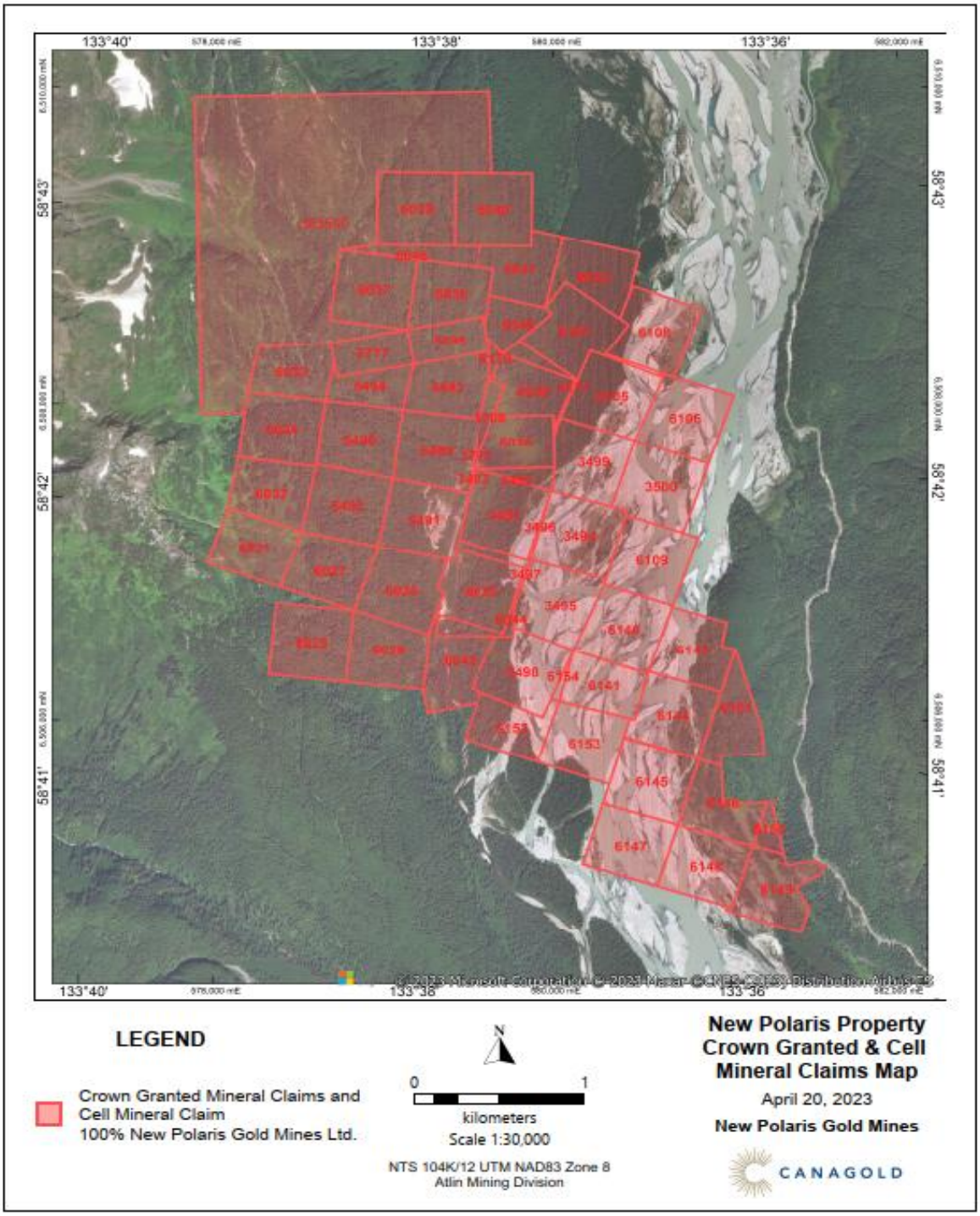
Table 4-1: Modified Grid Claim

Claim Name	Tenure No.	Issue Date	Expiry Date
W.W.1	353540	February 4, 1997X	February 4, 2026

Table 4-2: List of Crown Granted Claims

Claim Name	Lot No.	Folio No.	Claim Name	Lot No.	Folio No.
Polaris No. 1	6109	4472	Snow	3497	4545
Polaris No. 2	6140	5223	Snow No. 2	3495	5088
Polaris No. 3	6141	5223	Snow No. 3	3494	5495
Polaris No. 4	3498	4545	Snow No. 4	3499	5495
Polaris No. 5	6143	5223	Snow No. 5	6105	4472
Polaris No. 6	6144	5223	Snow No. 6	6106	4472
Polaris No. 7	6145	5223	Snow No. 7	3500	4472
Polaris No. 8	6146	5223	Snow No. 8	6107	4472
Polaris No. 9	6147	5223	Snow No. 9	6108	4472
Polaris No. 10	6148	5290	Black Diamond	3491	4472
Polaris No. 11	6149	5290	Black Diamond No. 3	6030	4944
Polaris No. 12 Fr	6150	5290	Blue Bird No. 1	5708	4545
Polaris No. 13 Fr	6151	5290	Blue Bird No. 2	5707	4545
Polaris No. 14	6152	5290	Lloyd	6035	5010
Polaris No. 15	6153	5290	Lloyd No. 2	6036	5010
Silver King No. 1	5489	4804	Rand No. 1	6039	5010
Silver King No. 2	5490	4804	Rand No. 2	6040	5010
Silver King No. 3	5493	4804	Minto No. 2	6033	4944
Silver King No. 4	5494	4804	Minto No. 3	6034	4944
Silver King No. 5	5491	4804	Jumbo No. 5	6031	4944
Silver King No. 6	5492	4804	Ready Bullion	6032	4944
Silver King No. 7	5495	4804	Roy	6042	5088
Silver King No. 8	5717	4545	Frances	6041	5010
Silver Queen No. 1	6026	4545	Eve Fraction	6170	5495
Silver Queen No. 2	6027	4545	Eve No. 1 Fraction	6171	5495
Silver Queen No. 3	6028	4944	P.T. Fraction	3493	5495
Silver Queen No. 4	6029	4944	Ant Fraction	3492	5088
Silver Strand	6037	5010	Atlin Fraction	3496	5088
Silver Strand No. 2	6038	5010	Powder Fraction	6043	5088
F.M. Fraction	6044	5088	Jay Fraction	6045	5088
Par Fraction	6154	5290			

Figure 4-1: Claim Location Map



Source: Canagold, 2023.

5 ACCESSIBILITY, CLIMATE, LOCAL RESOURCES, INFRASTRUCTURE AND PHYSIOGRAPHY

5.1 Physiography

The New Polaris Project area lies on the eastern flank of the steep, rugged, Coast Range Mountains, with elevations ranging from sea level to 2,600 m above sea level.

Extensive glaciation was the dominant factor in topographic development. The Taku and Tulsequah Rivers are the most prominent topographic features with broad valleys bounded by steep mountains. Numerous tributary streams flow from valleys filled with glaciers. Most of the glaciers are fingers branching from the extensive Muir ice cap, lying to the northwest of the Taku River. The Tulsequah glacier, which terminates in the Tulsequah valley, approximately 16 km north of the New Polaris Mine site, is one of the largest glaciers in the immediate area. It forms a dam causing a large lake in a tributary valley that breaks through the ice barrier (Jakülhlaup) during the spring thaw every year, flooding the Tulsequah and Taku valleys below for three to five days.

5.2 Accessibility

Site access is available by small aircraft from the nearest population centers: Atlin, BC, located 100 km north of the property, and Juneau, Alaska, approximately 60 km southwest. A short airstrip for light aircraft exists on the property. The nearest roads terminate 20 km south of Atlin and 10 km southeast of Juneau. In the past, shallow draft barges have been used to access the site via the Taku River, allowing for the transport of bulk supplies and heavy equipment to the site, as well as the shipment of flotation concentration from it. The property can be operated year-round.

5.3 Climate

The climate is one of heavy rainfalls during the late summer and fall months, and comparatively heavy snowfall, interspersed with rain during the winter. The annual precipitation is approximately 1.5 m of which 0.7 m occurs as rainfall. The snow seldom accumulates to a depth greater than 1.5 m on the level. Winter temperatures are not severe and rarely fall below –15ºC. Summer temperatures, in July, average 10ºC with daytime temperatures reaching the high 20's on occasion. The vegetation is typical of northern temperate rain forest, consisting primarily of fir, hemlock, spruce and cedar forest on the hillsides and aspen and alder groves in the river valley.

5.4 Local Resources and Infrastructure

Sufficient land is available within the mineral tenure held by Canagold for installations such as the process plant and related mine infrastructure. Surface rights for the areas occupied by the CSF, road to the airstrip and barge landing site, and the airstrip and barge landing sites lie with the Crown and land use permits will need to be obtained from the Province of British Columbia for these areas.

6 HISTORY

This section has been taken from the previous NI43-101 report completed by MMTS (MMTS, 2023).

From 1923 to 1925, the Big Bull and Tulsequah Chief properties were discovered along the east side of the Tulsequah River and opened up the Taku River district. In 1930, Noah A. Timmins Corporation optioned some of the claims that make up the New Polaris property and then conducted trenching and diamond drilling in 1931. The trenching exposed several veins, of which ten showed promising grades. A short exploration adit approximately 9 m long (30 feet) was also driven into the side of the hill and Timmins drilled 19 holes for a total of 1,615 m (5,297 feet) but was unable to correlate the intersections and elected to drop the option in September 1932.

The Alaska Juneau Gold Mining Company then optioned the property and conducted underground exploration from the "AJ" (Alaska Juneau) adit. Alaska Juneau drove a total of 190 m of drifting (625 feet) and, although they intersected "ore-grade" mineralization. They too had problems with correlation and dropped the property in the fall of 1934.

H. Townsend and M.H. Gidel of the Anaconda Corporation examined the property in 1934, carefully mapping the showings. They concluded that commercial ore bodies existed even though these showed irregularity due to faulting. Samples were sent to Geo G Griswold in Butte, Montana, who obtained gold recoveries from flotation tests in the order of 88%.

D.C. Sharpstone then secured an option on the property on behalf of Edward C. Congdon and Associates of Duluth, Minnesota. Congdon conducted 236 m (775 feet) of underground exploration in the "AJ" tunnel and collared 26 m (85 feet) into the canyon adit. The Polaris-Taku Mining Company was then incorporated in 1936 to take over the property from Congdon. Polaris-Taku erected a 150-ton per day flotation mill in 1937 and mined underground continuously until it was closed down in April 1942 due to labor restrictions brought on the Second World War. Mining operations resumed in April 1946 and continued until 1951 when the mine was closed due to high operating costs, a fixed gold price, and the sinking of a concentrate barge shipment during a storm in March 1951. Up to date, 231,604 oz of gold were produced at a recovered grade of 0.3 oz/ton.

An Edwards Roaster and a cyanide plant to produce bullion were installed and tested in 1949 in order to improve recovery and reduce shipping cost of concentrates to the Tacoma smelter. The addition of the roaster helped improve milling economics, but its capacity was somewhat limited as it could treat only approximately 45% of the concentrates produced from the flotation plant. After closure, the mill was leased to Tulsequah Mines Ltd (owned by Cominco) who modified it to process 600 t/d of massive sulphide polymetallic ore (containing gold, silver, copper, lead and zinc) from the Tulsequah Chief and Big Bull Mines. Tulsequah Mines Ltd used the mill from 1953 to 1957.

Numalake Mines acquired the property in 1953, changed their name to New Taku Mines Ltd (New Taku) and undertook rehabilitation work of the mine's plant. A negative feasibility study in 1973 halted this work. New Taku changed its name to Rembrandt Gold Mines Ltd in 1974. The property lay idle until Suntac Minerals Corp. (Suntac) optioned the property in 1988 and started surface exploration. Canarc merged with Suntac in 1992 and acquired a 100% interest from Rembrandt in 1994, subject to a 15% net profit interest, which Canarc can reduce to 10%. Canagold is challenging the validity of this historic net profit interest on the project. Canarc's subsidiary, New Polaris Gold Mines Ltd (formerly Golden Angus Mines Ltd), currently operates the property. Work on the property since 1988 is discussed in later sections of this report.

7 GEOLOGICAL SETTING AND MINERALIZATION

This section has been taken from the previous NI43-101 report done by MMTS (MMTS, 2023).

7.1 Regional Geology

The New Polaris Mine lies on the western edge of a large body of Upper Triassic Stuhini Group volcanic rocks, which has been intruded by a Jurassic-Cretaceous granodiorite body is part of the Coast Plutonic Complex, north of the mine. Older Triassic volcanic rocks and earlier sediments underlie the Stuhini volcanic rocks (Figure 7-1).

The structural trend in the area is northwest-southeast, paralleling major faults and folds to the east and intrusive alignment to the west. The Triassic volcanic rocks and older sedimentary rocks have been folded and sheared with the Stuhini Group rocks being deformed into broad to isoclinal, doubly plunging symmetrical folds with large amplitudes.

7.2 Project Geology

Canagold has carried out extensive mapping of the Polaris-Taku property since the early 1990's. The work has been done by several employees and contractors and is shown in Figure 7-2. The gold deposit is hosted within an assemblage of mafic (basalt and andesite units) volcanic rocks altered to greenschist metamorphic facies. The orientation of these units is inconclusive because there are no marker beds in the sequence. It is thought that the units are steeply dipping (70º to 80º) to the north based on the orientation of the limestone/basalt interface at the southern portion of the property.

A serpentinite unit is located to the northeast, which was identified in recent (1996/97) drilling and underground mapping. This unit appears to form the eastern extent of mineralization. The age relationship is unclear, but it is assumed that the serpentinite is a later stage feature possibly associated with tectonism in the area.

The 'vein' zones are structurally controlled shear zones and are typified by silicification and carbonatization cross cutting actual quartz-carbonate veins. These zones have sharp contacts with the wall rock and form anastomosing ribbons and dilations. These zones have been deformed several times, which makes original textures difficult to determine. The zones are generally tabular in geometry forming en-echelon sheets within the more competent host lithologies.

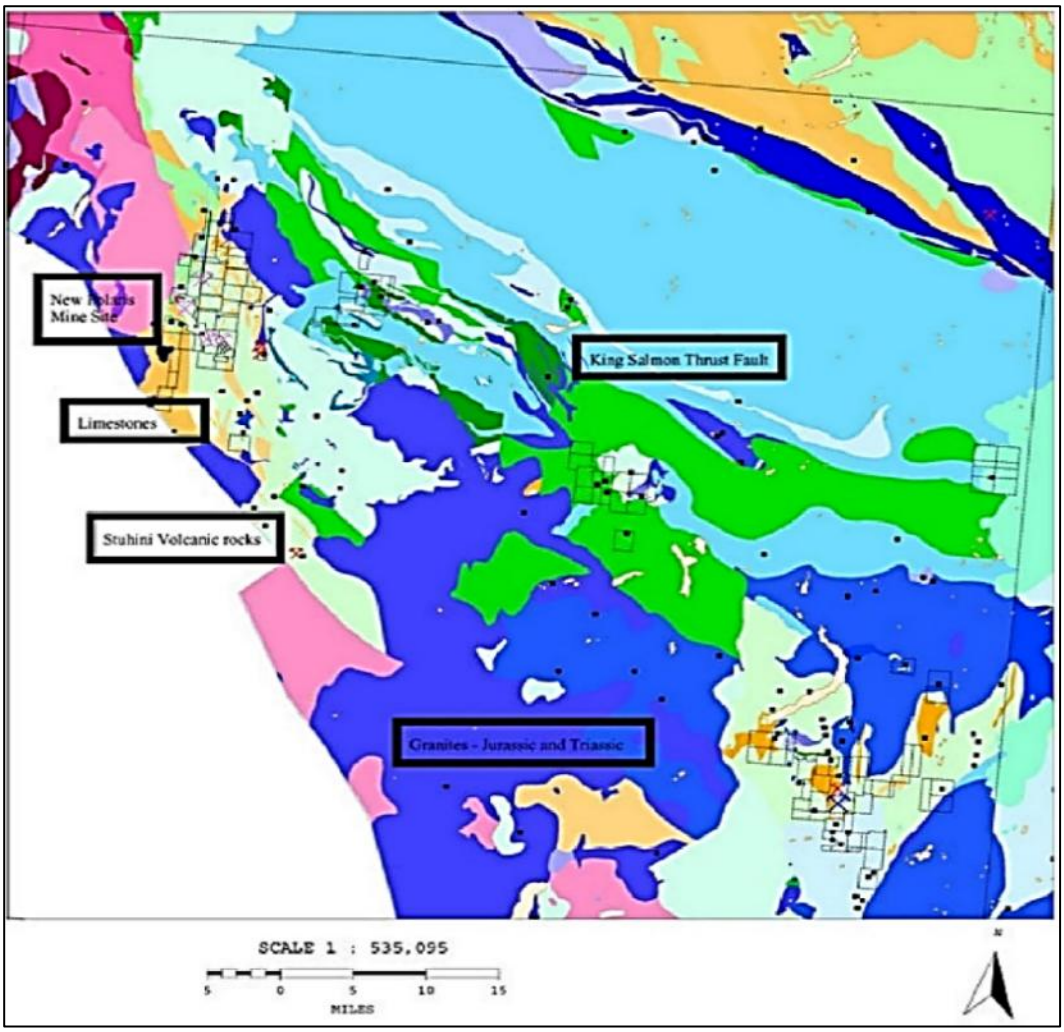
All of the strata within the property have been subjected to compression, rotation and subsequent extension. The plunge of folds appears to be variable though generally shallow. Small-scale isoclinal folds strike north-northwesterly and plunge moderately to the north. Numerous faults are found on the property, the more significant of which are discussed below.

The possible extension of the Llewellyn fault, termed the South Llewellyn fault, continues south from the Chief Cross fault along mine grid coordinate 4400 East. Slightly north of Whitewater Creek it is offset to the west by an east-west fault, the 101 fault, to continue in a more southeast orientation of the opposite side of Whitewater Creek. This northwest-southeast orientation structure was named the Limestone Fault due to its bedding parallel attitude within

a discontinuous limestone/marble horizon. It marks the southwest boundary of the “mine wedge”: the wedge-shaped package of rock within which all past production took place. The northern boundary of the “mine wedge” is further defined as mentioned above by the Whitewater Creek Schist Zone, a zone of schistose chlorite-amphibolite-serpentinite less than 100 m thick. A complex network of brittle faults is also found within this zone.

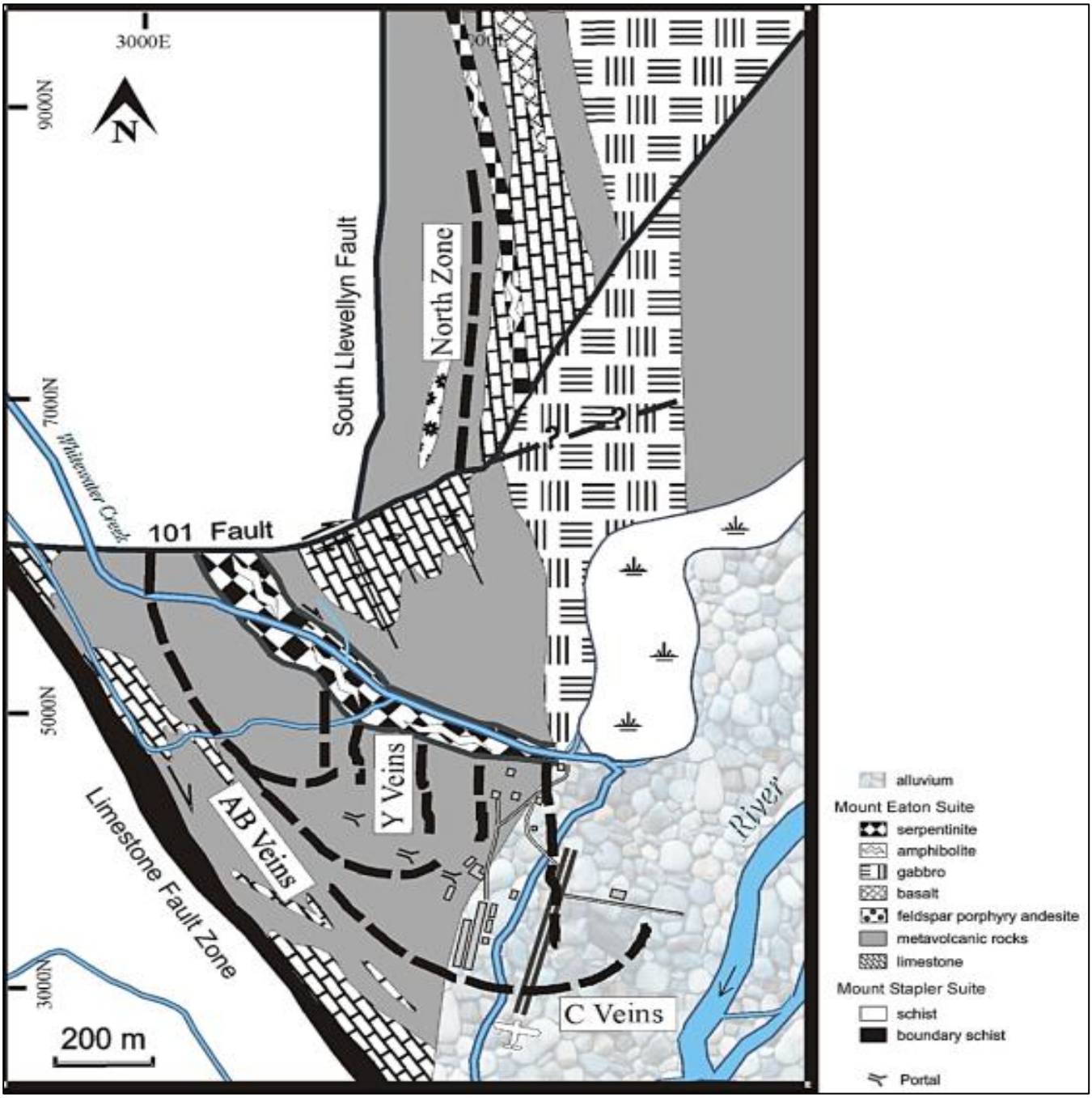
Three major faults, Numbers 1 and 5, and an unnamed fault, lie within the mine wedge. The No.1 and No.5 faults strike northwest-southeast, dipping approximately 45º to the northeast, and are sub-parallel to the unnamed fault, which dips steeply to the southwest. The No.1 fault has reverse displacement of up to 30 m while the displacement of the No.5 fault is poorly defined. The southwest dipping, unnamed fault showed no displacement, as it apparently parallels the A-B vein system. The mined-out areas indicate the wedge shape, the predominant orientations and continuity of the zones, and the overall plunge of the system to the southeast. An early interpretation of the structure showed that various veins appear to meet and form “junction arcs” where both thickness and grade improve.

Figure 7-1: Regional Geology



Source: MMTS, 2006.

Figure 7-2: Property Geology



Source: MMTS, 2006.

7.3 Mineralization

Mineralization of the New Polaris deposit bears strong similarities to many Archean lode gold deposits such as the arsenical gold camp of Red Lake, Ontario where the gold-bearing arsenopyrite is disseminated in the altered rock and in quartz-carbonate stringers.

The vein mineralization consists of arsenopyrite, pyrite, stibnite and gold in a gangue of quartz and carbonates. The sulphide content is up to 10% with arsenopyrite the most abundant and pyrite the next important. Stibnite is fairly abundant in some specimens but overall comprises less than one-tenth of 1% of the vein matter. The majority of the stibnite is concentrated in a high-grade core consisting of approximately 30% of the deposit while the remainder of the deposit having only very low stibnite concentrations. Alteration minerals include fuchsite, silica, pyrite, sericite, carbonate and albite.

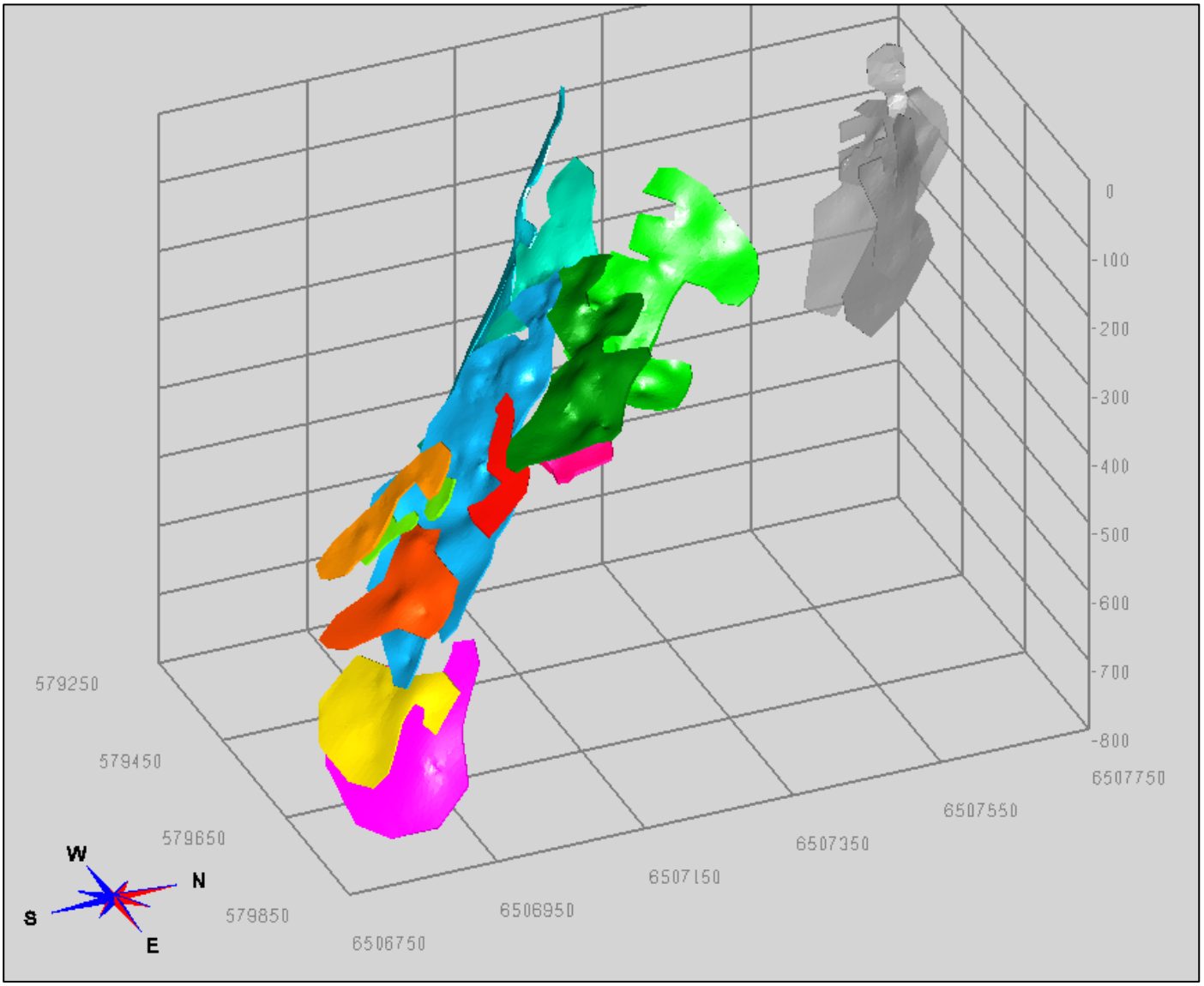
In general, the zones of mineralization ranging from 15 to 250 m in length with widths up to 14 m appear to have been deposited only on the larger and stronger shears. Their walls pinch and swell showing considerable irregularity both vertically and horizontally. Gold values in the veins have remarkable continuity and uniformity and are usually directly associated with the amount of arsenopyrite present. The prominent strike directions are north-south and northwest-southeast, which is interpreted to be within a major shear zone. Up to 80% of the mine production was from “structural knots” or what is now known as “C” zones. In detail the “C” zones are arcuate structures. (MMTS, 2023).

Figure 7-3 shows a 3D view of parts of the “C” and “Y” vein systems.

Vein mineralization has well marked contacts with the wall rock. The transition from mineralized to non-mineralized rock occurs over a few centimeters. The mineralization consists of at least three stages of quartz veining. The initial stage of quartz-ankerite introduced into the structure was accompanied by a pervasive hydrothermal alteration of the immediately surrounding wall rock. Arsenopyrite, pyrite and lesser stibnite were deposited with the alteration. Later stages of quartz-ankerite veining are barren and have the effect of diluting the gold grades in the structure. The sulphide minerals are very fine-grained and disseminated in both the wall rock and early quartz and ankerite veins. Free gold is extremely rare and to the end of 2005 had not been recognized in core samples. The majority of the gold occurs in arsenopyrite and to a lesser extent in pyrite and stibnite. Because there is no visible gold and the host sulphides are very fine-grained and disseminated there is little nugget effect and gold values even over short intervals rarely exceed 1 oz/ton.

Mineralization was observed by Sue Bird during the site visit both in drill core and underground. The description of the regional setting, local geology, and mineralization appears applicable to the New Polaris Project and is sufficiently well understood to support the estimation of Mineral Resources.

Figure 7-3: 3D Vein Model (colour = "C" veins, grey = "Y" veins)



Source: MMTS, 2003.

8 DEPOSIT TYPES

The New Polaris deposit is classified as a mesothermal lode gold deposit (Hodgson, 1993).

In general, it is quartz-vein-related, with associated carbonatized wall rocks. The deposits are characterized by a high gold/silver ratio, great vertical continuity with little vertical zonation, and a broadly syn-tectonic time of emplacement. They are commonly associated with pyrite, arsenopyrite, tourmaline and molybdenite. Mineralization may occur in any rock type and ranges in form from veins to veinlet systems, to disseminated replacement zones. Most mineralized zones are hosted by and always related to steeply dipping reverse- or oblique-slip brittle-fracture to ductile-shear zones.

The New Polaris Project is an orogenic lode gold deposit, also known as mesothermal vein deposit. Numerous examples of this type of deposit are known throughout the world including the Campbell Red Lake deposits in Ontario and the Bralorne deposit in British Columbia. Past exploration studies have demonstrated that the New Polaris vein systems have all the attributes of the orogenic vein gold deposit including, but not limited to association with major structural break, quartz-carbonate vein association, low-sulphide assemblage of pyrite and arsenopyrite, chloritic and sericitically altered wall rocks and persistent gold mineralization over a vertical distance of nearly 1 km.

The deposit type and model are considered by the QP as appropriate for a mesothermal lode gold deposit.

9 EXPLORATION

The New Polaris property represents an advanced exploration project on a former gold producer. The early exploration work in the area located gold mineralization on surface and subsequent exploration led to mining of approximately 689,500 tonnes of material grading 10 g/t of gold. More recent exploration work, since 1988, has been directed at gaining knowledge about the geology of the area and expanding the resource base of the mineralized zones.

Geological mapping, geochemical surveys, geophysical techniques, and drilling have added considerable value to the Project. Table 9-1 lists the relevant exploration work on the property along with contractor name and supervisor.

Table 9-1: Summary of Exploration Employees/Contractors

Year	Supervisor	Drilling Contractors	Laboratory	Underground
1988	Cloutier		TSL	
1989	Cloutier		TSL	
1990	Cloutier		TSL	
1991	Marriott	Arctic	Min En/Chemex	
1992	Marriott	Arctic	Chemex	
1993	Marriott/Moors	Arctic	Chemex	
1994	Moors	Arctic/Falcon	Chemex	
1995	Moors	Arctic/Falcon	Chemex	
1996	Karelse/Watkinson	Advanced	Northern	Main Street
1997	Karelse/Watkinson	Advanced	Northern	Main Street
2003	Moors	Hy-Tech	ALS Chemex	
2004	Moors/Aspinall	Hy-Tech	ALS Chemex	
2005	Moors/Aspinall	Hy-Tech	ALS Chemex	
2006	Moors/Cote	Hy-Tech	ALS Chemex	
2021	Gill	ITL	ALS	
2022	Gill	ITL	ALS	
2024	Liaghit	Atlas	ALS/MSA	

Surface mapping, geochemistry and geophysics over the “mine wedge” were completed by Orequest in 1988 and further surface mapping and geochemistry, on the “north grid” were completed in 1993 with results are shown in Figure 7-2.

Underground exploration included the rehabilitation of the AJ Level in 1988 and the rehabilitation of all the other levels, including the Polaris Portal, in 1996 and 1997. The underground rehabilitation also included a re-survey of the old workings so that the more recent surface work could be aligned with the old underground workings.

The procedures followed in the field and through the interpretation stage of exploration have been professional and are appropriate to the style of mineralization and current degree of geological knowledge and understanding of mineralization control.

Exploration work was conducted by crews under the supervision of professional geologists, and the data obtained is considered highly reliable. No exploration has occurred on site since 2024.

10 DRILLING

Diamond drill programs were carried out on the New Polaris Project when the project was reactivated in 1988 until 2024 (Table 10-1). Initially, the drilling focused on the down dip and along strike extensions of the Y veins. This work showed that the Y veins, while good grade, were narrow and less continuous than the AB vein system.

In 1990, drilling shifted to the area beneath the lowest-most C vein stopes. This drilling found that the vein system continued to depth and that gold grades in the 0.30 to 0.45 oz/ton range over an average true thickness of 3 m were present. From 1991 to 1993 most of the drillholes tested the C veins with fewer drilled on the Y vein system. Encouraging drill results from the C veins and to a lesser extent from the Y vein system led to further drilling on these two vein systems. Drilling on the C vein showed the veins to be open to depth and to have gold grades that ranged from 0.2 to 0.6 oz/ton over true thicknesses of 3 m.

In 1996 and 1997 the Y, C and AB veins were explored from underground. The plan was to closely test the upper portions of the Y, C and AB veins in order to allow calculation of a resource that might form the basis for resumption of mining. The results of the underground drilling program were mixed. The underground workings were for the most part driven along the vein structures with few crosscuts from which holes could be drilled to cut the down dip and along strike extension of the veins. As a result, except for those holes that tested the area immediately below the workings, most cut the veins at shallow angles. The very shallow angles that in places approach parallel to the vein make the use of these intersections inappropriate for a resource calculation (an example is hole 97-44 that cut 34.1 m grading 0.42 oz/ton). Despite the number of holes drilled during 1996 and 1997, the work did little to expand the extent of mineralization in the AB, C or Y vein systems. The work confirmed that the mineralized shoots in the lower most stopes on the Y and C veins were open to depth.

Poor market conditions after 1997 made financing of the New Polaris Project difficult. Drilling restarted on the property in 2003 with the objective of testing the extent of C vein mineralization. Canagold undertook a review of the Polaris Project and recommended additional drilling in order to test the continuity of the C vein zone mineralization at depth below the lower most mine workings. To this end, limited drill programs were conducted from 2003 to 2006. The 2003 to 2006 exploration programs targeted C vein extensions below the existing mine workings.

In 2021 to 2022 an additional two-phase drilling program was completed. Phase 1 was designed primarily to collect further details on the Inferred Resources of the C vein system, predominately in the West Main domain. The 52 infill drill holes had a target depth ranging between 300 m to 650 m and were designed to provide greater density of drill intercepts at approximately 25 m spacing along section lines. Two drill holes were considered abandoned with no attempt to re-drill. One very deep hole with two additional wedge holes were completed as part of the program to attempt to intersect the down dip projection of the CWM domain. The drilling confirmed the downdip extension of the C veins.

Phase 2 of the 2021 to 2022 drilling program was designed primarily to collect further details on the inferred Resources of the Y Vein system. The 45 infill drill holes had a target depth ranging between 29 m to 749 m and were designed to provide greater density of drill intercepts. One of the drill holes was aimed at intercepting the gap between two of the “C” veins and two holes were intended to infill the C East lenses.

The results of the 2021 to 2022 drilling of the "C" and "Y" vein systems confirmed the continuity of gold mineralization and the vein structure between the earlier drilled holes. As can be seen in the figures below, drill results show the "C" vein system to be an arc-like structure generally oriented east-west with a shift in strike in the west. The change in strike occurs across the No.1 fault. To the east of the No.1 fault, the vein splays into two or more branches. The dip of the vein system is to the south and southeast and has an average dip of approximately 50°, although east of the No.1 fault the vein appears to flatten and thicken in a sigmoid-like feature.

All of the holes in this period were drilled from surface and intersected a similar geologic sequence. From the collar, the holes penetrated 15 m to 79 m of overburden followed by inter-layered ash and lapilli tuff, volcanic wacke, and foliated andesite. The "C" vein system crosscuts the strike of the volcanic and volcaniclastic rocks at steep angles.

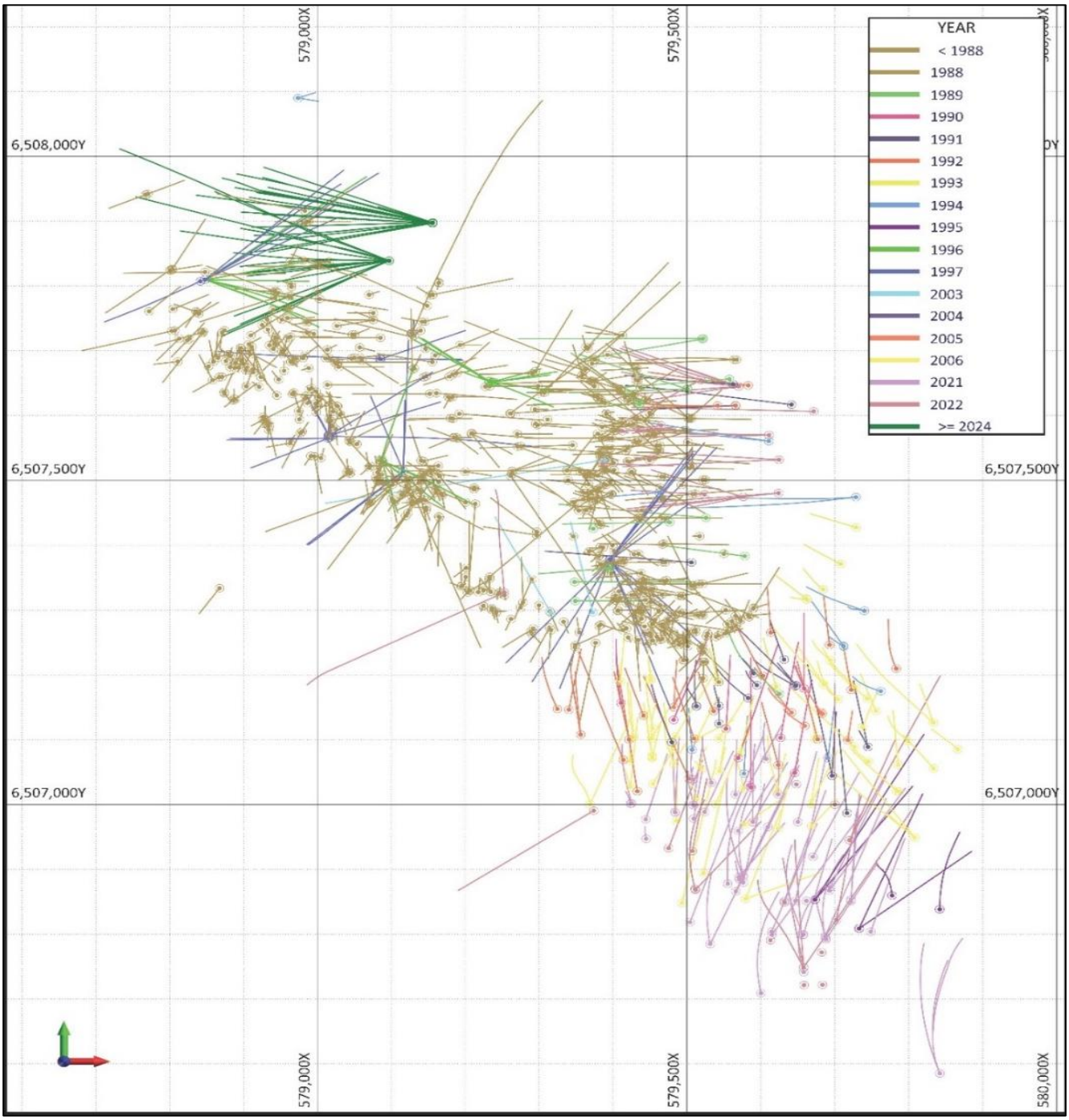
During 2024, 13,000 meters of drilling were completed in 34 holes targeting the North Zone. The North Zone is near the old mining area where there were some historic drill holes. The goal of the program was to add resources which could be accessed early in the mine life in order to reduce initial capital costs. Although mineralization was intercepted in a number of parallel veins, it wasn't significant enough to warrant up front mining or updates to the gold resource at this time.

A plan view of the collar locations for the drillhole used for the resource estimate are plotted on Figure 10-2. A representative long section plots showing the pierce points of drillholes and grades of the relevant drilling is presented in Figure 10-2.

Table 10-1: Diamond Drillhole Summary (1988 to 2024)

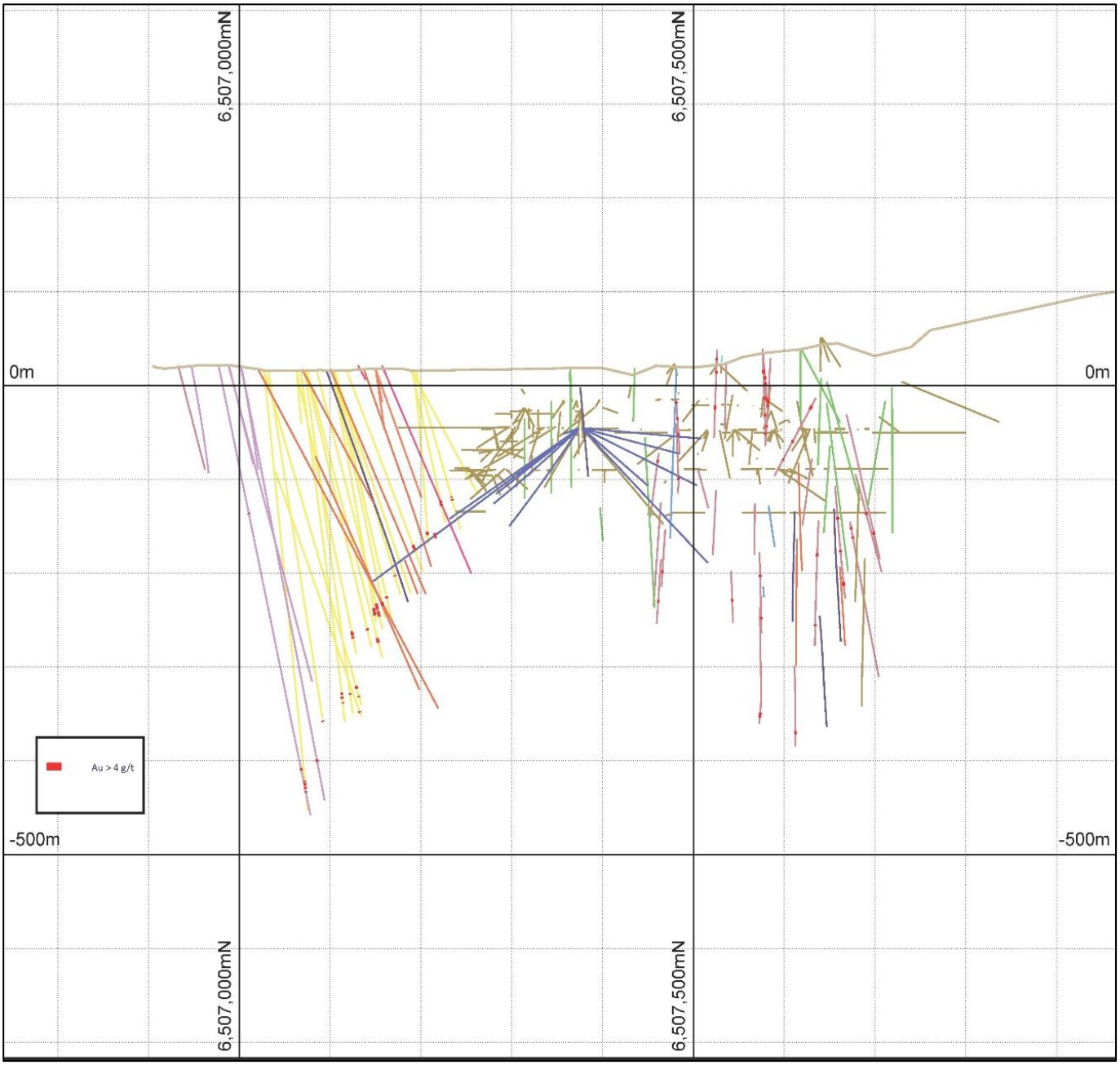
Year	Supervisor	Drilling Contractors	Laboratory	Number of Collars	Total Length Drilled (m)
<1988	-	-	-	778	32,483. 05
1988	Cloutier	-	TSL	8	1,027.79
1989	Cloutier	-	TSL	21	4,490.75
1990	Cloutier	-	TSL	9	2,862.46
1991	Marriott	Arctic	Min En/Chemex	11	3,436.77
1992	Marriott	Arctic	Chemex	21	5695.40
1993	Marriott/Moors	Arctic	Chemex	7	1,142.39
1994	Moors	Arctic/Falcon	Chemex	29	5,719.89
1995	Moors	Arctic/Falcon	Chemex	17	7,572.76
1996	Karelse/Watkinson	Advanced	Northern	24	3,204.69
1997	Karelse/Watkinson	Advanced	Northern	43	7,210.84
2003	Moors	Hy-Tech	ALS Chemex	3	1,560.89
2004	Moors/Aspinall	Hy-Tech	ALS Chemex	11	2,766.99
2005	Moors/Aspinall	Hy-Tech	ALS Chemex	9	2,389.63
2006	Moors/Cote	Hy-Tech	ALS Chemex	72	24,801.60
2021	McLaughlin/Dupuis	ITL Diamond Drilling Ltd	ALS Geochemistry	52	25,512.28
2022	Dupuis/Smith	ITL Diamond Drilling Ltd	ALS Geochemistry	45	14,324.40
2024	McCrory/Davies	Atlas Drilling	ALS & MSA labs Geochemistry	34	13,000

Figure 10-1: Plan View of Drill hole Locations by Year



Source: MMTS, 2025.

Figure 10-2: Section of the Drill holes and Gold Intercepts >4 g/t. Section depth ±50 m



Source: MMTS, 2025.

11 SAMPLE PREPARATION, ANALYSES, AND SECURITY

11.1 Sample Method and Approach

11.1.1 Year 2003–2006

Drilling of the vein was done by wire line diamond drills using NQ-size rods. Drill collar locations were surveyed in by total station surveying method. Drilling azimuth and dip were set using a Brunton compass and inclinometer. Routine downhole measurements of azimuth and dip were not done on the three holes drilled in 2003 and prior. In 2004, three different downhole survey systems were tried before settling on a Reflex system. The Reflex system was also used in 2005. The downhole survey tool was operated by the Hy-Tech drill crew. This information was entered into a GEMCOM program to plot the location of the collar and the pierce point of the veins.

Core recovery was very good and ranged from the low 90% to nearly 100% and is a fair sampling of the mineralization at the point where the drillhole pierced the vein.

Determining intervals of core for sampling was done by the geologist during logging of the core. The mineralized vein structures were marked out. Selections of core intervals for sampling were based in the presence of veining and sulphide mineralization, particularly arsenopyrite. Within the defined vein structure sample intervals ranged from 0.3 m to 1.5 m. Divisions were based on intensity of mineralization and veining. Sampling of the core over several meters on either side of the mineralized vein structures was also done to the point where hydrothermal alteration disappeared. No sampling of core from the unaltered rock was done.

The core was logged and stored in the camp. Access to the core was only available to the geologists and the core sampler. The core was brought from the drill to the logging facility by the geologist at the end of each shift. The core was geologically logged, recoveries calculated, and samples marked out in intervals of 0.5 to 1.0 m. The core was handed to the sample cutter who cut it into two equal halves with a diamond saw. Each half-core sample was individually wrapped in plastic bags for shipment while the corresponding other half remained in the core boxes for reference or future work. The sample intervals were easily identified and correlate well with drill logs.

In 2003, core samples were sent to ALS in Vancouver, BC, for prep and analysis. Sample preparation consisted of crushing to 70% passing 2 mm (CRU-31), homogenizing and generating a 250 g split using a riffle splitter (SPL-21), then pulverizing to 85% passing 75 microns (PUL-31). A 4-acid digestion with ICP-AES finish was selected (ME-ICP61), which reports 33 elements. Gold was analyzed using an industry-standard 30 g fire assay with atomic absorption spectroscopy (Au-AA23) finish. Samples returning over-limit gold results (>10 g/t Au) were reanalyzed using the same pulp with a gravimetric finish (Au-GRA21).

For samples taken in 2004, gold preparation and analysis followed the same procedures as in 2003, but instead of requesting a full ICP analysis from ALS on all samples sent, only selected high-grade gold samples were specifically analyzed for arsenic (As-AA46) and antimony (Sb-AA08). The Sb-AA08 method uses a $KClO_3$ and HCl based digestion with an atomic absorption spectrometry (AAS) finish.

The 2005 drill core samples were prepared and analyzed using the same methods as the samples from 2003, except for those from drill hole 05-1676E1, which were only analyzed for gold (Au-AA23 and Au-GRA21, where applicable). Over-limit antimony analyses (>10,000 ppm) from the ME-ICP61 method were not performed, which affected a total of five samples from three mineralized intercepts in 2005.

In 2006, all 1,635 samples taken that year were exclusively analyzed for gold using ALS Vancouver's ore-grade gravimetric method (Au-GRA21). The lower reporting limit for this gravimetric method is 0.05 g/t.

11.1.2 Years 2021-2022

During the 2021 to 2022 drilling program the core was geologically logged to identify the gold mineralized zones that were allocated unique sample number tickets and marked for cutting using a purpose-built diamond blade rock saw. Half-core samples were collected in labelled bags and the other half remains in the original core box stored on site. Quality control (QC) samples including certified reference material standards, blanks and duplicates were inserted into the sample sequence at intervals of one in ten on a rotating basis to monitor laboratory performance and provide quality assurance (QA) of the assay results. Several sample bags were transported together in rice bags with unique numbered security tags attached and labelled with company and lab contact information to ensure sample security and chain of custody during shipment to the lab.

The samples were submitted to the ALS Geochemistry lab in Whitehorse, YT in 2021 and Yellowknife, NT 2022 for preparation and assaying. ALS Canada Ltd is accredited by the Standards Council of Canada and is an ISO/IEC 9001:2015 and 17025:2017 certified analytical laboratory in North America.

The entire sample was crushed to 70% passing -2 mm, and a 250-g aliquot is split and pulverized to 85% passing -75 microns. Analysis for gold was by 30-g fire assay and gravimetric finish (Au-GRA21). A suite of 30 other elements including arsenic, antimony, sulphur and iron were analyzed by aqua regia digestion and Inductively Coupled Plasma Atomic Emission Spectroscopy (ICP-AES) finish (ME-ICP41a). The ME-ICP41a method has an upper reporting limit of 50,000 ppm for antimony. Fifteen of 2,634 drill core samples in 2021, and one of 1,561 samples in 2022, exceeded this limit; over-limit antimony analyses were not performed.

Sample preparation, analysis, and security procedures follow accepted engineering standards, and the quality of gold analytical data collected by Canagold is sufficiently reliable to support Mineral Resource estimation.

11.2 QA/QC – 2003-2006

As part of the QA/QC program during these years, samples were collected on site by Canagold personnel and shipped to the ALS Chemex laboratory in Vancouver. Beginning in 2005, blind blanks, field duplicates, and blind standards were inserted at regular intervals for quality assurance (see Table 11-1). The 2004 dataset is contamination-controlled by frequent blank insertions, and no QA/QC data is currently available for the 2003 assays. Overall, the QP believes that these earlier field programs were run by Canagold in an efficient and proper manner following accepted industry standards.

Table 11-1: 2003–2006 QA/QC Summary

Year	Lab	Core Samples	Blanks	CRMs	Field Dup	Coarse Dup*	Pulp Dup	QA/QC Total	% QA/QC	Check	Comment
2003	ALS	170	0	0	0	0	0	0	0.0%	0	
2004	ALS	463	16	0	0	0	0	16	3.3%	0	
2005	ALS	499	30	36	23	0	0	89	15.1%	0	
2006	ALS	1,474	56	60	44	1	0	161	9.8%	0	LABDUP in DB
Total		**2,606**	**102**	**96**	**67**	**1**	**0**	**266**	**9.3%**	**0**	

To demonstrate the performance of the blanks in this report, the available assay data is plotted against a warning threshold of 5 x DL (detection limit) of the respective analytical method and 10 x DL to indicate failure which should have triggered a data review and possible re-prepping and re-analysis at the lab at the time. MMTS is not aware of any re-run data certificates.

To control accuracy of results, Canagold used a set of two (2005) and three (2006) blind Au-certified reference materials or standards (CRM or STD) of variable Au grades. For this report, MMTS graphs the CRM results using a simple z-score normalization by subtracting the expected Au value (EV) from the reported assay value and divide the result by the 'in-between' standard deviation (SD) as was calculated in the certificate of analysis of the used CRM. A score of ±3 shows the threshold past which the result is classified as a 'failure' while a score exceeding ±2 indicates a warning only.

To illustrate reproducibility of results by today's standards, duplicates are to be taken at different stages of the sampling and sample size reduction processes. For 2005 and 2006, Canagold introduced field duplicates at acceptable rates by quarter-cutting the core and sending two individual samples representing the same drill core interval instead of only one. Coarse duplicate and pulp duplicate sampling in addition to the regular lab-internal QA/QC protocols at ALS have not been requested by the company.

11.2.1 Blanks 2003–2006

Blank samples represent material from the old mine, known to contain very low gold values. A total 102 blank samples were assayed, of which four exceeded the 10 x DL threshold for a failure rate of 3.9%. The blank statistics are shown in Table 11-2.

Table 11-2: 2003–2006 Blanks Performance Summary

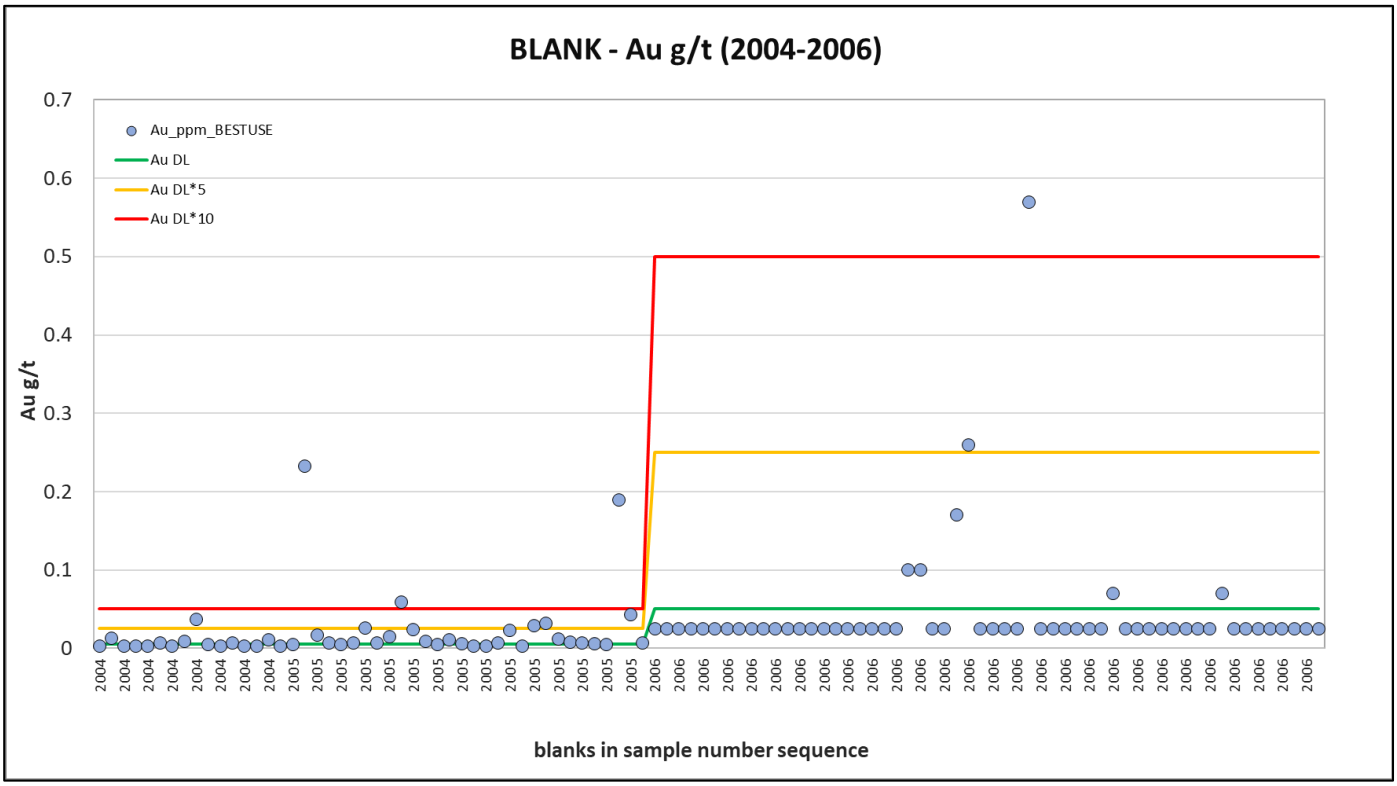
Year	Lab	Core Samples	BLK Count Au	Fails Au	% Fails Au
2003	ALS	170	0	0	0.0%
2004	ALS	463	16	0	0.0%
2005	ALS	499	30	3	10.0%
2006	ALS	1,474	56	1	1.8%
Total		**2,606**	**102**	**4**	**3.9%**

Figure 11-1 presents the test results of the 102 recorded blank samples, corresponding to an insertion rate of approximately 4% based on the total samples taken between 2003 and 2006. This is overall acceptable, but the 2003 data must be considered not independently controlled by the laboratory.

There are no indications of significant or systematic cross-sample gold contamination. However, it is noted that in the 2005 dataset, the blank insertions rarely follow high-grade gold intervals, and the blank material used may have been weakly mineralized. In 2006, the higher detection limit of the utilized Au-GRA21 method raised the failure threshold to 0.5 g/t Au, which resulted in only one failure of 0.57 g/t in sample C090930 in drill hole 06-1768E1A, following a 55 g/t Au result in the previous sample.

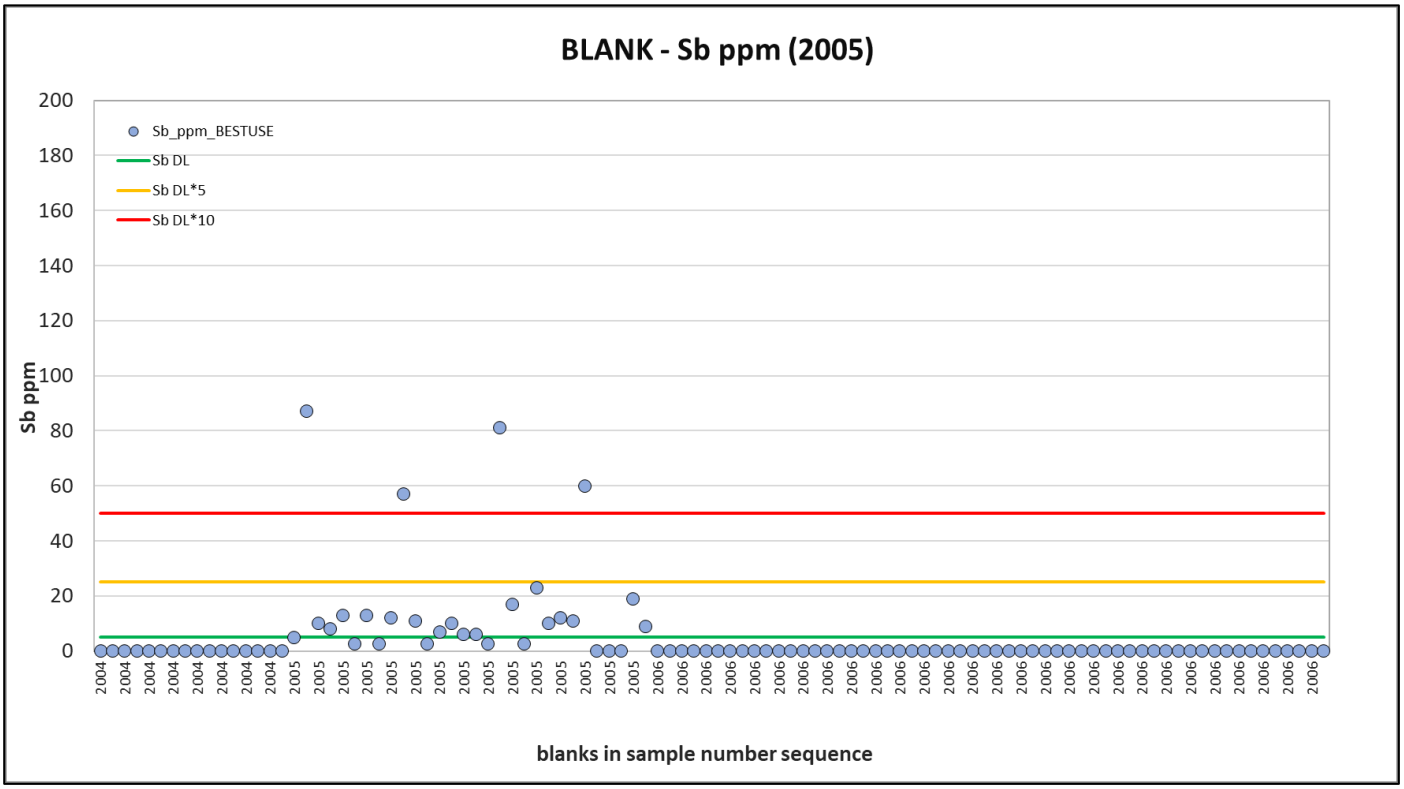
Figure 11-2 illustrates the blank performance with respect to antimony from 2003 to 2006. The graph keeps the same sorting and time frame as Figure 11-1 even though samples and therefore blanks were only analyzed for antimony consistently in 2005. There are four theoretical failures with a maximum of 87 ppm which is inconsequential. Spot-checking confirmed that the 'failure' blanks do not follow high-grade antimony intercepts, so these results are probably natural variability in the blank material.

Figure 11-1: Blank Performance 2004–2006 – Au



Source: MMTS, 2025.

Figure 11-2: Blank Performance 2005 – Sb



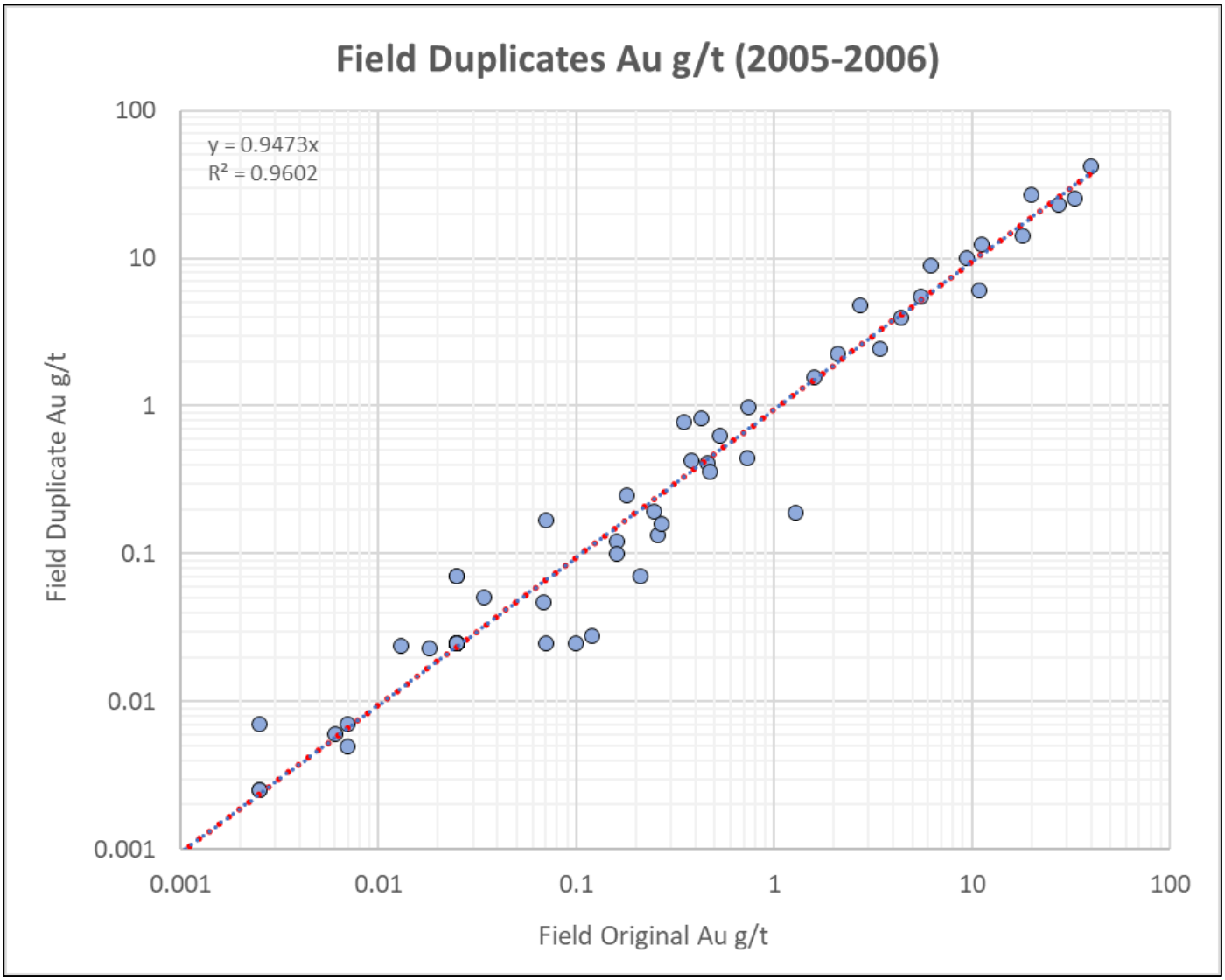
Source: MMTS, 2025.

11.2.2 Duplicates – 2003–2006

Field duplicate samples were made by cutting half core into quarters and submitting them as separate samples. A total of 65 field duplicate samples were taken and assayed in 2005–2006. No additional duplicate data is currently available, meaning that sampling, preparation, and assaying from 2003 and 2004 were not controlled for reproducibility. MMTS did not review the lab-internal duplication results for this report.

The scatter plot presented in Figure 11-3 shows how the field duplicate population correlates to the respective original samples (parents). The correlation coefficient ($R^2 \approx 0.96$), which is very good. The data does not indicate significant bias; the scatter is considered acceptable, with a half absolute relative difference (HARD) average of 7.7%. Also, the data covers a representative grade range from the detection limit to approximately 40 g/t Au. The scatter is assumed to be caused by mineralogical variability at the sample scale rather than by preparation or analytical error.

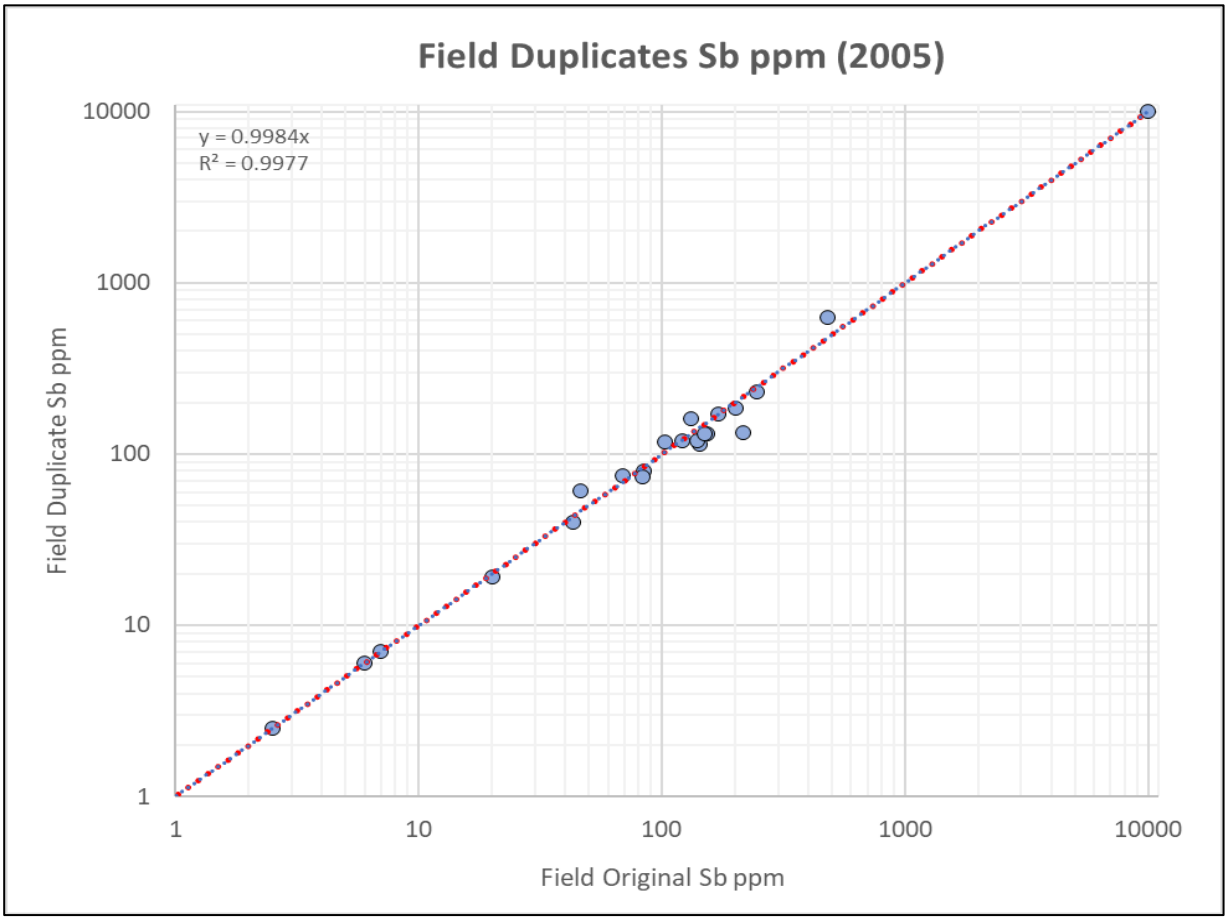
Figure 11-3: Field Duplicate Performance – Au



Source: MMTS, 2025.

Only 22 of the 65 field duplicate sample pairs have been analyzed for antimony (Sb), all in 2005. The small dataset indicates a very good correlation, with an R2 value of 1.00, as shown in Figure 11-4. However, the highest original-duplicate pair, at 10,000 ppm, was not analyzed using a suitable over-limit method, and is therefore plotted at 10,001 ppm on both axes. Removing the pair drops the R2 to 0.95, which is still strong. It should be noted that the expected grade range at New Polaris is not well represented in the duplicate dataset, as most grades fall below 600 ppm, while certain intervals from the 2005-2006 core can contain up to 16% Sb.

Figure 11-4: Field Duplicate Performance – Sb



Source: MMTS, 2025.

11.2.3 Standards – 2003–2006

Five different commercial gold standards were used during the 2005 and 2006 drilling and sample programs. These standards were inserted 'blind' at acceptable frequencies approaching 5% of all core samples to test the analytical accuracy of the laboratory. No standard insertions were recorded for the 2003 and 2004 drill seasons.

Table 11-3 summarizes the total number of counts, certification details, and failure results for the standards. The certified gold values of the selected standards range from high (2.37 g/t) to very high (16.21 g/t), which are suitable grades for a gold system such as New Polaris. The combined failure rate for the 96 standards of 10.4% is quite high, particularly due to the performance of high-grade gold standard PM916, which reported five low and two high results on 18 insertions in 2006. Given the overall acceptable performance of the other four standards, and the fact that laboratory and fire assay methods were unchanged, MMTS concludes that the poor result is probably caused by sub-optimal homogeneity of the PM916 standard itself.

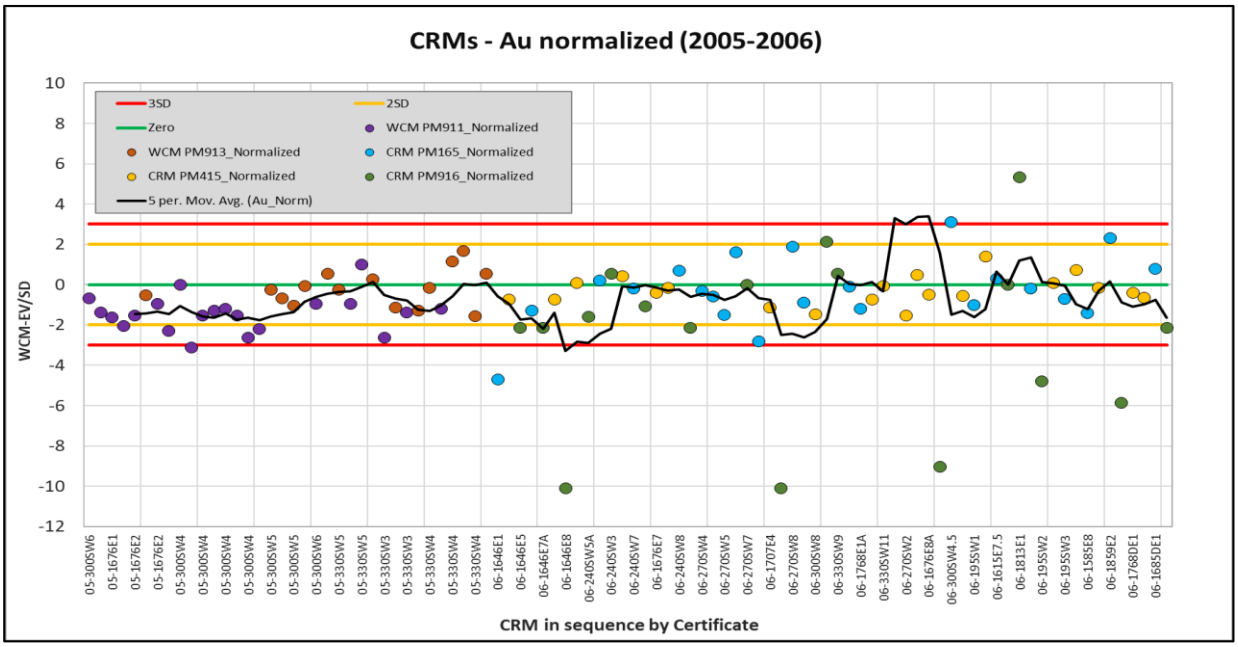
MMTS also notes that PM911 was regularly analyzed below its expected value, though it only failed once based on the ±3 z-score threshold, which results in a moderate low bias for 2005, as illustrated by the 5-sample moving average trend (black line) shown in Figure 11-5.

None of the five standards are certified for antimony. Based on the available assay results, PM913 averaged 19 ppm Sb, with reported values ranging between 14 ppm and 22 ppm. PM911 averaged even lower at 15 ppm (min 10 ppm and max 20 ppm), which would have made them unsuitable for the New Polaris system even with certification. The data is not plotted for this report.

Table 11-3: 2005–2006 Standards Performance Summary

Standard	Year	Count Au	Au g/t Mean	Au g/t EV	Au g/t SD	Au % Error	Au Fails High	Au Fails Low	Au Fails %
PM165	2006	22	6.48	6.51	0.10	-0.4%	1	1	9.1
PM415	2006	20	2.33	2.37	0.12	-1.6%	0	0	0.0
PM911	2005	21	15.36	16.21	0.59	-5.5%	0	1	4.8
PM913	2005	15	5.45	5.47	0.12	-0.4%	0	0	0.0
PM916	2006	18	12.58	12.70	0.09	-1.0%	2	5	38.9
Total		**96**					**3**	**7**	**10.4**

Figure 11-5: PCC Standards Performance 2005–2006 – Au



Source: MMTS, 2025.

11.2.4 Check-Assays - 2006

A preliminary set of sample pulps was selected for check-assay by a secondary laboratory. Acme Analytical Laboratory Ltd, a highly accredited lab in Vancouver, was chosen for this work. The results were generally very consistent, with the exception of one sample from the ALS Chemex Sample 1 set, which assayed 22 g/t, compared to 31.4 g/t by Acme and 31.4 g/t by ALS Chemex the second time. A complete analysis of the check-assay data is provided in the previous PEA report (MMTS, 2019).

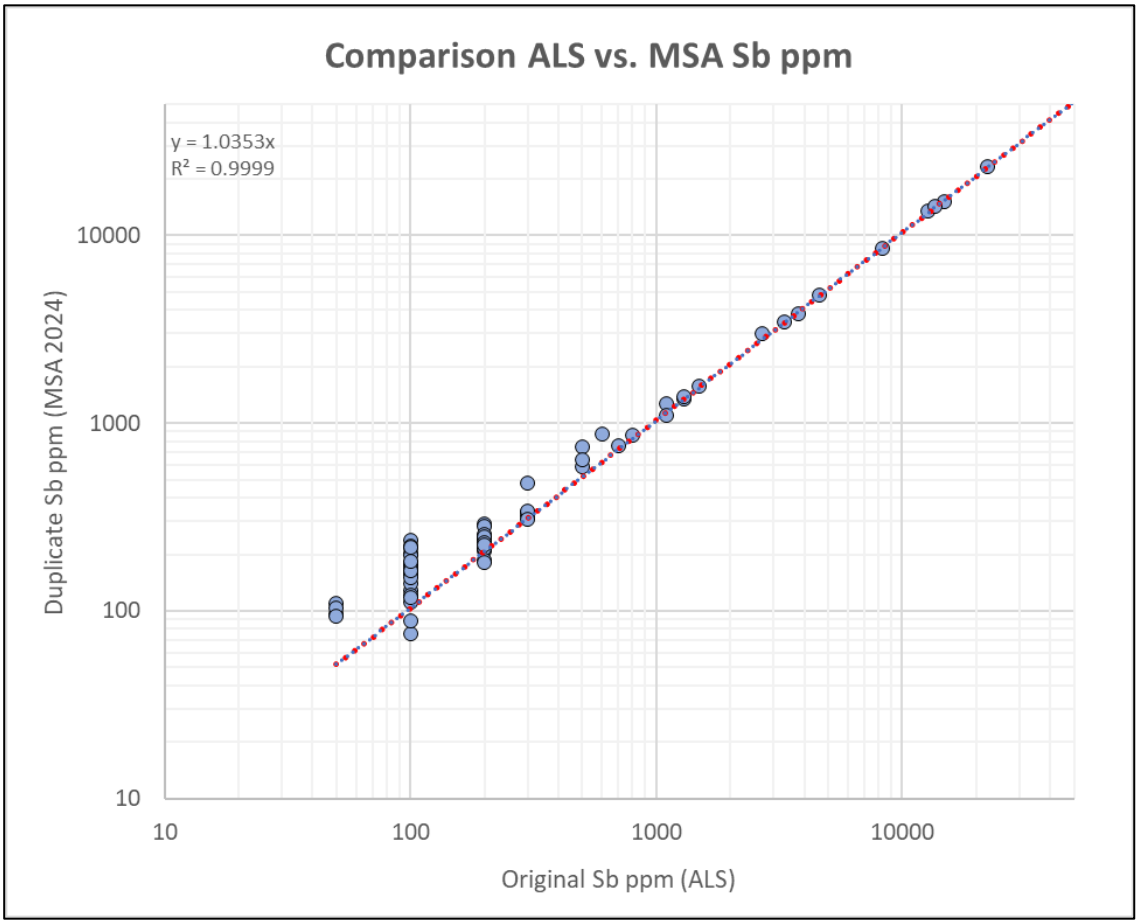
11.2.5 New Antimony Analysis of 2004–2006 Pulps

In late 2024, Canagold submitted 406 historical drill core sample pulps (from 2004 to 2006) to MSA Labs in Langley, BC, for re-assay. Sixty-nine (69) pulps were analyzed for antimony, and 337 pulps were selected to fill data gaps in high-grade gold intervals. MSA was asked to analyze the sample pulps using a standard 4-acid digestion on 0.25 g of material with an ICP-AES finish, to report 34 elements, including Sb and As, under method code ICP-230. Results exceeding 10,000 ppm Sb triggered an ore-grade analysis on a new 0.2 g aliquot of the sample material with the same 4-acid digestion method, higher dilution, and the same ICP-AES finish (method ICF-6Sb). Thirteen (13) samples exceeding 5% Sb were further analyzed using method STI-8Sb, which employs a titration finish.

Figure 11-6 compares the historical Sb results reported by ALS at the time against the recent Sb results from MSA. The correlation between the two datasets is very good ($R^2 \approx 1.0$), particularly for grades >0.1% Sb. The highest-grade sample reported approximately 8% Sb. Lower-grade samples (<800 ppm) are slightly biased towards the new 2024 MSA results, probably negatively influenced by the lower resolution of the original ALS.

MMTS considers the Sb data suitable for use in resource estimation.

Figure 11-6: Historical Pulp Re-analysis – Sb (MSA, 2024)



Source: MMTS, 2025.

11.3 QA/QC 2021–2022

The 2021-2022 QA/QC program followed a similar approach to the previous programs. Samples were collected on site by Canagold personnel and shipped to ALS Chemex laboratories in Whitehorse, YT (2021), and Yellowknife, NT (2022). For quality control and quality assurance purposes, blanks, duplicates, and blind standards were inserted into the core sample stream at a rate of approximately one in ten, in a rotating sequence, resulting in insertion rates of approximately 3.5% each.

Table 11-4 summarizes the QA/QC insertion counts. For both years, the total insertion rate was an industry-standard 10%, consisting of frequent blanks and standards, supplemented by coarse crush duplicates (LABDUP in the Canagold database). MMTS is not aware of any check-assay data.

Table 11-4: 2021–2022 QA/QC Summary

Year	Lab	Core Samples	Blanks	CRMs	Field Dup	Coarse Dup*	Pulp Dup	QA/QC Total	% QA/QC	Check	Comment
2021	ALS	2,368	96	88	0	86	0	270	10.2%	0	*LABDUP in DB
2022	ALS	1,561	63	57	0	58	0	178	10.2%	0	*LABDUP in DB
Total		**3,929**	**159**	**145**	**0**	**144**	**0**	**448**	**10.2%**	**0**	

11.3.1 Blanks – 2021–2022

A total of 159 blanks were inserted into the stream at regular intervals for an insertion rate of approximately 4%. For both drilling campaigns, Canagold used blank material CDN-BL-10, which consists of an unmineralized granite ground to <200 mesh in grain size, and certified to contain <0.01 g/t Au. As a fully pulverized material, this blank will not be able to capture cross-sample contamination that may occur during the crushing or pulverizing stages at the ALS preparation labs.

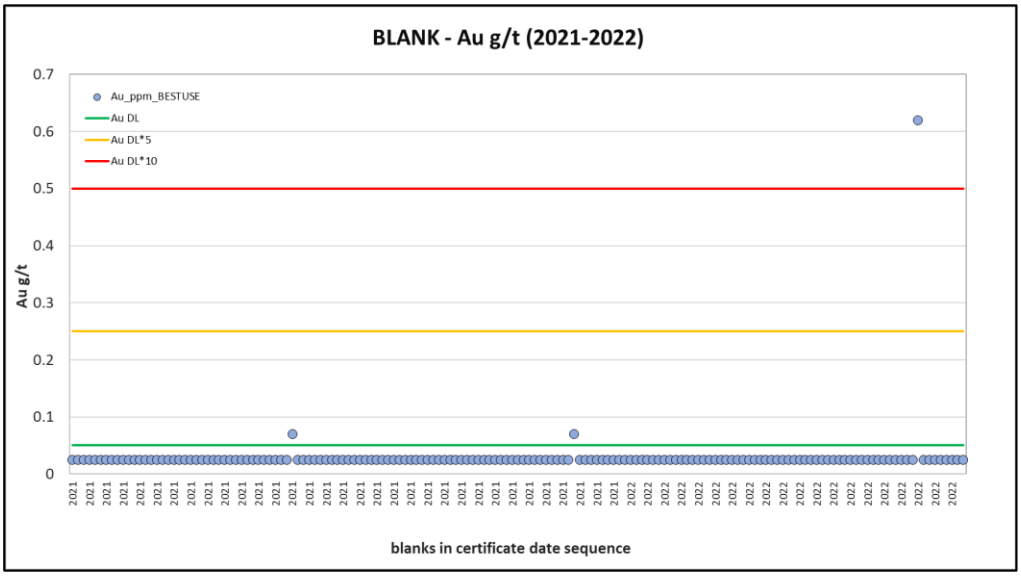
One failure was recorded for blank D937480 (0.62 g/t Au), which follows a moderately mineralized interval in drill hole P22Y25 (approximately 10 g/t Au at 324–326 m). However, the core sample immediately preceding the blank returned a lower grade of 0.4 g/t. MMTS is unclear how to interpret this potential contamination.

The blank statistics for gold are shown in Table 11-5, with the data illustrated in Figure 11-7.

Table 11-5: 2021–2022 Blanks Performance Summary

Year	Lab	Core Samples	BLK Count Au	Fails Au	% Fails Au
2021	ALS	2,368	96	0	0.0%
2022	ALS	1,561	63	1	1.6%
Total		**3,929**	**159**	**1**	**0.6%**

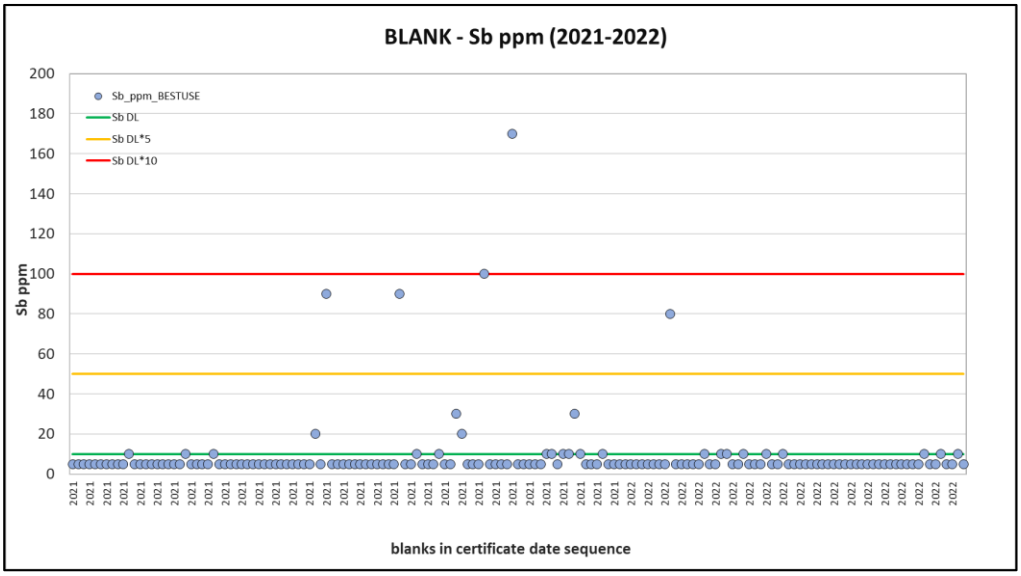
Figure 11-7: Blank Performance 2021–2022 - Au



Source: MMTS, 2025.

One theoretical failure is noted in the blank data for Sb (170 ppm, see Figure 11-8), following a very high-grade Au-Sb interval of approximately 4% Sb in drill hole 21-1800E3. Gold in the blank sample is not elevated, and MMTS does not view this as a concern.

Figure 11-8: Blank Performance 2021–2022 - Sb



Source: MMTS, 2025.

11.3.2 Duplicates – 2021-2022

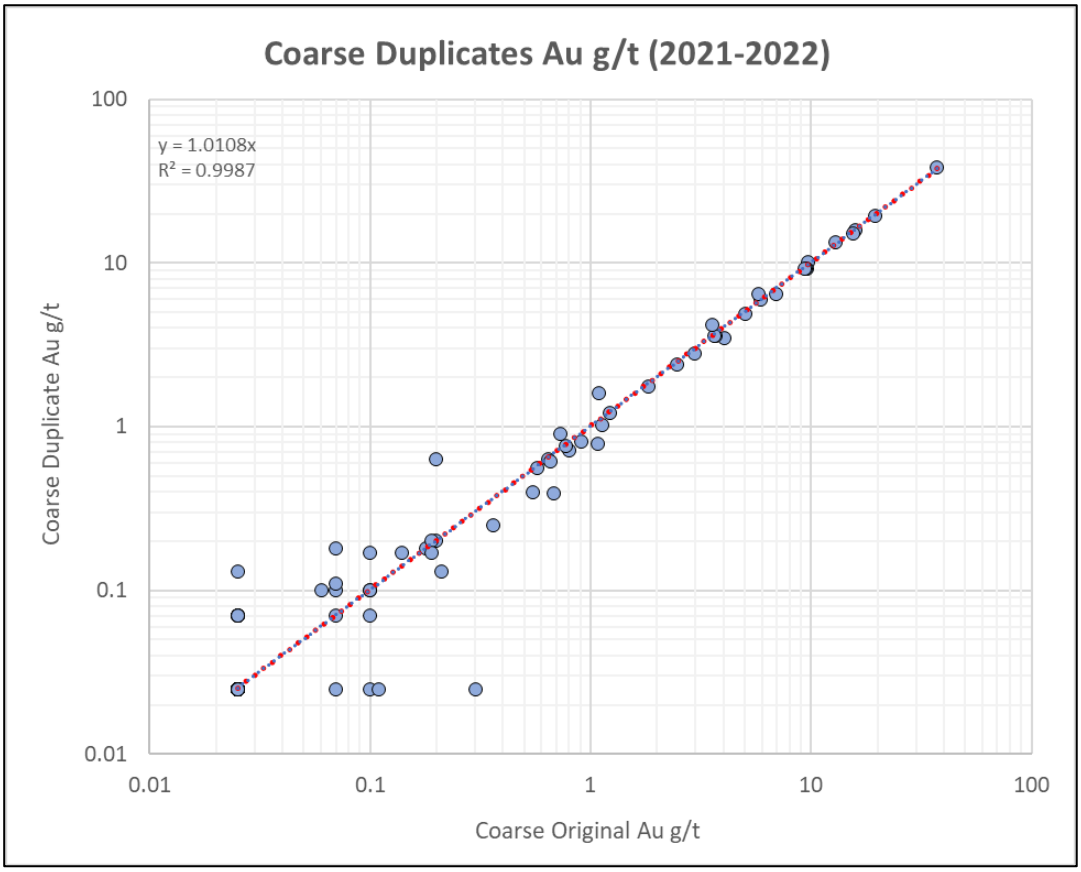
During the 2021-2022 drilling campaigns, Canagold requested ALS to generate what MMTS understands to be coarse duplicates at predetermined sample intervals, resulting in an insertion rate of 3.7% (144 duplicates out of 3,929 core samples). This is in addition to the lab's internal coarse duplicate protocols, the results of which are not reviewed for this report.

MMTS is not aware of any field duplicate datasets for these two years.

The original-duplicate pairs for Au and Sb are presented in simple scatter plots in Figure 11-9 and Figure 11-10, respectively.

The correlation between original and duplicate samples, and thus the reproducibility for gold, is very good ($R^2 \approx 1.0$). Grades of the selected intervals range from a detection limit of 0.05 g/t to nearly 40 g/t Au, which is representative of the New Polaris system, along with very good data point distribution along the grade range.

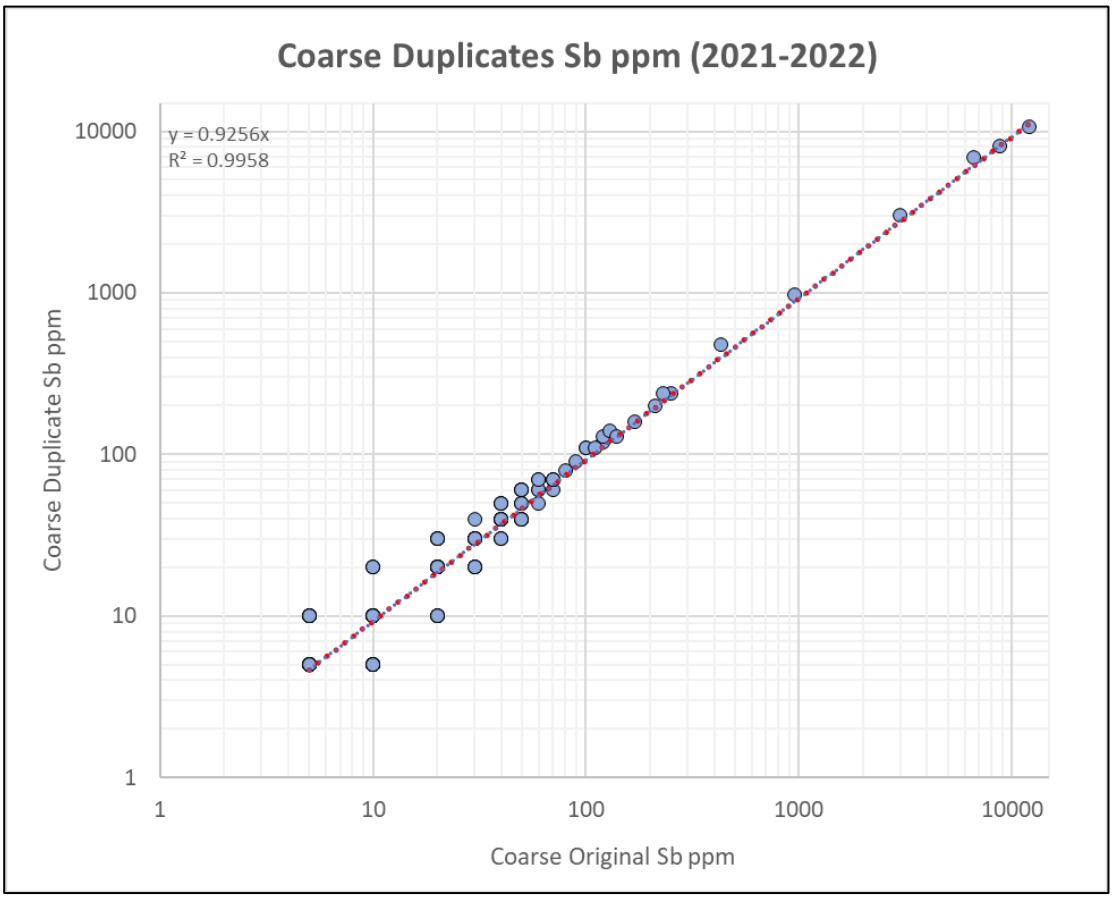
Figure 11-9: Coarse Duplicate Performance 2021–2022 – Au



Source: MMTS, 2025.

Similarly, the antimony assay results demonstrate near perfect correlation with very low variability between original and duplicate at the coarse crush stage of preparation at ALS. A very weak duplicate-positive bias is noted in Figure 11-10 for data between 100 ppm and 3,000 ppm; however, since multiple higher-grade pairs around 1% Sb show no bias, MMTS finds the results acceptable.

Figure 11-10: Coarse Duplicate Performance 2021–2022 - Sb



Source: MMTS, 2025.

11.3.3 Standards 2021–2022

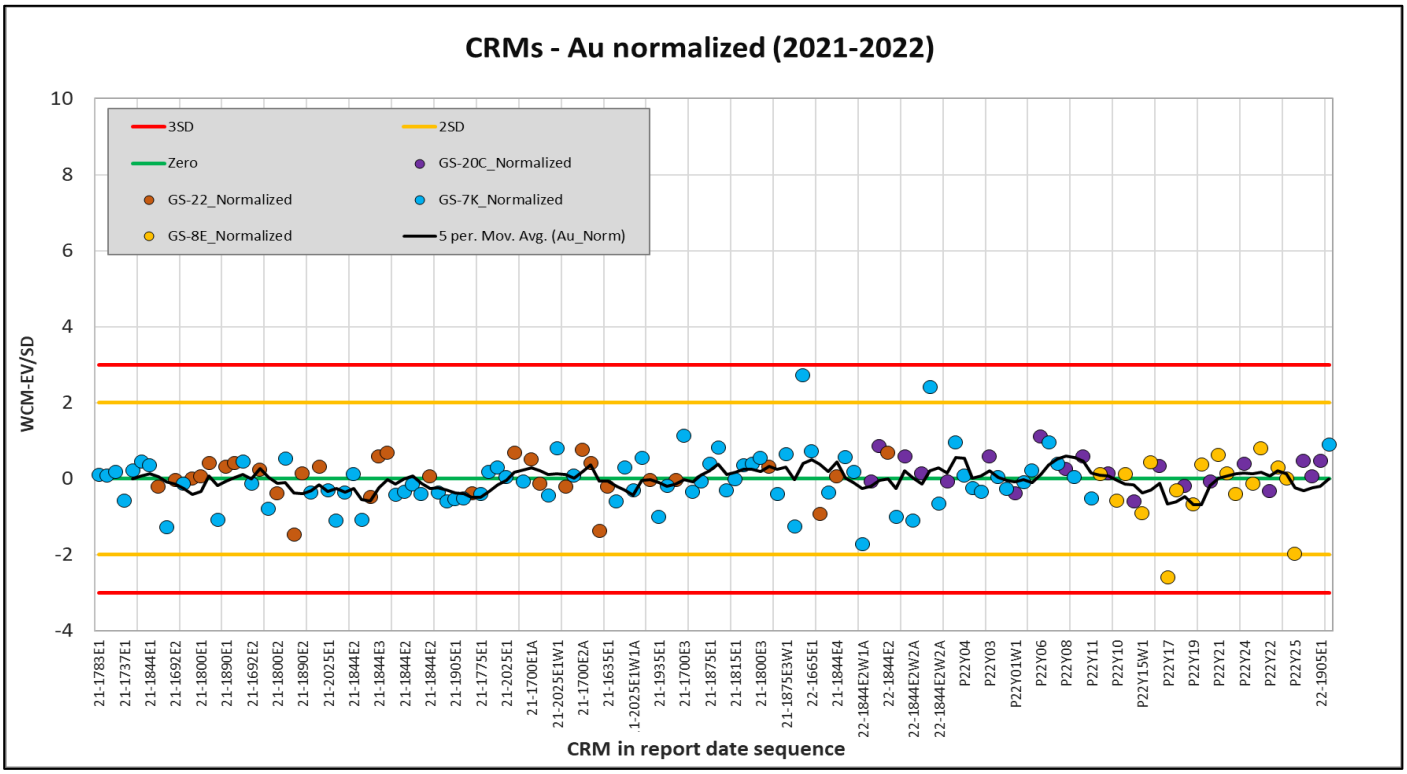
Certified Reference Materials (CRM) for the 2021-2022 campaigns were purchased from CDN Resource Labs in Langley, BC. The four CRMs used are certified for gold concentrations ranging from 7.12 g/t to 22.94 g/t, as listed in Table 11-5. These concentration values are appropriate for monitoring accuracy in the high-grade gold mineralization at New Polaris; however, MMTS recommends inserting a standard to cover the lower Au grades <5 g/t as well. The insertion rate during 2021-2022 was approximately 3.6%.

No failures are recorded for the 145 total standard analyses. Figure 11-11 shows the normalized Au performance of all standards and the distribution of the medium CRM (CDN-GS-7K and CDN-GS-8E) versus the high-grade CRMs (CDN-GS-20C and CDN-GS-22). The five-sample moving average underlines the acceptably accurate results.

Table 11-6: 2021–2022 Standards Performance Summary

Standard	Year	Count Au	Au g/t Mean	Au g/t EV	Au g/t SD	Au % Error	Au Fails High	Au Fails Low	Au Fails %
CDN-GS-20C	2022	20	19.81	19.65	0.76	0.8%	0	0	0.0
CDN-GS-22	2021-2022	30	22.97	22.94	1.12	0.1%	0	0	0.0
CDN-GS-7K	2021-2022	78	7.10	7.12	0.40	-0.2%	0	0	0.0
CDN-GS-8E	2022	17	8.52	8.62	0.35	-1.1%	0	0	0.0
Total		**145**					**0**	**0**	**0.0**

Figure 11-11: PCC Standards Performance 2021–2022 - Au



Source: MMTS, 2025.

11.4 Conclusions and Recommendations

With regards to contamination and accuracy, MMTS finds the available data sufficiently controlled from 2005 onwards and has no concerns about any systematic errors or bias for gold. Independent reproducibility QA/QC in the form of sets of duplicates, however, is only partial, and a review of the lab-internal coarse and pulp duplicates is recommended. For future drilling campaigns, MMTS also recommends adding a low-grade gold standard in the 1 g/t-Au-range to the medium and high-grade standards previously inserted to represent the expected gold concentrations more comprehensively.

Antimony was added to the resource estimation retroactively, which is why none of the inserted standards are certified for antimony. At New Polaris, antimony is seen as a by-product only, but MMTS recommends replacing at least one current standard with a standard that is certified for both gold and antimony to be able to independently control for accurate results.

Furthermore, the consistent insertion of a coarse blank is advised. MMTS understands that in the 2024 drilling campaign some blind blank insertions were of unmineralized rock in addition to CDN-BL-10.

Between 5 and 10% of samples should be check-assayed by a secondary laboratory, requesting comparable analytical methods, especially if a lab change occurred.

MMTS views the data as acceptable for resource estimation.

12 DATA VERIFICATION

12.1 Database Verification

The QP, Sue Bird visited the property on August 25, 2022. At the time of the site visit, core logging procedures from the 2022 program were examined, a general overview of the property was completed, including selected drill sites, historic core, an underground tour, and the condition of existing project infrastructure.

While on the property, the QP, along with Chief Geologist, Troy Gill, P.Geo., examined underground workings to confirm the nature of mineralization, as well as the dimensions and extent of the vein system. The QP also viewed a selection of core from key holes drilled from the early 1990's to the end of 2006 and compared observations with those documented in the drill logs. In both the case of the underground workings and the core, the author found that observations confirmed what was recorded in logs and sections. The QP also confirmed that core had been properly cut and stored.

The core logging facility was clean and orderly. The system of check assaying is adequate. The only issue that the QP has with the system is the use of quartered core for the duplicate samples. The sample size difference between the quartered and half core may account in part for the high relative difference between the original sample and the duplicate. In future, resubmission of pulps on a blind basis should be carried out to help separate variances caused by analysis from that due to sample size or bias cutting of the core.

In addition to the site visit, a detailed review of the database was completed. Forty-one drillholes were selected from the C vein area, and the drill logs and assay sheets were compared with the database. Only minor differences were observed between the hard copy material and the database. Additionally, the input of the database into the geology and resource modeling software was also checked.

The quality control systems in place prior to the 2003 program are poorly documented but seems to follow the norms of that period. Of concern is the manner in which the collar locations of drillholes were determined. Most of the holes were located using Brunton compass and chaining. Also, the down hole surveying was not consistently done. As a result, the exact location of the vein intersections is not as certain as those from the drilling after 2003. Some re-drilling of older holes is recommended, especially where there are discrepancies with respect to the vein location between the recently drilled holes and those drilled in the 1990's.

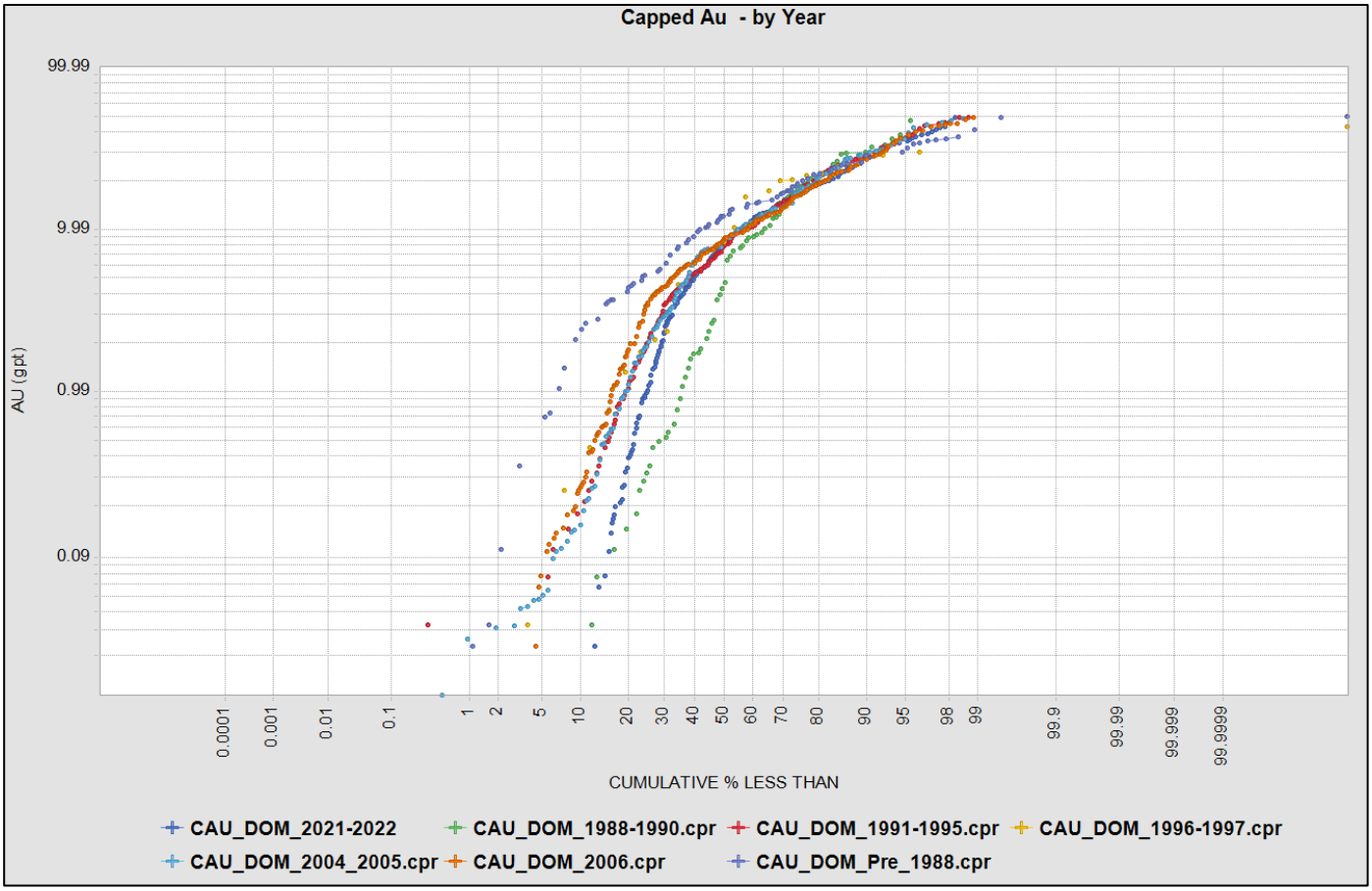
To further validate the drillhole database, comparisons have been made of the results by year.

Figure 12-1, the capped Au grades do not indicate any bias based on year drilled except possibly the pre-1988 holes. However, these holes are all within the high-grade central portion of the deposit and drilled from underground so are expected to be higher-grade. Therefore, a point validation has been completed to compare drillhole assays only within 20m of the 1988 data.

Figure 12-2 shows that the 1988 grade distribution is below the grade distribution of the surrounding data.

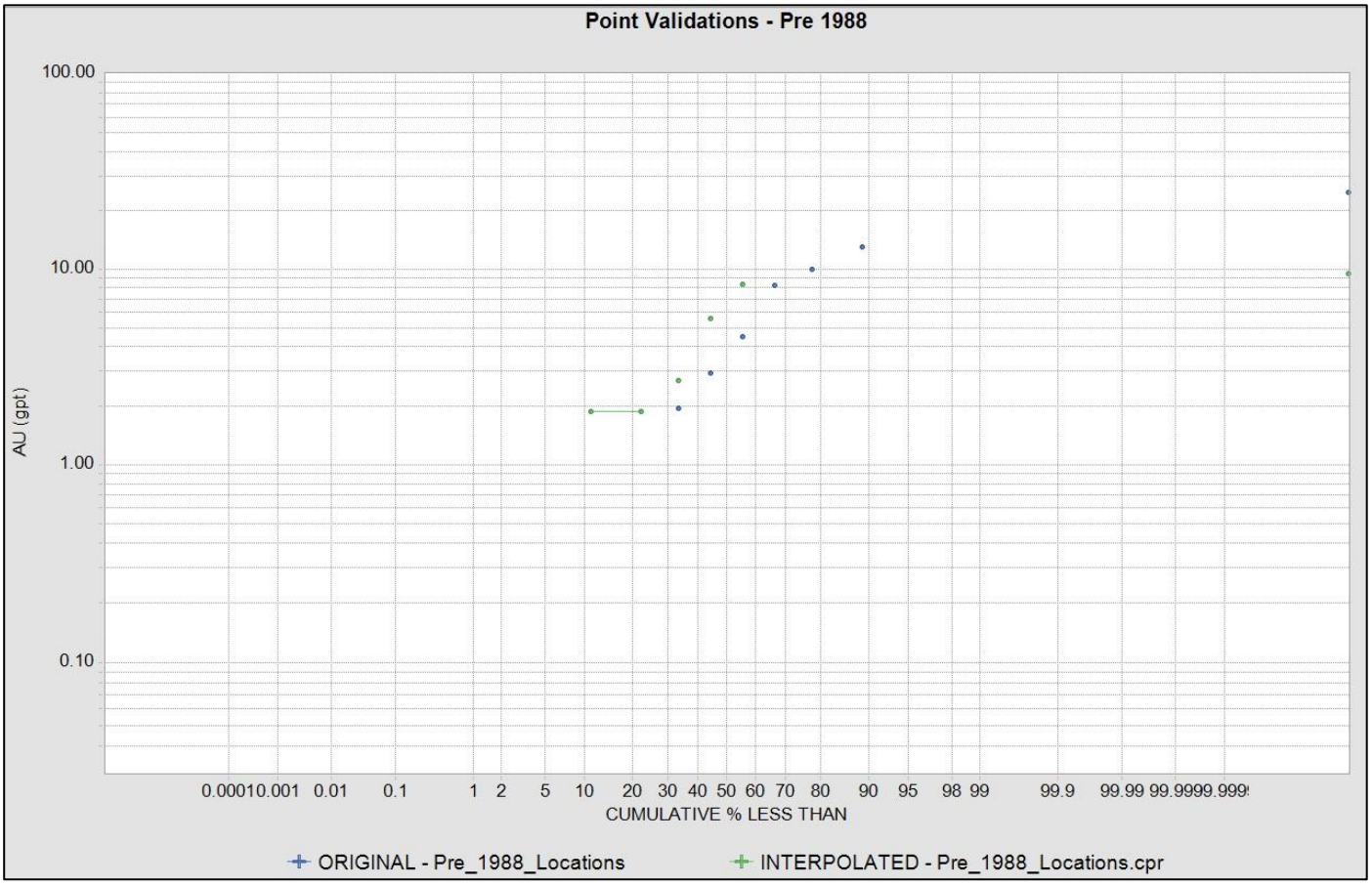
Figure 12-1 also shows that the high Au grades (>30 g/t) are lower for the pre-1988 than the other years. Both plots show that the pre-1988 data is somewhat conservative.

Figure 12-1: Comparison of Capped Au Grades by Year



Source: MMTS, 2023.

Figure 12-2: Point Validation of Pre-1988 Data



Source: MMTS, 2023.

The procedures used in the development of the database are considered by the QP to follow accepted industry standards. There is confidence that selected drillholes for the Mineral Resource Estimate are error-free data and are suitable to support Mineral Resource estimation. The data from those drillholes that do not have this confidence have not been used (prior to 1998).

12.2 Mineral Processing and Metallurgical Data

BIOX™ testing was completed by SGS South Africa Pty. Data from this program was not used for design, but this laboratory is accredited for the assaying and testing methods employed. Testing programs completed for this report were completed under the supervision of the QP. Base Metallurgical Laboratories Ltd. utilizes appropriate assaying and testing methods required for the testing completed in their scope of work. All metallurgical data was verified and is adequate for this technical report.

13 MINERAL PROCESSING AND METALLURGICAL TESTING

13.1 Introduction

The New Polaris deposit is primarily made up of finely disseminated arsenopyrite and pyrite grains containing highly refractory gold. Between 1937 to 1942 and 1946 to 1951, 231,604 oz of gold was produced in flotation concentrates from a head grade of approximately 10 g/t Au.

Preliminary batch testing in 2019 showed that Metso's BIOX™ process could oxidize flotation concentrate and liberate gold for recovery through cyanidation with high leach extractions of 96%. A flotation pilot plant was completed in 2022 to generate concentrate for pilot scale BIOX™ testing by Metso. As the pilot plant progressed, higher concentrations of antimony (Sb) caused issues with biological activity that required biological oxidation to be conducted at very dilute slurry density and very long retention time. Additionally, the high arsenopyrite concentrate requires external acid addition to maintain the required pH in oxidation. Failure to remove antimony prior to gold recovery would lead to a gold concentrate with elevated impurities. The increased capital and operating costs from these changes rendered this process route uneconomic.

Results from variability flotation testing completed at Base Metallurgical Laboratories Ltd (BaseMet) in 2023 to provide concentrate for BIOX™ testing by Metso were instead used for recovery modeling to support the sale of concentrates for third-party processing. Variability BIOX™ testing was not completed. The BaseMet program also included comminution testing.

A 2023 program conducted at Forte Analytical LLC (Forte) generated concentrate and tailings samples for solids liquids separation testing at FLSmidth Inc – Salt Lake City Operations (FLS).

Testing, both historical and recent testing, has focused on maximizing gold recoveries. No attempt has been made to optimize or improve antimony metallurgy. Further investigations are required to determine if the antimony content can be economically separated from the bulk gold concentrates. Antimony is typically a deleterious element for off-takers.

While the complex concentrates produced from flotation have very high gold grades, they also contain high arsenic (As) grades which is typically deleterious element for off-takers.

13.2 Metallurgical Testwork

The New Polaris project is a past- producing mine with historical testwork.

Table 13-1 summarizes the relevant testwork undertaken to develop this Feasibility Study.

Table 13-1: New Polaris Metallurgical Testwork Summary

Year	Reference: No.	Laboratory/Location	Testwork Performed
2022	KM6844	ALS Metallurgy, Kamloops, BC	Flotation pilot plant
2023	BL1189	Base Metallurgical Laboratories Ltd., Kamloops, BC	Comminution, variability flotation, and chemical analysis
2023	23058	Forte Analytical, LLC, Fort Collins, CO	Gravity concentration and flotation testing, bulk flotation to generate samples thickening, and filtration of concentrate and tailings
2023	-	Metso, SGS Johannesberg, SA	Bio-oxidation pilot plant trials
2024	Oracle Project Reference: 9232518616	FLSmidth Inc., Salt Lake City, UT	Concentrate and tailings thickening, and pressure filtration

13.2.1 ALS Metallurgy (2022) Pilot Plant and Metso BIOX™ Plant (2023)

Testing completed in 2019 showed that the Metso BIOX™ process could treat refractory New Polaris concentrates and liberate contained gold for downstream leaching. To demonstrate the viability of the process and generate process design information, a BIOX™ pilot plant would need to be operated. Operating a flotation pilot plant to generate sufficient concentrate for this purpose was the most effective approach.

13.2.1.1 ALS Metallurgy (2022) Pilot Plant

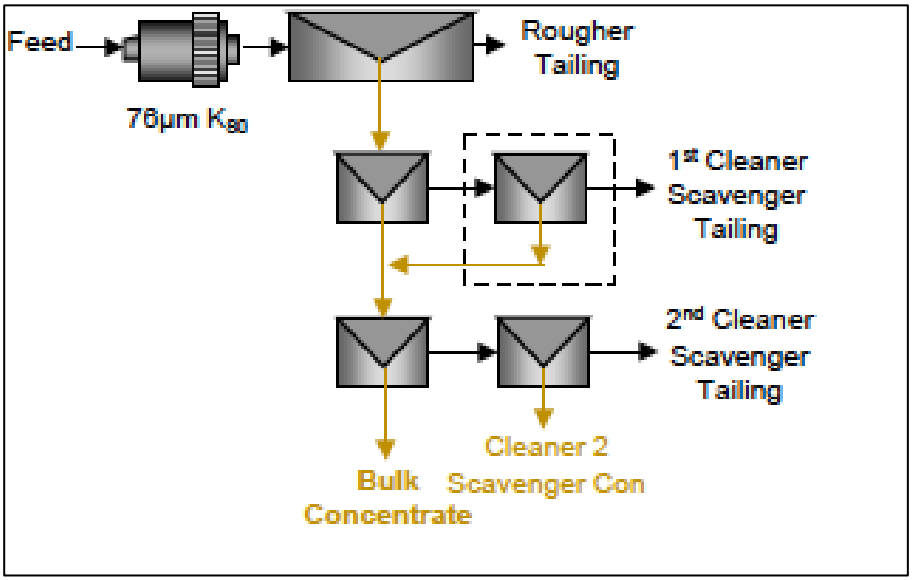
The primary objective of this program was to operate a flotation pilot plant to generate flotation concentrate for pilot scale BIOX™ testing. The program also validated the historical flotation flowsheet shown in Figure 13-1. The program used a large composite sample, 743 kg, identified as Master Composite 2022. The sample includes a number of diamond drill holes that provide broad spatial representation of the deposit. The composite sample head assays are shown in Table 13-2.

Table 13-2: Master Composite 2022 Sample Head Assays

Au (g/t)	Fe (%)	S (%)	As (%)	Sb (%)	C (%)	TOC* (%)	Hg (g/t)
11.3	4.16	2.19	1.34	0.44	3.87	0.06	<1

*TOC is total organic carbon.

Figure 13-1: 2022 ALS Metallurgy Flotation Pilot Plant Flowsheet



Source: ALS Metallurgy, 2022.

The first cleaner scavenger tailings are recycled to the rougher flotation while the second cleaner scavenger concentrate is recycled to the first cleaner flotation feed. The second cleaner scavenger tailings report to final the tailings along with the rougher tailings.

The pilot plant was run semi-continuously over six days at a feed rate of 40 kg/h. Flotation cells used were 15 L capacity and provided excess retention time in the cleaner stages. Samples were ground in 25 kg batches to 80% passing (P_{80}) of 70 to 75 µm and stored as a slurry for processing the following day. Chemical conditions and reagent addition rates for the pilot plant are shown in Table 13-3.

Table 13-3: 2022 ALS Pilot Chemical Conditions and Reagent Addition Rates

Stage	pH	Reagent Addition (g/t)				
		Na_2S	$CuSO_4$	PAX	MIBC	Na_2SiO_3
Grinding	8.5 – 9.0	-	-	-	-	-
Rougher Flotation	8.5 – 9.0	250	248 - 270	165 - 190	44 - 79	-
Cleaner Flotation	8.5 – 9.0	-	50 - 55	20 - 28	75 - 136	50 - 53

Flotation was performed at natural pH Sodium sulphide (Na_2S) is used as a sulphide mineral activator, typically when mineral surfaces are tarnished. Copper sulphate ($CuSO_4$) is used as an activator for arsenopyrite and pyrite to make them more amenable to flotation. Potassium amyl xanthate (PAX) is the flotation collector used. Sodium silicate (Na_2SiO_3) is used as a slimes dispersant in cleaning, MIBC (methyl isobutyl carbinol) is the frother used.

The pilot plant demonstrated high gold recoveries (average 95%) to concentrate could be achieved and also confirming that the historical flowsheet was suitable for processing current samples. The pilot plant was operated to achieve maximum gold recoveries to minimize downstream losses through BIOX™ processing. A summary of the final concentrates produced is shown in Table 13-4. On average the pilot plant recovered 13.8% of the feed mass to a concentrate with an average Au grade of 80.1 g/t and an average Au recovery of 95.4%. The average concentrate arsenic (As) and antimony grades were 9.29% and 3.07% respectively.

A total of 83 kg of concentrate was produced and sent to Metso in Johannesburg, SA for BIOX™ testing.

Table 13-4: 2022 ALS Metallurgy Pilot Plant Concentrate Results

PP	Mass	Balanced Concentrate Assays					Concentrate Distribution (%)				
Run	(%)	Fe (%)	S (%)	Au (g/t)	As (%)	Sb (%)	Fe	S	Au	As	Sb
P2	13.6	18.2	16.0	88.8	9.81	3.01	47.0	96.6	94.5	94.0	92.3
P3	14.6	16.1	15.1	74.1	8.64	2.95	47.2	97.1	94.7	94.4	93.5
P4	12.1	20.4	18.3	83.7	10.50	3.63	47.9	97.1	94.3	94.3	92.9
P5	14.4	18.0	15.2	78.1	9.16	2.89	48.7	97.1	96.0	94.8	90.6
P6	14.5	16.7	14.0	75.7	8.30	2.87	46.7	97.0	95.4	95.1	90.6
Averages	13.8	17.9	15.7	80.1	9.29	3.07	47.5	97.0	95.0	94.5	92.0

13.2.1.2 Metso BIOX™ Pilot Plant (2023)

Earlier testing showed the BIOX™ process could oxidize New Polaris concentrates to liberate gold finely disseminated in sulphide minerals and make it amenable to direct cyanidation. Pilot plant testing is required to demonstrate its viability on a larger scale and to provide process design data.

13.2.1.2.1 Sample Preparation

Samples were received at SGS South Africa Johannesburg, SA on December 19, 2022, and were dried, combined, and deagglomerated by passing through an 850 µm screen. In total, a dry mass of 82.5 kg concentrate was received and prepared.

Analyses by pycnometer determined the average specific gravity (SG) of the concentrate to be 3.59.

13.2.1.2.2 Mineral Composition

The concentrate received from Canagold was submitted for AutoSEM and X-ray Diffraction (XRD) analyses. The concentrate was determined to be primarily comprised of pyrite (24.1%), arsenopyrite (16.6%) and stibnite (2.4%) as the dominant sulphide-bearing minerals. The arsenic content was determined by chemical assays (9.1%) and directly correlates to the amount of arsenopyrite detected and verified via AutoSEM. Trace amounts of arsenian pyrite were detected.

The non-sulphide gangue consists of quartz, muscovite, and clinochlore. Dolomite content was determined to be around 8.6% which stoichiometrically translates to a carbonate content of around 5.6%.

13.2.1.2.3 Carbon-in-Leach (CIL)

The concentrate was submitted for a baseline standard carbon-in-leach (CIL) test under condition noted as follows:

- Solid concentration - 20%
- Carbon addition – 20 g/L slurry
- NaCN addition - 20 kg/t
- CaO addition - 1.45 kg/t
- NaCN consumption – 13.1 kg/t
- Mass Tie – 1.0

This CIL test verified the expectation that the New Polaris concentrate would have poor gold dissolution when subjected to direct cyanidation without any pre-treatment. Gold dissolution at a 20 kg/t sodium cyanide (NaCN) concentrate addition was recorded at 13.8% which confirmed the refractory nature of the concentrate.

13.2.1.2.4 BIOX™ Pilot Plant

The BIOX™ pilot plant trials started in December of 2022 and terminated in the first half of 2023 when BIOX™ results showed that significantly longer retention time at more dilute conditions plus the requirement to add sulphuric acid to the process would detrimentally impact project economics.

The BIOX™ process is a mixed culture of naturally occurring microbes which, under controlled conditions, can oxidize gold-bearing sulphide ores or concentrates due to a chemolithotrophic mode of metabolism. This means that they require inorganic compounds for the acquisition of both energy and carbon. The carbon requirements of the microbes for biosynthesis of cellular biomass are met by CO_2 in the atmosphere or from dissolution of carbonate in the ore.

The microbial culture in the BIOX™ reactors is not controlled but rather allowed to adapt to the concentrate and operating conditions. It is, therefore, important to control the pH, temperature, and other operating parameters to within certain prescribed ranges to maintain the right balance of microbial species to optimize the rate of oxidation.

Metso produced monthly progress reports from the BIOX™ pilot plant that captured the efforts made to optimize sulphide oxidation and downstream gold recovery.

13.2.1.2.4.1 Pilot Plant Facilities

The BIOX™ pilot plant consists of four continuously stirred stainless-steel reactors, configured as one primary reactor (Reactor P1) and three secondary reactors in series (Reactors S1, S2, and S3).

The average unaerated operating volumes of the reactors considered for the feed rate and retention time adjustments are shown in Table 13-5.

Table 13-5: Average Unaerated Operating Volumes

Reactor	Volume (L)
P1	20.3
S1	4.8
S2	4.9
S3	4.8
Total	**34.8**

The pilot plant includes:

- A single stock tank to store concentrate and adjust slurry density as required.
- An automated feed splitter unit controls the feed rates into the P1 primary reactor which enables feed slurry to be fed to the primary reactor at a predetermined flow rate and solids density.
- The P1 primary reactor product cascades into the first secondary (S1) reactor and then into S2, S3 and finally into the overflow bucket.
- The pilot plant is equipped with both heating and cooling circuits. The primary reactor and first secondary (S1) reactor are equipped with online temperature measurement and a temperature controller to maintain the temperature within the set point. This is achieved by circulating either hot water (42°C) through the stainless-steel heating coils or cold water through the stainless-steel cooling coils fitted in the primary and first secondary (S1) reactor tanks.
- The slurry is aerated by sparging compressed air through a single pipe sparger situated beneath the impeller in each vessel. A single rotameter is installed per reactor to control the supply of air to the reactors. The spargers are manufactured from 316 L stainless-steel tubing.

13.2.1.2.4.2 BIOX Pilot Plant Phases

The planned phases of the pilot plant program are described in Table 13-6. The plant started to receive feed in continuous mode from January 24, 2023.

Table 13-6: 2023 BIOX™ Pilot Plant Phases

Pilot Plant Phases	Plant Retention Time (Days)	Feed Slurry Solids Concentration (%)	Operational Days	Start Date	End Date
Inoculum Build-Up and Ramp-Up		20.0	~ 17	16-Jan-23	03-Feb-23
1 (2)	30	10.0	1	03-Feb-23	04-Feb-23
2 (2)	24	10.0	3	04-Feb-23	05-Feb-23
3 (2)	18	10.0	4	05-Feb-23	06-Feb-23
4 (2)	12	10.0, 12.5, 15.0, 17.5, 20.0	8	06-Feb-23	13-Feb-23
5 (3)	8	15.0	10	13-Feb-23	23-Feb-23
6 (Sampling Period 1)	7	15	25	23-Feb-23	19-Mar-23
7 (Sampling Period 2)	8	12.5	24	20-Mar-23	19-April-23
8(4) (Sampling Period 3)	9.3	12.5	14	20-April-23	2-May-23

Notes:
(1) n/a – not applicable.
(2) S1 serving as primary while P1 was reactivated.
(3) P1 re-instituted as primary reactor.
(4) Secondary reactor S4 added and FeSO4.7H2O supplementation.

13.2.1.2.5 BIOX™ Pilot Plant Results

The pilot plant testwork was extended beyond the originally planned time to address issues caused by elevated antimony level. Accumulated antimony solution levels during the pilot plant run impeded biological activity. As a work around, the slurry density in the BIOX trials were reduced to a maximum of 15% solids by weight to maintain the desired level of biological activity. An initial review of the pilot plant data indicated that instead of the typical 20% solids by weight with a residence time of 3-5 days, the high levels of antimony would limit the feed to 12% solids at a residence time of 9 days, effectively doubling the size of the BIOX™ reactors.

The low pilot plant feed pyrite content did not generate enough sulphuric acid during the BIOX™ process, to maintain the operating pH, requiring external sulphuric acid addition.

The BIOX™ pilot plant results indicated higher than expected capital and operating costs than had been indicated by prior small-scale batch testing and was therefore concluded to be a less attractive processing option for the New Polaris Project. The BIOX™ Process was therefore excluded from the design.

13.2.2 Base Metallurgical Laboratories Ltd. (2023)

BaseMet was selected to conduct a flotation variability testing program under project No. BL1189. The variability program was initially designed to provide concentrate for batch BIOX™ testing to generate recovery data. After the BIOX™ Processing option was dropped, variability testing was used to generate recovery data for the production of flotation concentrates for sale to third parties.

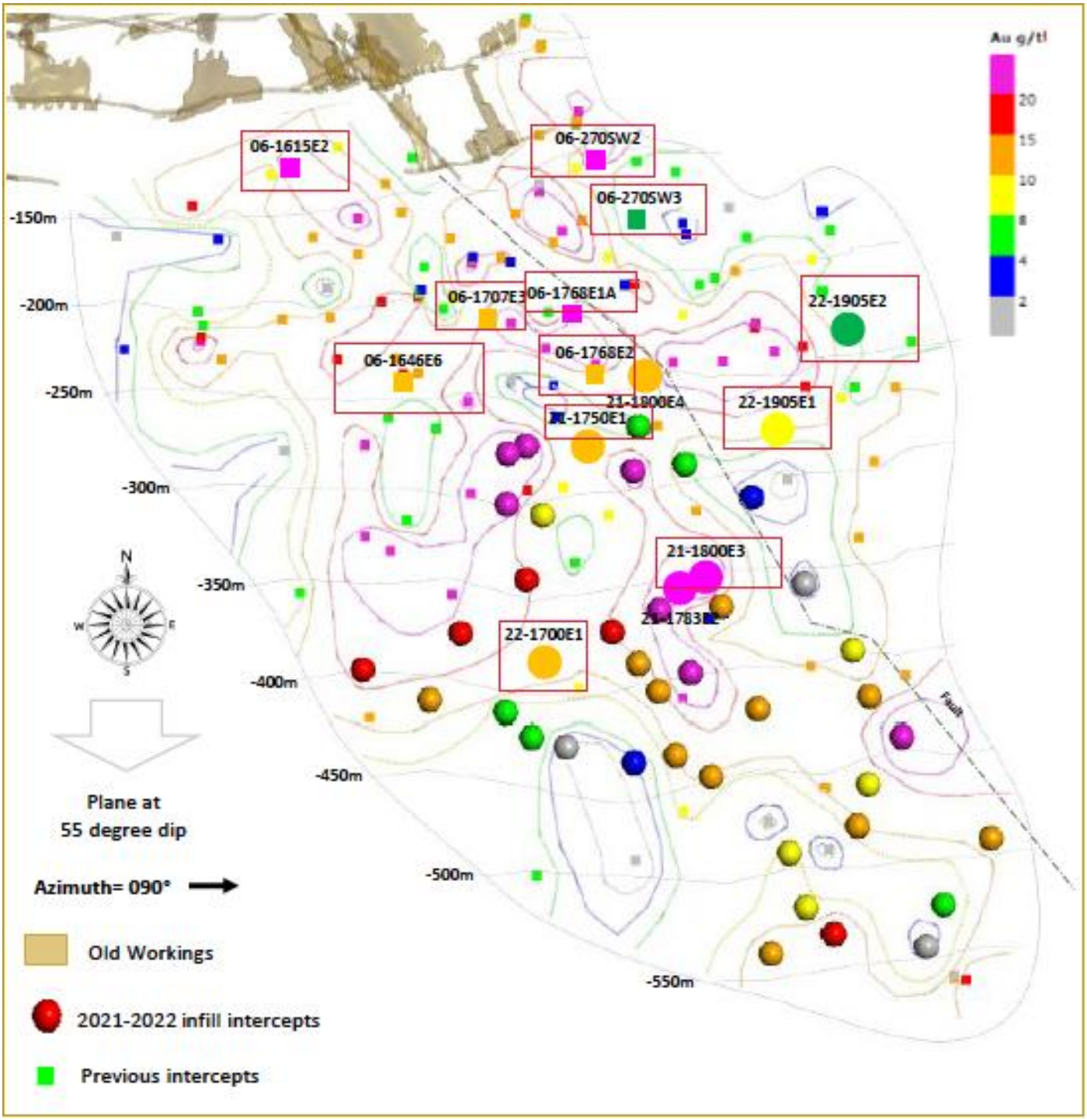
13.2.2.1 Sample Descriptions

A total of 22 comminution samples, and seven variability flotation composites were selected for testing. Several comminution samples (11) were also used for flotation testing. Samples were selected to provide spatial coverage of the deposit aligned with the major veins, C and Y. Samples were selected to cover grades from the cut-off grade of approximately 6 g/t Au up to a maximum of 25 g/t Au, with an overall average of approximately 10 g/t Au (resource average). A detailed mine plan was not available to guide grade selections. Intervals from each drill hole were minimum 6 m in length, approximating stope dimensions. The composites included:

- Twenty-two (22) comminution composites: COMM1 to COMM21 and ID9.
- Seven (7) Flotation composites: ID3, ID5, ID6, ID8, ID9, ID15, and ID16.

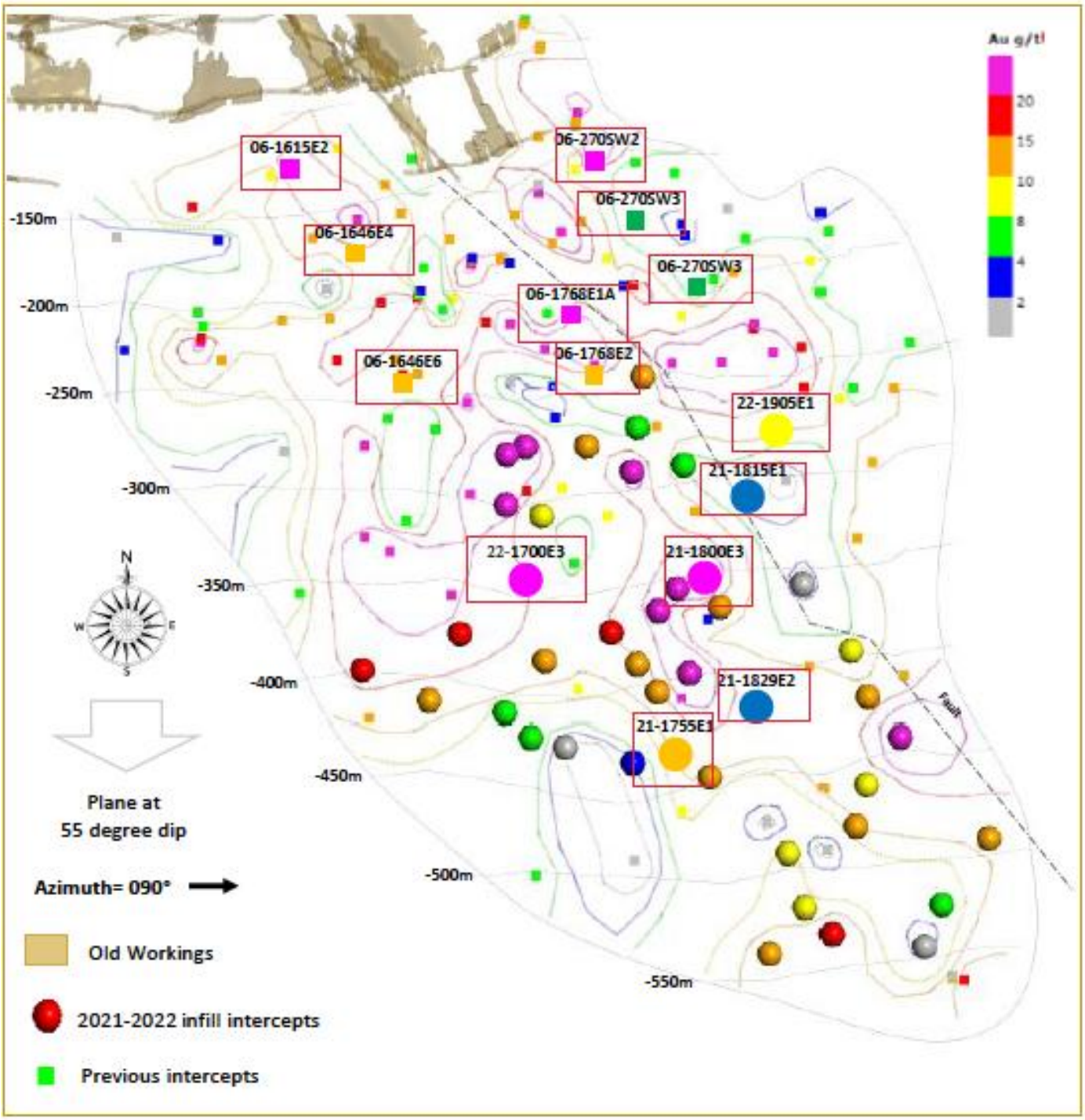
The location of the samples selected are shown in Figure 13-2 to Figure 13-6 inclusive.

Figure 13-2: New Polaris CWM and CHIE/CLOE Veins Flotation Samples



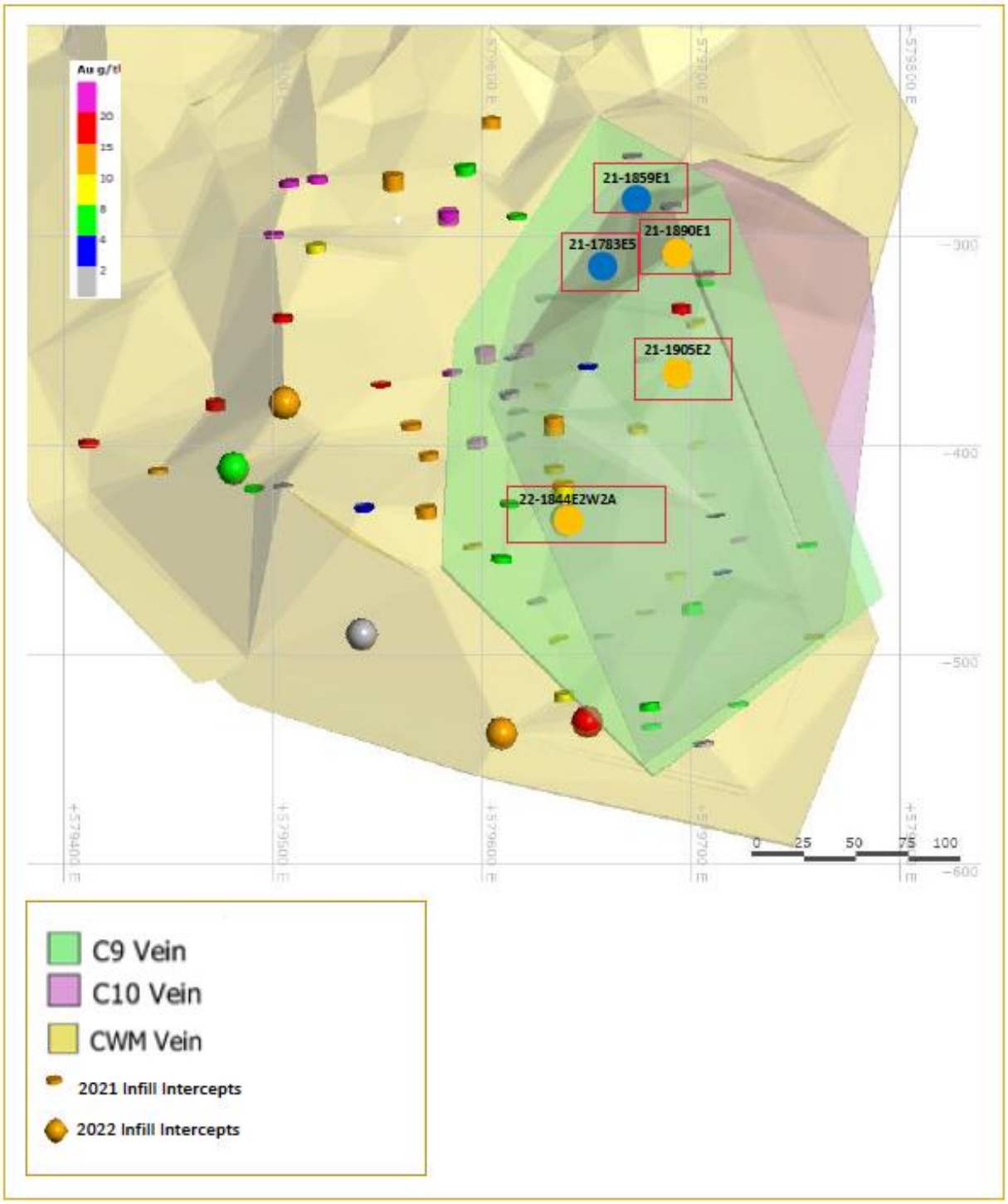
Source: Canagold, 2023.

Figure 13-3: New Polaris CWM and CHIE/CLOE Veins Comminution Samples



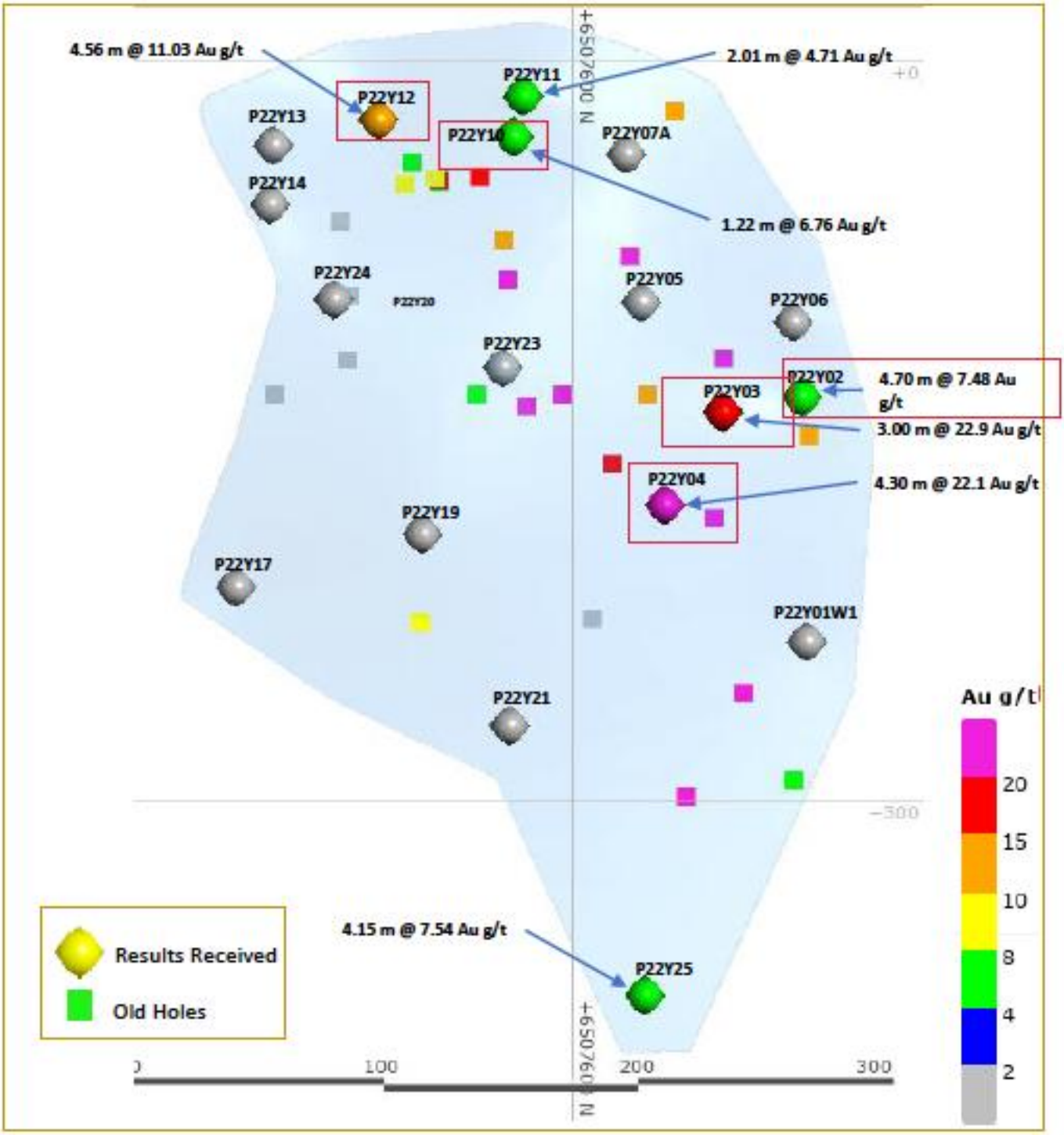
Source: Canagold, 2023.

Figure 13-4: New Polaris C9, C10 and CWM Veins Flotation and Comminution Samples



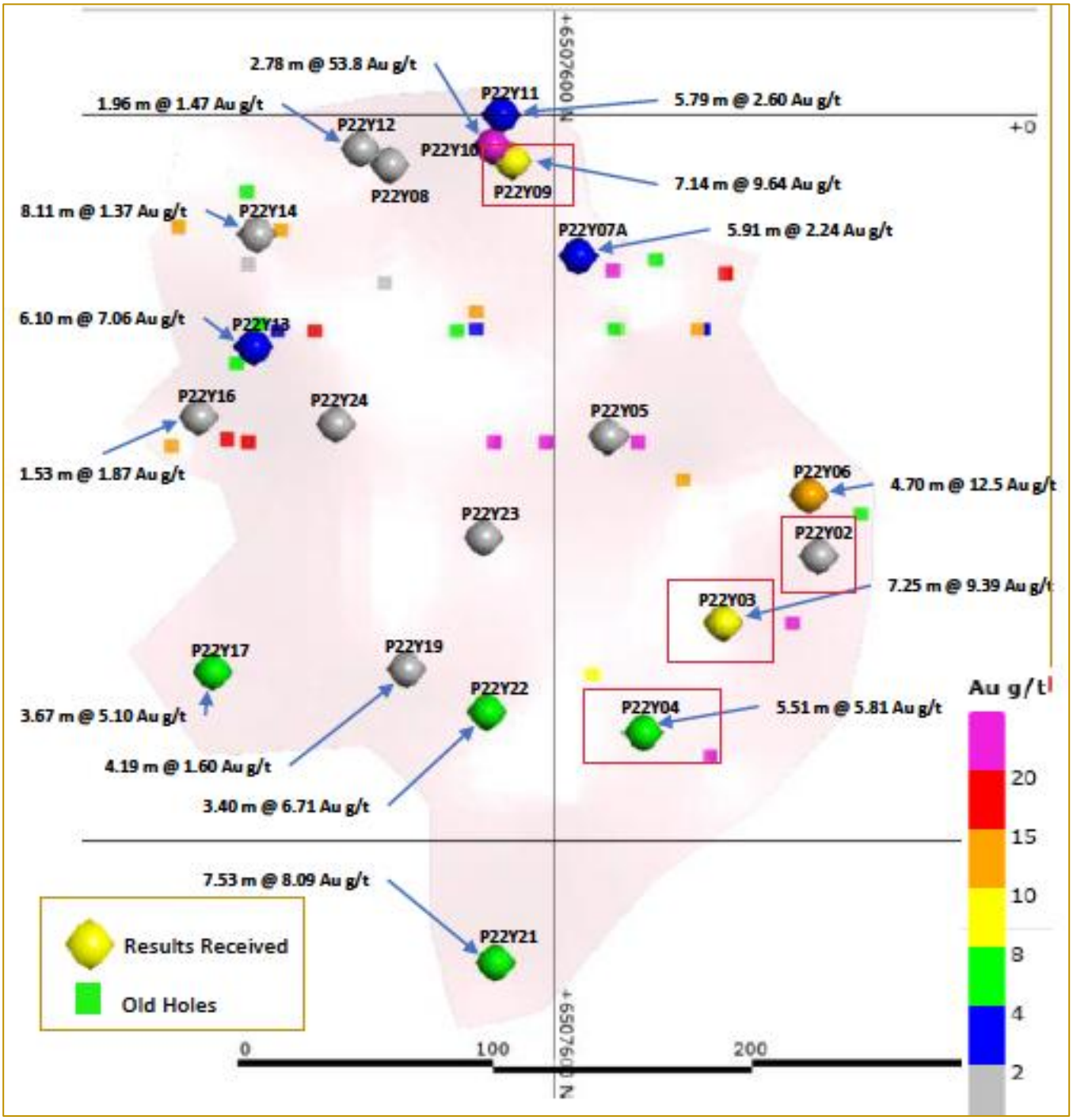
Source: Canagold, 2023.

Figure 13-5: New Polaris Y19 Flotation and Comminution Samples



Source: Canagold, 2023.

Figure 13-6: New Polaris Y20 Flotation and Comminution Samples



Source: Canagold, 2023. Intervals were selected to assemble grade composites low grade (LG), average grade (AG), and high grade (HG) for locked cycle flotation tests.

13.2.2.2 Sample Assays

All samples were crushed to -3.4 mm and prepared into charges for metallurgical testing. Head assay splits of 250 g were removed from a single test and pulverized. Assays for the samples are shown in Table 13-7.

Table 13-7: 2023 Variability Sample Assays

Sample	Vein	Au (g/t)	Ag (g/t)	S (%)	Fe (%)	C (%)	As (%)	Sb (%)
COMM1	CWM	4.97	1.40	1.52	5.40	4.04	0.99	0.01
COMM2	CWM	6.10	0.30	1.52	5.20	4.84	0.89	0.03
COMM3	CLOE	9.97	0.70	2.35	5.40	3.71	1.52	0.01
COMM4	C10	12.80	1.30	1.24	4.40	5.17	1.13	0.01
COMM5	Y20	7.90	0.30	1.69	5.57	3.45	1.36	0.01
COMM6	CWM	11.20	0.60	2.42	4.13	4.83	0.97	3.37
COMM7	CWM	14.90	1.80	1.25	5.60	3.99	0.56	0.85
COMM8	CWM	2.24	0.90	1.41	6.00	3.94	0.96	0.03
COMM9	CLOE	23.40	0.70	1.76	4.44	2.64	2.40	0.02
COMM10	C10	6.15	0.90	2.67	5.20	4.87	1.57	0.00
COMM11	Y19	6.01	1.10	1.41	5.10	4.12	0.65	0.01
COMM12	CHIE	7.18	0.50	1.93	5.94	3.77	1.93	0.02
COMM13	Y20	8.56	0.80	1.90	5.90	4.02	1.04	0.03
COMM14	C10	2.56	0.90	2.61	5.60	4.12	0.95	0.01
COMM15	CWM	3.00	0.80	1.49	5.50	3.26	0.59	0.01
COMM16	CWM	15.20	0.90	2.25	5.26	4.37	1.70	0.44
COMM17	CWM	3.64	1.60	3.01	6.05	2.92	2.83	0.07
COMM18	CWM	12.10	1.30	2.03	5.70	2.49	1.13	3.29
COMM19	CWM	11.40	1.10	2.29	5.84	3.14	1.13	1.91
COMM20	CLOE	24.30	0.50	1.58	6.16	3.73	1.11	0.01
COMM21	CHIE	11.30	0.50	2.02	4.88	5.16	1.82	0.01
ID3	CWM	11.40	0.60	1.94	5.40	3.42	1.49	0.01
ID5	C10	2.92	1.00	1.21	5.10	3.78	0.72	0.00
ID6	CWM	20.70	2.50	4.13	5.66	3.11	3.12	1.04
ID8	CWM	17.90	1.90	2.30	5.00	3.19	1.42	1.73
ID9	CWM	1.67	0.30	0.88	6.00	3.73	0.44	0.00
ID15	Y19	21.00	0.80	1.86	13.00	2.42	1.13	0.10
ID16	Y19	9.70	0.80	1.25	5.40	3.61	0.97	0.01
HG Comp	-	17.10	1.30	2.65	4.27	3.76	1.58	1.34
MG Comp	-	10.90	1.20	2.42	4.59	3.43	1.33	0.97
LG Comp	-	7.87	0.80	1.91	5.07	3.77	1.01	0.62

Gold assays ranged from 1.67 g/t to 24.3 g/t, averaging 10.52 g/t. Total sulphur (S) content ranged from 0.88% to 4.13%, averaging 1.96% with majority of sulphur species being sulphide sulphur S-2. Minor concentrations of total organic carbon (TOC) were found in most samples tested. Arsenic content ranged from 0.44% to 3.12%, averaging 1.30% reflecting the high arsenopyrite content. Antimony assays ranged from 0% to 3.37%, averaging 0.51%.

13.2.2.3 Comminution

The comminution tests were conducted on a set of 22 composites with specific tests selected for each composite and summarized in Table 13-8 and noted as follows:

- SMC testing: COMM5, COMM12, COMM16, to COMM21 and ID9
- Bond ball mill work index (BWi) testing: COMM1 to COMM15, COMM19 and ID9
- Bond rod mill work index (RWi) testing: COMM1 to COMM5
- Bond abrasion index (AI) testing: COMM1 to COMM15 and COMM19

Closing screen sizes of 1.41 mm and 106 µm for Bond RWi and BWi, respectively.

Table 13-8: 2023 Comminution Result Summary

Assay Statistic	Axb (SMC)	RWi	BWi	ai
	-	kWh/t	kWh/t	g
10th Percentile	30.3	-	-	-
25th Percentile	30.8	-	-	-
Average	33.6	18.6	20.0	0.098
75th Percentile	35.0	19.2	21.1	0.119
90th Percentile	39.1	19.4	22.0	0.203

The results show that the samples have moderately hard resistance to impact breakage based on the SAG Mill Comminution (SMC) Test results. The average RWi and BWi values of 18.6 (metric) and 20.0 (metric) are classified as hard. The average AI of 0.098 is considered non-abrasive and reflects the low silica and high carbonate content.

13.2.2.4 Flotation Testing

Each composite consisted of a 2 kg charge was tested using the same procedure outlined in Section 13.2.1.1. Flotation times included:

- Rougher flotation, 21.5 minutes.
- First cleaner flotation, 15 minutes.
- Second cleaner flotation, 12 minutes.

- Second cleaner scavenger flotation, 4 minutes.

Each product was filtered and assayed for gold, iron, sulphur, silver, arsenic, and antimony. The results are shown in Table 13-9.

Table 13-9: 2023 Variability Samples Flotation Tests Result Summary

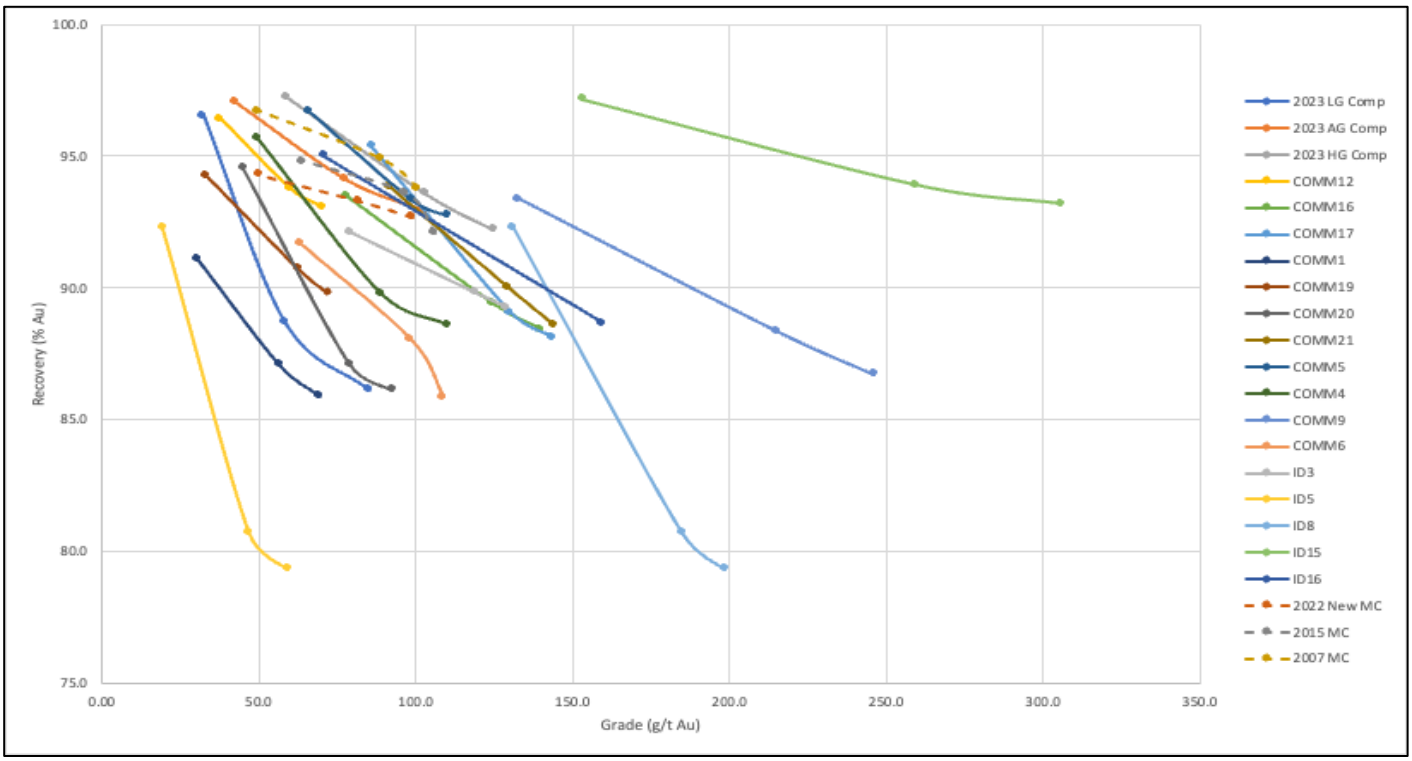
Comp	Calc. Feed Assays				Wt.	Concentrate Assay				Distribution (%)			
	Au (g/t)	Ag (g/t)	As (%)	Sb (%)	%	Au (g/t)	Ag (g/t)	As (%)	Sb (%)	Au	Ag	As	Sb
2023 LG Comp	8.26	0.8	0.82	0.62	8.0	96	7.5	9.67	4.31	87.7	86.4	87.7	57.3
2023 AG Comp	11.8	1.4	1.20	0.95	8.9	119	10.6	11.6	7.82	89.9	78.6	89.4	72.9
2023 HG Comp	15.7	1.3	1.35	0.49	8.3	85.3	7.7	8.37	2.54	86.1	79.0	84.6	84.6
COMM12	7.78	0.4	1.71	0.99	10.8	102	11.2	10.1	6.67	92.9	84.9	91.3	91.3
COMM16	17.0	0.8	1.51	1.30	11.6	125	9.1	10.6	7.73	92.2	82.8	90.8	90.8
COMM17	23.2	1.4	2.48	0.01	10.3	70.2	3.2	15.1	0.10	93.1	74.6	91.2	91.2
COMM1	6.4	0.9	1.06	0.39	10.7	140	5.2	12.0	2.45	88.4	72.5	84.9	84.9
COMM19	6.29	1.3	1.00	0.08	14.2	144	7.2	14.9	0.43	88.1	75.5	85.7	85.7
COMM20	6.35	0.4	0.96	0.01	8.0	70	9.0	11.2	0.08	85.9	77.9	83.7	83.7
COMM21	14.5	0.5	1.80	1.85	7.9	71.9	13.6	11.2	13.5	89.8	83.2	88.2	88.2
COMM5	10.9	0.5	1.46	0.02	5.9	92.7	4.8	13.7	0.22	86.1	69.2	83.9	83.9
COMM4	8.4	0.4	1.13	0.01	8.9	144	5.2	18.2	0.13	88.6	85.8	90.3	90.3
COMM9	21.3	0.5	1.81	0.01	9.2	110	4.4	14.7	0.05	92.8	89.0	92.5	92.5
COMM6	10.9	0.5	0.94	0.01	6.7	110	4.4	14.7	0.05	88.6	84.8	88.2	88.2
ID3	13.8	0.5	1.53	0.01	7.5	246	5.2	20.8	0.14	86.7	85.7	86.4	86.4
ID5	3.00	0.5	0.65	3.71	8.6	109	6.0	9.46	31.3	85.9	98.5	86.6	86.6
ID8	19.3	1.9	1.53	0.01	9.6	129	4.8	14.2	0.06	89.3	89.7	89.1	89.1
ID15	24.9	0.4	1.29	0.00	4.0	59.4	10.4	13.5	0.04	79.3	85.2	82.3	82.3
ID16	10.5	0.3	0.94	1.96	9.0	199	20.4	15.8	17.9	92.5	95.4	93.0	93.0

Gold recoveries to second cleaner concentrate ranged from 79.3% to 93.2%, averaging 88.8%, and generally followed head grades. Mass recoveries to second cleaner concentrate ranged from 4.0% to 14.2%, averaging 8.61%. Concentrate gold grades ranged from 59 g/t to 306 g/t, averaging 132 g/t. Concentrate arsenic grades averaged 13.75%, ranging from 8.37% to 20.8%. These arsenic concentrations are problematic in terms of concentrate sales as they are beyond the range that copper or lead smelters will typically accept. They are also beyond the limit that Chinese pyrite processors, the typical destination for complex gold concentrates, will typically accept. This is further discussed in Section 13.3.4

Overall projected gold recovery based on the cleaner concentrate ranged from 79% to 90% and average 89%. Mass recovery to final cleaner concentrate ranged from 4% to as high as 14%.

The location of the samples, whether it be C or Y veins, did not impact gold recoveries as recoveries generally followed gold head grades. This is demonstrated in Figure 13-7. Included in the figure are some historical test results to serve as a comparison, showing continuity in the results with the same flotation flowsheet.

Figure 13-7: Gold Grade Recovery Curves for Variability Samples



Source: Ausenco, 2025.

Locked cycle flotation tests (LCT) were completed on each of the three grade composites (LG, AG, and HG) described in Section 13.2.1. An LCT simulates closed circuit flotation operation by recycling cleaner flotation upstream to the appropriate stage as tests are completed over five to six cycles. The three locked cycle tests all achieved steady-state conditions. With higher circulating loads that can occur with some flowsheets, an increase in recovery is possible through recycle of these streams. A summary of the result is shown in Table 13-10.

Table 13-10: Locked Cycle Flotation Tests Result Summary

Comp	Calc. Feed Assays (% or g/t)				Mass	Conc. Assay (% or g/t)				Distribution (%)			
	Au	Ag	As	Sb	%	Au	Ag	As	Sb	Au	Ag	As	Sb
LG Comp	8.74	0.7	0.89	0.62	8.0	96	7.5	9.67	4.31	87.7	86.4	87.7	57.3
AG Comp	11.8	1.2	1.19	0.95	8.9	119	10.6	11.6	7.82	89.9	78.6	89.4	72.9
HG Comp	16.3	1.3	1.40	1.30	11.0	138	10.2	12.0	10.6	89.6	82.7	88.4	78.2

Results from the LCTs were comparable to the batch cleaner tests for each respective composite. Gold recovery into the concentrate measured between 88% and 90%, an increase of 1.5% for the LG Comp but decreased approximately 3% compared to the open circuit testing. However, the concentrate quality was improved from 20 to 30 g/t Au. The mass recovery ranged between 8 and 11%. The arsenic and antimony content in concentrate measured between 9.7% to 12% and 4.3% to 10.6%, respectively. These results show that the open circuit variability flotation test results can be used without adjustment for recovery estimates.

13.2.3 Forte Analytical LLC (2024)

A follow up program was completed at Forte in Fort Collins, CO with the objective of reducing concentrate mass recovery while maintaining overall gold recoveries using gravity concentration and other methods. The remaining 2023 variability samples were used for this program. The results did not improve on the 2023 program results and are not discussed here as the results were not used for process design. No final report was provided for this program. The remaining samples were used to generate flotation tailings samples, including simulated cyclone underflow from backfill preparation and flotation concentrate for solids liquids separation testing at FLSmidth.

13.2.4 FLSmidth Inc (2024)

FLSmidth was engaged by Canagold to conduct characterization, thickening, and filtration testing on samples from the New Polaris Project. These samples were produced by Forte Analytical in 2024. Four sets of samples were sent to their Salt Lake City Operations in Midvale, UT that were characterized as follows:

- Flotation Tailings – Thickening, pressure, and vacuum filtration testing
- Simulated Backfill Cyclone Overflow – Thickening and pressure filtration testing
- Simulated Backfill Cyclone Underflow – Pressure and vacuum filtration testing
- Flotation Concentrate – Thickening, pressure, and vacuum filtration testing

Flotation samples were crudely wet screened by Forte to simulate hydrocyclone underflow and overflow streams produced when backfill is being produced.

The primary objectives of this solid liquid separation test program were to measure solid liquid separation rates to predict operating parameters and equipment specifications for full scale thickeners, pressure filters, and horizontal belt filters (HBF).

13.2.4.1 Thickening

FLS accomplished thickening objectives by screening flocculants, determining feed well design criteria related to effective feed conditioning for flocculation, conducted dynamic settling tests to determine sizing and design criteria, and measured thickened tailings rheology to determine required rake torque and predict underflow manageability.

Testing determined that a high molecular weight low charge anionic polyacrylamide type flocculant was determined to be the most suitable for producing the best overflow clarity and settling velocities. Recommended flocculant dosages

for flotation tailings, flotation concentrate, and backfill cyclone overflow are approximately 25, 20, and 30 g/t of dry solids, respectively.

Optimum feed well suspended solids concentrations for flocculation was determined by flux testing and is approximately 8% by weight for the flotation tailings and backfill cyclone overflow, and 12% by weight for the flotation concentrate.

Thickening tests resulted in an expected solids loading rate of 25 t/d/m^2, or a unit area of 0.04 m^2/t/d for the flotation tailings, flotation concentrate, and backfill cyclone overflow. Thickeners can produce underflow solids concentrations of approximately 60, 70, and 53% by weight for the flotation tailings, flotation concentrate, and backfill cyclone overflow, respectively, in approximately 45-60 minutes of underflow residence time. Underflow yield stress of less than 20-Pa is expected at these solids concentrations.

13.2.4.2 Filtration

Vacuum filtration tests using filter feed solids concentrations ranging from 53 to 70 % by weight were performed. The solid concentrations used for filtration testing was determined by the results of the concentrate and tailings filter, but the backfill cyclone underflow was tested using the received solids concentration.

The backfill cyclone underflow samples received were too coarse for a traditional bench scale pressure filtration testing pump and was instead evaluated using a FLSmidth pneumapress with the intention of extrapolating data for pressure filter parameters. Rapid cake consolidation and short cycle times were seen under these testing parameters.

Vacuum filtration for both the concentrate and tailings resulted in wet cakes with moisture contents too high to be conveyed. The backfill cyclone underflow vacuum filtration test resulted in a fast-forming cake that appeared to be suitable for conveying. Fines were noticeably forming on top of the backfill cyclone underflow test which create the possibly of impeded filtration. The segregation of the fines could be mitigated by feeding the filter with a higher solids concentration.

Pressure filtration testing of the tailings, concentrate, and backfill cyclone overflow resulted in reasonable filtration rates and low cake moistures. Based on these observations, pressure filtration would be a suitable technology for dewatering of these sample streams. Results of the pressure filtration test is summarized below in Table 13-11.

Table 13-11: FLSmidth Pressure Filtration Test Results

Measurement	Flotation Tailings	Backfill Cyclone Overflow	Backfill Cyclone Underflow	Flotation Concentrate
Chamber Type	Recessed			
Filter Media	POPR 966			
Feed Sus. Solids (wt%)	60	53	60	70
Form Pressure (bar)	15	15	10	10
Form Time (s)	120	240	15	110
Cake Thickness (mm)	50	50	50	50

Measurement	Flotation Tailings	Backfill Cyclone Overflow	Backfill Cyclone Underflow	Flotation Concentrate
Air Blow Pressure (bar)	7.0	7.0	9.3	7.0
Air Blow Time (s)	162	444	45	156
Design Cake Moisture (wt%)	15	15	14	12
Dry Cake Density (kg/m^3)	1823	1792	1224	2252
Filtration Rate (kg/m^2/h)	326	212	403	414

13.3 Recovery Models

A total of 19 samples, three which are a low, medium, and high-grade composite, from the 2023 BaseMet flotation variability tests shown in Table 13-9 were used to establish the recovery models for the New Polaris Project.

Results between the locked cycle tests and open cycle tests were very similar. This was primarily due to the minimal recovery increase from recycling the second and first cleaner tailings. For these reasons, the recovery models were derived using the open cycle testwork, which would be a reasonable approximation to actual operational recoveries.

At the time of sample selection, the block model did not include elements such as arsenic, sulphur, and antimony that are needed for recovery modeling and at the time BIOX™ process design. The block model was updated to include these elements in 2025. The model range of grades for these elements are at or below the bottom end of the grades for these elements for the samples selected for recovery modeling. A comparison of these results is shown in Table 13-12. All values are in percent.

Table 13-12: 2023 Variability Samples Head Assay Statistics

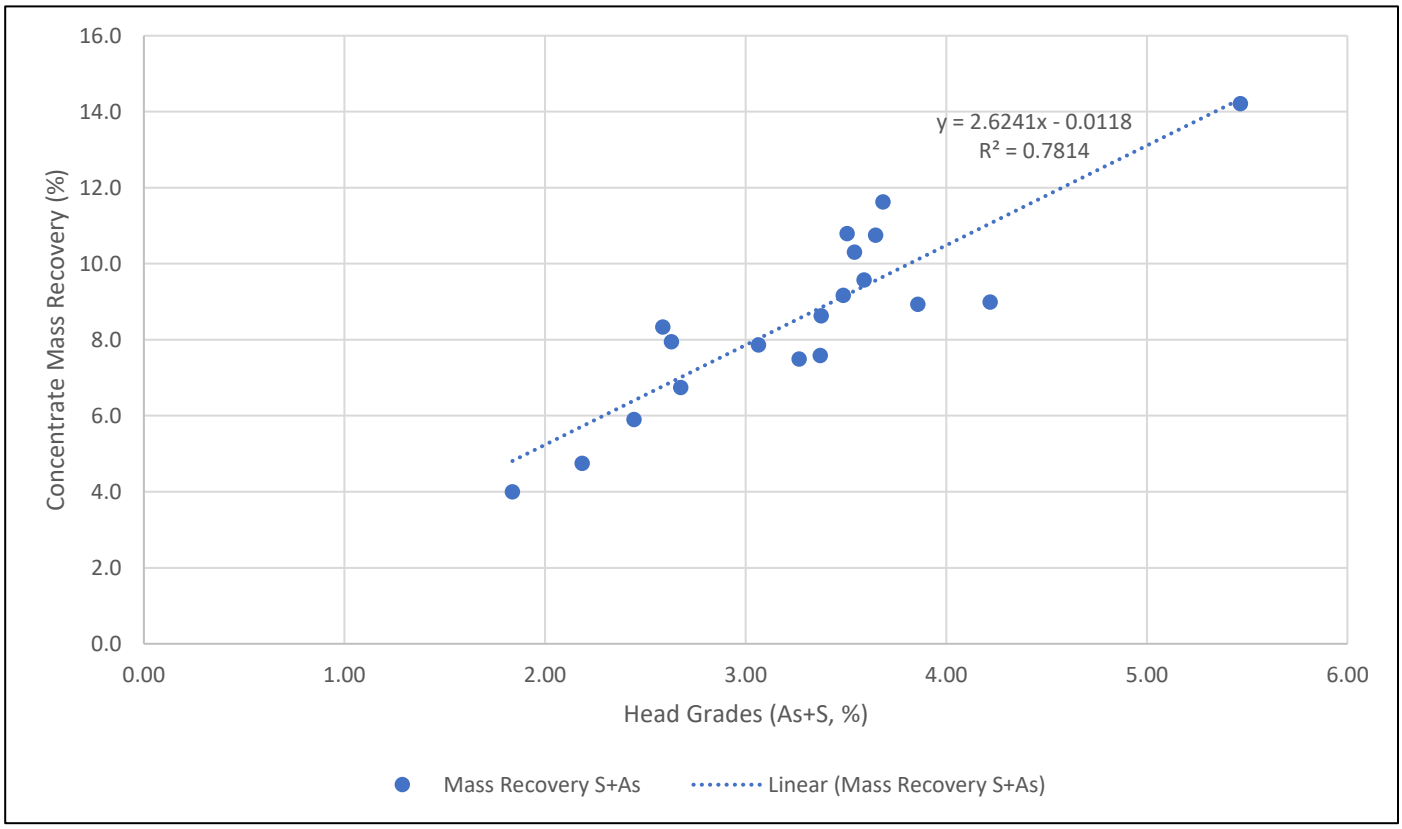
Element	Block Model			Samples in Recovery Model		
	Average %	Maximum %	Minimum %	Average %	Maximum %	Minimum
S	1.25	1.81	0.71	1.96	2.99	1.18
As	0.94	1.28	0.46	1.43	2.88	0.65
Sb	0.15	0.60	0.02	0.58	3.71	0.00

Although the sample grades are higher than the block model values, the largely linear relationships in the models that were developed permits use of the test data. Future testing should include samples with sulphur, arsenic and antimony aligned to grades in the block model.

13.3.1 Concentrate Mass Recovery Model

Gold, sulphur and arsenic assays were evaluated to support concentrate mass recovery modeling. In terms of mineralogy, the bulk of the gold is associated with arsenopyrite and pyrite, therefore the concentrate mass recovery was modeled against arsenic plus sulphur head grades as this provides the best fit. The 19 samples had a head grade range of As + S % of 1.84 to 5.47.

Figure 13-8: Concentrate Mass recovery as a Function of Sulphur (S) Plus Arsenic (As) Head Grades



Source: Ausenco, 2025.

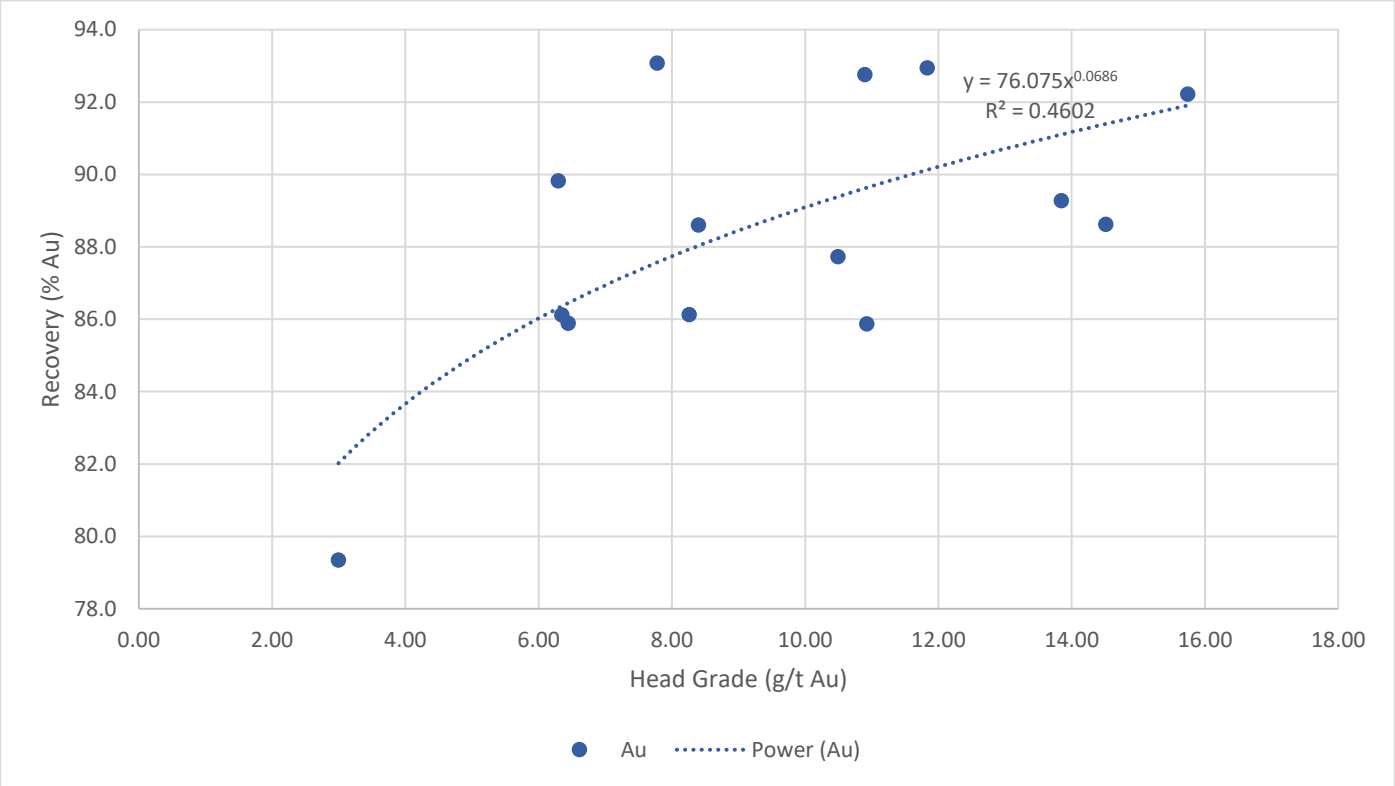
$$Concentrate\ Mass\ Recovery\ (\%) = 2.6241 * (As + S)\% - 0.0118$$

The linear best fit for plotting arsenic and sulphur head grades against concentrate mass recovery has a coefficient of determination (R^2) correlation of 0.78 which represents a very good correlation between the plotted data points and trend.

13.3.2 Gold Concentrate Grade Model

Gold recovery as a function of gold head grades was found to provide the best relationship to predict gold recovery. The modeled relationship is displayed in Figure 13-9.

Figure 13-9: Gold Recovery as a Function of Gold Head Grade



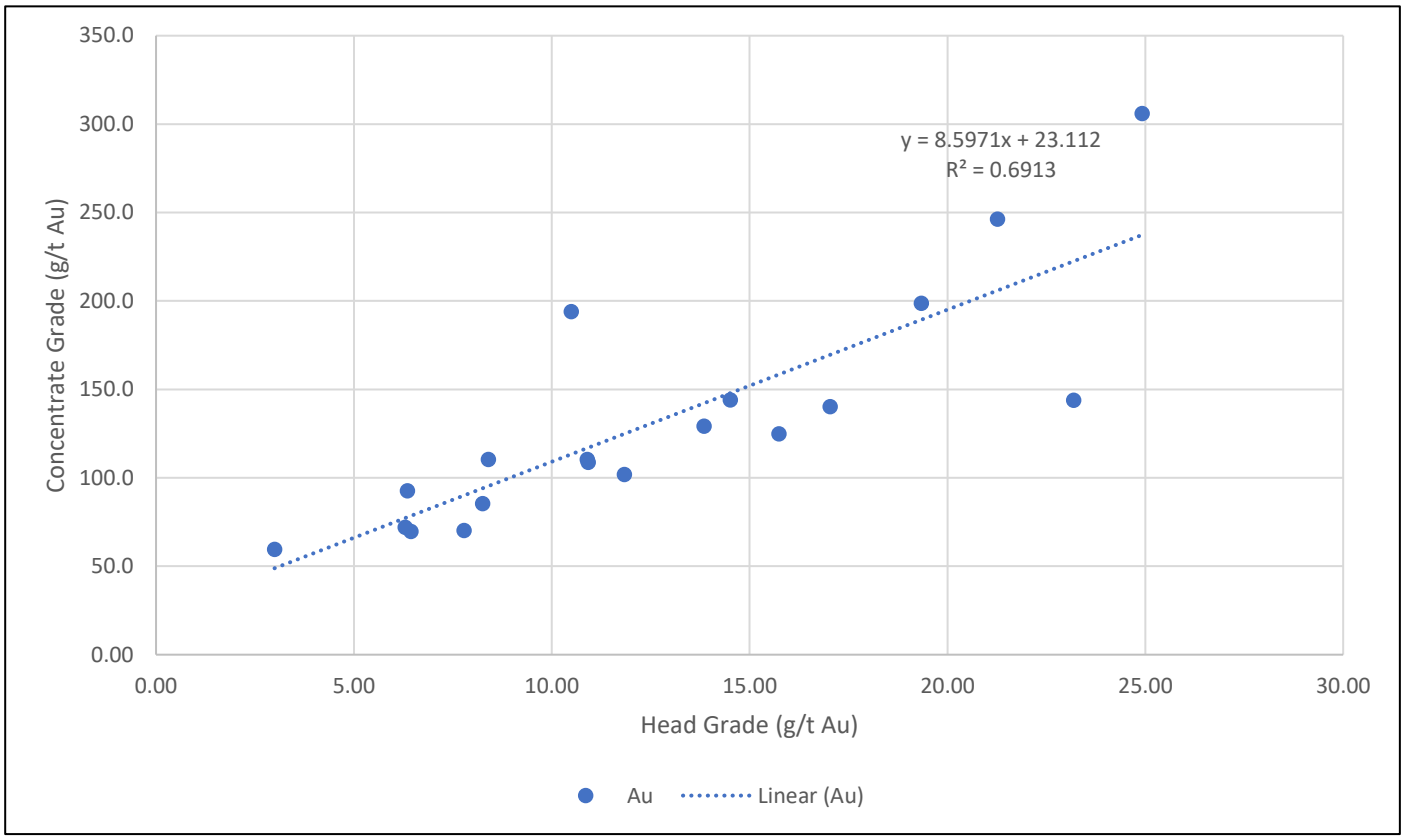
Source: Ausenco, 2025.

$$Au\ Recovery\ (\%) = 76.075 * (Au\ Head\ Grade, g/t)\text{^}0.0686$$

13.3.3 Gold Concentrate Grade Model

Concentrate gold grades are determined using the modeled gold recovery and the modeled concentrate mass recovery. A gold concentrate grade model was also developed to provide a check on the mass balanced grades. Gold head grade was plotted against concentrate gold grade in Figure 13-10. The predicted concentrate gold grades were generally below grades from mass balancing.

Figure 13-10: Concentrate Gold Grade as a Function Gold Head Grade



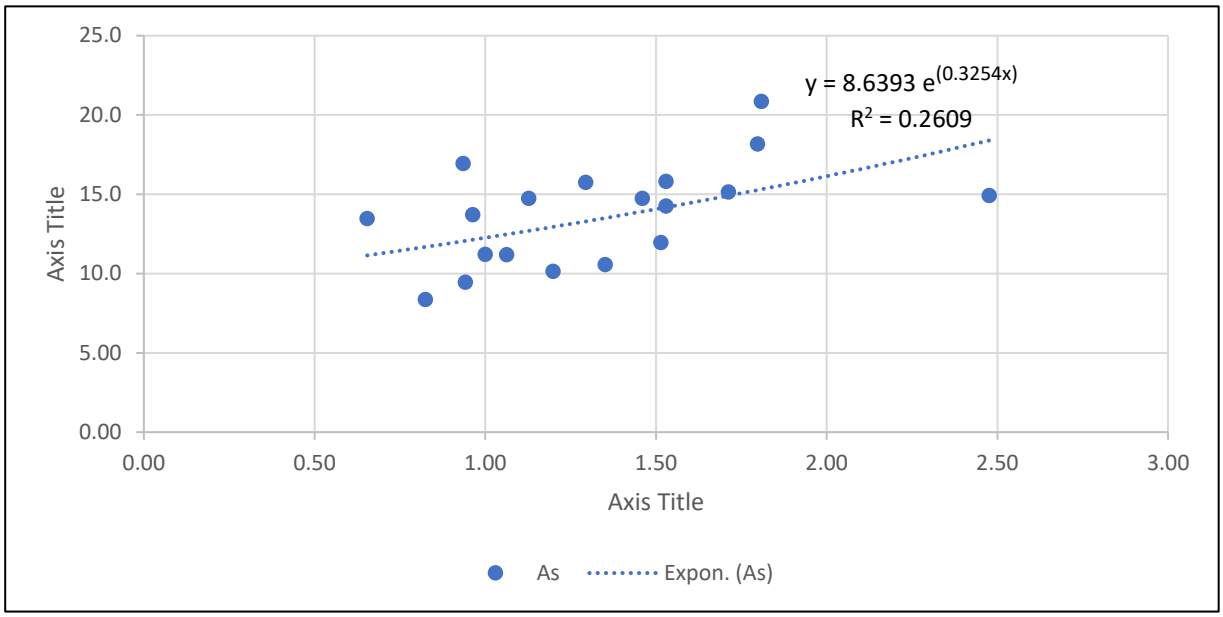
Source: Ausenco, 2025.

$$Concentrate\ Grade, Au\ (g/t) = 8.5971 * Head\ Grade, Au\ (g/t) + 23.112$$

13.3.4 Arsenic and Antimony Concentrate Grade Models

Arsenic and antimony sample grades and concentrate grades were also modeled to estimate their respective concentrate grades. Arsenic concentrations are expected to be high and may incur penalties or, in the worst case, make the concentrates unsaleable without blending with clean concentrates. Antimony was also modeled to determine if there was any economic potential for the antimony content or if it will incur penalties. Arsenic and antimony recovery models are plotted in Figure 13-11 and Figure 13-12, respectively.

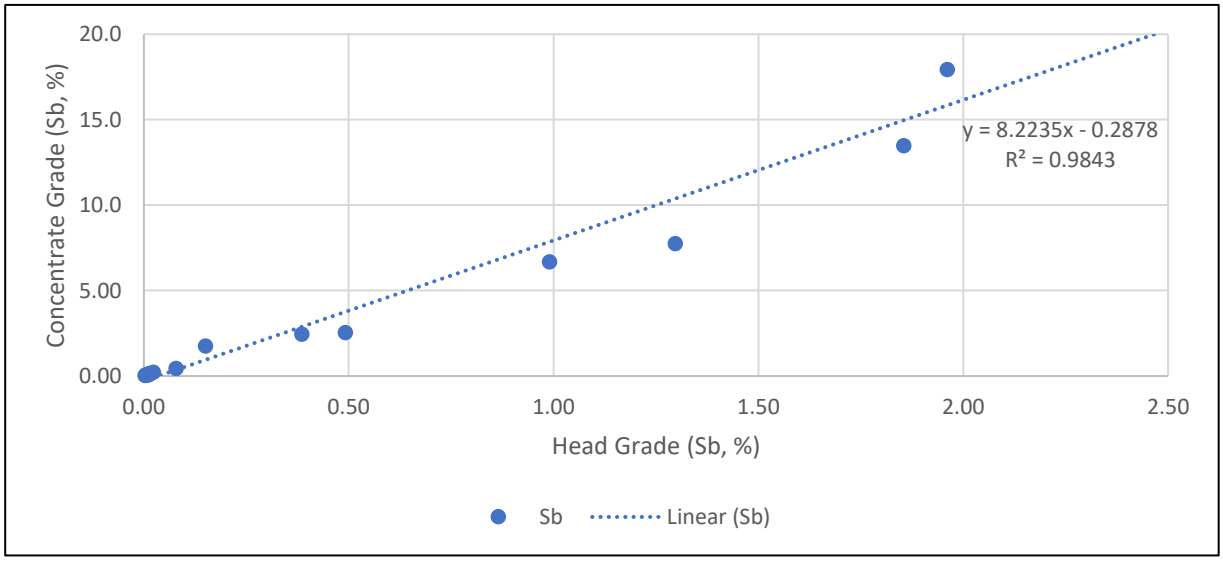
Figure 13-11: Arsenic Concentrate Grade as Function of Arsenic Head Grades



Source: Ausenco, 2025.

$$Arsenic\ Concentrate\ Grade\ (\%) = 8.6393 * EXP(0.3254 * Arsenic\ Grade\ (\%))$$

Figure 13-12: Antimony Concentrate Grade as a Function of Antimony Head Grade



Source: Ausenco, 2025.

$$Antimony\ Concentrate\ Grade\ (\%) = 8.2235 * Antimony\ Grade\ (\%) - 0.2878$$

13.4 Metallurgical Variability

The methodology used to select the 2023 variability samples is described in Section 13.3. The variability samples provide spatial representation of the deposit. The number of samples selected also reflect the relative amounts of the major vein systems. The samples are also representative of the gold grades across the deposit.

13.5 Deleterious Elements

As discussed, arsenic is a deleterious element that will be recovered to the concentrate at levels that are expected to incur penalties or render the concentrates unsaleable without blending with cleaner concentrates. Antimony concentration in the concentrate may also attract penalties from off-takers.

13.6 Recovery Estimates

Using the recovery models described in Section 13.3, recovery estimates, including concentrate grades are shown in Table 13-13. These values were derived from block model outputs.

Table 13-13: Recovery Estimates

Description	Unit	Value
Gold Recovery	%	88.9
Average Concentrate Mass Recovery	%	6.92
Concentrate Production	t	195,750
Gold Contained in Concentrate	oz	803,808
Average Concentrate Grade	g/t Au	128
	% As	12.34
	% Sb	1.22

14 MINERAL RESOURCE ESTIMATES

14.1 Introduction

The Mineral Resource Estimate completed in 2025 reported upon herein appends results for antimony and sulphur to the 2023 Mineral Resource Estimate. The gold resource remains unchanged. The Mineral Resources for the New Polaris Project have been completed by Sue Bird, P. Eng of MMTS in accordance with updated Canadian Institute of Mining, Metallurgy and Petroleum (CIM) Definition Standards (CIM 2014) and were estimated using the 2019 CIM Best Practices Guidelines. Updates from the previous model include re-analysis of historical samples to include antimony. Mineral resources for antimony were calculated as a sub-set within the confining shapes used for the 2023 Mineral Resource Estimate for gold.

14.2 Mineral Resource Estimate

The Resource Estimate for the New Polaris deposit is summarized in Table 14-1. A comparison with the previous (2019) resource estimate is also summarized. The resource has been summarized at various cut-off grades with the base case gold grade cut-off of 4.0 g/t highlighted. At each cut-off the total material within a potential confining mining shape is reported. Therefore, a separate mining shape has been created for each cut-off in the table. The antimony resource is reported as a sub-set of the Au resource and is summarized in Table 14-2.

The base case cut-off grade is 4 g/t Au and the Mineral Resource has been confined by "reasonable prospects of eventual economic extraction" shapes using the following assumptions:

- Metal prices of US$1,750/oz Au and foreign exchange rate (Forex) of US$0.75 to C$1.00
- Payable metal of 99% Au
- Off-site costs (refining, transport and insurance) of US$7.00/oz
- Mining cost of C$82.78/t, Processing costs of C$105.00/t and G&A and site costs of C$66.00/t
- Metallurgical Au recovery of 90.5%

The “reasonable prospects for eventual economic extraction” confining shape also considers a minimum mining width of 2.0 m, and removes shapes considered too small and separated from the primary mining volumes. Previous underground mining has been accounted for by restricting the modelled veins away from the mined-out shapes.

The effective date of this Resource Estimate is January 22, 2025, which includes the antimony in addition to the base case gold resource.

Table 14-1: Updated Mineral Resource Estimate and Comparison to the 2019 Resource

Class	Cut-off Au (g/t)	2025 Resource			2019 Resource			Difference as a Percent: (2025-2019)/2019		
		Tonnage (kt)	Au (g/t)	Au (koz)	Tonnage (kt)	Au (g/t)	Au (koz)	Tonnage	Au Grade	Au Metal
Indicated	**3**	3,118	11.21	1,124	1,798	10.40	601	73	8	87
	4	2,965	11.61	1,107	1,687	10.80	586	76	8	89
	5	2,769	12.11	1,078	1,556	11.30	565	78	7	91
	6	2,525	12.75	1,035	1,403	12.00	541	80	6	91
	7	2,270	13.45	981	1,260	12.60	510	80	7	92
	8	2,049	14.09	928	1,105	13.30	473	85	6	96
	9	1,814	14.81	864	947	14.10	429	92	5	101
	10	1,594	15.55	797	1,639	9.50	501	-3	64	59
Inferred	**3**	1,061	8.24	281	1,582	9.80	498	-33	-16	-44
	4	926	8.93	266	1,483	10.20	486	-38	-12	-45
	5	817	9.52	250	1,351	10.70	465	-40	-11	-46
	6	706	10.16	231	1,223	11.20	440	-42	-9	-48
	7	603	10.78	209	942	12.50	379	-36	-14	-45
	8	491	11.52	182	753	13.80	334	-35	-17	-46
	9	371	12.51	149	653	14.60	307	-43	-14	-51
	10	291	13.33	125	0	0.00	0			

Table 14-2: Mineral Resource Estimate for Antimony (Sb) within the Base Case Au Resource

Class	Tonnage	Sb (%)	Sb Metal (lbs)	Sb Metal (tonnes)
Indicated	859,989	0.6547	12,412,778	5,630.2
Inferred	99,581	1.2004	2,635,337	1,195.3

Notes for Mineral Resource Estimate:

1. Mineral resources are not mineral reserves and do not have demonstrated economic viability.
2. There is no certainty that all or any part of the mineral resources will be converted into mineral reserves.
3. Resources are reported using the 2014 CIM Definition Standards and were estimated using the 2019 CIM Best Practices Guidelines.
4. The base case Mineral Resource has been confined by "reasonable prospects of eventual economic extraction" shape using the following assumptions:
   a. Metal prices of US$1,750/oz Au and Forex of US$0.75=C$1.00.
   b. Payable metal of 99% Au.
   c. Off-site costs (refining, transport and insurance) of US$7.00/oz.
   d. Mining cost of C$82.78/t.
   e. Processing costs of C$105.00/t, and G&A and site costs of C$66.00/t.
   f. Metallurgical Au recovery of 90.5%.
5. The resulting Net Smelter Return equation is: NSR (C$/t)=Au*90.5%*US$74.72 g/t.
6. The specific gravity is 2.81 for the entire deposit.
7. The antimony Resource is reported as a subset of the total Mineral Resource within the 4.0 g/t Au underground mining shape.
8. Sb is a by-product of Au processing; therefore, it is reported using the same classification as the Au Resource within the 4.0 g/t Au shape.
9. Numbers may not add due to rounding.

The QP for the resource estimate is not aware of any environmental, permitting, legal, title, taxation, socio-economic, marketing, political, or other relevant factors that could materially affect the Mineral Resource Estimate other than those stated in this report.

Factors that may affect the estimates include: metal price assumptions, changes in interpretations of mineralization geometry and continuity of mineralization zones, changes to kriging assumptions, metallurgical recovery assumptions, operating cost assumptions, confidence in the modifying factors, including assumptions that surface rights to allow mining infrastructure to be constructed will be forthcoming, delays or other issues in reaching agreements with local or regulatory authorities and stakeholders, and changes in land tenure requirements or in permitting requirement.

14.3 Key Assumptions and Data Used in the Resource Estimate

14.3.1 Underground Workings and Topography

Underground workings and topography have been provided as solids and surfaces. They are the same as those used for the MMTS 2019 Resource Estimate with the exception that they have been converted to UTM coordinates (Underhill, 2021). The underground workings consist primarily of mined-out stopes and stoping development levels used for historic shrinkage and long hole mining with side hill ramp access.

The topography and bottom of the overburden are both above any veins considered for the Resource Estimate, and the underground workings have been considered in the modelling procedure by clipping the modeled veins away from them.

To the knowledge of the QP for the Resource Estimate and based on the tour of the property and discussion with the site geologist, there are no additional underground workings that have not been included in the shapes provided in the areas of the 2023 resource estimate. The topography and underground workings are shown in Figure 14-1.

14.3.2 Database

A summary of the total number of drillholes used for the Resource Estimate is found in Table 14-3 below. All zero value assays and missing assays values within the modeled vein shapes have been treated as zero on the assumption that they represent unmineralized dilution.

Table 14-3: Summary of Drillholes and Assays used in the Resource Estimate

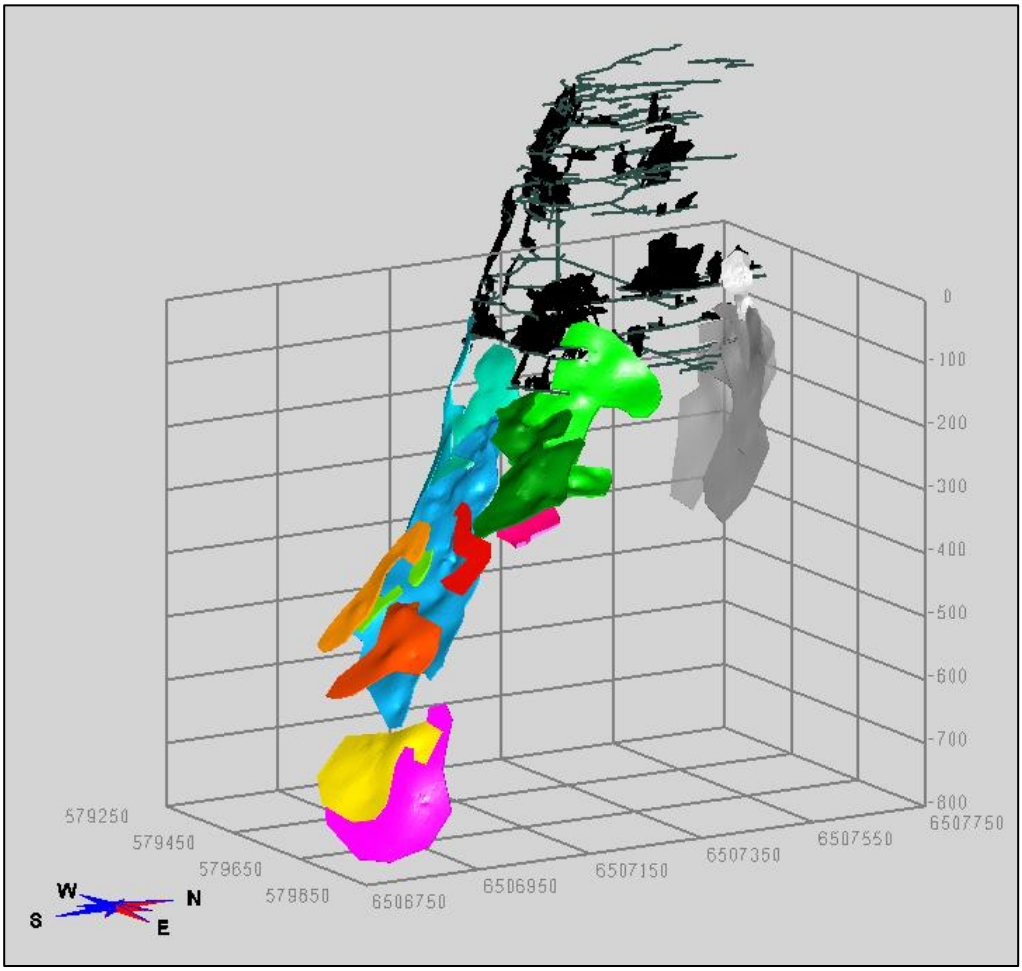
Year	Number of Drill Holes Intercepting Veins	Total Length drilled (m)	Assayed Length in Veins (m)	Number of Assays	% Length Assayed
<1989	48	1,430.41	253.69	211	18
1989	7	1,801.05	50.17	62	3
1990	6	1,923.58	41.37	50	2
1991	10	3,161.84	99.5	137	3
1992	14	3,897.08	79.24	77	2
1993	4	691.90	21.96	17	3
1994	7	2,019.00	35.99	45	2
1995	5	3,753.92	50.26	63	1
1997	3	756.21	31.66	26	4
2004	11	2,766.99	87.07	89	3
2005	8	2,295.14	96.21	124	4
2006	54	18,581.60	372.2	364	2
2021	40	19,477.00	249.35	297	1
2022	17	7,687.00	118.16	130	2
Total	**234**	**70,242.72**	**1586.83**	**1692**	**2**

14.4 Geologic Models

The mineralized veins discussed in Section 7 have been modeled in 3D using the Hexagon MinePlan Implicit Modeler® tool which uses the radial basis function (RBF similar to other industry software) to define surfaces based on user constraints.

A total of 17 veins (domains) have been modeled compared to 6 in the previous modelling. The increase in number of veins is due to the additional drilling, additional fault modeling and tighter control on dilution resulting in veins being split. The modeled shapes generally target a 4.0 g/t Au cut-off, while also honouring the known geological trends. Lower-grade dilution is occasionally included to maintain continuity and to ensure the veins are a minimum of 2 m in true thickness to align with the expected mining selectivity. The modeled shapes have also been clipped away from mined-out areas. An illustration of the veins is provided in Figure 14-1, with the “C” veins coloured and the “Y” veins to north shown in grey, and the existing underground workings shown in black.

Figure 14-1: Three-dimensional View – Veins and Existing Underground



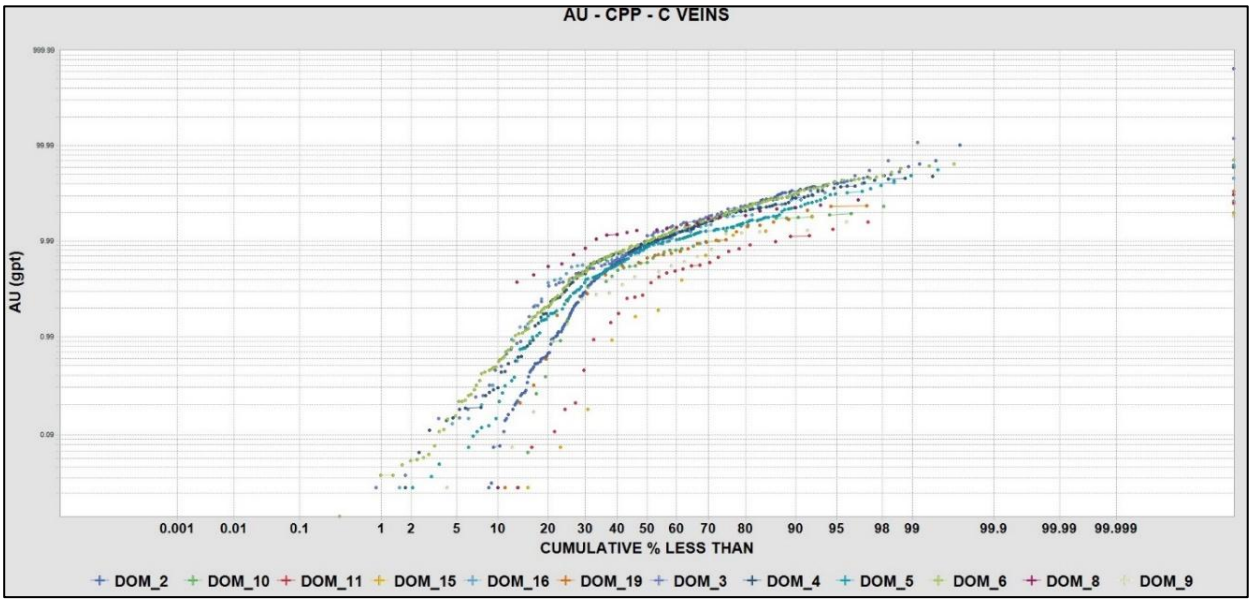
Source: MMTS, 2023.

14.5 Assay Statistics, Capping and Compositing

MMTS has examined the sample assays in the veins using boxplots, histograms, and cumulative probability plots (CPPs). The gold grade distribution is shown below in Figure 14-2 as CPPs by Domain. The grade distribution for Au is mainly lognormal except at very high grades where outliers are evident and therefore capping of assays has been done.

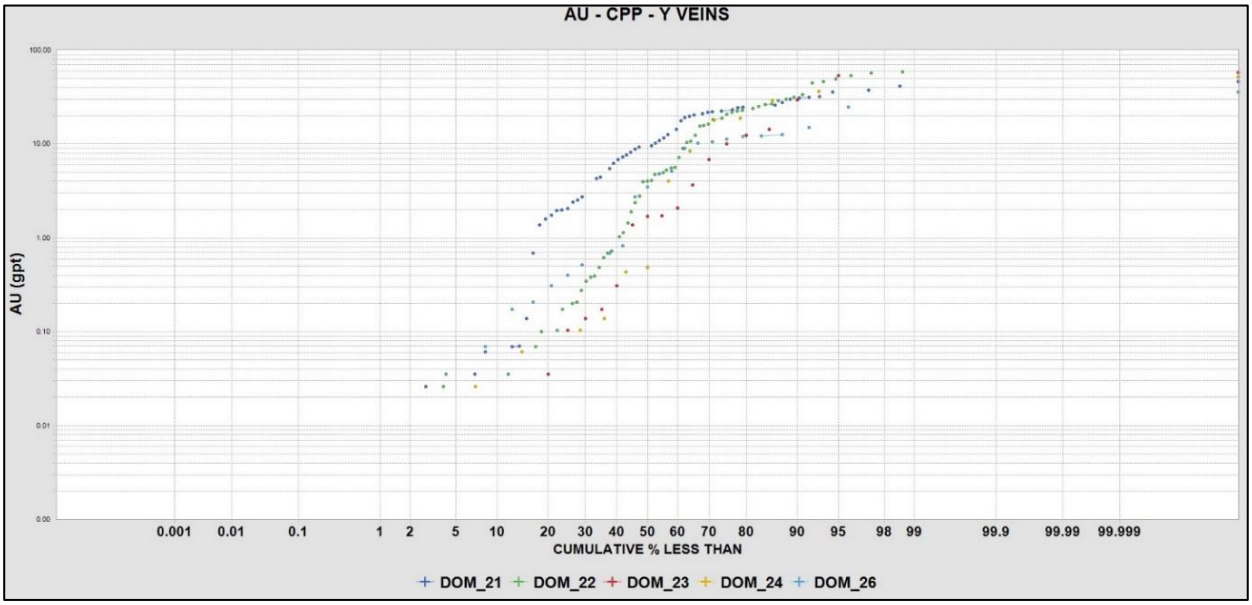
Table 14-4 summarizes the capping done on the assays prior to compositing. For clarity, also summarized in Table 14-4 is the Outlier Restrictions which has been applied to the composites during interpolation. For composite grades above the Outlier value provided, and at distances greater than 4m from the data, the value is essentially capped to the outlier.

Figure 14-2: Gold Grade (g/t) by Domain – "C" Veins



Source: MMTS, 2023.

Figure 14-3: Gold Grade (g/t) by Domain – "Y" Veins



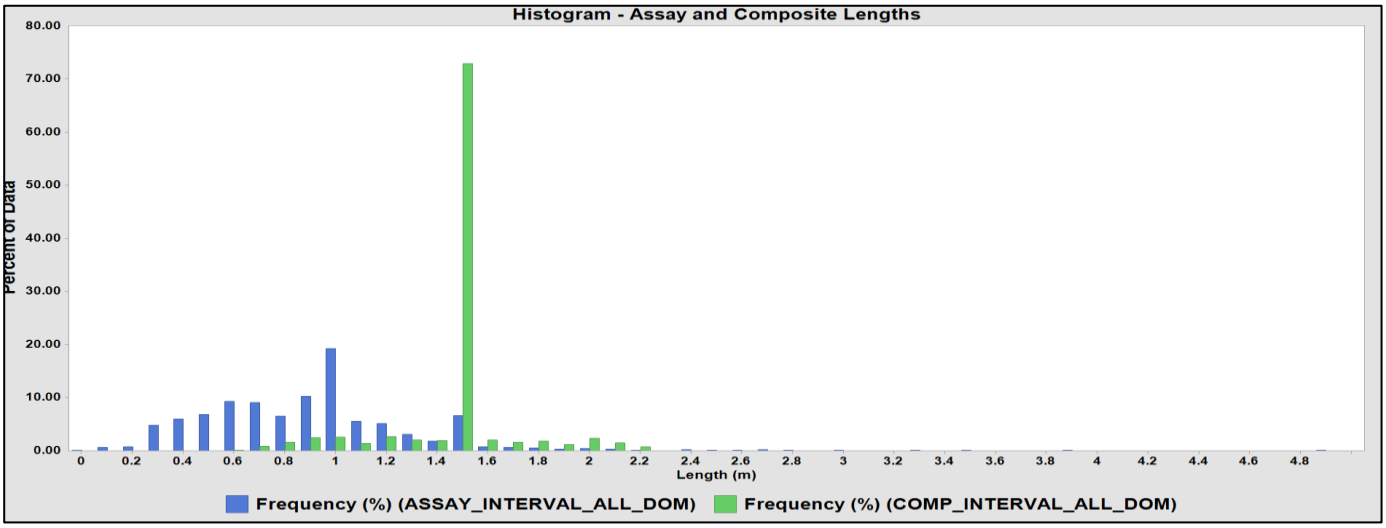
Source: MMTS, 2023.

Table 14-4: Capping of Assays and Outlier Restriction of Composites by Domain

Domain	Au Cap (g/t)	Au Outlier (g/t)	Outlier Distance
2	50		
3	50		
4	50	25	4
5	50	25	4
6	50	35	4
8	25	20	4
9	13	10	4
10	20	15	4
11	10		
15	12		
16	30	20	4
19	20		
21	30	20	4
22	50	35	4
23	30		
24	30		
26	15		

Assay sample lengths have varied with the drill programs. A histogram of the assay intervals for the New Polaris deposit is shown in Figure 14-4. The majority of sampling has been done using a 1.5 m interval length. Therefore, this is the value that has been used for the base length when compositing. The compositing also honored the domain boundaries. Assay intervals less than 0.75 m have been added to the previous composite.

Figure 14-4: Histograms of Assay and Composite Lengths within Mineralized Domains



Source: MMTS, 2023.

The assay and composite basic statistics within the modeled veins are summarized in Table 14-5. The Coefficient of Variation before capping is 1.6 indicating that linear interpolation is appropriate.

Table 14-5: Summary Statistics of Assays and Composites within the Domains

Parameter	Source		
	Assays	Capped Assays	Capped Composites
Num Samples	1616.00	1692.00	1069.00
Num Missing Samples	76.00	0.00	0.00
Min (g/t)	0.01	0.00	0.00
Max (g/t)	646.00	50.00	50.00
Weighted SD	20.61	11.78	9.76
Weighted CV	1.63	1.05	0.91

To ensure correct compositing the length and weighted mean grades of the composite have been compared to the original assay data in Microsoft Excel with the results summarized in the Table 14-6 illustrating that there is no bias introduced in the compositing process.

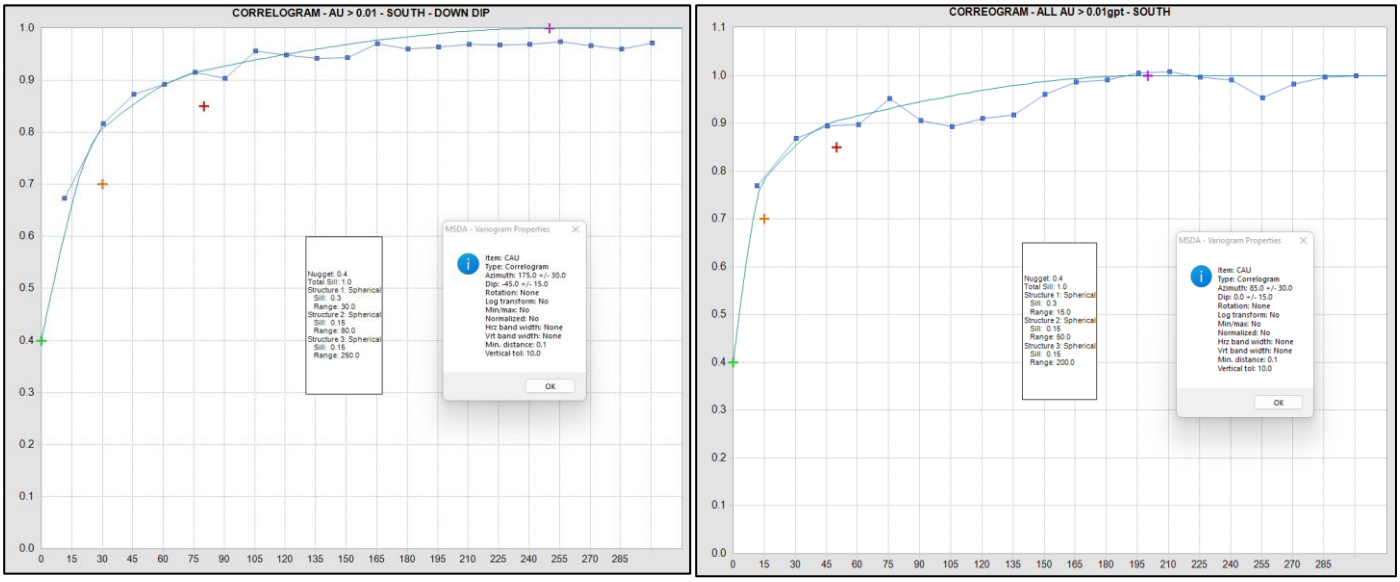
Table 14-6: Compositing Validation

Description	Grade X Length	Length (m)	Weight Average Grade (g/t)
All Au Assays in Database	53,704.4	16,903.4	3.177
All Au Assays in MinePlan Assay Table	53,704.4	16,903.4	3.177
Capped Au in MinePlan Assay Table	17,035.9	1,586.8	10.736
Capped Au in MinePlan Composite Table	17,035.9	1,586.8	10.736

14.6 Variography

The interpolations have been done by inverse distance squared. However, variograms have been used to determine drillholes spacing necessary to classify the material to Measured, Indicated and Inferred. The figure below illustrates variograms in the approximate strike and dip directions of the “C” veins.

Figure 14-5: Sample Variography for the "C" Veins



Source: MMTS, 2023.

14.7 Specific Gravity

A total of 76 specific gravity determinations are available for examination. All measurements are from core drilled in 1996 and 1997. Unfortunately, most of these samples are from parts of the deposit not included in this resource estimate (higher up on the C Vein). A total of 16 samples are from C Vein Domain 4. These 16 samples had an average specific gravity of 2.81 and this is the value used for all estimated blocks. The average value of all the gold-bearing samples is 2.88.

14.8 Block Model Estimation Parameters

Block dimensions are 5m x 5m x 5m with the extent of the block model summarized in UTM coordinates, in Table 14-7.

Table 14-7: New Polaris Model Extents

Direction	Minimum	Maximum	Size (m)	# Blocks
Easting	579,200	579,900	5	140
Northing	6,506,700	6,507,800	5	220
Elevation	-850	130	5	196

Modelling has been accomplished using inverse distance squared (ID2), which is a change from Ordinary Kriging (OK) which had been used previously. However, variograms were created on a global basis to aid in determination of Classification parameters.

Domain matching for each of the 17 domains has been used, and a search ellipse with axis in the same orientation as the primary axes of each domain has been used to select samples for interpolation. Composite values have also been restricted at higher cut-offs to reduce the impact of high-grade samples. The tables below summarize the interpolation search parameters.

Table 14-8: Summary of Search Parameters

Search Parameter		Pass1	Pass2	Pass3	Pass4
Search Distance (m)	Major	30	60	90	150
	Minor	30	60	90	150
	Vertical	10	20	30	30
Number of Composites	Minimum	4	4	4	4
	Maximum	12	12	12	12
	Maximum/DH	2	2	2	2
	Maximum/Quadrant	2	2	2	2

Table 14-9: Summary of Rotation by Vein

Domain	Rot-Y	Rot-X	Rot-Z
2	267	0	46
3	290	0	67
4	265	0	46
5	249	0	32
6	210	0	50
8	270	0	47
9	265	0	51
10	265	0	53
11	257	0	53
15	247	0	16
16	240	0	42
19	243	0	57
21	350	0	87
22	350	0	87
23	3	0	89
24	8	0	88
26	346	0	87

14.9 Classification

Classification is based primarily on anisotropic distances to drillholes with 50 m grid drill spacing being targeted. However, additional adjustments have been made to ensure a cohesive shape of Indicated material is produced. All blocks not classed as Indicated are classed as Inferred. Table 14-10 summarizes the classification parameters.

Table 14-10: Summary of Requirements for Classification to Indicated

Criteria		Distance (m)
1	Average Distance to closest 2 DHs	35
	Max. distance to closest 2 DH	50
	Minimum # Quadrants	2
2	Average Distance to closest 3 DHs	50
	Max. distance to closest 3 DH	70
	Minimum # Quadrants	2
3	Distance to closest DH	10
	Max. distance used	50
	Minimum # Drillholes used	3

14.10 Cut-off Grade and Reasonable Prospects of Eventual Economic Extraction

The base case cut-off grade is 4 g/t Au, based on the following economic assumptions and preliminary production rate estimates:

- Metal prices of US$1,750/oz Au and Forex of US$0.75=C$1.00
- Payable metal of 99% Au
- Off-site costs (refining, transport and insurance) of US$7/oz
- Mining cost of C$82.78/t, Processing costs of C$105.00/t and G&A and site costs of C$66.00/t
- Metallurgical Au recovery of 90.5%
- NSR (C$/t)=Au*90.5%*US$74.72 g/t

The "reasonable prospects of eventual economic extraction" has been further confirmed by:

- Ensuring all Resources are within modeled domains having minimum 2 m true thickness.
- Creating a resource a gradeshell for each reported cut-off and removing any outlying blocks too small to be considered “reasonably minable.”

14.11 Block Model Validation

The model has been validated by comparison of the Au grade and tonnage above cut-offs with the de-clustered grades and total metal. Further validations have been done by visually comparing the modeled grades to the assay data, as presented below.

14.11.1 Comparison of Modeled Grades to De-clustered Composites

To validate the block model, a Nearest Neighbour (NN) model has been created to compare the de-clustered composite data to the interpolated grades. Table 14-11 compares the relative metal content across grade-bins of Indicated Resources. Decreasing relative metal content as the grade increases confirms that interpolated grades are conservative compared to de-clustered composite values.

Although Domains 8 and 9 do not have a good comparison between modeled and NN model grades, they are very small domains that contributes little to the overall Resource. It is also established that Domain 9 contains assays at the peripheries of the solids, but still within the domains that are below cut-off, contributing to the values of lower de-clustered composite (NN) grades.

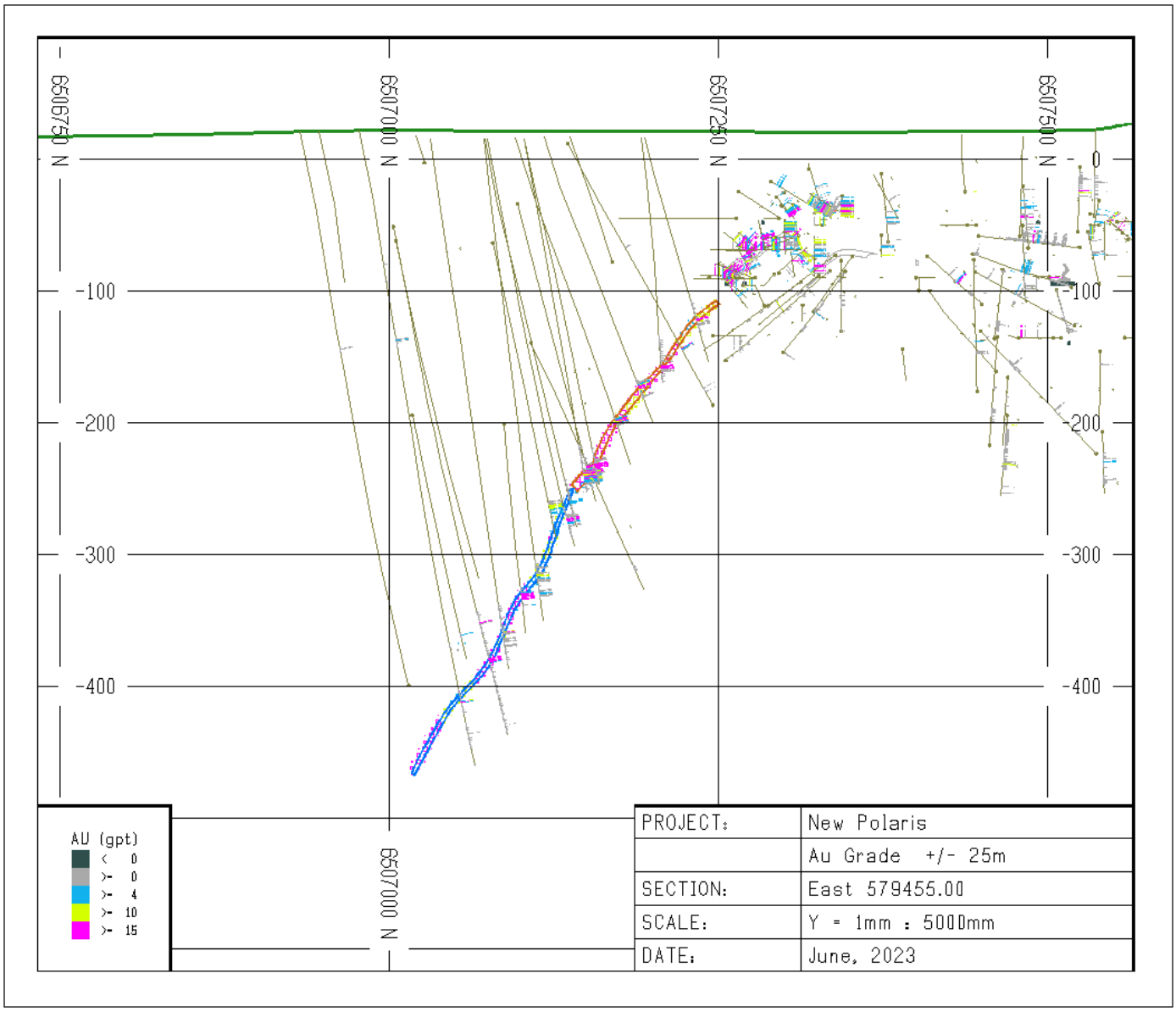
Table 14-11: Block Model and De-clustered Composite Comparison, Au, Indicated Resources

Domain	ID2/NN Contained Metal, Indicated Blocks only					
	Cut-off, g/t					
	1	4	6	9	15	% Total at 4 g/t Cut-off
2	92%	92%	92%	90%	68%	25.9%
3	96%	98%	98%	92%	70%	10.5%
4	96%	96%	98%	96%	77%	8.5%
5	96%	97%	94%	90%	85%	13.3%
6	97%	96%	93%	96%	90%	17.6%
8	95%	95%	93%	114%	106%	1.9%
9	131%	134%	113%	0%	0%	0.2%
10	96%	95%	84%	97%	0%	3.7%
11	91%	87%	54%	0%	0%	1.2%
15	76%	68%	58%	41%	0%	0.4%
16	90%	91%	91%	74%	59%	4.0%
19	102%	102%	100%	99%	0%	1.4%
21	84%	83%	82%	82%	45%	5.1%
22	95%	95%	95%	99%	69%	4.5%
23	78%	77%	70%	38%	6%	0.4%
24	72%	69%	86%	39%	29%	0.3%
26	90%	86%	91%	71%	0%	1.1%
All Domains	**94%**	**94%**	**93%**	**91%**	**72%**	**100%**

14.12 Visual Validation

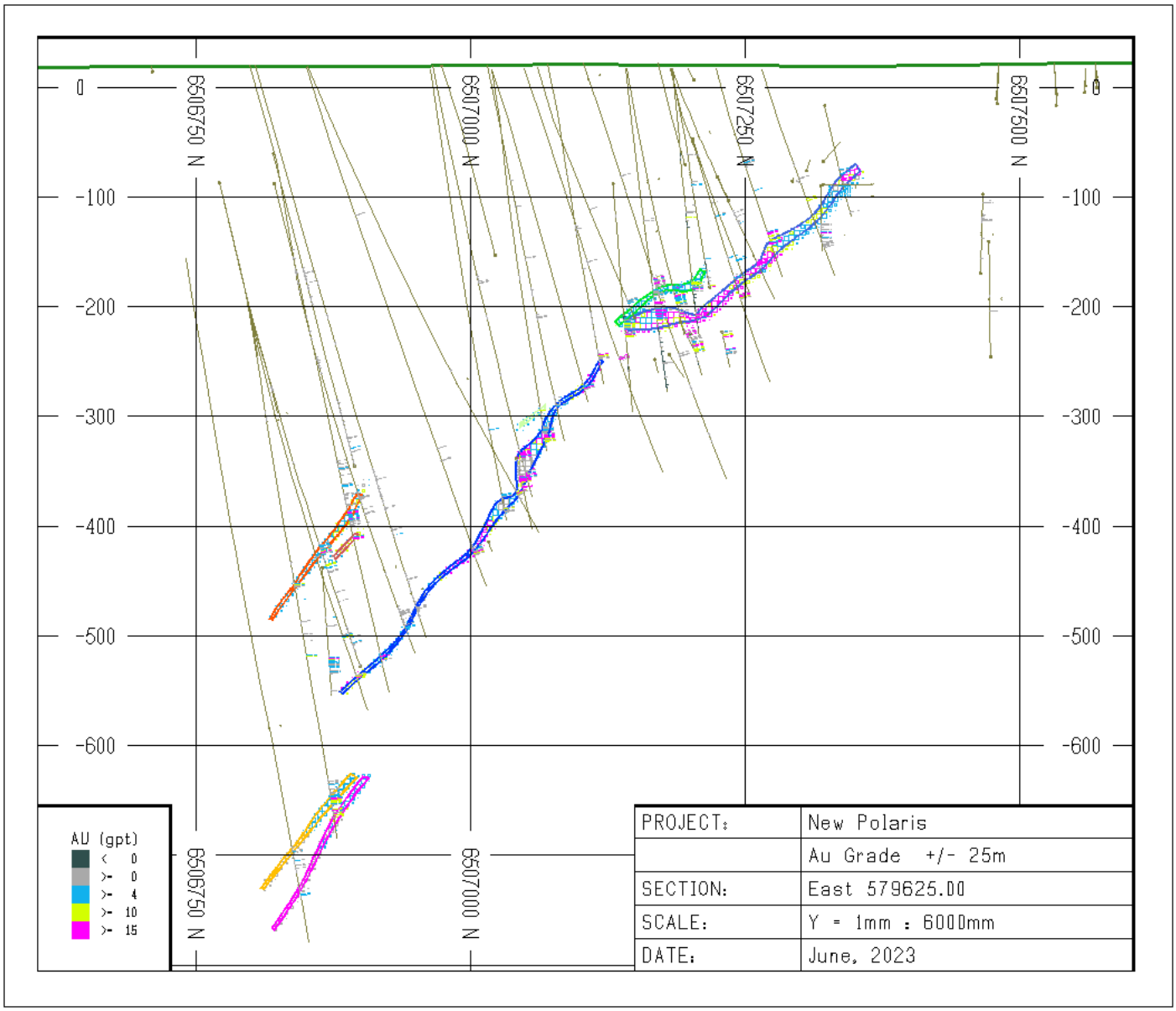
The modelled Au grades have been compared to the assay grades in section and plan to ensure the model matches the data. Examples of the sections are given in Figure 14-6 and Figure 14-7 for the "C" veins and in Figure 14-8 for the "Y" veins. The assay data has been projected ±25 m from the section. In the section plots, the block size is scaled according to the percent of the block coded to be within the domain to the percent of the block coded to be within the domain.

Figure 14-6: Comparison of Assay Au Grades and OK Block Grades – Section 579455E



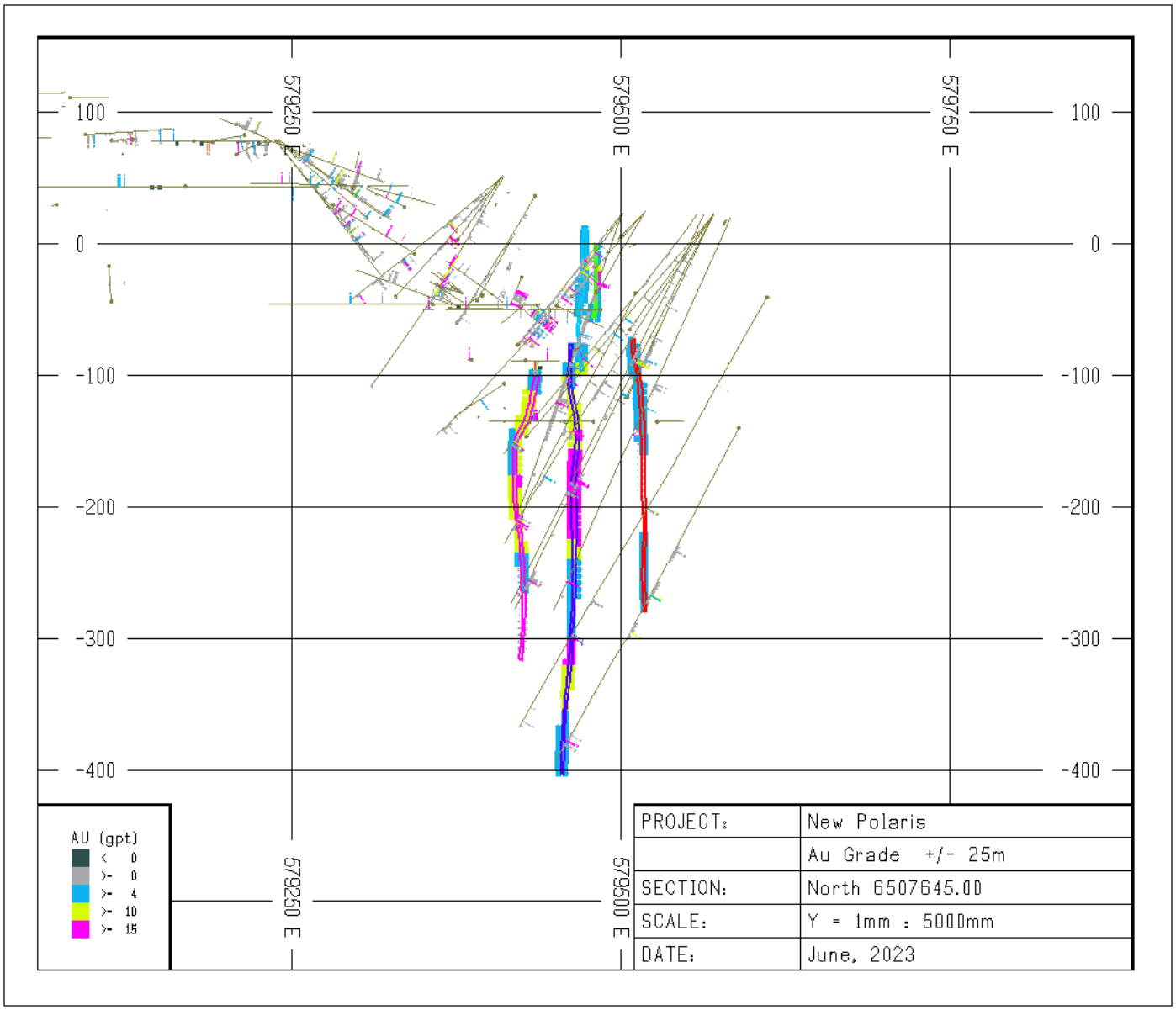
Source: MMTS, 2023.

Figure 14-7: Comparison of Assay Au Grades and OK Block Grades – Section 579625E



Source: MMTS, 2023.

Figure 14-8: Comparison of Assay Au Grades and OK block grades – Section 6507645N



Source: MMTS, 2023.

14.13 Independent Checks

An independent check on the modeling has been done by George Dermer, P.Eng. of MMTS who checked:

- Resource shapes

- Model coding
- “Reasonable prospect” shapes and inputs
- Interpolation runs
- Nearest Neighbour Validations

Table 14-12: Risk Assessment

No.	Description	Justification/Mitigation
1	Classification Criteria	Based on variography.
2	Geologic Model	Geologic interpretations and orientations of previous underground working considered when creating new geologic confining shapes for the resource interpolations. Faults used to define changes in orientations/offsets.
3	Metal Price Assumptions	Cut-off is based on US$1,750/oz Au, which is below the current prices and based on three-year trailing average.
4	High-Grade Outliers	Capping and outlier restriction applied to ensure modeled mean grade matches data. Grade-tonnage curves show modeled metal validates well with de-clustered composite data throughout the grade distribution.
5	Processing and Mining Costs	Assumed from comparable and based on mining method.
6	Previous underground mining	Underground workings examined during site visit. 2021 drilling did not hit unexpected voids.

15 MINERAL RESERVE ESTIMATES

15.1 Introduction

A Mineral Reserve is the economically mineable part of a Measured and/or Indicated Mineral Resource. It includes diluting materials and allowances for losses, which may occur when the material is mined or extracted and is defined by demonstrated studies at a Pre-Feasibility or Feasibility level as appropriate that includes application of Modifying Factors. This Feasibility Study includes adequate information and considerations on mining, processing, metallurgical, infrastructure, economic, marketing, environmental, and other relevant factors that demonstrate, at the time of reporting, that economic extraction could be reasonably justified.

Mineral Reserves are those parts of Mineral Resources, which, after the application of all mining factors, result in an estimated tonnage, and grade which, in the opinion of the Qualified Person(s) making the estimates, is the basis of an economically viable project after taking account all relevant Modifying Factors. Mineral Reserves are inclusive of diluting material that will be mined in conjunction with the Mineral Reserves and delivered to the treatment plant or equivalent facility. The term “Mineral Reserve” need not necessarily signify that extraction facilities are in place or operative or that all governmental approvals have been received. It does signify that there are reasonable expectations of such approvals.

Mineral Reserves are subdivided in order of increasing confidence into Probable Mineral Reserves and Proven Mineral Reserves. A Probable Mineral Reserve has a lower level of confidence than a Proven Mineral Reserve.

Mineral Reserves are reported in accordance with National Instrument (NI 43-101) “Standards of Disclosure for Mineral Projects” . The Mineral Reserves estimate follows the Canadian Institute of Mining, Metallurgy and Petroleum (CIM) 2014 “CIM Definition Standards for Mineral Resources & Mineral Reserves” (CIM, 2014) and the “CIM Estimation of Mineral Resources & Mineral Reserves Best Practice Guidelines” (CIM, 2019). The independent QP who takes responsibility for the Mineral Reserves estimate is Dino Pilotto, P.Eng. of JDS.

The Qualified Person (QP) has not identified any risk including legal, political, or environmental that would materially affect potential Mineral Reserves development.

15.2 Mineral Reserves Summary

All designs and scheduling have been completed to a level of detail appropriate for a feasibility study using a Mineral Resource model generated by Moose Mountain Technical Services on January 22, 2025. All proposed mining at the New Polaris site will be by underground methods.

Only Measured and Indicated Mineral Resources have been used for Mineral Reserves estimation. The breakdown of Mineral Reserves comprises of approximately 60% Drift and Fill and the remaining 40% of the tonnage being mined through LHS methods.

The estimated Proven and Probable Mineral Reserves for the envisaged New Polaris underground mine as of July 10, 2025 are summarized in Table 15-1.

Table 15-1: Mineral Reserves Summary as of July 10, 2025

Classification	Tonnage (kt)	Gold Grade (g/t)	Gold Metal (koz Au)	Antimony Grade (ppm)	Arsenic Grade (ppm)
Proven	-	-	-	-	-
Probable	2,830	9.94	904	1,828	10,960
Total	**2,830**	**9.94**	**904**	**1,828**	**10,960**

Source: JDS, 2025.* Metal conversion factors reflect total metal content. Notes:

1. Mineral Reserves are derived from Measured and Indicated Mineral Resources.
2. Figures in the tables may not compute exactly due to rounding.
3. Mineral Reserves are based on a cut-off grade of:
   a. 6.0 g/t Au for drift and fill stopes.
   b. 5.5 g/t Au for longhole stopes.
   c. 4.2 g/t Au marginal cut-off value for development ore.
   d. Cut-off grades are based on a gold price of US$2,245/oz.
4. Assumed exchange rate is C$1.39 = US$1.00.
5. Metallurgical recovery assumed to be 89.75%; payable assumed to be 90%.
6. Mineral Reserves are reported on a 100% ownership basis; Canagold owns 100% of the New Polaris property.

15.3 Mining Method

The New Polaris Mine was designed around the use of the drift and fill (DAF) and Longhole Stoping (LHS) mining methods. DAF was selected for the purpose of selectivity with a view to minimizing dilution in shallow dipping zones and areas with noted less than favourable ground conditions, LHS was selected as a less costly mining method with higher productivity rates.

Refer to Section 16 for additional details on mining method selection.

15.4 Input Parameters

15.4.1 Cut-off Grade

The cut-off grade (COG) is generally the minimum grade estimated for mineralized rock to be economically mineable. It is a key factor in mine design but is not the sole determinant of economic viability. Mineralized rock with a metal grade above the cut-off grade may not be economically mineable if its occurrence is such that the associated development and other costs cannot be amortized by the overall margin generated by processing and refining the contained metal. For example, the mineralized rock may be too remote and/or occur in small “pockets” that are not profitable to access. Conversely, mineralized rock with a grade that is below the cut-off value may be mined and processed if the mineralized rock is mined as a consequence of the overall mine plan. An example is mineralized development rock that must be mined to access other areas of the mine.

15.4.2 Mine Cut-off Grade Criteria

The gold COG was calculated using the parameters listed in Table 15-2 and assuming a nominal 1,000 t/d steady-state production rate. The break-even COG was derived by estimating the total cost of production divided by the economic value of recoverable gold. To allow for fluctuations in the COG input variables, the calculated value of 5.7 g/t Au was increased to 6.0 g/t Au for primary design of the DAF stopes and 5.5 g/t Au for the longhole stopes.

Table 15-2: Inputs for Break-Even Gold Cut-off Value Calculation

Items	Units	Value
Currency Exchange Rate	C$=US$1	1.39
Gold Price	US$/oz	2,245
Gold Recovery	%	89.75
Gold Payable	%	90
Mining Cost Drift and Fill (DAF)	C$/t	125.00
Mining Cost Longhole Stoping (LHS)	C$/t	85.00
Concentrate Transport	C$/t	120.00
Mine General and Tech Services	C$/t	45.00
Processing Cost	C$/t	88.00
General & Administrative Cost	C$/t	67.00
Sustaining Capital	C$/t	13.50
Calculated DAF COV	Au g/t	5.7
Calculated LHS COV	Au g/t	5.2
Calculated Development COV	Au g/t	4.2
Design DAF COV	Au g/t	6.0
Design LHS COV	Au g/t	5.5

15.4.3 Marginal Development Cut-off Grade

For mineralized rock that must be mined to access the various areas of the mine (e.g. development to access stopes), a marginal COG was utilized to determine whether it should be considered for inclusion into the Mineral Reserve estimate. The marginal COG of 4.2 Au g/t was derived using the inputs provided in Table 15-2 assuming no (zero) mining cost, given the material must be mined for the mine plan sequence to progress as designed and scheduled.

15.4.4 Dilution and Recovery

Unplanned dilution and mining recovery allowances have been applied to the Mineral Reserve estimate for all mining methods, the details of which are presented in Table 15-3. These allowances are considered reasonable given the mining methods employed.

Equivalent Linear Overbreak Sloughing (ELOS) dilution for DAF was estimated at 0.25 m for the hanging wall and footwall for a total overbreak of 0.5 m. Longhole stoping estimated 0.5 m on the hanging wall and footwall for a total overbreak of 1.0 m. In addition, mucking dilution (0.25 m) and backfill dilution from adjacent LHS of (0.5 m) is added. Further detail on ELOS is discussed in Section 16.3.

Ore losses (mining recovery factor) are related to the practicalities of extracting ore under varying conditions, including difficult mining geometry, rock mass conditions, losses of ore into backfill, and blasting issues.

Table 15-3: Dilution and Recovery Factors

Mining Method	Unplanned Dilution (%)	Recovery (%)
Drift and Fill	21	97
Development	21	100
Longhole stoping (blind upper)	31	90
Longhole stoping (top and bottom access)	31	95

15.4.5 Mineral Reserves Estimate

The New Polaris Mineral Reserves, in total and broken down by zone, are presented in Table 15-4.

Table 15-4: Mineral Reserves by Mining Area as of July 10, 2025

Classification	Mine Area	Tonnage (kt)	Gold Grade (g/t)	Gold Metal (koz)*	Antimony Grade (ppm)	Arsenic Grade (ppm)
Proven	Zone C	-	-	-	-	-
	Zone Y	-	-	-	-	-
	Total	**-**	**-**	**-**	**-**	**-**
Probable	Zone C	2,525	9.98	810	2,049	10,920
	Zone Y	306	9.63	95	0	11,291
	Total	**2,830**	**9.94**	**904**	**1,828**	**10,960**
Proven and Probable	Zone C	2,525	9.98	810	2,049	10,920
	Zone Y	306	9.63	95	0	11,291
	Total	**2,830**	**9.94**	**904**	**1,828**	**10,960**

Source: JDS, 2025. Notes:

1. Mineral Reserves are derived from Measured and Indicated Mineral Resources.
2. Figures in the tables may not compute exactly due to rounding.
3. Mineral Reserves are based on a cut-off grade of:
   a. 6.0 g/t Au for drift and fill stopes
   b. 5.5 g/t Au for longhole stopes
   c. 4.2 g/t Au marginal cut-off value for development ore.
   d. Cut-off grades are based on a gold price of US$2,245/oz.
4. Assumed exchange rate is C$1.39 = US$1.00.
5. Metallurgical recovery assumed to be 89.75%; payable assumed to be 90%.
6. Mineral Reserves are reported on a 100% ownership basis; Canagold owns 100% of the New Polaris property.

16 MINING METHODS

16.1 Introduction

The New Polaris Mine located in northwestern British Columbia was historically mined using underground methods from 1938 to 1942 and from 1946 to 1951 producing 740,000 tonnes of ore at an average grade of 10.3 g/t gold. With the exception of the rehabilitation and reuse of the existing portal and ramp/adit, the proposed underground mine design largely avoids the existing underground workings and focuses on ore body extensions and more recently defined zones.

Mining at the New Polaris mine will be performed via underground methods by way of a ramp from the surface with ore extracted from two zones (Zone C and Zone Y). Underground access through the existing Polaris Portal, will be slashed-out to 5.5 m wide (W) by 5.5 m high (H) to accommodate the underground mobile equipment as a single decline from the top of the Y zone two ramps will be developed, one following Zone C and the other following Zone Y.

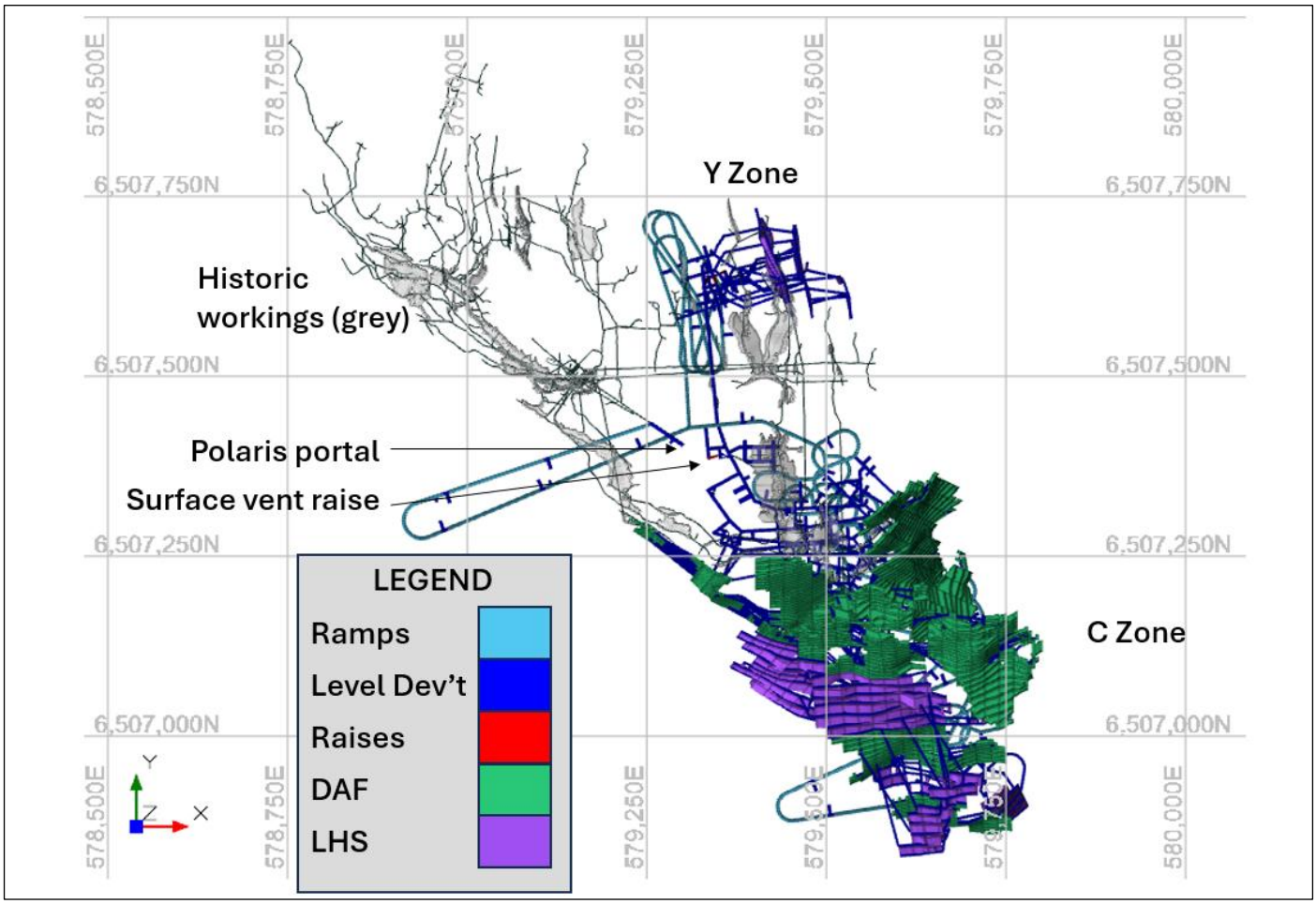
Mineralized zones will be extracted using a combination of Drift and Fill stoping and Longhole Stoping, with Drift and Fill stoping accounting for 60% of the mined reserve. Zone Y will be extracted exclusively using Longhole Stoping. Stopes will be backfilled using waste rock generated from development as well as a structural paste fill where required.

The mine will be ventilated by a series of Alimak raises, raise bore and drop raise vertical development, with the declines being utilized as fresh air. Dewatering of the mine will be accomplished by a network of connected sumps, settling sumps and pump stations. See Figure 16-1 and Figure 16-2 for plan and isometric views.

16.1.1 Deposit Characteristics

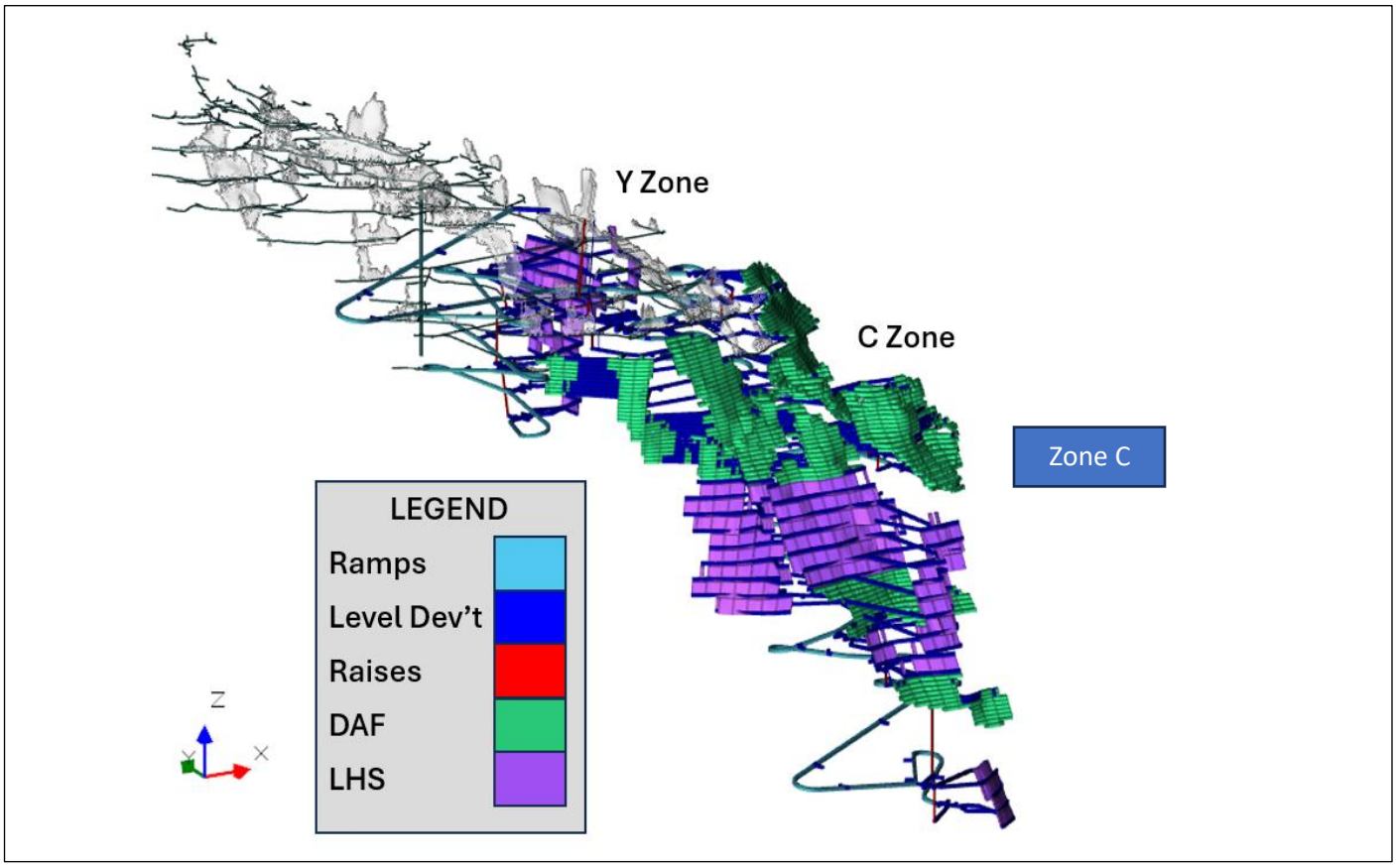
Zone C is the larger of the two mining areas. The proposed minining extends approximately 500 m horizontally in the east-west direction and has a vertical height of 520 m (-580 to -60 RL). Mineralization dips at roughly 50° to 60°. Sub-levels are spaced at 20 m vertical intervals with DAF mining lift heights of 5 m (see Figure 16-3). Zone Y extends approximately 150 m horizontally and has a vertical extent of 160 m (-260 to -100 RL). The zone dips at roughly 80 to 90°. Sub-levels are spaced at 20 m vertical intervals (see Figure 16-4).

Figure 16-1: Composite Plan View of the New Polaris Underground Mine



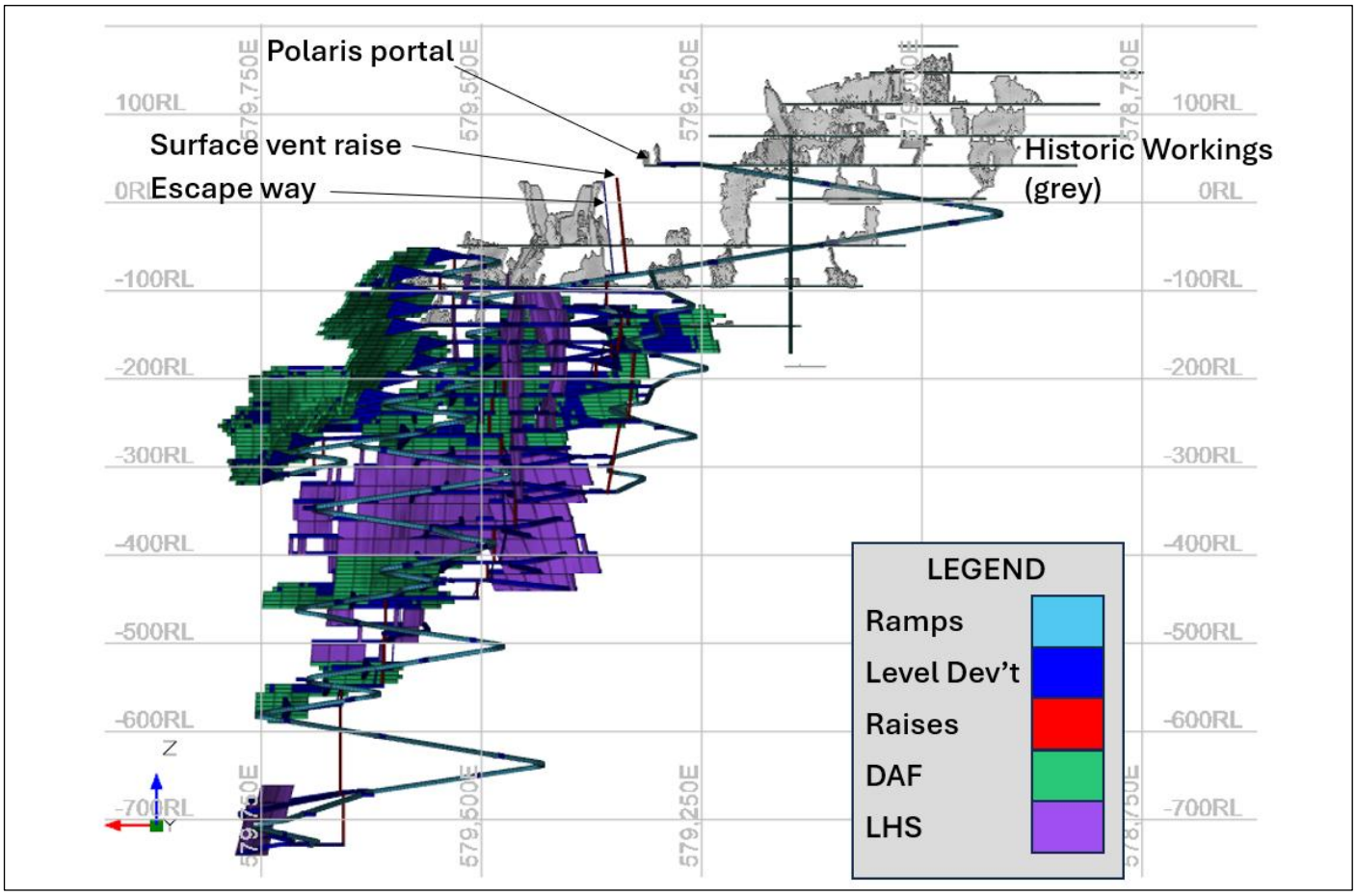
Source: JDS, 2025.

Figure 16-2: Isometric view of the New Polaris Underground Mine



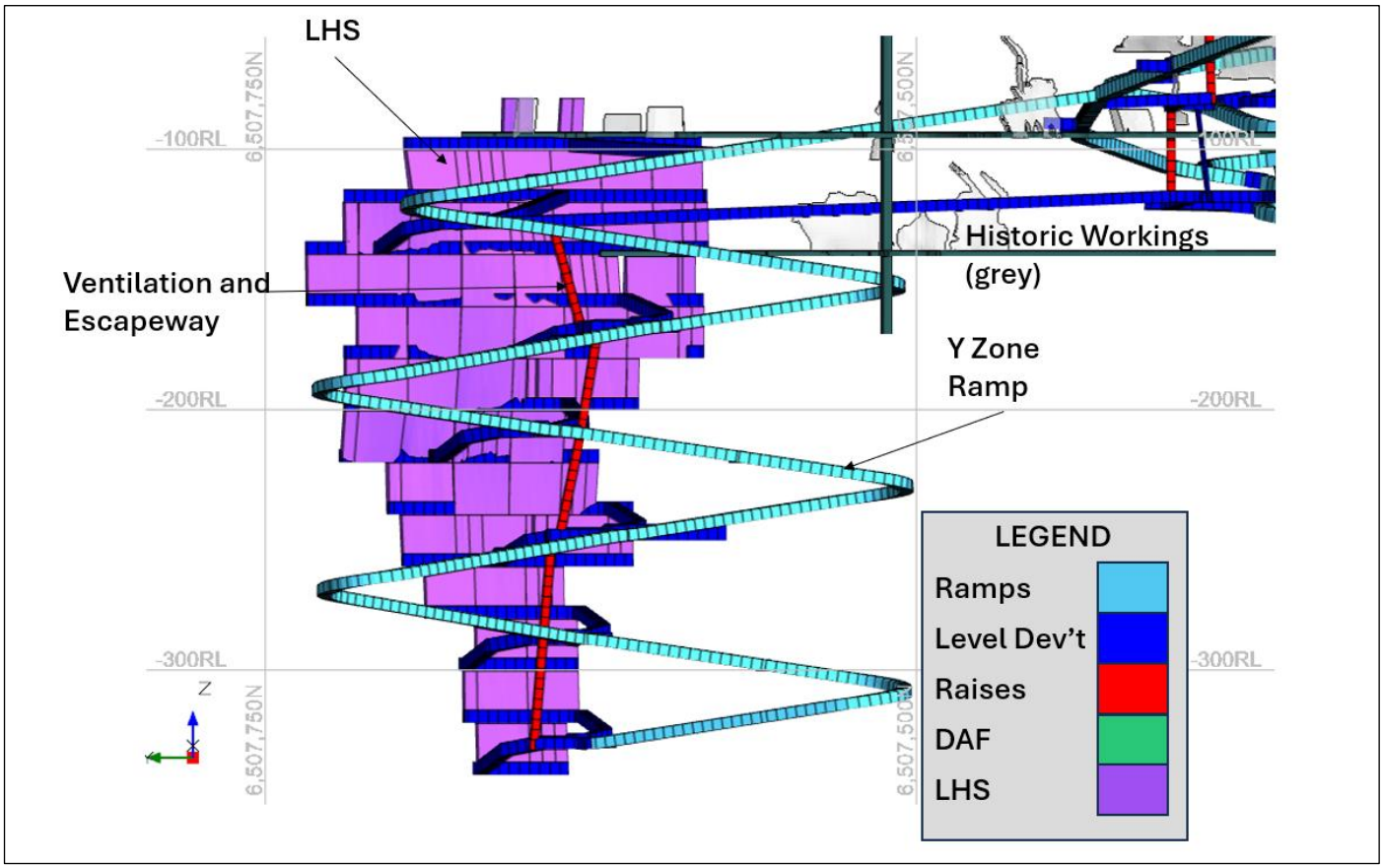
Source: JDS, 2025.

Figure 16-3: Section View of Zone C Mine Design (Looking North)



Source: JDS, 2025.

Figure 16-4: Long Section View of the Zone Y Mine Design (Looking West)



Source: JDS, 2025.

16.2 Geotechnical Analysis

16.2.1 Geotechnical Data Sources & Limitations

16.2.1.1 Geotechnical Core Logging Data

A geotechnical assessment was completed based on existing information which included historic geotechnical logging data and laboratory testing data obtained from the 2006, 2021 and 2022 drilling campaigns. The original geotechnical logging data was collected by Canagold geologists during the respective resource core logging programs and then reviewed for quality by geologists from Sun Valley Investments AG (Sun Valley) in 2025. Core photos are available for the 2006, 2021 and 2022 drillholes and were used to validate existing geotechnical logging data.

JDS conducted a thorough review of the geotechnical logging database from Sun Valley for accuracy based on core photo review and for internal consistency. In some cases, the data was found to be inconsistent internally and/or did not represent rock quality accurately, compared to conditions observed in the respective core photos.

Rock mass quality from the 2006 core logging data (typically 'Very Poor' to 'Fair' quality) was significantly lower compared to the 2021/2022 core logging (typically 'Good' to 'Very Good'). Although the majority of the 2006 drillholes were drilled in the upper portions of the C veins where the rock near the veins is commonly more weathered/altered and fractured, data for some zones was still considered unrepresentative and had to be adjusted to more accurately reflect ground conditions.

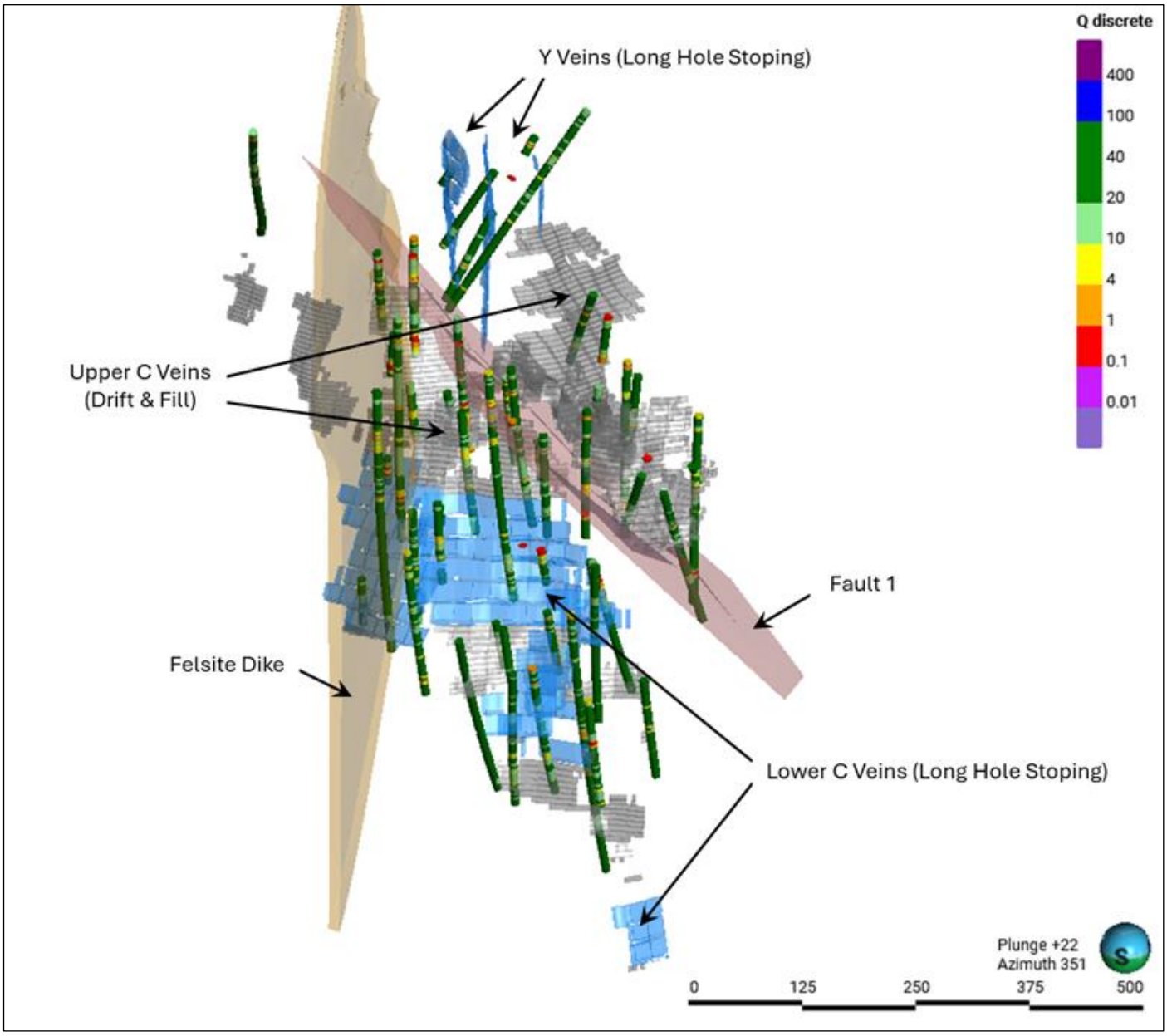
For these zones, basic logging parameters such as RQD based on core photos and joint conditions were adjusted based on other core intervals as well as observations made from underground exposures in the historic mining area. Core photos were also used to provide estimates of rock quality in key zones such as the hanging wall and vein for additional holes not previously logged. Good quality photos of unsplit core were available for nearly all holes; however, the core had been split for assay or the photos were poor quality for a few of the 2006 drillholes which may have resulted in more conservative estimates of rock quality.

A combined total of 6,928 m of core between 48 holes was assessed according to the Barton Q rock mass classification criteria (Barton & Grimstad, 1994) for the development of the final geotechnical logging database. The holes selected provide good spatial distribution across the veins except for the outside extents of the upper C veins where no recent drillhole exist with core photos. As a result, these areas were conservatively assumed to have lower rock quality and are currently planned to be mined with Drift and Fill mining.

The location of the holes used in the geotechnical assessment are shown relative to the stoping areas on Figure 16-5. The revised database is generally consistent with observations made underground in the historic mining area. JDS considers the final database suitable for a feasibility level study.

Oriented core data is available for most of the 2021–2022 drillholes; however, most of the data was focused on vein orientations with only 7% of the data set representing natural joints and faults/shears. The overall trends of the oriented available for joints and faults/shears are generally consistent with underground observations.

Figure 16-5: Isometric view of Location of Drillholes Used for Geotechnical Assessment



Source: JDS, 2025.

16.2.1.2 Laboratory Strength Testing

Laboratory strength testing was carried out on samples of core selected from the 2021-2022 drill program which included the Y veins and the lower sections of the C veins. The samples were selected by AMC (2024) to represent the

range of conditions anticipated for vein hanging walls and footwalls in the Mafic Tuff. No whole core was available for testing from earlier drilling campaigns or other rock types.

The amount of laboratory test results available is considered low for a feasibility study; however, the results are considered reasonable estimates based on published values associated with similar rock types. All test results, including samples that were may have been affected by pre-existing defects, were considered for the analyses which may result in conservative estimates of rock strength.

16.2.1.3 Geologic and Structural Models

A 3D geologic model exists for the New Polaris area based on lithology logging contained in the resource drillhole database which includes the historic (1988 through 1997) and most recent (2021-2022) drilling campaigns. Lithology logging information is not available for drillholes completed during the 2003 through 2006 programs.

Lithology logged during the historic core (1988 through 1997) consists mostly of Basalt and Pyroclastics in the historic mine area transitioning into mostly Tuff with lesser Basalt in the lower part of the deposit. The 2021-2022 drilling campaign was logged almost entirely as Mafic Tuff even where holes are within historic drillholes logged as Basalt and Tuff suggesting different lithology nomenclature was presumably used between the historic and recent drilling campaigns.

While the 3D lithologic model differentiates between Mafic Tuff, Tuff and Basalt, it was concluded that the Tuff and Mafic Tuff can be the same rock types logged with different nomenclature based on interpretations at that time. Distinct differences between these lithologies were also not evident in core photographs. There is also no surface geologic mapping of the New Polaris area to corroborate the rock types as the deposit is covered with a thick layer of alluvium without any outcropping. As such, the 3D lithologic model was not used to domain rock properties and, with the exception of the Felsite Dike, the host rock has been assumed to be a single unit comprised of Tuff/Mafic Tuff.

16.2.1.4 Mapping

A significant amount of geology mapping exists within the historic mining area, and relevant information was considered in the geotechnical assessment of the New Polaris area. Geology and ground conditions in the historic AJ level workings were also reviewed during a site visit as described in JDS (2025). There is no surface geologic mapping of the New Polaris area as the deposit is covered with a thick layer of alluvium without any outcropping.

16.2.2 Geologic and Structural Models

The following geologic description is based on MMTS (2023) and historical information published on the site and other nearby mineral deposits.

The lower portion of the deposit is composed predominantly of basaltic to andesitic volcaniclastic units. These units strike northwest to north and dip steeply towards southeast. Mineralization occurs along three major vein sets:

- AB shear veins – trending northwest-southeast dipping steeply to the southwest;

- Y extension veins – subvertical, north-northwest trending; and
- C shear veins – trending east-west to northwest-southeast with shallow to moderate dips toward south to southwest. The C veins are curve shaped, linking the AB with Y veins.

The various veins are similar, consisting of white quartz and carbonate veins with lenses and fragments of the host rock. Width of mineralization ranges from 0.3 to 15 m in thickness, but tends to average about 3 m. The feasibility study mine plan includes primarily C veins and a few Y veins.

The only known major fault structure within the New Polaris area is the No.1 Fault which strikes northwest to southeast and dips 45 to 50° northeast. The fault separates the C veins into two zones with differing strikes. To the west of the fault, veins No. 02 - 04, 08 - 11, 15 and 19 dip more south-southwest while veins No. 05, 06, 16 east of the fault dip more south-southeast. The No.1 Fault has reverse displacement of up to 30 m.

Based on core logging data, the fault zone ranges from a few centimeters to a couple of meters wide. No significant groundwater seepage was observed coming from the fault, where observed on AJ level during the site visit. The No.1 fault is not expected to have an adverse impact on stopes or mine scale stability but may affect local ground conditions were intersected in development.

Canagold and MMTS indicate there are no other large-scale faults have been observed in core or are known to exist within the New Polaris area. This is consistent with JDS observations made in the section of the AJ level during the site visit as well as observations from the core photographs and core logging data.

A north-northwest striking, steeply west-southwest dipping Felsite dike occurs near the western edge of the mine area. The dike is approximately 5 to 15 m wide and intersects only Vein No. 3. The location of the dike and Fault No. 1 are shown in Figure 16-5 relative to the planned stoping areas.

A thick layer of overburden exists above bedrock in the mine area varying from approximately 30 to 50 m above the Y vein area and 40 to 175 m of overburden above the C veins. The overburden likely consists of alluvial sand, gravel and cobbles deposited by the Taku River.

The upper few meters of rock below the overburden contact are typically slightly weathered and more fractured but will not impact mining except where mine infrastructure will daylight.

16.2.3 Geotechnical Domains and Rock Mass Quality

The finalized geotechnical core logging database was evaluated according to the host rock (Tuff/Mafic Tuff), mineralized veins and Felsite Dike rock types. The immediate vein hanging wall and footwall zones (within approximately 10 m from the mineralization contact) were also analyzed separately.

Results of the analysis indicate geotechnical characteristics within the host rock are generally consistent, except where intersected by the No. 1 Fault and mineralized veins. The veins are typically faulted within the vein itself occasionally in the immediate hanging wall. The fault zones vary from a few centimeters to a couple meters in thickness.

The average values and statistical variability of rock mass quality parameter (Q') derived from the analysis are summarized in Table 16-1 according to the hanging wall (HW), footwall (FW) and mineralized vein domains. Q' is the modified tunnelling quality index, which assumes the joint water reduction factor (Jw) and stress reduction factor (SRF) are equal to 1 (dry and medium stress conditions). The Q' values presented in Table 16-1 are anticipated to be representative of the overall rock mass behavior.

Table 16-1: Variability in Rock Mass Quality (Q') per Domain/Zone

Drilling Campaign (Domain)	Zone	Avg*	Standard Deviation*	Min	20th %*	Median	80th %*	Max
2006 (Upper C Veins)	HW	11	4.1	0.2	3.3	15	35	63
	Vein	9	3.6	0.2	3.1	10	24	77
	FW	17	3.1	0.6	8.5	18	44	98
2021 – 2022 (Lower C & Y Veins)	HW	24	2.9	0.2	14	27	59	95
	Vein	19	2.4	0.4	11	21	32	67
	FW	30	2.1	2.5	14	31	63	100
2006 and 2021 – 2022	No. 1 Fault	0.7	0.3	0.1	0.2	0.4	0.7	2.8
	Other Faults	0.7	0.4	0.2	0.2	0.5	1.1	2.9
	Felsite Dike	2.0	0.5	0.8	4.8	21	40	100

* Calculated values are weighted by interval length. Source: JDS, 2025.

16.2.3.1 Host Rock (Mafic Tuff/Tuff)

The host rock mass away from the mineralized veins is generally 'Good' to 'Very Good' quality according to the Barton Q rock mass classification criteria. Local zones of 'Poor' to 'Fair' rock quality exist where intersected by the No. 1 Fault.

16.2.3.2 Hanging Wall Zone

The rock mass in the immediate hanging wall of the upper, near surface portion of the C veins typically classifies as 'Fair to 'Good' quality with relatively narrow, localized zones of 'Poor' quality rock where faults exist. Hanging wall rock above the deeper portion of the C veins classifies as 'Good' to 'Very Good' quality. Based on analysis of the core logging database, the boundary between the upper and lower portions of the C veins is estimated to be at approximate elevation -280 m.

16.2.3.3 Mineralized Veins

The 'vein' zones as described by MMTS (2023) are structurally controlled shear zones and are typified by silicification and carbonatization cross cutting actual quartz-carbonate veins. These zones have sharp contacts with the wall rock and form anastomosing ribbons and dilations. These zones have been deformed several times. The zones are generally tabular in geometry forming en-echelon sheets within the more competent host lithologies.

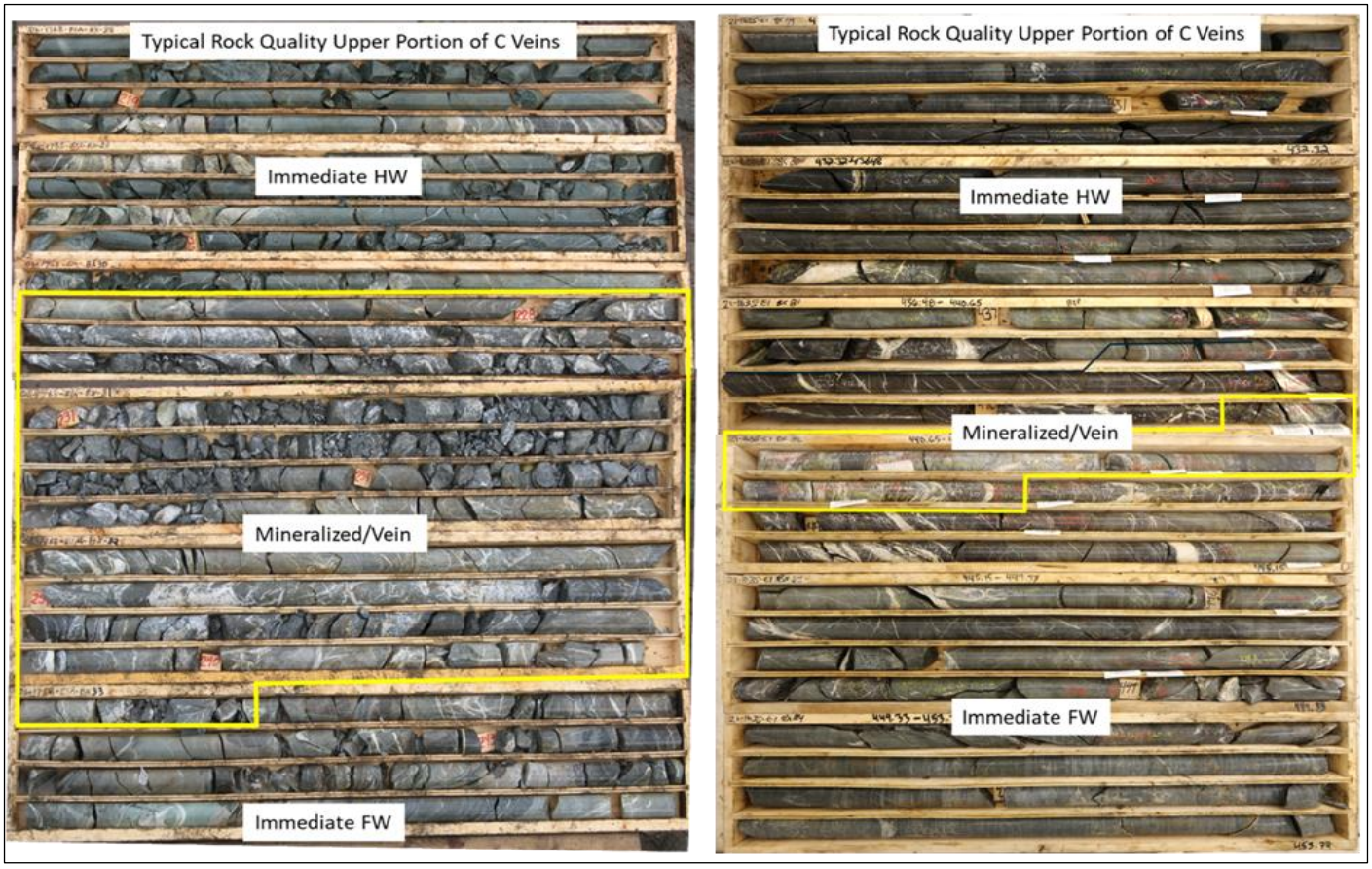
Overall, the vein rock mass quality is similar to the hanging wall but with a slightly higher density of fractures in some areas. The veins typically classify as 'Fair to 'Good' quality in the upper C veins with relatively narrow, localized zones of 'Poor' quality rock faulted. The lower C and Y veins are primarily 'Good' to 'Very Good' rock.

16.2.3.4 Footwall Zone

Rock quality in the vein footwall is generally higher than the hanging wall and vein zones, classifying mostly as 'Good' to 'Very Good' rock mass quality. Faulting is uncommon in the footwall.

Core photographs representing typical hanging wall, vein and footwall rock mass quality are shown in Figure 16-6. On Figure 16-6, photos on the left-hand side demonstrate the upper portion of the C Veins where the rock quality in the veins and immediate hanging wall is weaker and more heavily fractured. Photos on the right-hand side on Figure 16-6 show typical conditions of the hanging wall, vein and footwall for lower portion of the C veins and the Y Veins where the rock is significantly less fractured and quite competent.

Figure 16-6: Typical Rock Quality in Upper C (Drift & Fill) and Lower C and Y Veins (Long Hole)



Source: JDS, 2025.

16.2.3.4.1 Felsite Dike

The subvertical, north-northwest striking Felsite dike is approximately 5 to 15 m wide and occurs near the western edge of the mine area. The dike intersects only No. 3 vein.

Ground conditions within the dike are variable. The first few meters of the hanging wall and footwall in the dike are typically more heavily fractured and oxidized consisting of 'Poor' to 'Fair' quality rock. The core occasionally has heavily fractured zones but is generally 'Fair' to 'Good' quality. Raises and other critical infrastructure should avoid intersections with the dike where possible. The location of the dike is shown in Figure 16-5.

16.2.3.4.2 Fault No. 1

Ground conditions within the No. 1 Fault zone are typically very poor with highly fractured and oxidized rock. Core logging data and core photographs indicate the fault zone ranges from a few centimeters to a couple meters wide. This is also consistent with observations made the No. 1 fault where it intersected underground drifts in the AJ Level. No significant groundwater seepage was observed coming from the fault.

The No.1 fault is not expected to have an adverse impact on stopes or mine scale stability but may affect local ground conditions where intersected in development. Raises and other critical infrastructure should avoid intersections with the No. 1 Fault whenever possible. The location of Fault No. 1 is shown in Figure 16-5.

16.2.3.5 Intact Rock Strength

Laboratory uniaxial compressive strength (UCS), Brazilian tensile strength (BTS), and triaxial compressive strength (TCS) testing was completed on select core samples of Mafic Tuff from the 2021 to 2022 drilling campaign as part of the AMC (2024) investigation. Results of the laboratory UCS and BTS tests are summarized in Table 16-2 with TCS results summarized in Table 16-3.

Table 16-2: Intact Rock Strength and Elastic Properties

Zone	Stats	Density (kg/m^3)	UCS (MPa)	Young's Modulus (GPa)	Tensile Strength (MPa)	Poisson's Ratio
C veins	Mean	2831	88.9	48.2	9.5	0.29
	STD	72.5	75.9	13.8	3.2	0.05
	Count	7	6	6	6	6
	Maximum	2875	216.9	68.8	13.8	0.42
	Minimum	2774	31.7	33.6	5.0	0.19
Y veins	Mean	2875	76.2	45.0	12.4	0.30
	STD	92.5	45.1	11.9	2.3	0.06
	Count	8	6	6	4	6
	Maximum	2968	115.9	61.3	14.5	0.36
	Minimum	2736	32.6	25.5	10.3	0.20
All	Mean	2855	82.6	46.6	10.7	0.29
	STD	72.5	59.9	12.4	3.1	0.07

Source: AMC, 2024.

Table 16-3: Summary of TCS Test Results

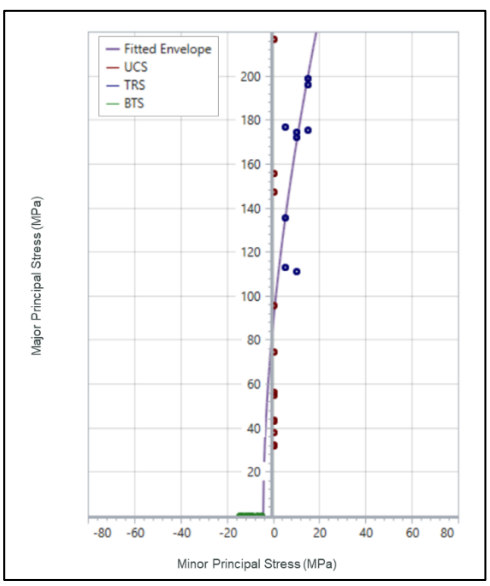
Zone	Drillhole ID	Depth (m)		σ_3(MPa)	σ_1(MPa)
		From	To		
C veins	22-1905E2	345.92	346.20	5	177.0
	21-1859E1	374.40	374.63	10	227.2
	21-1859E1	401.21	401.45	10	111.2
	21-1844E4	775.11	775.32	15	196.0
Y veins	P22Y06	198.00	198.34	5	113.2
	P22Y06	222.62	222.86	5	135.5
	P22Y04	220.46	220.80	10	174.6
	P22Y04	234.70	234.89	10	172.0
	P22Y25	440.72	440.98	15	198.9
	P22Y25	461.84	462.09	15	175.3

Source: AMC, 2024.

The laboratory test results and Hoek-Brown (Hoek, et al., 2002) strength envelope fitted to the data points are shown together in Figure 16-7. All of the test results were used which is considered conservative since a few of the tests broke on pre-existing defects however this is considered appropriate given the limited drill core available for testing was limited, based on the 2022 drillholes which were located Y veins or in the deeper section of the C Veins which generally showed higher rock quality near the veins than the 2006 drillholes. Test results are considered reasonable estimates based on published values associated with similar rock types.

Using the laboratory test results summarized above, the Hoek-Brown envelope fitted to the data indicates an intact rock strength (σ_{ci}) (MPa), defined as the intercept of the Hoek-Brown envelope with the Y axis, of 84.7 MPa with a Hoek-Brown material constant of 21.

Figure 16-7: Test Results and Hoek-Brown Intact Rock Strength Envelop



Source: AMC, 2024.

16.2.3.6 In-situ Stress

The magnitude and direction of in-situ stresses in the New Polaris area have not been directly measured. As such, stress conditions have been estimated based on regional geologic information and site topographic features. Based on a review of the World Stress Map (Heidbach, et al, 2016), the maximum horizontal stress may be oriented northeast-southwest, perpendicular to the vein strike. The World Stress Map maximum horizontal stress (σ_1) direction estimates nearest the New Polaris area were calculated from single focal mechanism (FMS) earthquake events and were rated as 'C' quality (±25° accuracy). The closest focal sources indicated a thrust faulting regime with σ_1 oriented approximately 230° azimuth.

It is assumed that the maximum horizontal stress magnitude is relatively low given the shallow (100 to 500 m) depth below ground surface of the planned stoping areas. No observations indicating high horizontal stress conditions were made underground during the visit on the AJ level or are known to have been made during historic mining.

It has been conservatively assumed that the ratio of maximum horizontal stress (σ_H) to vertical stress (σ_v) is approximately 1.5 to 2 with the minimum horizontal stress (sh) has been assumed to be approximately 1 to 1.5 times vertical stress (σ_v). The anticipated range of stresses has been considered in the calculation of stress parameter A for stope design as discussed below. Horizontal stresses may need to be assessed in more detail during operation.

16.2.3.7 Empirical Stope Design and Dilution Analysis for Long Hole Stopes

Empirical stope design analyses are based on stability graphs where the stability number (N') is plotted on the vertical axis against the hydraulic radius (wall area divided by wall perimeter) of the stope face on the horizontal axis. The stability number (N') is calculated based on the rock mass quality (Q'), and three empirical factors: A (induced stress conditions), B (geologic structure orientation) and C (dip angle of the stope face).

Maximum, non-entry stope dimensions that are stable without support were estimated using the Potvin (2001) method for the anticipated range of rock mass conditions and stope sizes. The Trueman & Mawdesley (2003) 'Stable' line was then used as a check against the Potvin (2001) results. Upper and lower bound estimates of stope face dimensions and rock quality (Q') were analyzed for the lower portion of the C veins and the Y veins. The weighted average and 20th percentile Q' values (Table 16-1) were used to represent the range in rock mass quality.

Empirical factors for the calculation of stability number (N') were based on the following:

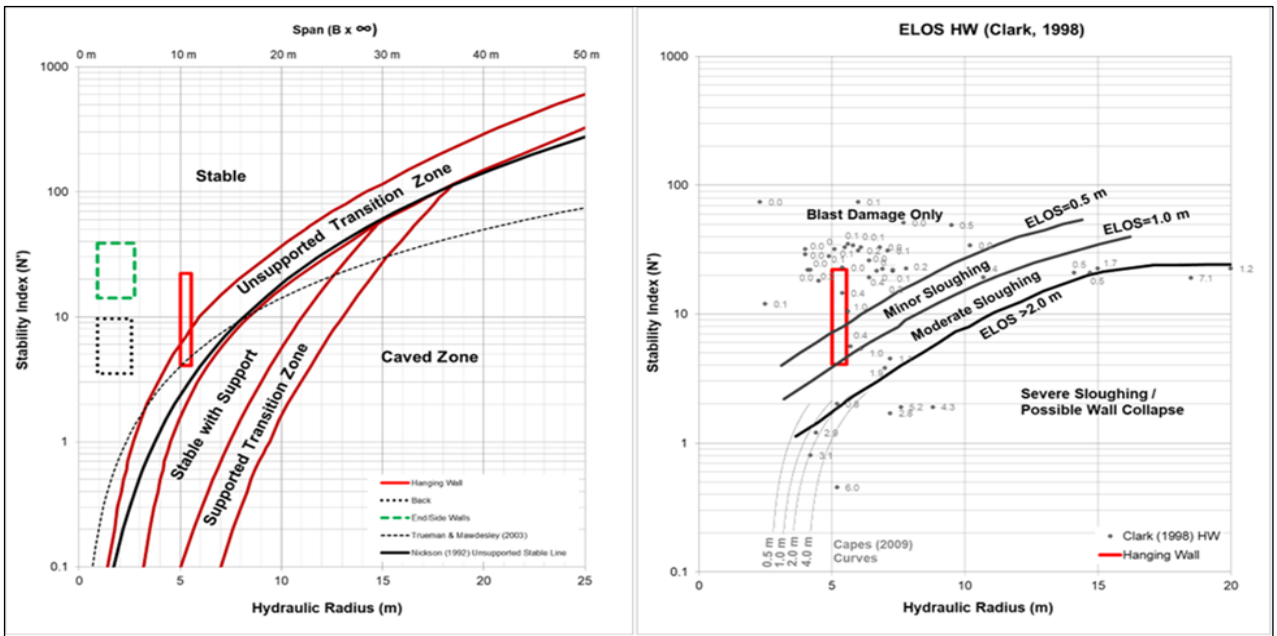
- Induced Stress Factor (A) based on the ratio of UCS and anticipated maximum horizontal stress estimated as a function of depth;
- Joint Orientation Factor (B) based on the orientation of the dominant joint sets relative to the stope wall orientation; and
- Gravity Factor (C) was calculated based on the dip angle for each stope wall using the equation: C =8 – 6 cos (face dip angle).

Maximum stable stope lengths were then determined from the stability graph based on a 20 m level spacing and the respective range in vein widths. The analyses indicate maximum stope lengths of approximately 20 m for the C veins and up to 60 m for the Y veins without support.

Unplanned dilution was estimated for stope hanging walls and footwalls using the equivalent linear overbreak/slough (ELOS) method developed by Clark (1998). The method is similar to the empirical stope stability charts discussed above with the Stability Number (N') and hydraulic radius but rather than showing stable/unstable conditions, the plot contains contours of dilution which approximate the average thickness, spread over the entire hanging wall or footwall. The analysis indicates approximately 0.5 m of dilution should be anticipated for the C veins while dilution for the Y veins will be controlled primarily by blasting practices, but up to 0.5 m.

Figure 16-8 and Figure 16-9 contain output plots from the empirical stability and ELOS analysis for long hole stopes in the C and Y veins, respectively.

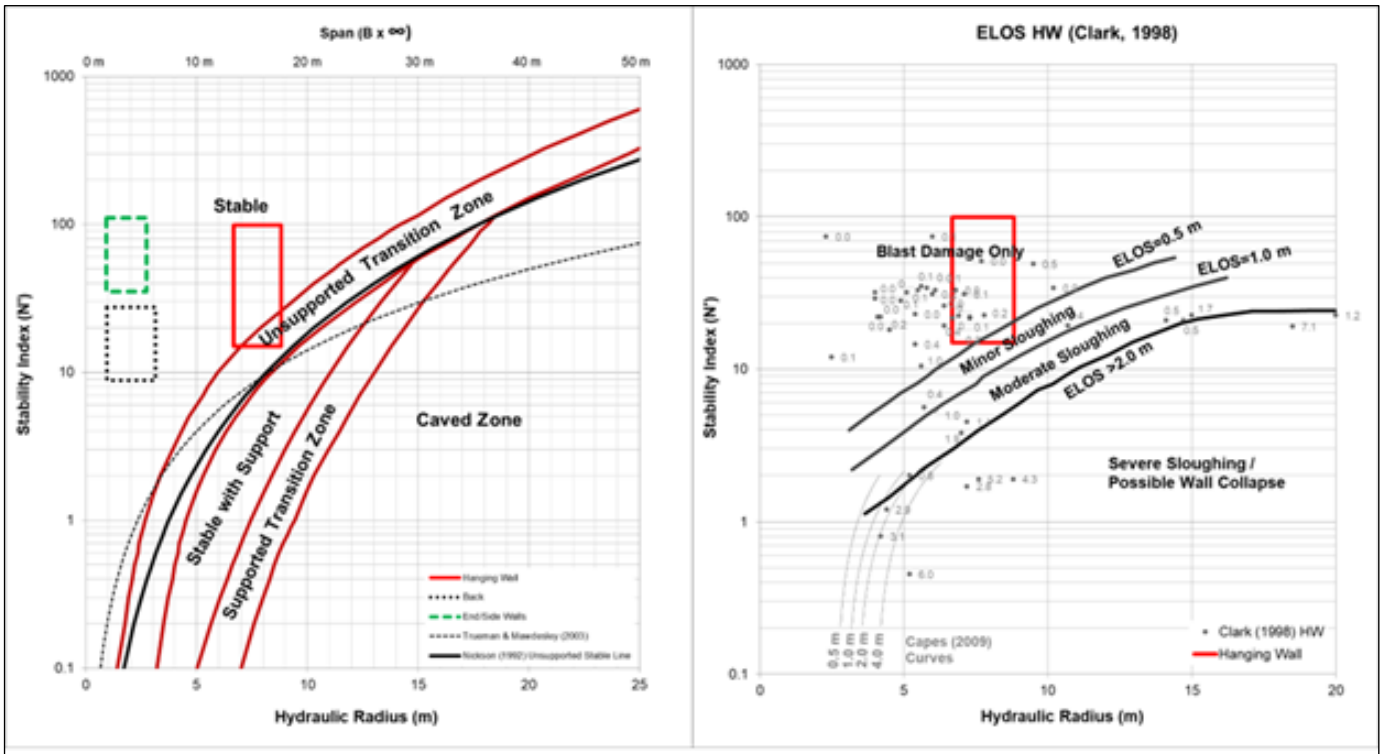
Figure 16-8: Empirical Stope Stability and Dilution Charts for C Veins



Stope Face	ID	Face Dip (deg)	Rock Mass Quality		Empirical Parameters			N'	Max. HR Allowed [C]	S
			Q'	RMR_{76} [A]	A (Stress) [B]	B (Joint Orientation)	C (Dip Angle)			
HW	HW - 14	55	14.0	67.8	0.32	0.2	4.6	4.1	6.0	5.6
Back	Back - 11	0	11.0	65.6	0.32	0.5	2.0	3.5	5.7	2.6
FW	FW - 14	55	14.0	67.8	0.32	0.2	8.0	7.2	7.3	5.6
End/Side	End/Side - 11	90	11.0	65.6	0.32	0.5	8.0	14.1	9.1	2.7
HW	HW - 27	70	27.0	73.7	0.46	0.3	5.9	22.2	10.7	5.0
Back	Back - 21	0	21.0	71.4	0.46	0.5	2.0	9.7	8.1	0.9
FW	FW - 31	70	31.0	74.9	0.46	0.3	8.0	34.2	12.3	5.0
End/Side	End/Side - 21	90	21.0	71.4	0.46	0.5	8.0	38.6	12.9	0.9

Source: JDS, 2025.

Figure 16-9: Empirical Stope Stability and Dilution Charts for Y Veins



Stope Face	ID	Face Dip (deg)	Rock Mass Quality		Empirical Parameters			N'	Max. HR Allowed [C]	S
			Q'	RMR_{76} [A]	A (Stress) [B]	B (Joint Orientation)	C (Dip Angle)			
HW	HW - 20	70	20.0	71.0	0.63	0.2	5.9	15.0	9.3	8.8
Back	Back - 14	0	14.0	67.8	0.63	0.5	2.0	8.8	7.8	3.1
FW	FW - 22	70	22.0	71.8	0.63	0.2	8.0	22.2	10.7	8.8
End/Side	End/Side - 14	90	14.0	67.8	0.63	0.5	8.0	35.3	12.5	2.7
HW	HW - 51	90	51.0	79.4	0.81	0.3	8.0	99.1	17.7	6.7
Back	Back - 34	0	34.0	75.7	0.81	0.5	2.0	27.5	11.5	1.0
FW	FW - 63	80	63.0	81.3	0.81	0.3	8.0	122.5	19.0	6.7
End/Side	End/Side - 34	90	34.0	75.7	0.81	0.5	8.0	110.2	18.3	0.9

Source: JDS, 2025.

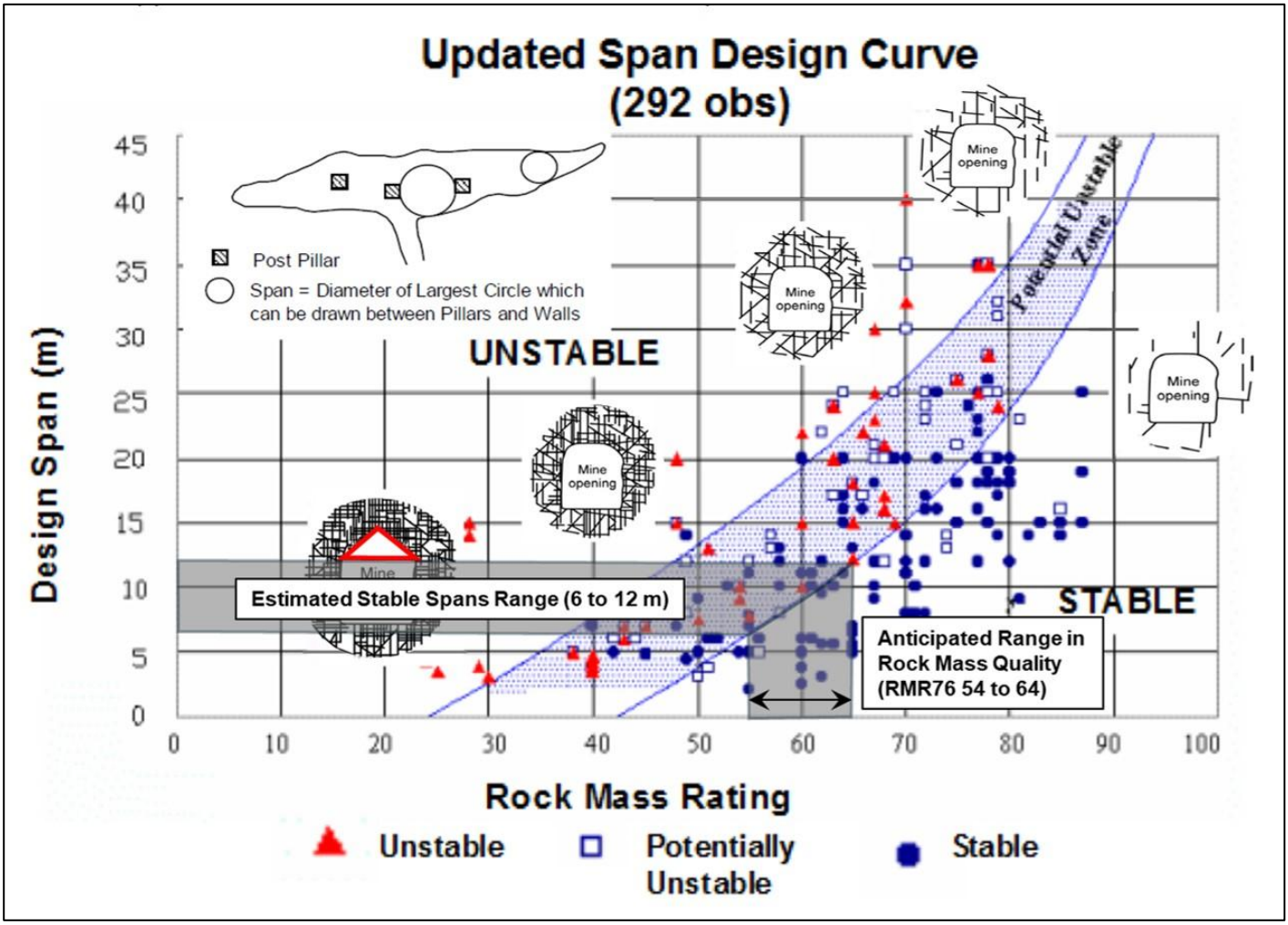
16.2.3.8 Drift and Fill Stope Stability

The maximum stable back span was assessed for the range of rock mass conditions anticipated in hanging walls of the Drift and Fill (upper C vein) areas using the Critical Span empirical method developed by Pakalnis (2007). The Critical Span is defined as the diameter of the largest circle that can be drawn within the boundaries of the exposed back. The method estimates the maximum stable span (Design Span) as a function of rock mass quality using the RMR (1976) classification system. The Design Span references the maximum stable back span with no support or local pattern bolting for surface control such as 1.8 m bolts on 1.2 x 1.2 m spacing. Local support is necessary to confine blocks that

may be loose or might open and fall because of subsequent mining in surrounding areas. Discrete wedges formed by intersection of structures are not considered in the method and must be supported prior to employing the Critical Span curve.

The weighted average and 20th percentile RMR values (Table 16-1) were used to represent the range in rock mass quality for the analysis. Results of the analysis are shown in Figure 16-10 which indicates maximum stable spans of approximately 6 to 12 m for the Drift and Fill stopes in the upper C vein area. Dilution in Drift and Fill stopes will be controlled primarily by blasting and excavation practices due to the small size of the openings and is anticipated to be minimal.

Figure 16-10: Critical Span Design Chart for Drift and Fill Stopes after Pakalnis (2007)



Source: JDS, 2025.

16.2.3.9 Extraction Sequence

To reduce potential stress impacts on mining, stopes should be sequenced to direct stresses away from the central mining areas and toward outer abutments. Where parallel veins exist, stopes should be mined in a hanging wall to footwall retreat sequence to divert, stresses away from the footwall stopes.

16.2.3.10 Backfill Strength Requirements

Long hole and Drift and Fill stopes require backfill to provide stability and allow full extraction, without leaving sill or rib pillars in the ore zone. Long hole stopes are planned to be backfilled with cemented paste backfill that will provide confinement on the stope walls and allow sequential stopes to be mined up against backfill of the previous stope. Drift and Fill stopes can use uncemented rock fill except where mineralized sill pillars will be left below the initial lift for each zone.

Requirements for cemented paste backfill uniaxial compressive strength (UCS) strength were developed analytically for various stope widths using the simplified wedge analysis originally developed by Mitchell (1983). A safety factor of 2.0 was assumed for the analysis which is considered appropriate for a Feasibility Study. It may be possible to reduce the safety factor and cement content during operation once site-specific experience is gained.

Recommendations for minimum cemented paste backfill UCS are as follows:

- Vertical exposures on the ends of the long hole stopes: 250 kPa;
- Sidewall exposures for Drift and Fill stopes (5 m high x up to 100 m long): 200 kPa; and
- Overhead/underhand exposures for Drift and Fill stope sills (5 m sill height): 800 and 6,000 kPa for 5 and 12 m sill widths, respectively.

16.2.3.11 Ground Support Design

Ground support has been designed based on rock mass classification and empirical design methods such as the Barton & Grimstad (2014) Q system and industry-standard rules-of-thumb. Ground support requirements were initially designed according to the Barton & Grimstad (2014) criteria for the range of anticipated rock quality (Q' values) as well as the size and expected life and use of the various mine openings. The Q-system also accounts for the life and use of the opening (ex. man-entry or equipment only) with the excavation support ratio (ESR) parameter.

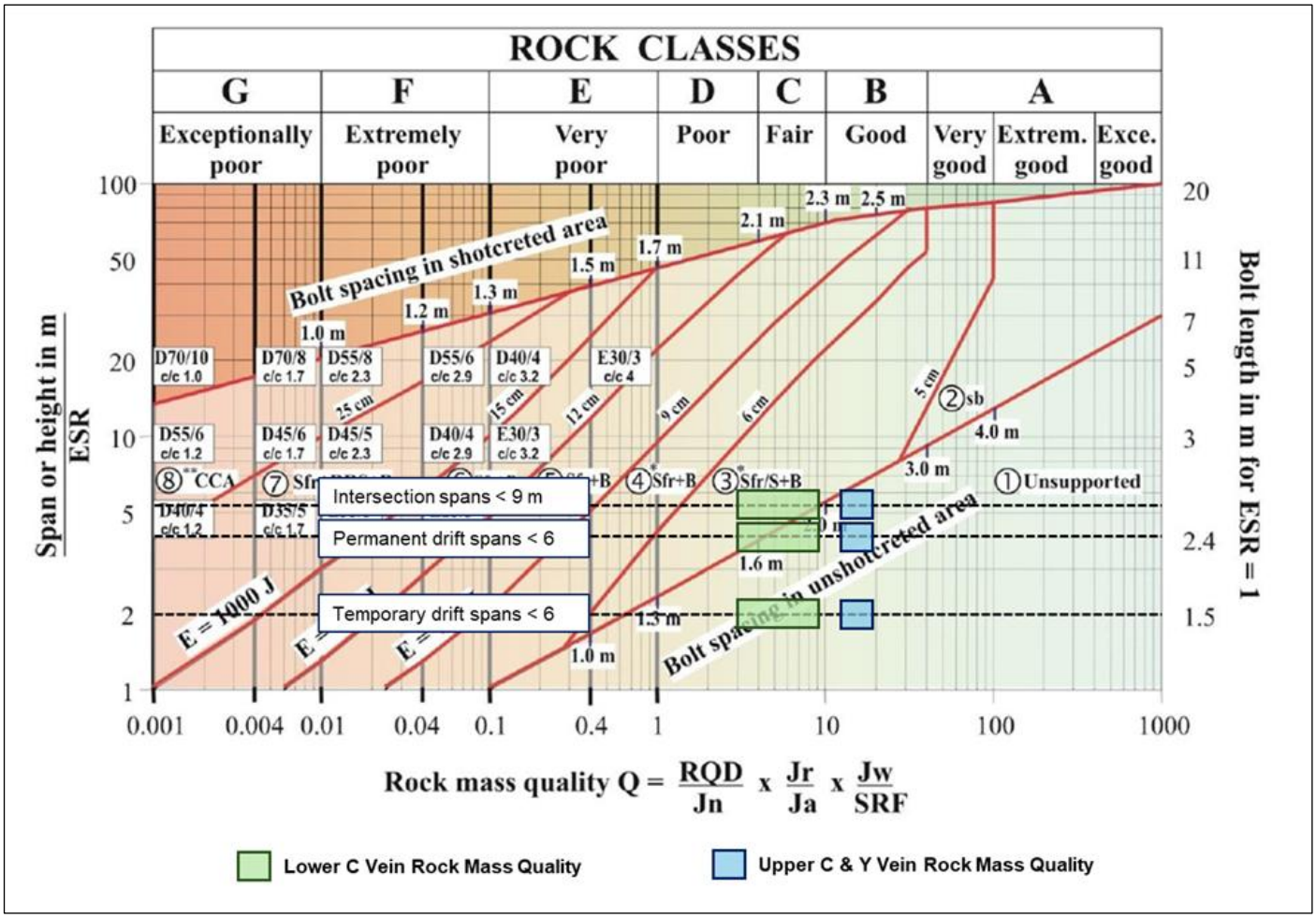
The ESR is used to adjust the design span to obtain the equivalent span (De) for use in the Q Support Diagram. In effect, it imposes a higher factor of safety on critical, permanent development (such as portals, ramps, crusher chambers underground workshops) and lower safety factors for short-term, temporary development such as ore drives. An ESR of 1.6 was assumed for permanent development and temporary, ore drives were assessed using an ESR of 3.

Results of the analysis are shown on the Barton Q ground support chart shown in Figure 16-11 which has three categories (equivalent dimensions) of openings:

- Standard permanent, decline profile with up to 6 m span (rounded up from 5.5 m to allow for overbreak);

- Maximum temporary, ore drive profile with up to 6 m span (rounded up from 5.5 m to allow for overbreak); and
- Intersections with up to 9 m spans or inscribed circles, as actually excavated (i.e., 3-way intersections with less than 2 m corner bevels or chamfer, or 4-way intersections with less than 1 m corner bevels or chamfer).

Figure 16-11: Q Ground Support Chart after Barton & Grimstad (2014)



Source: JDS, 2025.

The Barton Q ground support chart (Figure 16-10) indicates much of the development including temporary development in ore and all development in the upper C and Y veins require only spot bolting to maintain global stability; however, pattern bolting and welded wire mesh are recommended to control loose material for all development where miners will enter. Drifts and intersections with spans larger than 6.0 m in the upper C vein veins and immediate hanging walls will require the addition of 2 inches of fiber-reinforced shotcrete.

The recommended bolt lengths are selected based on the 1/3 of span rule-of-thumb for good rock quality. Openings with spans greater than 6 m (greater than 9 m in the case of intersections) require custom design that will include site-specific rock quality evaluations.

Ground support recommended for each type of permanent and temporary development are summarized as follows:

- Permanent Development (<6.0 m span):
  - Primary Support: 2.4 m #6 Fully Grouted Rebar, 1.2 m Square Pattern with Galvanized Welded Wire Mesh (6-gauge) to within 1.5 m of Sill; and
  - No secondary support required.
- Temporary Development (<6.0 m span):
  - Primary Support: 2.4 m FS 39 Split Sets, 1.2 m Square Pattern with Black Welded Wire Mesh to within 1.5 m of Sill; and
  - No secondary support required.
- Intersections (<9.0 m span or inscribed circle):
  - Primary Support: 2.4 m bolts, 1.2 m square pattern with galvanized welded wire mesh to within 1.5 m of sill; and
  - Secondary Support: 6.0 m, 26 tonne cable bolts, 1.8 m spacing within area of inscribed circle.
- Fault Zones/Poor Ground Conditions (assumed 5% of total development):
  - Primary Support: Same as above for respective development or intersections but reduce bolt spacing to 1.0 m and add 2 inches of shotcrete.

16.3 Mine Planning Criteria

The mine planning criteria for New Polaris are listed below:

- The pre-production mine development period will be approximately thirteen months, with the duration being split between surface preparations, portal rehab and slashing, underground ramp and infrastructure development.
- Ore will be extracted in the first quarter of year 1 and ramps up over a period of 8 months, achieving full production rate of 1,000 tonnes per day in month 9.
- It was estimated that due to weather conditions, access to the underground mine will be limited to 360 days per year:
  - Contractors will be utilized for underground activities until the start of production Year 2;
  - Conventional, trackless diesel-electric mining equipment will be utilized; and
  - Mined voids will be filled with rock fill or paste backfill.

Other key mine planning criteria are summarized in Table 16-4.

Table 16-4: Mine Planning Criteria

Parameter	Unit	Value
Operating Days per Year	Days	360
Shifts per Day	Shifts	2
Hours per Shift	Hours	12
Work Roster	Days on/days off	14/14
Ore Mining Rate	t/day	1,000
Ore Density (as per resource block model)	t/m³	2.81
Waste Density (as per resource block model)	t/m³	2.81
Swell Factor	%	35

Source: JDS, 2025.

16.4 Mining Methods

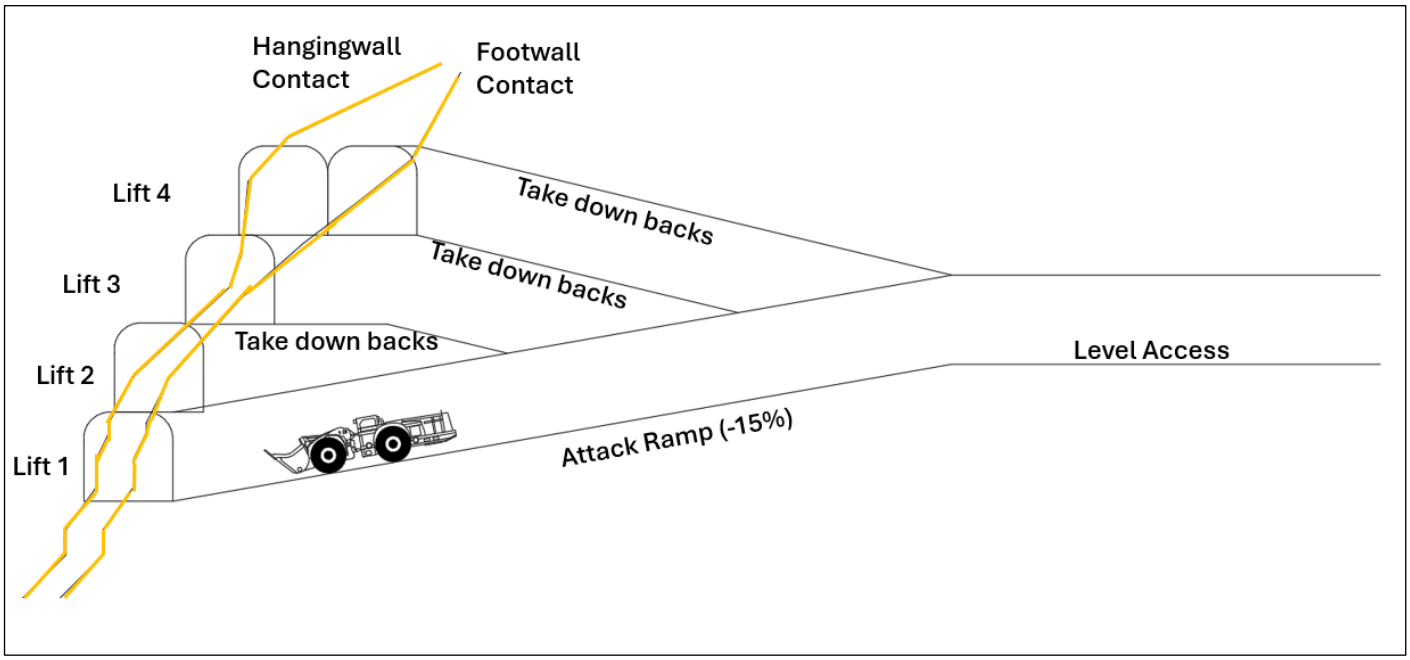
16.4.1 Drift and Fill

Drift and Fill (DAF) was selected for areas that have lower quality rock and/or where the resource geometry is not amenable to LHS. In this method, mining begins at the bottom of the chosen horizon and progresses upward. During the mining sequence, the back of the excavation is temporarily supported using rock bolts before the stope is backfilled to form the floor of the next level of mining. Backfill is designed to provide sufficient excavation support and a strong working floor for personnel and equipment. DAF stopes will be filled with paste fill on the bottom cut, this will be used as a pillar to mine up against from the production horizon below.

Stopes are accessed via attack ramps driven at a maximum ±15% gradient from a level access. Minimum drift size for DAF is 3.5 m W x 5.0 m H. DAF mining is planned as one or more passes, with an average mining width of 8 m. Where multiple-pass mining is required, it will be performed in a primary-secondary configuration whereby the primary cuts are taken first and filled with paste backfill. This provides structural wall support to permit mining directly adjacent to the primary cut. An example of primary/secondary cut is shown as lift 4 in Figure 16-12.

A typical single-pass DAF mining configuration is illustrated in Figure 16-12.

Figure 16-12: Typical Longitudinal Drift and Fill Mining Configuration



Source: JDS, 2025.

16.4.2 Longhole Stoping

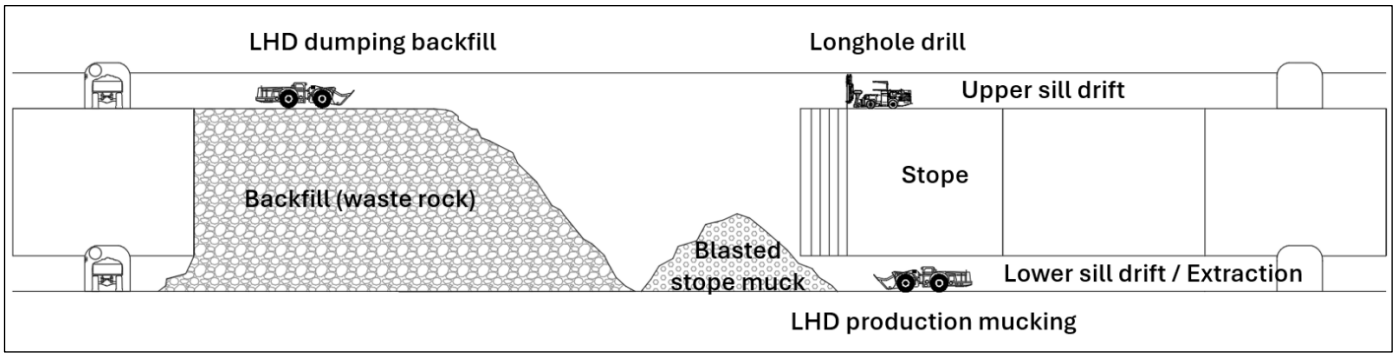
Longhole Stoping was chosen as the mining method for approximately 30% of the Mineral Reserve based on the configuration and grade of mineralization. In general, longitudinal retreat stoping will be utilized along with Avoca stoping. LHS is not a selective mining method when applied over long vertical distances due to drill hole deviation and vein geometry variations. To mitigate these effects, level spacing will be limited to 20 m. Typical stopes dimensions are 20 m long and 3.2 m wide and 5.1 m wide in the Y and C zones, respectively.

The longhole mining cycle will begin with blasting the slot raise to provide a free face for the first longhole round and initial empty volume for blasted swell. Production blasting will begin at the stope ends and retreat to the crosscut and/or access. Blasted material will report to the bottom of the stope where load-haul-dump (LHD) machines muck the material directly to the truck or dump into a remuck for rehandling.

Top sills will be utilized for drifting, loading, and blasting of the stope below, as well as providing access for LHDs for backfilling. Once the stope below is completed and backfilled, the drift will then be used as a bottom sill for mucking ore from the stope above. In the event a top sill is unavailable, the lower extraction sill may be used to drill up holes.

Longhole Avoca stoping will also be used. This method requires access on the top sill drive from both ends of the stoping block. As the ore is blasted, mucked and generally retreated, backfill is placed from the second top access limiting the exposed span along both the hanging wall and footwall. An example of Avoca stoping is shown in Figure 16-13 where the stope block is retreating left to right on the page.

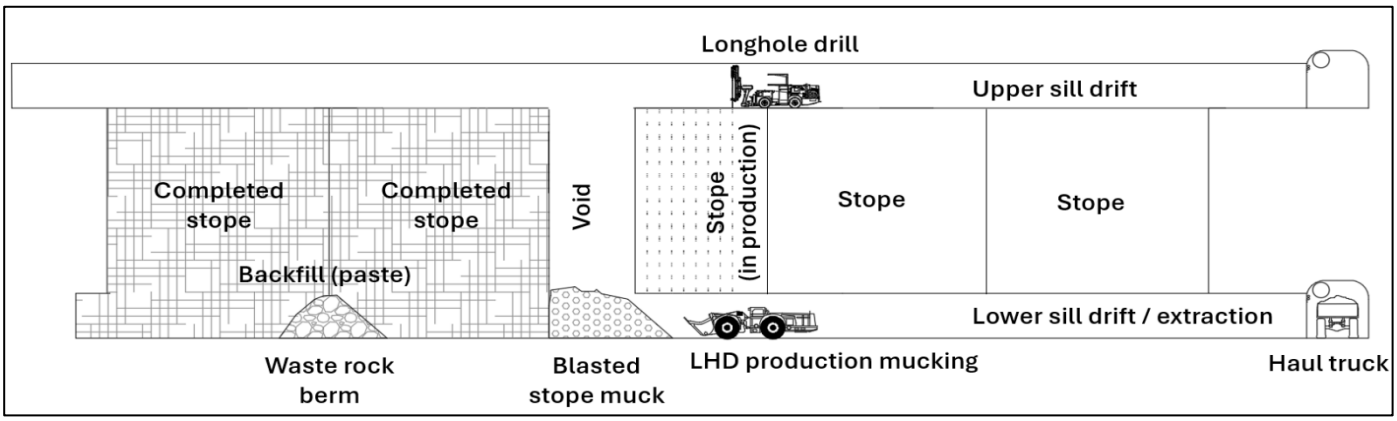
Figure 16-13: A Long Section Showing Avoca Longhole Stoping Method



Source: JDS, 2025.

Longitudinal retreat stoping will be utilized in areas not amenable or cost effective to use the Longhole Avoca stoping method. The stope cycle occurs in distinct phases where the backfill isn't placed until the stope is fully blasted and mucked clean. Stopes mined using this method will be backfilled with Paste fill. An example of longitudinal retreat stoping is shown in Figure 16-14.

Figure 16-14: A Long Section of Longitudinal Retreat Stoping



Source: JDS, 2025.

16.5 Mine Design

16.5.1 Stope Optimization

Multiple sets of stope shapes were created with Maptek™ Vulcan© mine stope optimizer (MSO) using different parameters (e.g. HW and FW angles, and widths and dips) until an optimized configuration was found that best captured mineralization whilst meeting mineability requirements. For DAF stopes there were two profiles analyzed. A narrow profile with a minimum width of 3.5 m up to 60-degree dip on the hanging wall (HW) and 90-degree footwall (FW), and a wide profile with a 4.75 m minimum mining width and 60-degree HW and FW. The LHS parameters were

2.8 m minimum mining width including Equivalent Linear Overbreak Sloughing (ELOS) dilution, 55-degree minimum dip angle.

Table 16-5 contains the final parameters used in the creation of stope shapes.

Table 16-5: Stope Optimizer Software Parameters

MSO Parameters	Unit	Drift and Fill Value (Narrow / Wide Profiles)	Longhole Stoping Value
Default density	mt/m³	2.81	
Cut-off grade	g/mt	6.0	5.5
Dip and strike		Dynamically controlled by vein wireframe	
Axis rotation		individual/custom based on strike angle of Vein Wireframe	
Stope length - Sections (U)	m	20	20
Stope height -Levels (V)	m	5	20
Stope minimum width	m	3.0 / 4.25	2.3
Dilution thickness – HW+FW	m	0.25+0.25	0.25+0.25
Diluted width	m	3.5 / 4.75	
Hanging wall dip angle	Deg °	60 to 90	55 to 125
Maximum change	Deg °	20	20
Footwall dip angle	Deg °	85 to 90 / 60 to 90	55 to 125
Maximum change	Deg °	20	20

Source: JDS, 2025.

Optimizations were run separately for each vein to allow for changing strike and dip angles. The resulting stope optimizer shapes were reviewed in twenty-meter horizons, and mining costs, recoveries and revenue were calculated for each method and the most profitable method for each horizon was selected. There was also consideration given to methods selected for veins in immediate proximity to those being analyzed.

16.5.2 Dilution and Mining Recovery

ELOS dilution for DAF was estimated at 0.25 m for the hanging wall and footwall for a total overbreak of 0.5 m. For LHS, dilution was estimated 0.25 m on the hanging wall and footwall for a total overbreak of 0.5 m. In addition, mucking dilution (0.25 m) and backfill dilution from adjacent LHS of (0.5 m) is added.

Ore losses (mining recovery factor) are related to the practicalities of extracting ore under varying conditions, including adverse mining geometry, rock mass conditions, losses of ore into backfill, and potential blasting damage. Dilution and recovery factors are summarized in Table 16-6.

Table 16-6: Dilution and Recovery Factors

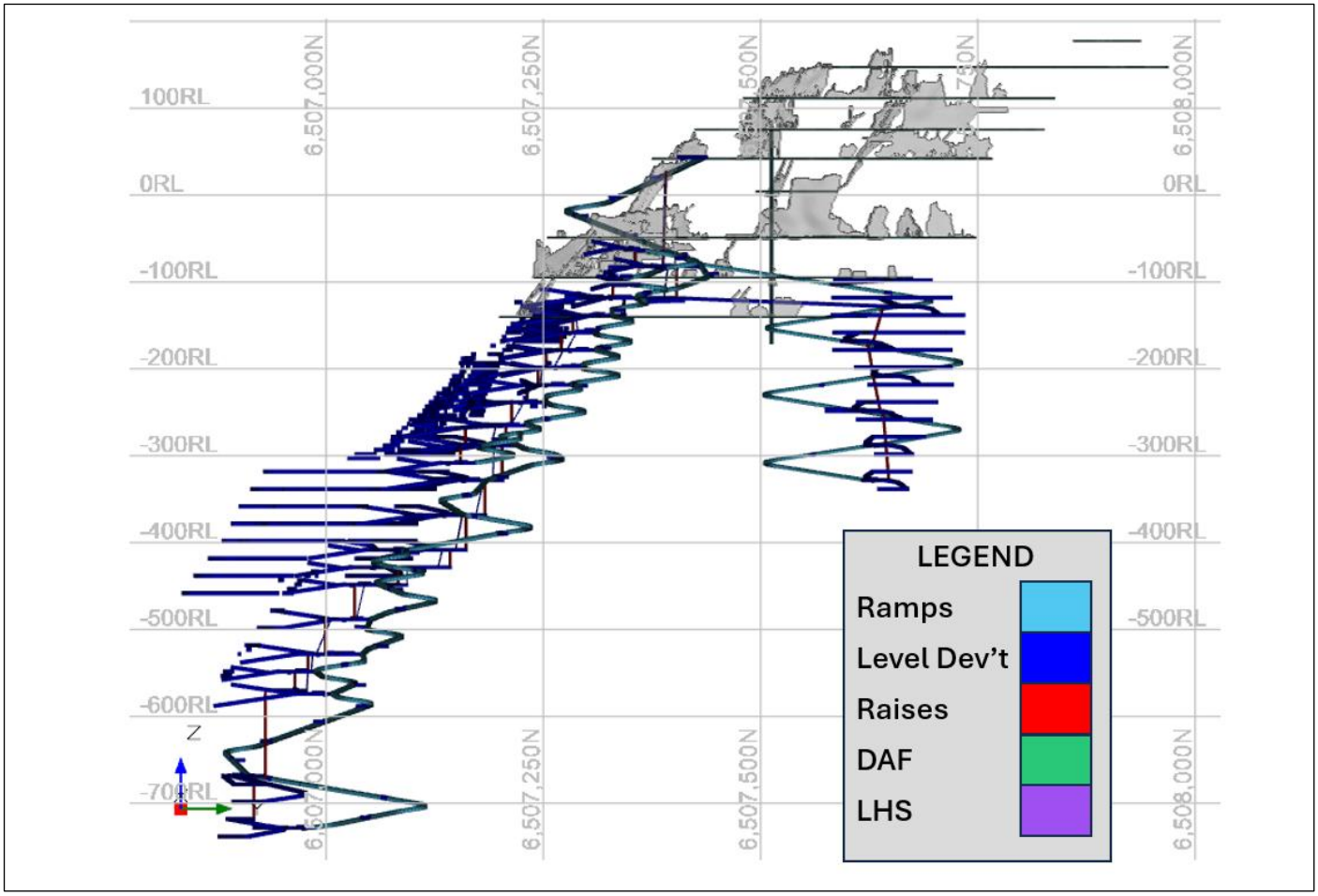
Mining Method	Unplanned dilution (%)	Recovery (%)
Drift and Fill	18	97
Development	18	100
Longhole stoping (blind upper)	24	90
Longhole stoping (top and bottom access)	24	95

Source: JDS, 2025.

16.5.3 Development

The lateral and vertical development for the mine is illustrated in and described in detail in the following sub-sections.

Figure 16-15: Overview of Planned Lateral and Vertical Development (Looking West)



Source: JDS, 2025.

16.5.3.1 Lateral Development

Access to underground will be through the existing Polaris Portal, which will be slashed-out following the existing development towards the historical No. 1 Main Shaft and a new decline then driven down to access the orebodies. At the top of the Y zone an additional ramp will be developed coming off the shared C zone ramp. Level accesses will be mined for each zone spaced 20 m vertically.

Lateral development heading sizes and lengths are presented in Table 16-7.

Table 16-7: Lateral Development Heading Sizes

Development Type	Width (m)	Height (m)	Total Length (m)
Portal slash	5.5	5.5	55
Main ramp	5.5	5.5	2,856
Internal ramp	5.0	4.5	4,454
Attack ramp	4.0	5.0	5,969
Take Down Backs	4.0	5.0	2,285
Level access	5.0	4.5	643
Footwall	5.0	4.5	1,431
Stope connector	3.5	5.0	4,605
Stope sill	3.5	5.0	5,848
Drift and Fill	3.5	5.0	34,396
Sump	4.5	4.0	243
Clean water sump	5.0	6.4	177
Settling sump	5.0	5.0	69
Remuck	5.0	4.5	1,083
Loadout bay	5.0	6.0	637
Workshop	6.0	6.0	170
Electrical substation	5.0	4.5	195
Powder magazine	7.0	6.0	36
Detonator magazine	5.0	4.5	8
Exploration bay	5.0	4.5	184
Return air access	5.0	4.5	2,172

16.5.3.2 Vertical Development

Vertical development in the mine is for ventilation/emergency egress purposes.

Raise development will occur either through drill and blast drop raising, Alimak (multi-level raises), or by Raise bore machine. The vertical development opening sizes are presented in Table 16-8.

Table 16-8: Vertical Development Sizes

Excavation Method	Zone	Purpose	Dimension (m)	Total Length (m)
Drop raise	Zone C	Egress	2.1 dia.	474
Drop Raise Total				**474**
Alimak/Raisebore	Zone C	Egress	1.5 x 1.5	109
		Return air raise	3.5 dia.	848
	Zone Y	Return air raise	3.5 dia.	194
Alimak/Raisebore Total				**1,151**

16.5.4 Infrastructure Development

Infrastructure designs include the following:

- Dewatering sumps
- Electrical substations
- Exploration bays
- Explosives (powder and detonator) magazines
- Footwall accesses (take-off point for attack ramps and sill development)
- Return air raise accesses
- Loadout bays
- Remuck bays
- Workshop bays

16.6 Mine Services

16.6.1 Mine Water Supply

The assessment of mine water supply for underground operations and infrastructure involved determining the total volume of water required considering the specific needs of the underground environment. The following are some of the key factors considered:

- Drilling operations
- Ground support operations
- Dust suppression, mining heading wash-down, and mucking.

Mine supply water from the surface freshwater tank will be distributed underground via pipelines installed in the decline, with pressure reducers, and distributed to the working areas. Peak underground usage was estimated at 4.4 L/s.

16.6.2 Dewatering

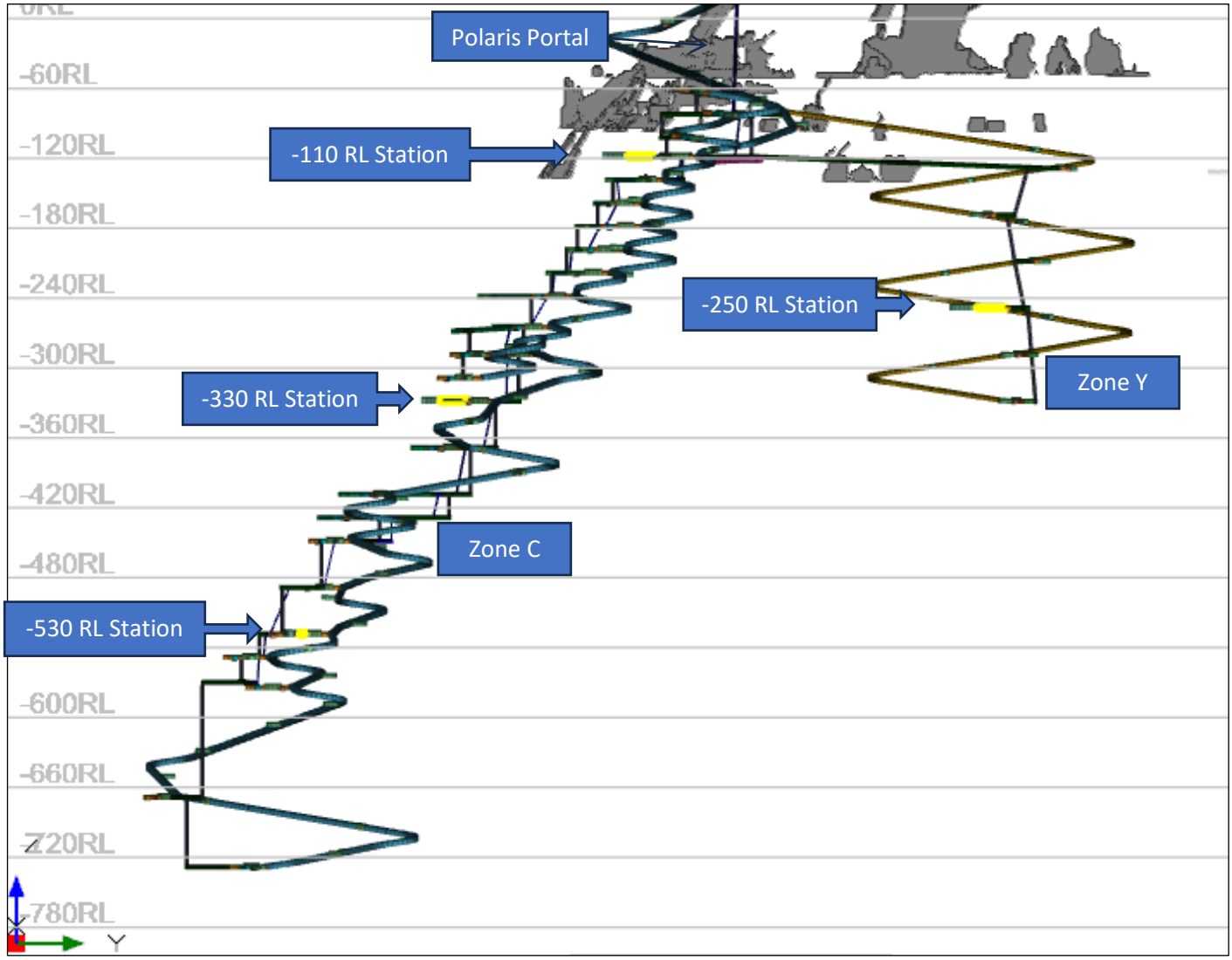
The current hydrogeological model has insufficient information to develop a definitive estimate of water inflow into the mine. Until such time that a more accurate estimate of flow can be determined by means of drilling and modelling, a flow of 60 L/s has been assumed for estimating power demand, capital and operating costs related to pumps, pipes, sumps and other dewatering related activities.

A contact water only dewatering system strategy has been assumed in which all water is pumped out of the mine in a single line. There is an opportunity for advance dewatering to allow for a dual system of contact and non-contact water handling to be considered in the future.

Dewatering will expand in four stages as the main ramp advances to the bottom of the mine in each zone. The mine dewatering system involves six pump stations and 25-level sumps. Ultimately, the pump stations on -110 RL Level will be responsible for collecting both mine water inflows from the decline above and all underground water. This combined flow will be pumped to a designated surface settling pond.

As mining progresses, pump stations will be established as illustrated in Figure 16-16. The pump stations will be stage connected, as required, to other secondary sumps and, ultimately, to the portal pump station. Small collection sumps on each level will drain through screened drill holes to the sumps and pump stations on levels below. The final secondary pump station will be located on the -570 RL Level to service the bottom of Zone C.

Figure 16-16: Underground Dewatering Pump Station Locations (Looking East)



Source: JDS, 2025.

16.6.2.1 Pump Station Configuration

Every pump station is equipped with a dirty water (settling) sump and a decant system that feeds into a clean water (reservoir) sump before pumping. This allows most of the slimes to settle out of the water column prior to pumping. Mitigating the presence of abrasive solids is important for extending the operational life of the pump impellers. Settling sumps are designed with sloped ramps to enable clean-out of solids by LHD equipment.

Each settling sump utilizes a single submersible pump to transfer water to the clean water sump (reservoir) before entering a multistage centrifugal pump, which pumps water to the next pump station above, or out of the mine in the case of the -110 LR pump stations.

16.6.3 Potable Water Supply

Large plastic containers will be used to deliver potable water underground to the workshop and refuge stations. Personnel will bring their own potable water in appropriate water containers to their work areas.

16.6.4 Compressed Air

Mobile equipment will come equipped with onboard compressors. The workshop will have a dedicated electrical air compressor.

16.6.5 Mobile Equipment Maintenance

The main maintenance workshop will be located on the surface. Maintenance on mobile equipment that ordinarily travel to surface and all major repairs will be undertaken in the surface workshop. A smaller underground workshop will be constructed to supplement the main surface workshop activities. The underground workshop will be a repair bay facility capable of servicing two pieces of mobile equipment.

16.6.6 Diesel Fuel Handling

Haul trucks and auxiliary mobile equipment will be re-fueled on surface at a fuel station. Drilling equipment and LHDs will be re-fueled underground with the fuel/lube truck.

16.6.7 Explosives Storage

16.6.7.1 Surface Storage and Manufacturing

An emulsion storage and production plant will be set up at the mine site. This site will have the following main components:

- Emulsion mixing plant
- Ammonium nitrate, Additive 1 (manufacturer proprietary) and surfactant storage
- Diesel storage and generator
- Packaged explosives and detonator storage magazines.

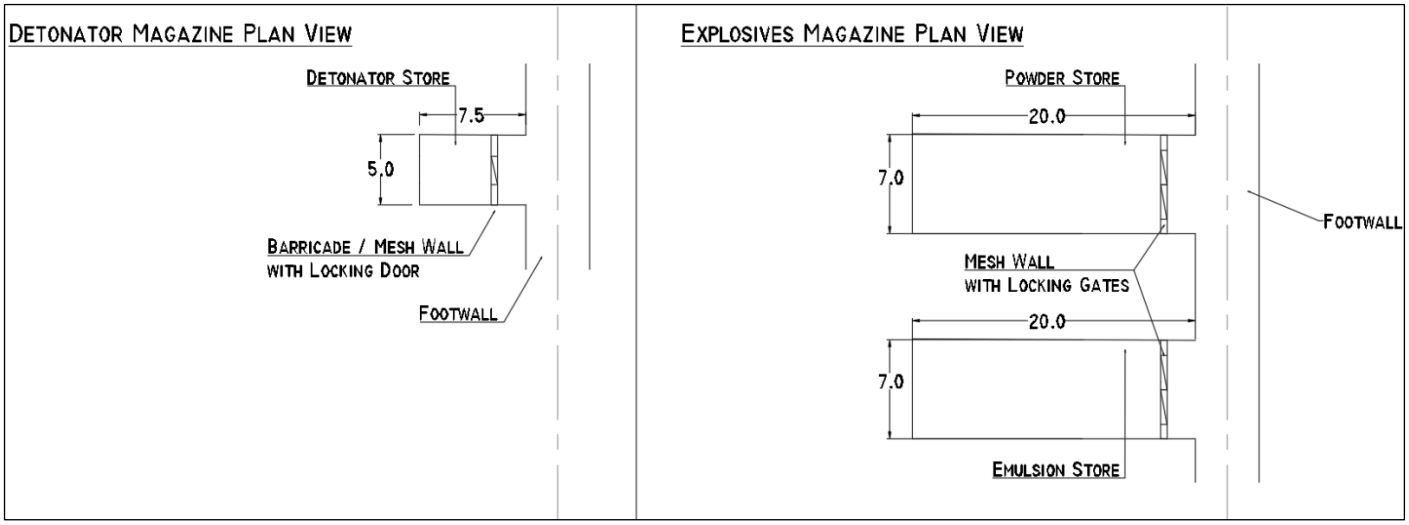
The emulsion plant will be constructed from ISO dry type containers with each container storing a component for emulsion production. Bulk Ammonium Nitrate Prill (ANP) will be shipped to site and stored in ISO dry type containers. A two months' supply of ANP will be kept on site at any one time. Two magazines are required to separately store packaged explosives and initiating system components (detonators).

16.6.7.2 Underground Storage and Handling

Underground storage magazines for explosives, detonators, and blasting accessories will be located on the -140 RL Level. There will be two explosives storage magazines and a separate magazine for storing detonators. Explosive products used for development and production mining will be securely handled and stored in the magazines. Explosives magazines will come with swing open wire mesh doors. The magazines will be equipped with fire extinguishers, wooden shelves, and concrete flooring. Bulk emulsion will be transported in specially designed totes on a deck truck with a boom. This boom will be used to load and unload emulsion totes.

The location of the explosives or detonator facility will be a minimum of 100 m from main access ramp, refuge station, transformer vault or a fuel storage or fuel transfer area. The detonator magazine will be no closer than 10 m to the explosives magazine. Packaged explosive and detonator products will be transported via special trucks from the surface via the ramp to the underground magazines.

Figure 16-17: Underground Explosives Magazines Basic Layout



Source: JDS, 2025.

16.6.8 Ventilation

Ventilation requirements were calculated based on the underground activities projected to be concurrently undertaken. The function of the ventilation system is to dilute/remove airborne dust, diesel emissions, explosive gases, and to maintain temperatures at levels necessary to ensure safe production throughout the life of the mine. The ventilation system has been designed to meet the requirement of the Health, Safety and Reclamation Code for Mines in British Columbia.

Each mining zone will be ventilated by a “Pull” or exhausting type ventilation system with the return air delivered through a network of raises and ventilation drifts connected through a final single exhaust raise on the -80-elevation daylighting at surface. Fresh air will enter the zones via the mine portal and then branch from the main ramp to each

zone at around the -80 RL horizon. In winter, air will be heated by glycol heat recovery system located on the surface near the Polaris Portal, capturing heat from the plant gensets. The heated air ducted to the intake airflow inside the portal.

Each level will have its own ventilation circuit. Fresh air will enter each level from the ramp and will be delivered to the working faces via auxiliary fans and ducting. Contaminated air exits the level through the return air raise (RAR) system via a regulator installed in the access to the RAR on each level. The proposed ventilation system was modelled using Ventsim software to check air velocities and practicality of the overall system.

Air volume requirements were calculated to ensure safe production. The amount of air required is determined by the number of active workplaces; particularly to ensure dilution and removal of dust and noxious gases as well as diesel particulate matter. Based on the scheduled production and development activities, peak airflow was calculated at 164 m^3/s.

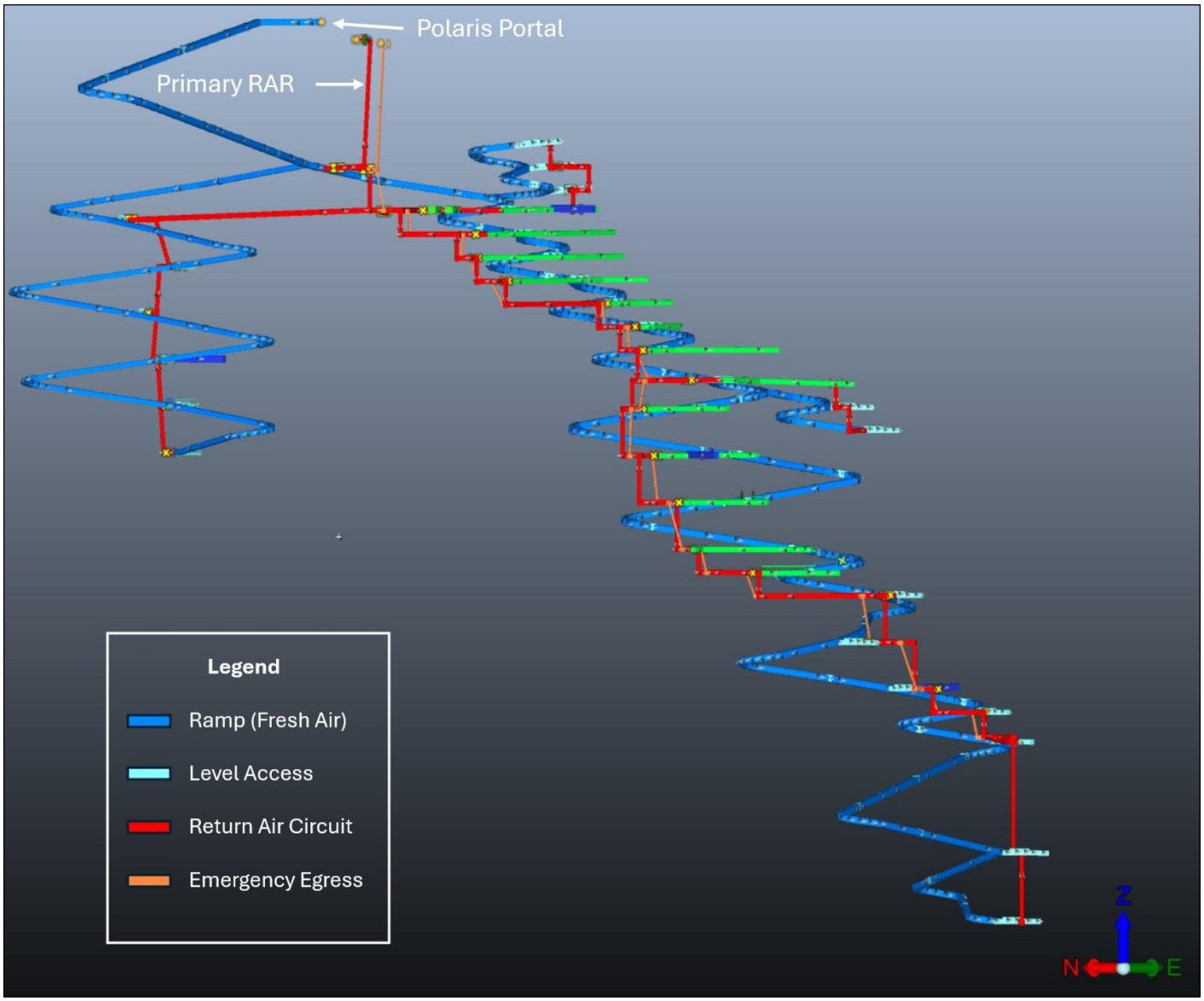
Intake air is through the main portal (Polaris Portal). The maximum total airflow through the portal is calculated as 200 m^3/s.

Based upon the equipment projected to be required and the activities being concurrently undertaken, a maximum total of 151 m^3/s airflow is required for Zone C and 48 m^3/s maximum airflow is required for Zone Y. This air is designated primarily to provide adequate dilution of exhaust emissions from the underground diesel equipment fleet and ensure sufficient supply of fresh air for personnel. Peak totals in the Y Zone and C Zone occur at different times and do not add to the 164 m^3/s peak.

The primary exhaust fans required for the ventilation system are 4 x 355 kW (450 hp) fans.

The ventilation strategy will transition as mining progresses and the RARs are completed. The system for the mine as a whole is shown in Figure 16-18.

Figure 16-18: Underground Mine Ventilation System (Looking Northeast)



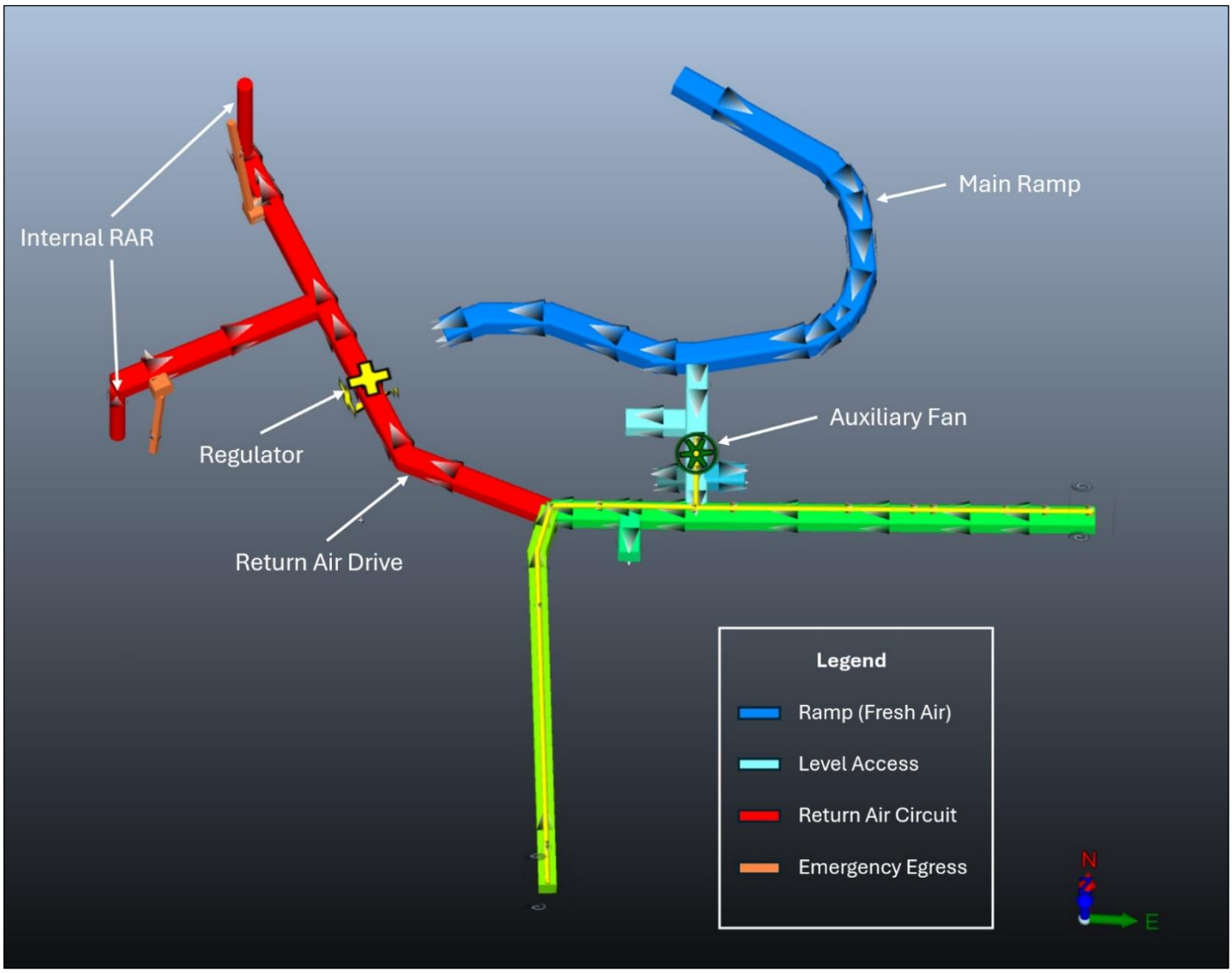
Source: JDS, 2025.

16.6.8.1 Auxiliary Ventilation

For ramp development, it is assumed that there will be a maximum of one loader and one truck in the decline heading. Air will be delivered through twin ventilation ducts with one duct extending to the face and the other to the truck loading bay. Each duct requires a twin 75 kW (total 150 kW or 200 hp) fan.

For level development and during the production cycle, it is assumed that trucks will not enter the ore drives and, instead, will be loaded at the level accesses. As such, the mining levels will need to accommodate ascoop as the largest piece of equipment. A typical level will have a maximum of two independent duct systems. Each duct system will have a 75 kW (100 hp) fan. Following completion of the ramp development for Zone Y, the ramp development fans will be repurposed for level production activities on the two levels, with the longest lateral extents in Zone C. Figure 16-19 shows the typical level ventilation plan.

Figure 16-19: Typical Level Ventilation Schematic



Source: JDS, 2025.

16.6.8.2 Emergency Egress and Refuge Chambers

In development of the ventilation strategy, consideration has been given to the potential for mine emergencies requiring evacuation. As such, the following criteria have been established.

On almost all levels, escape can be either to a ramp or to the escape ladderway in the internal raises.

Raises in Zone Y will be equipped with ladderways for personnel egress. Zone C has dedicated escape raises designed.

In the ramp, escape may either be up the ramp or down the ramp to a safe area.

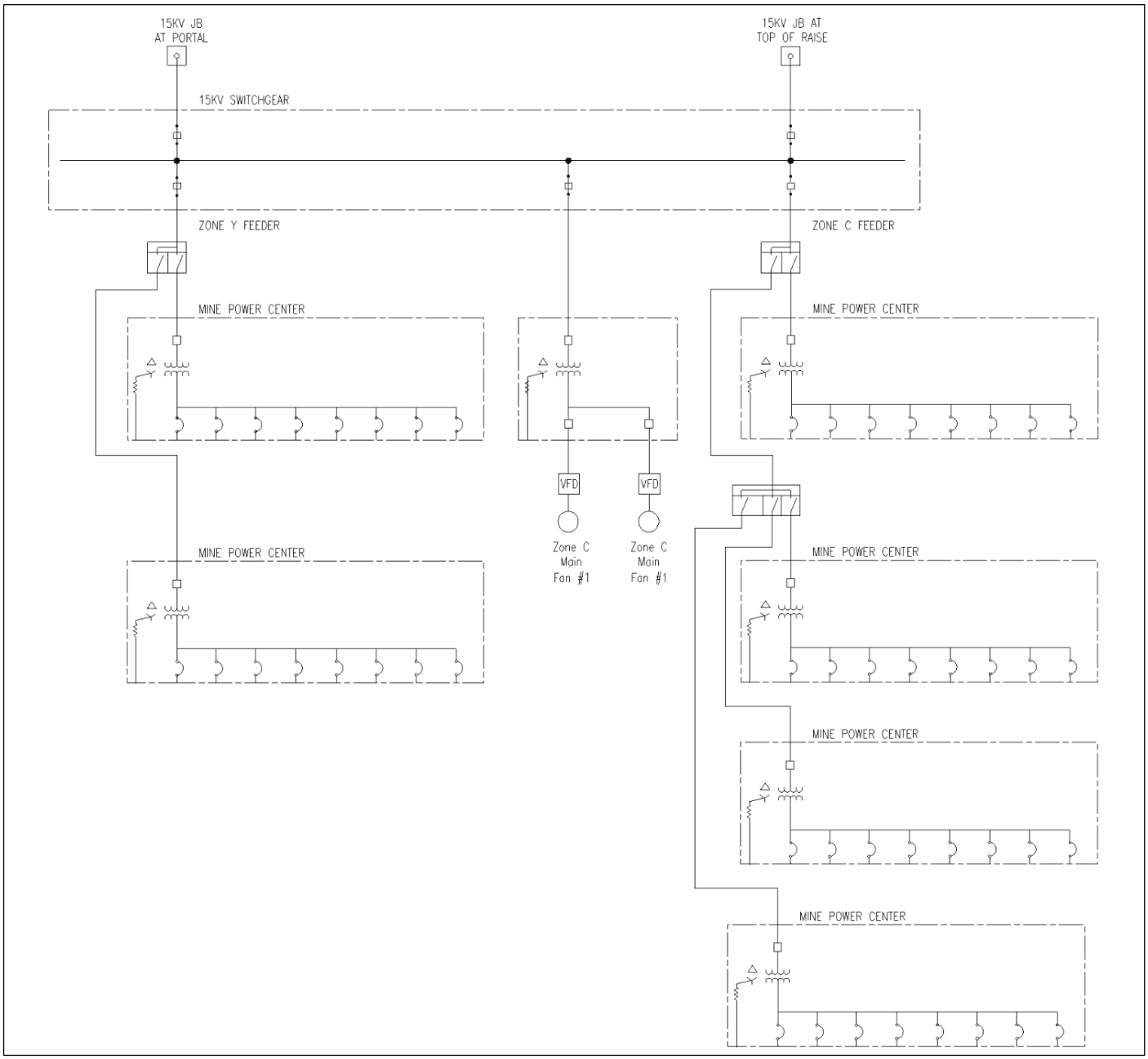
Portable refuge chambers will be installed at the most appropriate points in the mine.

16.6.9 Electrical

Power will be provided to the underground mine at 13.8 kV, 3 phase, 3W, 60 Hz. The system will be high resistance grounded with a maximum let-through current of 25A, pending charging current calculations. The underground mine will be provided with two feeders – one routed down the main ramp, and a second feeder down the surface vent raise once in place. The second feeder is intended to be a redundant power supply, with only one of the two feeders required to power the entirety of the underground mine.

On the -110 RL Level, both feeders will be connected to a 15 kV switchgear skid with two main breakers and three feeder breakers: one for Zone C, one for Zone Y, and one for the main ventilation fans. Generally, each substation will consist of a double gang load break disconnect switch and a mine power center (MPC) consisting of a main circuit breaker, 750 kVA 13.8-0.6 kV dry type transformer, 347V 5A continuously rated neutral ground resistor, and 600V distribution. A 400A skid-mounted load center will be provided for each level, with each MPC powering three to four level load centers. Load centers will be used to power mining and infrastructure related equipment on the associated level. The single line diagram (SLD) for the underground mine is shown in Figure 16-20.

Figure 16-20: Electrical Single Line Diagram for the Underground Mine



Source: JDS, 2025.

16.6.10 Communications

Similar to the power distribution network, redundant fiber optic cables will be run to the underground mine from surface via the main ramp and the Zone C ventilation raise. Communications cabinets will generally be located in each electrical substation bay for consolidation of fiber, leaky feeder, and wireless access cabling. Wireless access points will be distributed throughout the mine to provide data and communications connectivity to personnel and mobile and stationary equipment. Voice communications will primarily be via leaky feeder, process control via fiber optic cable, and equipment monitoring/data-based communications by Wi-Fi.

16.7 Unit Operations

16.7.1 Drilling

There are five principal drilling machines selected for New Polaris. Each has their own primary use:

- Two-boom electric-hydraulic jumbos for large dimension development and production headings.
- Single-boom electric-hydraulic jumbos for level slashing and drift and fill.
- Mechanized rock-bolters for ground support installation.
- Electric-hydraulic longhole drills for longhole stopes and drop raises.
- Jackleg and stoper drills for bolting in smaller Drift and Fill headings and raises, drilling safety bays, service drilling, ground support installation.

Drilling productivities (meter drilled/percussion hour) were built up from first principles and industry standards. Jumbo drilling rates vary from 60 m/h in small headings (1-boom) to 90 m/h in large headings (2-boom), and longhole drill machines average 14.5 m/h. Drilling productivities include penetration rates, reaming rates, repositioning, set-up/mark-up, and a 5% factor for re-drilling.

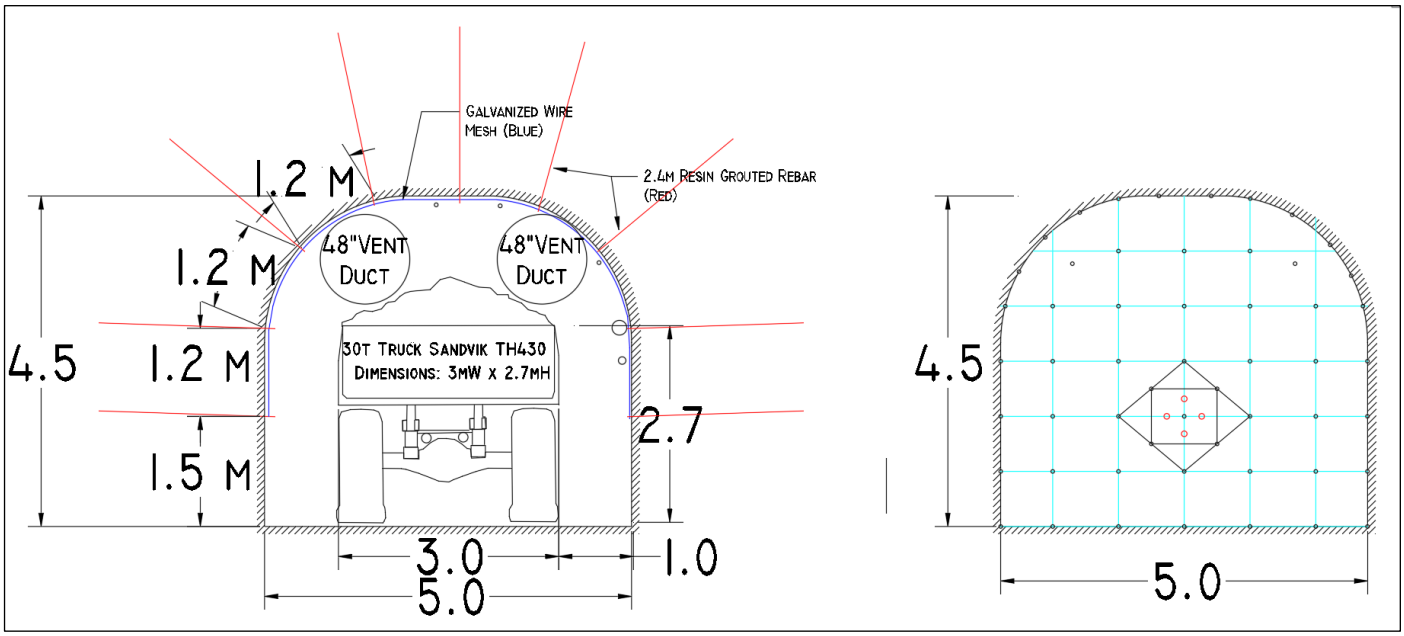
Drift and fill headings, attack ramps, sill drives and stope connection drives will be conducted by single-boom electric jumbo drills. The single-boom jumbos will be equipped with 3.7 m (12 ft) drill steel and will advance at an average rate of 3.8 m per round.

Ramps, footwall access, footwall drives, remucks, sumps and other large headings will be developed by two-boom electric jumbo drills. Jumbos will be equipped with 4.88 m (16 ft) drill steel and will advance at an average rate of 4.4 m per round.

Approximately 5% of all holes are anticipated to require re-drilling.

A typical development profile is shown in Figure 16-21, showing the 4.5 m H x 5.0 m W Internal access ramp.

Figure 16-21: Typical Ramp Profile



Source: JDS, 2025.

16.7.1.1 Blasting

Blasting crews will be trained and certified for explosives use. Bulk or packaged emulsion will be used for production blasting and development rounds.

Bulk emulsion is proposed as the primary explosive product to be used for lateral development, drift and fill, and Longhole Stoping. Emulsion can be handled in bulk deliveries to reduce costs in both pre-sensitized form and in its individual components for manufacturing on site, which is the preferred option for New Polaris. No ammonium nitrate and fuel oil (ANFO) has been incorporated into the mine plan. Boosters, primers, detonators, detonation cord and other ancillary blasting supplies will also be utilized. Smooth blasting techniques may be used as required in headings, with the use of trim powder/bulk emulsion for loading the perimeter holes.

For Alimak mining, packaged explosive products have been incorporated into the mine plan. It is also noted that, should poor quality/highly fractured ground conditions anywhere in the mine be encountered (e.g. fault zone), packaged explosives can be effectively used for all blasting in such zones.

Blasts will be initiated with a mix of non-electric and electric detonators. All blasting will be done at shift change. All personnel not clear from the underground will be required to be in a designated safe work area during blasting. During the production period, a central blast system will be used to initiate blasts for all loaded development headings and production stopes.

16.7.2 Ground Support

Ground support will be installed in accordance with specifications based on geotechnical analysis provided in Section 16.3 for the various rock qualities expected. Electric-hydraulic bolters, longhole drills, jackleg drills, and shotcrete spraying machines will be used.

Ground support will be installed post-mucking of the blasted drift. No additional development will be commenced in the heading prior to the installation of ground support. At no time will mine workers be under unsupported ground.

Different ground support criteria are recommended for various types of ground conditions, rated from good to poor, and largely associated with depth and active use of the heading. Discretion will be made by the development lead with support from the geotechnical engineer as to which ground support is required.

Regular pull tests will be conducted on-site to ensure adequate installation of rock bolts. Shotcrete, when required, will also be sampled by use of splatter boards and in-situ coring to be tested for strength and adequacy. Rock bolts and screens will be installed with a bolter machine.

Cable bolts will be installed in 3- and 4-way intersections and along sub-level LHS sill drifts when required. Cable bolts may be installed shortly after the development behind the development crew as to maintain the advance rate of the drift. When intersections are located in poor ground conditions, shotcrete and cable bolts will be installed prior to development of the intersection. Cable bolts will be drilled with a longhole drill and installed from a scissor deck manually.

Shotcrete application is not expected to require quantities to justify an on-site shotcrete batch plant, and pre-mixed bags of shotcrete have been budgeted for the project. Mobile shotcrete machines will apply shotcrete to the underground workings as needed.

16.7.3 Mucking

Mucking of all stope and development material will be by diesel-powered LHDs, and 30-tonne trucks.

Blasted material will be hauled by LHD a remuck bay where it will be stockpiled. When possible this will be direct-placed into a queuing haul truck. Stockpiled material will be loaded into a haul truck and then hauled to surface.

Management of traffic within the ramp system will be through two-way radio communications using equipment call-out (e.g., communicating ramp section, travel direction, equipment type).

16.7.3.1 Loader Productivity

Loader productivities will vary across different mining methods for each zone. Common assumptions used to determine loader productivity are as follows, with LOM average productivities summarized in Table 16-9.

- Swell factor of 1.35
- Average distance to remuck of 150 m

- Bucket fill factor of 90%
- Single bucket loading cycle time of 2.1 minutes
- Operator efficiency of 90%
- Average cycle time of 3.6 minutes for stoping and 3.1 minutes for lateral development.

Table 16-9: Life of Mine Average Loader Productivities

LHD Class (t)	Units	Lateral Development Productivity	LH Stoping Productivity
3.5	t/h	60	
6.7	t/h	121	100
10	t/h	173	

16.7.3.2 Truck Productivity

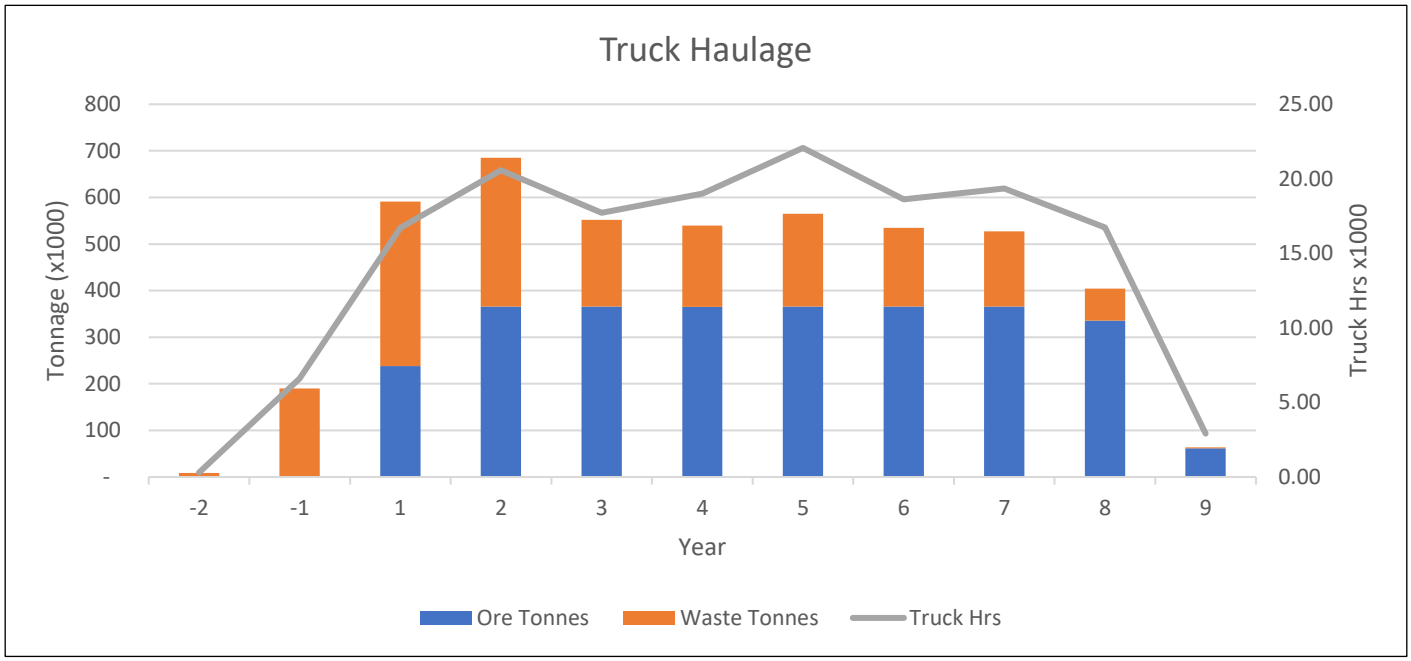
Haulage distances to the surface were estimated for every level of each zone in the mine. Truck productivities vary across the different mining configurations for each zone. Common assumptions used to determine truck productivity are as follows, with the average LOM productivities summarized in Table 16-10 and the truck haulage requirement in Figure 16-22.

- Swell factor of 1.35
- Three LHD bucket loads required to fill truck
- Truck fill factor of 78% to 90%
- Dump, manoeuvre, spot, wait, and fixed time of 2.5 minutes
- Variable loading time based on direct or remuck loading
- Operator efficiency of 90%
- Rated payload of 30 t.

Table 16-10: Life of Mine Average Truck Productivities

Zone	Units	Direct Loading to Truck	Remuck Loading to Truck
Zone C	t/h	24	31
Zone Y	t/h	31	39

Figure 16-22: Truck Haulage Requirement



Source: JDS, 2025.

16.7.4 Backfill

The principle method of backfill will be rock fill comprising of waste development rock. Rock fill will be used in both DAF and LHS stope voids where structural fill is not required. Paste fill will be used in DAF sill stopes, DAF primary stopes with secondary stope adjacencies, and longhole retreat stopes.

Life of mine backfill distribution by tonnage is estimated as follows:

- Rock Fill – Direct dumped from underground 39%
- Rock Fill – hauled from surface stockpile 28%
- Paste Fill 33%.

16.8 Backfill Plant

Mined voids are proposed to be filled with three types of fill. Waste rock backfill, low strength paste and high strength paste. Uphole longhole stopes are planned to be left open and account for less than 1% of the total void created.

Sixty seven percent (67%) of fill will be waste rock backfill, this will be sourced directly from underground waste development rounds or trucked from surface from the waste rock storage facility. Trucks will haul to the level, dump in the remuck and the LHD fleet will haul from the remuck to the stope and dump.

The paste backfill system will use classified mill tailings mixed with binder and accounts for 33% of total backfill.

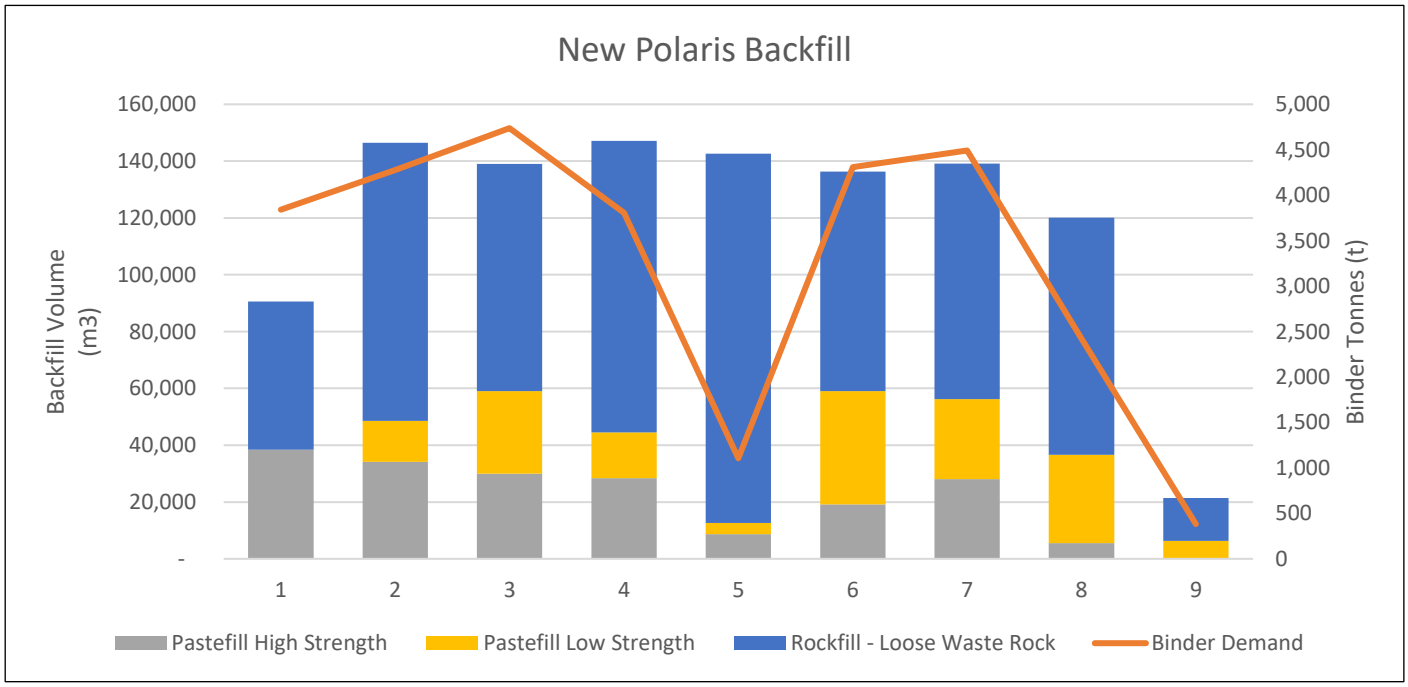
Dry bulk density of 1.41 t/m³, 3% and 5% binder content for low and high strength paste respectively are the assumed parameters. Tailings not used for backfill will be delivered for disposal at the surface tailings storage facility. It is estimated that the paste backfill will utilize 510 kt of tailings.

16.8.1 Backfill Production

Paste will be produced at the Backfill plant which will be directly attached to the mill building. The nominal placement rate for structural backfill is 27 m³/h.

Binder (slag: cement blend) will be supplied to site in bulk bags and stored on site. For day-to-day backfilling operations, bags will be transferred to a storage shed with a debagging unit adjacent to the backfill plant. The binder will be augured into the plant binder silo and metered into the backfill stream.

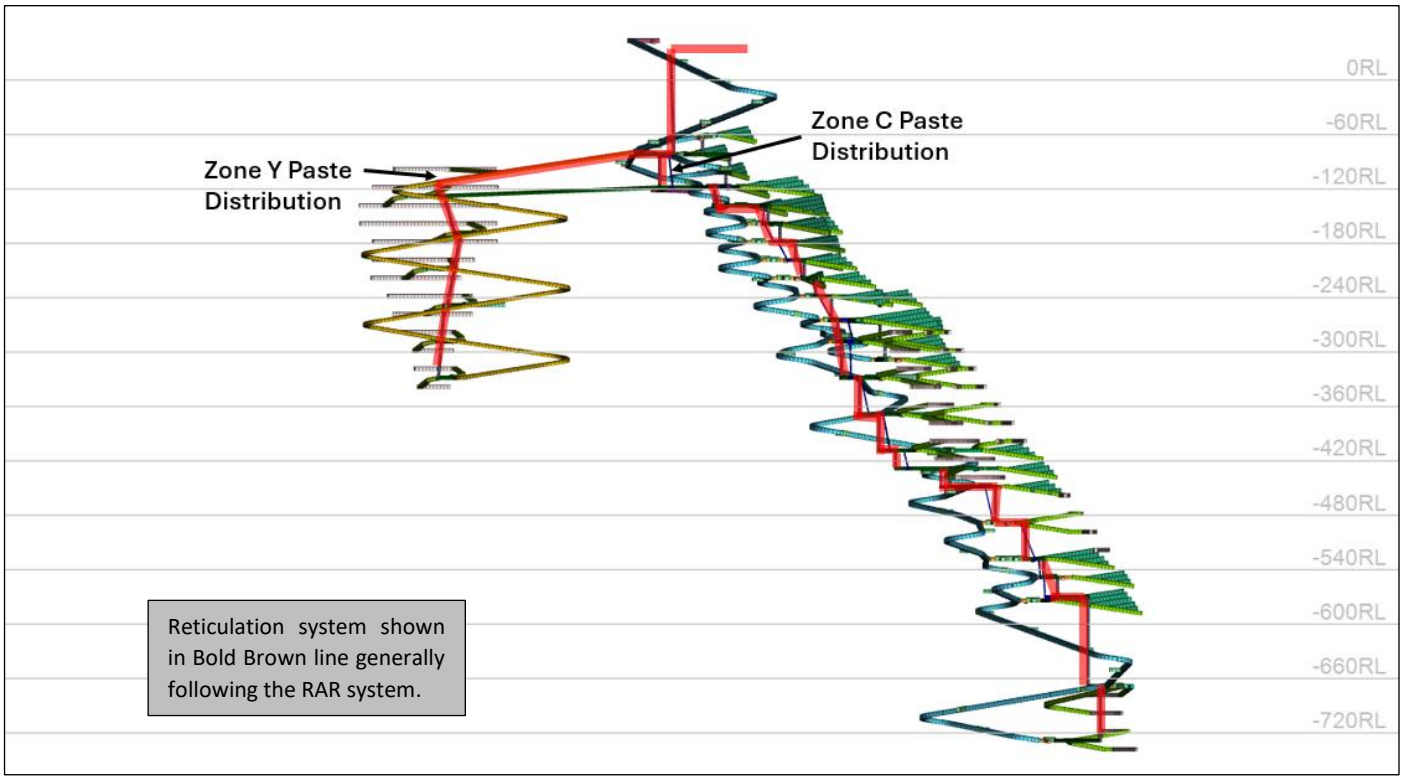
Figure 16-23: New Polaris Annual Backfill Volumes



Source: JDS, 2025.

The paste backfill system will leverage gravity and utilize booster pumps where required for material transport. The backfill piping will generally follow the ventilation return air network and then be distributed on the level. From the surface, a single pipe servicing both the Zone C and Zone Y reticulation network will be installed in the ventilation raise. Where it splits into the respective zones. The C zone follows the return air network of raises to the bottom of the mine, the Y zone network follows the access ramp across to Y zone where it then is installed in the raises.

Figure 16-24: Paste Backfill Reticulation System (Looking East)



Source: JDS, 2025.

16.8.2 Fill Placement

Paste will be placed in discrete runs of set times depending on stope dimensions, with filling being suspended after each run to enable drainage to take place. Placement in drives will require either rock bunding or structural barricades with some drainage in built to retain the fill. Longhole retreat stopes, DAF sill levels and DAF primary drifts will be filled with paste.

16.8.3 Backfill Quality Control

The backfill plant lab will contain a sea-can container-type test facility consisting of a UCS strength tester, scales and computer for testing equipment control and data collection. One half of the container will be set up to store paste samples collected each shift in a climate-controlled environment for strength testing at 7-, 14-, and 28-day curing times.

16.9 Mine Personnel

16.9.1 UG Personnel

The New Polaris underground mine will operate 360 days per year, 20 hours per day, seven days per week, on two shifts per day, 10 hours per shift. There will be a total of four crews. The mine will be a fly-in-fly-out (FIFO) operation with crew changeovers that are planned on the rotating two-weeks-on two-weeks-off roster.

Mining in the first twenty-five months will be undertaken by a contractor that provides and is responsible for all safety, mining supervision, drilling and blasting, ground support, load and haul, maintenance and mine services installation. Canagold will provide technical services, and key staff positions such as mine and site management during the tenure of the contractor. Canagold personnel will continue to work through to the end of the mine life.

At the start of Year 2, a transition from contractor to owner operated mining occurs.

Underground personnel requirements are summarized in Table 16-11.

Table 16-11: Underground Mining Personnel

	Max	Y -2	Y -1	Y 1	Y 2	Y 3	Y 4	Y 5	Y 6	Y 7	Y 8	Y 9
Mine General												
General Mine Manager	1	1	1	1	1	1	1	1	1	1	1	1
Administrator/Clerk	2	2	2	2	2	2	2	2	2	2	2	2
HSE Manager	1	1	1	1	1	1	1	1	1	1	1	1
Mine Rescue/Safety Officer/Trainer	2	2	2	2	2	2	2	2	2	2	2	2
Mining Superintendent	2	2	2	2	2	2	2	2	2	2	2	2
Mine Supervisor/Shift Boss	2	2	2	2	2	2	2	2	2	2	2	2
Maintenance Superintendent	1	1	1	1	1	1	1	1	1	1	1	1
Maintenance Supervisor/Shift Boss	2	2	2	2	2	2	2	2	2	2	2	2
Construction Superintendent	1	1	1	1	1	1	1	1	1	1	1	1
Mechanical Foreman	2	2	2	2	2	2	2	2	2	2	2	2
Electrical Foreman	2	2	2	2	2	2	2	2	2	2	2	2
IT Manager	1	1	1	1	1	1	1	1	1	1	1	1
IT Tech	1	1	1	1	1	1	1	1	1	1	1	1
Stores Clerk	2	2	2	2	2	2	2	2	2	2	2	2

	Max	Y -2	Y -1	Y 1	Y 2	Y 3	Y 4	Y 5	Y 6	Y 7	Y 8	Y 9
Drill & Blast												
Jumbo Operator	20	4	4	20	20	16	12	16	16	12	12	4
Production Drill Operator	8	4	4	4	4	4	4	4	4	8	8	8
Blaster	8	4	4	8	8	8	8	8	8	4	4	4
Drill & Blast Nipper	20	12	12	20	20	20	20	20	20	20	20	16
Load and Haul												
LHD Operator	28	4	12	28	28	24	24	24	20	20	20	12
UG Truck Driver	20	4	12	20	20	20	20	20	20	20	20	20
Surface Loader	4			4	4	4	4	4	4	4	4	4
Support Services												
Electrician	2	2	2	2	2	2	2	2	2	2	2	2
Development Service	12	6	12	12	12	12	12	12	6	6	6	6
Ground Support/Bolter/Shotcrete	15	8	12	24	24	20	20	20	16	16	12	12
Utility Vehicle Operator	4	4	4	4	4	4	4	4	4	4	4	4
Services Nipper	20	12	12	20	20	20	20	20	20	20	20	16
Backfill Mine Operations												
Backfill Plant Operator	4			4	4	4	4	4	4	4	4	4
Bulkhead / Backfill Miner	4			4	4	4	4	4	4	4	4	4
LHD Operator - Rammer Jammer	4			4	4	4	4	4	4	4	4	0
Mine Maintenance												
Maintenance Planner	2	2	2	2	2	2	2	2	2	2	2	2
Lube/PM Mechanic	2	0	0	2	2	2	2	2	2	2	2	0
HD Mechanic	4	0	2	4	4	4	4	4	4	4	2	2
Drill Mechanic	2	2	2	2	2	2	2	2	2	2	2	2
Overhaul Mechanic	4	0	0	0	4	4	4	4	4	4	0	0
Welder	2	2	2	2	2	2	2	2	2	2	2	2
Tyreman	2	2	2	2	2	2	2	2	2	2	2	2
Apprentice	4	4	4	4	4	4	4	4	4	4	4	4
Dry/Lapman/Bitman	4	4	4	4	4	4	4	4	4	4	4	4
Technical Services												
Technical Services Superintendent	1	1	1	1	1	1	1	1	1	1	1	1
Chief Mine Engineer	1	1	1	1	1	1	1	1	1	1	1	1
Senior Mine Engineer	1	1	1	1	1	1	1	1	1	1	1	1
Mine Planning Engineer	2	2	2	2	2	2	2	2	2	2	2	2
Mine Ventilation/Project Engineer	2	2	2	2	2	2	2	2	2	2	2	2
Geotechnical Engineer	2	2	2	2	2	2	2	2	2	2	2	2
Mine Technician	2	2	2	2	2	2	2	2	2	2	2	2
Surveyor	2	2	2	2	2	2	2	2	2	2	2	2
Surveyor Helper	2	2	2	2	2	2	2	2	2	2	2	2
Chief Geologist	1	1	1	1	1	1	1	1	1	1	1	1
Production Geologist	2	0	0	2	2	2	2	2	2	2	2	2
Geotechnical Technician/Sampler	4	0	0	4	4	4	4	4	4	4	4	4
Total Mining Operations	**226**	**96**	**124**	**222**	**226**	**214**	**210**	**214**	**200**	**196**	**186**	**156**

Source: JDS, 2025.

16.10 Mine Equipment

16.10.1 Mobile Equipment Requirements

A contractor will supply the personnel and mobile equipment to execute the initial capital development program and initial production mining for the first 30 months of the mine life. The only exception to this is the provision of light vehicles for the owner's staff. As part of the contractor to owner (Canagold) mine operator transition, owner (Canagold) will purchase all underground mobile equipment from the contractor. This mining equipment is required to maintain the steady-state production schedule and associated development. The procurement of additional equipment will also occur during the operating life of the mine. Table 16-12 shows the total mobile equipment numbers and types required to meet the development and production schedule.

Table 16-12: New Polaris Mine Equipment List

UG Mobile Equipment	Max	Y -2	Y -1	Y 1	Y 2	Y 3	Y 4	Y 5	Y 6	Y 7	Y 8	Y 9
Trucks	5	1	3	5	5	5	5	5	5	5	5	5
Scoops	8	1	3	8	8	7	7	7	7	7	7	7
Jumbos	5	1	1	5	5	4	3	4	4	4	4	4
Bolters	4	1	2	4	4	3	3	3	3	3	3	3
Longhole Drills	2	1	1	1	2	2	1	1	1	1	1	1
Scissor	2	1	2	2	2	2	2	2	2	2	2	2
Shotcrete	2	1	1	2	2	2	2	2	2	2	2	2
Explosive Chargers	2	1	1	2	2	2	2	2	2	2	2	2
Total	**30**	**8**	**14**	**29**	**30**	**27**	**25**	**26**	**26**	**26**	**26**	**26**

Source: JDS, 2025.

16.11 Development Schedule

The LOM development schedule presented in Table 16-13 is based on the parameters listed in Table 16-14. The mine schedule was created using Deswik Sched software. Development was sequenced to minimize preproduction duration and at the same time support a reasonable production ramp-up. Development constraints were utilized in both capital and operating headings to maximise utilization of equipment and delay non-essential development until it is required.

Table 16-13: Development Mining Advance Rate Assumptions

Aspect	Units	Assumption
Ramps and loadout bays	m/mo	150
Raises	m/mo	100
Larger drive profiles (e.g. laydowns, workshops)	m. equivalent/mo	150
Drift and Fill & LH Stope Sill Development	m/mo	105
All other development	m/mo	100

Table 16-14: Life of Mine Development Schedule

Development Type	Units	Total	Y -1	Y 1	Y 2	Y 3	Y 4	Y 5	Y 6	Y 7	Y 8	Y 9
Lateral Development												
Capital - Ramp	m	8,002	98	1,447	1,346	1,106	565	662	935	728	828	286
Capital - Infrastructure	m	6,411	5	884	2,171	982	607	516	256	470	372	148
Operating	m	53,032	0	0	6,564	10,283	6,751	6,791	8,920	6,160	4,336	3,226
Total	**m**	**67,445**	**103**	**2,331**	**10,082**	**12,370**	**7,924**	**7,969**	**10,111**	**7,358**	**5,536**	**3,660**
Vertical development												
Alimak	m	455	110	0	0	0	0	96	56	114	79	0
Raisebore	m	505	108	0	158	79	160	0	0	0	0	0
Drop raise	m	659	36	307	124	80	38	40	0	35	0	0
Total	**m**	**1,619**	**254**	**307**	**282**	**159**	**198**	**136**	**56**	**149**	**79**	**-**

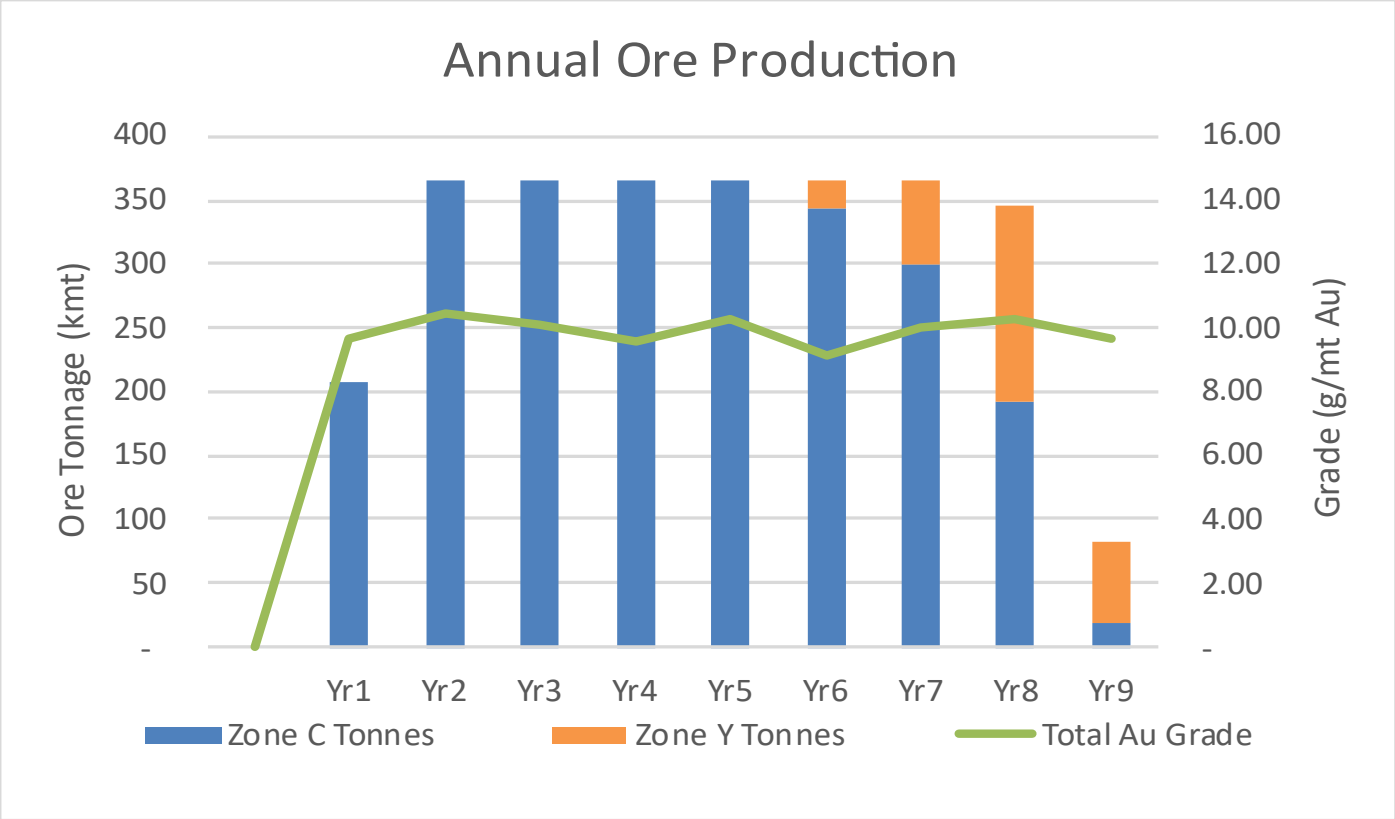
16.12 Production Schedule

Production stopes are sequenced in such a way to bring high value ounces online early in the schedule, taking into account both cost of accessing and cost of extracting those ounces. The projected LOM underground production schedule is summarized in Table 16-15 and Figure 16-25.

Table 16-15: Life of Mine Production Schedule

Aspect	Total	Y 1	Y 2	Y 3	Y 4	Y 5	Y 6	Y 7	Y 8	Y 9
Ore diluted tonnage (t)	2,830,172	207,281	365,998	365,999	364,900	365,998	365,994	365,997	345,251	82,754
Grade (g/t Au)	9.94	9.69	10.49	10.06	9.58	10.26	9.13	10.00	10.28	9.64
Contained metal (koz Au)	904	65	123	118	112	121	107	118	114	26
Antimony (ppm)	1,828	246	1,628	4,731	1,425	1,381	922	2,260	1,751	-
Arsenic (ppm)	10,960	9,469	10,339	9,972	11,888	11,673	11,291	11,061	11,153	11,841

Figure 16-25: LOM Underground Mine Production Schedule

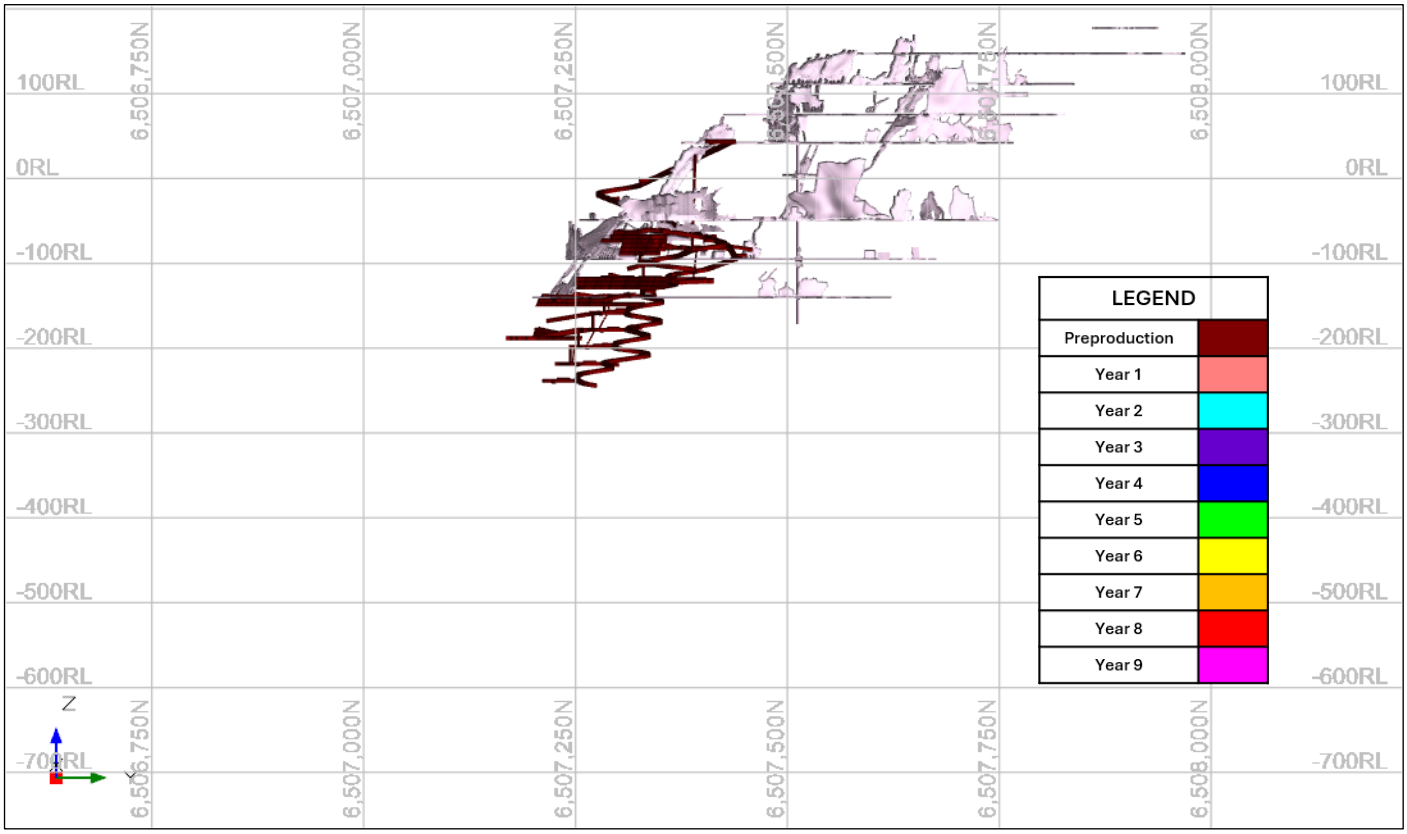


Source: JDS, 2025.

16.13 Underground Schedule

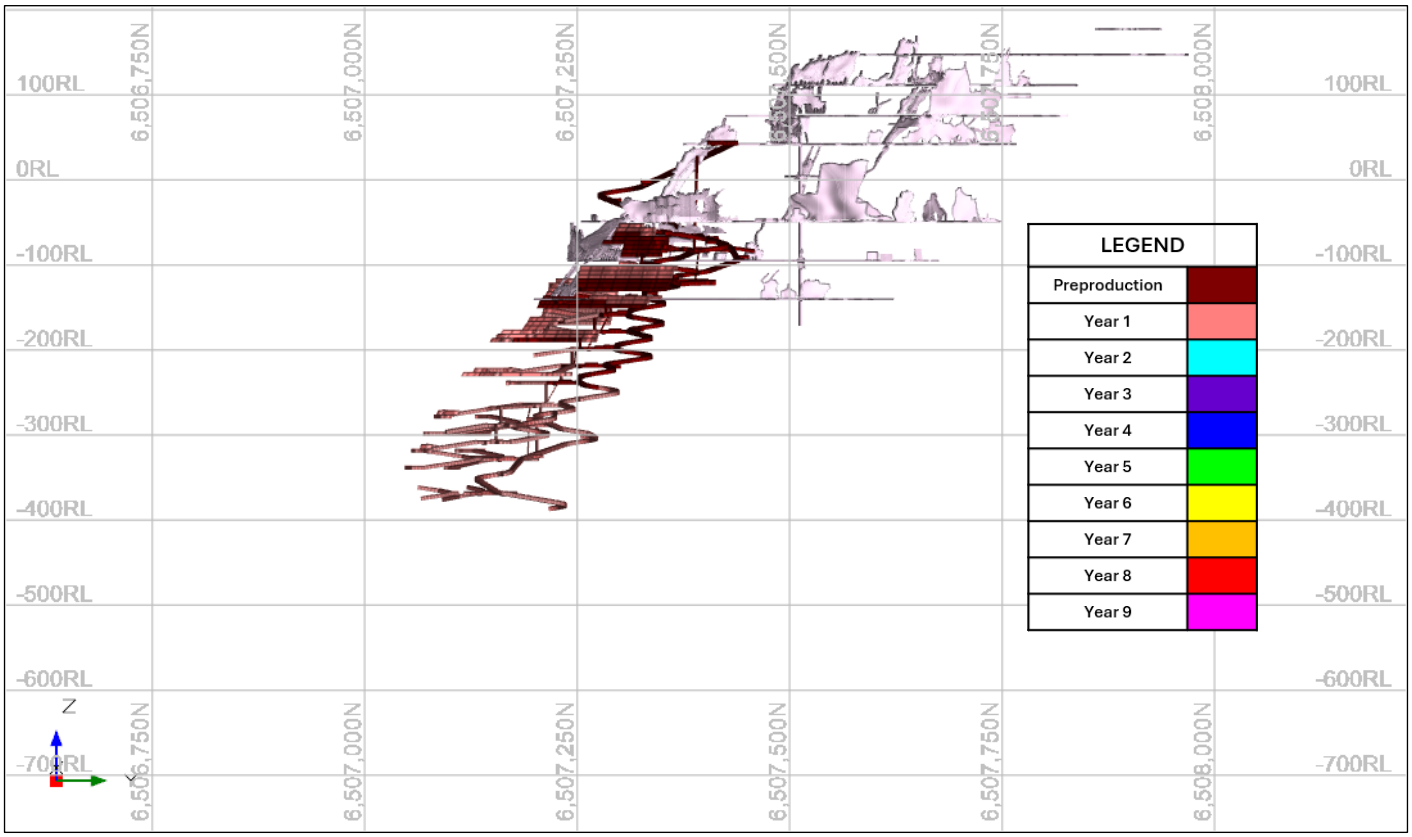
Figure 16-26 through Figure 16-34 show the overall mine sequence by years for the life of mine.

Figure 16-26: Long Section View of the Development Completed Pre-production



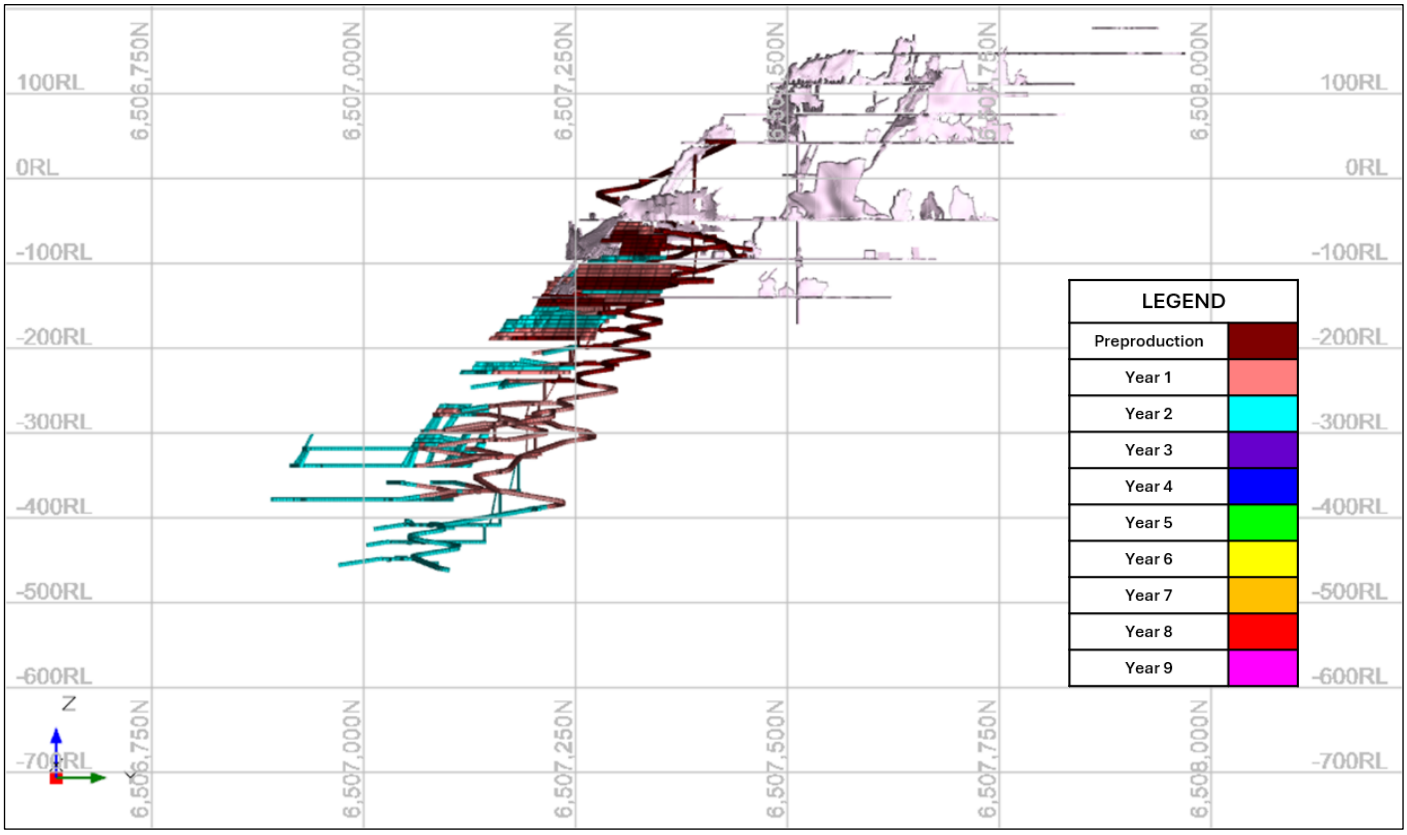
Source: JDS, 2025.

Figure 16-27: Long Section View of the Development Completed Year 1



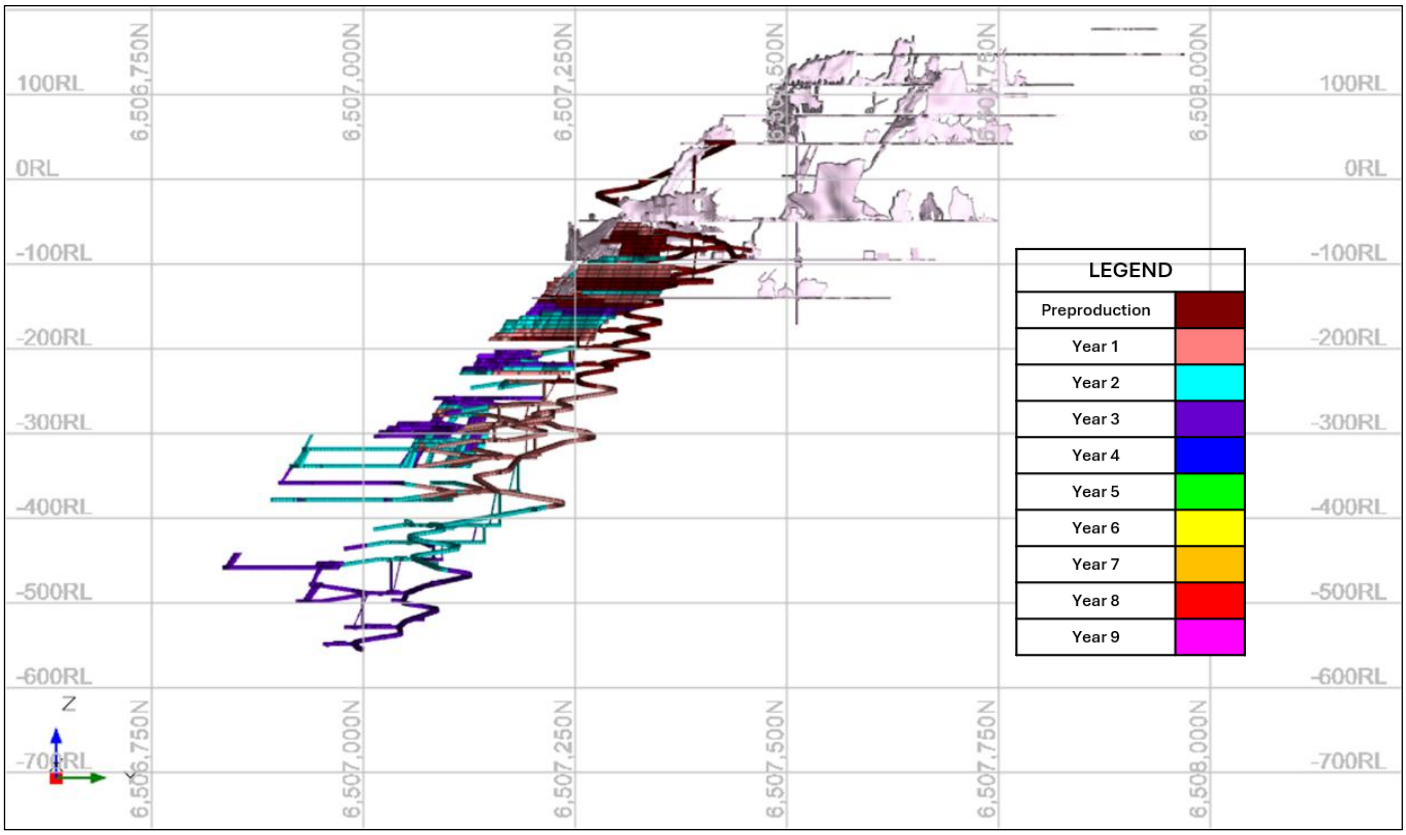
Source: JDS, 2025.

Figure 16-28: Long Section View of the Development Completed Year 2



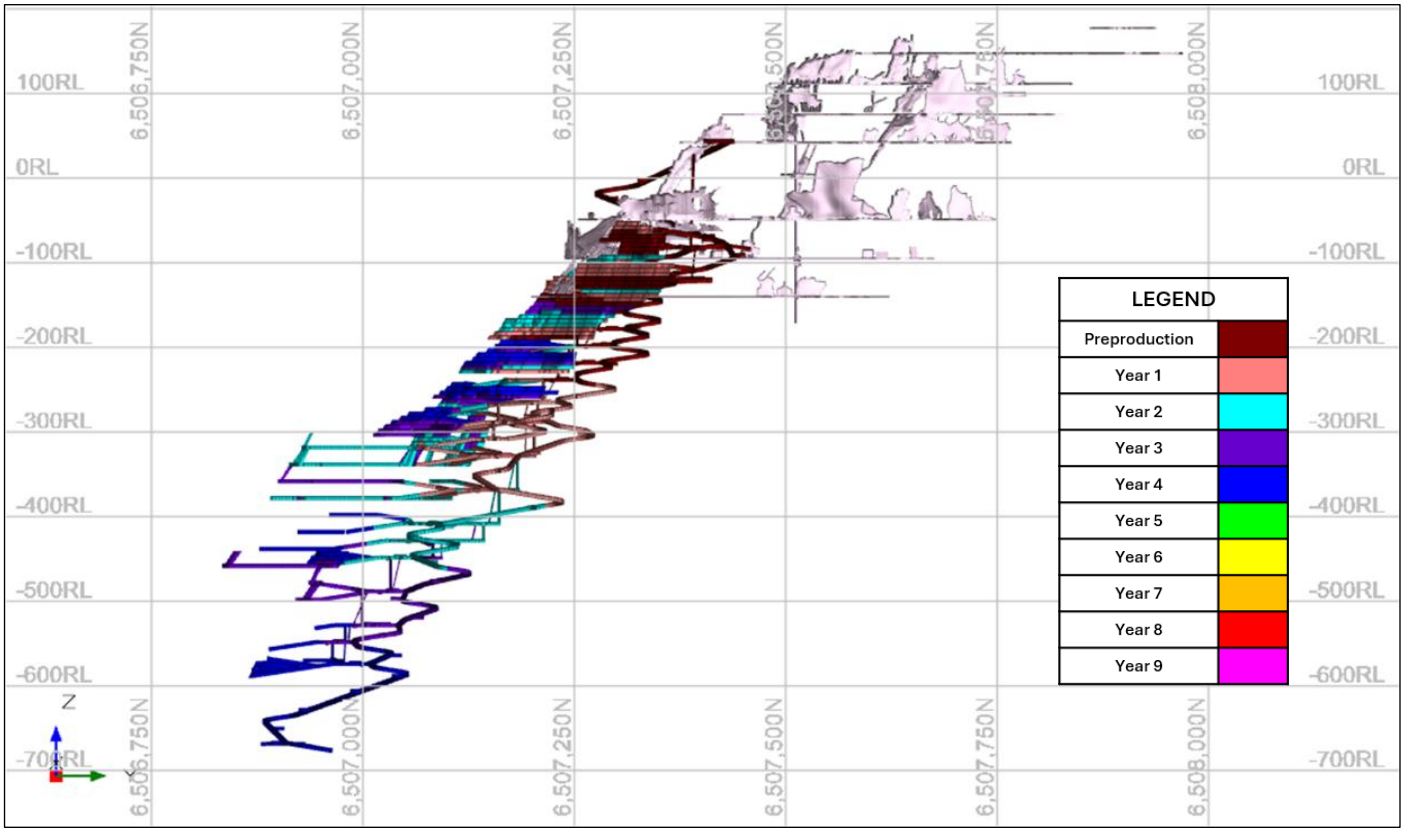
Source: JDS, 2025.

Figure 16-29: Long Section View of the Development Completed Year 3



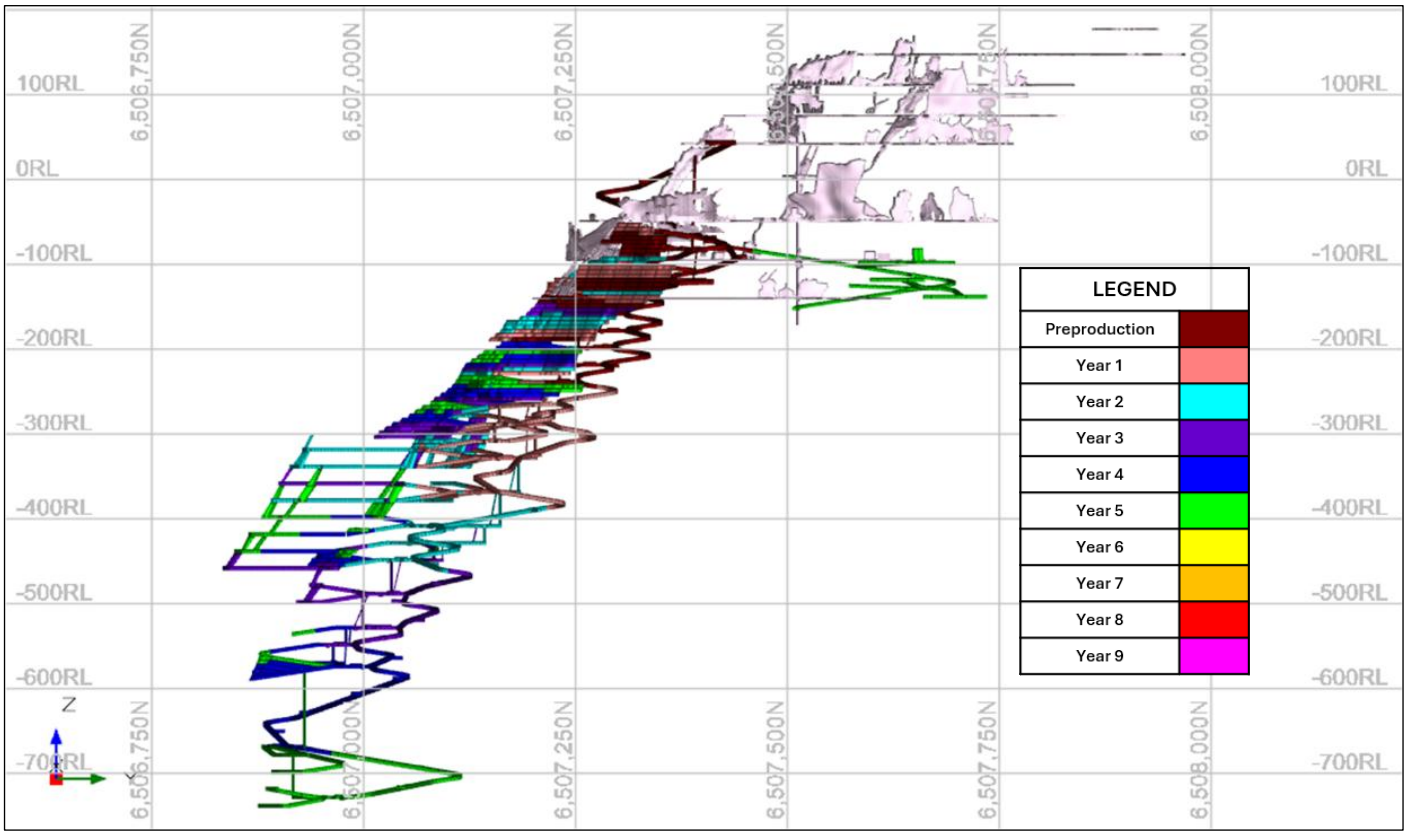
Source: JDS, 2025.

Figure 16-30: Long Section View of the Development Completed Year 4



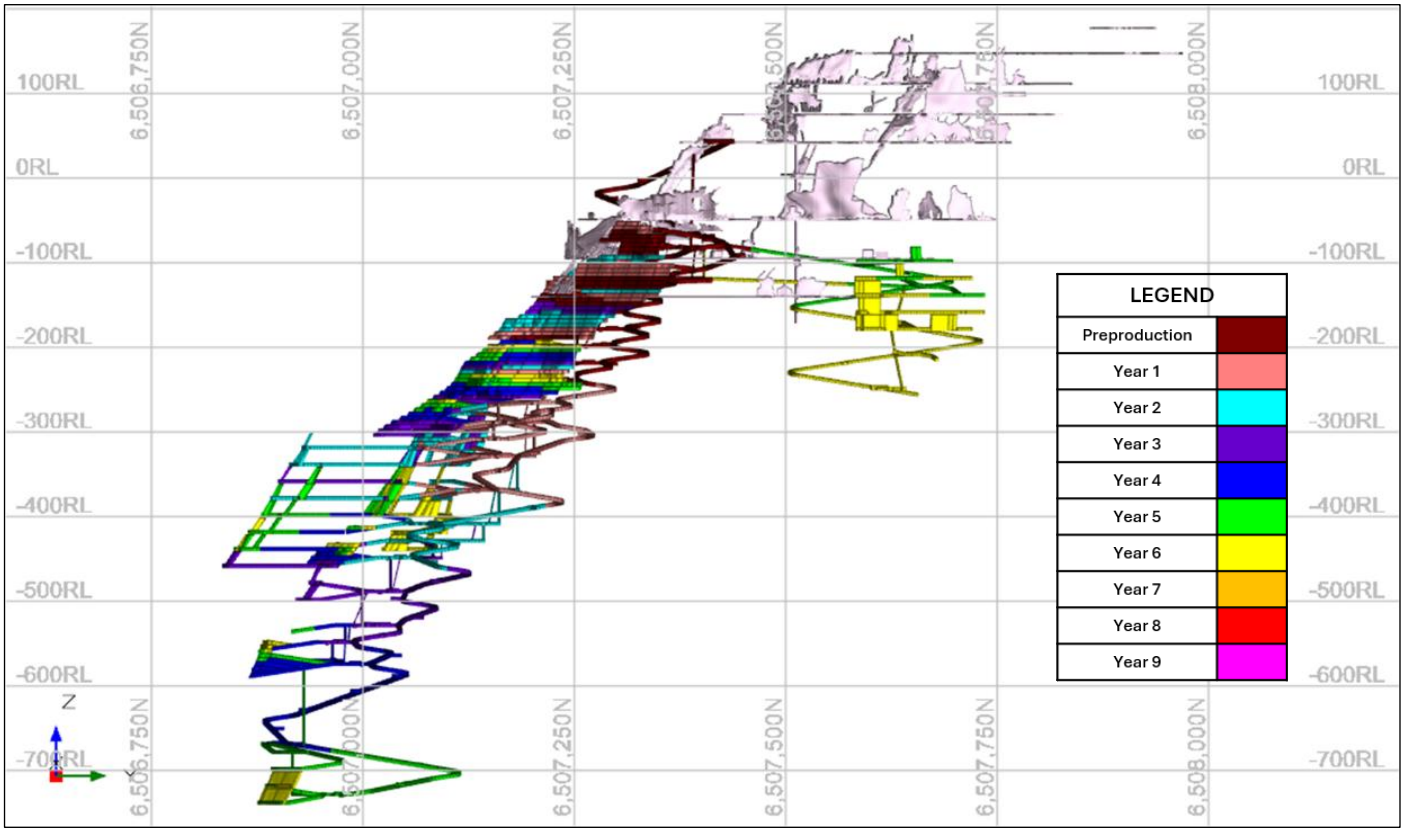
Source: JDS, 2025.

Figure 16-31: Long Section View of the Development Completed Year 5



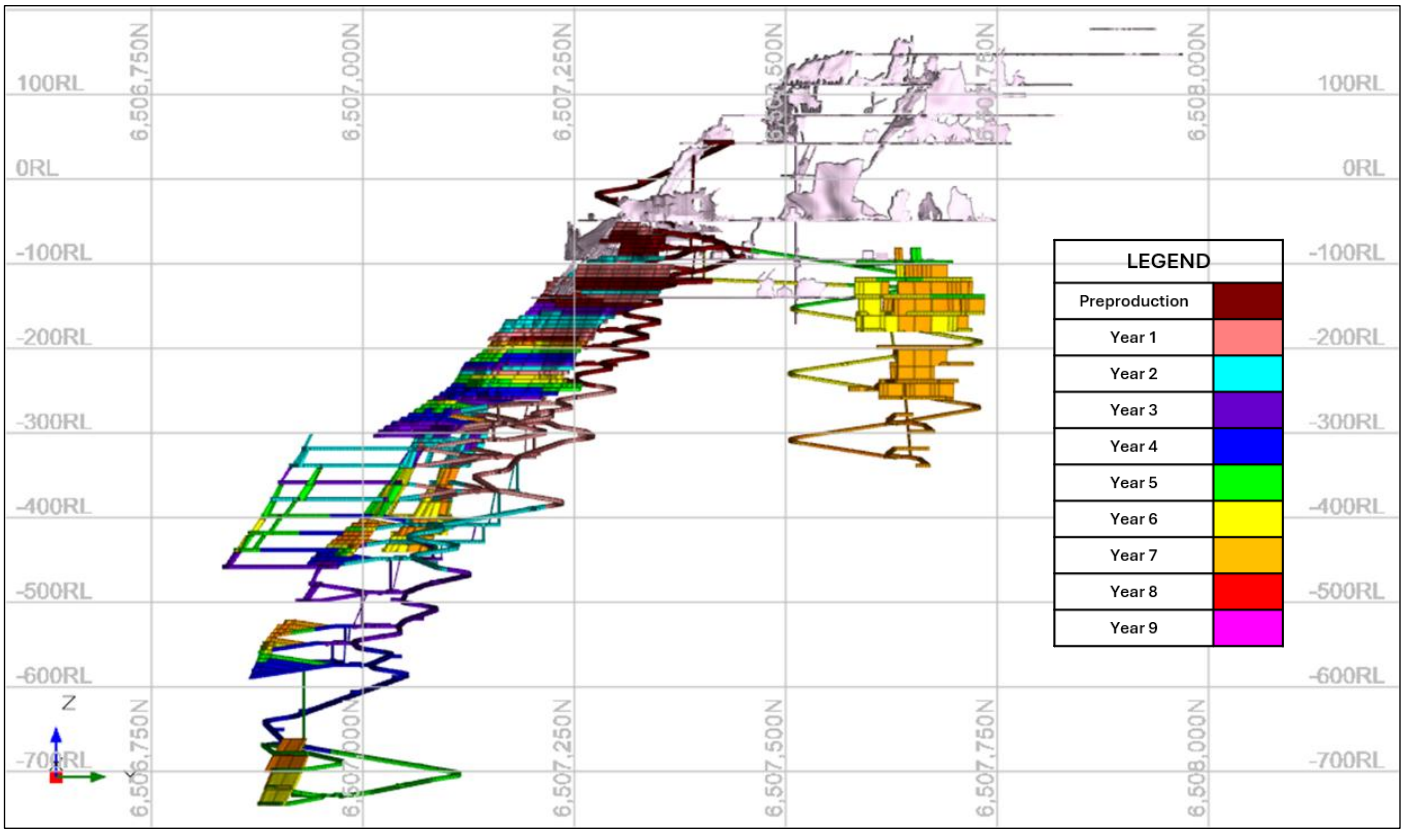
Source: JDS, 2025.

Figure 16-32: Long Section View of the Development Completed Year 6



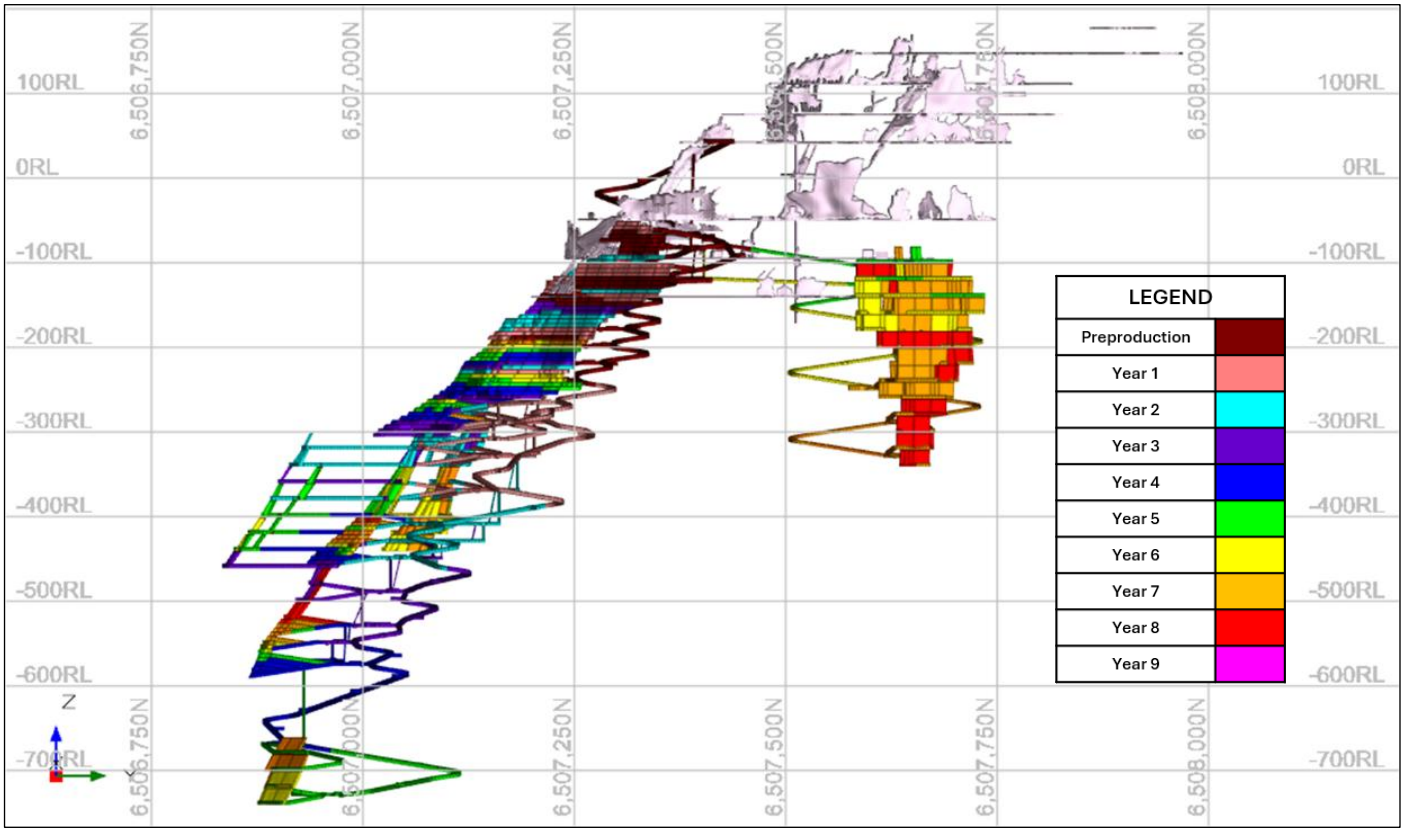
Source: JDS, 2025.

Figure 16-33: Long Section View of the Development Completed Year 7



Source: JDS, 2025.

Figure 16-34: Long Section View of the Development Completed Year 8 & 9



Source: JDS, 2025.

17 RECOVERY METHODS

17.1 Overview

The proposed process plant for the New Polaris Project will produce a gold concentrate from material sourced from the underground mine. The selected flowsheet utilizes three stages of crushing, consisting of a primary jaw crusher followed by a secondary and a tertiary cone crusher. Crushed ore will be stored in a silo.

Ore is reclaimed to a grinding circuit consisting of a single ball mill operating in closed circuit with a cyclone cluster. Ground ore will report to a flotation circuit designed to produce a gold concentrate with rougher and two stages of cleaner flotation.

The flotation circuit begins with the rougher conditioning and flotation cells, with the rougher concentrate reporting to the first cleaner cells. Tailings from the first cleaner will report to the first cleaner scavenger to increase gold recovery. The concentrate of the first cleaner and first cleaner scavenger cells reports to the second cleaner while the second cleaner tailings are recirculated to the first cleaner.

The final concentrate from the flotation circuit will report to a dedicated dewatering circuit that includes a thickener and filter press. The filtered concentrate will report to a dryer before packaging for transportation off-site.

Tailings from the flotation circuit will report to either the tailings thickener and filter press or to the tailings cyclone at the hydraulic backfill plant. The tailings cyclone overflow will be sent back to the tailings thickener. Filtered tailings from the dewatering circuit will be disposed of in the dry stack facility.

17.2 Process Flowsheet

The process design considers the following unit operations and circuits:

- Three stages of crushing
- Crushed ore silo and reclaim feeder
- Ball mill with trommel in closed circuit with a cyclone cluster
- Rougher flotation
- First cleaner and cleaner scavenger flotation
- Second cleaner flotation
- Final concentrate thickening, filtering, drying, bagging, and loadout
- Tailings cyclone and hydraulic backfill plant
- Tailings thickening, filtering, and dry stacking
- Reagent mixing and distribution
- Air services, water services, and utilities

The overall process flow diagram for New Polaris is provided in Figure 17-1 and described in the sections below. The overall plant layout is shown in Figure 17-2.

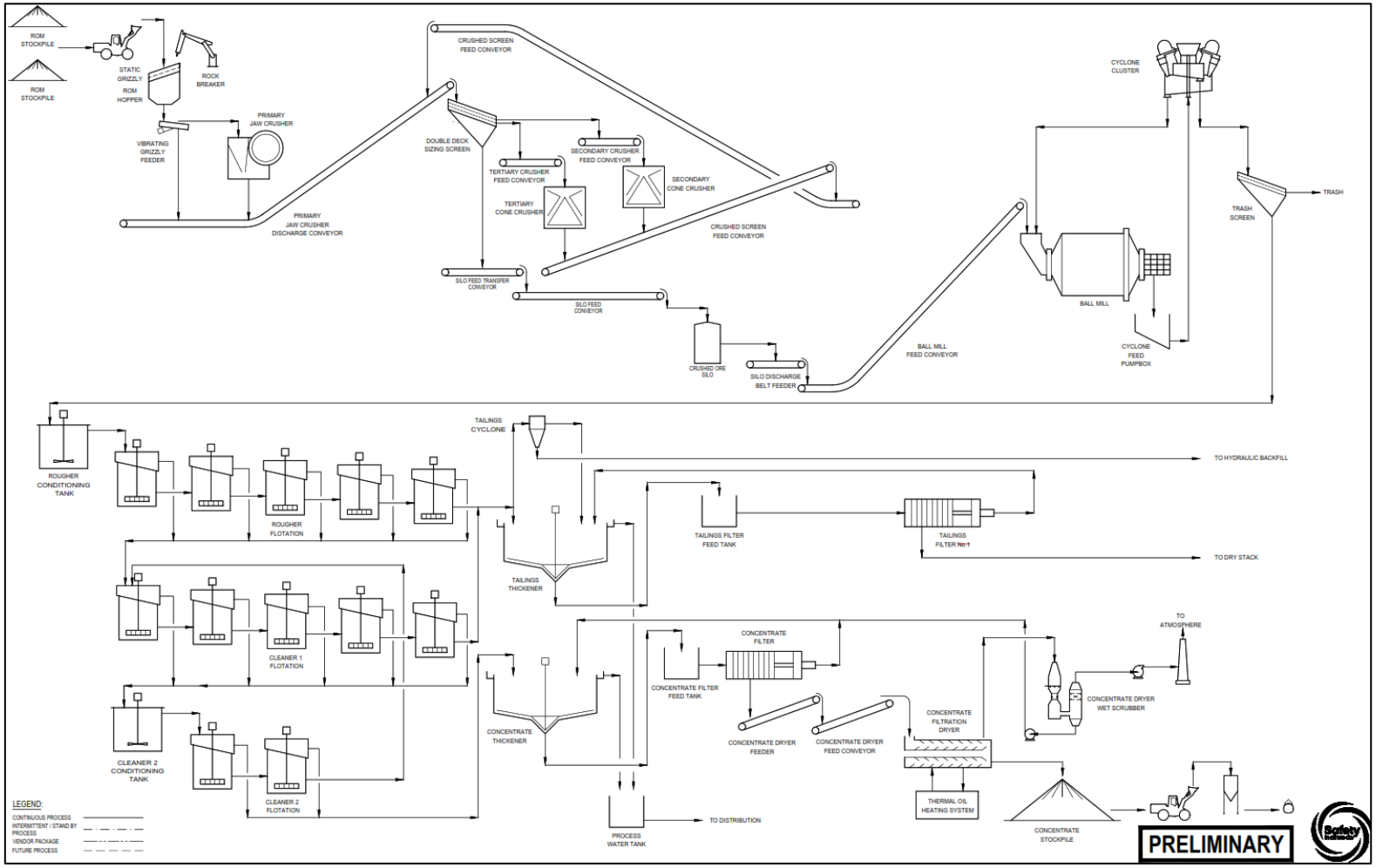
17.3 Plant Design

Key process design criteria for are listed in Table 17-1.

Table 17-1: Process Design Criteria

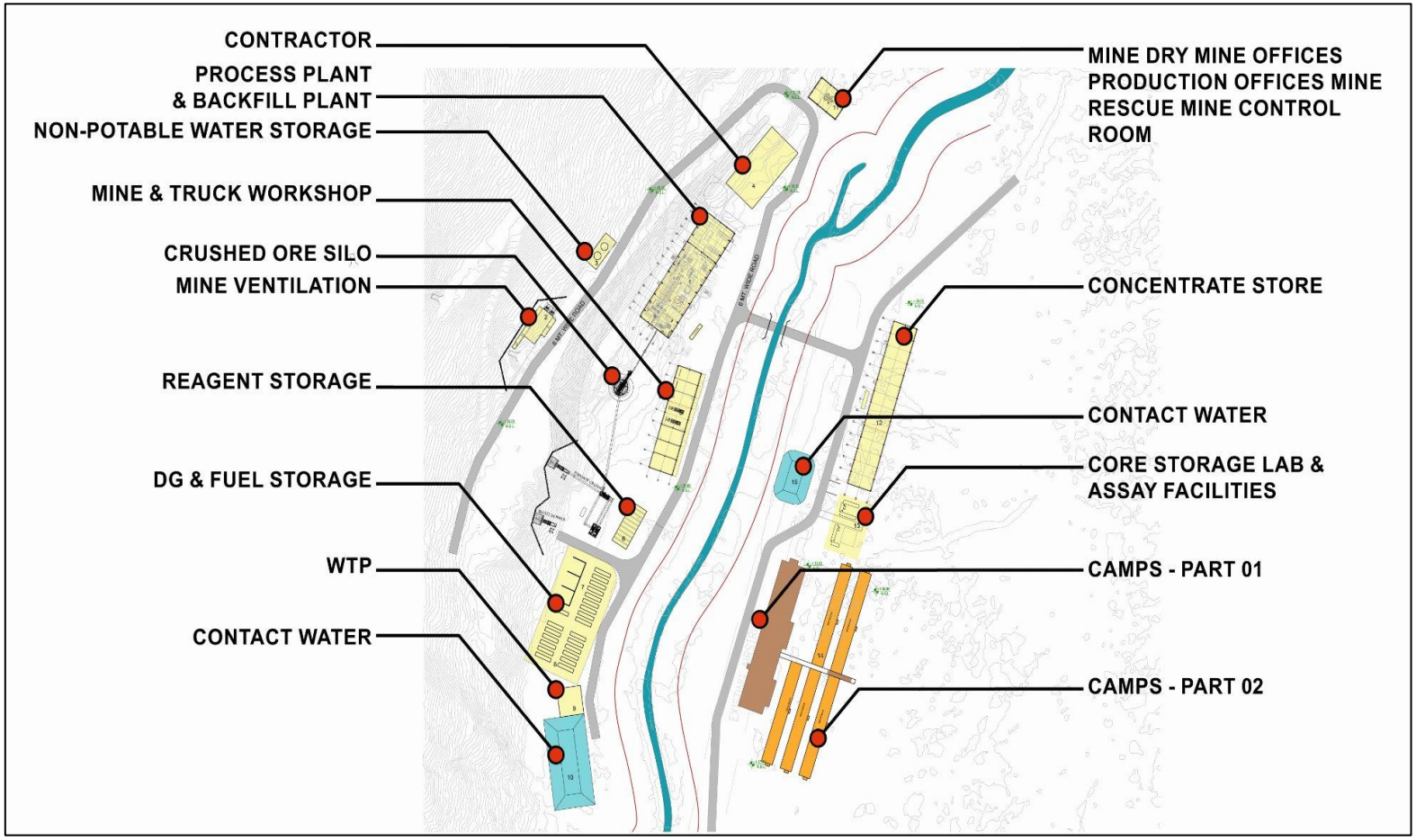
Criteria	Unit	Value
Annual Throughput	t/a	348
Daily Throughput	t/d	955
Operating Availability – Three Stage Crushing	%	66
Operating Availability – Grinding and Flotation	%	92
Operating Availability – Filtration	%	85
Average Head Grade – Au	g/t	9.94
Bond Rod Mill Work Index (RWi) – Design	kWh/t	19.2
Bond Ball Mill Work Index (BWi) – Design	kWh/t	21.1
Axb - Design	-	30.8
Bond Abrasion Index (Ai)	g	0.098
Crushing Circuit Feed Size, F100	mm	400
Crushing Circuit Feed Size, F80	mm	211
Crushing Circuit Product Size, 80% Passing	mm	13
Grinding Circuit Product Size, 80% Passing	µm	75
Rougher Flotation Stage Recovery – Au	% Plant Feed	95.7
First Cleaner and Scavenger Flotation Stage Recovery – Au	% Plant Feed	92.8
Second Cleaner Flotation Stage Recovery – Au	% Plant Feed	89.9
Flotation Recovery – Au	% Plant Feed	89.9
Rougher Flotation Mass Recovery	% Plant Feed	26.0
First Cleaner and Scavenger Flotation Mass Recovery	% Plant Feed	14.0
Second Cleaner Flotation Mass Recovery	% Plant Feed	6.9
Flotation Mass Recovery	% Plant Feed	6.9
Concentrate Thickener Unit Area Settling Rate	$t/m^2/h$	0.5
Tailings Thickener Unit Area Settling Rate	$t/m^2/h$	1.0
Thickened Concentrate, Pulp Density	%w/w	65
Thickened Tailings, Pulp Density	%w/w	50
Concentrate Filter, Design Filtration Rate	$t/m^2/h$	0.40
Tailings Filter, Design Filtration Rate	$t/m^2/h$	0.20
Concentrate Filter Cake, Moisture Content	%w/w	15
Tailings Filter Cake, Moisture Content	%w/w	15
Final Concentrate Grade, Au	g/t	128

Figure 17-1: Process Flowsheet



Source: Ausenco, 2025.

Figure 17-2: Processing Plant Overall Layout, including site G&A



Source: Ausenco, 2025.

17.3.1 Three Stage Crushing and Crushed Ore Silo

The crushing circuit is designed for an annual operating time of 5,782 hours or 66% availability.

The ROM ore will be crushed through a modular three stage crushing system designed to reduce the feed size from a F_{80} of 211 mm to a P_{80} of 13 mm. Ore will be directly tipped into the primary crusher blended ore bin with equipped grizzly. Oversized material unable to pass through the grizzly will be broken down using a rock breaker. Ore passing through the grizzly will feed the primary jaw crusher. A trash magnet located on the jaw crusher discharge conveyor will remove any tramp metal prior to screening. Downstream of the magnet, the jaw crusher discharge conveyor is equipped with a metal detector, which will automatically stop the conveyor if triggered. The crushed ore will then be classified by a double-deck screen with 30 mm apertures on the upper deck and 13 mm on the lower deck. Oversize material from the upper deck will feed the secondary cone crusher, while mid-size material from the lower deck will feed the tertiary cone crusher. The discharge from both cone crushers will be recirculated back to the double-deck screen for further classification. Material passing through the lower deck (P_{80} of 13 mm) will be conveyed to the crushed ore silo via the silo feed conveyor.

The silo will have sufficient capacity to store up to 24 hours of crushed ore, accommodating the expected lower availability of the crushing circuit relative to the grinding circuit. The crushed ore will be reclaimed by a belt feeder and feed the grinding circuit via the ball mill feed conveyor.

The area includes the following major equipment and facilities:

- modular crushing system consisting of:
  - primary jaw crusher, 90 kW
  - vibrating double-deck screen
  - secondary cone crusher, 90 kW
  - tertiary cone crusher, 90 kW
  - rock breaker
  - primary jaw crusher discharge conveyor, 800 mm W x 45 m L, 30 kW
  - double-deck sizing screen (30 mm upper deck and 13 mm lower deck)
  - secondary crusher feed conveyor, 650 mm W x 35 m L, 7.5 kW
  - tertiary crusher feed conveyor, 650 mm W x 35m L, 15 kW
  - crushed screen feed conveyor No.1, 650 mm W x 20 m L, 11 kW
  - crushed screen feed conveyor No.2, 650 mm W x 40 m L, 22 kW
  - silo feed transfer conveyor, 650 mm W x 40 m L, 15 kW
  - silo feed conveyor, 650 mm W x 15.5 m L, 22 kW

- crushed ore silo (24 hour live capacity), Carbon Steel, 5m Diameter

17.3.2 Primary Grinding

The grinding circuit is designed for an annual operating time of 8,059 hours, corresponding to 92% availability.

Primary grinding will consist of a single ball mill operating in closed circuit with a primary cyclone cluster. Crushed materials from the crushing circuit will be conveyed to the ball mill via the ball mill feed conveyor. Process water will be added to the ball mill to achieve the designed discharge pulp density of 75% w/w. Ground slurry will be discharged through an attached trommel. Oversize(scats) from trommel is directed to a scats bunker, which will be periodically emptied. Trommel undersize (mill discharge) report to the cyclone feed pumpbox.

The ball mill discharge will be pumped from the cyclone feed pumpbox to the primary cyclone cluster. The cyclone overflow, containing particles 80% passing 75 µm, will pass through a vibrating trash screen before feeding the flotation circuit. The cyclone overflow is designed to operate at 35% w/w solids. The cyclone underflow, containing coarser material, will return by gravity to the ball mill feed chute for further grinding.

Forged steel grinding media will be used in the ball mill and will be charged to the ball mill via a kibble system.

This area includes the following major equipment and facilities:

- ball mill, 3.6m D x 6.0m EGL, grate discharged, 1.6 MW
- cyclone cluster
- cyclone overflow trash screen, 7.5 kW
- process plant overhead crane, 25 t
- cyclone feed pump box and pump, 45 kW

17.3.3 Flotation

The flotation circuit consists of a series of rougher flotation cells, followed by two stages of cleaning. Concentrate from the second cleaner will be sent to the final concentrate dewatering.

17.3.3.1 Rougher Flotation

Discharge from the cyclone overflow trash screen will first report to the conditioning tank with a retention time of 10 minutes. Sodium sulphide, PAX, MIBC, and copper sulphate will be added to the conditioning tank. Post conditioning, the material will be gravity fed to a series of five conventional forced air flotation cells in a 1-2-2 configuration. The concentrate from each rougher cell will be collected in the rougher concentrate pump box through a launder and pumped to the cleaner flotation circuit. The tailings of each rougher cell will be gravity fed to the subsequent rougher cell with the tailings from the final rougher sent to the rougher tailings pump box from where the tailings will either be sent to the tailings thickener or to the tailings cyclone at the hydraulic backfill plant.

17.3.3.2 First Cleaner Flotation and Scavenging

Rougher concentrate is pumped to the first cleaner flotation and scavenging circuit, consisting of three conventional forced air cells for cleaning, followed by two scavenger cells, arranged in a 1-2-2 configuration. PAX and MIBC will be dosed into the first cleaner cell and scavenger cell. The cleaner concentrate from each cell will be gravity flow into a common launder, and reporting to a pump box, from which it will be pumped to the final stage of cleaning. The tailings of each cleaner cell will flow to the subsequent cell with the tailings from the final cell gravitating to the flotation tailings pump box.

17.3.3.3 Second Cleaner Flotation

The second cleaner flotation is the final flotation stage in the mill, and it will consist of a conditioning tank followed by two conventional forced air flotation cells. Concentrate from the first cleaner flotation and scavenging circuit is conditioned with PAX, MIBC, and copper sulphate to promote recovery, as well as sodium silicate as a depressant of deleterious material in the feed. The concentrate from both second cleaner cells is combined in an external launder, and pumped to the concentrate dewatering area.

The second cleaner circuit is designed to be elevated above the first cleaner flotation and scavenging circuit. This allows for the recirculation of the second cleaner tails to the first cleaner flotation and scavenging circuit without the use of pumps. The tailings from the first second cleaner cell will gravitate to the second cell before reporting back to the first cleaner flotation and scavenging circuit.

This flotation area includes the following major equipment and facilities:

- one rougher conditioning tank, 20 m^3 each
- one second cleaner conditioning tank, 6 m^3 each
- five rougher flotation cells, forced air, 20 m^3 each
- three first cleaner flotation cells, forced air, 6 m^3 each
- two first cleaner scavenger cells, forced air, 6 m^3 each
- two second cleaner flotation cells, forced air, 6 m^3 each
- associated pump boxes and pumps.

17.3.4 Concentrate Thickening, Filtering, Drying and Loadout

The design of the concentrate dewatering circuit consists of a thickener, filter press, and dryer. The concentrate thickener is designed for an annual operating time of 8,059 hours or 92% availability. The filter and dryer are designed for 7,446 annual operating hours or an availability of 85%.

The concentrate from the second cleaner flotation circuit will report to the concentrate thickener feed box and into the concentrate thickener. Flocculant will be dosed into the thickener to aid the settling process. Thickener overflow

will be sent to the process water tank for reuse in other areas of the mill. The thickener is designed to reduce the water content of the concentrate to 35% w/w before reporting to the filter feed tank sized for a residence time of 12 hours.

Thickened concentrate will be pumped to the concentrate filter feed tank designed to have a live capacity of 12 hours, from where it will be pumped to the filter press. A filter press (vertical chambers) will be utilized to further decrease the moisture content in the concentrate from 35% w/w to 15% w/w. A belt feeder below the filter will receive the filter cake and deposit onto the concentrate dryer feed conveyor. The filtrate will report to the concentrate filter air/water separator and into the filtrate tank where a pump will recycle the filtrate to the concentrate thickener feed box.

The concentrate drying system consists of a fuel-fired heater and wet scrubbing system, and will target a final product moisture content of 5% w/w. The dryer will discharge into a stockpile which will be manually reclaimed via front-end loader. The concentrate will be loaded into a concentrate hopper for bagging loadout.

The concentrate dewatering and loadout area consists of the following major equipment and facilities:

- High-rate thickener, 5 m Diameter
- Filter press (vertical chambers)
- Horizontal drum dryer, diesel burning
- Dryer dust collection system
- Associated pump boxes, pumps, feeders, and conveyors.

17.3.5 Tailings Thickening, Filtering, and Dry Stacking

The design of the tailings dewatering circuit consists of a thickener and filter press. The tailings thickener is designed for an annual operating time of 8,059 hours or 92% availability. The filter is designed for 7,446 annual operating hours or an availability of 85%.

Tailings from the flotation tailings pump box will report to either the tailings thickener or tailings cyclone at the hydraulic backfill plant depending on backfill demand. When the hydraulic backfill plant is not running, tailings will report to the dewatering circuit as normal. When generation of hydraulic backfill is required, the tailings will instead be pumped to the tailings cyclone. The coarse underflow fraction will be sent to the hydraulic backfill plant while the overflow will be pumped back to the mill and to the tailings dewatering circuit.

The tailings dewatering circuit begins with a thickening stage in the tailings thickener where flocculant will be dosed to promote settling of the solids to achieve an underflow of 50% w/w moisture content. Thickened tailings will be pumped by two centrifugal pumps in duty/standby arrangement to the tailings filter feed tank. The thickener overflow will be sent to the process water tank for reuse.

Two centrifugal pumps in duty/standby arrangement will pump the tailings from the filter feed tank to the tailings filter press. The filter is designed to achieve a filter cake moisture content of 15% w/w. Filtrate will be sent to the tailings filter air/water separator and held in the tailings filter filtrate tank before being pumped back to the thickener feed

box. The filter will deposit the filter cake in a tailings stockpile from where a front-end loader will load the filtered tailings into haul trucks and transported to the dry stack facility.

The tailings dewatering and loadout area consists of the following major equipment and facilities:

- High-rate thickener, 12 m diameter
- Filter press (vertical chambers)
- Tailings cyclone
- Hydraulic backfill plant
- Associated pump boxes, pumps, feeders, and conveyors.

17.3.6 Hydraulic Backfill Plant

See Section 16.8.

17.3.7 Reagents and Grinding Media

The reagent and grinding media areas are described below.

17.3.7.1 Copper Sulphate

Copper sulphate will be received on site on pallets in 25 kg bags and mixed once a shift in a mixing tank to a concentration of 15% w/w with a retention time of 24 hours. Copper sulphate will be dosed to the rougher flotation and second cleaner flotation circuit as an activator. Annual consumption is estimated to be 105 tonnes.

17.3.7.2 Sodium Sulphide

Sodium sulphide will be received on site in 25 kg bags on pallets and mixed once a shift in a tank to a concentration of 120 g/L, with a retention time of 24 hours. The annual consumption of sodium sulphide will be 145 tonnes.

17.3.7.3 Sodium Silicate

Sodium silicate will be delivered to site in liquid form in 200 L drums and will be dosed to the second cleaner as a depressant. Annual consumption will be 25 tonnes.

17.3.7.4 PAX

Potassium amyl xanthate (PAX) will be brought to site in pallets of 25 kg bags and mixed with fresh water to a solution concentration of 15%w/w in a 24-hour retention period tank once per shift. PAX will be dosed to the rougher, first cleaner, and second cleaner to promote the flotation of gold-bearing sulphides. The annual consumption rate of PAX is 70 tonnes.

17.3.7.5 MIBC

Methyl isobutyl carbinol (MIBC) frother will be received on site in liquid form 800 kg intermediate bulk containers (IBCs). The solution will be dosed to the rougher, first cleaner, and second cleaner. The annual consumption of flocculant will be 29 tonnes.

17.3.7.6 Flocculant

Magnafloc10 (or equivalent anionic polyacrylamide flocculant) will be supplied to site in pallets of 25 kg bags. The flocculant will be mixed to a concentration of 0.5% w/w once per shift in a 12-hour retention time tank. Flocculant will be dosed to the concentrate and tailings thickener as required. The annual consumption of flocculant will be 5 tonnes.

17.3.7.7 Ball Mill Media

Forged steel balls with a diameter of 75 mm will be delivered to site in 2500 kg bulk bags. Media consumption rate will be 0.035 kg/kWh or 0.8 kg/t of plant feed. The annual consumption of ball mill grinding media will be 280 tonnes.

17.3.8 Air Services, Water Services, and Utilities

17.3.8.1 Process Water

Process water in the mill is made up of the tailings and concentrate thickener overflow and make up water from the fresh water tank. When the hydraulic backfill plant is not running, the process water tank will see an excess and require no make up water. The excess water will be sent for effluent treatment at a rate of 1.6 t/h. The excess water is the result of the difference between the water entering the mill as moisture within the ROM ore and fresh water for us as gland seal water, and the retained water in the filtered concentrate exiting the mill.

When the hydraulic backfill plant is in operation, the majority of the flotation tailings are sent to the tailings cyclone, with only the overflow returning to the mill and feeding the tailings thickener. This will result in a net negative in process water for mill use and will require fresh water to be diverted from the fresh water tank to the process water tank at a rate of 5.8 t/h. No effluent treatment will be required due to there no longer be an excess amount of process water.

17.3.8.2 Fresh and Fire Water

Fresh water will be sourced from Whitewater Creek and held in a freshwater pond designed for a live capacity of two days. Fresh water will be pumped from the pond to the fresh and fire water tank with a capacity of 3 hours. This tank will maintain a certain minimum reserve capacity for fire water at all times. 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 fire water hydrant network and associated ancillary buildings. Fresh water used for reagent mixing in the mill will also be supplied from the fresh and fire water tank.

17.3.8.3 Potable Water

Potable water will be sourced from the fire and freshwater distribution pump and treated in the potable water treatment plant. Treated water from the potable water treatment plant will be held in a tank designed for a capacity of 48 hours and will supply all the potable water used on site.

17.3.8.4 Gland Water

Gland water will be directly drawn from the fresh and fire water tank and supply gland water to all the pumps.

17.3.8.5 Sewage Treatment Plant

The sewage treatment plant will be capable of handling an average sewage flowrate of 31 m^3/d and a peak of 82 m^3/d. The sewage treatment plant will come in two modules, one including the equalization chamber and mixer, denitrification, membrane filtration, and UV disinfection, and another for sludge mixing, dewatering, and sludge handling.

17.3.8.6 Effluent Water Treatment Plant

The effluent water treatment plant will consist of two parallel systems capable of each treating 30 L/s of effluent water. Each system will consist of slant clarifiers, pH treatment, coagulation dosing, and flocculation reactors two remove solids from the discharge stream. Each system will come with a sludge pumping system to remove the solids from the treatment plant.

This area includes the following major equipment and facilities:

- Process water tank, 150 m^3
- Fresh and fire water tank, 650 m^3
- Potable water treatment plant
- Sewage Treatment Plant (82 m^3/d, peak)
- Effluent Water Treatment Plant (two 30 L/s plants)
- Treated water tank
- Associated pumps

17.3.8.7 Air Services

Flotation cell blowers will supply air to the rougher, first cleaner and scavenger, and second cleaner circuit, and an air dryer will provide instrument air as required.

The concentrate and tailings filters will have dedicated compressors to service the blowing, membrane squeezing, and drying requirements.

The area includes the following major equipment and facilities:

- Rougher flotation air blower, 45 kW
- Cleaner 1 flotation air blower, 22 kW
- Cleaner 2 flotation air blower, 18.5 kW
- Three filter air receivers
- Plant air receiver
- Instrument air receiver
- Plant air compressor, 11 kW
- Air filters
- Air dryer

17.3.8.8 Power

All power delivered on site will be powered by diesel gensets. The process plant and ancillary buildings and camp will have a total installed power of 6.5 MW and see a typical operating load of 4.2 MW at an annual power consumption of 36.5 GWh.

Table 17-2: Power Requirements and Energy Consumption

	Installed Power (MW)	Operating Power (MW)	Energy Consumption (MWh/a)
Process Plant	4.4	2.8	24,622
Ancillary Buildings and Camp	2.1	1.4	11,927
Total	**6.5**	**4.2**	**36,549**

18 PROJECT INFRASTRUCTURE

18.1 Introduction

18.1.1 Overview

The New Polaris Project is located near the confluence of the Taku and Tulsequah Rivers, in a remote area of northwest British Columbia, Canada. The area is not accessible by road and has no nearby grid power to support the mine's operation. The site will include the following facilities and infrastructure:

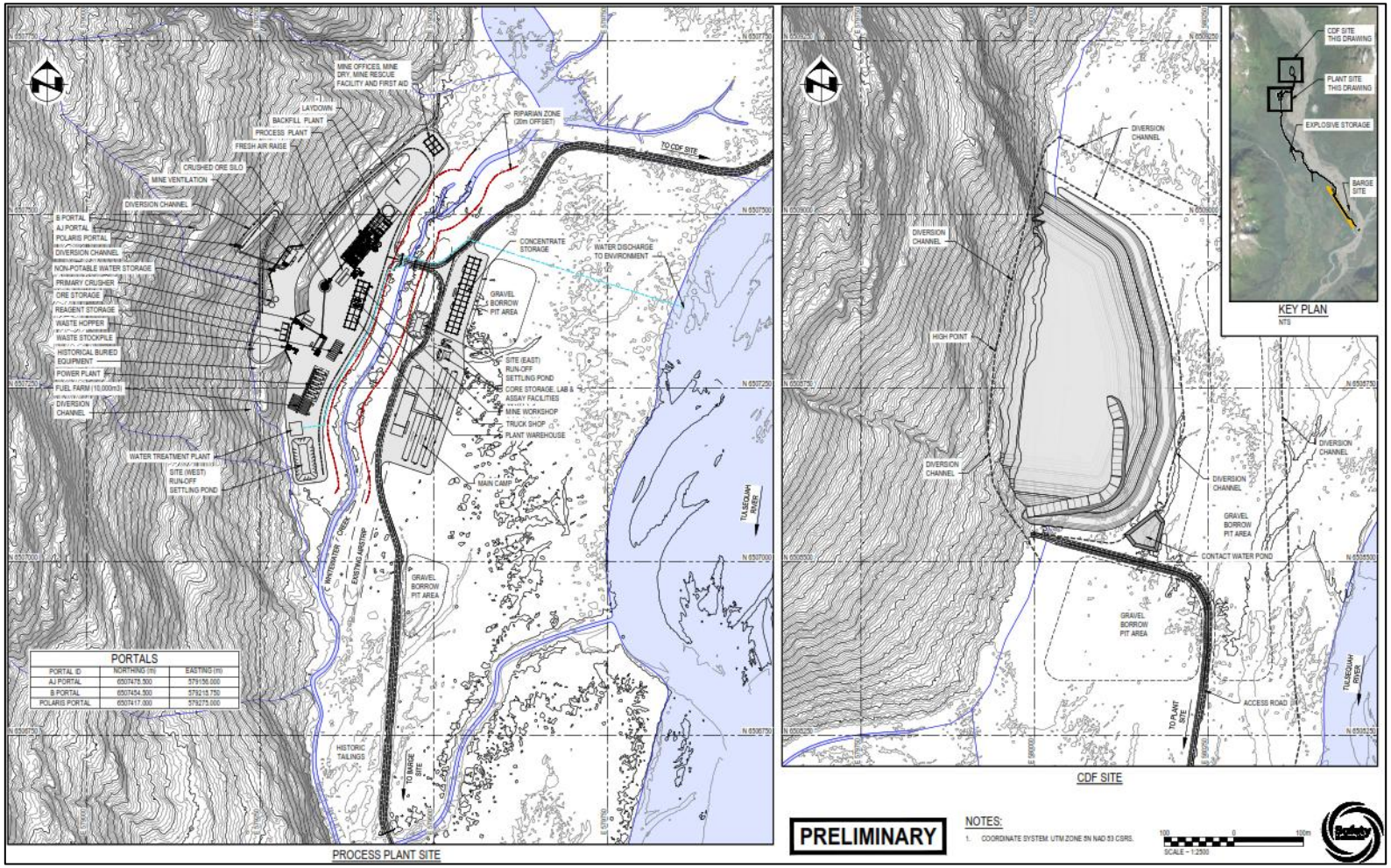
- mining facilities including mine administration offices, mine dry, truck workshop, explosives storage, fuel storage and distribution, ore stockpiles, and waste stockpiles.
- process facilities including the process plant, crushing facilities, backfill plant, process plant workshop, assay laboratory, reagent and concentrate storage.
- mine waste and water management infrastructure including: CSF for waste rock and tailings, catchments, ponds, and other site water management infrastructure.
- general facilities including the camp, power plant, new airstrip, new barge landing facility, access roads from the airstrip, barge landing facility and explosives facility to the mine site area, and haul roads from the mine site area to the CSF.

The location of site facilities was based on the following criteria:

- locate the facilities within the claim boundaries;
- locate the waste rock/ tailings storage facilities near the mine to reduce haul distance;
- locate the primary crushing and run-of-mine pad between the underground portal and the process plant to reduce hauling;
- utilize the existing site access road to the greatest extent possible; and
- leverage the natural terrain to optimize construction costs.

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

Figure 18-1: New Polaris Site Layout



Source: Ausenco, 2025.

18.1.2 Site-wide Geotechnical Field Investigations

The New Polaris 2023 geotechnical field investigation (the "Program") was carried out to support the New Polaris Project Feasibility Study. The Program included investigations in the proposed areas of the Project's main infrastructures such as the Process Plant, CSF, Airstrip, Barge Landing, and Explosive Storage Magazine. The investigations included Drilling, Test Pitting, Dynamic Cone Penetration testing and installation of monitoring wells to capture important design geotechnical-hydrogeological data for the Project's infrastructures regarding soil and bedrock geology, depth to bedrock, soil properties, and surface water hydrology. The field program commenced on August 13, 2023, and concluded on September 23, 2023. In all, 50 test pits, 41 Dynamic Cone Penetration tests, 120 m of infrastructure drilling, and three monitoring wells were executed during the field program.

Concurrent with the field program, collected samples underwent a series of geotechnical laboratory tests to determine the physical and mechanical properties of the soils and rocks. These tests included, but were not limited to, grain size analysis, Atterberg limit tests, compaction tests, proctor tests, flakiness index tests, and unconfined compressive strength tests. The laboratory program was critical in verifying the field observations to enable proper geotechnical assessment of the overburden and bedrock encountered at the Project's site.

18.2 Roads and Logistics

18.2.1 Site Preparation

The New Polaris site is a re-development of the former mine and town site of Polaris-Taku Mine, which operated between 1937 and 1951. Therefore, existing buildings require demolition, and existing infrastructure requires general site preparations. Site civil work includes design for the following infrastructure:

- access roads
- haul roads
- mine facility platforms and process facility platforms
- water management ponds and ditches and channels
- co-storage facility.

18.2.2 Access to Site

There is no road access to the New Polaris site. The site can only be accessed by air or water along the Taku River.

18.2.3 Roads

The existing site road network has not been actively maintained since the last mine closure in 1951 and is therefore in varying states of disrepair, subject to overgrowth, road wash outs, and other seasonal/ environmental effects. While some minor upgrading and maintenance has been completed to facilitate recent exploration at the site, it is assumed that to bring the site up to full operational levels, it will require a combination of new road works as well as upgrading

of existing road networks. Where possible the new road network will utilize existing alignments to reduce costs and environmental footprints.

The most significant road building and upgrading activity is to re-establish the main access road between the mine site and the proposed barge landing in order to facilitate transport of material goods and consumables, required for construction, into the site during the barging season. The remaining 7.5 km follows a similar alignment to the original access road, running southeast between the west bank of the Tulsequah River and the Flannigan Slough.

In addition, a haul road between the mine site and the CSF to facilitate transport of waste rock and filtered tailings will be constructed. The haul road is approximately 2 km in length, requiring balanced cut/fill road building.

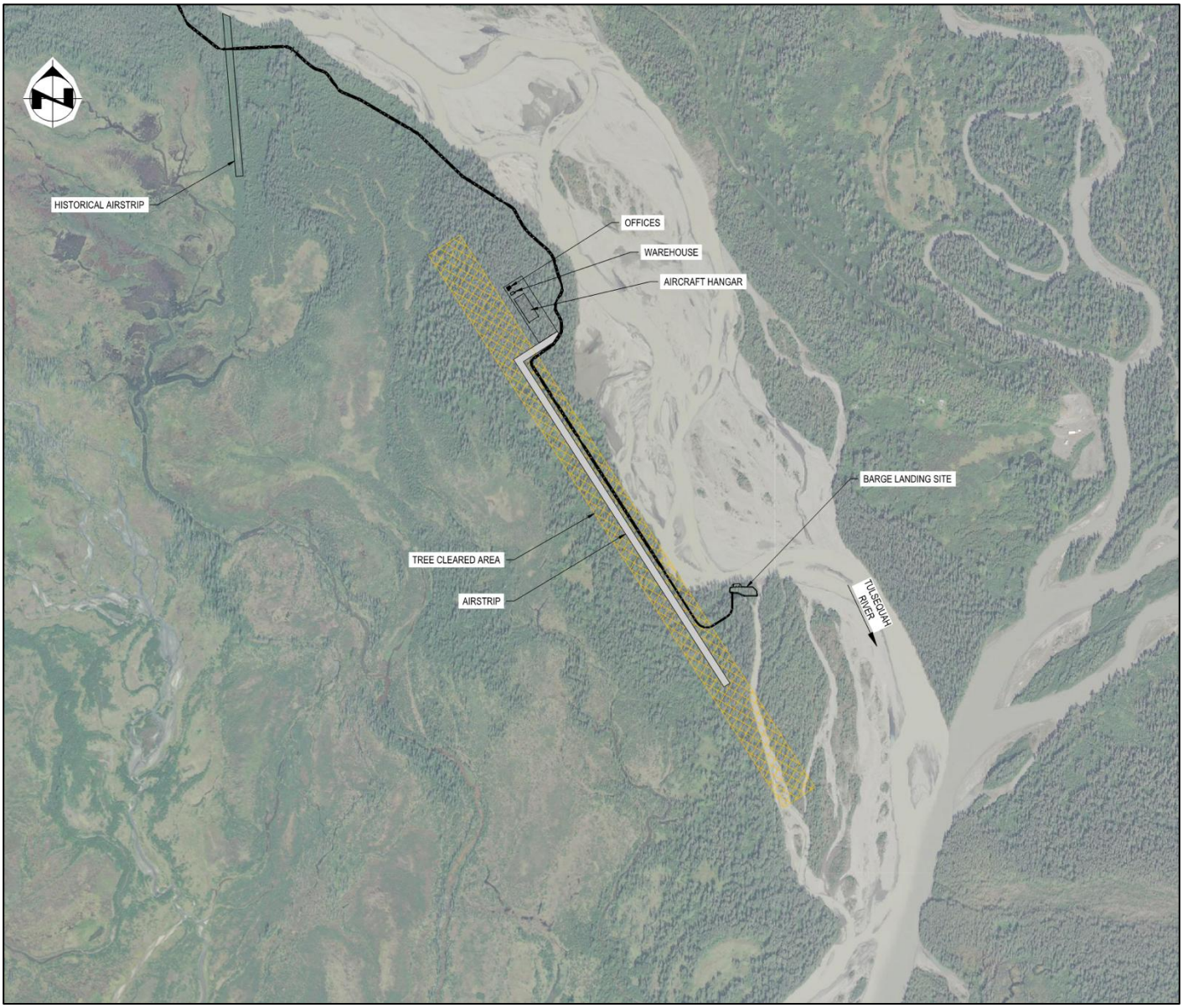
All roads will be 5 m in width for single lane traffic. Pull outs will be placed at intervals to allow passing of two-way traffic appropriate to the level of service of the road. Radio control of the road will also be required. The typical method of clearing, topsoil removal, and excavation will be employed, incorporating drains, safety bunds and backfilling with granular material and aggregates for road structure.

18.2.4 Airstrip

The existing New Polaris airstrip is too small to accommodate aircraft (ATR 72) appropriately sized for the mine during full operations, therefore a new 1,700 m airstrip located near the Tulsequah and Taku River confluence has been considered for this study (Figure 18-2).

An Air Operations Feasibility Study was completed by Jetpro Consultants Inc to evaluate the feasibility of airlifting consumables into the New Polaris Mine site and reciprocally airlift mine concentrate out of the site daily during full scale operations, including evaluation of the new proposed airstrip. The conclusion was that the plan is feasible with a robust air operations plan.

Figure 18-2: New Airstrip Location



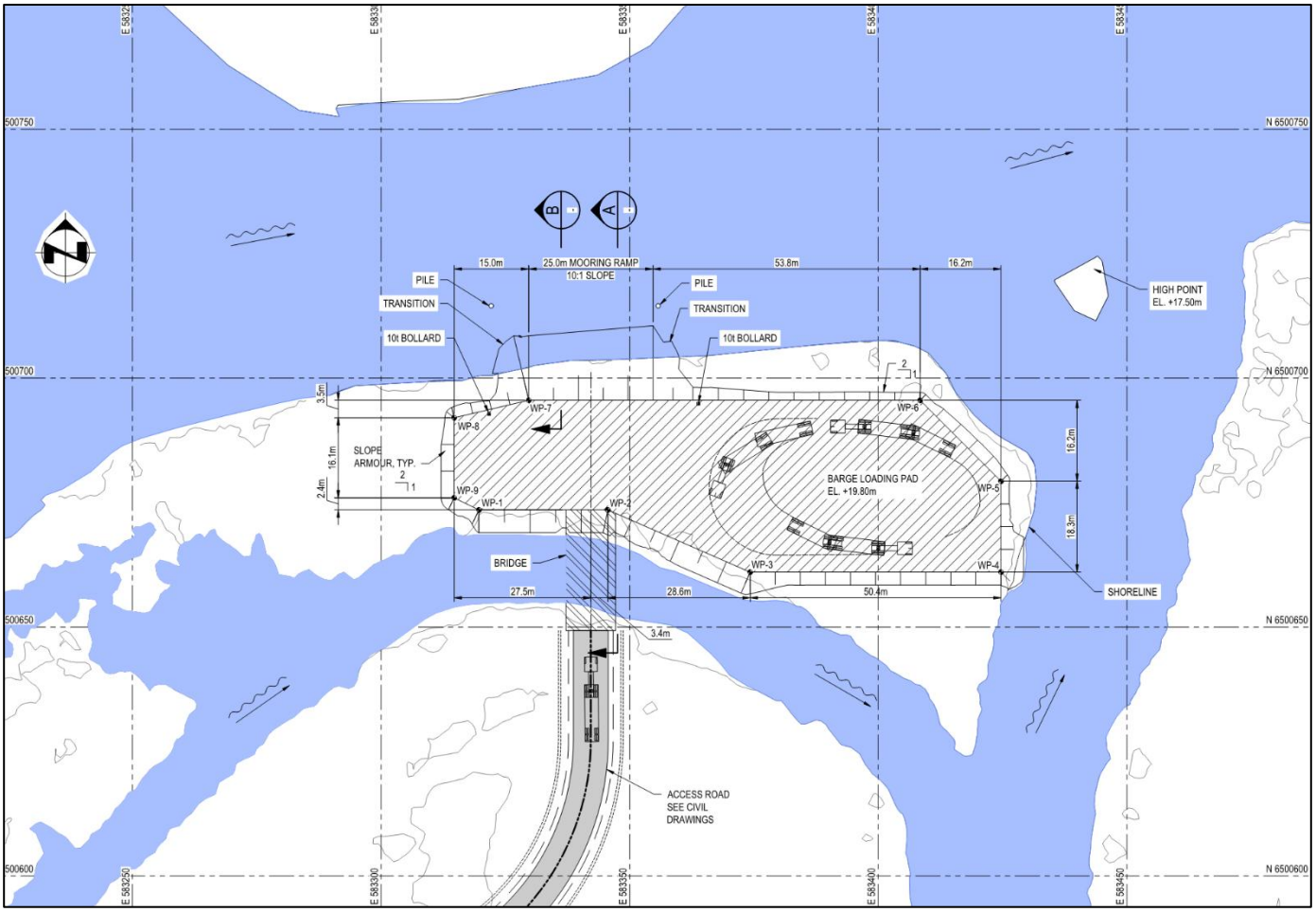
Source: Ausenco, 2025.

18.2.5 Barge Landing

The barge landing will be located immediately upstream of the confluence of the Tulsequah River and the Taku River, on the west side of the Tulsequah. The barge landing will be constructed from fill, protected by slope armour, to create a level platform along the river edge capable of docking up to two barges simultaneously and providing space for trucks

to be loaded. The barge landing facility will also include a small office trailer, genset, diesel fuel day tank, temporary storage area, and container handler and/or mobile crane.

Figure 18-3: Barge Landing Layout



Source: Ausenco, 2025.

18.2.6 Security

Because of the remote location of the site, no site security provision is considered.

18.3 Power and Electrical

New Polaris will obtain power from six 2-MW diesel generators (one generator as standby).

Five 2-MW diesel generators will form the main power supply for the site, with one additional 2-MW diesel generator for backup and maintenance. The generators will be placed adjacent to one another with day tanks servicing each unit, all housed within a power generating facility.

Power will be transformed down to 600 V for surface distribution of power to the process facility and other site infrastructure. The layout of the surface power distribution is subject to final layout of all site infrastructures.

18.4 Support Buildings

As shown in the site layout in Figure 18-1, the main plant site area consists of several buildings. Table 18-1 shows the building names, type of building, and size of buildings to be built.

Table 18-1: Description of On-Site Buildings

Building Name	Type	Length (m)	Width (m)	Height (m)
Process Plant (Grinding flotation, filtration and reagents)/ Backfill Plant	PEB	85	26	-
Mine Truck Shop/Workshop/ Truck Wash, Plant Warehouse	PEB	71	18	-
Mine Offices (Mine Dry, Mine Rescue Facility and First Aid)	Modular	24	16	3
Concentrate Storage	PEB	113	18	-
Air Strip Office	Modular	16	15	3
Airstrip Warehouse	Fabric	16	10	4
Reagent Storage	Fabric	30	12	3
Accommodation Camp	Modular	75	9	3

18.5 Stockpiles and Silos

The material from the underground mine will be diverted to two main destinations depending on material type: underground waste rock will be sent to the CSF, with ore reporting to and stored in ore storage bunkers adjacent to the top of the primary crusher. Fine ore will be stored in an ore storage silo after secondary crushing.

18.6 Rock Storage Facilities

A co-storage facility (CSF) is planned for waste rock from the underground mine and for plant tailings. Refer to Section 18.8 for details on the CSF.

18.7 Mining Infrastructure

18.7.1 Haul Roads

See Section 18.2.3.

18.7.2 Explosives Facilities

See Section 16.9.4.

18.7.3 Truck Shop/ Truck Wash/ Workshop/ Warehouse

The maintenance building will be a pre-engineered steel framed structure built on a concrete slab, sectioned into separate areas for the mine/truck/maintenance/electrical workshop, plant warehouse and truck wash. The facility will include four maintenance bays and one wash bay, with a single 20 t overhead crane to service all bays.

18.7.4 Mine Offices/ Mine Dry/ Mine Rescue Facility/ First Aid

The mine offices (mine dry, mine rescue facility and first aid) will be a modular building sectioned into separate units. Minimal installation requirements include clearing and grubbing, some minor foundation work, and hook up of site services.

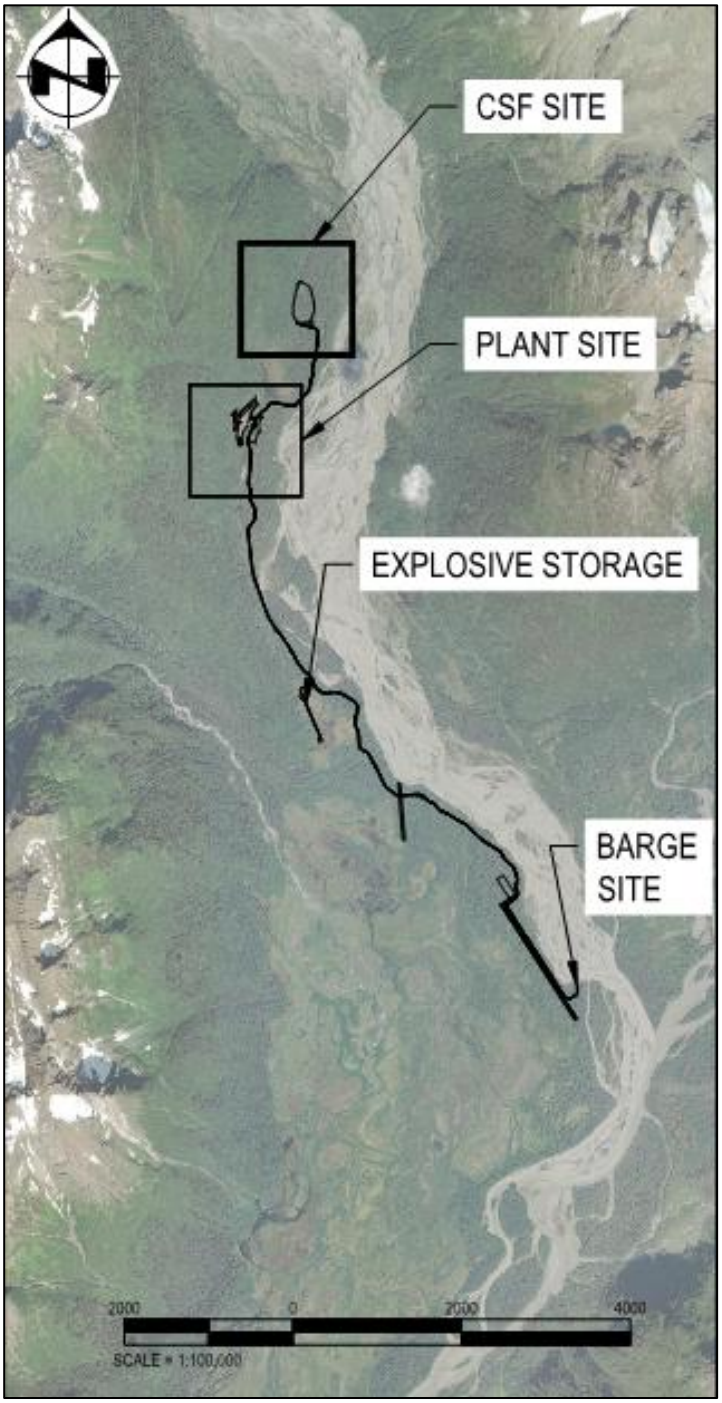
18.8 Co-Storage Facility (CSF)

New Polaris process tailings and waste rock are planned to be co-disposed in a facility constructed 2.5 km north of the deposit, portal and process plant. The location of this CSF is shown as the 'CSF Site' in Figure 18-4 below. The CSF facility has an ultimate elevation of 73 masl (height 38 m). The CSF site will have capacity to hold 3.21 Mt of tailings and waste rock. The facility will contain tailings with a waste rock cover over the slope. Bench slopes are designed to be 3:1 (H:V). Based on 2023 geotechnical investigations, the surficial geology below this facility is underlain by 1 to 0.7 m of sandy silt overlying 3 m of sandy gravel with silt.

Seepage from the facility will be captured in a constructed underdrain that conveys seepage to a small, lined pond located to the southern area of the footprint of this facility.

The CSF will encompass additional infrastructure, including a contact water collection system, collection pond and channels, underdrain collection system, and access road. These components are integrated into the areas designated for reclamation under the closure plan. Contact water will be collected and will be conveyed to the collection pond on the southern footprint of the facility. Noncontact water diversion channels will be built to reduce the generation of contact water from exposed regions of the CSF and will be discharged into natural ground. These channels are expected to stay in place, as they offer advantages by diverting water during low-return-period precipitation events.

Figure 18-4: CSF Site



Source: Ausenco, 2025.

The co-disposal methodology creates an integrated disposal facility with very little mixing between the filtered tailings and the waste rock. Benefits include:

- If any of the mine waste rock is potentially acid generating (PAG) and susceptible to metal leaching, the placement of waste rock with the tailings will reduce the access of oxygen to the rock, reducing the potential for acid generation and metal leaching.
- Inclusion of mine waste rock with the tailings improves the stability of the facility and reduces erodibility of the process tails.
- Co-disposal reduces the footprint area and simplifies the water management strategy, compared to disposing of waste rock and tailings separately.
- Co-disposal facilities, typically can be closed earlier when compared to standalone tailings and waste rock facilities, which reduces closure costs.

18.8.1 Design Criteria

The key design criteria for the CSF are as follows:

- The CSF will accommodate approximately 2.32 Mt of filtered tailings and 0.31 Mt of mine waste rock, which will be transported to the facility by haul trucks and co-disposed throughout the LOM.
- The primary guidelines considered for the design of the CSF are the Global Industry Standards on Tailings Management (GISTM) and the Canadian Dam Association (CDA) guidelines to provide a safe and environmentally acceptable facility for filtered tailings and waste rock.
- Based on the CDA, the CSF has a hazard classification of 'Significant.' Accordingly, it should be designed for a 1-in-1,000-year seismic event. However, we designed the CSF for a 1-in-2,500-year seismic event to be more conservative. The National Building Code of Canada Seismic Hazard Tool (NBC, 2020) was used to obtain the peak ground acceleration (PGA) values for these return periods, which are 0.163 g and 0.251 g, respectively. Temporary and permanent, contact and non-contact surface water management structures were designed for a 24-hour, 100-year return period storm event.
- The CSF involves stacking filtered tailings with a specific gravity of 2.77, maximum moisture content of 15% by weight with target placed dry density of 1.65 t/m^3, and waste rock with a specific gravity of 2.77 and density of 2.0 t/m^3.
- The CSF designed includes a liner to protect against any potential for ARD or metal leaching from tailings or waste rock.

While the design complies with the minimum factor of safety stipulated by the CDA, this does not indicate that the facility is limited to the minimum standard; engineering judgment and additional considerations have been applied to ensure robustness where appropriate

18.8.2 CSF Construction and Operation

18.8.3 Stability and Seepage Analysis

Stability analyses were carried out on a critical section of the planned ultimate CSF using both static and pseudo-static analyses. Based on similar projects and the National Building Code of Canada (NBC 2020), the 1-in-2500-year peak ground acceleration for Type C soil equal to 0.251 g is used in the stability analysis. Ausenco conducted stability analyses using GeoStudio (Slope/W). Minimum factor of safety (FOS) loading conditions according to (CDA, 2019) Limit equilibrium analysis was conducted using peak effective strength soil parameters taken from the laboratory triaxial tests. The analysis results for the CSF their ultimate configurations indicate satisfactory FOS under static and pseudo-static conditions.

Table 18-2: Target Factor of Safety (FOS)

Loading Conditions	Target FOS
Short Term (Static Condition)	1.3
Long Term	1.5
Pseudo-static	1.0

Seepage analysis was conducted to support the overall stability assessment of the structure. The analysis provided insight into the groundwater flow conditions and was essential in accurately estimating the phreatic surface. In addition, seepage flux was calculated as part of the analysis to quantify the rate of water movement through the soil. These results were used as input for the slope stability analysis to ensure a reliable evaluation of potential failure mechanisms.

18.8.4 CSF Water Management

CSF water management system includes structures for surface and seepage water management. The surface water management will include the following components:

- Non-contact water which will be diverted by channels around the CSF.
- Contact water from the CSF in diversion channels that run on the surface along the toe of the facility.
- Seepage water in the foundations of the CSF that will be captured using an underdrain system. The underdrain for the CSF has been designed to capture decant water (free water) from the tailings along with seepage from precipitation.

The diversion channels will collect non-contact surface runoff from the natural areas surrounding the CSF and convey it to drainage areas outside the CSF footprint. The CSF surface water collection system is designed to collect contact surface runoff during operations and direct it to the collection pond. Following closure, these collection systems will function as non-contact water collection ditches. Contact water, expected to contain only sediment from the facility, will be routed to the collection pond before being discharged to the environment. Water quality monitoring will be

conducted regularly to ensure that discharged water meets applicable standards. If the water does not meet these standards, it will be treated prior to discharge.

Temporary and permanent channels were designed to convey runoff from the 1-in-100-year storm events. The underdrain water management system will consist of HDPE dual wall pipe and drainage gravel wrapped in a non-woven geotextile to capture near-surface groundwater and seepage from the tailings; capturing these water sources will minimize build-up of the phreatic surface in the base of the CSF. In addition, the underdrain system will be utilized to drain surface runoff that is blocked by the CSF from following its original path. A ring-dyke filter system will be placed around the underdrain inlet to prevent solids from entering and potentially plugging the system.

18.8.5 Instrumentation and Monitoring

The stability of the CSF will be monitored using vibrating wire piezometers and GPS sensors strategically placed within the foundation and the CSF. The piezometers will monitor pore pressure below and above the liner system and the GPS sensors will monitor any movement of the CSF.

These instruments will be connected to a data logging system and telemetry system for real-time continuous and unattended monitoring. Additionally, water quality monitoring instrumentation will be implemented to monitor the surface contact water flows at the water collection pond.

18.8.6 Closure and Reclamation

The CSF will provide long-term storage of waste rock and filtered tailings and will remain in place beyond mine closure. The objective of closure for the CSF is to ensure its long-term physical, geochemical, hydrological, and revegetation stability. Closure strategy includes implementing a robust environmental closure design aligned with best practices to minimize environmental risks. Key goals include addressing and reducing safety concerns post-closure and protecting groundwater and surface water quality downstream of the CSF. Additionally, the closure strategy aims to maintain the long-term stability and integrity of the CSF while integrating it into the surrounding landscape. This involves restoring the site's natural appearance and supporting a return to its pre-mining land use and productivity level, ensuring the area can sustain ecological function and meet long-term land use goals. For reclamation, the CSF will be progressively reclaimed in stages following the completion of each outer permanent lift to maintain slope stability, minimize dust, minimize water infiltration, and contaminant migration while being visually compatible with the surrounding area. The outer slopes of the waste rock shell will be graded to 3.5:1 (H:V) as part of the operations to facilitate progressive reclamation. The outer slope will then be covered with topsoil and revegetated. The filtered tailings will be closed in the same manner after the last lift.

Contact water from the CSF will be collected in the water collection pond and treated, if required, before being released to the environment. Seepage water will be discharged directly to the environment if it meets the applicable water quality criteria; otherwise, it will be diverted to the collection pond and treated in the same manner as the contact water.

18.9 Site-wide Water Management

This section presents an overview of site-wide water managements, encompassing the design of water management structures, hydrologic considerations and the site-wide water balance. The review and interpretation of exiting climate data, encompassing factors such as precipitation, snowfall and evaporations are critical for accurately estimating water balance and designing effective water management structures.

18.9.1 Hydrometeorology

Meteorological monitoring at the New Polaris site is now complete and has been fully incorporated into this analysis. Site-specific measurements of precipitation depth and temperature (Table 18-3) were obtained from the operational weather station, replacing previous reliance on regional datasets such as Juneau International Airport. The on-site Pluvio Total Precipitation Gauge malfunctioned until recalibration in October 2024, rendering earlier precipitation records unusable. In the absence of reliable pre-2024 data, downscaled re-analysis products from Environment and Climate Change Canada's ClimateData.ca (2025) were used to derive 30-year mean monthly precipitation for the site grid cell. Comparison with the historical Tulsequah Chief dataset yielded a Pearson correlation coefficient near 1, confirming the suitability of ClimateData.ca as a precipitation proxy. The temperature data for 2024 was collected in two ways: from January to May, daily average, minimum, and maximum temperatures were manually downloaded, while from May to December the data was automatically recorded by the on-site New Polaris weather station and retrieved through DataGarrison.com. The combined dataset (Table 18-3)—site observations post-October 2024 and long-term re-analysis averages—provides robust monthly precipitation and temperature inputs, with temperature data integrated into the snowmelt model to improve water balance and infrastructure design. Ongoing station monitoring will enable refinement of the water management plan as conditions evolve.

Table 18-3: Monthly Precipitation (in mm) and Onsite Weather Station Monthly Temperatures (°C)

Month	Average Monthly Precipitation (mm)	Maximum 24-Hour Precipitation (mm)	Average Monthly Temperature (°C)
January	289.24	31.3	-8.4
February	195.37	23.5	-5.8
March	173.29	21.0	-2.9
April	97.00	13.0	1.5
May	81.18	14.8	7
June	85.82	15.3	9.6
July	108.09	17.1	11
August	158.43	26.2	9.7
September	239.72	35.8	7.6
October	313.89	38.8	3.8
November	269.11	32.6	-3.6
December	264.48	29.9	-9.2

Note: 30-year Monthly Average and 30-year Maximum 24-hour Precipitation from ClimateData.ca, 2025.

18.9.2 Water Management Strategy

Water management requires consideration of the water flow between facilities and the associated catchment areas. Surface water runoff that comes into contact with disturbed areas of the mine (contact water) must be managed prior to release to the receiving environment. The principal objectives of the site water management are to:

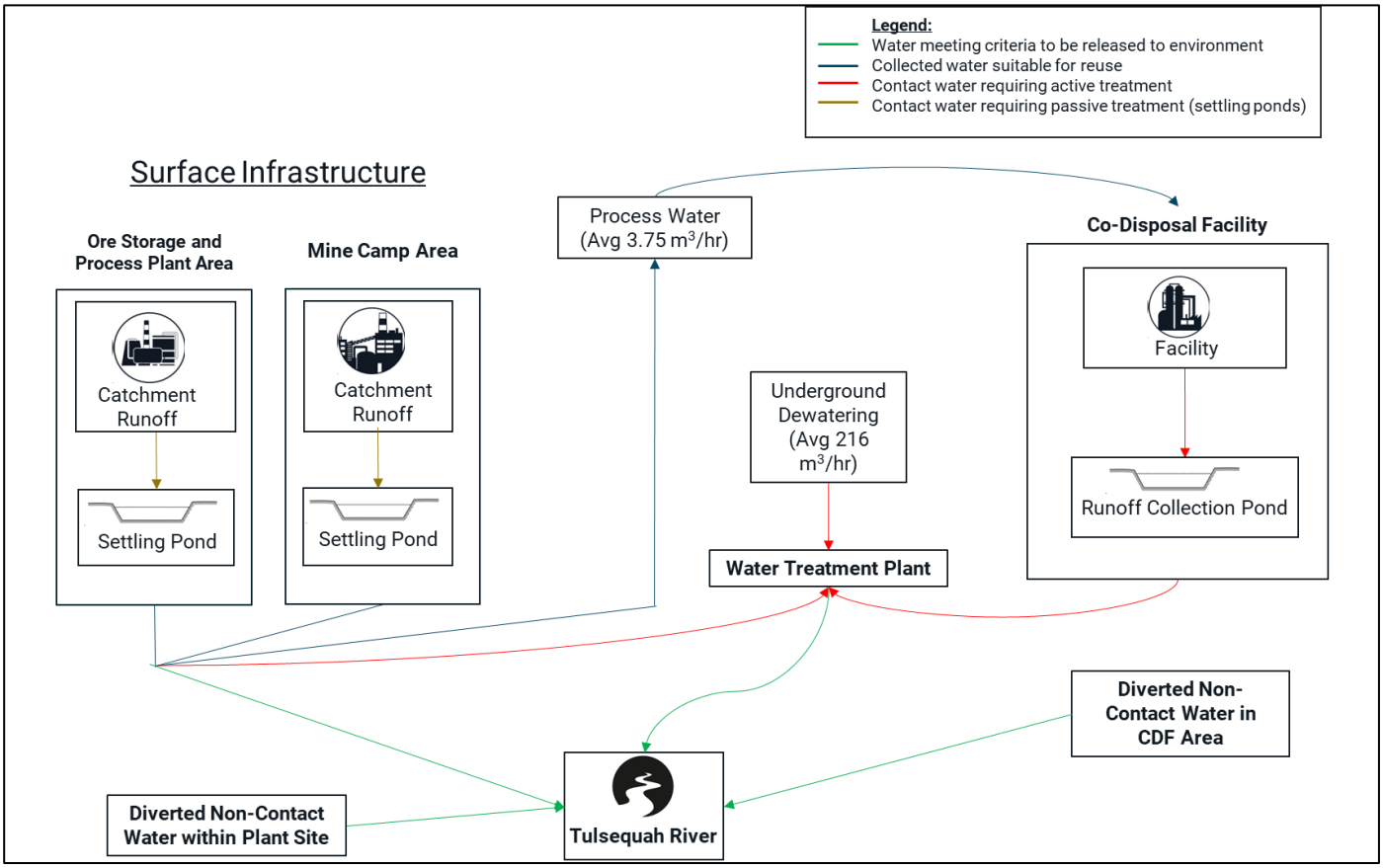
- Provide water for mining operations (milling/process, dust suppressant), including reuse of collected water to the extent feasible.
- Reduce potential flood effects to local receivers by controlling flooding on site and attenuating effluent discharge.
- Reduce water quality impacts on receivers by providing mine water sedimentation control and treatment.
- Reduce erosion and sedimentation with appropriate erosion and sedimentation controls.
- Maintain existing flows to the receivers to the extent feasible.
- Reduce site contact water as much as possible by incorporating surface water/overland flow diversion.

Major site infrastructure for water management includes:

- Diversion channels: Designed to convey non-contact water around disturbed areas and mine infrastructure. They have a design criterion of 1-in-100-year peak flow without overflow.
- Settling ponds: Sized to store a 1-in-100-year, 24-hour storm. Collected contact water will be released to the environment (pending water quality discharge requirements; otherwise, treated). Approximately 3.75 m^3/h of water released from the settling ponds will be reused as process water within the plant.

Site water management comprises the collection of contact water runoff from the three areas: Plant Site, Camp Area and the Co-Disposal Facility. The overall water management strategy is illustrated in Figure 18-5.

Figure 18-5: Water Management Schematic



Source: Ausenco, 2024.

18.9.3 Settling Ponds

The collection system for the main mine site (not including the Co-Disposal Facility) consists of ponds collecting water from:

- Processing Plant Area and Ore Stockpile
- Main Camp Area.

It is envisioned that each area will be graded toward the ponds to allow for gravity collection of contact runoff. Contact water will be treated for sediment within the ponds and discharged to the environment (if effluent meets discharge quality criteria) or sent to treatment. Settling ponds were sized to retain runoff from a 1-in-100 year, 24-hours storm without discharge, with a minimum freeboard of 0.5 m along with a 1.5 m permanent pool to aid in sediment settling. The estimated volume of each pond is noted in Table 18-4.

Table 18-4: Settling Pond Capacities

Setting Pond	Capacity (m^3)
Ore Storage and Plant Pond	6,200
Main Camp Pond	2,000

18.9.4 Diversion Channels

Diversion channels are required to divert non-contact runoff away from the facilities and to minimize the amount of contact runoff to be collected and managed. Road channel/ditching will intercept hillside runoff near the AJ Portal area and direct flow towards sawmill creek. Runoff upstream of the Plant site will be re-directed via a diversion toward Whitewater Creek. This diversion will be integrated into the grading of the process plant area.

18.9.5 Underground Dewatering

The underground mine workings at the site are currently flooded and will require dewatering prior to the excavation of a new ramp to provide access to the ore. As a result, mine dewatering will require active treatment and will be pumped directly from the mine workings at a rate of 216 m^3/h to the treatment plant. The dewatering rate was extrapolated from existing records from when the mine was last dewatered. An updated hydrogeological model is planned for the next stages of the project, with these updated values being incorporated into the revised water balance.

18.9.6 Water Balance

The water balance model for the New Polaris site was developed using GoldSim (GoldSim Technology Group LLC, 2022) to assess hydrological flows over a one-year period from January 1, 2025, to December 31, 2025. It uses site-specific meteorological data, including average monthly precipitation and average monthly temperatures, collected from the on-site weather station (precipitation from 2025 30-year averages and temperatures from 2021-2024). This data ensures a precise representation of local conditions, replacing earlier regional estimates.

Precipitation and temperature data drive the model's water balance calculations. Precipitation is categorized based on temperature: it accumulates as snow when monthly temperatures are at or below 0°C and is treated as liquid when above 0°C. Accumulated snow is converted to runoff during warmer months when temperatures exceed 0°C, with the melt rate proportional to the temperature increase up to a maximum of the accumulated snow. This conversion process ensures runoff reflects both liquid precipitation and snowmelt, adjusted for seasonal variations. Catchment areas across the site are factored into the model to calculate total runoff volumes, providing an accurate water balance for water management planning. The inflows to each of the settling ponds are shown in Table 18-5.

Table 18-5: Water Balance for Settling Ponds

Settling Pond	Daily Inflows (m^3/d) Average Year
Ore Storage and Plant Pond	347
Main Camp Pond	158

18.9.7 Water Treatment Requirements

At this stage, it is anticipated that treatment will be required for inflows from surface water runoff and seepage from the CSF and groundwater from underground dewatering of the mine. The anticipated average monthly inflow rate from the CSF requiring treatment will be 8.4 m^3/h. Concurrently, contact water from the underground mine will be treated at the water treatment plant at an inflow rate of 216 m^3/h. Discharge from the Plant Site settling ponds not meeting environmental requirements for release into the environment will also be routed to the water treatment plant for active treatment. The inflow rate of this will be established through predictive water quality modeling; planned for the next stages of the project. Once the specific effluent criteria and treatment systems have been established (through an assimilative capacity assessment and water quality modelling) sludge management measures will be reviewed. Table 18-6 outlines the estimated treatment rate required during operations for the Project.

Table 18-6: Estimated Treatment Rates

Flow Source	Inflow Rate (m^3/h)
Inflows from Underground	216
Average inflow from CSF	8.4
Average inflow from Site Settling Ponds (1)	52
Total	225 to 277

Note:

1. Water from Site Settling Ponds may require treatment. The above table provides a range of potential treatment rates if additional treatment is required for these ponds.
2. Pond optimization could include increasing the size to provide additional attenuation and reduce treatment inflow rate. This will be considered following the water quality assessment in the next stages.

19 MARKET STUDIES AND CONTRACTS

19.1 Introduction

19.1.1 Market Studies

Canagold employed Deno Advisory to complete a market study for New Polaris concentrate, summarized in the following section.

The New Polaris concentrate is a high gold concentrate containing high levels of arsenic and antimony. With the high gold price environment, the concentrate is attractive to multiple customers including gold roasters, copper and lead smelters and POX plants. For the base case, selling directly to traders for blending was selected with parameters shown in Table 19-1. Antimony is not considered payable at this time as further testing and investigation needs to be conducted into the production and sale of high-grade antimony concentrate.

Table 19-1: Off-take Agreements – Base Case

Delivery	Delivery CIP China, Callao or Manza (Peru) or parity
Gold Payables	If Au is above 80 g/t then 84.5% payable
	f Au is above 100 g/t then 86.75% payable
	If Au is above 120 g/t then 88.25% payable
	If Au is above 140 g/t then 89.5% payable
Treatment Charges	Not applicable
Penalties	None
Payment	Payment 15 days after bill of lading date
Weighing sampling and moisture determination	Outbound port
Air transport (US$/dmt)	594
Handling and barge (US$/dmt)	154
Sea freight (US$/dmt)	41
WSMD (US$/dmt)	2
Umpire + assay cost (US$/dmt)	2
Insurance cost (US$/dmt)	2
VAT (US$/dmt)	–
Letter of credit cost (US$/dmt)	–
Weight loss (US$/dmt)	12

The base case proposes material to be sold to traders for blending prior to shipment to Chinese gold roasters. Concentrate will be flown from the mine site to Juneau, shipped by barge to Tacoma then transported by ship to the customer.

The QP for this section has reviewed the market analysis and, in the QP's opinion, the data supports the assumptions in this technical report.

19.2 Commodity Price Projections

The project economics were estimated based on long-term metal prices of US$2,500/oz Au and an exchange rate of 1 CAD = 0.725 USD reflecting consensus forecasts from various financial institutions.

19.3 Contracts

No contracts for transportation or off-take of the concentrate are currently in place, but when they are negotiated, they are expected to be within 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.

19.4 Comments on Market Studies and Contracts

The QP is of the opinion that the marketing and commodity price information is suitable to be used in cashflow analysis to support the 2025 FS.

20 ENVIRONMENTAL STUDIES, PERMITTING, AND SOCIAL OR COMMUNITY IMPACT

20.1 Introduction

This section provides an overview of the setting of the New Polaris Project. It outlines existing biological and physical baseline conditions, proposed new baseline studies to support future permitting applications, existing permits, and future regulatory and permitting requirements including required management plans for water, site environmental monitoring, and waste disposal. In addition, this section also discusses socio-economic baseline conditions, the status of community consultation and engagement, and conceptual mine closure and reclamation planning for the project. Recommendations are also provided if the decision is made to progress the project through the permitting and construction phases.

The New Polaris project involves the re-development of the Polaris-Taku Mine located approximately 100 km south of Atlin, BC on the west bank of the Tulsequah River in British Columbia, Canada. New Polaris is located within the territory of the Taku River Tlingit (TRT). The Taku River/T'akú Téix' Conservancy lies approximately 10 km south of the project site and encompasses the British Columbia portion of the Taku River main stem from the Alaska border, to the confluence of the Nakina and Inklin Rivers (TRTFN and Province of BC, 2011). There is no proposed infrastructure or mining activities planned within this Conservancy; however, the proposed site access is by barge from Alaska via the Taku River which passes through the conservancy.

Refer to Figure 18-1 for site the site location and layout.

20.2 Environmental Considerations

The Tulsequah River flows south to a confluence with the Taku River, approximately 10 km downstream of the existing mine site. The Taku River is a transboundary river, flowing west into Alaska, and into the Pacific Ocean, 25 km downstream of the Canada-United States border. The Tulsequah River has a catchment area of approximately 2,300 km2 and is fed by glaciers and snowmelt. The river flows through a steep and rugged terrain, which results in rapid changes in water levels during periods of heavy rain or snowmelt. The Tulsequah River is an important habitat for a variety of aquatic species, including salmon, trout, and char, and its unique morphology plays a key role in supporting this diverse ecosystem. Peak flows typically occur from June to August and low flows primarily occur in January and February, while jökulhlaups (i.e., a glacial outburst flood) can occur at any time throughout the year. The Tulsequah glacier terminates in the Tulsequah valley approximately 16 km north of the project site and is the largest glacier in area. Historically the glacier forms a temporary dam part way up its stream, known locally as Lake No-Lake that forms in a tributary valley. The dam eventually breaks through the ice barrier during the latter periods of spring thaw every year, increasing flows in the Tulsequah and Taku rivers for three to five days (Ausenco, 2024).

Water rights will need to be acquired from the Province of British Columbia for sources of water for mining, processing and potable water during operations. Since a positive water balance prevails in the area, and plans include maximizing water recycling to minimize freshwater requirements, such approvals are generally granted, subject to acceptable conditions.

20.2.1 Baseline and Supporting Studies

20.2.1.1 Air Quality and Noise

Winds at the Project site tend to follow the orientation of the Tulsequah River valley, blowing from the northwest in the winter months and from the southwest in the summer months.

Since the Project site is remote and far from anthropogenic sources, it is anticipated that concentrations of particulate matter and gaseous pollutants would be low and that noise levels would be similar to that of natural remote alpine regions (Ausenco, 2023)

20.2.1.2 Fish and Aquatic Resources

The Taku and Tulsequah Rivers provide essential corridors to Whitewater Creek, Shazah Creek and Wilm's Creek that contain important spawning, rearing and overwintering habitat for fish, specifically salmonids (i.e., salmon, trout etc.). Fish and aquatic resources baseline studies were completed in 2021 and 2022 focused on Whitewater Creek, five unnamed tributaries located along the proposed tote road, as well as the Tulsequah River, Take River, and two reference sites that are not anticipated to be affected by the project (Ausenco, 2023).

An overview of key results from the aquatic baseline programs are provided below, included fish/fish habitat, primary productivity, and sediment quality:

- Whitewater Creek, Shazah Creek, and Wilm's Creek provide excellent rearing, overwintering and spawning potential for salmonids and other resident species.
- Spawning surveys identified coho salmon spawners within Whitewater Creek, Shazah Creek, and Wilm's Creek in 2021 and 2022. Sockeye salmon were found spawning in Whitewater Creek and pink salmon were found spawning in Wilm's Creek and Whitewater Creek.
- Mean selenium concentrations in coastrange sculpin tissue in Shazah Creek exceeded the selenium guideline for the protection of freshwater life.
- Exceedance of the methylmercury guideline for the protection of piscivorous wildlife occurred in coastrange sculpin within Whitewater Creek, Shazah Creek, and Wilm's Creek.
- Periphyton was found in limited quantities throughout the local study area with only three of the eight sites with periphyton present at the site.
- A decreasing trend of benthic invertebrate (EPT taxa) was observed in Whitewater Creek and is likely a function of substrate conditions within Whitewater Creek.
- The sediment quality results indicate elevated levels of various metals in both reference sites and test sites within the regional study area. Arsenic levels in Whitewater Creek exceeded the upper WSQG at all sites possibly indicating naturally elevated levels of arsenic at the site. Elevated levels of arsenic were also noted in Shazah Creek, and in the Tulsequah River at the upstream and downstream sites.

Table 20-1 lists the species documented within Tulsequah River (FIDQ, 2022). Due to hydraulic connectivity between Tulsequah River, Whitewater Creek and Taku River, fish presence is considered similar between watercourses.

Table 20-1 Fish Species Documented within Tulsequah River

Species	Scientific Name	Reduced Risk Work Window (BCMOF, 2018)
Arctic Grayling	Thymallus arcticus	January 1 – February 28
Chinook Salmon	Oncorhynchus tshawytscha	April 1 – September 15
Chum Salmon	Oncorhynchus keta	May 1 – July 31
Coastrange Sculpin	Cottus aleuticus	-
Coho Salmon	Oncorhynchus kisutch	April 1 – August 15
Cutthroat Trout	Oncorhynchus clarkii	September 1 – November 1
Dolly Varden	Salvelinus malma	July 1 – August 15
Eulachon	Thaleichthys pacificus	June 15 – March 15
Green Sturgeon	Acipenser medirostris	November 1 – April 30
Longfin Smelt	Spirinchus thaleichthys	-
Mountain Whitefish	Prosopium williamsoni	-
Northern Pike	Esox lucius	-
Pacific Lamprey	Entosphenus tridentatus	-
Pink Salmon	Oncorhynchus gorbuscha	March 15 – July 31
Rainbow Trout	Oncorhynchus mykiss	October 1 – November 30
River Lamprey	Lampetra fluviatilis	-
Round Whitefish	Prosopium cylindraceum	-
Slimy Sculpin	Cottus cognatus	-

Source: FIDQ, 2022.

The Taku watershed supports a variety of fish species, including all five species of Pacific salmon. None of these species are listed as 'Species at Risk' in Canada. Taku River Chinook salmon are a stock of Management Concern in Alaska, meaning there has been a chronic inability to meet escapement goals over recent years, despite the use of specific management measures. All fish and their habitats within the Local Assessment Area are protected by the Canadian federal *Fisheries Act.* Activities with the potential to harm fish or habitat must be authorized by the Minister of Fisheries and Oceans (Ausenco, 2025).

20.2.1.3 Vegetation

The vegetation in the Project area is typical of northern temperate rain forest, consisting primarily of fir, hemlock, spruce and cedar forest on the hillsides and aspen and alder groves in the river valley (BGC, 2015). Both mountain goat and moose are found year-round in the New Polaris study area (Gartner Lee Ltd., 2007).

Vegetation surveys were conducted in 2021 by Ausenco. A search of the Conservation Data Center's iMap indicates an occurrence of cryptic paw (Nephroma occultum) approximately 3.5 km south of the mine site and 4 km north of the Tulsequah-Taku Rivers confluence. This species is provincially blue-listed and is listed as endangered under Schedule 1

of Species at Risk Act. Table 20-2 summarizes vegetation and ecosystems that have been observed in the project area that are of federal or provincial conservation concern which may be sensitive to development.

Table 20-2: Vegetation Species and Ecosystems of Conservation Concern Observed in Project Area

Species or Ecosystem Common Name	Latin Name	Species at Risk Act (SARA) Schedule 1 Status	BC Status
Cryptic paw lichen	Nephroma occultum	Special concern	Blue
Floodplain	-	-	-
Old-growth Forest	-	-	-
Wetlands	-	-	-

Source: Ausenco, 2021

20.2.1.4 Ecosystems and Wetlands

EDI Environmental Dynamics Inc. (EDI) conducted desktop and field-based study at the Project in 2021 and 2024-2025. No federally or provincially listed rare plants were observed during the 2021 field program. Three Blue-listed ecological communities were mapped within the Coastal Western Hemlock Zone, Wet Maritime Subzone. A 2025 status review of potential species and ecological communities to occur (in the regional assessment area (RAA) as defined by the EIS, in progress) was conducted, with some changing in status. No invasive plants were observed during the 2021 field program. A desktop review in 2025 was conducted to update the list of invasive plants with the potential to occur in the RAA, along with their provincial designations and management priorities.

Wetland function surveys were conducted in 2024 on four representative wetlands in the RAA. All four surveyed wetlands received a “Moderate-High” wetland function rating, indicating a relatively high ecological value.

20.2.1.5 Terrestrial Wildlife and Wildlife Habitat

In 2021, wildlife studies were undertaken by Ausenco to obtain the necessary information including:

- Large mammal survey
- Bat survey
- Waterfowl survey
- Songbird and swallow point counts.

Large mammal surveys included a combination of winter moose (Alces alces) surveys and a remote camera program; Flannigan Slough has a significant moose population and the area East on the Taku River has been identified in the ATLUP as a mountain goat wintering habitat. Bat surveys involved the deployment of three remote acoustic recording units (ARUs) to assess species presence within the project Area. These ARUs were deployed from March to September 2021, during which time they recorded six species of bats including one federally listed species at risk, little brown myotis (Myotis lucifugus). Bird surveys in 2021 included a waterfowl survey, swallow point counts, and songbird point counts during the breeding season.

Amphibian species including western toad (Anaxyrus boreas), Columbia spotted frog (Rana luteiventris), wood frog (Rana sylvatica) and long-toed salamander (Ambystoma marcodactylum) have been documented within wetlands

adjacent to the Tulsequah River. Flannigan Slough, adjacent to the proposed barge landing, is noted to have presence of western toad, listed as of Special Concern within Schedule 1 of the *Species at Risk Act* (Rescan 1997).

Raptor, waterfowl, and passerine species have been documented in abundance along Tulsequah River (EDI, 2021). Grizzly bear, mountain goat (Oreamnos americanus) and moose populations have been identified within the proposed barge landing tote road (EDI, 2021). Table 20-3 summarizes the wildlife species that have been observed in the project area that are of federal or provincial conservation concern, which may be sensitive to development.

Priority wildlife baseline studies were expanded upon in 2023 to fill select data gaps identified in the 2020 and 2021 studies and incorporate the newly proposed barge route now identified as the marine transport route as access to the Project. Baseline studies for wildlife in 2023 included:

- acoustic bat surveys
- visual inspection of an identified bat residence
- aerial surveys for waterbirds and grizzly bear.

The 2023 bat surveys identified a total of six species of bats, including three Species at Risk, within the Project area.

Table 20-3: Wildlife Species of Conservation Concern Observed in the Project Area

Species' Common Name	Latin Name	SARA Schedule 1 Status	BC Status
Bald eagle	*Haliaeetus leucocephalus*	-	Yellow
Grizzly bear	*Ursus arctos*	Special concern	Blue
Hoary bat	*Lasiurus cinereus*	-	Blue
Little brown myotis	*Myotis lucifugus*	Endangered	Blue
Mountain goat	*Oreamnos americanus*	-	Blue
Northern goshawk, laingi ssp.	*Accipiter gentilis laingi*	Threatened	Red
Trumpeter swan	*Cygnus buccinator*	-	Yellow
Western toad	*Anaxyrus boreas*	Special concern	Blue
Wolverine	*Gulo gulo*	Special concern	Blue
Yuma myotis	*Myotis yumanensis*	-	Blue

Source: EDI, 2021

Bat investigations were completed in 2024 by EcoLogic Consultants Ltd for Canagold Resources Ltd, including deployment of automated bat detectors, and active monitoring of the summer bat roost in the existing abandoned workshop, mill building, and portal. Identified species include Little Brown Myotis (Ecologic Consultants Ltd., 2024).

It is unlikely that the bats using the maternity roost are hibernating within the same structure; the lack of thermally protected spaces within the workshop limits its value given the low winter temperatures in the region. No bats were observed in the workshop during the March 2024 site visit. The appearance of bats in the workshop in early April when average daily temperatures were still below zero (data provided by Canagold) suggest that the bats roosting in the workshop are local hibernators. The potential for bats roosting in the existing underground workings adjacent to the workshop during winter conditions should be considered in future bat monitoring programs.

The two primary options identified by EcoLogic Consultants Ltd for the management of the workshop roost were:

1. Leave the building in place and reinforce the structure (during the winter) to maximize roost longevity. The success of this option will be dependent on minimizing disturbance in the vicinity of the roost during the active periods. This may not be feasible given the operational requirements of the mine.
2. Create artificial roost structures that mimic the conditions of the roost in an area away from Project activity. This option would involve the installation of multiple artificial roost structures and subsequent monitoring to confirm use by local bats. Provincial and federal wildlife permits will be required in addition to the EA process.

20.2.1.6 Geochemistry

The geochemistry program to assess the risk of metal leaching/acid rock drainage (ML/ARD) from mine wastes was initiated by URS in 2006, with additional work carried out by Ausenco between 2023 and 2024. Table 20-4 provides a summary of the geochemistry baseline work carried out for the Project to date.

Table 20-4: Geochemistry Program Sampling and Analysis Summary

Reporting Date	Carried out by	Materials	Description	No. Samples	Analyses
March, 2007	URS	Waste rock (drill core) HW, FW	HW, FW simples bordering vein material	18	Mineralogy, ABA, metals, SFE
		Vein material (drill core)	C vein margins (assumed mineralized)	9	Mineralogy, ABA, metals, SFE
		Tailings	Flotation	5	Mineralogy, ABA, metals, SFE
June, 2007	URS	Waste rock (drill core),	Exploration drift/ FW decline, MV, GBRO, FD, AD	22	ABA, metals
November, 2007	URS	Vein material (drill core)	C vein, high S	1	ABA, metals, humidity cell
		Vein material (drill core)	C vein	16	ABA, metals, SFE
June, 2023	Ausenco	Low-grade ore	C vein	11	Mineralogy, ABA, metals, ABCC, SFE, humidity cell
			Y vein	6	
		Waste rock (drill core)	From development areas, basalt/ andesite	25	Mineralogy, ABA, metals, ABCC, SFE, humidity cell
		Co-disposal wastes	620g tailings + 380g waste rock	1	ABA, metals, SFE, humidity cell
		Historic waste rock	From 1950's, on site	5	ABA, metals, SFE
		Tailings	T1-flotation, from metallurgical test work	1	ABA, metals, SFE, humidity cell
Total No. waste rock samples				**65**	
Total No. low-grade ore samples				**17**	
Total No. mineralized vein samples (ore / low-grade ore / waste?)				**26**	
Total No. Co-disposal waste samples				**1**	
Total No. tailings samples				**6**	
Total No. historic waste rock samples				**5**	

Notes: ABA: Acid base accounting, SFE: Shake flask extraction, HW: Hanging wall, FW: Footwall, ABCC: Acid buffering characteristic curve. Source: Ausenco, 2024

The results of URS's geochemistry work in 2007 indicated that most mineralized vein rock samples were classified as non-potentially acid generating (NPAG) due to elevated carbonate content, with a minority of samples classified as potentially acid generating (PAG). Vein samples were mineralized and depending on the gold content may be considered as being ore, low-grade ore or waste rock. Footwall (FW) and hanging wall (HW) waste rock samples were generally classified as NPAG, with a minority of FW samples classified as PAG. All rock samples were found to potentially leach arsenic and antimony, with vein samples posing a higher leaching risk (URS, 2007).

Further geochemistry work was initiated by Ausenco in 2023 (Ausenco, 2023). This work included the collection of 17 low-grade ore (LGO) samples, 25 drill core waste rock samples, five historic waste rock samples and one tailings sample. Samples representative of waste rock, historic waste rock, co-disposal wastes, tailings, LGO and construction rock underwent static and kinetic testing to assess the ML/ARD risk (Ausenco, 2023). The majority of samples were classified as NPAG. Limited sampling indicated a potentially higher ARD risk from historic waste rock located on site (requires confirmation). All samples showed varying ML risk, with arsenic and antimony typically being the contaminants of concern. This conclusion is supported by the presence of elevated arsenic, and to a lesser extent antimony, in portal water samples collected from the legacy underground workings.

Tailings discharged by the legacy mining operations at New Polaris likely present sources of contamination to the Tulsequah River Floodplain. Previous analysis of grab sample of old tailings material (Personal communication, 2022) indicated elevated sulphur content. The extent of legacy tailings and their geochemistry and effects on water quality will be described in the project specific environmental effects assessment, currently underway.

A geochemistry gap analysis carried out by Ausenco in 2024 provided some recommendations for additional sampling and testing to confirm and further refine the current understanding of ML/ARD risks from the Project.

20.2.1.7 Surface Water Hydrology

The majority of the surface water draining the existing mine site flows within Whitewater Creek. Whitewater Creek is a significant tributary to the Tulsequah River, with a catchment area of approximately 250 km^2. The creek's hydrology is heavily influenced by the surrounding terrain, which includes steep mountains and glaciers. During the spring and summer months, the creek can experience significant increases in water levels due to snowmelt, while heavy rain events can also cause rapid changes in flow. The Creek cascades down a steep canyon located at the northern end of the study area, situated adjacent to the current camp infrastructure. After reaching the valley floor, the creek meanders southward, running parallel to the Tulsequah River. Whitewater Creek traverses through the current mine site, passing by the airstrip, and finally converges with the Tulsequah River approximately 3 km downstream. Additionally, Sawmill Creek, a minor tributary of Whitewater Creek, courses through the local study area, joining Whitewater Creek immediately south of the mine site. A small channel stemming from the Tulsequah River also feeds into Whitewater Creek, situated around 1 km south of the present mine site (Ausenco, 2023).

20.2.1.8 Surface Water Quality

Surface Water Quality Baseline sampling was completed in 2020 to 2022 and compared to water quality data obtained in 2006-2007 and 2015, to gain insight on potential long-term trends and assess water quality upstream and downstream of historical mining operations as well as in the receiving waters. Several water quality parameters do not comply with BC Water Quality Guidelines for freshwater aquatic life (BC WQG-FAL). Upstream of the historical mine

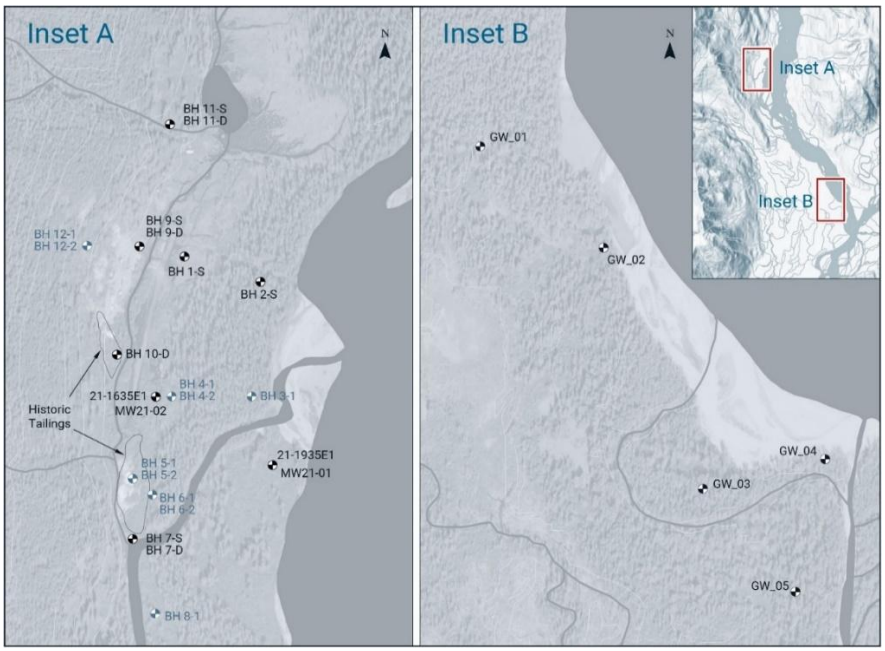
site, reference water quality stations show exceedances of the median concentrations of dissolved copper and zinc in Whitewater Creek. Downstream of historical mine operations, total arsenic exceeds BC WQG-FAL, in addition to dissolved copper and zinc. There are several maximum concentration metal exceedances through the historical mine area as well. Longer term trends, from data going back to 2006, suggest that total arsenic concentrations may be increasing while dissolved zinc and copper are decreasing. Trends in the Tulsequah River for total arsenic, dissolved zinc and copper concentrations, at two different downstream monitoring stations, show contrasting results reflecting the need for more frequent monitoring to better establish these trends (Ausenco, 2023).

20.2.1.9 Groundwater

A 2021 groundwater quality monitoring program involved conducting water quality sampling at nine groundwater wells between December 2020 and December 2021. Surface water sampling at selected locations in the project area has continued through 2022, 2023 and is continuing during 2024. Additional sampling locations have been added at the airstrip location with sampling beginning mid-2024. CCME-FAL guidelines were used as thresholds for concentrations of detectable parameters in groundwater samples. A map of the groundwater sampling locations is presented in Figure 20-1.

At the legacy tailings deposits to the south of New Polaris (BH7-D, BH7-S & BH10-D), dissolved zinc and manganese exceedances were identified. At the old mill site (BH9-D & BH9-S), exceedances of dissolved zinc and dissolved manganese were identified. At BH-1 and BH-2, exceedances of dissolved manganese were identified. Dissolved zinc concentrations also exceeded CCME-FAL at BH-1. Additionally, dissolved manganese and dissolved zinc exceedances were identified at MW21 02.

Figure 20-1 Ground Water Sampling Locations



Source: Ausenco, 2024.

Conceptually, groundwater at the mine site flows from upland areas located to the west through bedrock and overburden soils to the site. Piezometric surfaces constructed for May/June 2022 and August/September 2022 consistently indicate that shallow groundwater discharges to Whitewater Creek and to channels of the Tulsequah River. Whitewater Creek flows from a wetland to the north of the site towards the south where it discharges into a secondary channel tributary to the Tulsequah River. The catchment includes shallow groundwater flow from the west and up to 300 m to the east of the creek. Deeper groundwater may flow beneath Whitewater Creek and discharge to the Tulsequah River.

To assess the hydraulic conductivity of overburden soils, slug tests were conducted in September 2022 at ten monitoring wells (seven existing and two new monitoring wells installed in 2021). The hydraulic conductivity estimates calculated from the results of the slug tests range from 4.9×10^{6} m/s (BH11-S) to 8.2×10^{-4} m/s (BH10-S). To provide estimates of hydraulic conductivity of bedrock, packer tests were conducted at select intervals in drillholes 21-1635E1 and 21-1935E1 from October 14 to 25, 2021. Bedrock at the site is predominantly mafic tuff with aplite, mudstone, and diabase present. The mafic tuff hydraulic conductivity varies from 2.1×10^{-9} m/s to 1.5×10^{-8} m/s with a geometric mean of 6.4×10^{-9} m/s. The hydraulic conductivity of competent aplite ranges from 1.0×10^{-8} m/s to 1.2×10^{-6} m/s. A single test including mudstone indicated a hydraulic conductivity of 1.4×10^{-8} m/s. Fractures and a quartz-carbonate vein in mafic tuff displayed hydraulic conductivities ranging from 9.3×10^{-9} m/s to 3.4×10^{-7} m/s, or approximately an order of magnitude higher than competent mafic tuff. A fracture zone in the aplite had an estimated conductivity of 6.1×10^{-6} m/s while a vein in the mudstone had a conductivity of 5.4×10^{-6} m/s.

In terms of groundwater quality, two groundwater regimes are of note at the site: groundwater in bedrock and groundwater in overburden. All available monitoring wells at the site, with the exception of monitoring well BH11-D, are screened in overburden. Monitoring well BH11-D is screened in shallow bedrock and represents the only location where samples of groundwater from bedrock can be obtained. However, the seepage from the historical mine workings portal originates from bedrock groundwater that seeps into the mine workings, in effect providing samples of bedrock groundwater. The laboratory analytical results for samples of seepage from the portal are therefore used to assess the baseline condition of bedrock groundwater.

Groundwater in overburden has total and/or dissolved concentrations of the following parameters that exceed the CCME-FAL guidelines:

- Total Suspended Solids
- Aluminum
- Arsenic
- Cadmium
- Chromium
- Copper
- Iron
- Lead
- Manganese
- Silver
- Thallium
- Zinc

Groundwater in bedrock has only total and dissolved arsenic concentrations that exceed the CCME-FAL guidelines (Ausenco, 2023).

20.2.1.10 Terrain Stability Mapping

Orthic humo-ferric podzols, with a B horizon greater than 10 cm thick, are the main soils in the mine site area and road infrastructure to the north, including the CSF. Orthic humic gleysols with an organic-rich A horizon at least 10 cm thick overlying gleyed B horizons and parent material; and typic mesisols representing organic, wetland soils occupy floodplains and back-channel areas below the mine site and throughout much of the southern region of access roads. Cumulic regosols, with the parent materials underlain by buried A horizons, are dominant in treed and vegetated floodplain areas.

20.2.1.10.1 Surficial Material

The mine site area is underlain by poorly drained cobble, silt, sand glaciofluvial/fluvial deposits and silt, sand, gravel tills greater than 1 m thick and steep, potentially unstable terrace edges may be present. The floodplain region below the mine site and the Flannigan Slough along the southern region of the barge-to-mine site access road are underlain by imperfectly to poorly drained cobble, silt, sand glaciofluvial/fluvial deposits. The alluvial fans of Whitewater and Wilms creeks have thick accumulations of organic material that are poorly drained. The road from the mine site to the Co-storage Facility (CSF) crosses the toe of moderate rockfall hazard slopes and then back onto moderate to moderately steep colluvial and till covered slopes above a glaciofluvial terrace.

20.2.1.10.2 Geohazards & Terrain Stability

Potential geohazards and terrain stability concerns which may impact or be influenced by the proposed development include flooding, snow avalanches, debris slides, debris flows, rockfall, channel avulsion and/or migration and terrace edge instability (Madrone Environmental Services Ltd, 2024).

20.2.2 Environmental Considerations of Fuel and Explosives Storage

20.2.2.1 Fuel Storage

The proposed project includes an on-site tank farm for storing diesel fuel during the construction and operations of the project. An earth berm and impoundment would be constructed around the fuel tank farm using locally sourced sand and gravel. The berm and impoundment would be lined with an impermeable membrane to provide containment in the event of a spill.

During construction, fuel would be brought to site in containers via barge during the designated barging season. It would be offloaded and trucked to the bulk storage fuel tanks. Fuel would be distributed around the site by tanker truck. During operations, fuel will be flown to site in fuel bladders inside the ATR72 aircraft.

For daily mine construction and operations activities, smaller “day tanks” are proposed to be located at various locations throughout the site to supply fuel for equipment and power generation as required for the camp, mine, plant, truck maintenance facility, barge landing and other infrastructure sites. These day tanks would be filled by the tanker

as required. Day tanks would be installed in alignment with industry accepted standards, including containment to prevent release of any fuel to the surrounding environment.

20.2.2.2 Explosives Storage

Excavation of the below ground mine will require using explosives, therefore the proposed project contains an explosives storage facility. The facility would be secure and gated and would be designed in accordance with guidelines from Natural Resources Canada, including maintaining minimum distance to inhabited areas.

The facility design is comprised of eight 40-foot shipping containers. To ensure safe storage of the explosives, each container is designed with a surrounding earth berm. Each container is designed to store up to 23,000 kg of explosives.

Transport of explosives from the storage facility would be undertaken in accordance with relevant regulatory requirements using trucks.

20.2.3 Environmental Considerations of Mineral Processing and Tailings Management

The processing of ore will include a typical mill feed system closed off to the environment. The mineral processing circuit will include crushing, grinding and flotation to produce a bulk sulphide concentrate which will be shipped off-site for sale. The benefits of shipping concentrate for off-site refining include eliminating the use of cyanide and acid relative to conventional cyanide leach and doré production. Compared to on-site doré production, shipping concentrate reduces the Project's electrical power requirements by 50% which also reduces the on-site process plant's associated requirements for transportation and consumption of diesel fuel and reduces greenhouse gas generation.

Tailings are planned to be deposited in underground mining voids as thickened tailings and in the surface co-storage facility (CSF), as stacked filtered tailings. The CSF will be approximately 1 km north of the mine and process plant. Tailings to be disposed of in the CSF will be filtered to a semi-dry state then transported to the CSF via haul truck. The filtered tailings will be co-stored with waste rock from underground mining activities. Water recovered from the thickening and filtering of the tailings will be recycled for use in the process with any excess water being treated to ensure the quality meets all regulatory requirements before it is discharged into the environment.

The design of the CSF and water management facilities has considered the following:

- Staged development of the facility over the life of the project.
- Flexibility to accommodate operational variability in the waste rock and filtered tailings (filter plant shutdowns and ore variability, along with placement during variable climate conditions).
- Control, collection, and removal of contact water from the facility during operations for reuse as process water to the maximum practical extent.
- Any underground waste rock that comes to the surface will be stored within the CSF along with the filtered tailings.
- The facility will be 100 percent lined with a geomembrane to capture any seepage and runoff. The capture water will be utilized as part of the water for the plant or will be treated and released to the environment.

Co-placement of waste rock and tailings in one facility has a number of advantages over placing these in separate facilities including:

- Reducing the overall footprint area of tailings and waste rock storage allows for more simplified water management.
- Based on geochemistry testing the majority of waste rock is shown to be NPAG. The abundant carbonate content in the NPAG rock will provide excess alkalinity to mitigate any potential risk from the minor quantities of PAG waste rock. Co-storage of waste rock with tailings will also help to mitigate any low residual ARD risk by limiting oxygen ingress (retards sulphide mineral oxidation).

Detailed design of the CSF is advancing in parallel to the environmental assessment process as part of the Feasibility Study. Key design parameters to inform the detailed design of the CSF include:

- Non-contact surface water diversion ditches are built outside the facility to prevent surface runoff water from contacting the stored waste rock and tailings.
- Layers of waste rock and tailings would be compacted and sloped to manage runoff within the structure.
- Ditches are excavated around the inside of the CSF and lined with impermeable material to direct any seepage water to a lined settling pond that can be recycled or fed into a water treatment plant if additional treatment is needed.
- Progressive restoration of the CSF will take place to minimize the active areas.
- At closure, a rock/soil cover would be placed over the entire facility, encapsulating the tailings and mine waste rock. Efforts will be made to progressively reclaim slopes progressively, where possible.

The design standards for the CSF are based on the relevant federal and provincial guidelines for construction of mining tailings storage facilities in Canada.

The tailings may have constituents of concern (e.g. arsenic and antimony) that could enter the environment through normal precipitation seeping through the facility becoming contaminated. Therefore, the facility will be 100 percent lined with a geomembrane to minimize the risk of seepage reaching underlying groundwater. Excess water from the CSF will be recycled to the mill and/or will be treated on site prior to being released to the environment.

The CSF will be built in multiple phases.as described in Section 18.NAG waste rock will be utilized to construct exterior berms as the facility increases in height to contain the filtered tailings.

20.2.4 Environmental Monitoring

Existing monitoring completed as part of environmental baselines is described in the previous sections. Additional monitoring required during construction and operations is anticipated to include:

- Surface water quality monitoring, including monitoring of the barge landing area and downstream receiving environment.

- Groundwater quality monitoring, including installation of new groundwater monitoring wells upgradient, adjacent to, and downgradient of site infrastructure including process facilities and the CSF.
- Sediment quality monitoring in nearby streams, including benthic invertebrate surveys.
- Fish tissue sampling in potentially impacted streams and receiving waterbodies.

Monitoring is anticipated to be required throughout the construction, operations, and closure period. It is anticipated that environmental monitoring will be required for several years following closure of the mine and CSF to monitor for ML/ARD risk; monitoring requirements may be reduced should results indicate reduced environmental risk.

20.2.5 Water Management

To protect Whitewater Creek and other environmentally sensitive areas such as wetlands, the proposed project design includes protection zones that will be protected by berms with drainage ditches. This proposed design would capture surface drainage from the site facilities and divert it away for use in the facility process or allow water to settle out any solids prior to any discharge. On-site water that does not meet discharge criteria will be directed to a water treatment plant before discharge to the environment.

20.2.5.1 Diversion Channels

A series of diversion channels are proposed to divert non-contact water around and away from mining infrastructure. Diverted surface flow will not be moved to another watershed and the ultimate discharge locations will be as close as the existing flow path as possible. It is proposed that the diversion channels will be sized to convey a 1:200-year peak flow.

20.2.5.2 Collection Channels and Sediment Ponds

Material used for the construction of site infrastructure (i.e. road, laydown areas, foundations) will utilize borrowed material from around the site and be limited to non-acid generating and non-metal leaching material. It is expected that surface runoff from these areas will require sediment removal prior to treatment. Collection channels and ditches will be built to capture runoff from disturbed areas around the site and conveyed to settling ponds to facilitate removal of sediment before being discharged to the receiving environment.

The expectation is collected surface water will only require sediment removal prior to discharge; however, if elevated levels of other constituents of concerns are present, water will be pumped to the water treatment plant for additional treatment prior to discharge to the environment.

20.2.5.3 Co-storage Facility Water Management (CSF)

The CSF located north of the plant site, will require active treatment for contact water runoff from the facility. A collection pond will be constructed with the runoff being pumped towards the active water treatment facility at site before being released to the receiving environment, once discharge criteria is met. Above the CSF, a diversion channel

is also proposed to divert the non-contact runoff away from the area and to minimize the amount of contact runoff to be collected and managed.

20.2.5.4 Mine Dewatering

Mine dewatering will require active treatment and underground mine water will be pumped from the underground workings to the water treatment plant. The existing underground mine workings are currently flooded and will require dewatering prior to the advancement of an underground access ramp. A dewatering rate of 216 m^3/hour has been predicted based on extrapolated data from existing records documented when the mine was last dewatered. An updated hydrogeological model is planned for the next stages of the project, with these updates being incorporated into a revised water balance.

20.2.5.5 Water Treatment Plant

A water treatment plant will treat water from both the underground dewatering and contact water runoff from the CSF. The anticipated flow rates will be estimated from the water balance. At this stage it is envisioned that treatment will be a modular plant and any sludge or waste generated will be deposited in the CSF. Treated water will be discharged to the Tulsequah River via a pipeline. Diffusers or specific outlet configurations are being reviewed. The best available treatment technology will be installed to meet the projects treatment needs to meet permit discharge criteria. The specifications for the plant will be developed so as meet applicable effluent discharge and receiving water quality requirements.

Feed to the treatment plant will include mine discharge water, the CSF runoff, seepage water, and as required other contact water from around the site. Results from the baseline studies sampling of groundwater in bedrock showed the quality was generally good. The only parameters that exceeded CCME guidelines were total and dissolved arsenic concentrations.

Ground water in overburden has some concentrations of other parameters above the CCME-FAL guidelines. This suggests that the elevated concentrations in overburden groundwater are likely from surface sources related to historic mining, and not from bedrock. During dewatering of the mine in 2007 elevated levels of arsenic were present in the discharge water. The assay records taken during the dewatering, combined with the hydrogeology/groundwater baseline data will form the basis for determining the treatment requirements for the underground dewatering. The mine water and CSF runoff and seepage quantity and quality will be used to determine the treatment requirements.

The CSF will be a lined system that will minimize the risk of seepage leaving the facility. Geochemistry data show that acid rock drainage will not be an issue. Minimizing the leaching of arsenic and antimony from mine wastes will be an important consideration in the facility's design, and ongoing testing is being done to assess the contact water from the facility that will need treatment.

20.3 Permitting Considerations

The following sections describe the regulatory and policy framework for the project, and under the BC Environmental Assessment Act (EAA; SBC 2018, c 51) and the Canadian Impact Assessment Act (IAA; SC 2019, c 28, s 1), as well as the other permits and approvals anticipated to be required.

The project is a re-development of the former mine and town site of the past-producing Polaris-Taku Mine, which operated intermittently between 1937 and 1951. Cominco upgraded and leased the mill between 1951 to 1957 to process ore from the Tulsequah Chief Mine. Polaris-Taku did not have permits under the BC Mines Act as this mine pre-dated the BC Mines Act which only came into effect in the mid-1960's. Therefore, although the project is the re-development of an existing mine, it is considered, from a regulatory perspective, a new mine due to the lack of a previous Mines Act permit or environmental assessment.

Canagold has received a mineral exploration approval in the form of an Exploration Permit issued on September 11, 2020, under the Mines Act to complete exploration activities and geotechnical drilling to define and understand the potential Mineral Resource associated with the Project.

New Polaris does not require a review under the Impact Assessment Act of Canada as it would have a daily production capacity of 1,000 t/d, which falls below the 5,000 t/d limit outlined in the federal Physical Activities Regulation.

20.3.1 Environmental Permits and Regulations

The provincial EA process was initiated with the submission of the Project Description to the BC EAO, under the 2002 Environmental Assessment Act (EAA), on March 27, 2023. Canagold has also submitted the Engagement Plan as required by section 13(1) of the EAA and in accordance with EAO guidance. The Engagement Plan included a summary of engagement to date and Canagold's engagement plans.

The New Polaris Gold Mine constitutes a reviewable project pursuant to Part 3 of the Reviewable Projects Regulation (BC Reg. 607/19) of the Act, as a new project with a production capacity greater than or equal to 75,000 t/a of mineral ore. The project has a proposed production of 365,000 t/a, and is therefore reviewable

Canagold have been progressing an Environmental Assessment for the New Polaris Project. The BC EAO issued a Readiness Decision Report in 2024 recommending that New Polaris proceed to an EA. TRTFN provided its view that the project is ready to proceed to an EA. The EAO is aware that, at this stage, there are outstanding concerns from TRTFN and technical advisors. The EAO is of the view that these issues can best be addressed in subsequent phases of the EA and will carry these issues forward into process planning (BC EAO, 2024).

20.3.1.1 Climate Action Legislation

In 2024, British Columbia implemented its Output Based Pricing System (OBPS), aligning with Canada's nation approach to industrial emissions pricing. The British Columbia OBPS is integrated with Canada's broader emissions reduction framework and supports the country's Nationally Determined Contributions (NDCs) under the Paris Agreement. These NDCs represent Canada's commitments to reducing greenhouse gas emissions. The OBPS targets emissions from large industrial sources.

British Columbia's OBPS applies to facilities that produce a regulated product and emit more than 10,000 t CO_2e annually. Covered facilities are required to report emissions from specified sources and assess their emissions intensity against a product specific performance benchmark. Emissions that exceed the annual benchmark are subject to compliance obligations, with payment based on the federal price of C$95/t CO_2e (2025) increasing to C$170/t CO_2e by 2030. Facilities that operate below their benchmark may generate surplus units, which can be sold. Limited use of

carbon offsets is also permitted for compliance. The New Polaris Project is expected to be a covered facility under the British Columbia OBPS, as outlined in the 2025 CleanBC Industrial Incentive Program.

20.3.2 Mining Permits

This section summarizes the existing permits in place for the project and the federal and provincial legislation and associated permits, licenses and approvals that will apply or potentially will apply to the construction and operations of the project, as currently proposed.

20.3.2.1 Existing Permits

Canagold currently holds a Multi-Year-Area-Based Mines Act mineral exploration permit (MX-1-208) that expires on March 31, 2028. A description of Canagold's existing mineral tenures for the New Polaris Project is provided in Section 4 of this report.

20.3.3 Anticipated Permits and Authorizations

A summary of the potential federal and provincial major and minor permits required for the project is provided in Table 20-5.

Table 20-5: Provincial and Federal environmental approval requirements for the New Polaris Project

Legislation	Issuing Agency	Authorization	Purpose
Provincial			
BC Environmental Assessment Act	BC Environmental Assessment Office (BC EAO)	BC Environmental Assessment Certificate (BC EAC)	To assess the potential environmental and socio-economic impacts of the project prior to permitting.
Mines Act (MA)	BC Ministry of Energy, Mines and Low Carbon Innovation (EMLI)	*Mines Act* Permit M-29	Authorization for construction and mining activities.
Environmental Management Act (EMA)	BC Ministry of Environment and Climate Change Strategy (ENV)	*Environmental Management Act (EMA)* PE-00261	Authorization for effluent discharges.
		Environmental Management Act (EMA) PA-105340	Authorization for air discharges.
Drinking Water Protection Act	Interior Health Authority (IHA)	Drinking Water Permits	Drinking water systems.
Forest Act and Forest and Range Practices Act	Ministry of Forests, Lands, and Natural Resources Operations Region (FLNROR)	Occupant License to Cut (OLTC)	Cutting and removal of Crown timber from Crown land.

Legislation	Issuing Agency	Authorization	Purpose
Heritage Conservation Act (HCA)	Ministry of Forests (FOR)	Heritage Inspection Permit S 12.2	To assess the archaeological significance of the land and to recover information which otherwise might be lost as a result of site alteration of destruction.
		Site Alteration Permit (SAP) as needed S 12.4	Authorizes the removal of residual archaeological deposits if sites are identified during the inspection and/or investigation.
Land Act	Ministry of Forests (FOR)	Fee Simple Sale	The *Land Act* (1996) is the primary article of legislation that is used by the government to convey land to the public for community, industrial, and business use. If a mining project requires the use of Crown land for mining infrastructure in an area not covered under mineral tenure, a *Land Act* permit will be required.
		ROW update	
		License of Occupation	
Mineral Tenure Act	Ministry of Energy, Mines and Low Carbon Innovation (EMLI)	Mineral Claim Acquisition	Subsurface rights to minerals in a defined unit, up to 10,000 tonnes per year per unit.
	Ministry of Energy, Mines and Low Carbon Innovation (EMLI)	Mining Lease changes	Conversion of mineral claim to a mining lease is necessary before production can exceed above limits
Water Sustainability Act (WSA)	Ministry of Forest (FOR)	Change Approval S11	Changes in and about a stream are carried out under change approvals and notifications. These approvals authorize work in and around streams.
		New Water License S10	Water licences allow licensees to divert, store, and use specific quantities of water for one or more water use purposes. A water licence may also authorize works related to the diversion and use of the water.
Wildlife Act	Ministry of Forests (FOR)	Authorization Permits for general wildlife (relocation)	If work occurs within identified wildlife areas. A blanket permit applies across the entire project.
Federal			
Fisheries Act	Fisheries and Oceans Canada (DFO)	Section 35 Authorization	Where works may cause the harmful alteration, disruption, or destruction of fish habitat.
		Scientific License	License required to harvest fish for experimental, scientific, educational or public display purposes.

Legislation	Issuing Agency	Authorization	Purpose
	Metal and Diamond Mining Effluent Regulations, SOR/2002-222	Schedule 2 Amendment Authorization to deposit an effluent that contains a deleterious substance	Authorization for a tailings impoundment area in "waters frequented by fish". The definition of tailings in the case of the MDMER involves all mine related waste including mine rock and untreated effluent. Applies to mines that exceed an effluent flow rate of 50 m^3 per day, based on effluent deposited from all the final discharge points of the mine, and that deposit a deleterious substance in any water frequented by fish or in any place under any conditions where the deleterious substance or any other deleterious substance that results from the deposit of the deleterious substance may enter any such water. At this time, it is not known whether a Schedule 2 amendment will be required for the project.
Explosives Act	Ministry of Forests (FOR)	Manufacturing/Storage License	Required for the Manufacturing and Storage of explosives
Migratory Birds Convention Act	Canadian Wildlife Services	Damage or Danger Permit	Required for all activities with the potential to damage or danger migratory birds listed in the *Migratory Bird Convention Act, 1994*.
Canadian Navigable Waters Act (CNWA)	Transport Canada (TC)	Notice of works to the Minister under *CNWA* (Major and Minor Works in scheduled and unscheduled waters) Navigation Protection Program	Under *CNWA*, owners of works propose to construct, place, alter, rebuild, remove or decommission works that are in, on under, through or across any navigable waters may be required to apply to TC, for scheduled waterways, or go through public resolution for unscheduled waters.
Species at Risk Act (SARA)	Environment and Climate Change Canada (ECCC)	Species at Risk Permit	Required if the project crosses critical habitat, as designated under *SARA*. Requires regulatory consultation with ECCC.

Due to the seasonal nature of barge access to the New Polaris Gold Mine Project site, Canagold Resources Ltd. (Canagold) are proposing to pursue an Early Works Application package, in order to advance approvals for construction activities for the following facilities:

- Airstrip and support facilities
- Barge Landing and associated infrastructure
- Access road from airstrip and barge landing to Mine Site
- Construction camp.

The remaining facilities would be applied for under the Consolidated Application package, which would include activities for the construction and operation of the Mine Site. The EAO issued a Regulatory Coordination Plan for the New Polaris Gold Mine in January 2025 which describes the Early Works and Consolidated Application process (EAO, 2025).

The primary regulatory instruments anticipated for the proposed project for permit applications of both Early Works and the Consolidated Application Package (including if any permit applications would be reviewed during the EA[1]) are listed in Table 20-6, below. Proposed submission dates for these varous applications are currently in review and being determined.

Table 20-6: Primary Regulatory Instruments and Expected Permit Application Requirements – Canada

Regulatory Act	Early Works Required Permit	Consolidated Application Required Permit
Mines Act	Not applicable.	Approving a Mine Plan and Reclamation Program.
Drinking Water Protection Act, Drinking Water Protection Regulation	Water System Construction Permit Water System Operating Permit.	Not applicable.
Drinking Water Protection Act	Food Facility - Health Approval Application.	Not applicable.
Environmental Management Act	Air Discharge Permit.	Air Discharge Permit.
Environmental Management Act	Effluent Discharge Permit -May need a permit for construction activities on site and to manage contact water during the construction process – the Ministry of Environment and Parks will need this information to evaluate whether an	Effluent Discharge Permit.

[1] Permit applications may be reviewed at the same time as the EA; however, decisions on permits can not be made until after an EA decision per Section 8 of the Environmental Assessment Act, 2018.

Regulatory Act	Early Works Required Permit	Consolidated Application Required Permit
	Environmental Management Act permit is required.	
Environmental Management Act	Hazardous Waste Registration.	Hazardous Waste Registration.
Environmental Management Act	Refuse Permit.	Refuse Permit.
Environmental Management Act	Sewage System Registration.	Sewage System Registration.
Heritage Conservation Act	Heritage Inspection Permit, if required.	Heritage Inspection Permit, as needed.
Heritage Conservation Act	Heritage Investigation Permit, if required.	Heritage Investigation Permit, as needed.
Heritage Conservation Act Site Alteration Permit	As needed.	As needed.
Forest Act and Forest and Range Practices Act	Road Use Permit.	Not applicable.
Forest Act and Forest and Range Practices Act	Occupant Licence to Cut.	As needed during construction in new cut areas.
Forest Act and Forest and Range Practices Act	Special Use Permit (SUP).	
Land Act	- Lease Required for the barge landing site, new airstrip and portion of tote road.	
Land Act	License of Occupation , as needed.	License of Occupation, as needed.
Land Act	Investigative Use Permit or Temporary Use Permit , as needed for exploration.	Investigative Use Permit or Temporary Use Permit, a needed.
Mines Act	Not applicable.	Explosives Storage and Use Permit.
Park Act	Permit/Parks boundary amendment.	Permit/Parks boundary amendment, as needed.
Park Act	Park Use Permit.	Park Use Permit, as needed.
Water Sustainability Act	Change Approval.	Change Approval.
Water Sustainability Act	Water Use Licence.	Water Use Licence.

Regulatory Act	Early Works Required Permit	Consolidated Application Required Permit
Wildlife Act	Fish Collection Permit, if required.	Fish Collection Permit, as needed prior to fish salvage related to changes in and about a stream.
Wildlife Act	Wildlife Permit, if required.	Wildlife Permit, as needed prior to changes in and about a stream.
Canadian Navigable Water Act	Application for Approval, if required.	Not required.
Explosives Act Explosives Licence	Not required.	Explosives Licence.
Fisheries Act	Authorization	Authorization
Fisheries Act *MDMER Schedule 2*	Not required.	Amendment to the MDMER, Schedule 2, if required.
Migratory Birds Convention Act	Migratory Bird Permit, permit not expected. Ongoing documentation of environmental due diligence required to demonstrated compliance with the Act.	Migratory Bird Permit, permit not expected. Ongoing documentation of environmental due diligence required to demonstrated compliance with the Act.
Radio Communication Act	License	License
Species at Risk Act	Species at Risk Permit - permit not expected. Ongoing documentation of environmental due diligence required to demonstrate compliance with the Act.	Species at Risk Permit - permit not expected. Ongoing documentation of environmental due diligence required to demonstrate compliance with the Act.
Transportation of Dangerous Goods Act	Transportation of Dangerous Goods Permit.	Transportation of Dangerous Goods Permit.

Source: EAO, 2025.

Project permitting considerations relating to United States regulatory instruments are listed in Table 20-7.

Table 20-7: Primary Regulatory Instruments and Expected Permit Application Submissions – United States

Act and Required Permit	Early Works Required Permit	Consolidated Application Required Permit
NOAA Fisheries Regulator – Magnuson-Stevens Act	To be decided (TBD) pending discussions with regulatory agency.	TBD pending discussions with regulatory agency.
NOAA - Endangered Species Act		
Alaska Water Quality Standards		

Source: EAO, 2025.

20.4 Social Considerations

20.4.1 Engagement with Project Affected Peoples

The Initial Project Description and Engagement Plan for New Polaris Gold Mine were approved by the EAO on March 27, 2023. A 30-day public comment and engagement period was then held from May 9, 2023, to June 8, 2023, on the Initial Project Description. The EAO sought feedback from the public during two public sessions, which included one in-person open house in Atlin, BC on May 25, 2023, and one virtual information session on June 1, 2023. During this engagement period, technical advisors (including representatives from TRTFN), local governments, health authorities, provincial and federal government agencies, and U.S. federal and Alaska state agencies were invited to provide written comment. The EAO's Summary of Engagement was provided to Canagold on June 26, 2023 (BC EAO, 2024).

On September 26, 2024, the Environmental Assessment Office (EAO) issued a notice of the Readiness Decision for New Polaris to proceed to an environmental assessment under the Environmental Assessment Act (2018) (the Act). The two participating Indigenous nations for the New Polaris environmental assessment are Taku River Tlingit First Nation and Teslin Tlingit Council (TTC) (EAO, 2025). It is noted that Taku River Tlingit First Nation opposes TTC's assertion of Section 35 Rights in the area of New Polaris. However, on other aspects of the process planning documents, Taku River Tlingit First Nation indicated its agreement.

20.4.1.1 Taku River Tlingit First Nation (TRTFN)

Tlingit First Nation territory overlaps the New Polaris Gold Mine. On March 13,2007 the TRTFN, produced a comprehensive Taku River mining policy, the Taku River Tlingit First Nation Mining Policy (Tak River Tlingit First Nation, 2019). The TRTFN Mining Policy is the first of its kind in Canada. The policy outlines a clear set of rules for engagement, principles, procedures, and applications for mining related activities with the TRTFN within their territory, The goal of the policy is to provide a transparent and open guide for all potential and future partners.

On April 24, 2023, Taku River Tlingit First Nation confirmed its status as a participating Indigenous nation. This confirmation is posted on the EAO's EPIC website. As part of its own evaluation of the New Polaris Gold Mine, Taku River Tlingit First Nation established a Technical Working Group. This Technical Working Group will also engage bilaterally with the EAO and provide technical advice to the EAO on topics relating to Taku River Tlingit First Nation's rights and interests (BC EAO, 2023).

The New Polaris Project area has been and continues to be used by the TRT for cultural activities, and the Taku River is home to productive salmon fisheries, which are critically important to TRT community members. The New Polaris site is located within a management and protection area proposed by the TRTFN in 2023, known as the T'akú Tlatsini Indigenous Protected and Conserved Area (Takú IPCA). The Taku River/T'akú Téix' Conservancy also lies immediately south of the airstrip, which is part of the project site, and encompasses the British Columbia portion of the Taku River mainstem to the Alaska border. Both the Takú IPCA and Taku River/T'akú Téix' Conservancy aim to protect critical watersheds and ecosystems while allowing for economic diversification by establishing protected areas in tandem with areas where responsible development can occur.

20.4.1.2 Teslin Tlingit Council (TTC)

Teslin Tlingit Council (TTC) is a self-governing First Nation based in Teslin in Southern Yukon Territory. The Village of Teslin is set on the confluence of Nisutlin Bay and Teslin Lake, located 183 kilometers east of Whitehorse on the Alaska Highway. BC EAO has identified the TCC as a participating nation for the purposes of the Environmental Assessment process for the Project.

20.4.1.3 Input from Alaskan Tribes

It is the EAO's practice to notify Alaska Tribes who are nearby to projects undergoing a BC EA in the transboundary region. Throughout early engagement and the Readiness Decision phase the EAO provided information to five U.S. Tribes: Angoon Community Association, Central Council of Tlingit and Haida Indian Tribes of Alaska, Douglas Indian Association (including the Southeast Alaska Indigenous Transboundary Commission), Hoonah Indian Association, and the Organized Village of Kake (together, the Alaska Tribes).

During the public comment period for Early Engagement, the Alaska Tribes raised concerns related to the project's rationale and viability, regulatory engagement, requirements and oversight, barging, cumulative effects between Tulsequah Chief and New Polaris, and the potential impacts of New Polaris on culturally significant resources such as salmon and wild medicines. Canagold responded to these comments and provided additional information in the DPD in response to this feedback. These concerns will also be discussed further during review of the Application, if the project proceeds.

On July 24, 2024, the EAO sent a notification to each of the listed Tribes regarding the submission of the DPD, as this is a key milestone for notification as set out in the EAO's initial letters to Alaska Tribes (BC EAO, 2024).

20.4.2 Exploration Agreements

Canagold and TRTFN entered into a collaborative engagement agreement for the New Polaris gold exploration site on March 29, 2023, (TRTFN, 2023). The Collaborative Engagement Agreement provides capacity funding for TRTFN staff to fully engage with Canagold during the Feasibility and Environmental Assessment processes.

20.4.3 Traditional Land Use, Cultural Heritage, and Indigenous Interests

The project is located in the traditional territory of the TRTFN, and more specifically within the Tulsequah Valley/Tass Teiyi Héeini Resource Management Area, as described in the Wóoshtin wudidaa Atlin Taku Land Use Plan (ATLUP). The Project has the potential to affect various aspects of the TRTFN livelihood and traditional resource use. The potential effects on Indigenous Interests will be assessed and mitigation developed through a process of continuous and meaningful engagement and consultation. For the Tlingit people, the activities of fishing, trapping, hunting, berry picking and plant gathering have taken place in the region over generations.

Many TRTFN citizens exercise their aboriginal rights and meet substantial parts of their families' economic needs through traditional land-based activities such as hunting, fishing, gathering of plants for food and medicine, and trading in goods and services associated with these activities (TRTFN and Province of BC, 2011). Perhaps more importantly, the traditional territory of the TRTFN is also used to practice and promote their culture. "For the Tlingit community, the

traditional land-based way of life is not mere subsistence, or 'living off the land', but represents a complex set of social activities and relationships that lie at the heart of their culture and their khustìyxh ('way of life')" (TRTFN and Province of BC, 2011).

Canagold is currently in discussions with TRTFN to move forward with the collection of Traditional Knowledge and Use information in relation to the project. Results of this information will inform key aspects of the project and environmental assessment process.

The EAO Summary of Engagement outlines Taku River Tlingit First Nation preliminary interests and concerns (see Table 20-8).

Table 20-8 Summary of Taku Tlingit First Nation's Preliminary Indigenous Interests Received During Early Engagement

Topic	Interest details
Governance and process concerns	Acknowledgment of Taku River Tlingit First Nation's governance role in this area, as the only BC First Nation in whose territory the New Polaris Gold Mine falls.
	Acknowledgment of Taku River Tlingit First Nation's establishment of an Indigenous Protected and Conserved Area in the area of the New Polaris Gold Mine.
	Concerns about the issuances of tenures by the Province.
	Concerns about timing of the Environmental Assessment and the need to ensure Taku River Tlingit First Nation has the time needed to evaluate the effects of the New Polaris Gold Mine on its citizens and interests.
Biophysical effects of the mine	Concerns about effects to water, air, ecosystems, fish, and wildlife from the mine site.
	Concerns about physical works, including water diversions, barging of supplies, barge landing, and site roads, and the need to minimize effects as much as possible.
	Need for more information about how barging will be managed to minimize risks.
	Concerns about how fuel or hazardous materials will be safely stored and transported, as well as the potential for other accidents and malfunctions.
Effects to rights and cultural practices	Concerns about potential effects to fishing from barging operations.
Mine closure	Concerns about effects from mine closure on Taku River Tlingit First Nation citizens.
	Concerns about reclamation and how the mine site will be closed.

Source: BC EAO,2023.

20.4.4 Archaeological Assessment

An Archaeological Overview Assessment (AOA) Report was prepared in 2021 which identified archaeological potential and recommended an Archaeological Impact Assessment (AIA). Due to the proximity of the Tulsequah River, the Taku River, and multiple drainages, areas within the New Polaris Project area (Project Area) were assessed as having archaeological potential (Ecofor, 2021). Preliminary field reconnaissance was conducted in 2022, focused primarily on the areas that were identified during the AOA as having high archaeological potential several historic and modern mining structures and infrastructure were recorded. Three post-1846 culturally modified trees were recorded. The remainder of the area assessed was considered to have low archaeological potential due to saturated level terrain or

sloping terrain. Only a portion of the study area was assessed (Ecofor, 2022). The AIA survey focused primarily on eight study areas: Proposed right-of-way (ROW), Limestone Quarry, Airstrip Options 1 and 2, CSF Options 1 through 3, and CSF Access Roads. The historic and modern materials identified during survey are not protected under the Heritage Conservation Act (Ecofor, 2023).

20.4.5 Community Engagement

Canagold has held the New Polaris property since the early 1990s and previous engagement has occurred primarily with Taku River Tlingit First Nation (TRTFN) and potentially Affected Parties (i.e., government agencies and regulators, stakeholders, and the public). Engagement has been taking place in scale relative to the level of exploration and clean up activity at the site. From 2006 to 2021 little activity took place at the site therefore engagement with Affected Parties was also limited. With the resumption of exploration drilling in 2021, engagement has become more frequent, focused, and directed. More recent engagement activities have included forming of a joint Technical Working Group for sharing of project updates, plans, permit applications, meetings, and on-site visits along with monitoring, and field studies (Canagold, 2023).

Information regarding the identified Indigenous Nations, a rationale for their inclusion, any associated protocols, and agreements with Canagold and any known government to government agreements, are outlined in Table 20-9.

Table 20-9: Indigenous Nations Rationale, Protocols, and Agreements (Canagold, 2023)

Indigenous Nations	Rationale	Agreements	Engagement Protocols
Taku River Tlingit First Nation (TRTFN)	The Project Area is located within TRTFN Traditional Territory. Potential for sites of archaeological and heritage importance within the Project Area Project Area may overlap with current and Traditional Land Use areas, activities and otherwise inform the collection of Indigenous Knowledge	**TRTFN** • Hà khustìyxh / Our Way Agreement – 2003 • TRTFN Interim Agreement on Forest & Range Opportunities – 2007 • TRT Strategic Engagement Agreement (Whóoshtin yan too.aat Land and Resource Management and Shared Decision Making Agreement) – 2011 • Wóoshtin wudidaa Atlin Taku Land Use Plan – 2011 • TRTFN Clean Energy Revenue Sharing Agreement – 2013 • TRTFN Mining Policy – 2019 • TRT Forest Consultation and Revenue Sharing Agreement – 2022 • Hà khustìyxh / Our Way Agreement between TRTFN and Canagold – November 2022 • Capacity Funding Agreement related to TRTFN's involvement in the Environmental Assessment process – pending	• Directed to engage with designated Mining Liaison contact • Ongoing engagement with TRTFN designated departments i.e., Lands, Heritage, Environment (Fisheries and Wildlife, and Economic Development) on matters related to their role
Additional Indigenous Nations	To be determined during ongoing engagement and as per direction from BC EAO.		

Canagold will comply with TRTFN's relevant agreements and policies, including their Mining Policy and the Hà khustìyxh/Our Way Agreement that TRTFN recently signed with Canagold in November 2022. The key aspects of the TRTFN Mining Policy and the status of how Canagold is and will continue to comply with it are included in Table 20-10.

Table 20-10: TRTFN Mining Policy and Status

Policy and Section Number	Status
Agree to participate, complete, and execute a TRTFN Hà khustìyxh / Our Way Agreement (Section 6)	Signed November 2022
Provide prior notice of works (Section 5)	Ongoing
Share information regarding the proposed Mining Activity with TRTFN in accordance with Section 5	Ongoing
Meaningfully engage with TRTFN regarding the proposed Mining Activity in accordance with Section 5.2	Ongoing
Secure TRTFN consent in accordance with Section 4 including meeting Guiding Principles for TRTFN Decision Making (Section 3)	Ongoing

Source: Canagold, 2023.

A record of engagement since 2019 is available in the New Polaris Gold Mine Engagement Plan (Canagold, 2023); engagement and associated reporting are ongoing.

20.4.5.1.1 Summary of Engagement 2025

On November 26, 2024, from 5:00 PM to 7:00 PM, the EAO hosted a hybrid in-person and online community engagement session for the TRT community. The engagement session was attended by representatives from the EAO, the TRTTWG, and Canagold.

Questions and concerns relayed by TRT community members included:

- Potential for cumulative effects between New Polaris in combination with past mining operations in the area, namely Tulsequah Chief Mine.
- Concerns about whether Canagold would be held liable for remediation in the event of any major accident or malfunction resulting in severe environmental impacts.
- Feasibility of the proposed approach to transport equipment and supplies up the Taku River during the construction phase.
- Impact of New Polaris and the proposed airstrip on fisheries and wildlife in Flannigan Slough.
- Potential for New Polaris to bring employment and economic benefit to TRT.
- Protection of the land for future generations.

Additional comments from the TRT, TTC, and U.S. Tribes are included in EAO's 2025 Summary of Engagement and include requirements to assess transboundary impacts on Alaska Tribes, and to assess how historic tailings and past mining operations are currently affecting water quality, among others (EAO, 2025).

20.5 Mitigation and Management Plans

Table 20-11 summarizes the potential mitigation and management plans that Canagold anticipates will be required as part of the Project.

Table 20-11: Anticipated Mitigation and Management Plans

• Air Quality and Dust Management Plan • Mine Emergency Response Plan • Barge Emergency Response Plan • Avalanche Hazard Management Plan • Erosion and Sediment Control Plan • Dust Control Plan • Soil Management Plan • Vegetation Management Plan • Invasive Plant Management Plan • Wildlife Management Plan • Mine Site Traffic Control Plan • Hazardous Materials Management Plan • Reclamation and Closure Plan	• Spill Prevention and Contingency Plan for the Minesite and Barge • Construction Environmental Management Plan • Metal Leaching/Acid Rock Drainage Management Plan • Water Management Plan (including mine site water management and discharge management) • Aquatic Effects Monitoring Plan (updates to current protocols) • Archaeological and Paleontological Management Plan • Community Effects Monitoring and Management Plan • Occupational Health and Safety Management Plan • Chemicals and Materials Storage, Transfer and Handling Plan • Waste (Refuse and Emissions) Management Plan

During the course of regulatory, community and TRTFN engagement, additional management plans may be identified for development.

20.6 Closure and Reclamation Planning

20.6.1 Closure and Reclamation Plans

Under the BC Mines Act, anyone who engages in mining exploration work or mining operations determined by regulation must submit a reclamation plan. A conceptual reclamation and closure plan and a closure security estimate will need to be developed to support the submission of an EA report to the province or to the federal agency. The reclamation security will need to be posted to BC government prior to the commencement of construction the construction phase. The anticipated costs for closure and reclamation are provided in Section 21.2.7.

The current Conceptual Closure and Reclamation Plan for the project includes the following measures:

- Interim reclamation following construction phase to focus on cleared areas around the mine site (plant site, camp), road shoulders, and salvaged topsoil stockpiles.

- To reduce end of mine reclamation work, progressive reclamation measures will be implemented during the operating life of the mine in all disturbed areas that are no longer being used for ongoing operations.
- Airstrip, barge landing site, Micro-hydro project/road/transmission line will be decommissioned unless TRT requests they remain for their future use.
- All surface openings to the closed mine will be sealed.
- Mine site facilities (camp, plant site/equipment) to be decommissioned, dismantled and removed from site.
- Concrete slab supporting the plant site to remain in place and will be covered with a cover of alluvial gravels and stockpiled topsoil then revegetated.
- Mine site roads to be deactivated to access but not regraded.
- Water diversion channel around CSF to remain intact to retain the pre-existing surface channel connectivity leading downstream through Sawmill Creek.
- CSF to have 50 cm of alluvial sand and gravel (NAG) capping on final CSF contoured surface and have topsoil removed and stockpiled during construction placed on top to provide a growth medium.
- Active revegetation efforts to focus on large, disturbed areas like the mine site area and CSF. Linear features such as roads and transmission lines as well as areas along margins of undisturbed areas may be left to revegetation by natural ingress.
- Canagold to engage with TRT to establish on-site nursery and seed collection program.
- Restoration of the CSF and mine site area to take reclamation approach using native pioneer species working with primary successional pathways (favouring nitrogen fixing species such as alders, willow, legumes, grasses).
- Revegetation efforts will be a combination of broadcast seeding of grasses/legume seed mix and plug/whip planting of alder, willow.
- Reclamation progress will be monitored for trajectory to end land use objective (wildlife habitat/wildland). If there are issues with reclamation progression that may require intervention to achieve end land use objective, options will be evaluated through the reclamation monitoring program.
- Implementation of erosion and sediment control plan throughout construction, operations, closure and post-closure. Erosion mitigations and repairs to be implemented as/if project-induced erosional features develop.

21 CAPITAL AND OPERATING COSTS

21.1 Introduction

The capital and operating cost estimates presented in this FS provide substantiated costs that can be used to assess the economics of the New Polaris Project. The estimates are based on an underground mining operation; the construction of a process plant; associated waste rock and tailings CSF; off-site infrastructure including access roads, airstrip and barge landing facility; 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).

21.2 Capital Costs

21.2.1 Overview

The capital cost summary is presented in Table 21-1. The total initial capital cost for the New Polaris Project is C$250.4M, the LOM sustaining costs are C$225.0M. Closure costs are estimated at C$20.5M.

Table 21-1: Capital Cost Estimate

Description	Initial Capital Cost (C$M)	Sustaining Cost (C$M)	Closure Cost (C$M)	Total Cost (C$M)
Mining	63.3	196.1	-	259.4
Process Plant	43.0	-	-	43.0
Tailings Facilities	7.4	4.7	-	12.1
On-Site Infrastructure	38.5	-	-	38.5
Off-Site Infrastructure	9.4	-	-	9.4
Total Direct Costs	**161.6**	**200.8**	**-**	**362.4**
Project Indirects	42.3	-	-	42.3
EPCM Services	9.8	-	-	9.8
Owners Costs	7.8	-	-	7.8
Contingency	28.8	24.2	-	53.0
Total Indirect Costs	**88.8**	**24.2**	**-**	**113.0**
Project Total	**250.4**	**225.0**	**20.5**	**495.9**

Note: Values may not sum due to rounding.

21.2.2 Basis of Estimate

The capital cost estimate was developed in Q1 2025 Canadian dollars (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, processing, on-site infrastructure, tailings and waste rock facilities, 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 escalation added to the estimate.
- A growth allowance was included.

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

- mine schedules;
- FS-level engineering design by Ausenco, JDS, Tanka;
- topographical information obtained from the site survey;
- geotechnical investigations;
- budgetary quotes for process equipment and buildings from suppliers based primarily in China;
- budgetary quotes for surface mobile equipment and mine equipment from suppliers based in North America;
- budgetary unit costs from several 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.

Table 21-2: Exchange Rates

Currency Code	Currency Name	Exchange Rate
USD	United States dollar	1.00 CAD = 0.725 USD

21.2.3 Mining Direct Costs

21.2.3.1 Mining Basis of Estimate

21.2.3.1.1 Sources of Estimated Costs

The New Polaris mine site cost estimate is based on the construction of a new mine at a brownfield site having a nominal daily ore feed of 1,000 t/d. The capital and operating costs estimates for each discipline are developed by the following parties as shown in Table 21-3.

Table 21-3: Responsibility Matrix

Estimates	Responsible
Mine	JDS
Processing	Ausenco
Surface infrastructure	Ausenco
General & Administrative	Ausenco
Closure	Ausenco

The capital and operating cost estimates for the underground mine have been developed using engineering quantities (e.g. first principles estimates), third-party specialty data (e.g. salary surveys) and budgetary pricing obtained from vendors and contractors. Where required data was not available, estimates derived using in-house estimating procedures and database information have been provided.

21.2.3.1.2 Estimate Base Date and Validity Period Exchange Rate

The mining estimate was prepared with a base date of Q2 2025. All operating and capital costs were prepared in Canadian dollars (C$). Any conversion from other currencies has been done with the exchange rates as shown in Table 21-2.

21.2.3.1.3 Major Consumables

The major consumables and their respective costs used in the capital and operating cost estimate for the underground mine are shown in Table 21-4. Power cost is calculated based on the diesel-powered gensets using a rate of 3.7 kWh/L, with the inclusion of C$0.015/kWh for maintenance for parts and labour.

Table 21-4: Consumables Pricing

Consumable	Unit	Value
Diesel	C$/L	1.40
Power	C$/kWh	0.43

21.2.3.2 Mining Capital Costs

21.2.3.2.1 Contingency

Contingency has been applied to the capital cost estimate with input gathered from all parties involved. Each of the estimates has a contingency added, as shown in Table 21-5.

Table 21-5: Contingency Rates by Area

Area	Contingency (%)
Mining	15% (initial) 12% (sustaining)

21.2.3.2.2 Assumptions

The estimation is based on the project obtaining all relevant permits, as well as the following material assumptions.

- Underground mine:
  - Mined waste rock is considered suitable for stope backfill.
  - Skilled personnel and contractors are assumed to be readily available to the Project.
  - All surface works are performed by a contractor during construction and Canagold during operations.
  - Initial underground development and raises are mined by a contractor.
  - Other than for personnel (light) vehicles, mining contractor supplies all initial mobile equipment. At the end of the contract mining term, Canagold would purchase the used mobile equipment.
  - The underground geotechnical setting does not materially vary from that described in Section 16.2.
  - 100% of all mine operating costs are capitalized for the pre-production period.

21.2.3.2.3 Exclusions

The following items were not included in the mining capital cost estimate:

- Underground mining:
  - Provision for inflation, escalation, currency fluctuations and interests.
  - Operating costs other than those in Section 21.3.
  - Project financing costs.
  - Schedule delays.

- Mill commissioning delays.
- Delays receiving materials and consumables on site.
- Allowances for geotechnical drilling, analysis, and potential remedial works in relation to the historical underground voids present on the New Polaris site under planned infrastructure.
- Any site investigation and preparation work required for the emulsion mixing plant.
- Delays in recruiting personnel such that development, construction, or production is impacted.
- Force majeure.
- Work stoppages.
- Sunk costs.
- Mine mobile and fixed equipment salvage value at the end of mine life.

- The following items were included elsewhere in the capital cost estimate for the project:
  - Barging costs for all equipment and consumables.
  - Barging freight container logistics and related costs including the purchasing of containers.
  - Air freight shipping of bulk explosives and any other items to site.
  - Ore control assay laboratory costs.
  - Costs for transportatin of personnel to and from the New Polaris site and lodging while on site.

21.2.3.2.4 Mining Capital Cost Summary

The capital cost estimate for this project presented herein is considered to have an accuracy level of ±15%. The estimated LOM capital costs are presented in Table 21-5. Details of projected annual capital expenditure for the LOM are shown in Table 21-5.

Table 21-6: Life-of-Mine Mining Capital Costs Summary

Capital Costs	Initial (C$M)	Sustaining (C$M)	Total (C$M)
Mining and Contingency	64.7	219.6	284.3

Note: Numbers may not compute exactly due to rounding. *100% of operating costs are capitalized in the first 15 months.

21.2.3.2.5 Mining Capital Costs

The underground operation is planned to be operated by a contractor in the first two years inclusive of the pre-production period, which will then transfer operation to the Owner (Canagold). The start of underground development occurs in Year -2.

Initial mobile equipment capital costs cover only personnel (light) vehicles. All other initial mobile equipment is covered by the contractor's cost for all development, production, and backfilling. At the end of the contract mining term, the Owner (Canagold) would purchase the mobile equipment from the contractor.

Stationary equipment and infrastructure costs include all requirements for establishing the portal, refuge stations, ventilation, explosives storage, backfill system, mine water supply, dewatering system, underground workshop, electrical supply, and communications.

Owner's costs include contractor mobilization and demobilization and contractor indirects. Indirect costs include freight to get procured items to an ocean shipping port and critical spare parts for the mobile and stationary equipment and infrastructure. Table 21-7 shows the mining capital costs over the LOM. All initial underground capital costs are incurred in the first two years. Sustaining capital costs are from Year 1 and onwards.

Table 21-7: Life-of-Mine Annual Capital Costs

Description	Total (C$M)	Y –2	Y -1	Y 1	Y 2	Y 3	Y 4	Y 5	Y 6	Y 7	Y 8	Y 9
Surface Infrastructure	**14.4**	2.2	10.6	0.3	0.3	0.1	0.1	0.2	0.2	0.2	0.1	-
Capital Development	**161.1**	3.0	31.5	38.5	20.5	12.5	13.7	14.2	10.8	11.5	4.8	0.0
Underground Infrastructure	**0.7**	-	0.3	0.2	0.2	-	-	-	-	-	-	-
Water Management	**5.1**	0.1	0.5	2.4	0.7	0.2	0.1		0.5	0.5	-	-
Ventilation	**4.3**	0.3	1.7	1.6	0.3	0.1	0.1	0.0	0.1	-	-	-
Mobile Equipment	**49.5**	2.0	0.0	0.6	28.7	2.5	3.4	4.5	4.3	3.5	-	-
Electrical and Automation	**11.6**	0	2.5	3.4	2.4	1.9	-	-	1.4	-	-	-
Technical and Safety	**5.7**	1.1	0.4	0.5	0.5	0.5	0.4	0.5	0.4	1.1	0.3	0.1
Contingency	**32.0**	1.3	7.1	5.7	6.4	2.1	2.2	2.3	2.1	2.0	0.6	0.0
Total	**284.3**	**10.1**	**54.7**	**53.3**	**60.0**	**19.8**	**20.1**	**21.8**	**19.9**	**18.8**	**5.8**	**0.1**

Note: Numbers may not compute exactly due to rounding.

A subset of the total mining capital cost are the Capital development costs include ramps, infrastructure lateral development, raises, and service holes. All development includes operator and maintenance labour. A breakdown of the development costs is shown in Table 21-8.

Table 21-8: Total Mining Capital Development Costs

Description	Total (C$M)
Development Rehabilitation	0.5
Permanent Waste Drives	62.5
Vent Raises	4.6
Escape Way	7.8
Material Handling	26.9
Mine Maintenance	7.3
Mine General	51.6
Sub-Total	**161.1**
Contingency (12.6%)	20.4
Total	**181.4**

Note: Numbers may not compute exactly due to rounding.

Table 21-9 is a summary of total LOM mining capital costs.

Table 21-9: Life-of-Mine Mining Capital Costs Summary

Description	Initial (C$M)	Sustaining (C$M)	Total (C$M)
Surface Infrastructure	12.9	1.5	14.4
Underground Development	34.5	126.6	161.1
Underground Infrastructure	0.3	0.4	0.7
Water Management	0.6	4.5	5.1
Ventilation	2.0	2.3	4.3
Mobile Equipment	2.0	47.5	49.5
Electrical and Automation	2.5	9.1	11.6
Technical and Safety	1.4	4.3	5.7
Subtotal	**56.3**	**196.1**	**252.4**
Contingencies	8.4	23.5	32.0
Total	**64.7**	**219.6**	**284.3**

Note: Numbers may not compute exactly due to rounding.

21.2.4 Process Plant, Co-Storage Facility and On-Site Infrastructure Direct Costs

A summary of the Process Plant, Tailings Facility and On-Site Infrastructure capital cost estimate is presented in Table 21-10.

Table 21-10: Capital Cost Estimate Summary – Process Plant, Tailing Management Facility & On-Site Infrastructure

WBS Description	Initial Capital Cost (C$M) Phase 1	Sustaining Capital Cost (C$M) LOM	Total Cost (C$M)
Crushing, Stockpile/ Reclaim	9.2	-	9.2
Grinding	16.5	-	16.5
Flotation	3.1	-	3.1
Tailings Thickening and Handling	3.7	-	3.7
Concentrate Thickening	7.7	-	7.7
Reagents & Services	1.5	-	1.5
Plant Control System	1.3	-	1.3
Tailings and Waste Rock Co-Disposal	7.4	4.7	12.1
Bulk Earthworks	2.2	-	2.2
Infrastructure Buildings	3.4	-	3.4
Site Services	21.8	-	21.8
Mobile Equipment	4.3	-	4.3
Power Generation & Distribution	6.6	-	6.6
On-site Demolition	0.1	-	0.1
Total	**88.9**	**4.7**	**93.5**

Note: Values may not sum due to rounding.

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 21-11 and Table 21-12). For mechanical equipment costs, 91% of the value was sourced from budgetary quotes; the remainder was sourced either by factoring or by benchmarking against other recent North American mining projects and studies.

Pricing for the process plant equipment was from a single source budgetary quotation from an overseas vendor.

Table 21-11: Mechanical Equipment Price Basis

Source	Total Supply (C$M)	%
Budgetary Quotes	16.6	91
Estimated (database)	0.5	3
Factored	1.2	6
Total	**18.3**	**100**

*Note: costs exclude freight and growth.

Table 21-12: Mechanical Equipment & Packages

Package No.	Equipment
P0001	Modular Crushing Plant
P0002	Grinding Mills
P0004	Flotation Cells
P0005	Conveyors And Belt Feeders
P0006	Cyclones
P0007	Thickeners
P0008-B	Tailings & Concentrate Filters
P0009	Potable Water Treatment Plant
P0010	Effluent Water Treatment Plant
P0013	Sewage Water Treatment Plant
P0014	Vibrating Screens
P0017	Heat Recovery/HVAC
P0018A	Slurry Pumps
P0018B	Water Pumps
P0019	Reagents
P0020	Incinerator
P0021	Overhead Cranes
P0022	Agitators
P0023	Weigh Scale
P0024	Dosing Pumps
P0026	Plant Air Compressors, Air Receivers, Air Blowers
P0027	Air Filters & Air Dryers
P0028	Concentrate Filtration Dryer
P0029	Immersion Heater
P0031	Rock Breaker
P0032	Miscellaneous Pumps
P0033	Samplers
PS-104	Diesel Storage and Dispensing

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 21-13 and Table 21-14). For the electrical equipment, 88% of the value was sourced from budgetary quotations. The remainder was factored.

Table 21-13: Electrical Equipment Supply Price Basis

Source	Total Supply (C$M)	%
Budgetary Quotes	8.4	88
Factored	1.2	12
Total	**9.6**	**100**

*Note: costs exclude freight and growth.

Table 21-14: Electrical Equipment & Packages

Package No.	Equipment
P0201	Power Transformers - Supply
P0202	MCCs and VFDs - Supply
P0203	Diesel Generators (3.6MW)
P0204	MV Switchgear – included with gensets
P0205	Diesel Generators (600V) - standby
P0207	E-Room Package Integration

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, platework, piping, electrical and instrumentation bulks) to be derived from major commodities, as outlined in Table 21-15.

Table 21-15: Major Disciplines

Discipline	WBS Description
A	Architectural
B	Earthworks
C	Concrete
D	Demolition
E	Electrical equipment
F	Platework
I	Instrumentation
L	Electrical bulks
M	Mechanical equipment
N	Mobile equipment & ancillaries
O	Owner’s costs
P	Piping
R	Third-party estimates
S	Structural steelwork
T	Project delivery
U	Field indirects
V	Other (spares, fills, vendors)
Y	Contingency

After the derivation of all the bulk material quantities (earthworks, concrete, steel, piping, cables, etc.), for the process plant, tailings management facility and surface infrastructure areas, major construction contracts were formed and tendered for budgetary pricing bids, as per Table 21-16.

Table 21-16: Construction Contract Packages

Package No.	Equipment
C501	Earthworks
C502	Concrete & Batch Plant – Supply and Install
C503	SMP Installation
P0501	Structural Steel Supply
P0502	Platework Supply
P0503	Piping Supply
C505	Electrical and Instrumentation – Installation of electric equipment, supply & install of electric cabling
C506	Pre-Engineered Buildings – Supply and Install
C507	Modular Buildings – Supply and Install
C508	Fabric Buildings – Supply and Install

Budgetary supply costing for structural steel, platework, piping, process mechanical equipment, power plant, pre-engineered buildings and modular buildings was provided by overseas vendors. Canadian installation rates were used for the estimate.

Specific to the CSF, the design and material take-off outlined in Section 18 was applied against the contractor installation rates quotes for typical activities such as clearing, topsoil removal, excavation, backfilling with granular material and mine waste. The costs associated with this estimate also includes liner placement, drains, ground improvement, and associated supervision, monitoring, and indirect costs.

21.2.5 Off-Site Infrastructure Direct Costs

A summary of the off-site infrastructure cost estimate is presented in Table 21-17. The haul road from the process plant to CSF and access road to the explosives facility is covered under mine infrastructure capital (Section 1.1.1).

Table 21-17: Off-Site Infrastructure Capital Cost Estimate Summary

WBS Description	Initial Capital Cost (C$M)	Sustaining Capital Cost (C$M) LOM	Total Cost (C$M)
Main Access Road/Barge Landing	3.3	-	3.3
Airstrip	6.1	-	6.1
Total	**9.4**	**-**	**9.4**

21.2.5.1 Main Access Road/Barge Landing Facility

The estimate allows for diversions and upgrades to the existing site access road in addition to new routes from the new airstrip and barge landing location, as well as the construction of the barge landing facility. The access road will be upgraded including installation of culverts as well as grading of corners to ensure suitability for daily operational traffic.

Material take-offs for the road and barge landing facility were completed based on a Civil3D model design and was applied against the contractor installation rates quotes for typical activities such as clearing, topsoil removal, excavation, riprap installation, and backfilling with granular material.

21.2.5.2 Airstrip & Associated Buildings

The estimate allows for the earthworks to prepare the site and runway in addition to the supply and installation of an airstrip warehouse (fabric) and office (modular). A material take-off was completed based on a Civil3D model of the airstrip 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 Indirect Costs

Indirect costs include all costs that are necessary for project completion but not related to the direct construction cost, including project indirect costs, EPCM Services, Owner's costs, and contingency, as outlined below.

Project indirect costs include the following:

- temporary construction facilities and construction services
- construction camp, accommodation and messing costs
- vendor representative costs during commissioning and construction
- spares
- first fills and initial charges
- contractor commissioning assistance.

Owner's costs include the following:

- Owner's execution team
- Operational readiness
- Indirect costs are summarized in Table 21-18 and are described in the following sections.

Table 21-18: Indirect Costs

Description	Initial Capital Cost (C$M)	Sustaining Capital Cost (C$M) LOM	Total Capital Cost (C$M)
Project Indirects	42.3	-	42.3
EPCM Services	9.8	-	9.8
Owner's Cost	7.8	-	7.8
Contingency	28.8	24.2	53.0
Total Indirect Capital Cost	**88.8**	**24.2**	**113.0**

Note: Values may not sum due to rounding.

21.2.6.1 Project Indirect Costs

21.2.6.1.1 Temporary Construction Facilities and Construction Services.

The temporary construction facilities and construction services for the construction management team have been developed from similar historical projects and includes the following items:

- Personnel transportation to site (flights from Juneau/ Whitehorse to site).
- General construction services such as communications, snow removal, road maintenance, temp power.
- QA/QC services.
- Aggregate mobile crushing and screening plant.

21.2.6.1.2 Accommodation and Messing

The costs for establishing the camp facility, messing and operations are included in the cost estimate and is considered as free issued by Canagold.

The Project requires a 165-bed accommodation facility to support construction, and this facility will be maintained for operations personnel through the life-of-mine. Budgetary pricing for the camp was received from an overseas vendor and used in this estimate.

21.2.6.1.3 Vendor Representatives

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 flotation concentrator mining projects.

The total vendor representatives and assistance cost is estimated to be C$1M.

21.2.6.1.4 Spares

Commissioning spares 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. Allowance factors were based on a six-month period of capital spares.

The total spares cost is estimated to be C$1.9M.

21.2.6.1.5 First Fills

First fills include the costs for the initial construction first fills for installed equipment and commissioning first fills which consist of chemicals, fuels, and lubricants etc. and is an allowance based on historical data. Commissioning First Fills costs were developed from the Opex costs.

The total first fills cost is estimated to be to be C$0.4M.

21.2.6.1.6 Freight Costs – Barging

Freight costs for barging items, materials and equipment to site were calculated using an estimate of total tonnage, applied to a $/t unit rate given by a logistics subcontractor (AML). This rate was for barging from the Port of Seattle to the New Polaris site (via Juneau).

The total tonnage estimate for the Project was developed based on:

- Contractor estimates for materials and equipment
- Vendor information on mechanical/ electrical equipment supply
- Bulk material design quantities
- Contractor estimates on fuel usage during construction

The total barging cost is estimated to be C$13.3M.

21.2.6.1.7 Fuel Costs

Total fuel costs for the project during the construction period were estimated using contractor fuel projections given in their quotations.

The total fuel cost is estimated to be C$7.2M.

21.2.6.2 EPCM Services

EPCM service 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 Ausenco EPCM 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 is C$9.8M.

21.2.6.3 Owner's Costs (Operational Readiness)

Owner's costs of $7.8M have been provided by Canagold.

21.2.6.4 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 includes a provision to cover the risk from these uncertainties.

For Ausenco scope, a contingency rate of 10% to 15% has been used based on a Class 3 AACE estimate and the level of definition of the project scope. To develop the contingency value, 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. No contingency has been included for "project specific risks" such as items noted in Section 21.2.8, or management reserve. The @Risk simulation completed for Ausenco's scope, determined the contingency to be 14.5% of base accumulative costs at P_{50} confidence level.

The contingency percentages applied for JDS and Canagold were provided by each party and confirmed with Canagold.

The following contingency percentages were applied:

- Process Plant, Site Infrastructure, Tailings Management – 14.5% (P_{50} confidence level based @Risk simulation)
- Mining – 15%
- Owner – 15%

A summary of the contingency cost is Table 21-19.

Table 21-19: Estimate Contingency

Description	Initial Capital Cost (C$M)	Sustaining Cost (C$M)	Total Cost (C$M)
Process Plant/ Tailings/ Infrastructure	19.2	0.7	19.9
Mining	8.4	23.5	31.9
Owner's	1.2	-	1.2
Total Contingency	**28.8**	**24.2**	**53.0**

Note: Values may not sum due to rounding.

21.2.7 Closure Costs

An allowance of C$20.5M has been made for closure costs. See Section 20.6.1 for details on the conceptual closure and reclamation plan.

21.2.8 Growth Allowance

A growth allowance has 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;
- based on the accuracy of quantity take-offs and engineering lists based on the level of engineering and design undertaken at Feasibility Study level; and
- inclusive of pricing growth for the likely cost increase due to specification development and refinement, 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 21-20.

Table 21-20: Growth Cost Summary

Description	Growth (Average %)	Initial Growth Cost (C$M)	Sustaining Growth Cost (C$M)	Total Growth Cost (C$M)
Earthworks	7	1.5	0.3	1.8
Concrete	7	1.1	-	1.1

Description	Growth (Average %)	Initial Growth Cost (C$M)	Sustaining Growth Cost (C$M)	Total Growth Cost (C$M)
Structural Steel	7	0.2	-	0.2
Platework	7	0.3	-	0.3
Mechanical equipment	6	1.1	-	1.1
Piping	7	0.8	-	0.8
Electrical Equipment	6	0.6	-	0.6
Instrumentation	6	0.1	-	0.1
Electrical bulks	7	0.3	-	0.3
Architectural	6	0.3	-	0.3
Demolition & Relocation	7	0.0	-	0.0
Field Indirects	9	3.3	-	3.3
Other (Spares, Fills, Vendors)	3	0.1	-	0.1
Mobile Equipment & Ancillaries	6	0.2	-	0.2
Third-Party Estimates	7	0.3	-	0.3
Total Growth		**10.5**	**0.3**	**10.8**

Note: Values may not sum due to rounding.

21.2.9 Exclusions

The following costs and scope will be excluded from the capital cost estimate:

- Operating costs
- 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 Q1 2025
- Environmental impact assessment
- 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
- Financing costs.

21.2.10 Sustaining Capital

The total LOM sustaining costs for the New Polaris Project are C$225.0M. The values presented in Table 21-21 show total LOM sustaining costs.

Table 21-21: Total Sustaining Costs

Description	Sustaining Capital Cost (C$M)
Mining	196.1
Tailings Facilities (CSF)	4.7
Contingency	24.2
Project Total	**225.0**

Note: Values may not sum due to rounding.

21.3 Operating Costs

21.3.1 Overview

Operating costs include the ongoing costs of operations related to mining, processing, tailings disposal, and general administrative activities. Table 21-22 provides a summary of the life-of-mine average operating cost on a $/t milled basis, as well as the life-of-mine total operating cost.

Table 21-22: Operating Cost Summary

Cost Area	Total (C$M)	C$/t Milled	% of Total
Mining	383.3	135.45	50.6
Process	181.9	64.28	24.0
Barging	0.4	0.16	0.1
G&A	191.2	67.58	25.3
Total	**757.0**	**267.47**	**100.0**

Note: Closure reclamation bond (G&A) costs are included. Values may not sum due to rounding.

21.3.2 Basis of Estimate

Common to all operating cost estimates are the following assumptions:

- Cost estimates are based on Q1 2025 Canadian Dollars (CAD) pricing without allowances for inflation.
- Costs are expressed in Canadian Dollars (C$).

- For materials sourced in US Dollars, an exchange rate of US$0.725 to C$1.00 was used.
- For materials sourced in Euros (€), an exchange rate of €1.51 to C$1.00 was used.
- For materials sourced in Australian Dollars (AUD or A$), an exchange rate of A$0.91 to C$1.00 was used.
- Labour requirements are assumed to be satisfied by a mix of 30% local and 70% non-local personnel, but still within Canada.
- Processing unit operations were benchmarked against similar or comparable processing plants.
- Equipment and materials will be purchased new.
- Grinding media consumption rates have been estimated based on the measured ore characteristics.
- Reagent consumption rates have been estimated based on metallurgical testwork, testwork interpretations, and standard operating practices.
- Mobile equipment for processing, maintenance, and G&A include fuel, maintenance, spares, insurance (if applicable), and lease costs.
- All materials brought to site will be airlifted (C$300/t) from Juneau, AK, and in the case of oversized equipment or materials unsuitable for air transport, barged (C$500/t) from Juneau. All quoted consumables and reagents are priced to Juneau.

21.3.3 Mine Operating Costs

21.3.3.1 Summary

The estimated mining operating costs over the LOM are presented in Table 21-23.

Table 21-23: Life of Mine and Unit Operating Costs

Operating Costs	C$M	C$/t milled
Mining	383.3	135.4
Total	**383.3**	**135.4**

Note: Numbers may not compute exactly due to rounding.

Mining related items in Table 21-23 include labour costs, which were estimated based on the workforce necessary to operate and maintain the proposed mobile mining fleet and stationary equipment. Mining production rates and productivity, as well as equipment mechanical availability and utilization factors were considered in the operating cost estimate. Details of projected annual operating expenditures are shown in Table 21-24.

Table 21-24: Annual Mining Operating Costs

Description	Total	Y -1	Y 1	Y 2	Y 3	Y 4	Y 5	Y 6	Y 7	Y 8	Y 9
C$M											
Mining*	383.3	-	18.1	55.4	53.5	51.2	53.1	50.1	47.4	43.7	10.8
Total Mining Costs	**383.3**	**-**	**18.1**	**55.4**	**53.5**	**51.2**	**53.1**	**50.1**	**47.4**	**43.7**	**10.8**

Note: *Mining includes capitalized pre-production operating costs. Numbers may not compute exactly due to rounding.

21.3.3.2 Basis of Estimate

The mining operating cost estimate was based on Quarter 2 2025 assumptions and has an accuracy of ±15%. Cost estimates were based on the mine production schedule (see Section 16.8).

21.3.3.3 Contingency

No contingencies have been applied to the estimation of operating costs.

21.3.3.4 Assumptions

The following material assumptions were made in preparing the operating cost estimation:

- Underground mine:
  - The Owner (Canagold) will be the sole operator of the underground mine starting at the end of Year 1.
  - Mined waste rock is considered suitable for use as stope backfill.
  - Skilled personnel and contractors are assumed to be readily available for the Project.
  - The underground geotechnical setting does not materially vary from Section 16.2.

21.3.3.5 Exclusions

The following items were excluded from the mining operating cost estimate:

- Underground mining:
  - Delays receiving materials and consumables on site.
  - Delays in recruiting personnel such that development or production is impacted.
  - Force majeure.
  - Work stoppages.
  - Fluctuation of currency exchange rates.
  - Provision for inflation.

- The following items were included elsewhere in the mining operating cost estimate for the Project:
  - Air freight shipping of bulk explosives and any other items to site.
  - Ore control assay laboratory costs.
  - Costs for transportation of personnel to and from the New Polaris site and lodging while on site.

21.3.3.6 Underground Mine Costs

The start of underground development occurs in Year -2. Underground ore production will start in Year 1 and will last over the LOM until Year 9. The estimated LOM underground operating costs and the distribution of the unit cost items are shown in Table 21-25 and Table 21-26.

The mining operating costs include the following functional areas:

- **Temporary Waste Drives (Development)** – Costs related to the drilling, blasting, mucking, and hauling of waste development on the level.
- **Development Ore Drives** – Costs related to the drilling, blasting, mucking, and hauling of ore development related to longhole sills.
- **Longhole Stoping (LHS)** – Costs relating to the drilling, blasting, mucking, and hauling of longhole production ore (note: does not include sill development).
- **Drift and Fill Stoping** – Costs related to the drilling, blasting, mucking, and hauling of drift and fill stopes.
- **Backfill** – Costs related to backfill operations, including the paste plant, binder, fill barricades, and waste backfill.
- **Mine Maintenance** – Maintenance labour costs that support all other sectors.
- **Mine General** – Costs related to mine support activities, such as technical services, shared infrastructure (ventilation, dewatering, support equipment, and definition drilling.

Contractor quotations were used to estimate development and drift-and-fill production costs during periods when the contractor is on site. These contractor costs cover the contractor's labour and equipment, while all other costs are covered by the Owner (Canagold).

Table 21-25: Life of Mine Underground Operating Costs

Underground Operating Costs	C$M	C$/t Milled
Temporary Waste Drives	35.1	12.39
Development Ore Drives	17.2	6.09
Longhole Stoping (LHS)	33.8	11.94
Drift and Fill Stoping	98.4	34.76
Material Handling	59.1	20.88
Backfill	27.3	9.64

Underground Operating Costs	C$M	C$/t Milled
Mine Maintenance	21.6	7.62
Mine General	91.0	32.14
Total	**383.3**	**135.45**

Note: Numbers may not compute exactly due to rounding.

Table 21-26: Distribution of Underground Operating Costs (C$M)

Underground Operating Costs	Total	Labour	Equipment	Fuel	Materials	Power	Contractor Services
Temporary Waste Drives	35.1	20.8	2.0	0.9	11.3	0.0	0.0
Development Ore Drives	17.2	10.9	1.1	0.4	4.7	0.0	0.0
Longhole Stoping	33.8	13.9	2.1	0.7	17.1	0.0	0.0
Drift and Fill Stoping	98.4	57.9	6.6	3.3	30.6	0.0	0.0
Material Handling	59.1	42.5	5.9	8.4		0.0	2.3
Backfill	27.3	10.9	1.7	1.8	12.8	0.0	0.0
Mine Maintenance	21.6	20.9	0.2	0.5	0.0	0.0	0.0
Mine General	91.0	55.3	0.7	1.1	0.0	26.3	7.5
Total	**383.3**	**233.1**	**20.3**	**17.1**	**76.7**	**26.3**	**9.9**

Note: Numbers may not compute exactly due to rounding.

21.3.3.6.1 Underground Mining Labour

Underground mining staffing levels are built up based on the productivities (man-hours) required for mining activities occurring within a given time period. As such, mining manpower fluctuates throughout the mine life.

Mine labour (including supervision and support) related to development drifting is distributed between capital development (sustaining capital costs) and operating development (operating costs), based on the activities being performed within a given time period. As such, only a portion of the mine staffing is allocated within the mining operating costs.

21.3.3.6.2 Underground Equipment

Underground mining equipment usage costs are based on the equipment operating hours required to meet the life-of-mine plan. Equipment usage costs include unit costs for the following elements:

- Maintenance parts
- Tires

- Lubricants
- Boxes, buckets, and ground engaging tools.

Unit costs for the elements above have been obtained from equipment manufacturers databases. Equipment replacements and major rebuilds are included in the sustaining capital costs.

21.3.3.6.3 Underground Fuel Costs

Underground mining fuel consumption has been built up based on the required equipment operating hours dictated by the mine plan for development or production-based equipment, and annual allowances for support or fixed infrastructure equipment.

21.3.3.6.4 Materials and Consumables

Estimated from first principals, costs include:

- Ground Support
- Drilling consumables
- Explosives
- Ventilation consumables
- Services (Pipe, hangars, electrical cable, comms).

21.3.3.6.5 Power

All power costs have been included under Mine General. Power costs include usage estimates from mobile equipment, (jumbo drills, longhole drills, diamond drills, raise drills and bolters), main and auxiliary ventilation fans, dewatering equipment, maintenance facilities, compressors and refuge chambers.

21.3.3.6.6 Contractor Services

Additional fees incurred from use of the contractor. Including management costs, mobilization costs and demobilization costs.

21.3.4 Process Operating Costs

Unless stated otherwise, all costs presented in this section are in Canadian Dollars (C$) and are derived from first principles and benchmarking of Ausenco projects.

Table 21-27 summarizes the operating costs for the process plant on an annual basis over the life of mine. Preproduction costs are excluded from the operating cost estimate.

Table 21-27: Process Plant Opex Summary

Cost Area	LOM (C$M)	Y 1 (C$M)	Y 2 (C$M)	Y 3 (C$M)	Y 4 (C$M)	Y 5 (C$M)	Y 6 (C$M)	Y 7 (C$M)	Y 8 (C$M)	Y 9 (C$M)
Power	64.00	4.94	8.05	8.39	8.36	8.39	8.39	8.39	7.70	1.41
Reagents	11.31	0.95	1.46	1.46	1.46	1.46	1.46	1.46	1.34	0.25
Consumables	18.83	1.88	2.35	2.35	2.34	2.35	2.35	2.35	2.24	0.65
Labour	57.01	6.77	6.77	6.77	6.77	6.77	6.77	6.77	6.77	2.82
Maintenance	3.78	0.45	0.45	0.45	0.45	0.45	0.45	0.45	0.45	0.19
Mobile Equipment	16.58	1.97	1.97	1.97	1.97	1.97	1.97	1.97	1.97	0.82
Lab Services	1.73	0.21	0.21	0.21	0.21	0.21	0.21	0.21	0.21	0.09
Dry Stack	8.64	1.04	1.04	1.04	1.04	1.04	1.04	1.04	1.04	0.35
Barging	0.43	0.05	0.05	0.05	0.05	0.05	0.05	0.05	0.05	0.02
Total	**182.4**	**18.26**	**22.35**	**22.69**	**22.65**	**22.69**	**22.69**	**22.69**	**21.77**	**6.59**

*values may not sum due to rounding.

21.3.4.1 Labour

Staffing requirements were estimated by benchmarking the New Polaris Project against similar projects, both in terms of operations and locality. The labour costs incorporate staffing requirements for the plant operation, such as management, metallurgy, mill operations, mill maintenance, site services, and laboratory operations. Labour rates were estimated using a mixture of labour rates known from similar projects, as well as publicly available labour rates.

Table 21-28 summarizes the organizational staffing plans outlining the labour requirements for the process plant. Overall the process plant will always operate on a total headcount of 48 on site, and an annual total labour costs of C$6.8M.

Table 21-28: Labour Breakdown

Role	Department	Quantity
Chief Metallurgist (cross shift for Superintendent)	Process	1
Processing Superintendent	Process	1
Shift supervisor	Process	4
Crushing Operator	Process	4
Grinding Operator	Process	4
Control Room Operator	Process	4
Flotation Operator	Process	4
Flotation Helper	Process	2
Thickening/Filtration Operator	Process	4
Handling Operator	Process	2
Reagent Helper	Process	2
Chief Assayer	Laboratory	1
Lab Technician	Laboratory	4
Maintenance Manager	Maintenance	1
Millwright / Welder / Mechanic	Maintenance	4
Electrician	Maintenance	2
Electrical Helper	Maintenance	2
Inst. & Control Technician	Maintenance	2
Total	-	48

21.3.4.2 Electrical Power

The power costs of the process plant and ancillary facilities were calculated using the average power utilization based on individual equipment demand in the electrical load list. All site power will be generated by diesel burning gensets. Diesel will be supplied to the site by air.

Diesel price was based on an OPIS PNW spot weekly average metric in Q2 2025. Including air freight, a diesel price of C$1.40/L was used for the development of the power cost summary. A diesel energy density value of 0.231 L/kWh, was used to determine diesel consumption for electrical power.

The first two years of operations will not require the full design capacity of the mill and are considered a ramp-up period where the power draw is lower than the years after. Years 1 and 2 will have a power demand of 22.3 GWh and 23.6 GWh, respectively. Starting Year 3 the annual power demand will be 24.6 GWh. Most of the power savings is attributed to a lower power draw on the ball mill due to the gradual ramp-up of production. In terms of diesel consumption, Year 1 and 2 will require 5.2 and 5.5 million liters of diesel, respectively, and Year 3 onwards will require 5.7 million liters of diesel.

21.3.4.3 Reagents, Wear Items, and Grinding Media

Consumption rates of reagents are derived from the testwork summarized in Section 13. All quoted reagents were priced for delivery to the point where freight will be air lifted to the New Polaris property. Non-quoted consumables, including transportation to the air lift site, were priced using the in-house pricing database.

Mill media was determined to have a nominal consumption rate of 0.8 kg/t. The consumption rate was derived from the modified Bond formula and benchmarked against Ausenco's internal database using site operational data to ensure accuracy.

Crushing and grinding consumables requirements, such as liners and spares, were estimated based on benchmarks for replacement rates for each crusher or mill type. Maintenance consumable unit pricing was obtained from vendor quotations or Ausenco's internal database of benchmark costs.

Annual reagent costs are dependent on the mine plan, and range between C$1.34M to C$1.46M during the years of full ramped up operations (Y3-Y8). The final year (Y9) only consists of four operational months is not included in this range.

Similarly, consumable costs range between C$2.24M to C$2.35M, annually.

21.3.4.4 Maintenance Parts and Supplies

The process plant annual maintenance is derived from applying a small percentage factor of the mechanical equipment supply cost. The maintenance factor applied varies between 3-5% depending on the specific process task associated with the equipment, with an overall average of 3.6%.

The annual maintenance cost totals to C$449,600.

21.3.4.5 Mobile Equipment

Vehicle costs are based on a scheduled number of light and mobile equipment. Fuel, maintenance, spares, insurance (for off-site vehicles), and leasing costs are included. Mobile equipment requirements for plant operations and maintenance result in an annual cost of C$1.97M.

21.3.4.6 Dry Stacking

Filtered tailings from the mill will be disposed by method of dry stacking. The estimated operating costs associated with dry stacking includes the fuel and maintenance required to operating a compactor, dozer, and articulated haul truck, as well as cost of labour. The annual cost of dry stacking will be C$1.0M.

21.3.4.7 Barging

Oversized items and other non-air transportable items will be barged to site. An annual allowance of C$52,000 has been allocated.

21.3.4.8 On-site Laboratory Services

The operating cost estimate for the laboratory was developed from a scope that includes an estimate of the number of samples and assay requirements for use, as well as sampling for site environmental analysis such as raw and process water quality.

Annual laboratory services cost is C$206,000.

21.3.5 General and Administrative Operating Costs

General and administrative (G&A) operating costs are costs and expenses that are not directly related to production and include expenses not included in mining, processing, external refining, and transportation. All G&A costs were developed using Ausenco provided in-house data based on relevant and similar projects. The G&A costs are divided into the following areas:

- G&A related maintenance
- Labour
- Recruiting, training, and community relations and developments
- Administrative costs – materials
- Vehicles
- Health and safety – materials and services
- Environmental – monitoring, sampling, and damage mitigation
- IT & Telecommunications
- Legal – Contract services, insurance, consultation, licensing, and fees
- General Fees and taxes
- Power – G&A facilities and mobile equipment
- Travel & Camp for G&A personnel

G&A personnel (summarized in Table 21-28) will consist of a locally sourced labour force and will be stationed on site and follow the typical 14/14 rotation that applies to mill and mine operation personnel. The annual G&A cost will be C$21.3M as summarized in Table 21-29. The reclamation bond costs are not included in the G&A operating cost estimate.

Table 21-29: G&A Cost Summary

Cost Area	C$M/a
G&A Maintenance	0.52
Personnel	2.84
Human Resources	1.30
Site Admin/Maintenance	0.13
Vehicles	0.44
Health & Safety	0.24
Environmental	0.39
IT & Telecommunications	0.88
Contract Services, Insurance, Legal	3.98
Administrative Costs	0.56
Power	3.87
Travel & Camp	7.27
Closure Reclamation Bond	0.29
Total	**22.71**

Table 21-30: G&A Labour

Role	Department	Quantity
General Manager	G&A	1
Executive Assistant	G&A	1
OH & S Manager	G&A	1
HR Manager	G&A	1
Professional Associate for Procurement	G&A	1
HR Administration Officer	G&A	1
Environmental Coordinator - Mine	G&A	1
Recruitment Officer	G&A	1
Trainer Coordinator	G&A	1
Associate for Procurement	G&A	1
Accountant	G&A	1
OH & S Administrator	G&A	1
ICT Associate	G&A	1
Security	G&A	4
Environmental Technician	G&A	2
Janitor	G&A	2
Bus Driver	G&A	2
Env Field Assistants	G&A	2

Total	**-**	**25**

21.3.5.1 Travel and Camp

Travel and camp operating costs were developed using a benchmarked C$84/d cost for all site personnel that would cover all camp amenities, food, and lodging. The total expected headcount for mine, mill, contractors, non-employee visitor is 161. The camp will have a capacity of 165 beds. Camp service providers are excluded from the headcount but included are accounted for in the unit cost rate for mine employees.

Travel to and from camp will be by air. Site buses will take employees from camp to the air strip on the New Polaris site. Using Ausenco's internal database of flight travel costs, regional travel costs were benchmarked to be C$220 per person, while non-regional travel would be C$870 per person. Regional travel would be between the New Polaris site and Whitehorse, Yukon or Juneau, Alaska. Expected ratio of regional to non-regional travellers is 7:3.

Total camp costs are C$4.5M and all regional and non-regional travel will incur an annual operating cost of C$2.8M.

22 ECONOMIC ANALYSIS

22.1 Forward-Looking Information Cautionary Statements

The results of the economic analyses discussed in this section represent forward-looking information as 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. Forward-looking information includes:

- Mineral resource and reserve estimates.
- Assumed gold prices and exchange rates.
- The proposed mine production plan.
- Projected mining and process recovery rates.
- Assumptions as to mining dilution and ability to mine using underground mining methods as envisaged.
- Sustaining costs and proposed operating costs.
- Assumptions as to closure costs and requirements.
- Assumptions as to environmental, permitting, and social risks.

Additional risks to the forward-looking information include:

- Changes to costs of production from what are estimated.
- Unrecognized environmental risks.
- Unanticipated reclamation expenses.
- Unexpected variations in the quantity of mineralized material, grade, or recovery rates.
- 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 in 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.
- Accidents, labour disputes, and other mining industry risks.
- Changes to interest rates.

- Changes to tax rates.

This model assumes that permits must be obtained in support of operations, and approval for development to be provided by Canagold's Board.

22.2 Methodologies Used

An engineering economic model was developed to estimate annual pre-tax and post-tax cash flows and sensitivities of the Project based on an 5% discount rate. It must be noted, however, that tax estimates involve many complex variables that can only be accurately calculated during operations and, as such, the post-tax results are only approximations. A sensitivity analysis was performed to assess the impact on metal price, mill head grade, mill recovery, initial capital costs, operating costs and exchange rates.

The capital and operating cost estimates were developed specifically for this Project (presented in 2025 Canadian dollars). The economic analysis was run with no inflation (constant dollar basis).

22.3 Financial Model Parameters

The economic analysis was performed using the following assumptions:

- Project revenue is derived from the sale of gold concentrate. Potential revenue from antimony has not been included in the financial analysis.
- All cash flows are discounted back to the start of project execution.
- Construction period of two years.
- Full production is achieved by Q2 of the first year of operation.
- All products are assumed to be sold in the same year they are produced.
- The metal prices are based on consensus prices and assuming constant price US$2,500/oz Au. The forecasts used are meant to reflect the average metal price expectation over the life of the Project. Commodity prices can be volatile, and there is the potential for deviation from the forecast.
- A spot case was run using a spot price of US$3,300/oz Au sourced on July 11th.
- A mine life of 8.3 years.
- No adjustments for price inflation or escalation were applied.
- Capital costs funded with 100% equity (i.e., unlevered with no project debt or associated financing costs assumed).
- No binding contractual arrangements are currently in place.
- 1.00 CAD = 0.725 USD.

22.3.1 Taxes

The Project was evaluated on a post-tax basis to provide approximate value of the potential economics. A tax model was prepared by Canagold's appointed tax consultant. The calculations are based on the tax regime in place as of the date of this study.

At the effective date of the cashflow, the Project was assumed to be subject to the following tax regime:

- Corporate income tax of 27.0%: Federal Tax of 15 % and Provincial Tax of 12%.
- The BC Minerals Tax was modelled using a net current proceeds rate of 2% and a net revenue tax rate of 13%.
- Initial tax loss carry-forward of C$6.8M.

Total tax payments are estimated to be approximately C$343M over the LOM.

22.3.2 Working Capital

A high-level estimate of working capital has been incorporated in the cash flow, based on accounts receivable (30 days), inventory (30 days), and accounts payable (30 days).

22.3.3 Closure Costs

Closure cost includes a closure bond of C$20.5M for site closure and rehabilitation.

22.3.4 Royalties

No royalties have been included in the project economics.

Canagold is challenging the validity of the historic 15% net profit interest (that can be reduced to 10% with the issuance of 150,000 Canagold shares) on the project.

22.4 Economic Analysis

For the base case, the pre-tax NPV (5%) is C$666 M with an IRR of 38.4%, and payback period is 2.3 years. On a post-tax basis, the NPV (5%) is C$425 M, with an IRR of 30.9% and payback period is 2.4 years.

For the spot case, the pre-tax NPV (5%) is C$1,246 M with an IRR of 58.7%, and payback period is 1.6 years. On a post-tax basis, the NPV (5%) is C$793 M, with an IRR of 47.2% and payback period is 1.7 years.

A summary of Project economics is shown in Table 22-1 and shown graphically in Figure 22-1.

Table 22-2 shows a summary of the projects cashflow on an annual basis.

Table 22-1: Economic Analysis Summary

Item	Units	Base	Spot
Gold Price	US$/oz Au	2,500	3,300
Foreign Exchange Rate	USD/CAD	0.72	0.72
Discount Rate	%	5.0%	5.0%
Production			
Mine life	years	8.3	8.3
Total resource mined	kt	2,830	2,830
LOM Au grade	g/t	9.94	9.94
Average annual mined resource	kt/a	342	342
Total recovered Au	koz	806	806
Average annual recovered Au	koz/a	97	97
Total payable Au	koz	708	708
Average annual payable Au	koz/a	86	86
Revenue/Costs			
Total revenue	C$M	2,444	3,226
Average annual revenue	C$M/a	296	390
EBITDA	C$M	1,469	2,251
Average annual EBITDA	C$M/a	178	272
Total on site operating costs (mining, process, G&A)	C$M	757	757
Mining unit cost	C$/t mined	85.93	85.93
Mining unit cost	C$/t milled	135.45	135.45
Process unit cost	C$/t milled	64.44	64.44
G&A unit cost	C$/t milled	67.58	67.58
Total off-site operating costs (transport, treatment & refining)	C$M	218	218
Cash Costs			
Total cash cost*	US$/oz Au	998	998
All-in sustaining cost**	US$/oz Au	1,248	1,248
Capital Costs			
Initial CAPEX	C$M	250	250
Sustaining CAPEX	C$M	225	225
Closure CAPEX	C$M	21	21
Salvage value	C$M	19	19
Pre-Tax Economics			
NPV (5%)	C$M	666	1,246
IRR	%	38.42%	58.73%
Payback	year	2.3	1.6
NPV/initial capital	-	2.7	5.0
Post-Tax Economics			
NPV (5%)	C$M	425	793
IRR	%	30.90%	47.24%
Payback	year	2.4	1.7
NPV/initial capital	-	1.7	3.2

* Total cash cost consists of mining costs, processing costs, mine-level G&A, off-site charges and royalties.

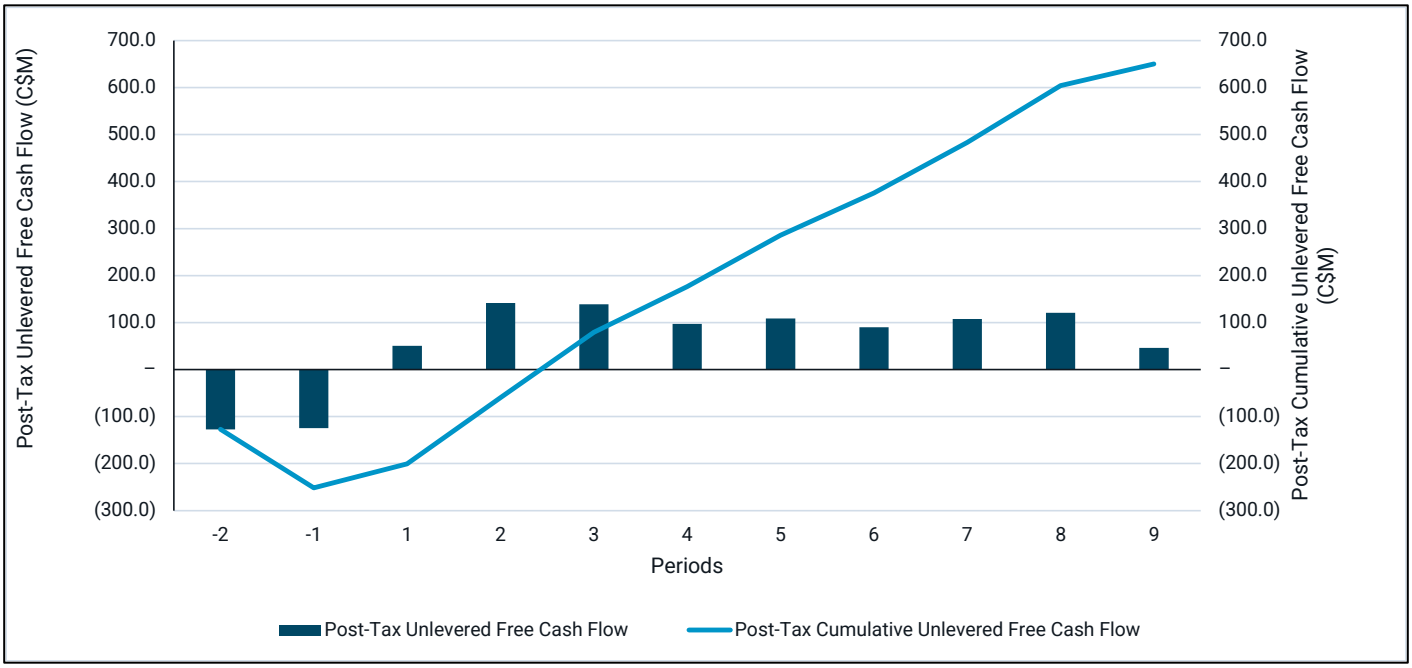
** AISC includes total cash cost, sustaining Capex and closure Capex.

Table 22-2: Cashflow Statement on an Annual Basis

Dollar figures in real 2025 C$M unless otherwise noted	Units	Total / Avg.	-2	-1	1	2	3	4	5	6	7	8	9	10	11	12
Macro assumptions																
Gold price	US$/oz	2,500	2,500	2,500	2,500	2,500	2,500	2,500	2,500	2,500	2,500	2,500	2,500	2,500	2,500	2,500
Exchange rate (USD/CAD)	CAD/USD	1.38	1.38	1.38	1.38	1.38	1.38	1.38	1.38	1.38	1.38	1.38	1.38	1.38	1.38	1.38
Free cash flow valuation																
Gross revenue	C$M	2,444.2	–	–	203.7	337.4	322.5	301.3	324.3	285.3	318.1	301.7	49.9	–	–	–
Operating expenses	C$M	(757.0)	–	–	(59.1)	(100.5)	(98.9)	(96.6)	(98.5)	(95.5)	(92.8)	(88.2)	(26.8)	–	–	–
Off-site charges	C$M	(218.3)	–	–	(12.9)	(25.4)	(27.2)	(30.2)	(29.4)	(28.5)	(30.4)	(29.1)	(5.2)	–	–	–
Royalties	C$M	–	–	–	–	–	–	–	–	–	–	–	–	–	–	–
EBITDA	**C$M**	1,468.9	–	–	131.7	211.5	196.4	174.5	196.4	161.2	194.9	184.4	17.8	–	–	–
Pre-production OPEX	C$M	(0.1)	(0.0)	(0.1)	–	–	–	–	–	–	–	–	–	–	–	–
Initial Capex	C$M	(250.4)	(127.1)	(123.3)	–	–	–	–	–	–	–	–	–	–	–	–
Sustaining Capex	C$M	(225.0)	–	–	(53.3)	(64.3)	(19.8)	(20.6)	(21.8)	(20.5)	(18.8)	(5.8)	(0.1)	–	–	–
Closure Capex	C$M	(20.5)	–	(1.0)	(1.0)	(1.0)	(1.0)	(10.3)	(6.2)	–	–	–	–	–	–	–
Salvage value	C$M	19.1	–	–	–	–	–	–	–	–	–	–	19.1	–	–	–
Change in working capital	C$M	–	–	–	(24.5)	(0.8)	(0.5)	1.2	1.5	0.3	(4.8)	5.6	21.9	–	–	–
Pre-Tax unlevered free cash flow	**C$M**	992.1	(127.1)	(124.4)	53.0	145.4	175.0	144.9	170.0	141.1	171.4	184.2	58.7	–	–	–
Pre-Tax cumulative unlevered free cash flow	C$M		(127.1)	(251.4)	(198.5)	(53.1)	121.9	266.8	436.8	577.8	749.2	933.4	992.1	992.1	992.1	992.1
Unlevered cash taxes	C$M	(343.0)	–	–	(2.6)	(4.2)	(36.2)	(47.6)	(61.2)	(51.1)	(64.0)	(63.5)	(12.5)	–	–	–
Post-Tax unlevered free cash flow	**C$M**	649.2	(127.1)	(124.4)	50.3	141.2	138.8	97.3	108.8	89.9	107.4	120.7	46.2	–	–	–
Post-Tax cumulative unlevered free cash flow	C$M		(127.1)	(251.4)	(201.1)	(59.9)	78.9	176.2	284.9	374.9	482.2	603.0	649.2	649.2	649.2	649.2
Production																
Total resource mined	**t**	**2,830,172**	**–**	**–**	**237,781**	**365,998**	**365,999**	**364,898**	**366,000**	**365,994**	**365,997**	**335,930**	**61,575**	**–**	**–**	**–**
Total material mined	**t**	**4,659,388**	**8,494**	**189,990**	**591,108**	**685,041**	**551,771**	**539,287**	**564,846**	**534,503**	**527,009**	**404,046**	**63,293**	**–**	**–**	**–**
Total mill feed	**t**	**2,830,172**	**–**	**–**	**237,781**	**365,998**	**365,999**	**364,898**	**366,000**	**365,994**	**365,997**	**335,930**	**61,575**	**–**	**–**	**–**
Mill head grade – Au	g/t	9.94	–	–	9.70	10.46	10.07	9.61	10.15	9.15	10.04	10.36	9.47	–	–	–
Metal recovered – Au in conc	oz	805,589	–	–	65,969	110,010	105,627	100,222	106,507	95,315	105,346	99,954	16,640	–	–	–
Total payable metal – Au	**oz**	**708,469**	**–**	**–**	**59,042**	**97,802**	**93,473**	**87,330**	**94,013**	**82,686**	**92,211**	**87,449**	**14,463**	**–**	**–**	**–**
Total revenue	**C$M**	**2,444.2**	**–**	**–**	**203.7**	**337.4**	**322.5**	**301.3**	**324.3**	**285.3**	**318.1**	**301.7**	**49.9**	**–**	**–**	**–**
Operating Costs																
Total operating costs	C$M	757.0	–	–	59.1	100.5	98.9	96.6	98.5	95.5	92.8	88.2	26.8	–	–	–
Mine operating costs	C$M	383.3	–	–	18.1	55.4	53.5	51.2	53.1	50.1	47.4	43.7	10.8	–	–	–
Mill processing costs	C$M	182.4	–	–	18.3	22.4	22.7	22.7	22.7	22.7	22.7	21.8	6.6	–	–	–
G&A costs	C$M	191.3	–	–	22.7	22.7	22.7	22.7	22.7	22.7	22.7	22.7	9.4	–	–	–
Off-site charges and royalties																
Total off-site costs	**C$M**	**218.3**	**–**	**–**	**12.9**	**25.4**	**27.2**	**30.2**	**29.4**	**28.5**	**30.4**	**29.1**	**5.2**	**–**	**–**	**–**
Total freight and marketing charge	C$M	218.3	–	–	12.9	25.4	27.2	30.2	29.4	28.5	30.4	29.1	5.2	–	–	–
Royalties	C$M	–	–	–	–	–	–	–	–	–	–	–	–	–	–	–
Cash costs																
Total cash cost*	US$/oz Au	998	–	–	883	933	978	1,052	986	1,087	968	972	1,606	–	–	–
AISC**	US$/oz Au	1,248	–	–	1,549	1,417	1,139	1,308	1,202	1,266	1,116	1,020	1,613	–	–	–
Capital Costs																
Total initial Capex	**C$M**	**250.4**	**127.1**	**123.3**	**–**	**–**	**–**	**–**	**–**	**–**	**–**	**–**	**–**	**–**	**–**	**–**
Mining	C$M	63.3	48.9	14.4	–	–	–	–	–	–	–	–	–	–	–	–
Process Plant	C$M	43.0	17.2	25.8	–	–	–	–	–	–	–	–	–	–	–	–
Tailings Facilities	C$M	7.4	3.0	4.4	–	–	–	–	–	–	–	–	–	–	–	–
On Site Infrastructure	C$M	38.5	15.4	23.1	–	–	–	–	–	–	–	–	–	–	–	–
Off-Site Infrastructure	C$M	9.4	3.8	5.7	–	–	–	–	–	–	–	–	–	–	–	–
Indirects	C$M	42.3	16.9	25.4	–	–	–	–	–	–	–	–	–	–	–	–
Project Delivery	C$M	9.8	3.9	5.9	–	–	–	–	–	–	–	–	–	–	–	–
Owner's Costs	C$M	7.8	3.1	4.7	–	–	–	–	–	–	–	–	–	–	–	–
Provisions	C$M	28.8	14.9	13.9	–	–	–	–	–	–	–	–	–	–	–	–
Total sustaining Capex	**C$M**	**225.0**	**–**	**–**	**53.3**	**64.3**	**19.8**	**20.6**	**21.8**	**20.5**	**18.8**	**5.8**	**0.1**	**–**	**–**	**–**
Underground Mining	C$M	196.1	–	–	47.5	53.6	17.7	17.9	19.5	17.8	16.8	5.2	0.1	–	–	–
Tailings and Waste Rock Co-Disposal	C$M	4.7	–	–	–	3.8	–	0.4	–	0.5	–	–	–	–	–	–
Contingency -AUSENCO	C$M	0.7	–	–	–	0.5	–	0.1	–	0.1	–	–	–	–	–	–
Contingency -JDS	C$M	23.5	–	–	5.7	6.4	2.1	2.2	2.3	2.1	2.0	0.6	0.0	–	–	–
Closure cost	C$M	20.5	–	1.0	1.0	1.0	1.0	10.3	6.2	–	–	–	–	–	–	–
Salvage value	C$M	19.1	–	–	–	–	–	–	–	–	–	–	19.1	–	–	–
Total capital expenditure incl. salvage value	**C$M**	**476.7**	**127.1**	**124.3**	**54.3**	**65.3**	**20.9**	**30.8**	**28.0**	**20.5**	**18.8**	**5.8**	**(19.0)**	**–**	**–**	**–**

* Total cash cost consists of mining costs, processing costs, mine-level G&A, off-site charges and royalties. ** AISC includes total cash cost, sustaining Capex and closure Capex.

Figure 22-1: Projected LOM Post-Tax Unlevered Free Cash Flow



Source: Ausenco, 2025.

22.5 Sensitivity Analysis

A sensitivity analysis was conducted on the Project's pre-tax and post-tax NPV and IRR assessing the impact of changes in gold price, opex, initial capex, head grade, recovery and exchange rate.

The analysis indicated that the Project is most sensitive to fluctuations in head grade and metal price, with less sensitivity to changes in exchange rate and recovery, as shown in Table 22-3, Table 22-4, Figure 22-2 and Figure 22-3.

Table 22-3: Pre-Tax Sensitivity

Pre-Tax NPV Sensitivity To Discount Rate

Discount Rate	Metal Price (20.00%)	(10.00%)	-	10.00%	20.00%
3.0%	374	577	781	984	1,188
5.0%	304	485	666	847	1,028
8.0%	219	372	525	678	831
10.0%	173	310	448	585	723
12.0%	133	257	381	505	629

Pre-Tax IRR Sensitivity To Discount Rate

Discount Rate	Metal Price (20.00%)	(10.00%)	-	10.00%	20.00%
3.0%	22.52%	30.92%	38.42%	45.28%	51.63%
5.0%	22.52%	30.92%	38.42%	45.28%	51.63%
8.0%	22.52%	30.92%	38.42%	45.28%	51.63%
10.0%	22.52%	30.92%	38.42%	45.28%	51.63%
12.0%	22.52%	30.92%	38.42%	45.28%	51.63%

Pre-Tax NPV Sensitivity To OPEX

OPEX	Metal Price (20.00%)	(10.00%)	-	10.00%	20.00%
(20.0%)	416	597	778	959	1,140
(10.0%)	360	541	722	903	1,084
--	304	485	666	847	1,028
10.0%	248	429	610	791	972
20.0%	192	373	554	735	916

Pre-Tax IRR Sensitivity To OPEX

OPEX	Metal Price (20.00%)	(10.00%)	-	10.00%	20.00%
(20.0%)	27.87%	35.69%	42.78%	49.31%	55.41%
(10.0%)	25.25%	33.34%	40.63%	47.32%	53.54%
--	22.52%	30.92%	38.42%	45.28%	51.63%
10.0%	19.68%	28.42%	36.16%	43.19%	49.69%
20.0%	16.70%	25.83%	33.84%	41.06%	47.70%

Pre-Tax NPV Sensitivity To Initial Capex

Initial Capex	Metal Price (20.00%)	(10.00%)	-	10.00%	20.00%
(20.0%)	352	533	714	895	1,076
(10.0%)	328	509	690	871	1,052
--	304	485	666	847	1,028
10.0%	280	461	642	823	1,004
20.0%	256	437	618	799	981

Pre-Tax IRR Sensitivity To Initial Capex

Initial Capex	Metal Price (20.00%)	(10.00%)	-	10.00%	20.00%
(20.0%)	28.21%	37.58%	45.98%	53.67%	60.80%
(10.0%)	25.15%	33.99%	41.91%	49.14%	55.85%
--	22.52%	30.92%	38.42%	45.28%	51.63%
10.0%	20.23%	28.25%	35.40%	41.93%	47.98%
20.0%	18.20%	25.89%	32.75%	39.00%	44.79%

Pre-Tax NPV Sensitivity To Head Grade

Head Grade	Metal Price (20.00%)	(10.00%)	-	10.00%	20.00%
(20.0%)	(31)	108	247	386	526
(10.0%)	140	300	461	621	782
--	304	485	666	847	1,028
10.0%	473	676	878	1,080	1,282
20.0%	641	864	1,088	1,311	1,534

Pre-Tax IRR Sensitivity To Head Grade

Head Grade	Metal Price (20.00%)	(10.00%)	-	10.00%	20.00%
(20.0%)	2.79%	11.99%	19.78%	26.67%	32.91%
(10.0%)	13.84%	22.45%	30.00%	36.81%	43.07%
--	22.52%	30.92%	38.42%	45.28%	51.63%
10.0%	30.31%	38.67%	46.23%	53.19%	59.67%
20.0%	37.15%	45.55%	53.22%	60.31%	66.94%

Pre-Tax NPV Sensitivity To Recovery

Recovery	Metal Price (20.00%)	(10.00%)	-	10.00%	20.00%
(20.0%)	(13)	129	270	411	553
(10.0%)	150	312	474	635	797
--	304	485	666	847	1,028
10.0%	462	662	863	1,064	1,265
20.0%	497	703	908	1,113	1,318

Pre-Tax IRR Sensitivity To Recovery

Recovery	Metal Price (20.00%)	(10.00%)	-	10.00%	20.00%
(20.0%)	4.10%	13.20%	20.96%	27.83%	34.06%
(10.0%)	14.42%	23.00%	30.54%	37.36%	43.62%
--	22.52%	30.92%	38.42%	45.28%	51.63%
10.0%	29.81%	38.17%	45.73%	52.68%	59.16%
20.0%	31.31%	39.67%	47.24%	54.21%	60.71%

Pre-Tax NPV Sensitivity To FX Rate

FX Rate	Metal Price (20.00%)	(10.00%)	-	10.00%	20.00%
(20.0%)	47	191	336	481	626
(10.0%)	175	338	501	664	827
--	304	485	666	847	1,028
10.0%	433	632	831	1,030	1,229
20.0%	562	779	996	1,213	1,431

Pre-Tax IRR Sensitivity To FX Rate

FX Rate	Metal Price (20.00%)	(10.00%)	-	10.00%	20.00%
(20.0%)	8.09%	16.63%	24.04%	30.68%	36.75%
(10.0%)	15.77%	24.17%	31.59%	38.31%	44.51%
--	22.52%	30.92%	38.42%	45.28%	51.63%
10.0%	28.63%	37.10%	44.73%	51.74%	58.26%
20.0%	34.25%	42.83%	50.61%	57.79%	64.48%

Table 22-4: Post-Tax Sensitivity

Post-Tax NPV Sensitivity To Discount Rate

Discount Rate	Metal Price (20.00%)	(10.00%)	-	10.00%	20.00%
3.0%	246	375	504	633	762
5.0%	194	310	425	540	655
8.0%	131	230	328	426	523
10.0%	96	186	274	362	450
12.0%	67	148	228	308	387

Post-Tax IRR Sensitivity To Discount Rate

Discount Rate	Metal Price (20.00%)	(10.00%)	-	10.00%	20.00%
3.0%	18.00%	24.80%	30.90%	36.44%	41.55%
5.0%	18.00%	24.80%	30.90%	36.44%	41.55%
8.0%	18.00%	24.80%	30.90%	36.44%	41.55%
10.0%	18.00%	24.80%	30.90%	36.44%	41.55%
12.0%	18.00%	24.80%	30.90%	36.44%	41.55%

Post-Tax NPV Sensitivity To OPEX

OPEX	Metal Price (20.00%)	(10.00%)	-	10.00%	20.00%
(20.0%)	266	381	496	611	726
(10.0%)	230	346	461	576	691
--	194	310	425	540	655
10.0%	158	274	389	505	620
20.0%	123	238	354	469	584

Post-Tax IRR Sensitivity To OPEX

OPEX	Metal Price (20.00%)	(10.00%)	-	10.00%	20.00%
(20.0%)	22.34%	28.70%	34.45%	39.72%	44.63%
(10.0%)	20.21%	26.78%	32.69%	38.10%	43.10%
--	18.00%	24.80%	30.90%	36.44%	41.55%
10.0%	15.72%	22.76%	29.05%	34.74%	39.97%
20.0%	13.35%	20.66%	27.15%	33.01%	38.36%

Post-Tax NPV Sensitivity To Initial Capex

Initial Capex	Metal Price (20.00%)	(10.00%)	-	10.00%	20.00%
(20.0%)	236	352	467	582	697
(10.0%)	215	331	446	561	676
--	194	310	425	540	655
10.0%	174	289	404	519	634
20.0%	153	268	383	498	613

Post-Tax IRR Sensitivity To Initial Capex

Initial Capex	Metal Price (20.00%)	(10.00%)	-	10.00%	20.00%
(20.0%)	23.34%	31.02%	37.87%	44.11%	49.85%
(10.0%)	20.46%	27.66%	34.10%	39.97%	45.36%
--	18.00%	24.80%	30.90%	36.44%	41.55%
10.0%	15.87%	22.32%	28.12%	33.40%	38.26%
20.0%	14.00%	20.14%	25.68%	30.73%	35.38%

Post-Tax NPV Sensitivity To Head Grade

Head Grade	Metal Price (20.00%)	(10.00%)	-	10.00%	20.00%
(20.0%)	(40)	65	158	247	336
(10.0%)	89	192	294	397	499
--	194	310	425	540	655
10.0%	302	431	560	688	816
20.0%	409	551	693	835	976

Post-Tax IRR Sensitivity To Head Grade

Head Grade	Metal Price (20.00%)	(10.00%)	-	10.00%	20.00%
(20.0%)	2.08%	9.56%	15.81%	21.36%	26.41%
(10.0%)	11.13%	17.94%	24.05%	29.58%	34.65%
--	18.00%	24.80%	30.90%	36.44%	41.55%
10.0%	24.30%	31.10%	37.21%	42.81%	48.02%
20.0%	29.87%	36.68%	42.85%	48.55%	53.85%

Post-Tax NPV Sensitivity To Recovery

Recovery	Metal Price (20.00%)	(10.00%)	-	10.00%	20.00%
(20.0%)	(27)	80	173	263	353
(10.0%)	96	199	303	406	509
--	194	310	425	540	655
10.0%	295	423	550	678	805
20.0%	318	448	579	709	839

Post-Tax IRR Sensitivity To Recovery

Recovery	Metal Price (20.00%)	(10.00%)	-	10.00%	20.00%
(20.0%)	3.08%	10.60%	16.75%	22.29%	27.35%
(10.0%)	11.60%	18.39%	24.49%	30.02%	35.09%
--	18.00%	24.80%	30.90%	36.44%	41.55%
10.0%	23.90%	30.70%	36.81%	42.40%	47.61%
20.0%	25.12%	31.91%	38.03%	43.65%	48.86%

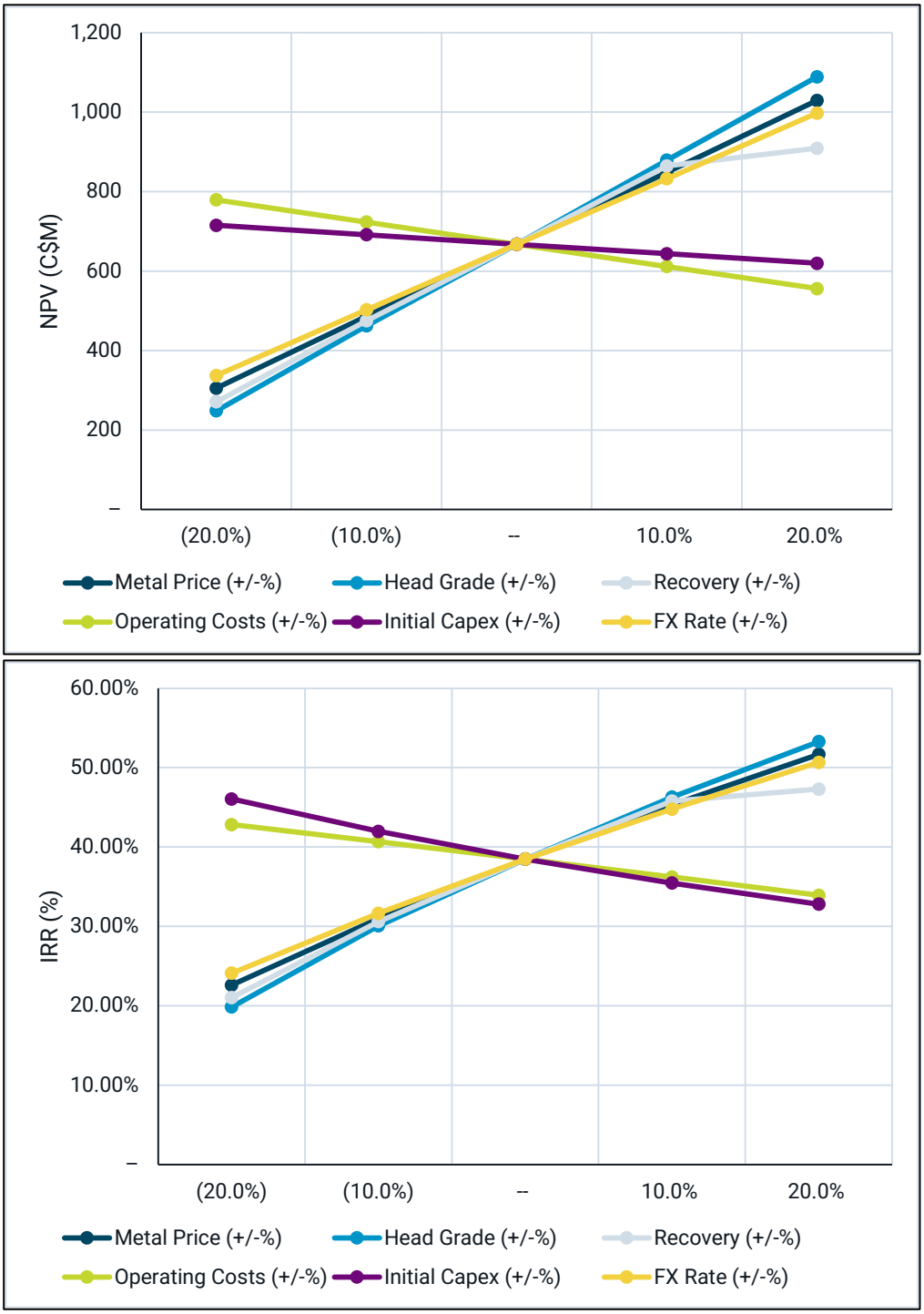
Post-Tax NPV Sensitivity To FX Rate

FX Rate	Metal Price (20.00%)	(10.00%)	-	10.00%	20.00%
(20.0%)	18	122	215	307	400
(10.0%)	112	216	320	424	527
--	194	310	425	540	655
10.0%	277	403	530	657	783
20.0%	359	497	635	773	911

Post-Tax IRR Sensitivity To FX Rate

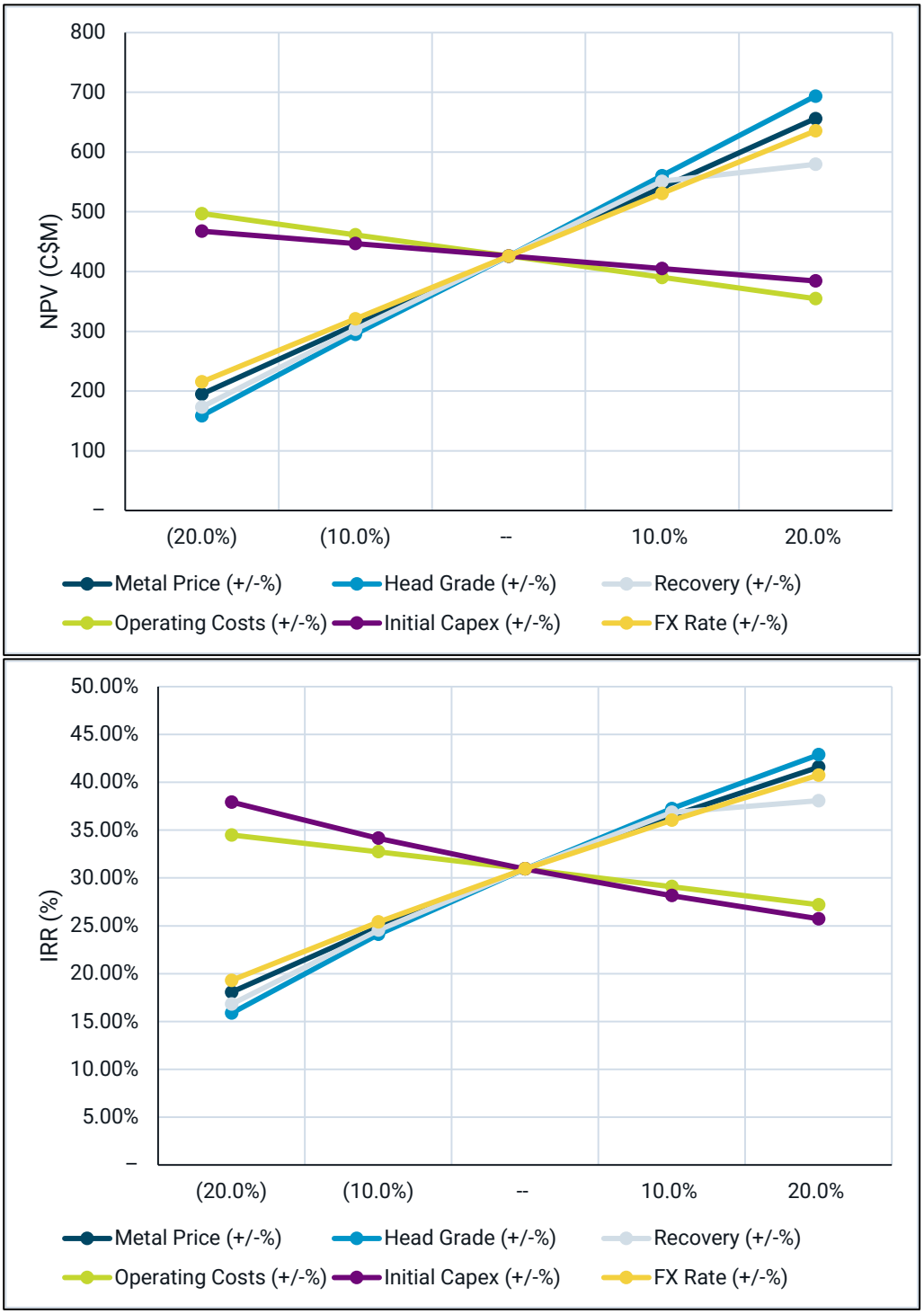
FX Rate	Metal Price (20.00%)	(10.00%)	-	10.00%	20.00%
(20.0%)	6.27%	13.31%	19.24%	24.61%	29.55%
(10.0%)	12.64%	19.33%	25.34%	30.81%	35.82%
--	18.00%	24.80%	30.90%	36.44%	41.55%
10.0%	22.94%	29.82%	35.99%	41.62%	46.86%
20.0%	27.49%	34.45%	40.71%	46.48%	51.83%

Figure 22-2: Pre-Tax Sensitivity Analysis



Source: Ausenco, 2025.

Figure 22-3: Post-Tax Sensitivity Analysis



Source: Ausenco, 2025.

23 ADJACENT PROPERTIES

No relevant information or data to disclose.

24 OTHER RELEVANT DATA AND INFORMATION

24.1 Project Execution Plan

The Project Execution Plan (PEP) addresses the overall project objectives, scope, strategies and roles and responsibilities, and provides a comprehensive plan for its development and implementation. The PEP covers the plan for engineering, procurement, construction, start-up, and commissioning of the project. The implementation strategy assumes an engineering, procurement, construction management (EPCM) implementation. The PEP will serve as the main communication tool to ensure the project team is aware and knowledgeable of the Project objectives and how they will be accomplished. A preliminary EPCM schedule has been developed and will be updated during the next project phase.

24.2 Project Organization and Alignment Strategy

The Project organization is based on an integrated team approach, minimising the duplication of roles and activities between the EPCM firm and the owner's team. Full-service delivery of the Project will be enabled from the Project's site office. The organizational chart for the Project team will be developed during the next project phase. The Project organization chart will be a live document, constantly being updated as the Project progresses.

The Alignment Strategy aims to create shared understanding of the Project vision and strategy to enable the EPCM firm, Canagold, and other internal/external stakeholders to achieve the Project objectives. The overall Project delivery team operates as one team with defined responsibilities, accountabilities, and authorities. The team is established and supported to deliver "Best for Project" outcomes in line with Canagold's expectations and critical success factors.

Establishment of the Project delivery team working relationships and agreeing acceptable desired outcomes will be conducted during facilitated alignment sessions. The alignment effort will be concentrated at the front-end of the project, although ongoing activities are planned throughout to increase overall effectiveness, commitment, and cohesiveness of Project team members.

24.3 Health, Safety & Environment Management Plan

The EPCM firm's alignment with Canagold's commitment to ensuring appropriate HSEC behaviours is key to the success of the Project and is reflected in the Project KPIs.

The HSEC Plan will detail the strategy, systems, process, procedures, and standards which are used on the Project and the mechanisms for recording, analysing, and reporting performance.

Home office and on-site selection and management of vendors and contractors as well as internal team members must take account of HSEC behaviour and performance.

HSEC Standards, Systems, and Safe Operating Procedures (SOPs) developed by Canagold will be supplemented by the EPCM firm's procedures and forms. Project "on-boarding" requirements are substantial and will be in accordance with

established Canagold programs. The EPCM firm will support and assist Canagold in the SOP development for the Project. The construction manager will be supported by a site based HSEC team, who will monitor and advise on HSEC performance and ensure alignment of Project team-members with Canagold's requirements.

24.4 Engineering Management Plan

The Engineering Management Plan (EMP) will detail the strategy, processes, and standards for the delivery of the engineering requirements of the project.

Key responsibilities for engineering on this Project include:

- integrating the engineering schedule with the overall schedule including sequencing of packages (through constructability input) and vendor data,
- interface between multiple engineering providers or locations needs careful management because of the different levels of design definition, and
- involvement of site operations team in key design reviews.

The EMP will outline the overall design and revision processes for Project activities. It sets out specification and flow diagram standards, and references all engineering procedures, systems, and forms to be used.

When required, the engineering team will provide technical advice to the procurement, construction management, commissioning, and quality departments throughout the project. Resources for ongoing field support during construction and commissioning phases will be drawn from the engineering teams.

24.5 Procurement and Contracts Strategy and Management Plan

The Procurement Strategy and Management Plan details the strategy for procuring equipment and materials for the Project, in addition to the processes and systems that will be used for procuring, expediting, shipping, receiving, storing, and issuing the equipment and materials.

The Contracting Strategy and Management Plan details the strategy that has been developed for identifying and engaging construction contractors for the project and the processes and systems that will be used to deliver the project. It also explains the management basis of the contract's formation and contracts administration procedure that will be applied along with an explanation of tools, operational controls and earned value performance monitoring techniques. A Contracting Plan Summary is appended and identifies all proposed work packages with their contract style, to be updated and confirmed during the next phase. A Contracting Matrix and Responsibility Matrix are also appended to highlight responsibilities that will be retained by Canagold versus those that rest with the EPCM firm or other contractors.

The procurement and contracting team will invite tenders and award purchase orders and contracts according to packages defined by the Contract Strategy and Management Plan supported by engineering and by Canagold's approved bidders list.

24.6 Construction Execution Plan

The Construction Execution Plan (CEP) will detail the processes to be used for the management of construction on-site and will be used in conjunction with the PEP.

Construction Management shall conform to the documented practices set out in the Project Quality Plan, Risk Management Plan and in strict accordance with the HSEC Management Plan.

25 INTERPRETATION AND CONCLUSIONS

25.1 Introduction

The QPs note the following interpretations and conclusions in their respective areas of expertise, based on the review of data available for this Report.

25.2 Mineral Tenure, Surface Rights, Water Rights, Royalties and Agreements

- The mineral tenure held is valid and sufficient to support the Mineral Resources.
- Surface rights will be required from the Crown before operations.
- Royalties are payable to third parties.
- There are no other significant factors and risks that may affect access, title, or the right or ability to perform work on the property that have not been discussed in this report.

25.3 Geology, Mineralization, Exploration, and Drilling

- The deposit is considered to be an example of Mesothermal lode gold mineralization.
- Knowledge of the deposit settings, lithologies, mineralization style and setting, and structural and alteration controls on mineralization is sufficient to support the Mineral Resource estimation.
- The quantity and quality of the lithological, collar and downhole survey data collected in the drill programs are sufficient to support Mineral Resource estimation.
- Canagold has been drilling on the Property since 1988. To date total drilling totals approximately 145 km in approximately 1,100 drillholes.
- The sample security, sample preparation and analytical procedures during the exploration programs by Canagold followed accepted industry practice appropriate for the stage of mineral exploration undertaken.
- Data verification has been conducted by Canagold, and no material issues have been identified by those programs.
- Data collected have been sufficiently verified for post 1988 drilling that these drillholes can support Mineral Resource estimation and can be used for mine planning purposes.

25.4 Metallurgical Testwork

- BIOX™ pilot plant testing showed that the required high capital and high operating conditions required to oxidize New Polaris concentrates with BIOX processing would not be economic.
- Flotation testing using the established flowsheet consisting of rougher flotation followed by two stages of cleaning generated concentrates at an average life-of-mine gold recovery of 88.9%.

- Flotation concentrates will contain arsenic at high concentrations that may incur penalties or impact the marketability of concentrates produced.
- Flotation concentrates will have a projected average grade of 128 g/t gold that will facilitate blending with cleaner concentrates for third-party processing.

25.5 Mineral Resource Estimate

The effective date of this Resource estimate is January 22, 2025. Mineral resources that are not mineral reserves do not have demonstrated economic viability.

- The base case Mineral Resource contains an Indicated resource of 2.97 Mt at 11.61 g/t Au for 1.1 Moz of Au and 0.859 Mt at 0.65% Sb for 12.4 Mlbs of Sb, and an Inferred resource of 0.93 Mt at 8.93 g/t Au for 0.27 Moz of Au and 0.100 Mt at 1.20% Sb for 2.6 Mlbs Sb.
- In total, 234 core drill holes from 1988 to 2022 inclusive have been used to determine the Resource estimate for Au, and 215 core drillholes from 1996 to 2024 inclusive have been used to determine the Resource estimate for Sb. A limited amount of historical data has been used and was validated.
- The updated resource accounts for "reasonable prospects of eventual economic extraction" by removing any portions of the mineralization that have a true thickness of less than two meters, and by removing discontinuous mineralization that is not of sufficient volume to have potential for underground mining at the current state of knowledge of the deposit.
- The QP for the resource estimate is not aware of any environmental, permitting, legal, title, taxation, socio-economic, marketing, political, or other relevant factors that could materially affect the Mineral Resource Estimate other than those stated in this report.

25.6 Mineral Reserve Estimate

The Mineral Reserve estimate is reported in accordance with National Instrument 43-101 Standards of Disclosure for Mineral Projects. The mineral reserve estimate follows the Canadian Institute of Mining, Metallurgy and Petroleum (CIM) 2014 "CIM Definition Standards for Mineral Resources & Mineral Reserves" (CIM, 2014) and the "CIM Estimation of Mineral Resources & Mineral Reserves Best Practice Guidelines" (CIM, 2019).

Mineral Reserves are reported for both Zone C and Zone Y, with a total Probable Reserve for the New Polaris property estimated at 2.830 Mt at 9.94 g/t of gold, 1,828 ppm of antimony, and 10,960 ppm of arsenic at a gold cut-off grade of 6.0 g/t drift and fill stopes, 5.5 g/t for longhole stopes, 4.2 g/t Au marginal cut-off grade for development ore and based on a gold price of US$2,245/oz.

25.7 Mining Methods

In terms of underground mining methods and plan, underground geotechnical assessment and mine Opex/Capex estimate, it is the conclusion of the QPs that this technical report contains adequate detail and information to support a feasibility level analysis. Standard industry practices, equipment and design methods were used in this Feasibility

Study and except for those outlined in this section, the report authors are unaware of any unusual or significant risks or uncertainties that would affect project reliability or confidence based on the data and information made available.

25.8 Recovery Methods

The recovery methods align with conventional base metal practices in the industry. Ore comminution, flotation recovery of payable metals and handling of tailings are achieved through typical processes that are commonly used in the industry for similar projects. Previous studies, along with new testwork results, were used to develop the resulting flowsheet suitable for the project.

25.9 Markets and Contracts

New Polaris will produce a high-grade gold concentrate which is attractive in the current high gold price market environment. For this study, selling to traders for blending was chosen. The proposed route for transport is via air cargo to Juneau, barge transport to Tacoma then ocean shipment to the customer.

Project economics were estimated based on long-term metal prices of US$2,500/oz Au and a CAD:USD exchange rate of 0.725, which was based on consensus forecasts from various financial institutions

25.10 Environmental, Permitting and Social Considerations

This section summarized key environmental and social factors which may influence permitting outcomes, construction readiness, and the Project's ability to meet regulatory, community, and investor expectations.

25.10.1 General Considerations

- The project is located in a remote, ecologically sensitive area near the Tulsequah and Taku Rivers, which support diverse aquatic life including salmonids.
- Mine waste materials contain elevated arsenic and antimony, with potential for leaching under neutral pH.
- Historical mining has contributed to elevated arsenic and metal concentrations in surface and groundwater. A comprehensive water treatment system is planned, including diversion channels, sediment ponds, and a lined CSF for tailings and waste rock.
- Future climate conditions may alter the volumes and timing of glacier runoff, and other hydrology, water, and precipitation regimes which affect the site during construction, operations, and closure.
- The area supports species at risk (e.g., grizzly bear, little brown myotis) and blue-listed ecosystems.
- Baseline studies indicate low levels of air and noise pollution due to remoteness. GHG emissions are expected to be below regulatory thresholds.
- Secure, lined fuel and explosives storage facilities are planned to prevent leakage and potential contamination.

25.10.1.1 Permitting Considerations

- The project is currently undergoing a full BC Environmental Assessment (EA) and requires numerous provincial and federal permits. It is not subject to federal Impact Assessment due to its scope of activities and size.
- Permitting is planned in a phased approach to support early works construction activities (e.g., airstrip, barge landing) on a seasonal basis in preparation for future construction and operation of the Mine Site.
- The project must align with BC's Climate Change Accountability Act and GHG reporting requirements.

25.10.1.2 Social Considerations

- The project is within the traditional territory of the TRTFN, who have established an IPCA nearby. Engagement includes a signed Hà khustìyxh/Our Way Agreement and ongoing collaboration.
- Archaeological assessments have been conducted; no significant sites were identified. Archaeological and Paleontological Management Plans will be required for the Project.
- Key issues include potential impacts to fisheries and aquatic health, and cumulative effects including from the nearby Tulsequah Chief Mine, water quality, barge transport impacts, and long-term site reclamation.
- Potential for local employment and economic benefits, particularly for Indigenous communities.

25.11 Capital Cost Estimate

The capital cost estimate was developed in Q1 2025 Canadian Dollars (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 conforms 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).

The total initial capital cost for the New Polaris Project is $250.4M, the LOM sustaining costs are $225.0M. Closure costs are estimated at C$20.5M.

25.12 Operating Cost Estimate

The operating cost estimate was developed in Q1 2025 Canadian Dollars (CAD) from budgetary quotations and Ausenco's in-house database of projects and studies as well as experience in similar operations.

The estimate includes mining, processing, and general and administrative (G&A) costs.

25.13 Economic Analysis

Based on the assumptions in this report, the Project has a positive cash flow. For the base case, the pre-tax net present value discounted at 5% (NPV5%) is C$666M, the IRR is 38.4%, and the payback period is 2.3 years. On a post-tax basis, the NPV5% is C$425M, the IRR is 30.9%, and the payback period is 2.4 years. For the spot case, the pre-tax net present

value discounted at 5% (NPV5%) is C$1,246 M, the IRR is 58.7%, and the payback period is 1.6 years. On a post-tax basis, the NPV5% is C$793 M, the IRR is 47.2%, and the payback period is 1.7 years.

A sensitivity analysis was conducted on the base case pre-tax and post-tax NPV and IRR of the Project, using the following variables: gold price, operating costs, initial capex, head grade, recovery and exchange rate. The analysis revealed that the Project is most sensitive to changes in gold price and head grade followed by, to a lesser extent, exchange rate and recovery.

25.14 Risks and Opportunities

The following discussion of risks and opportunities involves forward-looking statements that are based on reasonable expectations and informed by the recent past. Readers are cautioned that such forward-looking statements involve uncertainties and unknowns that may cause actual outcomes to differ from those implied by these forward-looking statements.

25.14.1 Risks

25.14.1.1 Metallurgical Testwork

The arsenic, sulphur and antimony grades of the variability samples tested are higher than modelled grades in the mine plan. The resulting recovery models show linearity towards lower grades but may not be accurate with respect to concentrate mass produced and arsenic grades. The testing completed to date has not focused on antimony.

No metallurgical testing has been completed to support the production of saleable antimony concentrates. A flowsheet that demonstrates that saleable antimony concentrates is required to convert antimony resources to reserves.

25.14.1.2 Mining

25.14.1.2.1 Mine Engineering

Risk is present in any development project. Feasibility study engineering formulates design and engineering solutions to reduce that risk common to every mining project such as resource uncertainty, mining recovery, geotechnical and dilution control, metallurgical recoveries, environmental and social impact, schedule and cost overruns, and labour sourcing. The QPs are of the opinion that these risks have been clearly identified, and mitigation measures have been considered.

In terms of the underground mine plan, Table 25-1 presents identified material risks, their potential impact, and potential mitigation measures.

Table 25-1: Underground Mining Risks

Risks and Potential Impact	Mitigation Approach
Risk: Not achieving development and/or production priorities or mobile equipment productivities. Potential impact: Reduction in production rate and an increase in operating costs.	Consider automation such as tele-remote drilling, mucking and hauling between shifts
Risk: Qualified personnel availability. Potential impact: Reduction in production rate and an increase in operating costs.	Hire additional staff more than levels recommended in this study. Consider extending the contract mining duration beyond the planned 30 months until skilled site operations and maintenance personnel persist.
Risk: Personnel cost related inflation. Potential impact: Increase in operating cost.	Evaluate technological solutions to automate more aspects of the mining process and to increase efficiency in those mining processes that cannot be automated. Review short term, long term, and life of mine plans to adjust for changing costs and revenues.
Risk: Delays with the air transportation of bulk explosives, consumables or other key supplies. Potential impact: Reduction of production and development mining and an increase in operating costs.	Increase amount of stock on hand of key products.
Risk: Ramp congestion reduces truck productivity.	Take a tiered approach with a view to reducing total cost impact to the site: • Implement a 2-way radio type traffic light system triggered by mobile equipment operators to indicate the presence of equipment in the ramp. • Implement a fleet management or similar vehicle tracking system that would provide more data and have greater effectiveness with ramp traffic management.
Risk: Lower mobile equipment mechanical availability. Potential impact: Production shortfall resulting in increased operating costs.	Assess effectiveness of the mobile equipment reliability maintenance programme and put controls in place to address root causes. Assess capital spare components for mobile equipment should availability of repair parts be a contributing factor to the reduced mechanical availability. If component reliability on either re-build or re-manufactured equipment is found to be a contributing factor, purchase new replacement machines.
Risk: Higher than planned dilution Potential impact: Reduction in annual gold production and increased tailings generation.	Review drilling and blasting practices, engaging expert professionals as required until the root causal factors are understood and controlled. Should dilution be coming from paste backfilled stopes, assess binder content in addition to drilling and blasting practices. Ensure where possible the drilling control system onboard the Jumbo is utilized to increase drill pattern execution compliance.

25.14.1.2.2 Geotechnical Engineering

The geotechnical risks identified for New Polaris are presented in Table 25-2. These risks could impact mining conditions, support requirements, production rates and costs. However, given their nature, these risks can be mitigated by further geotechnical and hydrogeological investigation, thorough assessment, and appropriate design modifications.

Table 25-2: Geotechnical Risks

Risks and Potential Impact	Mitigation Approach
Geotechnical drilling coverage is limited, and the geotechnical data is sparse in the upper sections of No. 3 and No. 6 veins. The rock mass conditions in these areas with poor coverage are Inferred to be similar to other areas. Poorer rock mass conditions could be encountered. This can cause instability due to inadequate ground support for poorer ground.	• Collect geotechnical data (geotechnical core logging and testing) of addition drilling through the planned mining areas with poor drilling spatial coverage and limited data. • Conduct regular mapping and characterization of the rock mass as development advances. • Additional ground support can be installed where required. • Modify ground support design as required for poor rock mass conditions.
Limited structural fabric orientation data is available from the HW and within the ore zones. As a result, dominant joint sets identified may be not representative of actual conditions. This can affect ground support designs based on wedge analyses, where joint set orientations are the major inputs.	• Collect structure orientation data of drillholes in vicinity of mining and development areas. • Conduct regular underground face mapping as development advances. • Update kinematic assessments based on updated orientation data and modify ground support designs as required.
Higher external dilution could be encountered for several reasons (e.g. rock mass conditions worse than expected, poor drilling and blasting techniques). Higher external dilution can lead to lower mining recovery, lower grades, and higher operating costs.	• Implement and maintain good drilling and blasting practices for development and stoping to reduce blast-induced damage. • Survey stope blastholes for drilling accuracy as required. • Conduct regular mapping and characterization of the rock mass as development advances. • Modify ground support designs as required.
Backfill strength lower than expected can cause higher levels of dilution from backfill.	• Implement a quality control program for the backfill at the start of mining.

25.14.1.3 Recovery Methods

There is a risk that the process plant will not achieve throughput and/or recovery estimates but further variability testwork will help mitigate this.

As a result of the geographical constraints, concentrate produced by the New Polaris Project is limited to air transportation. A concentrate dryer was added later into the design to reduce the moisture content of the concentrate filter press to a level suitable for bagging and air freight. Several risks are involved with concentrate drying:

- Health and safety concerns in drying and bagging area with the potential release of dust containing high arsenic may result in the need for increased dust control and collection equipment.
- Filterability of the concentrate results in higher % moisture than design resulting in insufficient dryer capacity and higher operating and transportation costs. This can be mitigated with further variability testwork.

A single source, overseas process plant vendor quote has been used for cost estimating, including supply of all process and ancillary mechanical/electrical equipment and mechanical/ electrical bulks. There is a risk of price and schedule fluctuations due to availability and quality of equipment and materials when depending on a single supplier. This can be mitigated through increased quality and schedule monitoring during fabrication.

25.14.1.4 Project Infrastructure

25.14.1.4.1 Roads

The road alignment from the barge landing facility to the mine site area, and from the mine site area to the CSF, are at low elevation and so at risk of flooding. Therefore, road maintenance will be critical for the upkeep of mine operations and it may be common for road reparation activities to occur.

25.14.1.4.2 Co-Storage Facility

A limited geotechnical field and laboratory investigation was conducted for the FS due to some technical issues with the drilling in the CSF footprint. Therefore, an additional geotechnical field and laboratory program needs to be completed to determine whether the foundation conditions (especially at depths greater than 3 mbgs investigated by the test pitting program) are similar or how they might potentially affect the CSF. This could possibly require modifications to the design that might increase initial capital, sustaining capital, and operating costs.

Both the tailings and waste rock stored in the CSF are classified as non-acid generating (NAG) based on laboratory tests. However, there is neutral metal leaching of arsenic and antimony based on the laboratory program. Currently, we do not have a long-term water quality model or an understanding of the potential impacts. During operations, seepage through the pile and surface water will be directed to the contact water pond. Depending on the water quality, it will either be discharged to the environment or sent to the water treatment plant. At closure, seepage through the pile will be directed to the Tulsequah River.

If a future water quality model shows that the discharge water quality would not meet regulatory standards, then adding a barrier, such as a geomembrane liner or tailings with a bentonite cap, would be necessary for the

closure cover system. The cost would be approximately C$500k to C$1M at closure or could be included as part of a progressive closure.

25.14.1.5 Environmental Studies, Permitting and Social or Community Impact

Potential risks may affect project schedule, costs, or to otherwise affect project outcomes, a non-exhaustive list of risks is provided below:

- Risk of arsenic and antimony leaching from tailings and waste rock, including circum-neutral leaching.
- Historical / legacy contamination and potential for further degradation if planned mitigative measures fail to adequately address existing and future impacts.
- Proximity to the Tulsequah Chief Mine, which still discharges acid mine drainage, raises cumulative impact concerns
- Complex permitting process with transboundary and multiple Indigenous governance considerations. The proposed site access is by barge from Alaska via the Taku River, therefore there are permitting considerations related to several United States regulatory requirements.
- Potential for unresolved concerns from the public, TRTFN, TTC, Alaskan Tribes, or others may results in opposition and delay or halting of the project.
- Availability of suitable construction materials for mining infrastructure and environmental management infrastructure (e.g., CSF cover system).

25.14.1.6 Project Economics

Imposition or fluctuation of Canadian tariffs on Chinese-manufactured equipment and materials may increase procurement costs or cause sourcing issues.

The ability of mining companies to fund the advancement of their projects through exploration and development is influenced by commodity prices. Although a marketing study was done to validate the pricing, variations in the commodity prices may lead to reduced or elevated revenues compared to those projected in this study.

There is no off-take agreement in place and the ability to market a high-grade gold concentrate with high levels of arsenic may become more difficult depending on gold price and/ or increased import controls on arsenic concentrates that could negatively impact the revenues compared to those projected in this study.

No royalties have been included in the project economics.

Canagold is challenging the validity of the historic 15% net profit interest (that can be reduced to 10% with the issuance of 150,000 Canagold shares) on the project. The addition of royalties will impact the project economics negatively and increase the cost in the LOM.

25.14.1.7 Foreign Exchange

Unfavourable foreign exchange rates could increase the effective cost of overseas imports for equipment and materials.

25.14.2 Opportunities

25.14.2.1 Metallurgical Testwork

A flotation development program focused on producing a separate antimony concentrate will determine if saleable antimony concentrates can be produced. Producing a separate antimony concentrate will add value to the project and eliminate any antimony penalties from the processing of the gold sulphide concentrate.

25.14.2.2 Mining

A material opportunity exists within the mine plan to change the stope cross-cut access strategy between two parallel veins (e.g. FW to HW vein). By employing an "Attack-Ramp" style approach, such as has been designed in the mine plan from the level access to the first vein contact, a significant amount of development could be reduced. From preliminary analysis work completed, there could be roughly a 50% reduction in the Zone C and 30% reduction in the Zone Y stope cross-cut development.

A trade-off study is recommended on the economic viability of using predominantly the Longhole Stoping mining method. The scope of the analysis should encompass the logistics and costs of binder transport and storage on site, as well as additional costs associated with the construction and operation of a paste plant to enable the use of paste backfill. In narrower areas, waste rock could be a major component of backfill. The surface waste rock storage requirement for DAF versus the additional cost to process more diluted LHS ore and storage of the additional tailings should be assessed.

A trade-off study is recommended to quantify the potential benefit of using rockfill as a significant stope backfill constituent versus HF. The study should examine the benefits of not stockpiling as much waste rock as the developed mine plan indicates and offsetting that with additional tailings storage, as well estimating potential reductions in mining operating cost.

While the BEV assessment discussed in Section 16.5 did not result in BEVs being the preferred production mobile equipment (LHDs and trucks) for this study, Canagold should continue to evaluate this technology and its potential value to the New Polaris site. BEVs should be considered not just for production mobile equipment, but also for auxiliary equipment.

- **Inferred Resources**: Inferred resources are not included in the production schedule; however, a plan to infill drill specific areas could increase Measured and Indicated resources. This resource increase could improve project economics and mine life. Operational definition drilling will test Inferred resources as part of the production sequence. Additional Measured and Indicated resources would have a compounding positive effect in that the development required per ore tonne and vertical mining advance rate would be decreased.

- **Used Equipment:** Assess the opportunity to purchase good quality used equipment that meets design specifications.
- **Geotechnical:** Additional geotechnical information could revise the geotechnical model and increase the amount of longhole stoping in the mine plan and would reduce operating cost and potentially increase productivity.
- **Additional Infill Drilling:** Additional drilling would increase the confidence in the Mineral Resource and could lead to higher modeled grades. Bulk sampling and test stoping at New Polaris could have a significant positive reconciliation of gold grade against the resource model.
- **Battery Electric Vehicles:** Continue to evaluate BEV technology and its potential value to the New Polaris site. BEVs should be considered not just for production mobile equipment, but also for auxiliary equipment.

25.14.2.3 Recovery Methods

Further variability testwork may provide the opportunity to reduce comminution energy requirements, and/ or reduce equipment sizing and costs.

25.14.2.4 Infrastructure

25.14.2.4.1 Co-storage Facility

The CSF has growth potential if more resources are found in future exploration efforts.

25.14.2.4.2 Micro-Hydro

There is potential for micro-hydro power to supplement or replace diesel-generated electricity at the site, providing a reliable and renewable energy source. The continuation of micro-hydro studies into the next project phase presents an opportunity to reduce long-term operating costs, lower greenhouse gas emissions, and improve environmental sustainability. Continued study in the next project phase may also support long-term energy security and help mitigate risks associated with fuel pricing and supply disruptions.

25.14.2.5 Environmental Studies, Permitting and Social or Community Impact

Opportunities exist to advance the project schedule, reduce costs, or to otherwise improve project outcomes, a brief list of potential opportunities is provided below:

- Maintain strong Indigenous relationships. Agreements with TRTFN provide a framework for collaboration and shared benefits; advancing existing relationships presents opportunity to increase support for the project. Expediting the development of a formal process with TRTFN for review and input on engineering designs, especially for infrastructure near culturally sensitive areas during design development. A formal and comprehensive design co-review may improve existing relationship between Canagold and TRTFN and may reduce subsequent review and permitting timeframes. Continue to engage as needed with other Indigenous nations.
- Continue to look for opportunities to reduce emissions and energy use specific to the Project.

- Water treatment measures including passive treatment for long-term management of effluent may present opportunity to reduce cost and risks associated with ML/ARD.
- Regarding hydrological, hydrogeological, and geochemical studies, there are opportunities to work closely and collaborate with the geotechnical, water resources, and mineralized material processing engineering teams and hence, reduce effort and costs.
- Positioning the project as a responsible re-development of a legacy site (with modern environmental safeguards) may provide opportunity to improve public and regulatory support. Detailed early planning for legacy remediation and mine closure activities may provide opportunity to obtain additional support for the project.
- Early and sustained engagement with Indigenous communities and regional stakeholders on training, employment, and business opportunities may foster goodwill and improve perception of the Project.
- The inclusion of detailed designs supported by management and monitoring plans (e.g., air, water, wildlife, and cultural heritage monitoring and management plans) may be used to demonstrate preparedness to alleviate concerns from the public, TRTFN, TTC, Alaskan Tribes, and regulators.
- Early engagement with provincial and federal regulators regarding species at risk identified at or near the Project site will optimize relationships with regulators and ensure that their views and recommendations are included in Project plans. For example, current plans for bat hibernacula include plugging all openings to surface; design and construction of portal and adit closure infrastructure to provide bat hibernacula post-closure presents opportunities to work closely with regulators to support biodiversity and reduce risks of impacting bat species in the vicinity of the Project.

26 RECOMMENDATIONS

26.1 Introduction

The following recommendations are suggested to further derisk the Project by progressing key areas such as exploration, mining, hydrogeology, infrastructure and environmental and permitting considerations. Canagold intends to make a decision on FEED design following completion of the Feasibility Study, in line with permitting timeline. This phase will further define the Project, confirm costing, and serve as the starting point for the Engineering, Procurement and Construction Management (EPCM) process. EPCM activities will begin when the Company makes a construction decision, which will follow receipt of permits and completion of the financing process. Complete a flotation test program focused on producing a separate antimony concentrate to determine if saleable antimony concentrates can be produced. Producing a separate antimony concentrate will add significant value to the project and eliminate any antimony penalties from the processing of the gold sulphide concentrate.

Initiate field programs and engineering evaluation for micro-hydro power to supplement or replace diesel-generated electricity at the site, providing a reliable and renewable energy source.

Table 26-1 below summarizes the recommended work to be completed after the Feasibility Study and prior to a construction decision.

Table 26-1: Phase 1 Recommended Work Program

Program Component	Estimated Total Cost (C$M)
Metallurgical Testwork	0.8
Mine Engineering	0.2
Backfill	0.9
CSF	0.8
Hydro Power Feasibility & Field Investigations	6.7
Environmental Studies	0.4
Total	**9.8**

26.2 Metallurgical Testwork

Testing of samples with arsenic, sulphur and antimony grades aligned with grades in the most recent mine should be completed to refine recovery models. This may result in lower concentrate mass recovery and lower concentrate arsenic grades. The samples should be selected based on the latest mine plan focusing on the first five years of production.

Materials handling tests on suitable feed samples should be completed to engineering design of the crushing circuit and crushed ore storage areas.

A standalone program to develop a flowsheet to produce saleable antimony concentrates as described in Section 25.13.1.1 must be completed. A total estimated costs for these activities are $750,000.

26.3 Assay Sampling and QA/QC

Independent reproducibility QA/QC in the form of sets of duplicates is only partial, and a review of the lab-internal coarse and pulp duplicates is recommended. For future drilling campaigns, the QP recommends adding a low-grade gold standard in the 1-g/t gold range to the medium and high-grade standards previously inserted to represent the expected gold concentrations more comprehensively.

Antimony was added to the resource estimation retroactively, which is why none of the inserted standards are certified for antimony. At New Polaris, antimony is seen as a by-product only, but MMTS recommends replacing at least one current standard with a standard that is certified for both gold and antimony to be able to independently control for accurate results.

Furthermore, the consistent insertion of a coarse blank is advised. MMTS understands that in the 2024 drilling campaign some blind blank insertions were of unmineralized rock in addition to CDN-BL-10.

Between 5 and 10% of samples should be check-assayed by a secondary laboratory, requesting comparable analytical methods, especially if a lab change occurred.

26.4 Underground Mining

Below is a summary of the recommended work program for advancing the New Polaris Project. Details of each component can be found in the following sub-sections.

Table 26-2: Recommended Work Program – Underground Mining

Discipline	Component	Estimated Total Cost (C$)
Mine engineering	Resue split face test mining for narrow drift and fill areas	150,000
Backfill	Review backfill program in support of longhole stoping	850,000

26.4.1 Mine Engineering

The following further works are recommended:

- Assessment of the economic viability of using resue mining to control dilution. Drilling and blasting the waste and ore separately to control excess dilution on narrower veins.

26.4.2 Geotechnical Engineering

Recommendations for future work include:

- Conduct additional geotechnical drilling and logging in areas with poor drillhole coverage such as the upper sections of No. 3 and No. 6 veins to confirm rock mass conditions ahead of mining.
- Collect structural orientation data (HW, Orebody, and FW) from future drilling programs using oriented core and/or downhole televiewer and underground mapping to confirm dominant discontinuity sets and their variability across domains. Conduct supplemental laboratory strength testing on core on samples of the major rock units, particularly the Felsite Dike and mineralized vein rock. Supplemental testing data would also help in further definition of intact rock strength envelopes for numerical modelling, as required in future.
- Standardize lithology logging to allow integration of recent and historic logging programs to serve as the basis for 3D geologic model development.
- Update the geotechnical models with newly collected data.
- Update all geotechnical assessments, ground support designs, and reassess stability and ground support with new information, including updated hydrogeology data, geotechnical data including stress measurement, hydrogeology assessments, further geotechnical drilling, logging, and laboratory testing for material properties.

Develop a Ground Control Management Plan to include all geotechnical conditions and ground support designs for each development profile.

26.4.3 Backfill

For backfill, complete a full review of paste backfill considerations using different binders and ad-mixes (plasticizers) to support LHS.

26.4.4 Underground Infrastructure

For dewatering, further work in obtaining a more accurate water inflow rate into the mine is recommended. Through a better understanding of water inflow rates, the requirements for an appropriate operational dewatering system can be better defined, along with more accurate operating and capital cost estimates.

For infrastructure, the next phase should involve basic and then detailed design of identified systems.

For ventilation, air flow requirements have been established. The next phase requires discussions with fan and heater suppliers for pricing and deliveries.

26.5 Project Infrastructure

26.5.1 Co-storage Facility

26.5.1.1 Geotechnical Investigations

A limited geotechnical field and laboratory investigation was conducted in 2023 for the FS due to some technical issues with the drilling in the CSF footprint. Therefore, an additional geotechnical field and laboratory program needs to be completed to determine whether the foundation conditions (especially at depths greater than 3 mbgs investigated only by the test pitting program in 2023) are similar or how they might potentially affect the CSF and its access road. This could potentially require modifications to the design that might increase initial capital, sustaining capital, and operating costs.

26.5.1.2 Design

Following activities are recommended to bring the design and analysis of the CSF to a FEED level of detail:

- Conduct a site-specific seismic hazard assessment and update the seismic parameters used in the stability analysis and design of the CSF accordingly.
- Review and update the water balance and surface water and sediment management including the meteorological and hydrology information for the CSF.
- Update geotechnical parameters for tailings, waste rock, construction material, and foundation conditions by conducting additional laboratory testing.
- Review and update the stability model according to the above updated information.
- Perform a liquefaction assessment with consideration of updated information on material properties for the tailings along with foundation for the CSF and LRTF.
- Review and update the seepage model and seepage control measures for the CSF along with degree of saturation during stacking operations and its effect on the stability of the CSF.
- Optimize the tailings and waste rock handling and deposition strategy, including material handling equipment (dozer and compactors for the CSF).
- Solicit additional budgetary quotes for earthworks and geosynthetics to get more accurate pricing for the next cost estimates.
- Develop FEED level design of the CSF.
- Develop cost estimates (i.e., capital, sustaining capital, and operating costs) for earthworks, construction of the CSF and associated seepage and surface water management structure.

The estimated cost for the recommended work is C$770,000.

26.5.2 Hydro Power Feasibility & Field Investigations

Initiate field programs and engineering evaluation for micro-hydro power to supplement or replace diesel generated electricity at the site, providing a reliable and renewable energy source.

The estimated cost for the recommended work is C$6,700,000.

26.6 Environmental Studies, Permitting and Social Considerations

The following recommendations are made regarding future studies and activities related to areas of environment, permitting and community engagement. These studies and activities are recommended to support the project through the detailed design, EA preparation, and permitting stages and will reduce associated risks:

- Conduct supplemental geochemical characterization testing including static and kinetic testing (e.g., ABA, SFE, HCT) on fresh drill core samples from development areas including:
  - Overburden and borrow materials
  - Cemented paste backfill
  - Construction rock
  - Water treatment sludge
- Develop geochemical source terms for water quality modelling.
- Conduct construction material surveys of potential borrow sources for construction and CSF cover materials. Assess physical properties (grain size, permeability, compaction) and chemical stability (ARD/ML potential).
- Further refinement of the hydrological and hydrogeological models by means of focused monitoring and testing programs in areas requiring further refinement for water balance and effluent predictions.
- Develop detailed plans for addressing bat roosts which exist on site and may impede permitting and construction. Identify and develop potential alternative bat roost locations which will support continuity of bat population and allow for demolition of the existing workshop building, and relocation of portal roost outside of the roosting season. It is recommended to complete planning and relocation to support permit approvals.

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