NI 43-101 Technical Report on the Donlin Gold Project, Alaska, USA

Prepared for: NOVAGOLD RESOURCES INC.

Prepared by: Edwin Peralta, PE, Technical Director Mining & Geology, Wood Group USA, Inc.
Henry Kim, P.Geo, Principal Resource Geologist, Wood Canada Limited.
Alan Drake, P.L.Eng, Principal Metallurgist, Wood Canada Limited
Jennifer Pretorius, P.Geo, Senior Hydrogeologist, Geosyntec Consultants, International, Inc.
Paul Baluch, P.Eng, Technical Director, Civil/Structural/Architectural, Wood Canada Limited Paul
Dockweiler, P.Geo, Principal Geologist, Geosyntec Consultants International, Inc.
Rick Sisson, PE, Principal Geotechnical Engineer, Geosyntec Consultants International, Inc.

Effective date: 30 November 2025

Important Notice

This Report was prepared as a National Instrument 43-101 technical report for NOVAGOLD RESOURCES INC. (NOVAGOLD) by Wood USA Group Inc. (Wood) and Geosyntec Consultants International, Inc. (Geosyntec) (collectively the Consultants). The quality of information, conclusions, and estimates contained herein is consistent with the terms of reference, constraints and circumstances under which the Report was prepared by the Consultants and are 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 to be used by NOVAGOLD subject to terms and conditions of its contract with each of the Consultants. That contract permits NOVAGOLD to file this Report as a technical report with Canadian securities regulatory authorities pursuant to provincial and territorial securities law. Except for the purposes legislated under Canadian provincial and territorial securities law, any other use of this Report by any third party is at that party's sole risk.

CERTIFICATE OF QUALIFIED PERSON

Edwin Peralta, P.E.
Wood Group USA Inc
17325 Park Row
Houston, TX 77084

I, Edwin Peralta, P.E. am employed as Technical Director of Mining and Geology with Wood Group USA Inc.

This certificate applies to the technical report entitled "NI 43-101 Technical Report on the Donlin Gold Project, Alaska, USA" with an effective date of November 30, 2025 (the "Technical Report").

I am registered as a Professional Engineer in the state of Nevada, as a Professional Engineer in the state of Alaska, and am a Registered Member of the Society for Mining, metallurgy and Exploration (RM SME, #04033387). I graduated with a B.S. degree from the Colorado School of Mines in 1995, and with an M.S. degree from the same university in 2000.

I have practiced my profession for 29 years. I have experience in Mineral Reserve estimation, mine planning and design for open pit and underground mining operations, and scoping, prefeasibility, and feasibility studies.

As a result of my experience and qualifications, I am a Qualified Person as defined in National Instrument 43–101 *Standards of Disclosure for Mineral Projects* (NI 43–101).

I visited the Donlin property from September 8, 2025 to September 9, 2025

I am responsible for Sections 1.1-1.3, 1.12, 1.15, 1.16, 1.19, 1.21-1.24, 1.26, 2, 3, 12.3, 12.7.2, 15.1.1-15.1.4, 15.2-15.4, 16.1, 16.3-16.7, 16.9-16.11, 18.5.1, 19, 21.1.1, 21.1.1.1-21.1.1.3, 21.1.1.10-21.1.1.12, 21.1.2, 21.2.1-21.2.3, 21.2.5, 21.2.6, 22, 24, 25.1, 25.7, 25.8, 25.14, 25.16, 25.17, 26.1, 26.3, 26.10 and 27 of the Technical Report.

I am independent of NOVAGOLD RESOURCES INC. as independence is described by Section 1.5 of NI 43–101.

I have had no previous involvement with the Donlin Gold Project.

I have read NI 43–101, and sections of the Technical Report that I am responsible for have been prepared in compliance with that Instrument.

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

"*signed and sealed*"

Edwin Peralta, P.E.

Dated: January 22, 2026

CERTIFICATE OF QUALIFIED PERSON

Henry Kim, P.Geo., CPG
Wood Canada Limited
111 Dunsmuir St, Suite 400
Vancouver, British Columbia, Canada, V6B 5W3

I, Henry Kim, P.Geo., CPG, am employed as Principal Resource Geologist with Wood Canada Limited.

This certificate applies to the technical report entitled "NI 43-101 Technical Report on the Donlin Gold Project, Alaska, USA" with an effective date of November 30, 2025 (the "Technical Report").

I am registered as a Professional Geoscientist with Engineer & Geoscientists of British Columbia (P.Geo.) since 2015, as a Certified Professional Geologist (CPG) with the American Institute of Professional Geologists, and as a Certified Professional Geologist (CPG) in the state of Alaska. I graduated from the University of British Columbia in 2008 with a B.Sc. in geology. I completed the Applied Geostatistics Citation Program with the University of Alberta in 2014, and the Specialized Training Cycle in Geostatistics (CFSG) program with the MINES ParisTech and Geovariances in 2022.

I have practiced my profession for 18 years. I have been involved in exploration drilling programs involving core logging, sampling, QAQC, and database validation. I have conducted onsite grade control and management of mine operation crews for an open-pit base metal mine in eastern Canada. I have conducted audits and due diligence exercises on geological models, sampling databases, drill hole spacing studies, conditional simulations, preparation of resource models, validation of mineral resource estimates, and mineral resource estimates on advanced mining studies and active mine operations.

As a result of my experience and qualifications, I am a Qualified Person as defined in National Instrument 43–101 *Standards of Disclosure for Mineral Projects* (NI 43–101).

I visited the Donlin property most recently from August 18, 2025 to August 22, 2025.

I am responsible for Sections 1.1, 1.2, 1.4-1.12, 1.14, 1.24, 1.25.1, 1.25.2, 1.26, 2, 3.1, 4-11, 12.1, 12.2, 12.7.1, 14, 23, 25.1, 25.2, 25.3, 25.4, 25.6, 25.18.1, 25.18.2, 26.1, 26.2, 26.10, and 27 of the Technical Report.

I am independent of NOVAGOLD RESOURCES INC. as independence is described by Section 1.5 of NI 43–101.

I have been involved with the Donlin Project since 2020 and was responsible for the mineral resource estimate dated June 1, 2021.

I have read NI 43–101, and sections of the Technical Report that I am responsible for have been prepared in compliance with that Instrument.

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

"*signed and sealed*"

Henry Kim, P.Geo., CPG

Dated: January 22, 2026

CERTIFICATE OF QUALIFIED PERSON

Alan Drake, P.L.Eng.
Wood Canada Limited
111 Dunsmuir St, Suite 400
Vancouver, British Columbia, Canada, V6B 5W3

I, Alan Drake, P.L.Eng., am employed as a Principal Metallurgist with Wood Canada Limited.

This certificate applies to the technical report entitled "NI 43-101 Technical report on the Donlin Gold Project, Alaska, USA" with an effective date of November 30, 2025 (the "Technical Report").

I am a Professional Licensee Engineering with Engineers & Geoscientists of British Columbia. I graduated from the Technicon Witwatersrand with a National Higher Diploma in Extraction Metallurgy in 1993.

I have practiced my profession for 33 years. I have been directly involved in metallurgical plant operations, process design, construction and commissioning of minerals processing and hydrometallurgical facilities for base and precious metals.

As a result of my experience and qualifications, I am a Qualified Person as defined in National Instrument 43–101 *Standards of Disclosure for Mineral Projects* (NI 43–101).

I have not visited the Donlin property.

I am responsible for Sections 1.1, 1.2, 1.12, 1.13, 1.17, 1.21, 1.22, 1.24, 1.25.1, 1.25.6, 2, 3.1, 12.4, 12.7.3, 13, 17, 21.1.1, 21.1.1.1, 21.1.1.2, 21.1.1.5, 21.1.1.10, 21.1.1.11, 21.1.1.12, 21.2.1, 21.2.2, 21.2.4, 25.1, 25.5, 25.10, 25.16, 25.18.1, 25.18.6 and 27 of the Technical Report.

I am independent of NOVAGOLD RESOURCES INC. as independence is described by Section 1.5 of NI 43–101.

I have had no previous involvement with the Donlin Gold Project.

I have read NI 43–101, and sections of the Technical Report that I am responsible for have been prepared in compliance with that Instrument.

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

"*signed and sealed*"

Alan Drake, P.L.Eng.

EGBC Permit to Practice: 1000679

Dated: January 22, 2026

CERTIFICATE OF QUALIFIED PERSON

Jennifer Pretorius, P.Geo.
Geosyntec Consultants International Inc
111 Research Drive, Suite 218
Saskatoon, Saskatchewan, Canada, S7N 3R2

I, Jennifer Pretorius, P.Geo., am employed as a Senior Hydrogeologist with Geosyntec Consultants.

This certificate applies to the technical report entitled "NI 43-101 Technical Report on the Donlin Gold Project, Alaska, USA" with an effective date of November 30, 2025 (the "Technical Report").

I am a Professional Geoscientist with Association of Professional Engineers and Geoscientists of Saskatchewan, Association of Professional Engineers and Geoscientists of Alberta, and Association of Professional Engineers and Geoscientists of British Columbia. I was previously registered with the South African Council for Natural Scientific Professions (SACNASP) for 11 years from 2008 to 2019. I graduated from the University of Johannesburg, South Africa with a B.Sc. in Geology and Chemistry in 1994, a B.Sc. Honours in Geohydrogeology/Hydrology from University of the Free State, South Africa in 1997, a M.Sc. in Geohydrogeology from University of the Free State, South Africa in 2000, and a Ph.D. in Geohydrogeology from University of the Free State, South Africa in 2007.

I have practiced my profession for 26 years. I have been directly involved in hydrogeological investigations for the mining sector, focused on tailings seepage management, mine dewatering, and groundwater quality monitoring. I have contributed to pre-feasibility, feasibility, active operations, and closure planning.

As a result of my experience and qualifications, I am a Qualified Person as defined in National Instrument 43–101 *Standards of Disclosure for Mineral Projects* (NI 43–101).

I visited the Donlin property from August 18, 2025 to August 22, 2025.

I am responsible for Sections 1.1, 1.2, 1.12, 1.18, 1.21, 1.24, 1.25.1, 1.25.8. 1.26, 2, 3.1, 12.5.3, 12.7.6, 16.2.2.4, 16.8, 18.5.2, 18.7, 21.1.1, 21.1.1.1, 21.1.1.2, 21.1.1.3, 21.1.1.10, 21.1.1.11, 21.1.1.12, 25.1, 25.12, 25.16, 25.18.1, 25.18.8, 26.1, 26.8, 26.10 and 27 of the Technical Report.

I am independent of NOVAGOLD RESOURCES INC. as independence is described by Section 1.5 of NI 43–101.

I have had no previous involvement with the Donlin Gold Project.

I have read NI 43–101, and sections of the Technical Report that I am responsible for have been prepared in compliance with that Instrument.

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

"*signed and sealed*"

Jennifer Pretorius, P.Geo.

Dated: January 22, 2026

CERTIFICATE OF QUALIFIED PERSON

Paul Baluch, P.Eng., PE
Wood Canada Limited
111 Dunsmuir St, Suite 400
Vancouver, British Columbia, Canada, V6B 5W3

I, Paul Baluch, P.Eng., PE, am employed as Technical Director, Civil/Structural/Architectural with Wood Canada Limited.

This certificate applies to the technical report entitled "NI 43-101 Technical Report on the Donlin Gold Project, Alaska, USA" with an effective date of November 30, 2025 (the "Technical Report").

I am a member of Engineers & Geoscientists of British Columbia, Association of Professional Engineers and Geoscientists of Alberta, Association of Professional Engineers and Geoscientists of Saskatchewan, Professional Engineers Ontario and a member of Idaho Board of Professional Engineers and Professional Land Surveyors. I graduated from the Slovak Technical University in Bratislava, Slovakia with a Diploma in Civil Engineering in 1980.

I have practiced my profession for 43 years. I have been directly involved in site investigations, site development, infrastructure and civil works scoping studies, prefeasibility and feasibility studies and detailed engineering on mining, infrastructure, and other industry projects.

As a result of my experience and qualifications, I am a Qualified Person as defined in National Instrument 43–101 *Standards of Disclosure for Mineral Projects* (NI 43–101).

I visited the Donlin property from September 8, 2025 to September 9, 2025.

I am responsible for Sections 1.1, 1.2, 1.12, 1.18, 1.21, 1.24, 1.25.1, 1.25.7, 1.26, 2, 3.1, 12.5.1, 12.7.4, 18.1-18.4, 18.8-18.10, 18.11.1, 18.12, 21.1.1, 21.1.1.1, 21.1.1.2, 21.1.1.4, 21.1.1.7-21.1.1.12, 25.1, 25.11, 25.16, 25.18.1, 25.18.7, 26.1, 26.7, 26.10 and 27 of the Technical Report.

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I have had no previous involvement with the Donlin Gold Project.

I have read NI 43–101, and sections of the Technical Report that I am responsible for have been prepared in compliance with that Instrument.

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

"*signed and sealed*"

Paul Baluch, P.Eng., PE

EGBC Permit to Practice: 1000679

Dated: January 22, 2026

CERTIFICATE OF QUALIFIED PERSON

Paul Dockweiler, CPG, CEM
Geosyntec Consultants
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I, Paul Dockweiler, CPG, CEM, am employed as a Senior Geologist with Geosyntec Consultants.

This certificate applies to the technical report entitled "NI 43-101 Technical report on the Donlin Gold Project, Alaska, USA" with an effective date of November 30, 2025 (the "Technical Report").

I am a Certified Professional Geologist with American Institute of Professional Geologists and a Certified Environmental Manager with the state of Nevada. I graduated with a B.S. in Geological Science from Michigan State University in 2001.

I have practiced my profession for 25 years. I have been directly involved in mineral exploration, resource development, permitting and environmental management within the mining industry. I have been involved in GISTM standardization of tailings facilities, groundwater injection studies, geotechnical investigations of highwalls, tunnel and long linear civil works projects.

As a result of my experience and qualifications, I am a Qualified Person as defined in National Instrument 43–101 *Standards of Disclosure for Mineral Projects* (NI 43–101).

I visited the Donlin property from August 18, 2025, to August 22, 2025.

I am responsible for Sections 1.1, 1.2, 1.12, 1.20, 1.24, 1.25.1, 1.25.9, 1.26, 2, 3.1, 12.6, 12.7.7, 20, 25.1, 25.15, 25.18.1, 25.18.9, 26.1, 26.9, 26.10 and 27 of the Technical Report.

I am independent of NOVAGOLD RESOURCES INC. as independence is described by Section 1.5 of NI 43–101.

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I have read NI 43–101, and sections of the Technical Report that I am responsible for have been prepared in compliance with that Instrument.

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

"*signed and sealed*"

Paul Dockweiler, CPG, CEM

Dated: January 22, 2026

Geosyntec consultants

CERTIFICATE OF QUALIFIED PERSON

Richard Sisson, PE, P.Eng., Ph.D.
Geosyntec Consultants
803 2nd St
Davis, CA 95616

I, Richard Sisson, PE, P.Eng., Ph.D., am employed as a Senior Consultant with Geosyntec Consultants.

This certificate applies to the technical report entitled "NI 43-101 Technical report on the Donlin Gold Project, Alaska, USA" with an effective date of November 30, 2025 (the "Technical Report").

I am a Professional Engineer with Association of Professional Engineers and Geoscientists of British Columbia and the Association of Professional Engineers of Alberta and a Professional Engineer with the states of California and Montana. I graduated with a B.Sc. in Civil Engineering with high honors from Michigan State University in 1979, a M.Eng. in Geotechnical Engineering from the University of California in 1982, and a Ph.D. in Geotechnical Engineering from the University of California in 1990.

I have practiced my profession for 38 years. I have been directly involved in engineering projects for tailings and water retention structures, dam safety reviews, slope analyses and design and foundation designs. I serve on independent tailings review boards to provide guidance and review for the design of tailings dams and waste rock facilities for mining operations from conceptual design through closure. I have conducted or managed investigation analyses of slope stability for design of pit slopes in soil and soft rock, highways slopes, and levees.

As a result of my experience and qualifications, I am a Qualified Person as defined in National Instrument 43–101 *Standards of Disclosure for Mineral Projects* (NI 43–101).

I visited the Donlin property from September 8, 2025 to September 9, 2025.

I am responsible for Sections 1.1, 1.2, 1.12, 1.18, 1.21, 1.24, 1.25.1, 1.25.3-1.25.5, 1.26, 2, 3.1, 12.5.2, 12.7.5, 15.1.5, 16.2.1, 16.2.2.1-16.2.2.3, 16.2.2.5, 16.2.2.6, 18.5.3, 18.6, 18.11.2, 21.1.1, 21.1.1.1, 21.1.1.2, 21.1.1.6, 21.1.1.10-21.1.1.12, 21.1.2, 25.1, 25.9, 25.13, 25.16, 25.18.1, 25.18.3, 25.18.4, 25.18.5, 26.1, 26.4, 26.5, 26.6, 26.10 and 27 of the Technical Report.

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I have read NI 43–101, and the sections of the Technical Report that I am responsible for have been prepared in compliance with that Instrument.

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

"*signed and sealed*"

Richard Sisson, PE, P.Eng., Ph.D.

Dated: January 22, 2026

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1.0 SUMMARY

1.1 Introduction

NOVAGOLD RESOURCES INC. (NOVAGOLD) requested Wood Group USA Inc. (Wood) and Geosyntec Consultants International, Inc. (Geosyntec) to update the National Instrument (NI) 43-101 Technical Report on Donlin Gold Project, Alaska, USA, with an effective date of 1 June 2021 (Donlin 2021 Technical Report) so that it is current to 30 November 2025. Work done since 2021 on the property with respect to exploration, drilling, permitting and minor project design changes, as a result of recent permitting activities, are summarized in this 30 November 2025 technical report (Report). Updated content includes all new drill data, analytical data, Mineral Resource estimate, Mineral Reserve estimate, pit design, production schedule, permitting, operating costs, capital costs, taxes, long term gold price and the economic analysis on the Donlin Gold project (Project). A data verification exercise was completed by each qualified person (QP) co-authoring the Report.

1.2 Terms of Reference

On 3 June 2025, Barrick Gold Corporation (Barrick) divested their 50% ownership in Donlin Gold LLC. Through their wholly-owned subsidiary, NovaGold Resources Alaska Inc., NOVAGOLD increased their ownership interest in Donlin Gold LLC to 60%. The remaining 40% ownership interest in the Donlin Gold Project is Donlin Gold Holdings LLC, a subsidiary of Paulson Advantage Plus Master Ltd. and Paulson Advisors LLC. (together "Paulson").

Wood and Geosyntec have used information completed for earlier studies, as well as more recent data and information available on the Project to support and make current the information in this Report. QP authors conducted due diligence reviews on the information supplied by Donlin Gold LLC and made adjustments to the results of the 2021 feasibility study (2021 FS) update based on the outcome of those reviews.

Mineral Resource estimates were completed in accordance with the Canadian Institute of Mining, Metallurgy and Petroleum (CIM) Estimation of Mineral Resources and Mineral Reserves Best Practice Guidelines, November 29, 2019 (CIM MRMR Best Practice Guidelines) and reported in accordance with the CIM Definition Standards for Mineral Resources and Mineral Reserves, May 10, 2014 (CIM Definition Standards).

The Report uses American English. Unless otherwise specified in the text, monetary amounts are in U.S. dollars and units are metric except where noted.

1.3 Principal Outcomes

The principal outcomes of this study are outlined in Table 1-1.

Table 1-1: Key Project Outcomes

Item	Unit	Value
Total Mined	Mt	3,803
Ore Treated	Mt	504.8
Life of Project[1]	years	27
Strip Ratio	W/O	6.5
Total Gold Recovered	Moz	29.5
Gold Recovery	%	90.0
Gold Payable	%	99.9
Gold Price (Cash Flow)	\$/oz	2,100
Total Before Tax Cash Flow	\$M	25,415
Total Before Tax $NPV_{5\%}$	\$M	7,516
Before Tax IRR	%	12.5
Before Tax Payback Period	years	4.9
Total After Tax Cash Flow	\$M	19,614
Total After Tax $NPV_{5\%}$	\$M	5,058
After Tax IRR	%	10.3
After Tax Payback Period	years	6.5
Gross Revenue	\$M	61,952
Selling Costs	\$M	51
Operating Costs (Including Royalties)	\$M	24,504
Initial Capital	\$M	9,233
Sustaining Capital	\$M	2,325
Total LOM Capital	\$M	11,558
Closure Costs	\$M	423
Taxes	\$M	5,801

Note: (1) Includes 24 years of mining and three additional years of stockpile reclaim

1.4 Location, Climate, and Access

The Donlin deposits are situated approximately 450 km west of Anchorage and 250 km northeast of Bethel up the Kuskokwim River. The closest village is the community of Crooked Creek, approximately 20 km to the south, on the Kuskokwim River (Figure 1-1). There is no road or rail access to the site. All access to the Project site for personnel and supplies is by air.

Access to Bethel and Aniak, the regional centers, is limited to river travel by boat or barge in the summer and air travel year-round. The Kuskokwim River is a regional transportation route and is serviced by commercial barge lines.

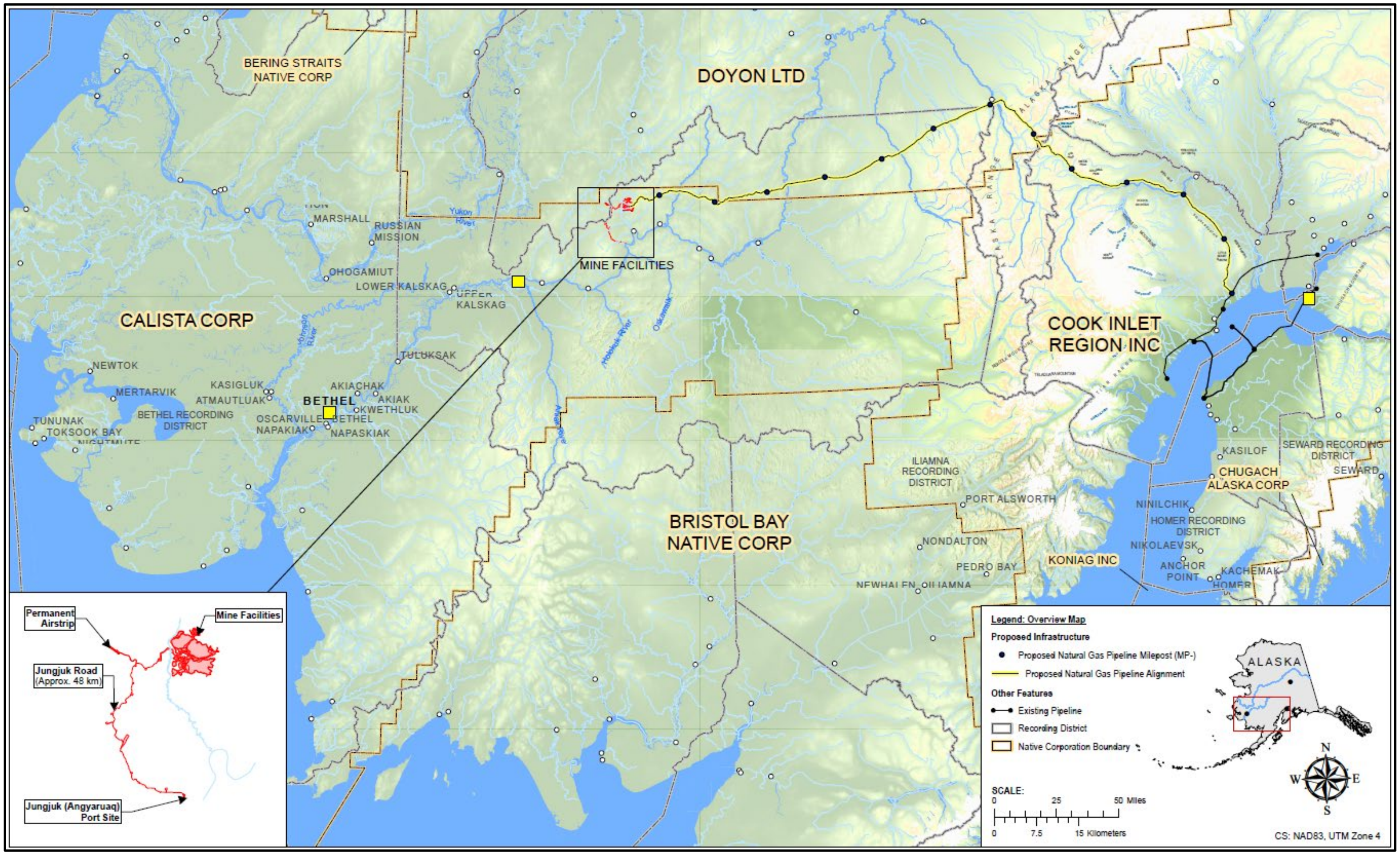
The area has a relatively dry interior continental climate with typically about 500 mm of total annual precipitation. Permafrost is sporadic, typically confined to valley bottom and mid-slope, with thickness ranging from 1.5 m to over 15 m (average ~4 m). The site is in a seismically active region of Alaska, influenced by the collision of the Pacific and North American plates.

1.5 Mineral Tenure and Surface Rights

The Calista Lease (which runs through April 2031, with provisions to extend beyond that time) currently includes a total of 72 complete sections in the vicinity of the deposits. Additional partial sections associated with the Project infrastructure are leased from Calista Corporation (Calista), an Alaska Native Corporation that holds the subsurface (mineral) estate for Native owned lands in the region. In addition to the approximately 19,988 ha leased from Calista, Donlin Gold LLC holds 493 Alaska State mining claims comprising approximately 29,008 ha, on State and State-selected lands. These are located in the vicinity of the leased lands in the Kuskokwim and Mount McKinley recording districts, bringing the total mineral land package to approximately 48,996 ha (Property).

A separate Surface Use Agreement (SUA) with The Kuskokwim Corporation (TKC), an Alaska Native Village Corporation that owns the majority of the private surface estate in the area, grants non-exclusive surface use rights to Donlin Gold LLC on at least 64 sections. These overly or are in the vicinity of the mineral deposit, with provisions allowing for adjusting that area in conjunction with adjustments to the subsurface included in the Calista Lease. The SUA was revised and restated and executed by Donlin Gold LLC and TKC to expand the TKC surface lands included in the SUA and update other provisions, effective 6 June 2014.

Figure 1-1: Location Map



Source: NOVAGOLD, 2025

The term of the SUA runs through 30 April 2031, with provisions to continue beyond that time so long as the Calista Lease remains in effect. Calista and Donlin Gold LLC executed an agreement making limited further amendments to the Calista Lease, effective 6 June 2014, in conjunction with the revision of the SUA between Donlin Gold LLC and TKC. The SUA provides Donlin Gold LLC with surface rights to approximately 16,923 ha of TKC-owned land.

A small surface estate of 5.7 ha to which the Lyman family has title, is within the Snow Gulch area lying immediately to the north of the Project's pit shell. Lyman Resources in Alaska, Inc. (Lyman Resources), the Lyman family and Donlin Gold LLC executed a Surface Lease and Assignment of Mining Lease assigning the Lyman placer lease, located within the Calista Lease area, to Donlin Gold LLC for mining use.

1.6 Royalties

The terms for the Calista Lease and TKC SUA include various royalty and other payment provisions and considerations such as shareholder employment and contracting opportunities.

Royalty terms of the Calista Lease include:

- Annual advance minimum royalty (variable) to 2030
- All advance minimum payments are recoverable as a credit against the net smelter return (NSR) royalty and net proceeds payment
- NSR of 1.5% for the earlier of the first five years following commencement of commercial production or until initial capital payback
- Conversion to 4.5% NSR after the earlier of five years or initial capital payback
- Net proceeds royalty of 8% of the net proceeds realized by Donlin Gold LLC commencing with the first quarter in which net proceeds are first realized.

Payment terms of the TKC SUA include:

- Annual advance minimum payment (variable per milestones)
- All advance minimum payments are recoverable as a credit against the milled tonnage fee and net proceeds payment
- Milled tonnage fee of $0.40/t processed for the first 10 years of production
- Conversion of the milled tonnage fee to $0.50/t processed for all production after 10 years
- Net proceeds payment of 3% of the net proceeds realized by Donlin Gold LLC commencing with the first quarter in which net proceeds are first realized.

There are currently no Government royalty obligations.

1.7 Geology and Mineralization

The Donlin mineralization model is a high-level, reduced intrusion-related vein system. The ACMA (named after the American Creek magnetic anomaly)-Lewis part of the district is a low sulfidation, reduced intrusion related, epizonal system with both vein and disseminated mineral zones.

The Donlin deposits lie in the central Kuskokwim basin of southwestern Alaska, which contains a back-arc continental margin basin fill assemblage of the Upper Cretaceous Kuskokwim Group, and Late Cretaceous volcano-plutonic complexes. The Project area is underlain by an 8.5 km long x 2.5 km wide granite porphyry dike and sill swarm hosted by lithic sandstone, siltstone, and shale of the Kuskokwim Group.

The deposits are hosted primarily in igneous rocks and are associated with an extensive Upper Cretaceous gold–arsenic–antimony–mercury hydrothermal system. The northeast, elongated, roughly 1.5 km wide x 3 km long cluster of gold deposits has an aggregate vertical range that exceeds 945 m. These areas consist of the ACMA and 400 Zone, Aurora and Akivik mineralized areas (grouped as ACMA) and the Lewis, South Lewis, Vortex, Rochelieu, Queen and North Akivik mineralized areas (grouped as Lewis).

Gold occurs primarily in sulfide and quartz–carbonate–sulfide vein networks in igneous rocks and, to a much lesser extent, in sedimentary rocks. Broad disseminated sulfide zones formed in igneous rocks where vein zones are closely spaced. Sub-microscopic gold, contained primarily in arsenopyrite and secondarily in pyrite and marcasite, is associated with illite–kaolinite–carbonate–graphite-altered host rocks.

1.8 Exploration

Placer gold was first discovered at Snow Gulch, a tributary of Donlin Creek, in 1909. Early-stage exploration in the modern era was performed by Resource Associates of Alaska (1974–1975), Western Gold Exploration and Mining Co. LP (WestGold) during 1988–1989 and Teck Exploration Ltd. (Teck) in 1993. Exploration included geological mapping, trenching, rock and soil sampling, an airborne magnetic and VLF survey, ground magnetic surveys, and initial Mineral Resource estimates.

The work completed on the Project has been undertaken by Placer Dome Inc. (1995 to 2000, and again from 2002 to 2005), NOVAGOLD (2001 to 2002), Barrick (2006) and by Donlin Gold LLC (from 2007 to date).

Activities have included construction of infrastructure to support exploration activities; reconnaissance and geological mapping; aerial photography; rock chip and soil sampling; trenching; max-min electro-magnetic (EM) geophysical surveys; airborne geophysical surveys; Reverse circulation (RC) and core drilling for resource infill, geotechnical, engineering, condemnation, waste rock, and metallurgical purposes; environmental baseline studies; community consultations; detailed metallurgical testwork; geotechnical and hydrogeological studies; sampling of prospective calcium carbonate source areas; exploration and auger drilling program for sand and gravel sources; a series of Mineral Resource and Mineral Reserve estimates; and initial mining and engineering studies. This work culminated in a feasibility study in 2007, and updates to this study in 2009, 2011, 2021, and this Report.

1.9 Exploration Potential

The Property retains exploration potential. The Akivik and East ACMA areas have good potential for lateral extensions of mineralization to the northwest and southeast of the pit footprint. In addition, known gold mineralization is likely to extend at depth, below the base of the designed pit, and in some areas immediately adjacent to the planned pit floor, which has been intersected by current drilling. Several drilled prospects and other exploration targets along the 6 km igneous trend north of the Mineral Resource area remain under-explored.

1.10 Drilling

A total of 2,145 core and RC holes of 516,779 m were completed from 1995 through 2025. Holes drilled by previous operators between 1995 and 2000 include 347 core holes (86,298 m) and 77 RC holes (16,338 m). Holes drilled by Donlin Gold LLC include 491 core holes (121,383 m) and more recently 10 dual rotary holes (630 m).

Core sizes used on the Property include: NQ3 (45.1 mm core diameter), NQ (47.6 mm), HQ3 (61.2 mm), HQ (63.5 mm), and PQ (85 mm).

Standard logging and sampling conventions were used to capture information from the drill core and, where applicable, RC chips. Data captured included lithology, mineralization, alteration (visual), structural, and geotechnical.

A survey of nearly 200,000 core recovery records in the database revealed an overall length-weighted average core recovery of 95%. Average recovery increases from 80-95% from 0-40 m and then ranges from 95-100% below 40 m where overburden and surface weathering effects are generally absent.

Traditional (transit) collar survey methods were used up to 2001 with modern global positioning system (GPS) technology introduced in 2002. Various down-hole survey methods and tools have been used over the years including Sperry Sun single-shot camera, Reflex EZ Shot, Boart Longyear TruShot, SPT GyroMaster survey and Index OMIx42 survey tools. Televiewer data was collected from 2020-2025.

The quantity and quality of the lithological, geotechnical, and collar and down-hole survey data collected in the exploration and delineation drill programs are sufficient to support Mineral Resource and Mineral Reserve estimation in the opinion of the QPs.

1.11 Sample Preparation, Analysis and Security

Core is digitally photographed and sawn in half with water-cooled diamond saw blades. Drill holes are sampled from the top of bedrock to the end of the hole. Prior to 2017, the maximum sample length in zones consisting of intrusive rocks or that contain appreciable sulfide/arsenic minerals was typically 2 m, whereas sample lengths in sedimentary rock zones that lack appreciable sulfide/arsenic minerals could be 3 m. A minimum of three additional 2 m sample intervals are placed before and after each intrusive rock or mineralized zone. From 2017 onward, samples were typically taken at maximum of 2 m intervals.

Specific gravity (SG) data measurements were collected in 2006, 2008-2010, 2017 and 2019, using the wax immersion, water displacement method. The weighted average of all SG data points was 2.65 for the intrusive units, and 2.71 for the sedimentary units.

The primary laboratory for most assaying up to 2020 has been ALS Global – Geochemistry Analytical Laboratory in Vancouver, BC (ALS). During the exploration programs, ALS held accreditations typical for the time, including, at various times, ISO9001:2000 and ISO 9002, and from 2005, ISO/IEC 17025 accreditations. Since 2020, the primary laboratory for gold analysis has been Bureau Veritas, Vancouver, British Columbia (BV).

Sample preparation procedures have varied slightly since 2005 with core either split (2020 onwards) and/or crushed onsite (2005-2010) before being shipped to an independent laboratory facility (2005-2010), or whole core shipped off-site for splitting and sample preparation (2006, 2017). Crushing requirements for sample preparation are typically 70% minus 2 mm, with ALS and BV subsequently further reducing the splits of crushed core to better than 85% passing minus 75 µm.

Approximately 30 g subsample of the pulp was assayed using fire assay-atomic absorption spectroscopy (AAS). Before 2007, the primary gold assay method was Au-AA23, which had an analytical range of 0.005-10 g/t Au. The Au-AA25 gold assay method was initiated in 2007 and had an analytical range of 0.01-100 g/t Au. Samples that exceeded the analytical limit for a given method were re-assayed by fire-assay and gravimetric finish or ore grade fire-assay AAS. In 2017, the program returned to using Au-AA23 as the based fire assay method using a 30 g subsample, with an overlimit (>10 g/t) trigger to a fire assay with a gravimetric finish.

ALS continues to analyze samples for sulfur content according to the CNL/Leco method. The Leco method was also used to analyze samples flagged for acid base accounting (ABA) for carbon content as well as to determine neutralization potential (NP) and acid potential (AP) according to the industry-standard ALS ABA procedure.

Most trace and major element data for drill holes located within the resource model boundary were acquired prior to the 2005 program by various laboratories using industry-standard acid digestions followed by atomic absorption (AA) or inductively coupled plasma (ICP) instrumental determinations. Subsequent multi-element trace analyzes were performed at ALS using aqua regia or four-acid digestions followed by ICP ± mass spectrometry with BV performing this analysis on all samples starting 2025.

Quality assurance and quality control (QA/QC) programs have been in place since 1995, and consist of the insertion of blank, standard reference material (SRM) and duplicate samples.

Sample security measures include moving of core from the drill site to the storage yard adjacent to the geology office at the end of each drill shift and tracking of sample shipments using industry-standard procedures. Core storage is secure because of the remote site and camp and access is strictly controlled and unauthorized camp personnel have been excluded from the core cutting facility.

1.12 Data Verification

The QP authors verified the data used in preparing the study through various activities including but not limited to site visits, reviewing source documents, QA/QC integrity, and discussions with site personnel.

1.13 Metallurgical Testwork

Between 2004 and 2007, several mineralogical investigations were conducted by organizations, including Amtel, Hazen Research, G&T Metallurgical Services, Barrick Technology Centre and SGS Lakefield. In 2018, additional work was done by FLSmidth Dawson Metallurgical Laboratories, and AuTec. The studies revealed that gold is associated with quartz-carbonate-sulfide veins. Pyrite is the dominant sulfide mineral, with marcasite present at a ratio of approximately 1:7 compared to pyrite. Arsenopyrite serves as the primary arsenic carrier, while stibnite is the main carrier of antimony. Arsenopyrite is the primary host mineral for gold, containing the majority of gold in a solid solution, while pyrite and marcasite also serve as gold carriers, though to a lesser extent.

Initial comminution testwork was undertaken in the 1990s with additional work completed in 2002-2003 at Hazen Research, Golden, Colorado (Hazen) that was managed by NOVAGOLD. Placer Dome Inc. initiated further work by SGS Lakefield Research Limited, Lakefield, Ontario (SGS Lakefield) in 2004 and on fresh core in 2007 that formed the basis for the semi-autogenous milling with ball milling and pebble crushing (SABC) grinding circuit design. The ores have a moderate hardness, indicated by an average Ball Work Index of 15 kWh/t and a SAG power index of 88 minutes, and amenability for SAG milling. Ore hardness is largely influenced by the type of rock lithology.

Flotation bench scale and pilot testing conducted between 2004 and 2007 at G&T Metallurgical Services, Kamloops, British Columbia (G&T) and SGS Lakefield confirmed a mill chemical-float, mill chemical-float (MCF2) flowsheet.

Additional bench flotation testwork was completed in 2018 and 2019 at AuTec. Results show opportunity for plant optimization and potential reductions to process operating and capital costs. However, the recommendations from the testwork include the need for further testing, locked-cycle testing and continuous pilot plant, and more detailed economic evaluation of impacts of any new design parameters. Therefore, the results of the 2017 and 2018 testwork programs are not yet definitive and have not been incorporated into the process design.

Bench-scale and pilot plant runs were initiated in 2006 and 2007 to test grinding, flotation, pressure oxidation (POX), and neutralization to develop process parameters and expand engineering information.

Testwork completed by SGS Lakefield, Hazen, G&T, Barrick Technology Centre, Vancouver, British Columbia (BTC) under Barrick's supervision showed that the Donlin ore requires pre-treatment prior to cyanidation to recover the gold. Process development work has determined that POX is the preferred method of pre-treatment.

Extensive testwork on composites has shown that acceptable gold recoveries can be produced through a combination of flotation pre-concentration, POX, and carbon in leach (CIL) cyanidation.

Flotation using the MCF2 flowsheet provides recoveries ranging from 81.45-97.7% depending on geological domain. For partially geologically oxidized ores, the recoveries were reduced with an average recovery rate of 75%. CIL recoveries after POX are approximately 96.6% for an estimated average LOM combined plant total gold recovery of 90.0%.

1.14 Mineral Resource Estimates

Mineral Resources take into account geologic, mining, processing and economic parameters, and have been constrained within an appropriate pit shell. Mineral Resources are reported in inclusive of Mineral Reserves in Table 1-2 and exclusive of Mineral Reserves in Table 1-3 at a commodity price of $2,400/oz gold and a NSR cut-off of $26.86/t. The figures in Table 1-3 are a subset of and are not additive to the figures in Table 1-2.

Table 1-2: Mineral Resources Statement Inclusive of Mineral Reserves

Category	Tonnage (kt)	Au Grade (g/t)	Contained Au (koz)	S Grade (%)
Measured	9,243	2.67	793	1.25
Indicated	550,727	2.21	39,195	1.12
Total Measured and Indicated	**559,970**	**2.22**	**39,988**	**1.12**
Inferred	88,886	2.03	5,812	1.09

Note: (1) The effective date of the Mineral Resource estimate is 30 November 2025. The QP for the Mineral Resource estimate is Mr. Henry Kim, P.Geo., an employee of Wood.

(2) Mineral Resources are prepared in accordance with CIM Definition Standards and the CIM MRMR Best Practice Guidelines.

(3) Mineral Resources are inclusive of Mineral Reserves.

(4) The cut-off date for the sample database used in the Mineral Resource estimate is 31 December 2024. However, more recent drilling data up to 30 November 2025 was used to validate the Mineral Resource model as remaining current.

(5) Mineral Resources are constrained within a pit shell using the following assumptions: gold price of $2,400/oz; reference mining cost of $2.68/t mined incremented $0.0041/t mined/m with depth from the 220 m elevation (equates to an average mining cost of $3.23/t mined); mining sustaining cost of $0.41/t mined; variable metallurgical recoveries by rock type and geological domain, ranging from 29.4% in oxide to 94.17% in intrusive rocks in the Akivik domain; process operating cost of $20.01/t processed; process sustaining cost of $2.14/t processed; G&A cost of $4.57/t processed; stockpile reclaim cost of $0.30/t processed assuming a reclaim percentage of 45%; refining recovery of 99.9%; selling cost of $1.71/oz gold; royalty considerations of 4.5% NSR and $0.50/t processed; and variable pit slope angles, ranging from 22 to 47°.

(6) The NSR value for each block is determined using the gold grade, processing and refining recoveries, gold price, selling costs, and royalties.

(7) Mineral Resources are reported using a marginal NSR cut-off value of $26.86/t based on a total process cost of $22.15/t processed, G&A operating cost of $4.57/t processed, and a stockpile reclaim cost of $0.30/t processed assuming a reclaim percentage of 45%.

(8) The average LOM process recovery for Mineral Resources is 89.8%.

(9) Sulfur is not an economic contributor to the Project; however, it does impact the POX process.

(10) Tonnage and grade measurements are in metric units. Contained gold ounces are reported as troy ounces.

(11) Rounding may result in apparent summation differences between tonnes, grade, and contained metal content.

Table 1-3: Mineral Resources Statement Exclusive of Mineral Reserves

Category	Tonnage (kt)	Au Grade (g/t)	Contained Au (koz)	S Grade (%)
Measured	1,432	1.18	54	1.05
Indicated	175,224	1.32	7,439	1.00
Total Measured and Indicated	**176,656**	**1.32**	**7,493**	**1.00**
Inferred	74,426	1.87	4,483	1.06

Note: (1) The effective date of the Mineral Resource estimate is 30 November 2025. The QP for the Mineral Resource estimate is Mr. Henry Kim, P.Geo., an employee of Wood.

(2) Mineral Resources are prepared in accordance with CIM Definition Standards and the CIM MRMR Best Practice Guidelines.

(3) Mineral Resources are exclusive of Mineral Reserves.

(4) The cut-off date for the sample database used in the Mineral Resource estimate is 31 December 2024. However, more recent drilling data up to 30 November 2025 was used to validate the Mineral Resource model as remaining current.

(5) Mineral Resources are constrained within a pit shell using the following assumptions: gold price of $2,400/oz; reference mining cost of $2.68/t mined incremented $0.0041/t mined/m with depth from the 220 m elevation (equates to an average mining cost of $3.23/t mined); mining sustaining cost of $0.41/t mined; variable metallurgical recoveries by rock type and geological domain, ranging from 29.4% in oxide to 94.17% in intrusive rocks in the Akivik domain; process operating cost of $20.01/t processed; process sustaining cost of $2.14/t processed; G&A cost of $4.57/t processed; stockpile reclaim cost of $0.30/t processed assuming a reclaim percentage of 45%; refining recovery of 99.9%; selling cost of $1.71/oz gold; royalty considerations of 4.5% NSR and $0.50/t processed; and variable pit slope angles, ranging from 22 to 47°.

(6) The NSR value for each block is determined using the gold grade, processing and refining recoveries, gold price, selling costs, and royalties.

(7) Mineral Resources are reported using a marginal NSR cut-off value of $26.86/t based on a total process cost of $22.15/t processed, G&A operating cost of $4.57/t processed, and a stockpile reclaim cost of $0.30/t processed assuming a reclaim percentage of 45%.

(8) The average LOM process recovery for Mineral Resources is 89.8%.

(9) Sulfur is not an economic contributor to the Project; however, it does impact the POX process.

(10) Tonnage and grade measurements are in metric units. Contained gold ounces are reported as troy ounces.

(11) Rounding may result in apparent summation differences between tonnes, grade, and contained metal content.

1.15 Mineral Reserve Estimates

Mineral Reserves were optimized for all Measured and Indicated blocks assuming a gold selling price of $2,100/oz. Using the proposed open pit mining method, modifying factors have been applied to the Measured and Indicated Mineral Resources to determine Proven and Probable Mineral Reserves. Mineral Reserves are summarized in Table 1-4.

Table 1-4: Mineral Reserves Statement

Category	Tonnage (kt)	Au Grade (g/t)	Contained Au (koz)	S Grade (%)
Proven	9,487	2.29	698	1.15
Probable	495,324	2.02	32,099	1.09
Total Proven and Probable	**504,811**	**2.02**	**32,797**	**1.09**

Note: (1) The effective date of the Mineral Reserve estimate is 30 November 2025. The QP for the Mineral Reserve estimate is Mr. Edwin Peralta, PE, an employee of Wood.

(2) Mineral Reserves are prepared in accordance with CIM Definition Standards and the CIM MRMR Best Practice Guidelines.

(3) Mineral Reserves are constrained within a engineered pit design using the following assumptions: gold price of $2,100/oz; reference mining cost of $2.68/t mined incremented $0.0041/t mined/m with depth from the 220 m elevation (equates to an average mining cost of $3.23/t mined); mining sustaining cost of $0.41/t mined; variable metallurgical recoveries by rock type and geological domain, ranging from 29.4% in oxide to 94.17% in intrusive rocks in the Akivik domain; process operating cost of $20.01/t processed; process sustaining cost of $2.14/t processed; G&A cost of $4.57/t processed; stockpile reclaim cost of $0.30/t processed assuming a reclaim percentage of 45%; refining recovery of 99.9%; selling cost of $1.71/oz gold; royalty considerations of 4.5% NSR and $0.50/t processed; and variable pit slope angles, ranging from 22 to 47°.

(4) Mineral Reserves are reported using an economic NSR cut-off value of $29.95–32.36/t followed by an elevated gold cut-off grade of 0.75 g/t. The NSR value for each block is determined using the gold grade, processing and refining recoveries, gold price, selling costs, and royalties.

(5) The average LOM processing recovery for the Mineral Reserves is 90.0%.

(6) Sulfur is not an economic contributor to the Project; however, it does impact the POX process.

(7) Tonnage and grade measurements are in metric units. Contained gold ounces are reported as troy ounces.

(8) Rounding may result in apparent summation differences between tonnes, grade, and contained metal content.

1.16 Mine Methods

The proposed Project will be a conventional, large-tonnage, open-pit operation designed to provide for a nominal process throughput of 53,500 t/d. The operating mine life is estimated at 24 years based on the planned processing rate.

The ACMA ultimate pit has been divided into seven phases, the Lewis pit into four phases. The initial phases of the two pits are independent, but they partially merge later in the mine life, forming a final single pit. The mine design, complete with haulage access, includes 504.8 Mt of ore and has a strip ratio of 6.52:1.

Mineable pit phases were designed based on optimized nested pit shell guidance, gold grade, strip ratio, access, and backfilling of the ACMA phases. Ramps in final walls have a design width of 40 m and a gradient of 10%. A nominal minimum mining width of 150 m was used for phase design.

Dates in this Report are relative to the start of process production. Pre-production has been defined as starting in early Year -1 and finishing second quarter Year 1, when the main orebody is exposed. Process production starts in mid-Year 1.

The schedule incorporates long term and short-term ore stockpiles. The long-term stockpile will hold all ore produced at the mine in excess of plant feed, separated into three sections according to sulfur grade for blending purposes.

The amount of sulfur in the feed can be controlled through a blending strategy combining ore feed directly from the mine and from stockpiles.

Waste will be stored in a single ex-pit waste rock facility (WRF) with a total capacity of 2,458 Mt located in the American Creek Valley, east of the pit area. Commencing in Year 18, waste can be stored in the ACMA backfill WRF with a capacity of 940 Mt. Two additional overburden WRF will be used for reclamation and tailings dam construction.

1.17 Recovery Methods

The process design is based on conventional and proven technology for the concentrator, flotation, POX, and cyanidation facilities for large, modern gold processing plants. The process plant is based on an average daily throughput of 53,500 t/d that considers a 93% plant availability.

ROM ore will be crushed in a gyratory crusher followed by a SAG mill and two-stage ball milling, addition of chemicals, and a flotation circuit. The primary ball milling circuit will produce a P_{80} particle size of 120 to 150 µm as feed to the primary rougher flotation section. The Rougher tails are fed to the secondary ball milling circuit, which will produce a P_{80} particle size of 50 µm as feed to the secondary rougher flotation section.

Gold-bearing sulfides, recovered by flotation, generate a concentrate containing 7% sulfur. The concentrate is refractory and will be treated in a POX circuit prior to cyanidation. Excess acid from the autoclave circuit will be neutralized with flotation tailings and slaked lime. Tailings from the process will be impounded in a TSF; water reclaimed from the TSF will be reused in the process plant.

Mineralogical studies have shown that the gold is not visible. Testwork analysis indicates a high level of association of gold with arsenopyrite. Other sulfides such as pyrite and marcasite are also present, with reduced tenors of gold. Organic carbon, a potential preg-robber, is present in the sedimentary ore. It is also present at lower levels in the intrusive ores, believed to be in the form of well-ordered graphite. This form of organic carbon is possibly less likely to preg-rob.

Given the plan to use stockpiles to manage the ore blend into the process from the perspective of gold, sulfur, carbonate, and hardness, allowances were made for ore aging or stockpile degradation for the LOM feed. Ore oxidized through weathering will have a slower flotation response than fresh rock. In general, ore at Donlin does not contain highly reactive sulfide species, and testwork has shown no statistical deviation over a one-year period. While data from a longer timeframe are not presently available, the testwork results for oxidized material show some degradation. Consequently, there is no effect on flotation recovery for material stockpiled for less than one year (sulfide “fresh” material).

1.18 Project Infrastructure

The Project will require construction of significant infrastructure to support the planned producing facilities. The planned infrastructure for the Project site includes process plant ancillary facilities, a power plant, TSF and water diversion and retention structures, accommodation camp as well as extensive off-site facilities designed to support the mine, including a buried steel natural gas pipeline that originates from the Beluga gas pipeline near Cook Inlet. Electric power for the Project site will be generated from a dual-fueled (natural gas and diesel) reciprocating engine power plant. There is currently no road access to the Project site. Supplies will be shipped on ocean barges to a port at or near Bethel where cargo will be transferred to river barges that will transport supplies and fuel to the Jungjuk Port. An all-season access road will connect Jungjuk Port to the mine site. The site also features a gravel airstrip for personnel transport and emergency response. Water supply for processing and potable use is sourced from wells and reservoirs, with modular water treatment plants for both construction and operations. A tailings storage facility (TSF) located in the Anaconda Creek valley will be supported by a seepage collection dam downstream and two temporary freshwater diversion dams upstream to limit surface water reporting to the tailings dam. Ore stockpile and WRFs manage material on site. Two contact water dams (CWDs) will manage and retain water that has been in contact with mine operations, including runoff, seepage, and pumped flows from the open pit and WRF.

1.19 Market Studies and Contracts

NOVAGOLD will be able to market its share of gold produced from the Project. Sales contracts that could be negotiated would be expected to be within industry norms. However, most of production would be expected to be spot marketed.

1.20 Environment, Permitting and Social and Community Impact

There was a focused effort to collect comprehensive environmental baseline data between 1995 and 2013 to lay the groundwork with local and regulatory stakeholders for the successful permitting and development of the Project. Baseline data collected included studies covering wetland delineation, water quality, fish and aquatic habitats, air quality, wildlife habitats, cultural resources and heritage, subsistence, traditional knowledge, socioeconomics, health, mercury data, overburden, ore and waste rock characterization studies, noise, visual aesthetics, and river and land use.

The National Environmental Policy Act (NEPA) process and formal permit applications require the preparation of an environmental impact statement (EIS). Upon completion of the Final EIS, a Joint Record of Decision (JROD) between U.S. Army Corps of Engineers (USACE) and the Bureau of Land Management (BLM) was finalized on 13 August 2018 in which Alternative 2, the North Route Pipeline option was chosen. The EIS and JROD describe the conditions of the approval and explain the basis for the decision. The State permitting process has been ongoing to comply with the NEPA mitigation requirements and state permitting requirements.

The Report assumes that Donlin Gold LLC will establish as planned a trust fund to fund closure and post-closure obligations as allowed by State statutes. That fund must be sufficient to generate the cash flow to cover all reclamation, closure, and post- closure costs and is estimated at $7.8 million provided annually over the construction and operating period, for a total of $412 million accrued to the trust fund at the start of closure.

In addition to the trust fund, financial assurance in the form of letters of credit and/or surety bonds will be required to construct and operate the mine. Per the Donlin Gold Project Reclamation Plan Approval from Alaska Department of Natural Resources (ADNR), financial assurance in the amount of approximately $322 million must be submitted in a form and substance approved by ADNR. The cost to maintain this financial assurance is assumed to be approximately $1.3 million per year, paid from the start of construction through the end of operations.

Sustained consultation with Alaska Native Corporations, Tribal Councils, and regional stakeholders has remained integral to Project planning. Commitments include regional workforce development, cultural heritage preservation, and local infrastructure improvements that align with sustainable development principles.

1.21 Capital Costs

The capital cost estimate is based on updated, fourth quarter 2025 pricing applied to the engineering designs and material take-offs (MTOs) from the feasibility study. The level of accuracy for the estimate is ±25% of estimated final costs, with a blended 13.8% contingency.

Apart from the following three design changes, no other changes to engineering or MTOs were made:

- Mine plan quantities and sequencing, including fleet purchase schedules
- The natural gas pipeline was updated for an increase in pipe diameter from 12″ to 14″ and for modifications made to the route (i.e., the North Route Alignment) between mile post (MP) 85 and 112.

Initial and sustaining capital estimates were updated for all areas. Closure and reclamation costs were based on Donlin Gold LLC's filed closure plan with the ADNR. Warehouse inventory is excluded from the capital cost estimate but is included in the financial model as part of the working capital provision.

The total initial capital cost estimate is $9,233 million (Table 1-5) and total sustaining capital is $2,325 million (Table 1-6).

Table 1-5: Initial Capital Estimate

Description	Estimate ($M)
Direct Costs	
Mining	429.4
Site Preparation and Roads	334.1
Process Facilities	1,817.8
Tailings Storage Facility and Reclaim Systems	169.4
Utilities	1,875.4
Ancillary Buildings and Facilities	437.9
Off Site Facilities	346.4
Subtotal Direct Cost	5,410.4
Owner's Costs	665.7
Indirect Costs	2,038.1
Contingency	1,119.1
Total Initial Capital Cost	**9,233.4**

Table 1-6: Sustaining Capital Estimate

Description	Estimate ($M)
Mining	1,125.0
Site Preparation and Roads	4.0
Process Facilities	-
Tailings Storage Facility and Reclaim Systems	876.9
Utilities	-
Ancillary Buildings and Facilities	119.5
Off Site Facilities	24.9
Subtotal Direct Cost	2,150.3
Owner's Costs	-
Indirect Costs	174.7
Contingency	-
Total	**2,325.1**

1.22 Operating Costs

The operating cost estimates were updated to fourth quarter 2025 pricing by updating key cost drivers like energy, labor, consumables, and freight. Mining costs have been updated to align with the new design and schedule. The level of accuracy for the estimate is ±25% of estimated final costs. The updated estimated LOM operating costs are $48.54/t processed (see Table 1-7).

Table 1-7: LOM Operating Costs

Area	Total LOM ($M)	$/t Processed
Mining	11,946.7	23.67
Processing	7,718.3	15.29
G&A	2,070.4	4.10
Land and Royalty Payments	2,768.8	5.48
Total	**24,504.3**	**48.54**

1.23 Economic Analysis

Certain information and statements contained in this section of the Report are forward-looking in nature and are subject to known and unknown risks, and that actual results of the economic analysis may vary from what is forecast. Examples of forward-looking information include gold price assumptions, cash flow forecasts, projected capital and operating costs, mine and metallurgical recoveries, mine life and production rates, and other assumptions used in this feasibility study identified in the relevant sections of this Report. Material risk factors are identified in Section 1.25.

The overall economic viability of the Project has been assessed using both undiscounted and discounted cash flow techniques. Undiscounted techniques include total net cash flow and payback period (measured from start of production). Discounted cash flow techniques include net present value (NPV) and internal rate of return (IRR). Discounted values are calculated using a 5% discount rate and a discrete end-of-year convention relative to Year -6, the start of basic and detailed engineering.

Based on the economic evaluation using a forecast long term gold price of $2,100/oz, the Project generates positive before and after tax economic results. After tax NPV_5 is $5,058 million and the after tax IRR is 10.3%. After tax payback is achieved 6.5 years following the start of production. Key evaluation metrics are summarized in Table 1-8.

Sensitivity analyses performed on gold price, gold grade, operating costs, and capital costs showed the Project is most sensitive to changes in the gold price and gold grade.

Table 1-8: Financial Results

Item	Unit	Value
Before Tax Valuation Indicators		
Total Before Tax Cash Flow	$M	25,415
Total Before Tax $NPV_{5\%}$	$M	7,516
Before Tax IRR	%	12.5
Before Tax Payback Period	years	4.9
After Tax Valuation Indicators		
Total After Tax Cash Flow	$M	19,614
Total After Tax $NPV_{5\%}$	$M	5,058
After Tax IRR	%	10.3
After Tax Payback Period	years	6.5

1.24 Conclusions

This Report updates the technical and economic information in the feasibility study on the Project and provides a current Report. Under the assumptions in this Report, the Project shows a positive financial return.

1.25 Risks

1.25.1 Summary

The Project is subject to risks that are commonly expected to exist with large, undeveloped mines and risks that are specific to Arctic conditions. These include the following sections.

1.25.2 Mineral Resource Estimates

Risks identified for Mineral Resources are uncertainties in, or changes from, what was assumed in the estimate for:

- Gold price
- Unrecognized variability in the metallurgical recoveries

- Uncertainties in the geotechnical characteristics of the rock mass and the impact on the pit slope angles
- Uncertainties to the inputs to the resource cut-off
- Gold threshold for defining the indicator mineralized shells
- Uncertainties in the interpretations of fault geometry, in particular the Vortex and Lo faults
- Search orientations used for grade estimation
- Uncertainty in the geological model
- Mineral Resource confidence classification criteria.

1.25.3 Mine Geotechnical

Risk factors associated with pit design are:

- The orientation and location of faults: faults with unfavorable orientation (e.g., dipping out of the slope face) may act as failure planes for planar or wedge-type collapses.
- Strength, spacing, and persistence of weak joints and strata (e.g., ash layers and shale slickensides) may cause pit slope failures.
- Unrecognized structural complications, low strength of joint with slickensides, faults, or ash layers, or unfavorable groundwater conditions (especially at faults) could introduce unfavorable pit slope stability conditions.
- Limited information available on how major faults affect the groundwater systems.
- Unexpected differences from the assumed water encountered in the pit walls producing extensive seepage into the pit requiring additional horizontal drains and pumping wells.

1.25.4 Waste Rock Facility

The risks associated with the WRF are:

- A risk associated with the stability of the WRF design is the presence of organic and ice-rich overburden deposits in the interior foundation of the WRF that could result in instability. This could limit overburden stripping due to lack of available dump space with possible impacts on ore stripping, or could pose operational safety issues.
- The presence of organic and ice-rich overburden deposits in the interior foundation of the WRF could result in instability that limit overburden stripping due to lack of available dump space with possible impacts on ore stripping, or could pose operational safety issues.

1.25.5 Tailings and Geotechnical

Risks associated with the TSF are:

- The proposed use of a geotextile for filter protection of the TSF underdrain system could delay approval for construction of the tailings dam.
- Weak ash layers or shale joints with slickensides could exist in the foundation of the TSF, and in conjunction with undrained or strain-softening behavior could require flatter slopes than the current design, increasing required fill volume and cost.
- Insufficient or inadequate borrow material could delay TSF embankment construction resulting in delays or limitations for production.
- A failure of a temporary water diversion dam could create dam safety or operational issues for the TSF with a resultant impact on production.

1.25.6 Recovery Methods

Risk associated with the process design are:

- Sulfur content variability in mill feed poses a risk to production. Years with lower sulfur content may result in reduced processing efficiency, making it challenging to achieve planned production levels.
- Stockpiling sulfide ore may lead to oxidation that could negatively impact flotation performance and gold recovery.

1.25.7 Project Infrastructure

Risk associated with the infrastructure design are:

- There are known to be intermittent areas of permafrost and poor ground conditions at the various facility sites that could affect foundation design and site preparation.
- Concrete retaining walls are used within the plant site footprint to separate tiers of the facility. Over the 27 year operation of the facility, these retaining walls pose a risk if they begin to fail.
- The design of plant site buildings and other structures adopted the seismic provisions of the 2006 International Building Code which may not be in accordance with the current code 2021 edition.

- The overall number of proposed Horizontal Directional Drill crossings (HDDs) on the natural gas pipeline is currently eight. Additional HDDs may be required based on the number of streams along the proposed route. Further construction evaluation and regulatory agency consultations will be necessary to determine the total number of HDDs required.

1.25.8 Water Management

Risks associated with water management are:

- Groundwater data near major faults remains sparse, increasing uncertainty in hydrogeologic interpretations
- Use of outdated MODFLOW versions limits accuracy for complex pit simulations and future pit lake modeling
- Without appropriate controls, widespread arsenic contamination of water resources could occur.

1.25.9 Environmental, Permitting, and Social and Community Impact

The Project reliance on natural gas introduces a strategic risk due to Alaska's limited supply and potential permitting delays. These factors could affect energy availability, impacting construction timelines, and operational continuity.

1.26 Recommendations

The QP authors have identified recommendations to be considered in the planned FS update. These include updates or completion of the following:

- Mineral Resource estimates with 2025 drilling data
- Certain aspects of the mining study
- Pit slope design report
- Aspects of the TSF
- Aspects of the natural gas pipeline design
- Inputs used to determine the water management strategy
- Aspects to environmental and permit plans.

Estimated costs of the recommendations total $1.64 million. This total does not include the planned feasibility study update.

2.0 INTRODUCTION

NOVAGOLD requested Wood and Geosyntec update content on the Donlin Gold project (the Project) and to prepare a current National Instrument (NI) 43-101 technical report (the Report).

NOVAGOLD is filing this Report on System for Electronic Document Analysis and Retrieval Plus (SEDAR+) to update the scientific and technical information on the Property and to support NOVAGOLD's year-end annual report filing.

2.1 Terms of Reference

Wood and Geosyntec have used information completed for earlier studies, as well as more recent data and information available on the Project to support and make current information in this Report. QP authors conducted due diligence reviews on the information supplied by Donlin Gold LLC and made adjustments to the results of the 2021 feasibility study update based on the outcome of those reviews.

Mineral Resource estimates were completed in accordance with the Canadian Institute of Mining, Metallurgy and Petroleum (CIM) Estimation of Mineral Resources and Mineral Reserves Best Practice Guidelines, November 29, 2019 (CIM MRMR Best Practice Guidelines) and reported in accordance with the CIM Definition Standards for Mineral Resources and Mineral Reserves, May 10, 2014 (CIM Definition Standards).

The Report uses American English. Unless otherwise specified in the text, monetary amounts are in US dollars and units are metric except where noted.

2.2 Qualified Persons

The following individuals are the QPs as defined in National Instrument 43-101, Standards of Disclosure for Mineral Projects, and are the co-authors of this report:

- Edwin Peralta, PE, Technical Director Mining & Geology, Wood
- Henry Kim, P.Geo, Principal Resource Geologist, Wood
- Alan Drake, P.L.Eng, Principal Metallurgist, Wood
- Jennifer Pretorius, P.Geo, Senior Hydrogeologist, Geosyntec
- Paul Baluch, P.Eng, Technical Director, Civil/Structural/Architectural, Wood
- Paul Dockweiler, CPG, Senior Geologist, Geosyntec
- Rick Sisson, PE, Senior Consultant, Geosyntec

2.3 Site Visits and Scope of Personal Inspections

The QPs conducted site visits of the Property as shown in Table 2-1.

Table 2-1: QPs, Areas of Report Responsibility, and Site Visits

Qualified Person	Site Visits	Report Sections of Responsibility
Edwin Peralta	8-9 September 2025	1.1-1.3, 1.12, 1.15, 1.16, 1.19, 1.21-1.24, 1.26, 2, 3, 12.3, 12.7.2, 15.1.1-15.1.4, 15.2-15.4, 16.1, 16.3-16.7, 16.9-16.11, 18.5.1, 19, 21.1.1, 21.1.1.1-21.1.1.3, 21.1.1.10-21.1.1.12, 21.1.2, 21.2.1-21.2.3, 21.2.5, 21.2.6, 22, 24, 25.1, 25.7, 25.8, 25.14, 25.16, 25.17, 26.1, 26.3, 26.10, 27
Henry Kim	18-21 September 2020 & 18-22 August 2025	1.1, 1.2, 1.4-1.12, 1.14, 1.24, 1.25.1, 1.25.2, 1.26, 2, 3.1, 4-11, 12.1, 12.2, 12.7.1, 14, 23, 25.1-25.4, 25.6, 25.18.1, 25.18.2, 26.1, 26.2, 26.10, 27
Alan Drake	N/A	1.1, 1.2, 1.12, 1.13, 1.17, 1.21, 1.22, 1.24, 1.25.1, 1.25.6, 2, 3.1, 12.4, 12.7.3, 13, 17, 21.1.1, 21.1.1.1, 21.1.1.2, 21.1.1.5, 21.1.1.10-21.1.1.12, 21.2.1, 21.2.2, 21.2.4, 25.1, 25.5, 25.10, 25.16, 25.18.1, 25.8.6, 27
Jennifer Pretorius	18-22 August 2025	1.1, 1.2, 1.12, 1.18, 1.21, 1.24, 1.25.1, 1.25.8, 1.26, 2, 3.1, 12.5.3, 12.7.6, 16.2.2.4, 16.8, 18.5.2, 18.7, 21.1.1, 21.1.1.1-21.1.1.3, 21.1.1.10, 21.1.1.11, 21.1.1.12, 25.1, 25.12, 25.16, 25.18.1, 25.18.8, 26.1, 26.8, 26.10, 27
Paul Baluch	8-9 September 2025	1.1, 1.2, 1.12, 1.18, 1.21, 1.24, 1.25.1, 1.25.7, 1.26, 2, 3.1, 12.5.1, 12.7.4, 18.1-18.4, 18.8-18.10, 18.11.1, 18.12, 21.1.1, 21.1.1.1, 21.1.1.2, 21.1.1.4, 21.1.1.7-21.1.1.12, 25.1, 25.11, 25.16, 25.18.1, 25.18.7, 26.1, 26.7, 26.10, 27
Paul Dockweiler	18-22 August 2025	1.1, 1.2, 1.12, 1.20, 1.24, 1.25.1, 1.25.9, 1.26, 2, 3.1, 12.6, 12.7.7, 20, 25.1, 25.15, 25.18.1, 25.18.9, 26.1, 26.9, 26.10, 27
Rick Sisson	8-9 September 2025	1.1, 1.2, 1.12, 1.18, 1.21, 1.24, 1.25.1, 1.25.3-1.25.5, 1.26, 2, 3.1, 12.5.2, 12.7.5, 15.1.5, 16.2.1, 16.2.2.1-16.2.2.3, 16.2.2.5, 16.2.2.6, 18.5.3, 18.6, 18.11.2, 21.1.1, 21.1.1.1, 21.1.1.2, 21.1.1.6, 21.1.1.10, 21.1.1.11, 21.1.1.12, 21.1.2, 25.1, 25.9, 25.13, 25.16, 25.18.1, 25.18.3-25.18.5, 26.1, 26.4-26.6, 26.10, 27

QP Peralta visited the Donlin property on 8-9 September 2025. While at site he participated in a fly over of the proposed pit to evaluate terrain, slope stability, and geotechnical conditions, as well as the proposed locations of the tailings storage facility, water containment dams and port. He visited the core shack and observed the geotechnical logging of two drill holes. While on site he was able to verify the proximity to mineralized zones and proposed haulage routes and infrastructure corridors, as well as inspect the proposed location of the waste rock storage facility, assess surface drainage patterns and identify potential constraints related to topography, overburden removal and pit wall orientation.

QP Kim initially visited the Donlin property on 18-21 September 2020. While at site he discussed the preparation of the DC9 geological and Mineral Resource model with the independent resource consultant. QP Kim also compared geological logs to the core, compared modeled lithologies to surface outcrops, reviewed logging and sampling protocols and observed the handling of core and sample preparation. He also compared available 2020 core logs to the DC9 geological model.

During QP Kim's recent visit to the Donlin property (18-22 August 2025) he conducted the following activities:

- discussed regional and local geology with site geologists
- visited the onsite core storage facility
- reviewed core of selected drill holes and reviewed the logged geological units and assay intervals matching them with visible mineralization, observed major structures and matched assay certificates to the assay database
- reviewed geotechnical and geological logging procedures
- visited active drill sites and observed drilling procedures
- verified the collar locations of several drill holes using a handheld global positioning system (GPS).

QP Pretorius conducted a site visit to the Donlin property from 18-22 August 2025. Activities conducted while at site included field inspections of natural water bodies and drainage channels, assessments of water sources for various operational requirements, and detailed reviews of water management plans and hydrogeological data. An aerial survey via helicopter provided an overview of the project area, including infrastructure situated beyond the immediate mining zone. She evaluated drainage patterns, runoff controls, and prospective locations for water treatment and storage facilities. Additionally, she inspected current waste management practices and geological features to support future infrastructure development and water management strategies.

QP Baluch visited the Donlin property on 8-9 September 2025. While at site he participated in a fly over of the proposed sites for the process plant, TSF, access road from the Jungjuk port, the Jungjuk port and the new airstrip. He also walked the locations of the proposed process plant, primary crusher, TSF and waste rock storage facility and visited the core shack and sample preparation facility. While on site he was able to discern the proximity to mineralized zones and proposed haulage routes and infrastructure corridors, and observed topography, drainage patterns and surface geotechnical aspects of the proposed sites.

QP Dockweiler conducted a site visit to the Donlin property from 18-22 August 2025. During the site visit, he performed baseline environmental assessments of water systems, infrastructure sites, and the surrounding ecology. The scope of activities included evaluating water distribution and monitoring facilities, observed the collecting of data on local vegetation and wildlife, reviewing proposed construction areas, and determining water management requirements. Additional tasks involved assessing permitting processes, consulting on stakeholder engagement strategies, and assessing the relationship with local native villages.

QP Sisson's site visit to the Donlin property from September 8-9, 2025, focused on geotechnical engineering aspects essential for mine development. During his flyover of the proposed pit, he evaluated terrain features, slope stability, and geotechnical conditions relevant to supporting the mine pits, WRF, plant site, and TSF. At the core shack, he observed geotechnical logging of drill holes to inform pit design and assess foundation conditions for the tailings and WRFs.

QP Drake did not conduct a site visit as there is currently no process facility to observe at site.

2.4 Effective Dates

The effective date of the Mineral Resources, Mineral Reserves, and the overall Report is 30 November 2025.

2.5 Previous Technical Reports

NOVAGOLD previously filed the following technical reports on the Project:

- Hanson, K., Woloschuk, M., and Kim., H, 2021: NI 43-101 Technical Report on the Donlin Gold Project, Alaska, prepared for NOVAGOLD RESOURCES INC., effective date 1 June 2021
- Lipiec, T., Seibel, G., and Hanson, K., 2012: Donlin Creek Gold Project Alaska, USA, NI 43-101 Technical Report on Second Updated Feasibility Study, prepared for report prepared for NOVAGOLD RESOURCES INC., effective date 18 November 2011, amended 20 January 2012 (Donlin 2011 Technical Report)
- Francis, K., 2008: Donlin Creek Project, NI 43-101 Technical Report, Southwest Alaska, U.S., prepared for NOVAGOLD RESOURCES INC., report date 5 February 2008

- Dodd, S., Francis, K. and Doerksen, G., 2006: Preliminary Assessment Donlin Creek Gold Project Alaska, USA, NI 43-101F1 Technical Report, prepared for NOVAGOLD RESOURCES INC., report date 22 August 2006
- Dodd, S., 2006: Donlin Creek Project 43-101 Technical Report, NI 43-101F1 Technical Report, prepared for NOVAGOLD RESOURCES INC, report date January 2006
- Juras, S. and Hodgson, S., 2002: Technical Report, Preliminary Assessment, Donlin Creek Project, Alaska, NI 43-101F1 Technical, Report, prepared for NOVAGOLD RESOURCES Inc., report date March 2002
- Juras, S., 2002: Technical Report, Donlin Creek Project, Alaska, NI 43-101F1 Technical Report, report prepared for NOVAGOLD RESOURCES INC., report date 24 January 2002.

2.6 Information Sources

The primary data sources for this Report are the following:

- Documentation from the Donlin Gold Project, Alaska, Feasibility Study Update, prepared by Wood for Donlin Gold LLC, dated 30 July 2021
- AMEC Americas Ltd., 2011: Donlin Creek Gold Project, Alaska, Feasibility Study Update 2, dated November 2011.

Reports and documents listed in Section 3 and Section 27 were also used to support preparation of the Report.

3.0 RELIANCE ON OTHER EXPERTS

The QPs have relied upon the following experts, who provided information regarding mineral rights, surface rights, agreements, and tax content of this Report as noted below.

3.1 Mineral Tenure, Surface Rights, Agreements, and Royalties

The QP authors have not reviewed the mineral tenure, nor independently verified the legal status, ownership of the Project area, underlying property agreements or permits. The QPs have fully relied upon, and disclaim responsibility for, information derived from Donlin Gold LLC experts and experts retained by Donlin Gold LLC for this information through the following documents:

- Email from NOVAGOLD entitled "Donlin Technical Report Section 4" with attachment of the content for inclusion as Section 4 in this Report, dated 15 January 2026

This information is used in Section 4 of the Report and was also used to support considerations of reasonable prospects of economic extraction and declaration of Mineral Resources in Section 14, for consideration of appropriate modifying factors for declaration of Mineral Reserves in Section 15, and for royalties and encumbrances and property agreements considered for the economic analysis in Section 22.

3.2 Taxation

QP Peralta has not reviewed the taxation information. QP Peralta has fully relied upon, and disclaims responsibility for, information supplied by NOVAGOLD for information related to taxation contained in the following document:

- Taxation letter Title: Taxation information and tax inputs to the financial model used in the Donlin Gold Project Feasibility Study National Instrument 43-101 Technical Report prepared by Wood Group USA ("Wood") for NOVAGOLD RESOURCES INC. ("NOVAGOLD"), dated 14 January 2026.

This information is used to summarize the tax information and in support of the after-tax economic analysis in Section 22, which demonstrates economic viability of the Project in support of the Mineral Reserve estimation in Section 15.

4.0 PROPERTY DESCRIPTION AND LOCATION

4.1 Location

The Donlin deposits are situated approximately 450 km west of Anchorage and 250 km northeast of Bethel up the Kuskokwim River. The closest village is the community of Crooked Creek, approximately 20 km to the south, on the Kuskokwim River (Figure 4-1).

The resource areas are within T. 23 N., R. 49 W., Seward Meridian, Kuskokwim and Mt. McKinley Recording Districts, Crooked Creek Mining District, Iditarod A-5 USGS 1:63,360 topography map. The mineralization is centered on approximately 540222.50 east and 6878534.36 north, using the NAD 83 datum.

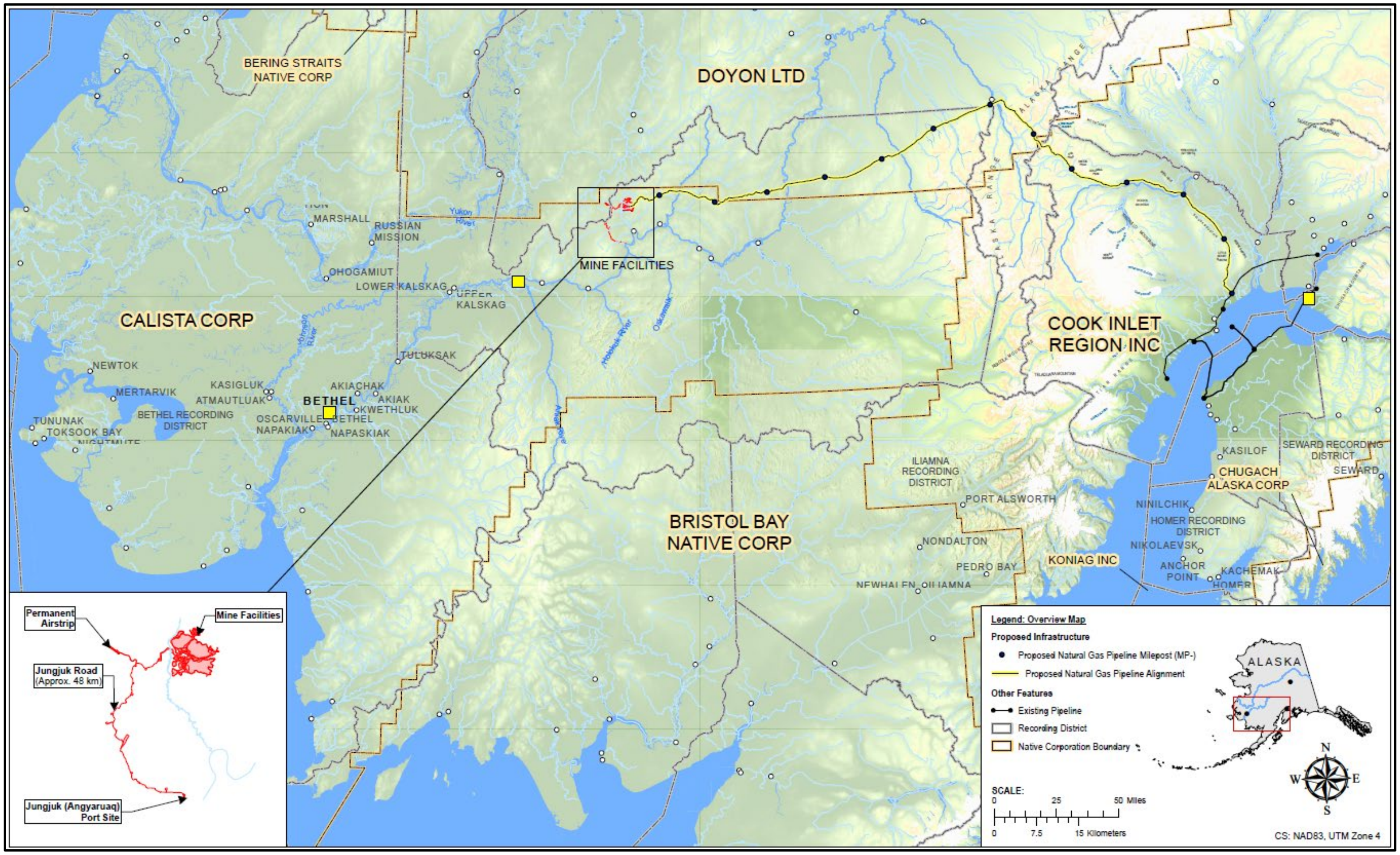
The area of the Donlin Gold property is 48,996 ha as described in Section 4.3.

The mineral resource areas consist of the ACMA and 400 Zone, Aurora and Akivik mineralized areas (grouped as ACMA) and the Lewis, South Lewis, Vortex, Rochelieu and Queen mineralized areas (grouped as Lewis). The proposed project configuration, including the pit outline for the combined ACMA and Lewis areas, in relation to the Calista Corporation (Calista) Lease boundary and The Kuskokwim Corporation (TKC) Surface Use Agreement boundary (SUA) (refer to Section 4.3) is shown as Figure 4-2.

4.2 Origin of Native Title to the Lands

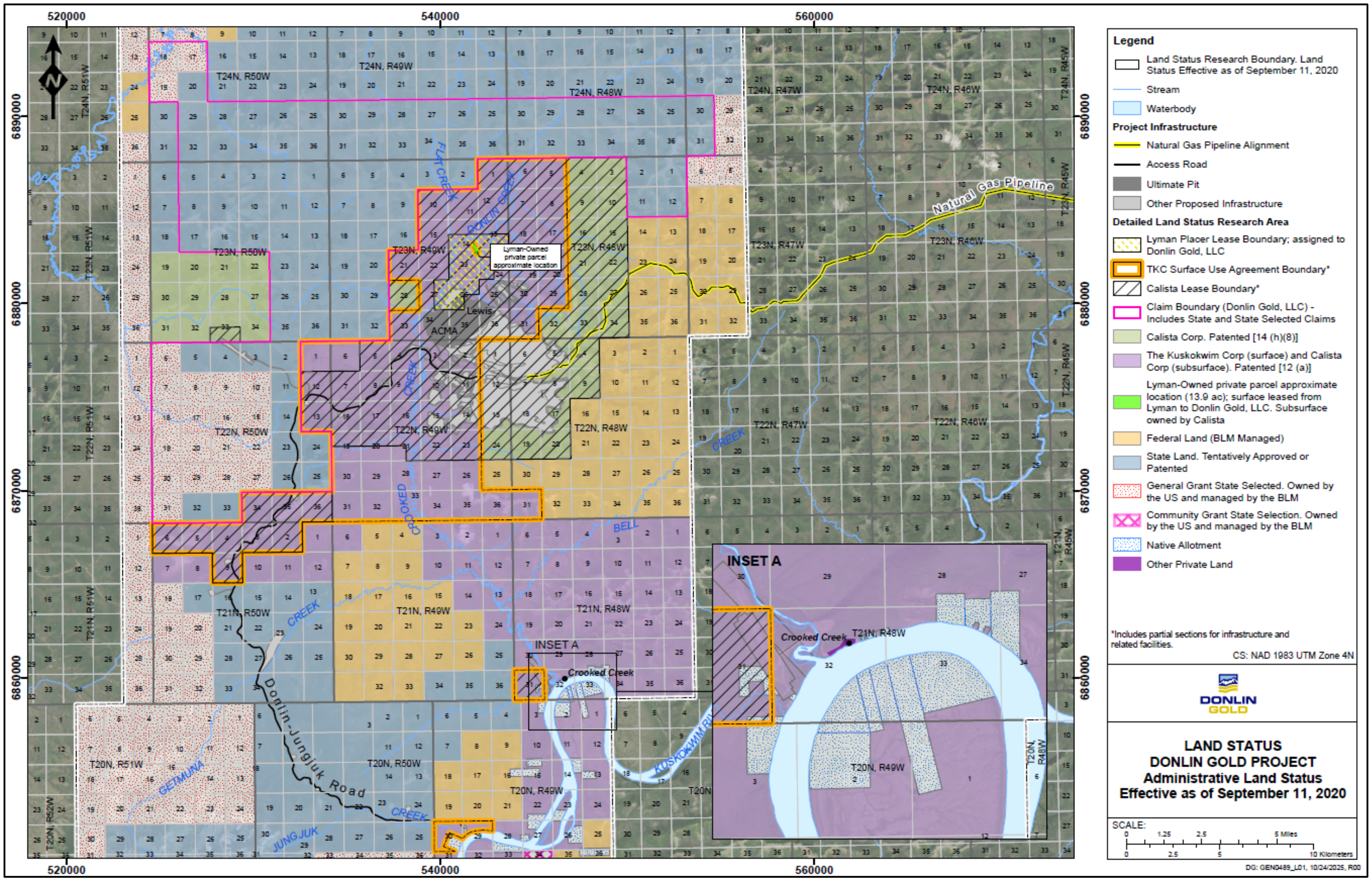
The surface and subsurface estates held by Calista Corporation, a regional corporation, and The Kuskokwim Corporation, a village corporation, arise pursuant to the Alaska Native Claims Settlement Act (ANCSA). Section 12(a) of ANCSA entitled each village corporation to select surface estate land from an area proximal to the village in an amount established by its population. Under ANCSA, the regional corporation receives conveyance of the subsurface estate underlying the surface estate conveyed to the village corporation. ANCSA also allowed regional corporations to select lands to which they received both the surface and subsurface estates. Pursuant to ANCSA, the United States has conveyed portions of the Project lands to Calista Corporation as to the surface and subsurface for certain lands and subsurface only for other lands, and The Kuskokwim Corporation (TKC) as to the surface estate only. Donlin Gold LLC holds rights to those portions of the Project lands owned by Native corporations pursuant to the agreements described below.

Figure 4-1: Location Map



Source: NOVAGOLD, 2025

Figure 4-2: Donlin Gold Property Land Status Map



Source: Donlin Gold LLC, 2025

4.3 Mineral Tenure

4.3.1 Calista Lease

Calista, an Alaska Native Corporation, holds the subsurface estate (including minerals) beneath the Village-owned surface estate that is part of the Project lands.

Mineral rights in the Project lands were originally leased from Calista in 1995. A Restated Exploration and Lode Mining Lease was executed in February 2011, to reflect all amendments and assignments to the prior lease up to and including 11 February 2011 (Lease). The Lease grants to Donlin Gold LLC, with respect to the lands subject to the Lease, the exclusive right to explore for, develop, and mine all minerals in or under the leased property. The Lease also grants the right to construct and use buildings, roads, tailings ponds, waste dumps, and other improvements reasonably required under the purposes of the Lease. The Lease was amended again effective 6 June 2014; however, the amendment does not affect the land subject to the Lease as restated in 2011.

The Calista Lease currently includes a total of 72 complete sections of land and portions of an additional 13 sections of land in the vicinity of the Donlin deposits, and associated with the Project infrastructure. These lands comprise approximately 19,988 ha that have been conveyed to Calista by the Federal Government. Calista also owns the surface estate on a portion of these lands.

4.3.2 Lyman Lease

Lyman Resources in Alaska, Inc. (Lyman Resources) has an existing placer mining lease covering approximately 1,040 ha (partially covering six sections) within the Calista Lease area (Lyman-Calista Lease). The Lymans also have title to approximately 5.7 ha of surface estate within the Snow Gulch area. The lands subject to the Lyman-Calista Lease lie immediately to the north of the Project's planned open pit footprint. The Calista Lease grants priority to extraction of the lode mineralization in the event of a conflict of use between lode and placer mining operations, provided that a two-year notice is provided to Lyman Resources of activities that would deprive Lyman of the opportunity to recover placer gold.

Lyman Resources, the Lyman family, and Donlin Gold LLC entered into a Surface Lease and Assignment of Mining Lease effective 9 May 2012 leasing the Lyman surface estate and assigning the Lyman Lease to Donlin Gold LLC for mining use (Lyman Lease). The Lyman Lease has an initial term of 20 years but shall be extended while Donlin Gold LLC conducts operations within an area of interest defined in the Lyman Lease.

4.3.3 State Mining Claims

In addition to the approximately 19,988 ha leased from Calista, Donlin Gold LLC holds 493 Alaska State mining claims comprising approximately 29,008 ha, on State and State-selected lands in the vicinity of the leased lands in the Kuskokwim and Mount McKinley recording districts, bringing the total mineral land package to approximately 48,996 ha (the Property).

Of these claims:

- 84 are on State-selected lands
- A total of 409 are located on lands that have been conveyed by the United States to the State of Alaska by Tentative Approval or Patent.

The mining claims abut and largely surround the northern and western boundaries of the lands subject to the Calista Lease and TKC SUA. State mining claims held by Donlin Gold LLC show as active on Alaska Department of Natural Resources online records, indicating that appropriate claim payments have been made and annual affidavits of annual labor have been timely recorded.

None of the claims held by Donlin Gold LLC have been surveyed.

All claims are either 64.8 ha or 16.2 ha in size.

4.4 Surface Rights

A separate Surface Use Agreement with TKC grants non-exclusive surface use rights to Donlin Gold LLC on at least 64 sections of land overlying much of the minerals leased from Calista, with provisions allowing for adjusting the SUA boundary in conjunction with adjustments to the mineral rights included in the Calista Lease. The SUA was originally entered into effective 5 June 1995, and revised and restated by Donlin Gold LLC and TKC to expand the TKC surface lands included in the SUA and update other provisions, effective 6 June 2014. The term of the SUA runs through 30 April 2031, and on a year-to-year basis thereafter, so long as the Calista Lease remains in effect. Upon termination of the Lease with Calista, the SUA will automatically terminate. The SUA provides Donlin Gold LLC with surface rights to approximately 16,923 ha of TKC-owned land.

The Calista Lease, TKC SUA, and Lyman Lease grant to Donlin Gold LLC most of the surface rights that will be required to support mining operations in the proposed mining area. Additional land rights are required for off-site infrastructure, such as the Jungjuk port site, the road from the port site to the mine site, and natural gas pipeline. These facilities are to be situated on Native, State of Alaska, and BLM lands. Rights-of-way are required from these underlying land owners for the road and pipeline corridors.

4.5 Royalties and Encumbrances

The terms of the Calista Lease and TKC SUA include various royalty and other payment provisions and considerations such as shareholder employment and contracting opportunities. The Lyman Lease provides for rent and certain other payments.

Royalty Terms of the Calista Lease include:

- Annual Advance minimum royalty (variable) to 2030
- All advance minimum payments are recoverable as a credit against the NSR royalty and net proceeds payment.
- NSR of 1.5% for the earlier of the first five years following commencement of commercial production or until initial capital payback, increasing to 4.5% thereafter
- Net proceeds royalty of 8% of the net proceeds realized by Donlin Gold LLC commencing with the first quarter in which net proceeds are first realized.

Payment Terms of the TKC SUA include:

- A Surface Use Fee paid annually on a per acre basis
- An Exclusive Use Fee for acres dedicated to certain uses or for Donlin Gold LLC’s exclusive use unless TKC elects to have Donlin Gold LLC purchase that portion of the surface estate
- Milestone payments due upon the occurrence of specific events
- Annual advance minimum payment (variable based on project status)
- Milled tonnage fee of $0.40/t processed for the first 10 years of production and $0.50/t processed for all production after 10 years
- Net proceeds payment of 3% of the net proceeds realized by Donlin Gold LLC commencing with the first quarter in which net proceeds are first realized
- All advance minimum payments are recoverable as a credit against the milled tonnage fee and net proceeds payment.

Additional estimated costs associated with various landowner and lease agreements, not already covered in initial capital or G&A operating costs, average approximately $8.6 million per year during the six pre-production years and $2.5 million per year during the 27 operating years.

Annual rent, labor expenditures and filings are required to maintain Alaska State mining claims on State land.

Mining license tax payments also apply and are discussed in Section 22.

4.6 Permits

Permits required to support Project development are discussed in Section 20 of the Report.

4.7 Environmental Liabilities

Environmental studies, closure plans and costs, and environmental liabilities and risk issues are discussed in Section 20 of the Report.

4.8 Significant Risk Factors

The relatively isolated location of the Property in Alaska makes the Project subject to the risk of delays to mine development and increased costs caused by difficult terrain and harsh seasonal weather conditions. Risks posed by these conditions are discussed in Section 18 of the Report.

5.0 ACCESSIBILITY, CLIMATE, LOCAL RESOURCES, INFRASTRUCTURE, AND PHYSIOGRAPHY

5.1 Accessibility

The Project site is approximately 450 km west of Anchorage and 250 km northeast of Bethel up the Kuskokwim River. The closest village is the community of Crooked Creek, approximately 20 km to the south, on the Kuskokwim River. Bethel, approximately 30 km upstream from the mouth of the Kuskokwim River, is the regional center for the Yukon-Kuskokwim Delta area of Southwest Alaska. The town of Aniak, also on the Kuskokwim River and about 80 km southwest of the Project site, is the regional center for the Upper Kuskokwim Valley.

There is no road or rail access to the site. The nearest roads are in the Anchorage area. Access to Bethel and Aniak, the regional centers, is limited to river travel by boat or barge in the summer and air travel year-round. The Kuskokwim River is a regional transportation route and is serviced by commercial barge lines.

All current access to the Project site for personnel and supplies is by air. The Project has an all-season, soft-sided camp sufficient to support recent field activities. An adjacent 1,500 m long airstrip is capable of handling aircraft as large as C-130 Hercules, with a payload of 18,000 kg, allowing efficient shipment of personnel, some heavy equipment, and supplies. The Project can be serviced directly by charter air facilities out of both Anchorage and Aniak.

5.2 Climate

The area has a relatively dry interior continental climate with typically about 500 mm of total annual precipitation. Summer temperatures are relatively warm and may exceed 30°C. Minimum temperatures may fall to well below -42°C during the cold winter months.

Exploration is possible year-round, though snow levels in winter and wet conditions in late autumn and in spring can make travel within the Project area difficult. It is expected that mining operations will be able to be conducted year-round.

5.3 Local Resources and Infrastructure

Local resources necessary for the exploration and possible future development and operation of the Project are in Bethel and the Yukon-Kuskokwim region. Some resources would likely have to be brought in from the Anchorage area or other parts of Alaska.

Alaska and the adjacent Canadian Province of British Columbia have a long mining history and a large resource of equipment and skilled personnel. The Property is currently isolated from power and other public infrastructure. The exploration camp has capacity for exploration and other on-site field work. Power is provided by on-site diesel generators. Water sources are described in Section 18.7.

Infrastructure assumptions and the proposed infrastructure layout for the Project are discussed in Section 18 of the Report.

5.4 Physiography

The Project area is one of low topographic relief on the western flank of the Kuskokwim Mountains. Elevations range from 150 to 640 m. Ridges are well rounded and easily accessible by all-terrain vehicles.

Hillsides are forested with black spruce, tamarack, alder, birch, and larch. Soft muskeg and discontinuous permafrost are common in poorly drained areas at lower elevations and along north-facing slopes.

Permafrost is sporadic, typically confined to valley bottom and mid-slope, with thickness ranging from 1.5 m to over 15 m (average ~4 m).

The site is in a seismically active region of Alaska, influenced by the collision of the Pacific and North American plates.

5.5 Sufficiency of Surface Rights

Regarding future mining operations, sufficient space is available to site the various facilities, including an open pit operation, personnel housing, stockpiles, tailing storage facility, waste rock storage facilities and processing plants.

It is a reasonable expectation that any additional surface rights to support Project development and operations can be obtained.

6.0 HISTORY

6.1 Prior Ownership

Calista Corporation (Calista), an Alaska Native Corporation, has held the mineral rights to the Project mine site lands since 1974.

Placer Dome Inc. acquired a 20-year lease from Calista effective 1 May 1995.

On 13 November 2002, NOVAGOLD Resources Alaska, Inc., a wholly-owned subsidiary of NOVAGOLD, earned a 70% interest in the Property by expending $10 million on exploration and development of the Project. Placer Dome Inc. retained an option to buy back into the Property.

On 11 February 2003, Placer Dome Inc. exercised its back-in right and assumed management of the continued development of the Property. In January 2006, Barrick acquired Placer Dome Inc. and assumed Placer Dome Inc.'s joint venture responsibilities for exploration and development activities on the Property.

On 1 December 2007, NOVAGOLD entered into a limited liability company agreement with Barrick (the Donlin LLC Agreement) that provided for the conversion of the Project into a new limited liability company, the Donlin Creek LLC (DCLLC), which was jointly owned by NOVAGOLD and Barrick on a 50/50 basis. In July 2011, the Board of Donlin Creek LLC voted to change the name of the company to Donlin Gold LLC.

On 3 June 2025, Barrick divested their 50% ownership in Donlin Gold LLC NOVAGOLD through their wholly-owned subsidiary NovaGold Resources Alaska Inc., increased their ownership interest to 60%. The remaining 40% ownership interest in the Donlin Gold Project is Donlin Gold Holdings LLC, a subsidiary of Paulson.

6.2 Exploration History

Placer gold was first discovered at Snow Gulch, a tributary of Donlin Creek, in 1909. Intermittent small-scale placer gold production by Lyman Resources continued through 2014. Resource Associates of Alaska (RAA) carried out a regional evaluation for Calista in 1974-1975. This work included a soil grid and three bulldozer trenches in the Snow area immediately north of the current resource area. Calista followed up with prospecting activities between 1984 and 1986 and completed minor auger drilling in 1987.

The first substantial exploration drill program was carried out by WestGold in 1988 and 1989. WestGold completed geological mapping, trenching, rock and soil sampling, an airborne magnetic and very low frequency (VLF) survey, and ground magnetic surveys. WestGold also tested biogeochemical sampling and ground penetrating radar with positive results. Based on this information, WestGold performed an initial Mineral Resource estimate.

Teck carried out a limited trenching and soil sampling program in the Lewis area in late 1993 and updated the Mineral Resource estimate.

Placer Dome Inc. explored the Property from 1995 to 2000. Placer Dome Inc. constructed an exploration camp and airstrip, undertook reconnaissance and geological mapping, aerial photography, completed rock chip and soil sampling, trenching, max-min (electromagnetic) geophysical surveys, airborne geophysical surveys, RC and core drilling, carried out detailed metallurgical testwork, and prepared a series of Mineral Resource estimates and initial mining and engineering studies.

Placer Dome Inc. formed the Donlin Creek joint venture (DCJV) with NOVAGOLD as operator in 2001. During the period of the DCJV, NOVAGOLD undertook trenching, core and geotechnical drilling, updated Mineral Resource estimates, and completed a Preliminary Assessment. Placer Dome Inc. reassumed management of the Property as operator in late 2002. From 2002-2005, work comprised additional core drilling, condemnation, geotechnical, and water drilling, geotechnical and hydrogeological studies, geological mapping and sampling of prospective calcium carbonate source areas, exploration and auger drilling program for sand and gravel resources, and updated Mineral Resource estimates.

Barrick acquired Placer Dome Inc.'s interest in the DCJV through a merger with Placer Dome Inc. in early 2006. Work completed in the period 2006-2007 included core drilling for resource infill, geotechnical, engineering, condemnation, waste rock, and metallurgical purposes, and updated Mineral Resource estimates.

The DCJV partners formed Donlin Creek LLC in late 2007, with the subsequent name change to Donlin Gold LLC occurring in 2011.

Work on the Property included soil and stream sediment sampling, core drilling for resource infill, geotechnical, engineering, condemnation, waste rock, and metallurgical purposes, and estimation of Mineral Resources and Mineral Reserves.

An initial feasibility study was completed on the Project in 2007, and updated in 2009, and then updated again in 2011.

Between 2011 and 2025 the following types of activities have been completed on the Property: resource infill and extension drilling, trenching, geotechnical work, metallurgical testing, monitoring to support permitting, advancement of permits and certificates for the project, consultation with local communities, community support activities and sponsorships, and infrastructure design work.

A summary of the exploration programs completed on the Property is summarized in Table 6-1.

Table 6-1: Exploration Work History Summary for Donlin Gold Property

Year	Company	Work Performed	Results
1909-1956	Various prospectors and placer miners	• Gold discovered on Donlin Creek in 1909 • Placer mining by hand, underground, and hydraulic methods	• Total placer gold production of approximately 30,000 oz
1970s-present	Robert Lyman and heirs	• Resumed sluice mining in Donlin area and placer mined Snow Gulch	• Small scale placer mining
1974, 1975	RAA	• Regional mineral potential evaluation for Calista • Soil grid and three bulldozer trenches in the Snow Gulch area	• Anomalous gold values in soil, rock, and vein samples
1984-1987	Calista	• Minor work	• -
1986	Lyman Resources	• Auger drilling for placer evaluation encounters sulfide-rich clay near Quartz Gulch	• Initial discovery of Far Side (Carolyn) prospect
1987	Calista	• Rock sampling of ridge tops and auger drill sampling of Far Side prospect.	• Anomalous gold values from auger holes
1988, 1989	WestGold	• Airborne geophysics, ground geophysics, geological mapping, and soil sampling over most of Project area • Trenching at all prospects First metallurgical tests and petrographic work	• Initial work identified eight prospects (Snow, Dome, Quartz, Carolyn, Queen, Upper Lewis, Lower Lewis, and Rochelieu) • Drilling at most of these prospects led to identification of the Lewis areas as having the best bulk-mineable potential • Early resource estimate performed • WestGold dissolved by early 1990

Year	Company	Work Performed	Results
1993	Teck	• Trenching and soil lines in Lewis area • Petrographic, fluid inclusion, and metallurgical work	• Identified new mineralized areas and expanded property, completed updated resource estimate • Metallurgical tests not favorable, property dropped
1995-2000	Placer Dome Inc.	• 87,383 m of core, 11,909 of RC drilling, and 8,493 m of trenching	• Drilled the American Creek magnetic anomaly (ACMA), discovered the ACMA deposit • Numerous Mineral Resource estimations
2001, 2002	DCJV (Placer Dome Inc. / NOVAGOLD)	• 46,495 m of core including 89.5 m of geotechnical drilling, 11,589 m of RC drilling, and 268 m of water monitoring holes • Mineral Resource estimate	• Expanded the ACMA resource
2003-2005	DCJV (Placer Dome Inc. / NOVAGOLD)	• 25,448 m of core and 5,979 m of RC drilling • Calcium carbonate exploration drilling • Induced polarized (IP) lines for facility condemnation studies.	• Infill drilled throughout the resource area demonstrated continuity • Discovered a calcium carbonate resource • Poor quality IP data not useful for facility studies
2006	DCJV (Barrick / NOVAGOLD)	• 92,804 m of core drilling for resource conversion, slope stability, metallurgy, waste rock, carbonate exploration, facilities, and port road studies	• Geological model and internal resource updates

Year	Company	Work Performed	Results
2007	DCJV	• Core drilling totaled 72,257 m and included resource delineation, geotechnical and engineering, and carbonate exploration • 13 RC holes for monitor wells and pit pump tests totaled 1,043 m • Updated Mineral Resource estimate	• Improved pit slope parameters • positive hydrogeological results • exploration for carbonate mineral source was negative.
2008	DCLLC	• 108 core holes totaling 33,425 m for exploration and facility related geotechnical and condemnation studies • Metallurgical testwork: flotation variability and cyanide (CN) leach • 54 test pits and 37 auger holes completed for overburden characterization	• Resource expansion indicated for East ACMA • CN leach resource potential indicated for the main resource area, Snow, and Dome prospects • Facility sites successfully condemned • Updated resource estimates utilizing applicable data through 2007.
2009	Donlin Gold LLC (name change)	• 19 geotechnical core holes totaling 950 m in facility sites and to address hydrology • Mineral Reserve and Mineral Resource estimate update	• -

Year	Company	Work Performed	Results
2010	Donlin Gold LLC	• Six geotechnical core holes totaling 2,090 m to evaluate slope stability of expanded pit • Drilled 90 auger holes totaling 585 m and dug 59 test pits to further evaluate overburden conditions and gravel supplies within TSF area • Mineral Reserve and Mineral Resource estimate update	• Pit slope stability of new pit design remained acceptable • Evaluation of construction suitability of surficial materials in TSF is ongoing
2017	Donlin Gold LLC	• 16 HQ core holes totaling 7,040 m drilled within the resource area • Acoustic televiewer surveys were completed on 12 holes. Five of the holes were also logged by geotechnical engineering consultants for pit slope geotechnical data collection • Metallurgical sample collection was also conducted.	• Geologic, geotechnical, and assay data were incorporated into project database for internal geologic modeling and optimization updates • Metallurgical samples were tested in 2018, primarily for flotation optimization.
2019	Donlin Gold LLC	• 30 geotechnical core holes totaling 1,060 m were drilled as part of a site investigation program in support of detailed dam design	• Geotechnical data were incorporated into a site investigation dataset to be utilized for detailed dam design and permitting once the field program is complete.

Year	Company	Work Performed	Results
2020	Donlin Gold LLC	• 85 holes and 23,361 m HQ core drilling in ACMA and Lewis resource areas • Objectives on this program was to validate and increase the confidence in recent geologic modelling concepts and support future resource updates. • Acoustic and optical televiewer surveying were completed on most of the holes. • Geotechnical logging was performed on core from 10 holes.	• Available geologic and assay data were incorporated into the project database for internal geologic modeling and optimization updates. • 2020 drilling geological logs generally agrees with the DC9 geological model while suggesting local adjustments.
2021	Donlin Gold LLC	• 79 core holes totaling 24,263 m in both the ACMA and Lewis deposits to validate recent geologic modeling • concepts and test for extensions of high-grade zones	• 2021 drilling geological logs and preliminary assays results generally agree with the DC9 model while suggesting local adjustments
2022	Donlin Gold LLC	• 141 core holes totaling 42,331 m in both the ACMA and Lewis deposits in-pit and below pit in sparsely drilled areas • Platform mapping, waste rock facility condemnation drilling and geotechnical drilling for the Alaska Dam Safety certificates	• Mapping to confirm mineralization continuity and key geological controls in representative areas of the deposit and studies to support the future update of the feasibility study

Year	Company	Work Performed	Results
2023	Donlin Gold LLC	• 42 core holes totaling 1,833 m and 13 RC holes totaling 1,279 m were drilled as part of a site investigation program in support of detailed dam design, hydrogeologic studies and seismic surveys.	• Work supports the Alaska Dam Safety certificates and mine planning and design work
2024	Donlin Gold LLC	• Metallurgical testwork, field and geochemical data collection and advancement of the Donlin Gold mineral resource model	• Work performed will support the future update of the feasibility study including closure planning.
2025	Donlin Gold LLC	• 47 core holes totaling 18,056 m comprising of infill drilling, in-pit exploration and geotechnical drilling • 26 holes totaling 399 m geotechnical drilling at Jungjuk Port Road material sites	• Work performed will support the future update of the feasibility study.

7.0 GEOLOGICAL SETTING AND MINERALIZATION

7.1 Regional Geology

The Donlin deposits lie in the central Kuskokwim Basin of southwestern Alaska, and the basin is northeast-trending which subsided between a series of amalgamated terranes. Rock types within the basin include Mesozoic marine volcanic rocks, Paleozoic clastic and carbonate rocks, and Proterozoic metamorphic rocks.

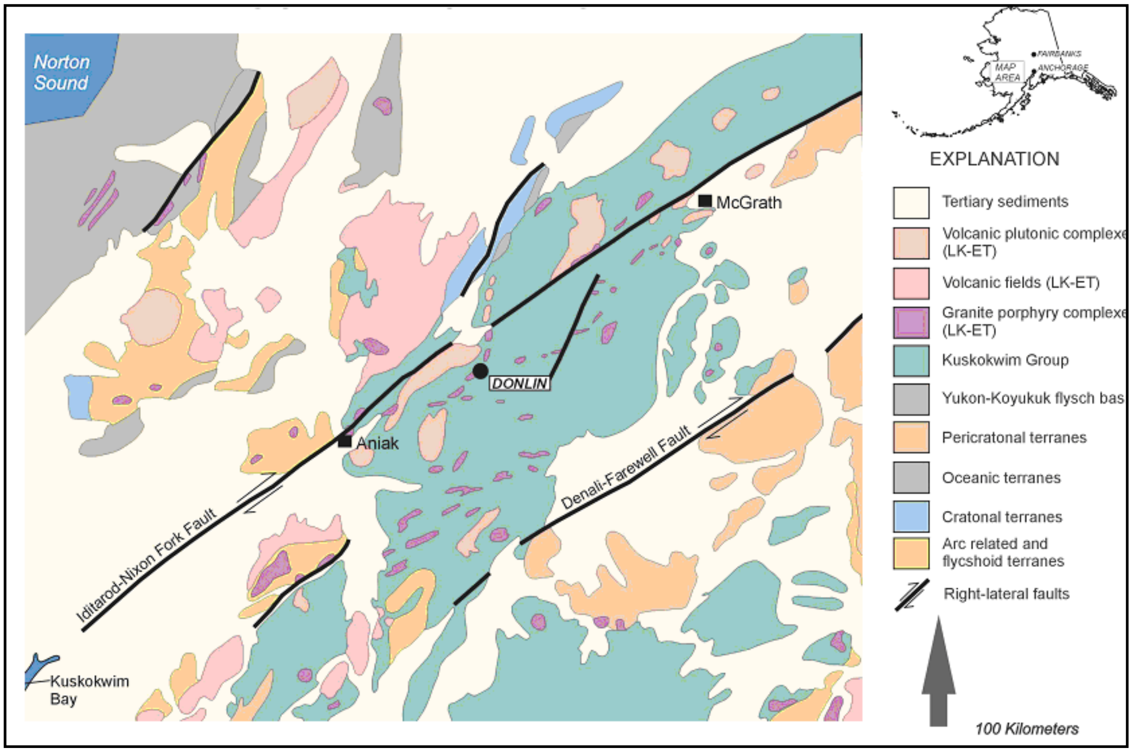
The Kuskokwim Basin is predominately underlain by the Upper Cretaceous Kuskokwim Group, a back-arc continental margin basin fill assemblage that formed in response to a change in the angle of convergence between the Kula oceanic plate and the Cretaceous North American continental margin. Sediments primarily consist of a coarse- to fine-grained turbidite comprising sandstone, siltstone, and shale with minor conglomerate.

Late Cretaceous and Early Tertiary volcano-plutonic complexes intrude and overlie the Kuskokwim Group sedimentary rocks. Volcanic components of these complexes consist of intermediate tuffs and flows. Subaerial volcanic tuffs, flows, and domes are regionally extensive and dominantly andesitic, locally include dacite, rhyolite, and basalt. Associated plutons are calc-alkaline in composition, ranging from monzonite to granodiorite. Felsic to intermediate hypabyssal granite to granodiorite porphyry dikes, sills, and plugs are also widely distributed and often intruded into northeast-striking extensional faults. Volumetrically minor Upper Cretaceous intermediate to mafic intrusive bodies are also common.

The center of the Kuskokwim Basin lies between two continental-scale, dextral slip-fault zones: the Denali–Farewell Fault system to the south and the Iditarod–Nixon Fork Fault system to the north. Fold-and-thrust-style deformation formed the earliest structures in response to subduction-related compression shortly after deposition of the Kuskokwim sediments. Eastward-trending folds and thrust faults are common in the central Kuskokwim Basin, including the Donlin Gold project area. Younger north–northeast-trending folds are dominant near the Iditarod–Nixon Fork Fault and Denali–Farewell Fault but also formed throughout the region in response to basin-scale dextral movement. Most of the folds predate emplacement of the volcano-plutonic complexes. Pre-, syn-, and post-(?) intrusion, northeast-striking normal and oblique slip faults formed during subsequent late compressional and extensional events and focused intrusive igneous rocks and hydrothermal systems across the basin.

A regional geological plan is included as Figure 7-1.

Figure 7-1: Regional Geology of Central Kuskokwim Area



Source: Donlin Gold LLC, 2025

7.2 Project Geology

Fluvial deposits of flood plains, outwash plains, and alluvial fans blanket much of the major river valleys. Gravel deposits occur on rock benches and terraces overlooking flood plains, thought to be of Pliocene age (5.3-2.6 Ma).

Although glaciers formed throughout Alaska in the Pleistocene (2.6 Ma to 11.7 ka), the Project area remained unglaciated. Loess (windblown silt) is widespread below elevations of 300-450 m and was deposited during Illinoian and Wisconsinan time (191-11 ka), derived from outwash plains and river floodplain deposits. Periglacial weathering, permafrost, and solifluction occurred throughout the area, generating extensive deposits from mass wasting and frost action.

Outcrop is limited and of generally poor quality, therefore property-scale geology is largely interpreted from trenches, drill holes, aeromagnetic surveys, and soil geochemistry.

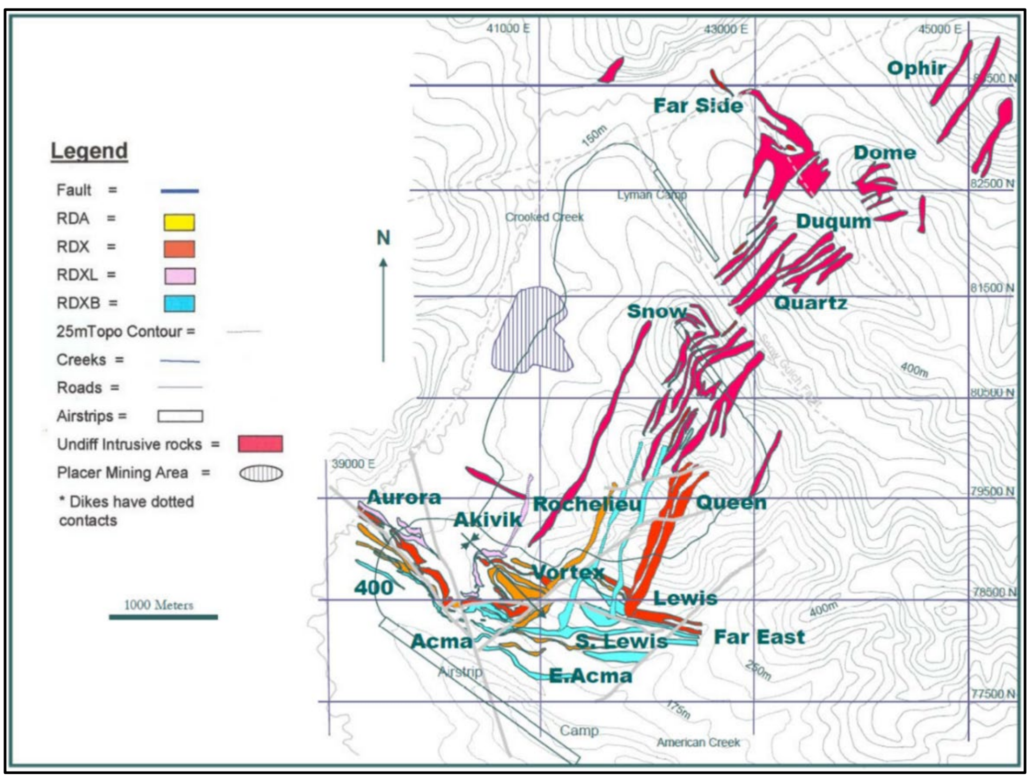
The Project area is underlain by a 8.5 km long x 2.5 km wide granitic porphyry dike and sill swarm hosted by lithic sandstone, siltstone, and shale of the Kuskokwim Group.

7.2.1 Lithologies

The oldest igneous rocks at the Property are intermediate to mafic dikes and sills. They are not abundant but occur widely throughout the Property as generally thin and discontinuous bodies. The younger and much more voluminous granite porphyry intrusive rocks vary from about one meter to 60 m wide and occur as west–northwest-trending sills in the southern resource area and north–northeast-trending dikes farther north. The granite porphyry dikes and sills all have similar mineralogy, and the porphyry texture indicates relatively shallow emplacement. Although these rocks belong to the regionally important granite porphyry igneous event, geologists working on the Property classify them into five textural varieties of rhyodacite. These units are chemically similar, temporally and spatially related, and probably reflect textural variations of related intrusive events.

Figure 7-2 illustrates the interpreted property-scale distribution of igneous rocks, including the mineral resource area between the Queen deposit area on the northeast and the airstrip on the southwest.

Figure 7-2: Interpreted Property-Scale Igneous Rocks



Source: Donlin Gold LLC, 2025
Note: RDA = Aphanitic Porphyry; RDX = Crowded Porphyry; RDXL = Lath-Rich Porphyry; and RDXB = Blue Porphyry.

7.2.2 Structure

The Project is located in a structurally complex area about 25 km southeast of the Iditarod–Nixon Fault (refer to Figure 7-1). Sedimentary bedding generally strikes northwest and dips 10° to 50° to the southwest. Overall, sedimentary structure in the northern resource area is monoclinal, while sedimentary rocks in the southern resource area display open eastward trending folds. East–southeast-trending and plunging folds or monoclinal warps are the oldest recognized structures and are associated with north-vergent thrust faults. Thrust faults are generally southwest-dipping, parallel to the bedding plane, and account for imbrication of the sedimentary rocks and locally moderate to steep southwest and northeast dips. Younger, low-amplitude north–northeast-trending folds crop out in the airstrip exposures along American Creek and are recorded on historical trench geology maps. Lack of cleavage or other evidence of dynamic recrystallization suggests that folds and thrust faults formed at relatively shallow depths.

7.3 Deposit Setting

Within the ACMA-Lewis area, a northeast, elongated, roughly 1.5 km wide x 3 km long cluster of gold deposits has an aggregate vertical range that exceeds 945 m. The deposits are hosted primarily in igneous rocks and are associated with an extensive Upper Cretaceous gold– arsenic–antimony–mercury hydrothermal system. Gold occurs primarily in sulfide and quartz–carbonate–sulfide vein networks in igneous rocks and, to a lesser extent, in sedimentary rocks. Broad disseminated sulfide zones formed in igneous rocks where vein zones are closely spaced. Sub-microscopic gold, contained primarily in arsenopyrite and secondarily in pyrite and marcasite, is associated with illite–kaolinite–carbonate–graphite altered host rocks.

7.4 Paragenesis

Fluid inclusion studies and field and drill hole observations define three distinct styles of gold mineralization that are locally telescoped and cross-cut one another. The earliest is a porphyry-style stockwork vein system at the Dome prospect.

Dome is located within the same dike-and-sill swarm that hosts the ACMA–Lewis resource, but the Kuskokwim sedimentary rocks are thermally metamorphosed to a siliceous hornfels. Quartz veins have a Au–Ag–Cu–Zn–Bi ± Te trace metal signature (Ebert et al., 2003c; Drexler, 2010) with up to 3% arsenopyrite–pyrite–chalcopyrite–pyrrhotite ± Fe-rich sphalerite and trace amounts of electrum, native bismuth, and bismuth tellurides and selenides. Veins cut both the hornfels and porphyry dikes.

ACMA–Lewis-style mineralization post-dates the Dome veins and consists of sparse Au–Ag–As–Sb–Hg ± W (Ebert et al., 2003c; Drexler, 2010), trace metal-bearing quartz–Fe–dolomite veins with <3% auriferous arsenopyrite–pyrite ± stibnite ± late realgar, native arsenic, and graphite. Veins and related disseminated sulfide zones are primarily hosted in illite–carbonate–kaolinite-altered rhyodacite dikes and sills but also occur in Kuskokwim Group sedimentary rocks near igneous contacts.

Variations between Dome and ACMA–Lewis vein habits, vein mineralogy, wall rock alteration, geochemical signatures, stable isotope variations (Drexler, 2010), and fluid inclusion chemistry (Ebert et al., 2003c) indicate that hydrothermal fluids were sourced at depth northeast of the Dome prospect, precipitated the base metal assemblage at Dome from metals sequestered in the vapor phase, and then migrated southwestward to the more distal ACMA–Lewis environment, where gold-bearing minerals were precipitated due to mixing with meteoric waters and boiling.

The last event consists of gold-bearing quartz–stibnite veins up to 1 m thick with variable carbonate, pyrite, and arsenopyrite found mainly around the margins of Dome and partially overlapping ACMA–Lewis. Quartz–stibnite veins also contain anomalous Au–As–Cu–Zn–Bi and have fluid chemistry and temperatures intermediate between Dome and ACMA–Lewis (Ebert et al., 2003). In the opinion of Donlin Gold LLC geologists, these veins do not contain significant gold mineralization.

7.5 Deposit Geology

Most of the detailed trench, road cut, and outcrop maps have not yet been compiled into a geological “fact map” in the resource area. The surface geology illustration in Figure 7-3 is a projection of the 3D geological model of intrusive rock units and faults shown in a perspective view of an orthophoto-draped digital elevation model (DEM) image.

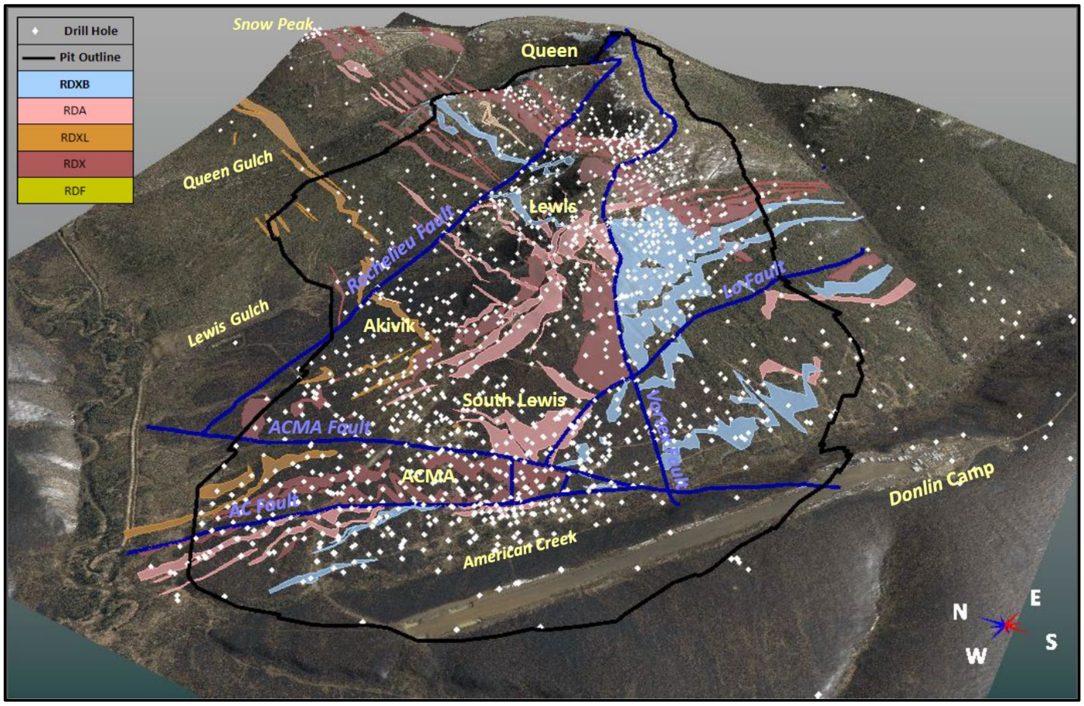
7.5.1 Sedimentary Rocks

Informal sedimentary stratigraphy in the immediate deposit area is shown in Table 7-1. The approximate thicknesses of each unit are from the southern or ACMA resource area.

The stratigraphy in the deposit area consists of basin margin clastic rocks (MacNeil, 2009) dominated by greywacke (lithic sandstone) units with complex transition zones of interbedded siltstone, shale, and greywacke. Marker beds are not yet recognized, so absolute stratigraphic breaks are difficult to identify. Greywacke is dominant in the northern part of the resource area

(Lewis, Queen, Rochelieu, Akivik), whereas shale–siltstone-rich units are common in the southern part (South Lewis, ACMA).

Figure 7-3: Interpreted Surface Geology of Resource Area



Source: Donlin Gold LLC, 2025

Note: Oblique view looking northeastward showing igneous rock units, faults, drill holes, and Mineral Reserve pit outline.

Table 7-1: Donlin Gold Project Stratigraphy

Assigned Nomenclature	Principal Rock Type	Apparent Thickness (m)
Upper Greywacke	Greywacke	100+
Upper Siltstone	Siltstone/shale	50
Main Greywacke	Greywacke	80
Main Shale	Shale/siltstone	Up to 140 (with sills)
Basal Greywacke	greywacke	200+

Note: After Piekenbroke and Petsel (2003)

7.5.2 Igneous Rocks

The mafic igneous rocks and the five textural varieties of rhyodacite recognized in the Donlin deposits were also shown in Figure 7-3. Table 7-2 lists the intrusive rocks from oldest to youngest.

Table 7-2: Donlin Gold Project Stratigraphy

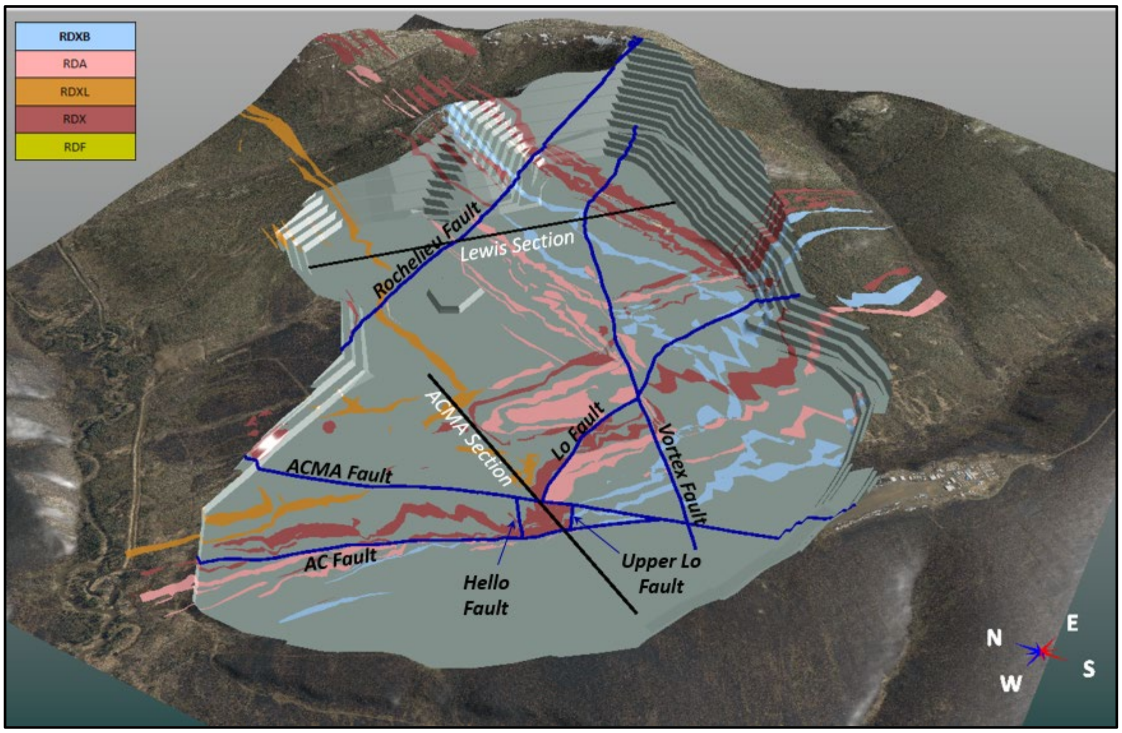
Name	Code	Age	Rock Types
Mafic Dikes/Sills	MD	Oldest	Intermediate to mafic dikes and sills; locally host high-grade gold; generally less than 3 m thick. In the transition area between Akivik and ACMA, mafic sills are extremely abundant within the Lower Greywacke, immediately below the Main Shale.
Fine-Grained Porphyry	RDF	-	Earliest rhyodacite intrusions recognized. Grey, typically fine-grained, felsic porphyries. RDF intrusives occur as two main northeast-striking, 5 to 10 m wide dikes in the Lewis zone and possible discontinuous bodies in early eastward-trending compressional faults, e.g., the Lo Fault
Crowded Porphyry	RDX	-	Volumetrically the most significant intrusive phase. Grey, characterized by a uniformly crowded feldspar porphyry texture. Present as two 50 to 100 m wide dike zones in the eastern edge of the north to north-northeast mineralized trend of Lewis/South Lewis. RDX is also found as sills throughout ACMA near the basal part of the sill sequence.
Lath-Rich Porphyry	RDXL	-	Characterized by sparse, elongate plagioclase laths; significant coarser-grained biotite. Occurs as two important dikes in the Akivik area that strike south into the center of the ACMA deposit. In Akivik and ACMA, RDXL occurs as a significant sill immediately below the RDX sill. The RDXL sill continues to the west but pinches out to the east. RDXL dikes are also present within the main Lewis area RDX dike trend, but here they are volumetrically insignificant.
Aphanitic Porphyry	RDA	-	Rhyodacite rock with a salt-and-pepper texture of fine biotite phenocrysts and variable quartz and potassium feldspar phenocrysts. Numerous (up to eight) RDA dikes strike south from the Vortex/Rochelieu (Lewis) area into the East ACMA/ACMA area. The dikes are typically found west of the Vortex Fault but are also present between the Lo and Vortex faults and below the Lo Fault. An extensive sill package of RDA lies immediately above the RDX sills in the ACMA area. In West ACMA, the RDA sills are buttressed against, and locally cross-cut, RDX sills. Another package of RDA sills is found south of the AC Fault, in the Aurora domain.
Blue Porphyry	RDXB	Youngest	Final intrusive event; coarsely porphyritic with large blocky feldspars set in a graphite- and sulfide-rich matrix. Locally hosts important high-grade disseminated sulfide material in addition to gold-bearing veins. RDXB occurs as two major dikes, the Lewis Blue Porphyry dike and the Vortex Blue Porphyry dike. Extensive thin RDXB sills are found in the uppermost part of the sill sequence in the South Lewis and ACMA areas, and RDXB sills are present as both distinct sills and co-mingled with RDA in the core of ACMA and in the Aurora domain.

Note: After Piekenbroke and Petsel (2003)

7.5.3 Structure

The morphology of intrusive rocks in the deposit is largely governed by the rheology of sedimentary rocks and pre-intrusion faults and folds. Faults in the geological model (from earliest to youngest) are the American Creek (AC) Fault, Lo and Rochelieu faults, Vortex Fault, and ACMA Fault. Figure 7-4 shows an oblique view of the faulting in the deposit area, and cross-sections through the ACMA and Lewis areas, respectively in Figure 7-5 and Figure 7-6.

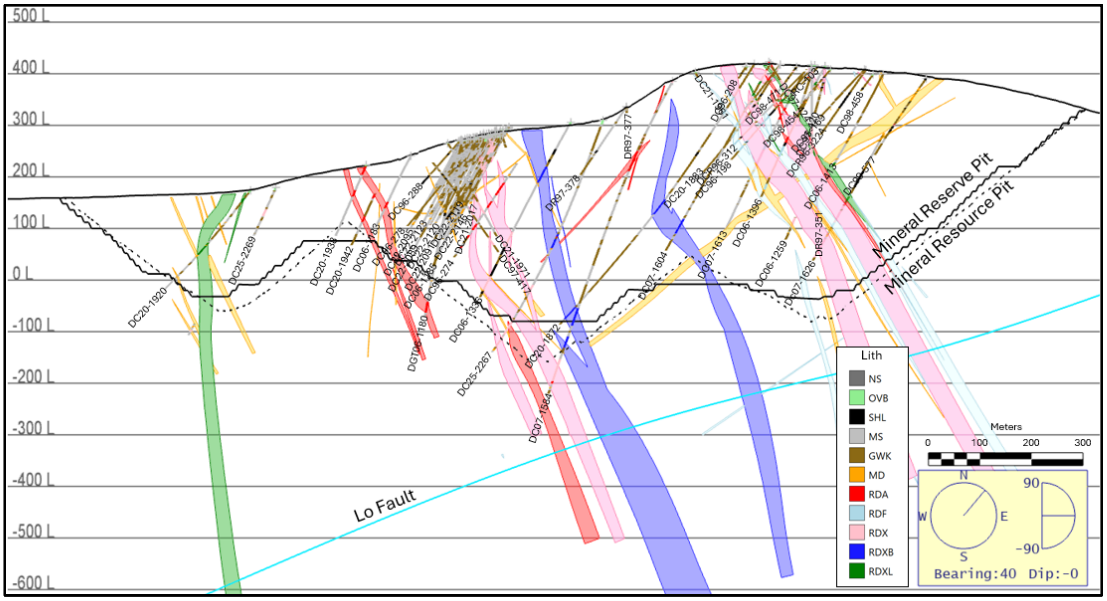
Figure 7-4: 100 m Bench Level Geology



Source: Donlin Gold LLC, 2025

Note: Oblique view looking north-eastward of the 3D geological model projected on the 100 m pit bench level and the Mineral Reserve pit outline.

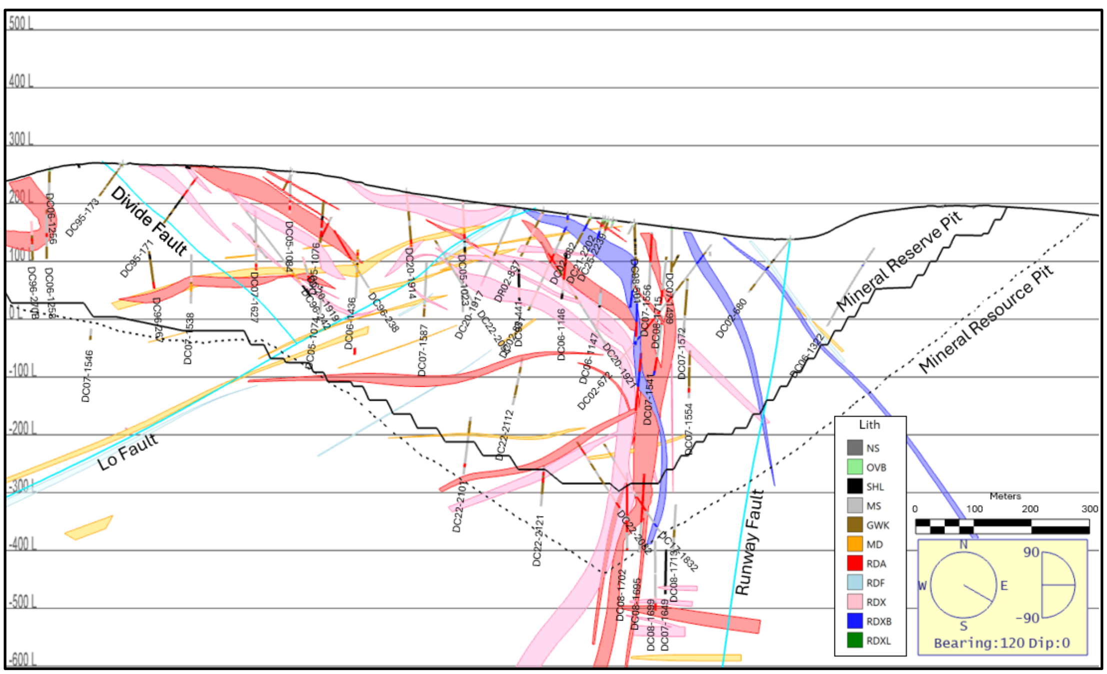
Figure 7-5: Lewis Area Section



Source: Donlin Gold LLC, 2025

Note: Shows intrusive rocks, faults, drill holes, Mineral Resource pit and Mineral Reserve pit, looking northeast.

Figure 7-6: ACMA Area Section



Source: Donlin Gold LLC, 2025

Note: Shows intrusive rocks, faults, drill holes, Mineral Resource pit and Mineral Reserve pit, looking southeast.

7.6 Deposits

The Donlin deposits include eleven mineralized areas that exhibit slightly different geological settings but generally fall into two geologically similar deposit areas: ACMA and Lewis. ACMA, or the intrusive sill and shale–siltstone sedimentary setting, includes the Aurora, 400, Akivik, ACMA, and East ACMA mineralized zones. Lewis, or the massive greywacke-hosted intrusive dike setting, includes the South Lewis, Lewis, Vortex, Rochelieu, Queen, and North Akivik mineralized zones.

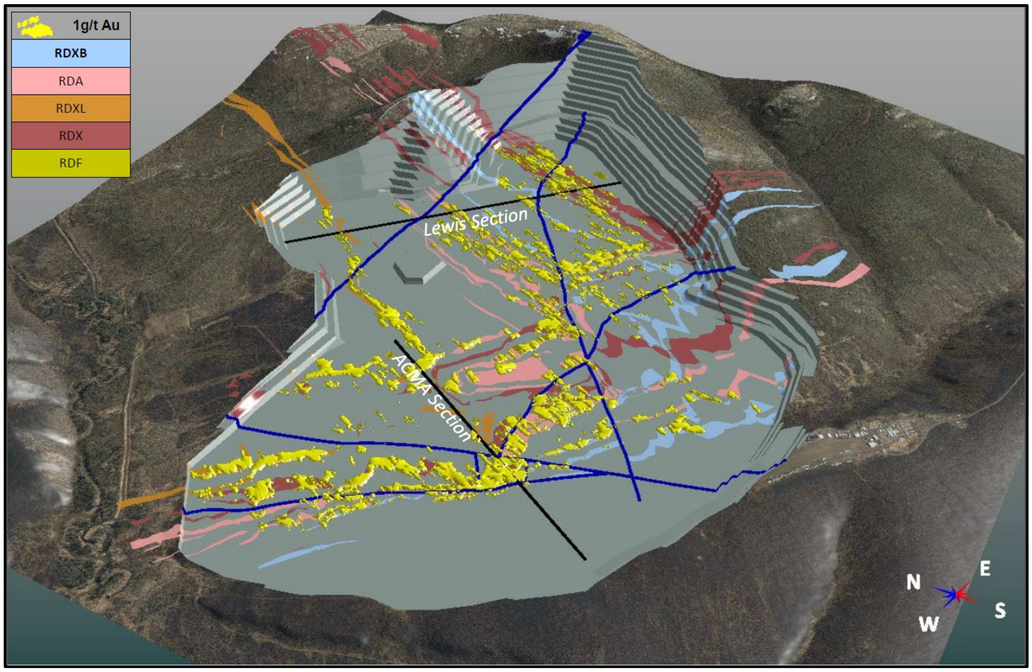
Veins in north–northeast-striking, east- or west-dipping faults and fracture zones are the primary control on gold distribution and are ubiquitous in all mineralized areas. Northwest- and northeast-striking veins occur locally but are relatively rare. Veins are narrow (typically <1 cm wide), highly irregular, discontinuous, and generally sparsely distributed, although vein density can locally range up to two to eight per meter in higher-grade zones. Vein zones vary from 2 to 35 m wide and 100 to 350 m long. Individual vein zones generally display limited lateral and vertical continuity; however, swarms of many anastomosing vein zones form larger mineralized corridors characterized by extensive lateral and depth continuity.

Vein corridors are more apparent in the north–northeast-trending dikes of Lewis than in the west–northwest-trending ACMA sill zone. The greater width of the sill-hosted ACMA mineralized zone makes discreet corridors less obvious (but still present). Mineralized zones follow steeply dipping dikes and sills beyond the depth limits of current drilling, or over a vertical range of at least 945 m.

Veins are best developed in relatively more brittle intrusive rocks and massive greywacke. Small, irregular, carbonate-altered mafic bodies often host very high-grade gold as sulfide dissemination, replacement, and breccia fill. Structural breccias in sedimentary rocks are also favorable sites for high-grade gold. Gold distribution in the deposit closely mimics the intrusive rocks. The more steeply dipping sills in the ACMA sill sequence host the highest-grade and most continuous igneous-hosted mineralized zones, particularly where intersected by northeast-striking "feeder" dikes and faults. Gold grade is directly proportional to vein density and intensity of overlapping disseminated sulfide vein aureoles. The dike-dominant Lewis deposit areas consist of sheeted veins with limited disseminated sulfide in the wall rocks and are characterized by lower-grade and less continuous mineralized zones.

Gold distribution in the planned pit area is shown in Figure 7-7, as a bench plan.

Figure 7-7: 100 m Bench Level Gold Distribution (>1 g/t Au grade blocks)



Source: Donlin Gold LLC, 2025

7.7 Mineralization

Gold-bearing zones are coincident with quartz–carbonate–sulfide veins and related disseminated sulfide aureoles in hydrothermally altered rhyodacite bodies and, to a lesser extent, in sedimentary rock near igneous contacts. Continuity and grade of mineralized material within the rhyodacite host rocks varies directly with vein spacing and the amount of vein and disseminated arsenopyrite, the principal gold-bearing mineral. Gold in sedimentary rocks and minor mafic igneous bodies is generally limited to small and discontinuous vein and breccia fill occurrences.

7.7.1 Vein and Disseminated Mineralization

Veins in the ACMA–Lewis area are subtle in appearance and vary from <1 mm to 20 cm wide, averaging <1 cm. They formed in brittle fractures and are typical of open-spaced fillings with vugs, drusy quartz-lined cavities, vein wall-banded and cockscomb quartz, and bladed carbonate. Veins are composed of gray to clear quartz, white to tan carbonate, and as much as 3% sulfides. Table 7-3 contains a summary of the gold-bearing vein stages.

Table 7-3: Vein Stages

Vein	Description
V1	Thin, irregular, and discontinuous sulfide (>50%) veins with pyrite and trace arsenopyrite, little or no quartz (<30%) or carbonate (<50%). Broad disseminated selvage of pyrite and poorly crystalline illite and Fe–carbonate alteration. Barren or very low grade.
V2	Thin, discontinuous quartz (>30%) sulfide veins contain variable pyrite and arsenopyrite. May have broad, often pervasive selvages of fine-grained, needle-like arsenopyrite. Broad pyrite aureole may surround the arsenopyrite selvage. Open-space vuggy textures common. Trace amounts stibnite. Have moderate gold grade and strong illite alteration aureoles with variable Fe–carbonate replacement of the host rock.
V3a	Higher-grade veins. Thicker, more planar and continuous, open-space quartz veins with Fe-dolomite, pyrite, arsenopyrite, native arsenic, and variable amounts of stibnite. Commonly show broad arsenopyrite-rich selvages with little to no Fe–carbonate as wall rock alteration.
V3b	Thicker, more continuous, and planar quartz veins with open-space textures and complex mineralogy, including pyrite, arsenopyrite, stibnite, native arsenic, realgar, and trace other sulfides in intensely illite altered material. Gold grades are commonly much higher than the average grade of the deposit.
V4	Latest vein phase. Barren carbonate-quartz (>50% and <50%, respectively) vein sets that post-date mineralized veins. Primarily barren white and clear quartz veinlets and calcite ± ankerite veinlets with no sulfides.

Mineralized zones are consistently oriented sub-parallel to the main δ1 axis (024) of the compressive structural regime (Piekenbrock and Petsel, 2003). Veins in the ACMA–Lewis resource evolved through a continuum (V1 through V3) of changing mineralogy and increasing gold grade while maintaining a generally consistent north-northeast strike and southeast dip. The final carbonate–quartz vein set (V4) has a broader range of orientation.

MacNeil (2009) found that the average vein orientation for all veins is 024/71 degrees. This orientation is generally consistent across all domains and vein types, which indicates that veins in the Donlin deposits formed during the same mineralizing event.

A comparison by host rock shows that veins in igneous rocks strike more easterly and dip more steeply than veins in sedimentary rocks, probably due to refraction across lithologic contacts.

Several quartz and carbonate phases have been recognized, including pre-gold-stage Mn–calcite veins and wall rock replacement and cockscomb quartz veins; Fe–dolomite in main gold stage veins; and post-gold-stage clear quartz veins and ankerite stringer veins.

Euhedral and porous replacement pyrite are the earliest sulfide phases, followed in order by marcasite, arsenopyrite, realgar, and native arsenic. Stibnite is most abundant in later veins. Most accessory sulfides are relatively early, while boulangerite is relatively late. Arsenopyrite occurs as both coarse (up to 1 cm) crystals and very fine (0.1 to 0.2 mm) euhedral grains. Fine-grained arsenopyrite contains five to 10 times more gold than the paragenetically earlier coarse-grained phase.

7.8 Alteration

Rhyodacite bodies are ubiquitously altered to an illite–carbonate–kaolinite–chlorite / smectite ± quartz ± graphite assemblage.

Mafic igneous rocks are strongly altered by carbonate ± fuchsite and contain locally high-grade gold with disseminated, massive replacement or breccia filling sulfide.

Altered sedimentary rocks consist of relict quartz grains in a matrix of illite, kaolinite, carbonate, hematite, and <1% pyrite and trace sphalerite (Drexler, 2010).

Pyrite is widespread in all altered rocks (0.5% to 2%) but is more abundant (1% to 4%) in mineralized zones. Alteration is most intense near veins and is typically zoned outward from illite ± kaolinite to kaolinite ± illite and then to a distal zone of chlorite ± smectite ± quartz.

Silica is dominantly restricted to veins in the ACMA–Lewis area and is not generally expressed as pervasive silicification. Vein relationships show an increase in quartz content from early sulfide-dominant veins to late silica-dominant veins. Some increased silicification has been noted in the Queen area (Ebert et al., 2003b).

Short-wave infrared reflectance (SWIR) spectroscopy data, collected between 2007 and 2011, are interpreted by Donlin Gold LLC geologists to show that higher-grade gold is most strongly correlated with an alteration suite dominated by NH4–illite (ammonium–illite), whereas kaolinite-bearing zones contain lower-grade gold.

7.9 Minor Elements

The most abundant minor elements associated with gold-bearing material are iron, arsenic, antimony, and sulfur. These are contained primarily in the mineral suite associated with hydrothermal deposition of gold, including pyrite, arsenopyrite, realgar, native arsenic, and stibnite. Minor hydrothermal pyrrhotite and marcasite, and syngenetic or sedimentary pyrite, also account for some of the iron and sulfur.

Much less abundant elements such as copper, lead, and zinc are contained in relatively rare or accessory hydrothermal mineral species observed in the deposit, including chalcopyrite, chalcocite, covellite, tennantite, tetrahedrite, bornite, native copper, galena, sphalerite, and boulangerite. Small amounts of silver in the deposit are most likely accommodated within the crystal structures of tetrahedrite and galena, and to a lesser extent in some of the other sulfides. Molybdenum occurs in rare molybdenite. Very minor nickel has been observed in the secondary sulfide mineral millerite and minor cobalt in various secondary minerals in sedimentary rocks. The nickel and cobalt probably have a sedimentary origin.

Three elements of particular processing significance are mercury, chlorine, and fluorine. Graphitic carbon and carbonate minerals also have the potential to negatively affect the metallurgical process.

8.0 DEPOSIT TYPES

According to Donlin Gold LLC geologists, the Donlin deposits share characteristics of several gold deposit genetic models. It has been classified as:

- Granite porphyry-hosted gold polymetallic (Bundtzen and Miller, 1997)
- Distal or high-level epizonal intrusion-related (Hart et al., 2002)
- Low-sulfidation epithermal (Ebert et al., 2003a)
- Orogenic- or intrusion-related (Goldfarb et al., 2004)
- Reduced porphyry to sub-epithermal Au–As–Sb–Hg (Ebert et al., 2003c; Hart, 2007).

Hart (2007) classifies the deposit as a high-level, reduced intrusion-related vein system to account for the reduced ilmenite series intrusions, near contemporaneous age of mineralization, and the apparent genetic relationship to the higher-temperature hydrothermal system at Dome (Drexler, 2010).

The ACMA-Lewis part of the district is clearly a low sulfidation, reduced intrusion related, epizonal system with both vein and disseminated mineral zones and conforms most closely to the Hart (2007) classification.

9.0 EXPLORATION

A summary of the exploration programs completed on the Property is summarized in Table 6-1.

9.1 Grids and Surveys

Historically, the Property has used a designated Geologic survey datum. The "Geologic Datum" was Universal Transverse Mercator (UTM) Zone 4 (meters), NAD83/NGVD29.

In 2023, Donlin Gold LLC contracted Brice Environmental Services Corporation to convert all geologic data to UTM, Zone 4 (meters), NAD83(2011)/NGVD88; allowing an alignment with Engineering work and more precise locations. These transformed data were field checked against survey monuments and known collar locations to confirm successful transformation into the new datum.

9.2 Geological Mapping

Geologic mapping of the Project area has occurred both locally and on larger, quadrangle scales but, due to the natural terrain, has provided a limited amount of detail. The Donlin prospect is partially hosted in low-lying wetland terrains with limited and poor-quality outcrops. Shallow water tables, particularly in the ACMA portion of the deposit, lead to intense oxidation. Upland portions of the deposit, such as the Lewis area, are more accessible but are mostly vegetated with little to no naturally occurring outcrops. Successful mapping programs utilized support from more detailed data obtained from trenches and core drilling.

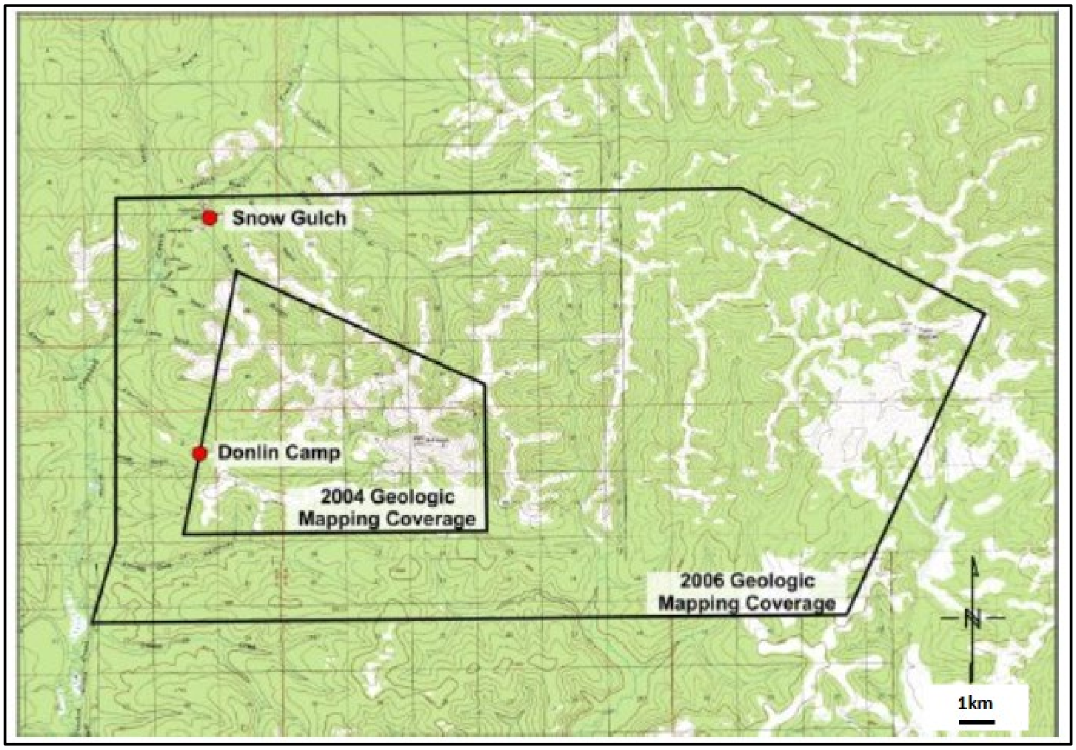
Localized geological mapping was performed by WestGold between 1988-1989. Reconnaissance mapping was undertaken by Placer Dome Inc. from 1996 to 1998. In 1999, Placer Dome Inc. completed a 1:10,000 scale geological mapping program over the entire Project area. In 2004 and 2006, Donlin Gold LLC conducted a geologic mapping exercise covering approximately 285 square kilometers east of Donlin Creek (Figure 9-1) to identify and track the distribution of calcareous sedimentary horizons.

In 2009, Donlin Gold generated an interpretive surface geologic map of rhyodacite intrusive rock units within the Project area given data and interpretations available at the time (Figure 7-2). The map was generated from data captured both during drilling and from historical surface mapping efforts and displays the distribution of the five visually distinct rhyodacite units described in Section 7.

After 2006, physical surface mapping efforts have been restricted to trench, road, and drill pad excavations. These small area maps have not been consolidated into a single map repository but will be discussed in subsequent sections.

Mapping is generally limited by the poor quality and limited extent of outcrop. Information from the mapping programs was used to support more detailed data obtained from trenches and core drilling.

Figure 9-1: 2004 and 2006 Mapping Campaigns



Source: Buntzen and Laird, 2008

9.3 Geochemical Sampling

From 1988 to present, 28,422 geochemical samples have been collected by the property operators as part of regional prospectivity evaluations. A summary of geochemical sampling organized by the different project operators through time can be found in Table 9-1. Sample types included: rock chip samples, stream sediment samples, soil samples, and biogeochemical samples.

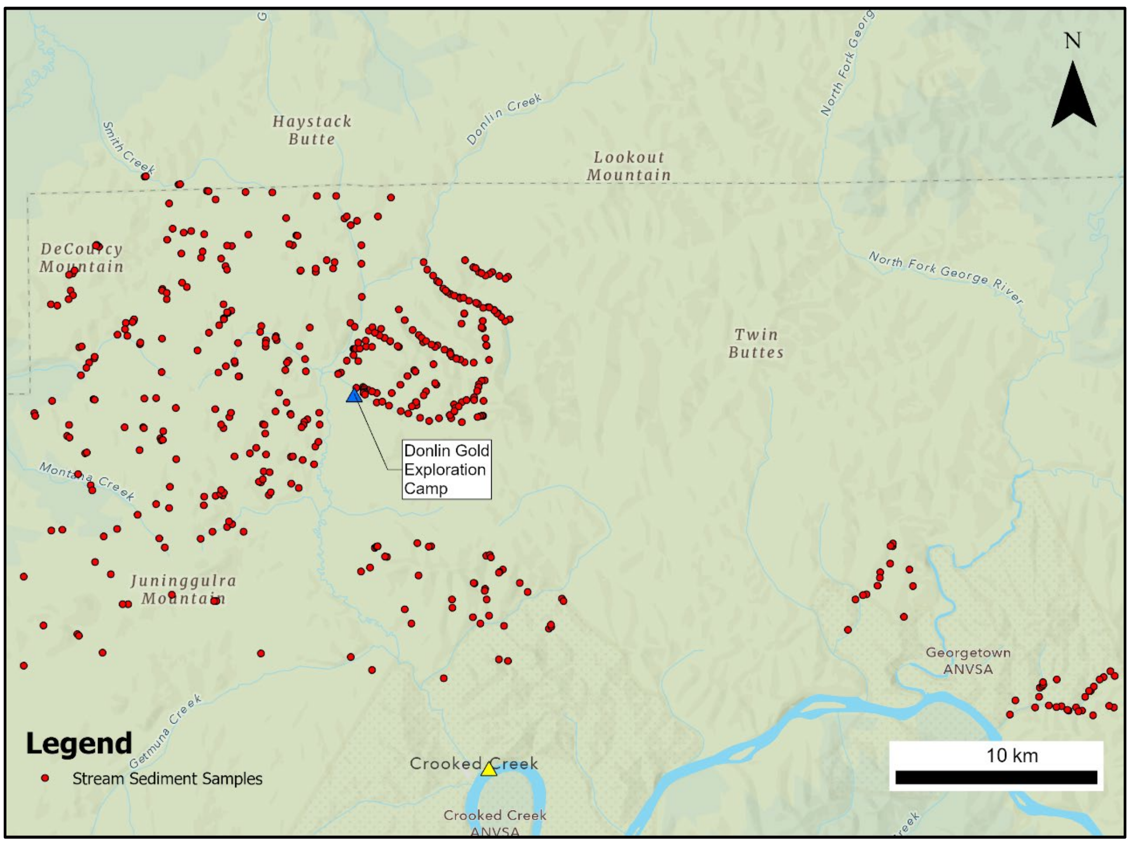
Stream sediment samples were collected along numerous drainages as part of an exploration campaign by Placer Dome in 1998 and by Donlin Gold LLC in 2006, 2008, and 2010 (Figure 9-2).

Table 9-1: Geochemical Sampling Summary

Year	Company	Auger	Rock*	Sediment	Soil
1988-1989	West Gold	-	3,662	-	8,713
1993-2005	Placer Dome Inc.	-	5,167	240	2,727
2006-2025	Donlin Gold LLC	107	1,776	440	5,590
Total		**107**	**10,605**	**680**	**17,030**

Note: * Sample type identified as "Rock" includes trench samples.

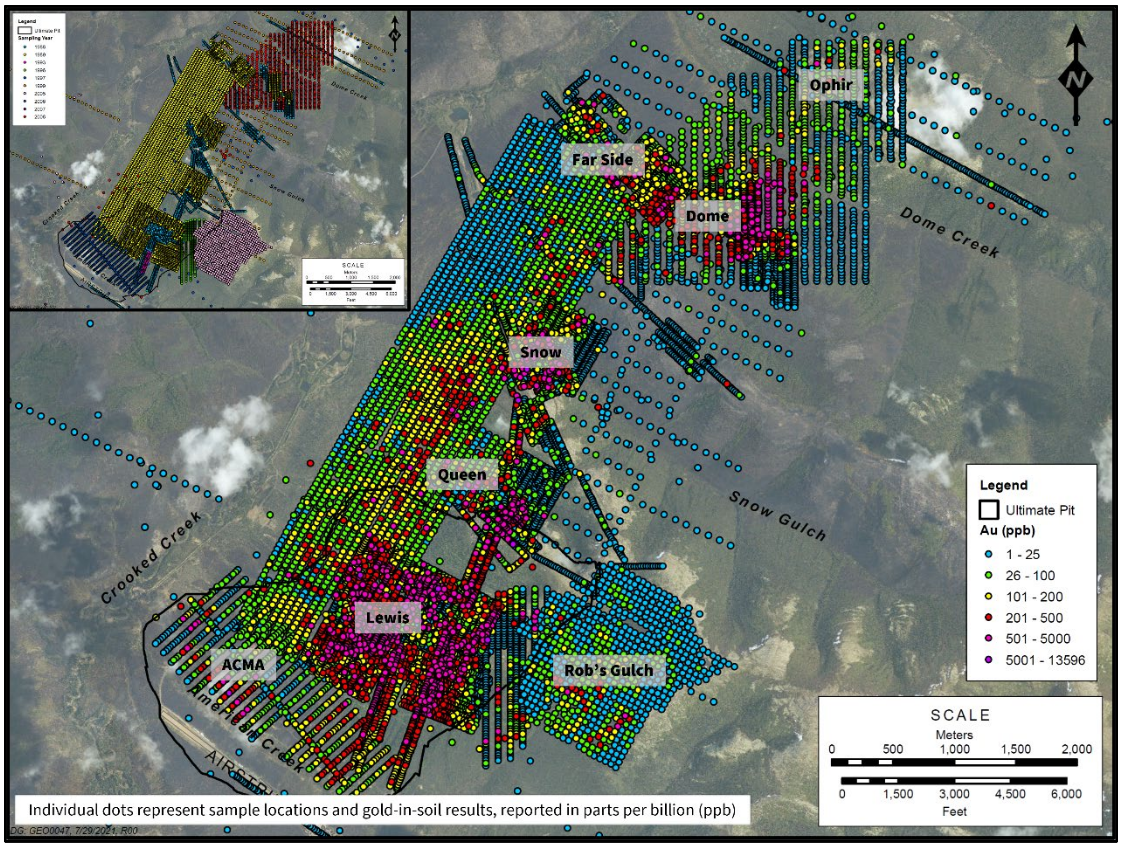
Figure 9-2: Map of Stream Sediment Sampling Locations



Source: Donlin Gold LLC, 2025

Extensive soil sampling campaigns were conducted on the Property in efforts to locate and determine the extents of mineralization (Figure 9-3). Samples collected along ridgetops or higher elevations show more continuity of mineralization than samples collected in low-lying areas due to the amount of overburden and difficulty retrieving viable samples in wetland terrain. Results from analysis of the soil samples show gold-in-soil anomalies over the Project area. Gold-in-soil anomalies were also identified on prospects to the north of the Project area, along trend of known mineralization. Prospects near the Donlin deposit with soil sampling data include Queen, Snow, Rob's Gulch, Far Side, Dome, and Ophir.

Figure 9-3: Overview of Soil Sampling Campaigns Completed on the Property



Source: Donlin Gold LLC, 2025

Note: Individual dots represent sample locations and gold-in-soil analytical results, reported in parts per billion (ppb). The 2025 soil sampling campaign has not been included as results were pending at time of writing.

9.4 Geophysics

WestGold performed an airborne magnetic and very low frequency (VLF) survey and ground magnetic surveys during 1998-1999. The company also trialed ground-penetrating radar.

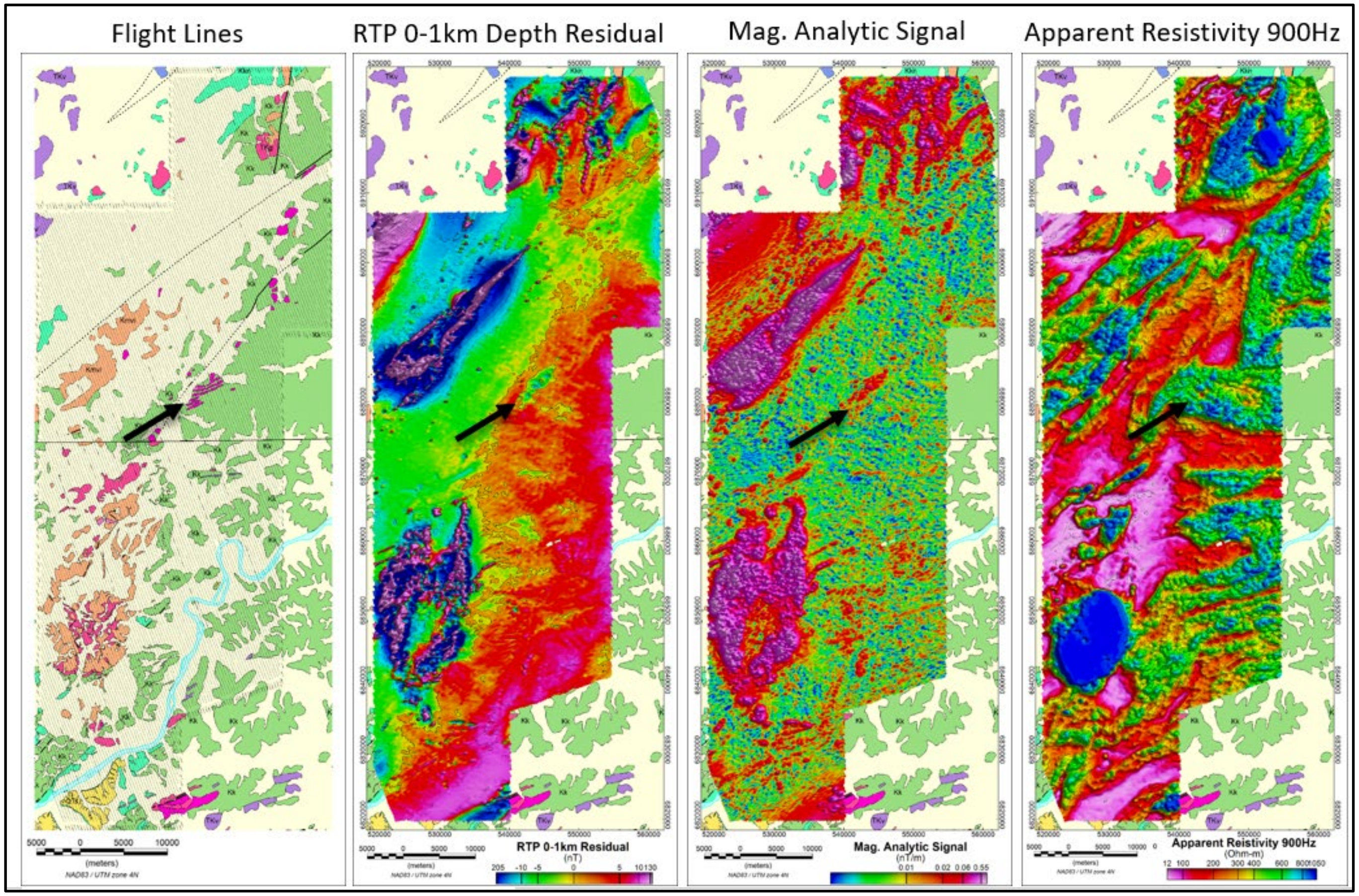
Placer Dome Inc. completed four geophysical surveys from 1995-1998 using an in-house Minimag system. In 1995, the Main ACMA to Dome area was surveyed with 200 m line spacing at an azimuth of 300º at 100 m height. In 1996, the survey extended from Dome to the northeast and was captured with the same parameters as 1995. In 1997, a higher resolution survey was conducted over the Main ACMA to Dome area with 50 m line spacing at 300° at 100 m height. In 1998, the survey area was extended to the southwest with 140 m line spacing at 300° at 70 m height.

During 1999, Placer Dome Inc. completed 25 km of max-min electromagnetic (EM) geophysical survey in the ACMA and southern Lewis areas and 1,800 km of aeromagnetic survey at 50 m line spacing and 50 m elevation, and 17.7 km of induced polarization (IP) and resistivity lines. In 2000, an additional 41.6 km of IP/resistivity lines were completed.

In 2000, the state of Alaska flew a frequency-domain multi-coil system (known as DIGHEM) airborne electromagnetic/resistivity/magnetic survey over the Iditarod Quadrangle, which includes the Donlin Property (Figure 9-4). The survey consisted of approximately 9,186 line-km, including 964 line-km of tie lines. The nominal line separation is approximately 400 m on azimuth 340° at 30 m height. Tie lines were flown perpendicular to the flight line direction with a separation of 5 km (Stephens, 2000).

In 2021, historic aerial geophysics data was compiled and re-interpreted using a new 3D magnetic vector inversion for a core subset of merged 1995, 1996, and 1998 magnetics (Figure 9-5). Vector inversion accommodates remanent magnetization, providing the strength and direction of magnetization. The re-interpretation utilized 50 m x 50 m x 25 m cell sizes to an approximately 4 km depth. The interpretation of the data suggests two different intrusive styles; the strongest response from reverse-polarity characterizes the rhyodacite dike corridor between ACMA and Dome, although evidence of demagnetization is present in the ACMA portion of the deposit. The second intrusion style interpreted from this work is a normal-polarity, deep-rooted intrusion, located both north of Dome and west of ACMA (Williams, 2021).

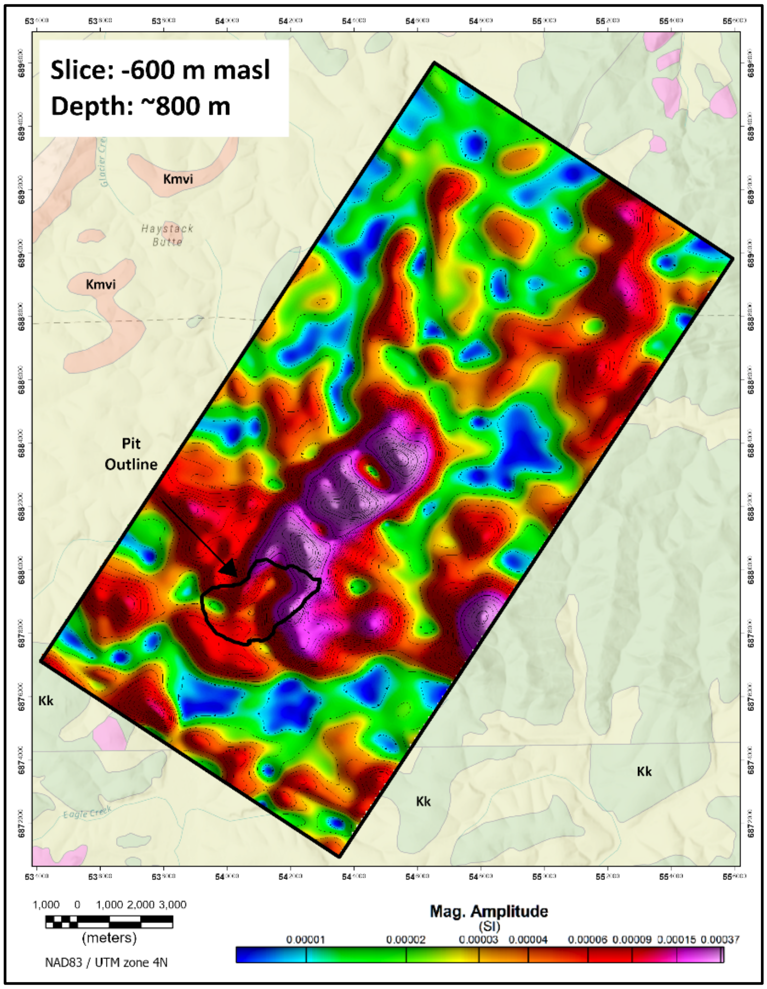
Figure 9-4: Fugro DIGHEM Geophysical Survey Flown Over Parts of the Sleetmute and Iditarod Quadrangles



Source: Modified after The State of Alaska, 2000

Note: Black arrow indicates Donlin Gold LLC Property location

Figure 9-5: Plan View of Magnetic Vector Inversion Model Sliced at -600 masl



Source: Modified after Williams, 2021

9.5 Excavations and Trenches

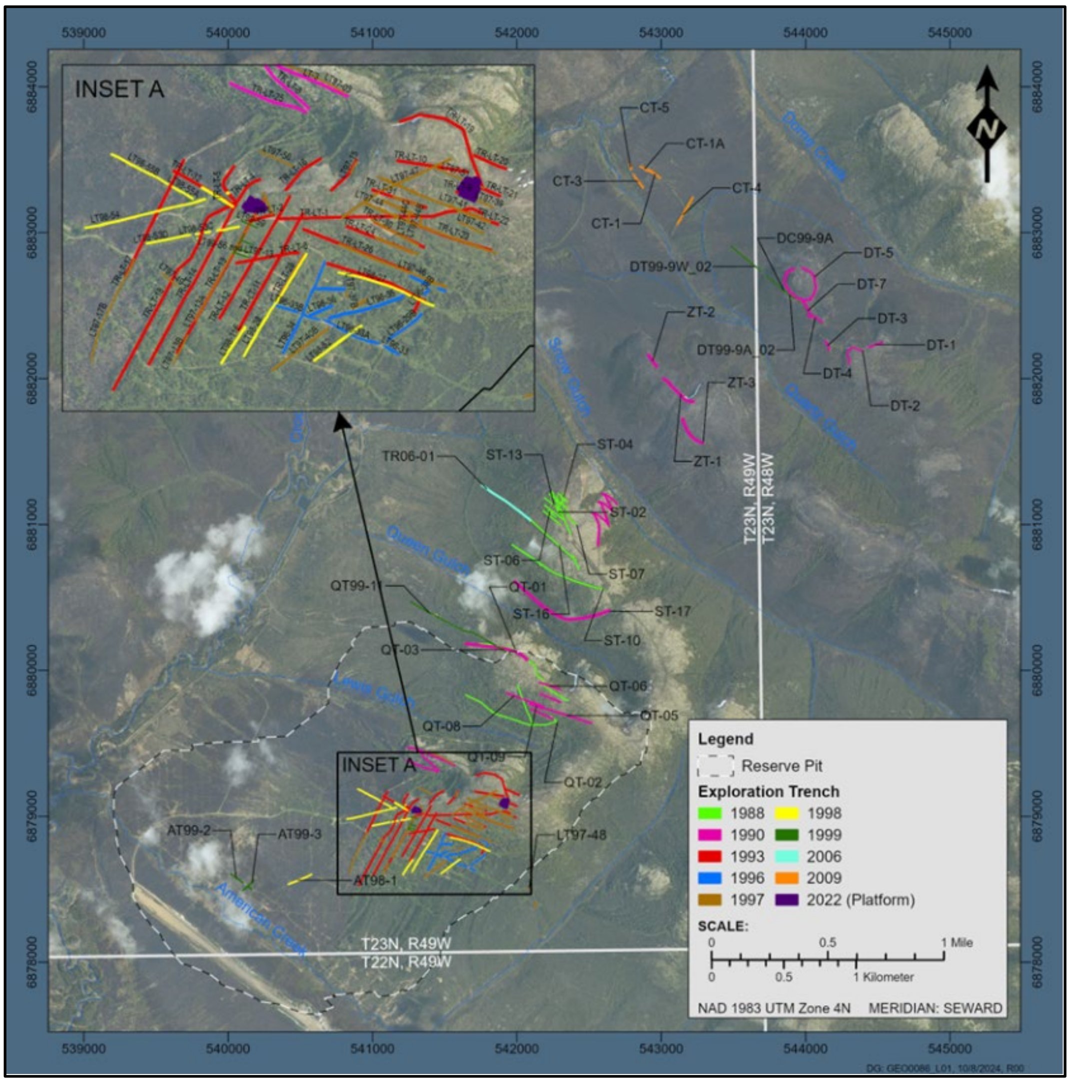
The lack of natural outcrops within the Property, resulting from thick vegetated cover of bedrock in wetland and tundra habitat, and mild topographic relief, necessitates localized excavations for the purpose of geologic mapping and sampling. To date, a total of 26,587 linear meters of trenches were excavated between 1988 and 2022 as part of exploration programs conducted by WestGold, Placer Dome, and Donlin Gold LLC (see Table 9-2 and Figure 9-6).

Table 9-2: Summary of Trenches Completed from 1998 to 2022

Year	Company	No. of Trenches	Meters	Prospects
1988-1989	WestGold	185	14,173	Lewis, Queen, Snow, Wheetie
1993-1999	Placer Dome Inc.	166	11,029	Lewis, Lewis-Vortex, ACMA, Dome, Queen, Far Side
2020-2022	Donlin Gold LLC	32	1,385	Lewis, Divide, Far Side
Total		**383**	**26,587**	

Note: Work completed by either NOVAGOLD or Barrick after the development of the joint venture, Donlin Gold LLC., is recorded under Donlin Gold LLC.

Figure 9-6: Summary of Trenches Completed from 1998 to 2022



Source: Donlin Gold LLC, 2025

9.5.1 Legacy Excavations and Trenches

Excavations conducted by previous operators from 1988-1999 were linear and comprised excavations that were a few meters wide and up to 400 m long and created with small 20 -tonne class excavators and/or 10 -tonne class dozers. Geological data such as lithology information, structure type and orientation data, and geochemistry and assay data were collected from these excavations. These trenching activities are summarized in Table 9-2.

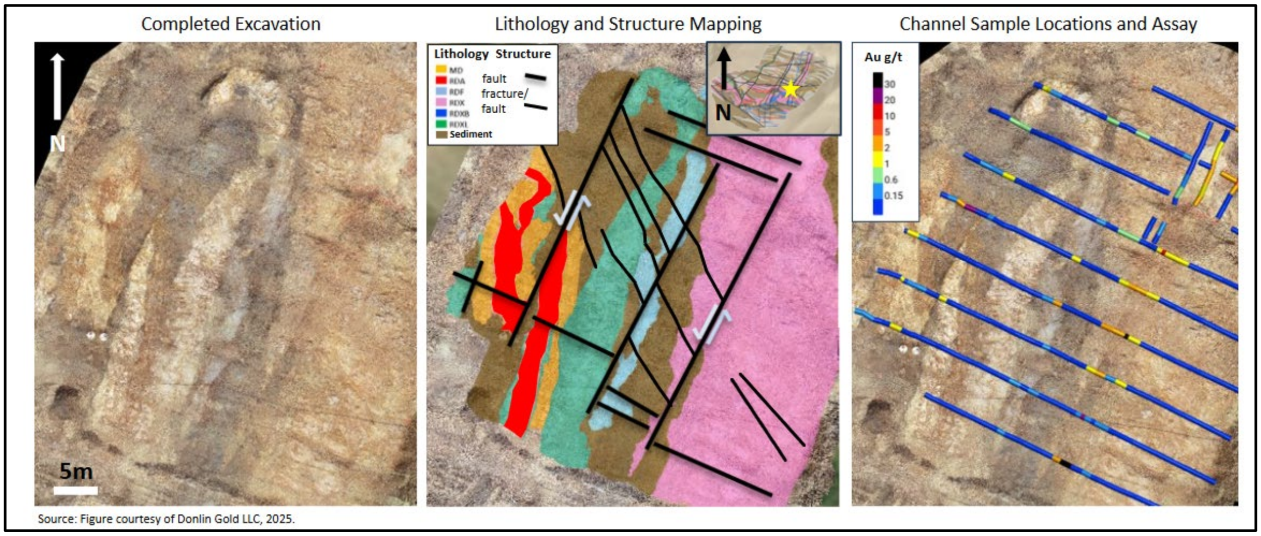
9.5.2 Recent Excavations and Trenches

A multi-year, phased, trenching and excavation program on Lewis Hill (within the Lewis area) was undertaken by Donlin Gold LLC from 2020 to 2022. In 2020, a 30 m x 100 m trench was designed to crosscut modeled mineralized features and excavated for the purpose of locating, characterizing, and collecting samples from the modeled features. In 2021, the second phase of the Eastern Lewis Hill trench was designed along strike length of the mineralized feature unearthed in 2020 and a 23 m x 80 m trench was excavated perpendicular to the original. In 2022, Donlin Gold LLC excavated the third and final phase of the Eastern Lewis Hill Trench (Figure 9-7) and an additional excavation, the Divide Trench, which is located on the western flank of Lewis Hill (Figure 9-8). The objective was to excavate in a different way relative to the legacy trenches (Section 9.5.1) to allow for true 2D representation of mineralization continuity. To this end, an approximately 60 m x 50 m surface area was excavated to improve geologic understanding of lithologic relationships, structures and mineralization continuity. All overburden and unconsolidated material overlying bedrock were cleared and the exposed bedrock was swept and pressure washed clean for detailed geologic mapping and sampling.

The Lewis Hill excavation provided insight into the complexity of the prospect (Figure 9-7). Five intrusive lithologies (RDX, RDF, RDXL, RDA and mafic dikes) were exposed in addition to the Kuskokwim sediments. A dominantly 020° trending set of structures with orthogonal cracking was mapped across the exposure, with lesser structures trending ~290°. The mapped mineralization was generally associated with the 020° structures. The excavation was sampled by laying out 1-m channels perpendicular to the expected mineralization trend, every 6 m. The channels were divided into bins less than 2 m wide based on their geology. Bulk samples, weighing approximately 2.5 kg were taken from each bin and sent for laboratory analysis.

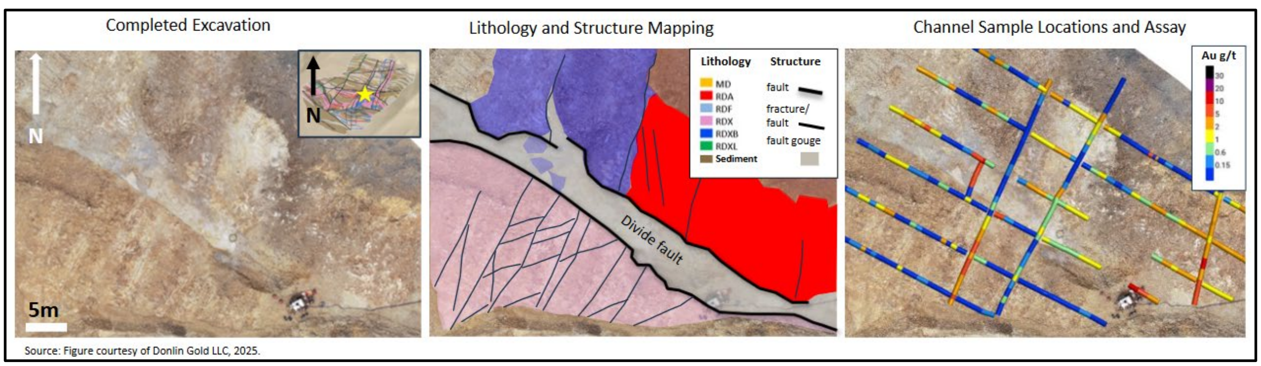
The Divide Trench location was selected to better understand the structural relationship between the Lewis domain and the ACMA domain (Figure 9-8). At the center of the excavation, the Divide Fault was exposed and serves as the contact between three different intrusive units; RDX on the west, RDXB and RDA on the east. The fault strikes approximately 125° and is intersected by several 020° mineralized structures, similar to those present in the Lewis Trench.

Figure 9-7: Lewis Hill Trench



Source: Donlin Gold LLC, 2025

Figure 9-8: Divide Trench



Source: Donlin Gold LLC, 2025

9.6 Exploration Potential

Exploration potential in the vicinity of the Project open pit designs include extensions to the south and west of ACMA and to the north and east of Lewis. Mineralization remains open at depth under the current pit limits. Mineralization also remains open to the north of the planned pit and has been tested by shallow trenching and soil sampling, with limited drilling undertaken to date.

Exploration potential also extends outside the areas that have been the subject of the mine design. Gold mineralization is associated with an overall north–northeasterly trending high level dike/sill complex and includes the Ophir, Dome, Far Side, Quartz/Duqum, and Snow prospects. Soil sampling surveys were conducted between 1988 and 2008 by WestGold, Placer Dome Inc., and Donlin Gold LLC.

Figure 9-3 and Figure 9-5 shows gold-in-soils analytical results for nearby prospects adjacent to the reprocessed geophysical data which highlights the reverse polarity magnetic trend along the same orientation as mineralization (Willliams, 2021).

Summaries of the prospect potentials identified by Donlin Gold LLC are derived from Buchanan (2009), Chamois (2009), Francis (2011), and collaborative assessment within the joint venture and Owners.

9.6.1 Far Side

The Far Side prospect has been tested by three Donlin Gold LLC core drill holes (735 m) along 300 m of strike and 29 RC holes that were drilled by WestGold. Drill results for the core drilling are indicated in Table 10-4. The prospect is situated in an area where dikes of a generally easterly trend intersect the more dominant northeasterly trend.

9.6.2 Duqum

The Duqum prospect is the site of the first recorded lode gold discovery in the Donlin deposit area. It is about 1 km long, and mineralization is associated with a narrow porphyry dike. Three core holes have been drilled (1,043 m) by Donlin Gold LLC. Drill results are summarized in Table 10-5

9.6.3 Snow / Quartz

The Snow and Quartz prospects are hosted in a dike-related, gold-bearing corridor that is about 1.5 km wide and approximately 4 km long. In the area of this dike swarm, the porphyry dikes are 20 m to >100 m wide, discontinuous bodies. Gold mineralization is associated closely with the dikes and is hosted either within the dikes themselves or in the adjacent sedimentary rocks. limited drilling has been completed. Better drill results are included in Table 10-6.

9.6.4 Dome

The Dome prospect is situated under a prominent, rounded hill about 5 km north of the planned ACMA-Lewis pits. Several mineralized felsic porphyries intrude into a greywacke unit and have hornfelsed the sedimentary rocks over wide intervals. Mineralization consists of stockworks of veinlets containing arsenopyrite, pyrite, pyrrhotite, and minor chalcopyrite. Preliminary metallurgical testwork indicates that mineralization may be less refractory than that encountered in the ACMA-Lewis area.

Fourteen widely spaced drill holes have been completed over an area of approximately 500 m x 500 m. Mineralization is open to the north, east, south and to depth, and may be open to the west at depth. Better drill results are included in Table 10-7.

9.6.5 Ophir

The Ophir Hill is the highest topographic feature in the Donlin district. Surface mapping over an area of about 1.5 km x 750 m indicates Cretaceous sedimentary rocks have been intruded by felsic to intermediate intrusions, which may be dikes. Surface exposures are completely oxidized, but boxworks after sulfides indicate arsenopyrite, pyrite and other sulfides occur as disseminations and thin veinlets. Soil sampling has identified a strong gold-in-soil anomaly on the southwestern flanks of the hill. Four soil-sampling campaigns were completed between 1988 and 2010, comprising 149 samples (20 m spacing on a single line) in 1988, 108 samples (100 m × 400 m grid) in 1999, 518 samples (25 m × 100 m grid) in 2008, and 175 samples (50 m × 200 m grid) in 2010. No drilling has been undertaken.

10.0 DRILLING

A total of 2,145 core and RC holes of 516,779 m were completed from 1995 through 2025 as summarized in Table 10-1. Holes drilled by previous operators between 1995 and 2000 include 347 core holes (86,298 m) and 77 RC holes (16,338 m). Holes drilled by Donlin Gold LLC include 491 core holes (121,383 m) and more recently 10 dual rotary holes (630 m).

A total of 1,737 core (456,450 m) and 387 RC (37,457 m) holes conducted since 1995, were used to inform the resource estimate. Results from the 2025 drilling were compared to the geologic model and resource model and were found to support the interpretation of geology and estimation.

Drill hole location plans are provided in Figure 10-1 for the Property, and in Figure 10-2 for the area where Mineral Resources and Mineral Reserves were estimated.

10.1 Drill Methods

Core sizes used on the Property include: NQ3 (45.1 mm core diameter), NQ (47.6 mm), HQ3 (61.2 mm), HQ (63.5 mm), and PQ (85 mm). Most of the core drilled between 1995 and 1999 was HQ size, since all holes were started with HQ tools and reduced to the smaller diameter NQ size as necessary.

Depth limits for HQ size holes varied by hole conditions from 475 m to 650 m.

10.2 Geological Logging

Standard logging conventions were developed by Placer Dome Inc. and refined over the durations of the drilling programs.

Standard logging and sampling conventions were used to capture information from the drill core and, where applicable, RC chips. Types of data captured in separate tables include lithology, mineralization, alteration (visual), structural and geotechnical. Remarks by the logging geologist were also recorded. Lithology was recorded in a two to four letter alpha code. The mineralization table recorded visual percent veining (by type) and sulfide (pyrite, arsenopyrite, stibnite and realgar). Specific alteration features including iron oxide and carbonate alteration were also recorded using a qualitative scale. Structural data collected consisted of the type of structure, measurements relative to core axis and oriented core measurements, if applicable. The geotechnical table recorded percent recovery and rock quality designation (RQD) for the entire hole, and fracture intensity where warranted.

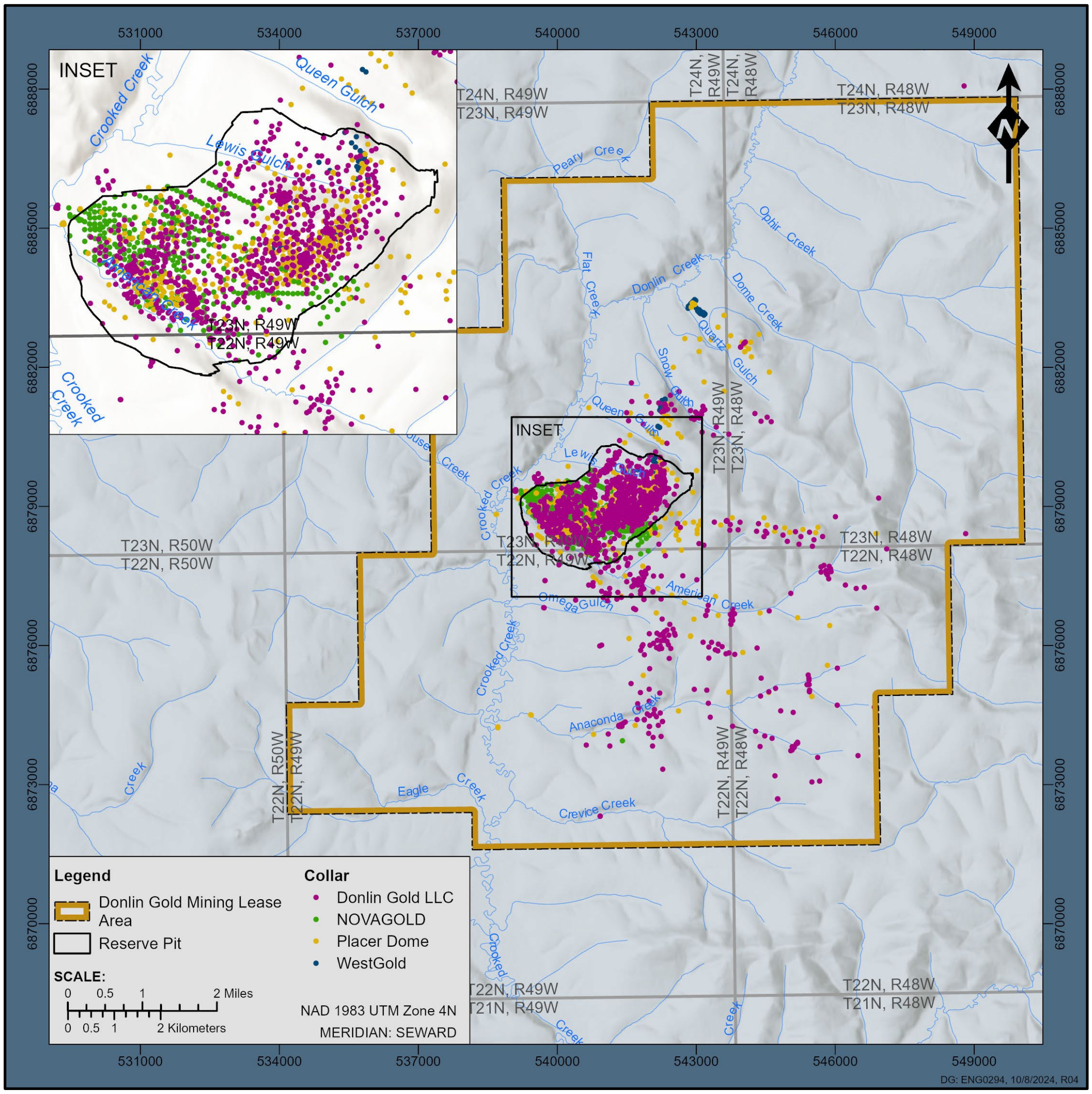
Table 10-1: RC and Core Drill Summary Table

Year	Company	No. of Drill Holes	Hole Type	Meters Drilled	Comments
1988-1989	WestGold				Core and auger holes were drilled on the Property.
1995	Placer Dome Inc.	32	core	6,117	Thirty in Lewis, one at Rochelieu Ridge, and one near the mouth of Queen Gulch
1996	Placer Dome Inc.	26	RC	8,413	Seventeen of the holes twinned earlier core holes. Four water wells (three in camp, one in Lewis) were drilled with the RC drill, and five core holes in the 400 area were pre-collared through deep overburden.
		113	core	30,214	All but eight of the core holes were drilled in Lewis or Queen. The others were distributed north of the current resource area in the Dome, Far Side, and Snow prospects.
1997	Placer Dome Inc.	51	RC	7,925	Lewis, Queen, Rochelieu, ACMA, 400 Area, Vortex, alongside the American Ridge runway, and Snow. Includes two water wells
		66	core	15,241	Lewis, Queen, 400 Area, ACMA, and north of the resource area at Quartz, Duqum, and Dome
1998	Placer Dome Inc.	96	core	24,132	The drilling was done in two phases: four holes in the ACMA–400 area in March and April, and 41 closely spaced holes in the Lewis area in June to October to test variography. Resource expansion drilling in the Lewis, Queen, and ACMA areas was also conducted from June to October.
1999	Placer Dome Inc.	33	core	9,190	Twenty-six of these, totaling 6,690 m, were resource definition holes drilled in ACMA–400.
2000	Placer Dome Inc.	7	core	1,404	Five at Dome and two at Quartz, for an evaluation of IP anomalies and potential for high-grade deposits

Year	Company	No. of Drill Holes	Hole Type	Meters Drilled	Comments
2001	Placer Dome Inc.	42	core	7,493	Evaluation of the potential for significant resource growth in the ACMA area
2002	NOVAGOLD	146	RC	11,589	One hundred and forty-one exploration and resource expansion holes in the ACMA, 400, Lewis, Akivik, Rochelieu, Vortex, and Far East prospects. Three water wells were drilled near the mouth of American Creek, and two were drilled in the Low Road on the south face of Lewis.
		196	core	39,092	Two of the core holes are geotechnical holes in the Anaconda Creek valley.
2004	Placer Dome Inc.	17	RC	2,335	Condemnation holes in the Anaconda Creek and upper American Creek valleys
		3	core	852	Geotechnical core holes
2005	Placer Dome Inc.	30	RC	3,644	-
		90	core	24,596	Infill in ACMA and Lewis
2006	DCJV	327	core	92,804	Pit slope stability, metallurgy, waste rock studies, facilities condemnation, and engineering, and calcium carbonate resource bulk sampling, delineation, and exploration
2007	DCJV	13	RC	1,043	Monitor wells and pit pump tests
		248	core	75,257	Pit resource infill, pit expansion, carbonate exploration, geotechnical, and engineering studies
2008	DCLLC	108	core	33,425	Exploration, resource infill, condemnation, and geotechnical studies
2009	Donlin Gold LLC	19	core	950	Geotechnical and hydrological core holes
2010	Donlin Gold LLC	6	core	2,090	Geotechnical core holes
2017	Donlin Gold LLC	16	core	7,040	Infill, geotechnical and geometallurgy tests

Year	Company	No. of Drill Holes	Hole Type	Meters Drilled	Comments
2019	Donlin Gold LLC	30	core	1,060	Geotechnical core holes in planned TSF and other planned water retention facilities in support of engineering and permitting of those facilities
2020	Donlin Gold LLC	85	core	23,361	Infill to confirm recent geologic modeling concepts and test potential high-grade extensions
2021	Donlin Gold LLC	79	core	24,263	Resource infill and grid drilling
2022	Donlin Gold LLC	141	core	42,331	Resource infill and grid drilling
2023	Donlin Gold LLC	42	core	1,833	Geotechnical drilling
		10	dual rotary	630	Geotechnical drilling
2025	Donlin Gold LLC	47	core	18,056	Infill resource drilling and geotechnical drilling
		26	core	399	Jungjuk Port Road material sites

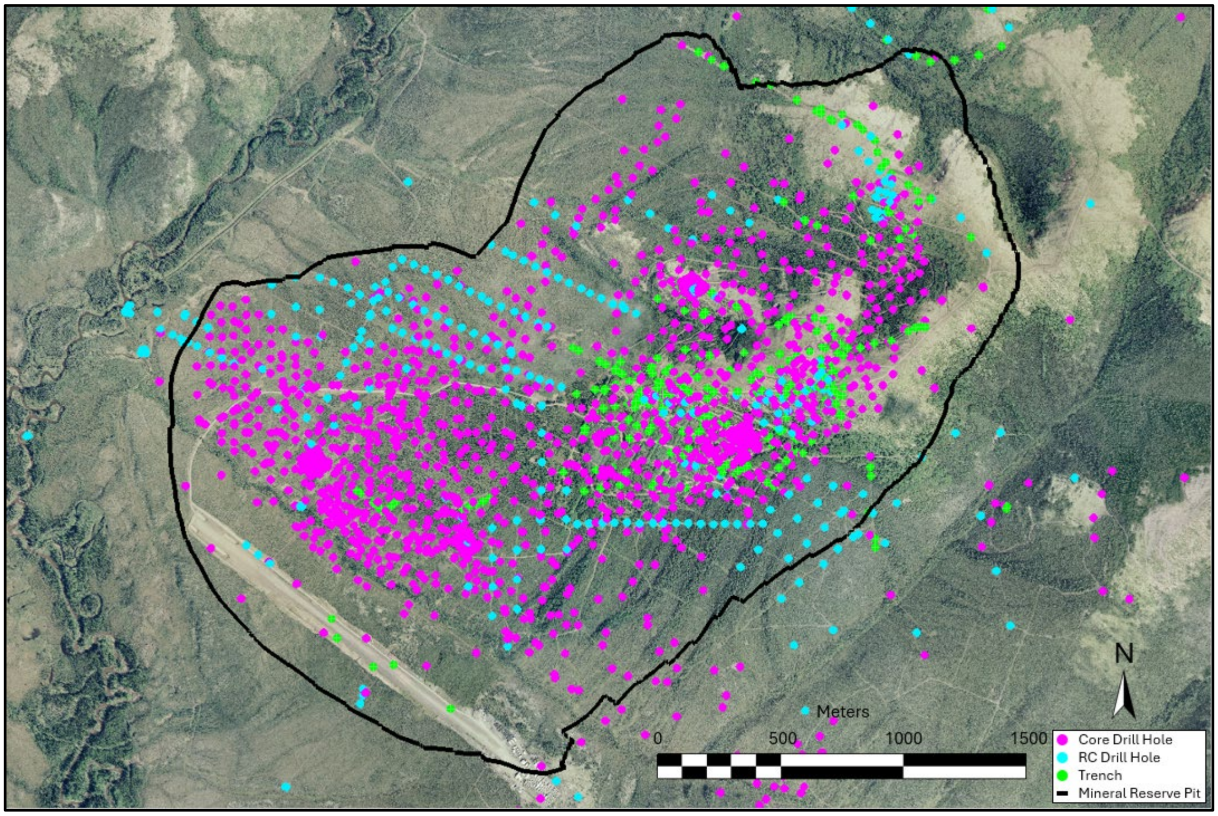
Figure 10-1: Project Drill Hole Location Plan



Source: Donlin Gold LLC, 2025

Note: Donlin Gold Mining Lease area that is available for subsurface exploration (e.g., drilling) under lease agreements with Calista and TKC.

Figure 10-2: Resource Area Drill Holes



Source: Donlin Gold LLC, 2025

10.3 Recovery

A survey of nearly 200,000 core recovery records in the database revealed an overall length weighted average core recovery of 95%. Material variation across the deposit areas is not expected as the recovery is uniformly high. Average recovery increases from 80-95% from 0-40 m and then ranges from 95-100% below 40 m where overburden and surface weathering effects are generally absent.

10.4 Collar Surveys

Traditional (transit) survey methods were used to locate all 1995-1999 and 2001 drill collars and trenches. Modern GPS technology involving a base unit and up to two roving units was introduced in 2002. The roving instruments were operated in the field to collect stationary readings over the drill collars.

Based on surveys of control points, the approximate maximum horizontal and vertical variances of drill hole collar surveys under optimal conditions were considered to be 0.2 and 0.6 m, respectively.

In 2017, surveyors utilized Global Navigation Satellite System (GNSS) receivers and Real Time Kinematic (RTK) measurements. During the 2017 drill campaign, all drill holes, except one, were surveyed with a horizontal and vertical accuracy within 8 cm.

From 2020-2022, data were collected using GPS dual frequency receivers using RTK GPS methodology. No positions differed by more than 2 cm, horizontally or vertically.

From 2023-2025, data were collected using Post-Processed Kinematic (PPK) GPS methodology with a horizontal and vertical accuracy within 1 cm.

10.5 Down-hole Surveys

The Sperry Sun single-shot camera method was used through 2000 for directional surveys to determine down-hole deviation. Reflex EZ Shot instrumentation was introduced in 2001 and used until 2017. In 2020, the Boart Longyear TruShot tool was used. A SPT GyroMaster survey tool was used in 2021 and 2022 with a combination of the SPT GyoMaster and the Board Longyear TruShot tools being used in 2023. In 2025, Donlin Gold LLC switched to an Index OMIx42 downhole survey tool.

Televiewer data was collected from 2020-2025.

10.6 Geotechnical and Hydrological Drilling

Geotechnical and most hydrogeological drilling is conducted with methods and standards similar to resource drilling. Where drill holes were not consumed for testing work, samples were commonly collected for assay analysis and, if within reasonable distances to the core of the deposit, were considered useful for the purposes of resource estimation. Geotechnical and hydrological drilling is included in the drill totals in Table 10-1, and are included in the drill location plan in Figure 10-2. The interpretation of results is presented in Section 16 and Section 18.

10.7 Metallurgical Drilling

Specific drill holes were completed for comminution testwork. These holes, although not broken out by collar, are included in the drill location plan in Figure 10-1. Interpretation of the results are presented in Section 13.

10.8 Condemnation Drilling

Condemnation drilling was performed to identify potential infrastructure sites. Condemnation drilling is conducted with methods and standards similar to deposit drilling. Where drill holes were not consumed for testing work, samples were commonly collected for assay analysis and, if within reasonable distances to the core of the deposit, were considered useful for the purposes of resource estimation.

Locations of the condemnation drill holes are included in Figure 10-1. The results of the condemnation drilling were factored into the location of key surface infrastructure.

10.9 Twin Drilling

Core and RC holes were compared in 1996 when 17 core holes in the Lewis area were twinned with RC holes. This study found that, in most instances, composite assay intervals from the RC holes were shorter, less continuous, and lower in grade than in the twinned core holes (Szumigala, 1997).

10.10 Summary of Drill Intercepts

A summary of a number of drill hole intercepts from the ACMA area are shown in Table 10-2 and the Lewis area in Table 10-3. Interpretation of drill results with examples of the drill hole geometry, and drill hole intercepts are shown in Figure 10-3 (ACMA) and Figure 10-4 (Lewis), and demonstrate that the drilling was designed to intersect the mineralization as perpendicular as possible.

A summary of a number of drill hole intercepts from the areas that show exploration potential are shown in Table 10-4 to Table 10-7.

Table 10-2: ACMA Area Drill Hole Intercept Summary Table

Hole ID	Northing	Easting	Elevation	Azimuth	Dip	Area	Drill Intercept From (m)	Drill Intercept To (m)	Drilled Thickness (m)	Gold Grade (g/t Au)
DC06-1114	6878385.36	539899.82	127.97	294.85	-65.4	ACMA	178.00	218.19	40.19	4.14
DC06-1114	6878385.36	539899.82	127.97	294.85	-65.4	ACMA	234.00	304.68	70.68	4.10
DC06-1114	6878385.36	539899.82	127.97	294.85	-65.4	ACMA	310.28	316.51	6.23	3.79
Total/Average									**117.10**	**4.10**
DC06-1115	6878270	540117	133	288.55	-58.2	ACMA	194.00	198.00	4.00	1.37
DC06-1115	6878270	540117	133	288.55	-58.2	ACMA	232.00	242.00	10.00	4.58
DC06-1115	6878270	540117	133	288.55	-58.2	ACMA	248.00	252.98	4.98	19.37
DC06-1115	6878270	540117	133	288.55	-58.2	ACMA	261.00	280.29	19.29	5.64
DC06-1115	6878270	540117	133	288.55	-58.2	ACMA	286.00	304.00	18.00	2.19
DC06-1115	6878270	540117	133	288.55	-58.2	ACMA	310.00	332.94	22.94	3.49
DC06-1115	6878270	540117	133	288.55	-58.2	ACMA	343.05	368.00	24.95	5.87
Total/Average									**104.16**	**5.01**
DC06-1120	6878411.6	539846.32	127.22	295.85	-61.2	W. ACMA	145.00	160.00	15.00	2.48
DC06-1120	6878411.6	539846.32	127.22	295.85	-61.2	W. ACMA	175.00	205.00	30.00	1.11
DC06-1120	6878411.6	539846.32	127.22	295.85	-61.2	W. ACMA	235.50	271.46	35.96	2.89
Total/Average									**80.96**	**2.15**
DC06-1126	6878684.93	539605.65	123.7	299.05	-60.3	W. ACMA	189.04	204.50	15.46	2.56
DC06-1126	6878684.93	539605.65	123.7	299.05	-60.3	W. ACMA	222.00	229.30	7.30	4.21
DC06-1126	6878684.93	539605.65	123.7	299.05	-60.3	W. ACMA	259.90	264.00	4.10	1.21
DC06-1126	6878684.93	539605.65	123.7	299.05	-60.3	W. ACMA	303.00	309.00	6.00	2.32
DC06-1126	6878684.93	539605.65	123.7	299.05	-60.3	W. ACMA	317.00	335.00	18.00	5.37
DC06-1126	6878684.93	539605.65	123.7	299.05	-60.3	W. ACMA	345.00	365.00	20.00	3.27
DC06-1126	6878684.93	539605.65	123.7	299.05	-60.3	W. ACMA	374.00	382.00	8.00	2.40

Hole ID	Northing	Easting	Elevation	Azimuth	Dip	Area	Drill Intercept From (m)	Drill Intercept To (m)	Drilled Thickness (m)	Gold Grade (g/t Au)
Total/Average									**78.86**	**3.43**
DC06-1134	6879104.23	539709.27	149.47	297.35	-61.3	Akivik	17.00	27.00	10.00	1.64
DC06-1134	6879104.23	539709.27	149.47	297.35	-61.3	Akivik	35.00	43.00	8.00	4.22
DC06-1134	6879104.23	539709.27	149.47	297.35	-61.3	Akivik	187.00	201.00	14.00	5.54
Total/Average									**32.00**	**3.99**
DC06-1136	6879210	539771.1	150.99	297.55	-59	Akivik	33.00	47.00	14.00	2.90
DC06-1136	6879210	539771.1	150.99	297.55	-59	Akivik	61.00	69.00	8.00	2.79
Total/Average									**22.00**	**2.86**
DC06-1138	6878826.27	539729.18	136.51	298.35	-61.6	Aurora	13.00	40.00	27.00	2.07
DC06-1138	6878826.27	539729.18	136.51	298.35	-61.6	Aurora	54.63	68.00	13.37	2.70
DC06-1138	6878826.27	539729.18	136.51	298.35	-61.6	Aurora	104.23	117.00	12.77	1.51
DC06-1138	6878826.27	539729.18	136.51	298.35	-61.6	Aurora	285.50	288.00	2.50	12.40
DC06-1138	6878826.27	539729.18	136.51	298.35	-61.6	Aurora	306.00	316.00	10.00	4.05
DC06-1138	6878826.27	539729.18	136.51	298.35	-61.6	Aurora	405.00	409.00	4.00	4.54
DC06-1138	6878826.27	539729.18	136.51	298.35	-61.6	Aurora	460.00	474.00	14.00	3.88
DC06-1138	6878826.27	539729.18	136.51	298.35	-61.6	Aurora	494.00	506.00	12.00	2.19
DC06-1138	6878826.27	539729.18	136.51	298.35	-61.6	Aurora	526.00	530.00	4.00	3.25
Total/Average									**99.64**	**2.96**

Hole ID	Northing	Easting	Elevation	Azimuth	Dip	Area	Drill Intercept From (m)	Drill Intercept To (m)	Drilled Thickness (m)	Gold Grade (g/t Au)
DC06-1245	6878553.04	540311.42	170.41	301.65	-58.7	E. ACMA	5.33	23.00	17.67	1.62
DC06-1245	6878553.04	540311.42	170.41	301.65	-58.7	E. ACMA	42.00	62.00	20.00	1.84
DC06-1245	6878553.04	540311.42	170.41	301.65	-58.7	E. ACMA	94.00	106.00	12.00	5.53
DC06-1245	6878553.04	540311.42	170.41	301.65	-58.7	E. ACMA	112.00	126.00	14.00	2.33
DC06-1245	6878553.04	540311.42	170.41	301.65	-58.7	E. ACMA	128.00	148.97	20.97	2.85
DC06-1245	6878553.04	540311.42	170.41	301.65	-58.7	E. ACMA	166.00	178.00	12.00	1.21
DC06-1245	6878553.04	540311.42	170.41	301.65	-58.7	E. ACMA	187.00	193.65	6.65	2.33
DC06-1245	6878553.04	540311.42	170.41	301.65	-58.7	E. ACMA	210.00	245.50	35.50	8.35
DC06-1245	6878553.04	540311.42	170.41	301.65	-58.7	E. ACMA	254.00	276.00	22.00	1.89
Total/Average									**160.79**	**3.68**

Note: Drill thicknesses may not represent true thicknesses.

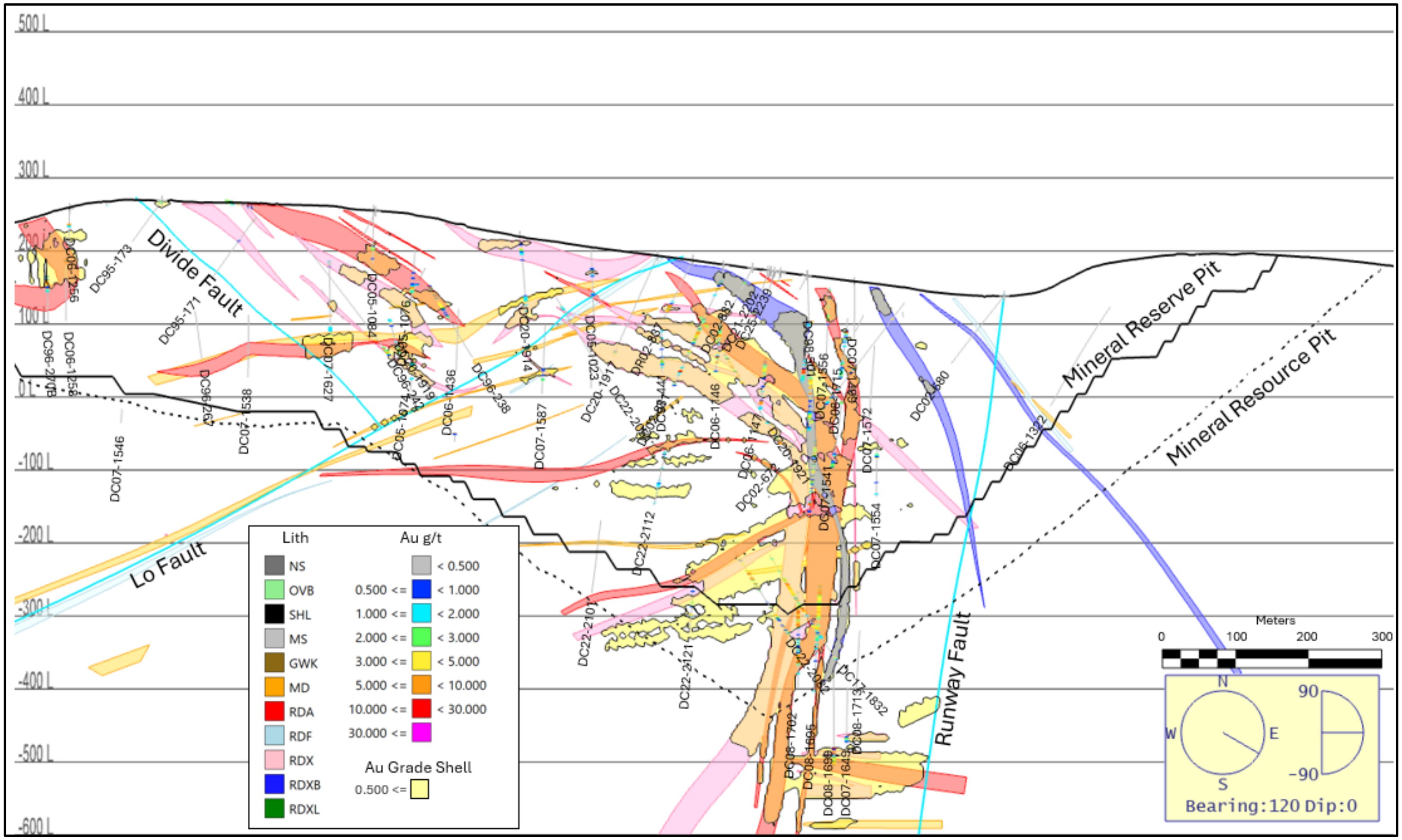
Table 10-3: Lewis Area Drill Hole Intercept Summary Table

Hole ID	Northing	Easting	Elevation	Azimuth	Dip	Area	Drill Intercept From (m)	Drill Intercept To (m)	Drilled Thickness (m)	Gold Grade (g/t Au)
DC06-1252	6878663.18	541745.81	302.68	303.35	-59.6	Lewis	45.21	53.21	8.00	3.98
DC06-1252	6878663.18	541745.81	302.68	303.35	-59.6	Lewis	104.49	108.49	4.00	1.78
DC06-1252	6878663.18	541745.81	302.68	303.35	-59.6	Lewis	209.10	215.10	6.00	3.07
DC06-1252	6878663.18	541745.81	302.68	303.35	-59.6	Lewis	240.90	259.00	18.10	2.56
DC06-1252	6878663.18	541745.81	302.68	303.35	-59.6	Lewis	265.00	281.00	16.00	4.80
DC06-1252	6878663.18	541745.81	302.68	303.35	-59.6	Lewis	297.00	309.00	12.00	2.73
Total/Average									**64.10**	**3.39**
DC06-1253	6879030.63	541736.09	400.88	300.35	-58.9	Lewis	9.00	14.33	5.33	3.00
DC06-1253	6879030.63	541736.09	400.88	300.35	-58.9	Lewis	26.00	32.50	6.50	1.84
DC06-1253	6879030.63	541736.09	400.88	300.35	-58.9	Lewis	39.69	42.50	2.81	3.39
DC06-1253	6879030.63	541736.09	400.88	300.35	-58.9	Lewis	74.50	85.70	11.20	1.17
DC06-1253	6879030.63	541736.09	400.88	300.35	-58.9	Lewis	90.00	120.00	30.00	1.60
DC06-1253	6879030.63	541736.09	400.88	300.35	-58.9	Lewis	293.00	305.00	12.00	1.26
DC06-1253	6879030.63	541736.09	400.88	300.35	-58.9	Lewis	311.00	340.50	29.50	3.29
DC06-1253	6879030.63	541736.09	400.88	300.35	-58.9	Lewis	349.00	363.00	14.00	1.21
Total/Average									**111.34**	**2.05**
DC06-1183	6879518.7	541253.71	240.06	299.65	-61.8	Rochelieu	42.00	51.00	9.00	2.22
DC06-1183	6879518.7	541253.71	240.06	299.65	-61.8	Rochelieu	60.00	63.00	3.00	2.01
DC06-1183	6879518.7	541253.71	240.06	299.65	-61.8	Rochelieu	96.00	102.00	6.00	6.72
DC06-1183	6879518.7	541253.71	240.06	299.65	-61.8	Rochelieu	110.00	116.00	6.00	3.64
Total/Average									**24.00**	**3.67**

Hole ID	Northing	Easting	Elevation	Azimuth	Dip	Area	Drill Intercept From (m)	Drill Intercept To (m)	Drilled Thickness (m)	Gold Grade (g/t Au)
DC06-1185	6879443.2	541416.5	288.87	295.35	-61.9	Rochelieu	29.80	55.93	26.13	3.61
DC06-1185	6879443.2	541416.5	288.87	295.35	-61.9	Rochelieu	63.84	78.00	14.16	1.85
DC06-1185	6879443.2	541416.5	288.87	295.35	-61.9	Rochelieu	196.00	204.77	8.77	6.66
DC06-1185	6879443.2	541416.5	288.87	295.35	-61.9	Rochelieu	237.00	251.00	14.00	6.25
DC06-1185	6879443.2	541416.5	288.87	295.35	-61.9	Rochelieu	283.00	289.00	6.00	1.82
Total/Average									**69.06**	**4.02**
DC06-1267	6879724.6	542229.44	374.72	299.05	-59.4	Queen	131.67	135.09	3.42	1.54
DC06-1267	6879724.6	542229.44	374.72	299.05	-59.4	Queen	190.00	206.00	16.00	2.41
DC06-1267	6879724.6	542229.44	374.72	299.05	-59.4	Queen	212.00	216.00	4.00	4.27
DC06-1267	6879724.6	542229.44	374.72	299.05	-59.4	Queen	242.00	248.00	6.00	1.46
DC06-1267	6879724.6	542229.44	374.72	299.05	-59.4	Queen	266.00	272.00	6.00	2.27
DC06-1267	6879724.6	542229.44	374.72	299.05	-59.4	Queen	316.43	332.80	16.37	2.10
DC06-1267	6879724.6	542229.44	374.72	299.05	-59.4	Queen	362.00	377.04	15.04	1.47
Total/Average									**66.83**	**2.09**
DC06-1268	6879663.53	542219.04	356.56	293.95	-58.2	Queen	159.55	165.00	5.45	2.08
DC06-1268	6879663.53	542219.04	356.56	293.95	-58.2	Queen	234.00	243.00	9.00	8.06
DC06-1268	6879663.53	542219.04	356.56	293.95	-58.2	Queen	254.00	257.00	3.00	2.43
DC06-1268	6879663.53	542219.04	356.56	293.95	-58.2	Queen	288.00	292.34	4.34	2.54
Total/Average									**21.79**	**4.69**

Note: Drill thicknesses may not represent true thicknesses.

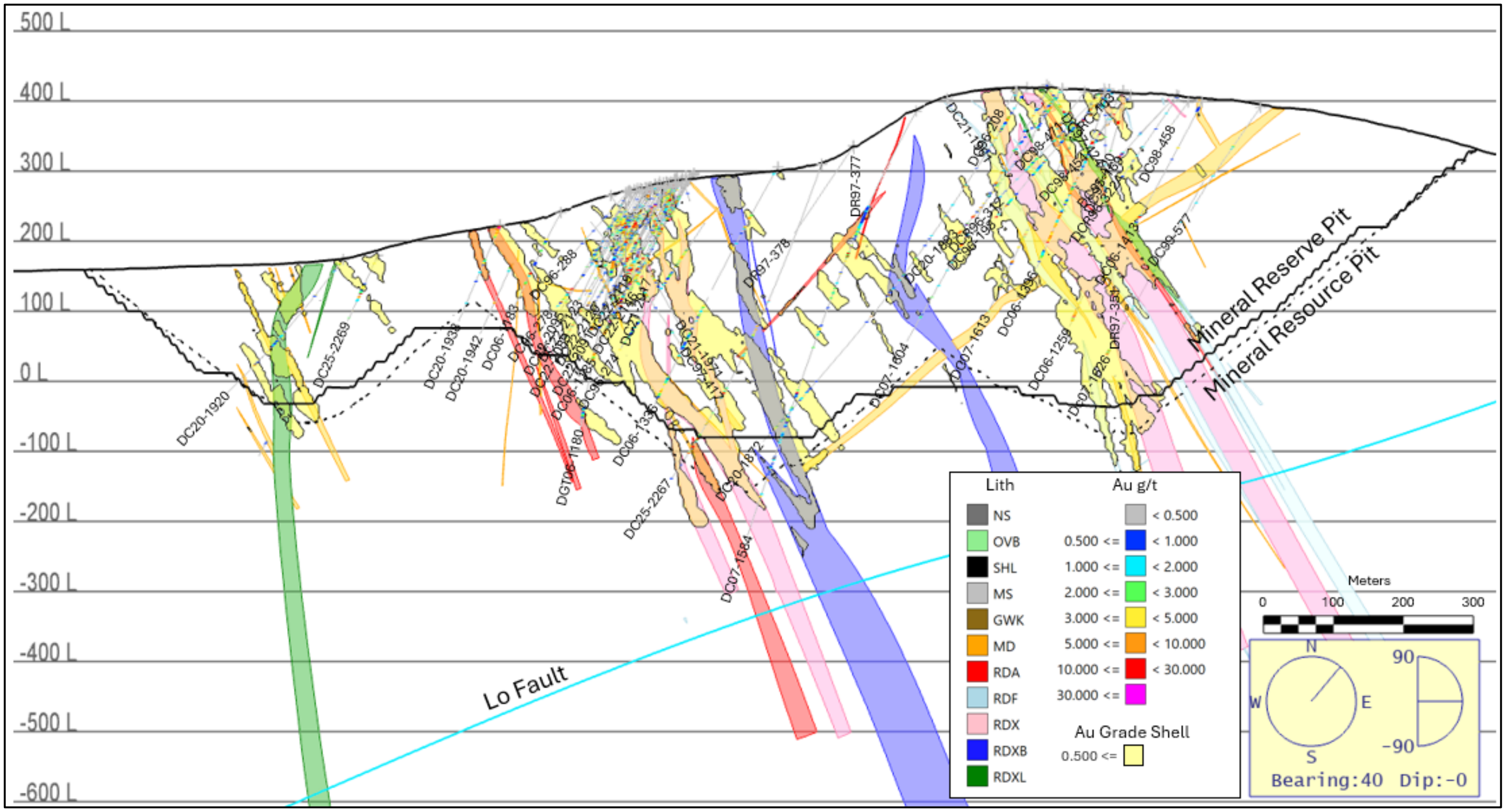
Figure 10-3: Example Drill Cross-Section ACMA



Source: Donlin Gold LLC, 2025

Note: 2025 drilling included

Figure 10-4: Example Drill Cross-Section, Lewis



Source: Donlin Gold LLC, 2025

Note: 2025 drilling included

Table 10-4: Far Side

Hole ID	Northing	Easting	Elevation	Azimuth	Dip	Drill Intercept From (m)	Drilled Thickness (m)	Gold Grade (g/t Au)
DC96-254	6,883,383	542,914	179	320	65	23.2	16.8	4.60
DC96-255	6,883,356	542,960	182	320	65	122.0	14.0	3.00
DC96-256	6,883,296	542,888	175	320	65	130.0	15.6	5.86

Note: These are illustrative of better drill results, may not be representative of the deposit in general. Drill thicknesses may not represent true thicknesses. Northings, eastings, and elevations are at the collar locations.

Table 10-5: Duqum

Hole ID	Northing	Easting	Elevation	Azimuth	Dip	Drill Intercept From (m)	Drilled Thickness (m)	Gold Grade (g/t Au)
DC97-387	6,882,450	543,692	274	310	55	338.0	16.0	2.39
DC97-388	6,882,605	543,070	221	310	55	210.0	12.0	5.03
DC97-388	6,882,605	543,070	221	310	55	236.0	10.0	2.29
DC97-389	6,882,607	543,074	221	40	55	90.0	10.0	3.86
DC97-389	6,882,607	543,074	221	40	55	202.0	10.0	2.79
DC97-389	6,882,607	543,074	221	40	55	320.0	16.0	3.79

Note: These are illustrative of better drill results, may not be representative of the deposit in general. Drill thicknesses may not represent true thicknesses. Northings, eastings, and elevations are at the collar locations.

Table 10-6: Snow / Quartz

Hole ID	Northing	Easting	Elevation	Azimuth	Dip	Drill Intercept From (m)	Drilled Thickness (m)	Gold Grade (g/t Au)
DC97-383	6,880,373	541,449	230	295	50	16.0	23.0	2.77
DC97-384	6,880,537	541,563	194	295	50	52.0	10.0	3.34

Note: These are illustrative of better drill results, may not be representative of the deposit in general. Drill thicknesses may not represent true thicknesses. Northings, eastings, and elevations are at the collar locations.

Table 10-7: Dome

Hole ID	Northing	Easting	Elevation	Azimuth	Dip	Drill Intercept From (m)	Drilled Thickness (m)	Gold Grade (g/t Au)
DC08-1785	6,882,542	544,056	342	110	70	139.0	10.0	2.19
DC08-1785	6,882,542	544,056	342	110	70	163.0	10.0	3.89
DC08-1785	6,882,542	544,056	342	110	70	211.0	22.6	3.29
DC08-1785	6,882,542	544,056	342	110	70	248.0	25.0	2.94
DC97-392	6,882,486	544,041	340	130	65	94.0	52.0	3.21
DC97-392	6,882,486	544,041	340	130	65	185.0	61.0	3.30
DC97-392	6,882,486	544,041	340	130	65	258.0	14.0	3.99

Note: These are illustrative of better drill results, may not be representative of the deposit in general. Drill thickness may not represent true thickness. Northings, eastings, and elevations are at the collar locations.

11.0 SAMPLE PREPARATION, ANALYSES, AND SECURITY

11.1 Sampling Methods

Drill hole sampling protocols were developed by Placer Dome Inc. and refined over subsequent drill programs.

Trenching was completed on the Project area but was not used for Mineral Resource estimation.

Holes are sampled from the top of bedrock to the end of the hole. Overburden, excluding the organic layer, is also sampled if core recovery was good and if the interval is abnormally thick and composed of abundant rock clasts. Prior to the 2017 drilling program, core sample intervals were typically based on rock type, rock type breaks, and presence of visible sulfide/arsenic minerals. The maximum sample length in zones consisting of intrusive rocks or that contain appreciable sulfide/arsenic minerals was typically 2 m, whereas sample lengths in sedimentary rock zones that lack appreciable sulfide/arsenic minerals could be 3 m. A minimum of three additional 2 m sample intervals were taken before and after each intrusive rock or mineralized zone. From 2017 onward, samples are typically taken at maximum of 2 m intervals.

Numbered paper tags are stapled to the core box at each sample break. A sampling cutting list is generated that also specifies the insertion points for control samples.

The core is then digitally photographed and split in half with an electric rock saw equipped with water-cooled diamond saw blades. Core cutters orient the core in the saw to ensure a representative split. One-half of the core is returned to the core box for storage at site, and the other half is bagged for sample processing. Occasionally, whole core has been transported off site for cutting and sampling, after which the remaining half-core is stored off site.

11.2 Metallurgical Sampling

Typically, metallurgical sampling consisted of taking half-core samples which were used in flotation and pressure oxidation tests. Whole core samples were only taken for drop weight and SAG mill comminution (SMC) tests.

11.3 Specific Gravity Determination

Historically, only two SG values were used in tonnage calculations: 2.65 for the intrusive units and 2.71 for the sedimentary units. Additional SG measurements were collected in 2006, 2008-2010, 2017 and 2019 to provide better coverage of deposit rock units and geographic sub-regions. Statistical evaluations of these SG values showed that they were similar to the historical intrusive and sedimentary rock SG values. Therefore, the historical values were used for the Mineral Resource estimate.

The following methodology was used to determine SG:

- Samples of whole core approximately 5 to 10 cm in length were first weighed dry and then weighed in water. The dry weighing tray assembly was replaced with a wire basket and the sample was submerged in a five-gallon bucket of water. A small tare weight (to compensate for the removed weighing tray) was attached midway up the wire assembly to facilitate alternating wet and dry measurements.
- The formula for SG calculation was: *Weight in Air/(Weight in Air – Weight in Water)*. The SGs were automatically computed in acQuire when the weights were entered into the database.
- Measurements were collected for all rock types at a minimum frequency of one sample from all logged rock type intervals and one sample every 15 to 20 m in the longer rock unit intervals. Mineralized rock takes precedence over unmineralized rock in a given rock type interval, but sufficient measurements of unmineralized material were also collected to document potential variability.

The weighted average of all SG data points was 2.69.

11.4 Analytical and Test Laboratories

The primary laboratory for all assaying up to 2020 was ALS Vancouver, British Columbia. Other ALS laboratory locations were also utilized. During the exploration programs, ALS held accreditations typical for the time, including, at various times, ISO9001:2000 and ISO 9002, and from 2005, ISO/IEC 17025 accreditations. Prior to ALS, Placer Dome Inc. did utilize their own laboratory Placer Dome Technical Services, Vancouver, British Columbia (PDTS) for their earlier drilling campaigns. ALS is independent of Placer Dome Inc. and Donlin Gold LLC. Since 2020, Donlin Gold LLC have been using Bureau Veritas Commodities Canada Ltd, Vancouver, British Columbia (BV) as the primary laboratory for all assaying. BV is accredited with ISO/IEC 17025:2017. Donlin Gold LLC are independent of BV.

Metallurgical test facilities have included AuTec and Barrick Technology Centre (BTC), both formerly PDTS, SGS-Lakefield Research, Lakefield, Ontario (SGS Lakefield), and Hazen Research (Hazen), Golden, Colorado, G&T Metallurgical Services, Kamloops, British Columbia (G&T), and FLSmidth Minerals Ltd., Scarborough, Ontario (FLSmidth) who are independent, recognized metallurgical testing laboratories. Work has also been performed by test facilities operated by Placer Dome Inc. and Barrick. Metallurgical test facilities are not typically accredited.

11.5 Sample Preparation and Analysis

Most core samples from 2005 through 2010 were split and crushed at the Donlin camp sample preparation facility and pulverized at the ALS Vancouver laboratory facility. Samples of 2006 core that were split by Alaska Earth Sciences in Anchorage were shipped to an ALS preparation laboratory for crushing and pulverizing. In 2017, whole core was shipped off-site, split and sampled in Fairbanks by a contractor independent of ALS. Sampled half-core was then crushed, pulverized, and split at ALS' facility in Fairbanks, and pulps were shipped to and analyzed primarily at the ALS Vancouver laboratory. In 2020, drill core was split at the Donlin camp facility, shipped primarily to ALS in Fairbanks for sample preparation, and pulps were shipped and analyzed at ALS in Vancouver and Lima, Peru. Since 2021, half core was sent to BV in Juneau or Fairbanks for the preparation and analysis for gold with all other analyses completed by BV and ALS.

Typical sample preparation procedures are as follows:

- The entire bagged sample is dried in an oven heated to 90- 95°C for 12 hours.
- The sample and sample tag are placed into trays for processing.
- Blank samples (one of three QA/QC control samples) are inserted into the sample stream.
- The sample is crushed until the end product passes 70% minus 10 mesh (2 mm). Sieve analyses are performed daily to check crush quality, and the crusher jaws are adjusted as necessary. The crushers are cleaned with blank material four times per 12-hour shift and before a new hole is started. Cutting lists also specify special cleaning frequency when unusually sulfide-rich material is processed.
- Crushed sample is then passed through a riffle splitter four to six times to obtain a nominal 250 g (or 1 kg between 2017-mid 2025) split. This subsample is put into a numbered pulp bag, and the remainder, or coarse reject, is put back into the original sample bag. The splitter and sample pans are cleaned with compressed air.
- Two additional control samples—standard reference material (SRM) and a duplicate split of crushed sample—are inserted as specified on the cutting list prepared by the geologist. Two of each control sample type, including SRM, duplicates, and blanks, are included in every batch of 70-100 samples. The blank is prepared by processing a sample from a bin of gravel-size crushed rock through the jaw crusher and riffle-splitting it to 200 g. When a duplicate is required, the crushed core sample is passed once through the riffle splitter, and each half is split repeatedly to obtain a 200 g sample.

Final sample preparation and chemical analysis at various ALS facilities consisted of the following:

- Splits of crushed core were reduced to rock flour or pulp (better than 85% passing minus 75 µm in a ring-and-puck grinding mill.
- A 30 g subsample of the pulp was assayed by fire assay-atomic absorption spectroscopy (AAS). Before 2007, the primary gold assay method was Au-AA23, which had an analytical range of 0.005-10 g/t Au. The Au-AA25 gold assay method was initiated in 2007 and had an analytical range of 0.01-100 g/t Au. This switch was made to reduce the cost and time delay associated with re-assaying samples with values above the 10 g/t Au analytical limit. In 2017, the program returned to using Au-AA23 as the based fire assay method using a 30 g subsample, with an overlimit (>10 g/t) trigger to a fire assay with a gravimetric finish.
- Trace multi-element analysis utilized ME-MS61m (4-acid ICP/MS suite with Hg by MS42). Overlimit analysis was performed for As using As-OG62 and Sb using Sb-AA08. Prior to 2017, this multi-element work was not performed on all samples, but a subset. Starting into 2017 to present, all samples undergo multi-element work.
- Samples that exceeded the analytical limit for a given method were re-assayed by fire-assay and gravimetric finish or ore grade fire-assay AAS. ALS determined the sulfur content of each sample according to the Leco method. The Leco method was also used to analyze samples flagged for acid base accounting (ABA) for carbon content as well as to determine neutralization potential (NP) and acid potential (AP) according to the industry-standard ALS ABA procedure.
- Prior to 2017, small subsets of samples underwent CNL and Leco analysis. Since 2017, all samples with Au >0.25 g/t are submitted. CNL was performed using Au-AA13: 30 g subsample leached in 60 mL of NaCN solution and finished using AAS. Leco was performed for Total C and S (ME-IR08), non-carbonate C (C-IR06a), and sulfate S by $NaCO_3$ leach with gravimetric finish (S-GRA06). One out of 10 samples from this subset were submitted to sulfide S short method (S-IR06a) and those within certain limits for this method triggered to sulfide S long method (S-IR07).
- Most trace and major element data for drill holes located within the resource model boundary were acquired prior to the 2005 program by various laboratories using industry-standard acid digestions followed by atomic absorption (AA) or inductively coupled plasma (ICP) instrumental determinations. Subsequent multi-element trace analyses were performed at ALS using aqua regia or four-acid digestions followed by ICP ± mass spectrometry.

On occasion, if ALS Vancouver was unable to complete the preparation, another ALS laboratory would be used. ALS Vancouver currently analyzes samples for CNL/Leco as well as the umpire laboratory for core drilled since 2025.

Final sample preparation and chemical analysis have occurred at BV since 2021 using the following:

- Split core was sent to BV, Juneau and BV, Fairbanks for crushing from 2021-2023. Crushing was performed to 70% passing 2 mm sieve. 1 kg crush splits were then sent to Vancouver for pulverization to 85% passing 75µm and submitted for analysis. BV's practice was to split the 1,000 g pulp into a 750 g archive and a 250 g analytical split. Starting in 2025, BV's Fairbanks preparation laboratory expanded to accommodate more throughput and all of prep went through BV-Fairbanks. This included completing pulverization in Fairbanks. Total of 250 g pulps splits were shipping to Vancouver for analysis.
- Total of 30 g fire assay with AAS finish (FA430) was employed for all samples. Detection limits were 0.005–10 g/t. An overlimit trigger 30 g fire assay with gravimetric finish was employed (FA530).
- From 2022-2024, BV's trace multi-element package was customized to replicate ALS MEMS61m results by modifying their procedure to use incipient dryness instead of total digestion. The resulting method was called MD250 and was used on all samples starting in 2025.

11.6 Quality Assurance and Quality Control

11.6.1 Standard Reference Materials

Placer Dome Inc. produced four standards from Donlin material with two used up to the beginning of 2005 when additional reference material was purchased from Analytical Solutions and CDN Laboratories once these SRMs were depleted. After the 2005 season, two additional SRMs were created from Donlin coarse reject material. These two new standards and one from the CDN Laboratories were used during the 2006 season.

Nine matrix matched SRMs of varying gold grade were added in early 2007, and the older standards were eventually phased out. These SRMs were created from coarse reject samples from throughout the deposit. Composites of this material were pulverized and homogenized at CDN Laboratories in Vancouver, BC. A Barrick geochemist certified the 2007 SRMs after industry-accepted round-robin assay and statistical analyses. The final SRMs included four each from unoxidized igneous and sedimentary host rocks and one oxidized igneous rock SRM.

More recently, in 2022 SRMs were produced and certified for gold FA-AA by ORE, Melbourne. Eight standards were created which sought to create a selection of grades for sedimentary and intrusive matrices with two average grade oxide standards. These were all developed using coarse reject material from 2021 drilling.

11.6.2 Blank Materials

Washed river gravel produced by Anchorage Sand and Gravel was used for blanks through early 2006 and then replaced by granite crushed material until 2018. Between 2018 and 2021, decorative granite and marble chips were purchased from hardware stores and used as blanks. In 2022, blank material was purchased from AGGPRO in Juneau, Alaska which supplied ¾ inch washed river rock sourced by DuPont Quarry outside Olympia, Washington and underwent round-robin testing before included in the sample stream.

11.6.3 1995–2002 Protocol

Placer Dome Inc. initiated the first QA/QC protocol during the 1995 drilling campaign. Coarse reject duplicate splits from 10% of the drill hole samples were submitted to an outside laboratory (Bondar Clegg, North Vancouver, British Columbia). SRM assay standards and blanks were added in 1996, and an outside laboratory (ALS Chemex, North Vancouver, British Columbia) performed check assays, presumably of coarse reject duplicates. Check assays by a secondary assay laboratory were discontinued after 1996. A more structured assay QA/QC protocol, consisting of SRMs, blanks, and duplicates inserted in rotation every 15 m down-hole, was initiated in 1997. This protocol evolved to random and blind insertion of SRMs, blanks, and coarse reject duplicates through the 2002 NOVAGOLD program.

From 1996–2002, SRMs and coarse-reject duplicates were inserted at an average rate of one per 24 samples, and blanks were inserted at an average rate of one per 25 samples. After the 1995 to 2002 drilling campaigns, almost all samples associated with SRM and blank control samples that returned values beyond acceptable tolerance limits were re-assayed until the control sample results were either acceptable or validated by duplication.

11.6.4 2005–2006 Protocol

No resource delineation drilling was conducted in 2003 and 2004. Placer Dome Inc. implemented a slightly modified QA/QC protocol in 2005, which Barrick continued in 2006. Three QA/QC samples, consisting of one blank, one coarse reject duplicate, and one SRM, were randomly inserted into every block of 20 sample numbers. Thus, every block of 20 sample numbers contained 17 drill hole samples and three QA/QC control samples.

11.6.5 2007–2010 Protocol

The batch size submitted to ALS was increased from 20 samples to 78 in 2007. To avoid sample mixing with products from other sources in the fusion process, the ALS protocol was based on a fusion batch size of 84 samples, where the laboratory added six internal control samples, leaving space for 78 client samples in a given batch.

Each batch of 78 samples shipped to ALS for sample preparation and analysis contained nine control samples (12%) consisting of three each of standards, blanks, and crushed duplicates. Spacing of the SRMs within the batch was left to the judgement of the geologist. Up to 5% field duplicates (remaining half split of core) were added to the sample batch at the discretion of the geologists for geologic reasons.

11.6.6 2017 Protocol

At least nine quality control samples (three blanks, three standards, and three field duplicates) were inserted into each batch of 69 samples, totaling 78 samples. Coarse blanks consisted of blank silica material purchased from Analytical Solution, Ltd. Additional change to protocol was the use of a 1,000 g split for subsampling at the crushing stage, where previous programs used a 250 g split. All analytical work continued to operate through ALS with preparation occurring in Fairbanks and analysis at ALS, Vancouver.

11.6.7 2020 Protocol

Sample dispatches to the laboratory were comprised of 80 samples, including QA/QC samples. Each group of 80 samples had 14 QA/QC samples inserted into the sample sequence which included four of each of the following: a SRM, and a coarse granite blank (blank-granite); as well as two of each the following: a pulp blank, a prep (crush) duplicate, and a pulp duplicate.

11.6.8 2021–2023 Protocol

QC sample insertions sought to adhere to a protocol of 5% for standards, 2.5% coarse blanks, 2.5% pulp blanks, 2.5% crush duplicates, and 2.5% pulp duplicates. Additionally, laboratory repeats were performed by the analytical laboratory.

11.6.9 2025 Protocol

Modifications made to the 2021–2023 QC protocol include:

- Preparation of all samples at BV Fairbanks
- Pulverization in Fairbanks rather than in Vancouver
- Completion of multi-element analysis in BV Vancouver using their custom MD250 method
- Re-introduction of a 250 g subsample at the crush stage rather than 1,000 g subsample
- Pulp splits of an extra 250 g specifically for ALS to perform their work was discontinued. Rather the remaining pulp material following BV analysis was sent to ALS.
- Finally, 250 g archive pulp splits were created by BV during prep which Donlin retain in long term storage in Chugiak, AK.

11.7 Databases

The work completed by Placer Dome Inc. and predecessors before 2001 was collected and compiled into a main Microsoft Access database. NOVAGOLD compiled the Placer Dome Inc. database into an updated Access database and added information from work completed in 2001 and 2002.

Placer Dome Inc. contracted ioDigital to convert the Access database to an MS SQL Server database in early 2005 using an acQuire Technology Solutions data model (acQuire). Data obtained after the conversion were imported directly into the acQuire database.

Donlin Gold LLC continues to use acQuire software to capture drill hole data.

11.8 Sample Security

For all drill programs following the initial involvement of Placer Dome Inc. in the Project, core samples are transported from the field and brought to the yard adjacent to the geology office and logging tents at the end of each drill shift.

Core storage is secure because the Property is remote and camp access is strictly controlled.

Unauthorized camp personnel have generally been excluded from the core cutting facility.

Splits of core, along with the control samples, are packed in a shipping bag, secured with a numbered security seal, and sealed in boxes for shipment. The coarse rejects and remaining split core are returned to a storage yard on site or to a secure storage facility near Anchorage.

Previously, the typical sample shipment procedure had boxed samples flown from the Donlin camp to ALS in Fairbanks, for sample preparation and forwarded to ALS in Vancouver (or other ALS facility as required) for analysis. When BV was initiated in 2021, bagged samples were shipped to Anchorage where either Lynden Transport moved the samples by road to Fairbanks or by Alaska Air Cargo to Juneau and then couriered by Eagle Raven Transport to the laboratory for sample preparation.

All samples, when stored at the prep facilities were stored in fenced, locked storage yards or within a warehouse where only laboratory staff were able to access the samples.

11.9 QP Comments on Section 11

Sample collection, preparation, analysis and security for all Placer Dome Inc., NOVAGOLD, Barrick, the DCJV, DCLLC and Donlin Gold LLC core drill programs are in line with industry-standard methods for gold deposits.

QP Kim is of the opinion that the adequacy of sample preparation, security and analytical procedures implemented for the Placer Dome Inc., NOVAGOLD, Barrick, the DCJV, DCLLD and Donlin Gold LLC drill programs are sufficiently reliable to support Mineral Resource and Mineral Reserve estimation without limitations on Mineral Resource and Mineral Reserve confidence categories.

12.0 DATA VERIFICATION

12.1 Drill Hole Database

12.1.1 2002–2008

The types of data verification activities include:

- CRM standards, blanks, duplicates, check assays
- how quality control failures were addressed
- collar locations against the topography
- downhole checked for anomalous deviation
- sampling intervals checked for gaps or overlaps
- assays checked against original documentation.

QP Kim has reviewed the 2002 to 2008 drill hole database and QA/QC results and concludes that the data is reliable for Mineral Resource estimation.

12.1.2 2009–2025

QP Kim performed a database integrity exercise on 5% of holes drilled between 2009 and 2022 and confirmed that the database reflects the original certificates for collar, downhole surveys and assays ensuring consistency and reliability of the database.

QP Kim reviewed the 2021 and 2022 QA/QC information for the drill hole data used to construct the Mineral Resource estimate. Following are the outcomes of the review of the 2021 QA/QC samples:

- *CRMs*: A failure rate of 3.9% was realized during this period with 42 reruns initiated and resolved after rerun or investigation. A failure is defined as a CRM that returns a value of ≥3 standard deviations from the expected mean or two consecutive CRMs that return values ≥2 standard deviations from the expected mean.
- *Blanks*: Eleven coarse blank failures were identified with six found to be sample swaps. After initiating reruns only one was confirmed to be a result of contamination; however, results demonstrated that the contamination did not affect downstream samples and so was rejected.

Following are the outcomes of the review of the 2022 QA/QC samples:

- *CRMs*: A failure rate of 1.97% was realized during this period with 33 reruns initiated and resolved after rerun or investigation. A failure is defined as a CRM that returns a value of ≥3 standard deviations from the expected mean or two consecutive CRMs that return values ≥2 standard deviations from the expected mean.
- *Blanks*: Eleven coarse blank failures were identified with five found to be sample swaps. After initiating reruns, three were confirmed to be a result of contamination; however, results demonstrated that the contamination did not affect samples that followed.

Based on his data verification, QP Kim concludes the data is reliable for Mineral Resource estimation.

12.1.3 SRK Audit (2022)

In 2022, SRK was contracted to perform an audit of the Donlin database. This audit was conducted in two phases:

- *Phase 1*: Population assay audit whereby SRK recreated the assay database from original source documentation and compared it to the data housed within acQuire. Any discrepancies identified were addressed. Less than 5% of assay data within the database was not validated as original source documentation was not available.
- *Phase 2*: Database export audit that compared the resource modeling databases with the acQuire database to ensure data integrity between the datasets. This included evaluation of acQuire's configuration, metadata, key data points such as collar and downhole surveys, and the data workflow. Data Entry objects were evaluated for proper validation and recommendations were made.

QP Kim reviewed the results of the audit report and considers them supportive of the reliability of the assay database.

12.2 Geology and Mineral Resource Data

12.2.1 Wood (2025)

QP Kim performed the following data verification procedures:

- Discussed the development of the 2025 geologic and mineral resource model with the modeler reviewing the following:

  - Resource estimation procedures
  - Geological and Mineral Resource models
  - Validation checks on the models
- During the August 2025 site visit, QP Kim:
  - discussed regional and local geology with site geologists
  - visited the onsite core storage facility
  - reviewed core of selected drill holes and reviewed the logged geological units and assay intervals matching them with visible mineralization, observed major structures and matched assay certificates to the assay database
  - reviewed geotechnical and geological logging procedures
  - visited active drill sites and observed drilling procedures
  - verified the collar locations of several drill holes using a handheld GPS
- Completed a database review comparing 5% of the database against original documentation for assays, collar and downhole surveys
- Reviewed on screen all of the 2025 drilling results not used in the 2025 updated and compared them to the Mineral Resource model as a validation check.

Based on his data verification, QP Kim concludes the data is reliable for Mineral Resource estimation.

12.3 Mining

QP Peralta completed a 3-day site visit at the Project site during September 2025. During the site visit the following information was verified:

- Open pit location, including pit orientations aligned with the mineralization for both zones ACMA and Lewis
- WRFs location, including the natural containment features, alternative haulage routes, potential for staged expansion and reclamation
- Mine access constructability, including staging areas identified near camp and processing facilities
- Surface drainage patterns and potential constraints mine design related to topography
- Seasonal constraints, snow, freeze-thaw that could potentially affect the mine plan
- Ongoing drilling and geotechnical work, visited the core shack and observed the geotechnical logging of two drill holes
- Potential constraints to pit footprint, and pit wall orientation.

12.4 Metallurgy and Mineral Processing

QP Drake reviewed the design documents and engineering studies that are associated with the mineral processing facility, including updates to the engineering and designs that were included in the feasibility study, and additional metallurgical testing completed in 2018.

12.5 Project Infrastructure

QP Baluch, QP Sisson and QP Pretorius visited the Donlin Project in September 2025 as detailed in Section 2.

12.5.1 Utilities, Onsite Buildings and Off-site Infrastructure

QP Baluch completed a general technical review of design and original source documents for the Donlin utilities, onsite buildings, and off-site infrastructure including, but not limited to:

- Power plant construction requirements and power distribution
- Fueling logistics and storage requirements
- Civil layout and facility locations
- Site topography to support facility installation
- Location of proposed Jungjuk port and access road alignment to project site
- Location of airfield as suitable for purpose
- Natural gas pipeline design including natural gas pipeline subject matter expert reports.

12.5.2 Tailings and Geotechnical

QP Sisson's data verification focused on confirming data provenance, QA/QC integrity, sequential coverage, and consistency with permitting and design requirements. Data verification activities include:

- Reviewed the source documents
- Examined previous drilling logs, geophysical surveys, and laboratory reports to confirm subsurface conditions and structural geology
- Cross-referenced geological models with new interpretations
- Compared slope angles and stability criteria against design acceptance criteria
- Compared designs against ADNR dam safety guidelines and hazard potential classification
- Analyzed slope angle impacts on strip ratio and operational costs
- Reviewed liner and filter designs to mitigate potential weaknesses in the design

- Examined stability and deformation for seismic and flood events
- Reviewed investigation completeness and adequacy of the design stage.

12.5.3 Water Management

QP Pretorius' data verification focused on confirming data provenance, QA/QC integrity, sequential coverage, and consistency with permitting and design requirements. Verification activities include:

- Spot checks during the site visit to determine the existence of field investigation and monitoring locations
- Review of field logs and laboratory reports to confirm consistency with reported data and summaries
- An examination of prior technical reports for methodological soundness and internal consistency
- A determination that the hydrogeologic model was updated and recalibrated with data collected since 2014
- Evaluated the waste rock characterization program for consistency with industry standards
- Assessed the WTP process flow and datasets
- Reviewed referenced reports and supporting data were methodologically sound and internally consistent.

12.6 Environmental

QP Dockweiler's data verification focused on confirming data provenance, QA/QC integrity, sequential coverage, and consistency with permitting and design requirements. Data verification activities include:

- *Baseline sources and coverage* – Verified the compilation of environmental baseline studies and references (2021–2025), including hydrologic/hydrogeologic models and aquatic monitoring updates, as cited in Donlin Report 2025 and the baseline matrix.
- *Sampling methods and QA/QC* – Reviewed the Integrated Waste Management Monitoring Plan and associated QAPP to confirm field and laboratory procedures, station lists, parameters, QA/QC frequencies, and reporting requirements.
- *Temporal completeness* – Assessed monitoring program timelines (1996–2015; resumed 2019–present) and confirmed sufficiency for current condition characterization.

- *Model inputs and updates* – Verified the lineage and calibration sources for numerical hydrogeologic models and hydrometeorological syntheses (BGC 2011, 2014; updates in 2023).
- *Geochemistry and pit lake predictions* – Confirmed ARD/ML testwork coverage for mine materials and reviewed pit lake/post closure treatment assumptions and citations (e.g., sulfate, selenium).
- *Permitting consistency* – Cross-checked environmental datasets against APDES, IWMP/WMP, 401 certification, and related permits, using the permit matrix and correspondence.
- *Monitoring and reporting* – Verified that monitoring plans include parameters, frequencies, and reporting protocols for construction, operations, closure, and post closure phases.
- *Site visits and consultations* – Documented QP site visit activities and confirmed agency/stakeholder reviews that validated environmental datasets and key issue lists.

12.7 QP Comments on Section 12

12.7.1 Database, Geology and Mineral Resources

QP Kim considers that the data supporting the Mineral Resources are reliable and adequate for the purposes used in the Report.

12.7.2 Mining

QP Peralta considers that the data supporting the Mineral Reserves are reliable and adequate for the purposes used in the Report.

12.7.3 Metallurgy and Mineral Processing

QP Drake considers that the data supporting the metallurgy and mineral process designs are reliable and adequate for the purposes used in the Report.

12.7.4 Utilities, Onsite Buildings and Off-site Infrastructure

QP Baluch considers that the data supporting the Project infrastructure are reliable and adequate for the purposes used in the Report.

12.7.5 Tailings and Geotechnical

QP Sisson considers that the data supporting the tailings and pit design are reliable and adequate for the purposes used in the Report.

12.7.6 Water Management

QP Pretorius considers that the data supporting the water management are reliable and adequate for the purposes used in the Report.

12.7.7 Environmental

QP Dockweiler considers that the data supporting the environmental permitting are reliable and adequate for the purposes used in the Report.

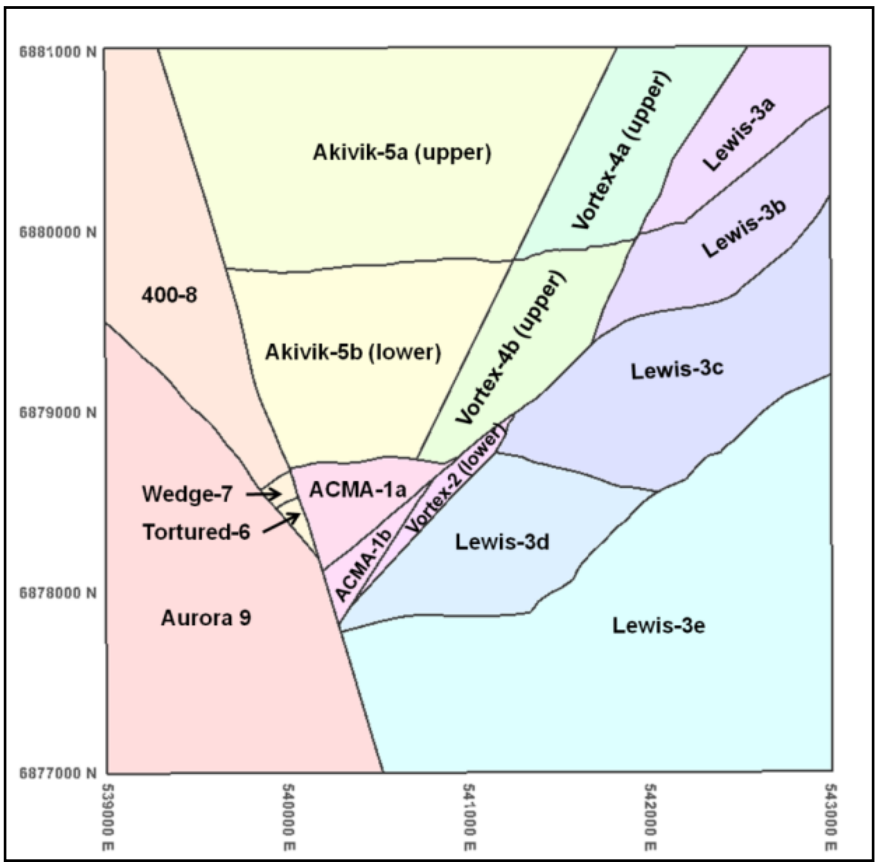
13.0 MINERAL PROCESSING AND METALLURGICAL TESTING

13.1 Geology and Mineralogy

The two main pits (Lewis and ACMA) are each subdivided into spatial geological domains typically separated by faults or other key geological structures as defined in Figure 13-1.

Discreet geological domains were generated by recognizing the main pit spatial domains (Lewis, ACMA, Akivik, 400, Aurora, and Vortex) for the intrusive rocks, but separating out the sedimentary lithologies from every spatial area, and differentiating into the two sedimentary global domains, greywacke (GWK) and shale (SHL). For metallurgical interpretation, all types of domain categorization are considered, used, and applied as appropriate.

Figure 13-1: Geological Domains



Source: Donlin Gold LLC, 2011

13.1.1 Mineralization

A number of mineralogical investigations were undertaken between 2004 and 2007, including work carried out by Amtel, London, Ontario (Amtel), Hazen Research, Golden, Colorado (Hazen), G&T Metallurgical Services, Kamloops, British Columbia (G&T), Barrick Technology Centre, Vancouver, British Columbia (BTC) and SGS Lakefield, Lakefield, Ontario (SGS Lakefield). In 2018, work was performed by FLSmidth Dawson Metallurgical Laboratories, Salt Lake City, Utah and AuTec, Vancouver, British Columbia. (AuTec). Typical sulfide mineralization presence based on these investigations is summarized in Table 13-1.

Gold is associated with quartz-carbonate-sulfide veins defined by four vein stages (Section 7.7).

Pyrite is the dominant occurring sulfide within the deposit. Marcasite is also present, at an approximate ratio of 1:7 as marcasite to pyrite. Arsenopyrite is the main carrier of arsenic within the deposit. Stibnite is the main carrier of antimony.

Table 13-1: Typical Sulfide and Metals Mineralization in the Donlin Ores

Typical Occurrence	Sulfides	Chemical Formula
Major to Minor	Pyrite	FeS_2
	Marcasite	FeS_2
	Arsenopyrite	FeAsS
Minor to Trace	Native Arsenic	As
	Realgar	AsS
	Stibnite	Sb_2S_3
Trace	Chalcopyrite	$CuFeS_2$
	Sphalerite	(Zn,Fe)S
	Tetrahedrite	$(Cu,Fe)_{12}Sb_4S_{13}$
	Galena	PbS
	Pyrrhotite	FeS
	Molybdenite	MoS_2
	Bornite	Cu_5FeS_4
	Covellite	CuS
	Mercury Sulfide	HgS in pyrite
	Native Copper	Cu

13.1.2 Gold Deportment

The primary host mineral of gold is arsenopyrite, which contains 80-90% of the gold as a solid solution, with gold atomically distributed within the arsenopyrite crystal. Fine arsenopyrite is typically highest in gold grade at 500-1,500 g/t (mineral gold content), compared to coarse arsenopyrite at approximately 100-500 g/t. A proportion of the arsenopyrite (particularly coarser grained arsenopyrite) has been shown to have a preferential concentration of gold around the rim of the crystal.

Pyrite is the second most important gold carrier, also in a solid solution form. Typical pyrite gold grades are 1-50 g/t. Similarly, marcasite is also a gold carrier.

Free gold is a minor source of gold (rare occurrence) within the deposit, at less than 1% of the contained gold being free liberated particles less than 20 µm in diameter. Native arsenic is an insignificant carrier of gold in terms of both grade and quantity.

13.1.3 Mercury, Chlorine, Carbonates and Organic Carbon Deportment

For occupational health and environmental considerations, mercury is an important species considered for the process plant design.

Detailed deportment of mercury was undertaken by Amtel in 2007, which indicated that, based upon the concentrate samples tested, pyrite is the principal carrier (66%) of mercury as mercury sulfide in solid solution within the sulfide mineral matrix, followed by the sulfide marcasite (18%). The Amtel study also indicated that fine grained pyrite, marcasite, and stibnite are relatively enriched in mercury content. Arsenopyrite has relatively low mercury content and is an insignificant carrier. No specific mercury minerals have been found or identified in any of the samples examined by Amtel in the deportment work completed.

Donlin mineralization contains chloride with an average concentration of approximately 22 ppm. Detailed testing of chloride deportment carried out by Amtel in 2007 indicated that the principal carrier of chloride in flotation concentrate was muscovite (white mica) as $KAl_2(Si_3Al)O_{10}(OH,F,Cl)_2$, followed by hydroxylapatite, $Ca_5(PO_4)_3(OH,F,Cl)$.

In addition to the refractory nature of the ore, Donlin mineralization contains organic carbon in both sedimentary and intrusive lithologies. The sedimentary ores are relatively abundant in organic carbon, which can exhibit preg-robbing behavior, while the intrusive ores also contain organic carbon, typically in the form of well-ordered graphite. Although this graphitic carbon is believed to be less susceptible to preg-robbing, it remains a factor to consider.

The form of carbonates is an important consideration as carbonates within the flotation tails stream are used to neutralize the acidic liquor coming from the autoclave. The major forms of carbonate identified by Amtel in 2007 were calcite ($CaCO_3$), siderite ($FeCO_3$), and dolomite ($CaMg(CO_3)_2$) and ankerite ($Ca(Fe,Mg,Mn)(CO_3)_2$). The predominant carbonate species remaining in flotation tails was dolomite and ankerite. Calcite was only identified in two test samples and siderite in only one test sample. For the samples tested, the Aurora geological domain consistently contains calcite as the dominant carbonate.

13.2 Metallurgical Testwork

13.2.1 Samples

The current process flowsheet remains as per the 2011 design. The 2006 and 2007 testwork is the basis for informing the flowsheet. The samples used for the metallurgical testwork during 2006-2007 were obtained from a variety of sources. The summary of samples used during the 2006 and 2007 testwork campaigns with the relevant testwork performed is listed in Table 13-2 and Table 13-3. Samples for metallurgical testwork were selected to represent the various lithologies and geological domains.

Additional samples were collected and tested in 2017 and 2018. The main objective of the 2017 drilling campaign was to confirm resource modeling and improve geologic understanding. Some of the drill core from this campaign was made available and was utilized for metallurgical testing.

The metallurgical test results from these samples were reviewed in 2020 and again for this update. Consistent with the findings from 2020, these results did not lead to any changes in the process design.

The test samples are considered representative of the various types and styles of mineralization and the mineral deposit as a whole.

Table 13-2: Summary of Samples Used in 2006 Metallurgical Testing

Preparation Date	Samples Generated	Major Testwork Use
Q1, 2006 HQ Core Drilled 2005	Three bulk composites formed by Hazen and separately tested: • ACMA Int (2005) • Lewis Int (2005) • Sedimentary (2005)	• Hazen bench/pilot flotation (2006) • G&T bench/pilot flotation (2006) • Concentrates from both pilot flotation used for: Bench and pilot pressure oxidation testwork at BTC – Aug to Nov 2006 BIOX Testing at SGS South Africa • Flotation Tails from pilot flotation used for: Bench and Pilot neutralization • Test program products also tested for: Mineralogy (Amtel) Tailings geotechnical / geochemical Thickening and Rheology
Q4, 2006 HQ Core Drilled in 1999, 2001	12 x Variability samples (selected by lithology)	• Bench flotation testing • Mineralogy (variability)
Q4, 2006 HQ Core Drilled in 1999, 2001	155 x Variability samples	• Crushing/grinding hardness variability testing
Q4, 2006 HQ Core Drilled in 1999, 2001	• Bulk composite sample (~5 tonne) based on lithology • Including 9x bulk variability samples	• SGS Lakefield Pilot Flotation Dec 2006 (conventional float) • Concentrates generated used for bench and pilot pressure oxidation at BTC (February 2007, Feasibility Phase I test program) • Flotation tails used for bench and pilot neutralization (February 2007) • Pilot test products also tested for: Mineralogy (Amtel and SGS Lakefield) Tailings geotechnical / geochemical Bench and pilot CIL Thickening and rheology
Q4, 2006 PQ Core Drilled 2006	9 x Variability (selected by lithology)	• Crushing/grinding hardness testing • JK drop weight testing

Table 13-3: Summary of Samples Used in 2007 Metallurgical Testing

Preparation Date	Samples Generated	Major Testwork Use
Q1, 2007 HQ Core Drilled 2006	Bulk Composite (5 tonnes) based on lithology	• SGS Lakefield pilot flotation (February 2007, Feasibility Phase II) • Conventional flotation pilot • MCF2 flotation pilot • Concentrates from pilot flotation used for: Bench and pilot pressure oxidation testwork (BTC) – May 2007 • Flotation tails from pilot flotation used for: Bench and pilot neutralization • Test products also tested for: Mineralogy (Amtel) Tailings geotechnical/geochemical Geochemical Thickening and rheology
Q1, 2007 HQ Core Drilled 2006	9 x Bulk Variability samples (selected by lithology)	• Samples derived from the components forming the bulk composite • Bench flotation testing
Q1, 2007 HQ Core Drilled 2006	149 x Variability samples	• Crushing/grinding hardness testing
Q4, 2006 HQ Core Drilled 2006	102 x Variability samples	• Bench flotation and bench autoclave tests (BTAC) • Mineralogy (SGS Lakefield) Chlorine/fluorine neutron activation

13.2.2 Comminution

Initial grinding testwork was undertaken in the 1990s with additional work completed in 2002–2003 at Hazen that was managed by NOVAGOLD.

Placer Dome Inc. initiated some further work by SGS Lakefield in 2004, which tested ACMA material. During this program, JK and Bond testwork were completed in conjunction with testing the applicability of using high-pressure grinding rolls (HPGRs). During 2006/2007, Barrick initiated three major testwork programs at SGS Lakefield:

- SGS Lakefield 2006 HQ half-core testwork
- SGS Lakefield 2006 PQ whole-core testwork
- SGS Lakefield 2007 HQ half-core testwork.

Testwork completed on fresh core from exploration drilling conducted in 2006 formed the basis for the design of the grinding circuit for the feasibility study.

13.2.2.1 2002–2003 Testwork

A summary of results of the testwork undertaken in 2002–2003 is shown in Table 13-4. The testwork summary indicates that the material tested at that time was moderately hard. These results align with more recent testwork.

Table 13-4: Grinding Testwork Results from Hazen

Pit	Location	Ore Type	BWI (kWh/t)
ACMA	-	Intrusive	14.3
	-	Sediment	13.0
Lewis	Rochelieu	Intrusive	14.1
	Rochelieu	Sediment	13.3
Lewis	North	Intrusive	13.9
	North	Sediment	13.1
Lewis	South	Intrusive	15.1
	South	Sediment	12.1

13.2.2.2 2004–2005 Testwork

Placer Dome Inc. initiated a testing program at SGS Lakefield in 2004. This work was performed on two large samples from the Donlin deposits with the objective of comparing the power efficiency of using HPGR as opposed to semi autogenous grinding to prepare the ore ahead of ball milling. The results of the JK and Bond work are summarized in Table 13-5.

Table 13-5: Summary of Grindability Testing

Sample Composite	CWI (kWh/t)	A x b	ta	Density (g/cm^3)	RWI (kWh/t)	BWI (kWh/t)	Ai (g)
ACMA Sedimentary	9.9	38.7	0.39	2.76	14.7	14.0	0.205
ACMA Intrusive	11.3	52.8	0.31	2.69	13.5	14.7	0.181

Note: CWI = Bond low-energy impact; RWI = rod work index; BWI = Bond ball work index; Ai = abrasion index

The following key items were identified from the testwork:

- The sedimentary rock sample was described as moderately hard with respect to resistance to impact, as measured by the impact work index (CWI) and drop-weight test (A x b).
- The intrusive rock sample measured as hard in terms of low-energy impact and medium in the drop weight test (DWI).
- The sedimentary rock sample was moderately hard with respect to resistance to abrasion breakage (ta), while the intrusive sample was hard.
- Both samples can be categorized as medium in terms of Bond rod mill (RWI) and ball mill work indices (BWI).
- Both samples were only mildly abrasive (Ai).

The two samples were also submitted to a series of bench-scale HPGR tests. The tests showed that the HPGR successively reduced the 13 mm material feed to ball mill feed size with an energy input of 2.07 and 1.94 kWh/t for the sedimentary and intrusive rock samples, respectively. The BWI's of the resultant samples were tested at 75 µm and produced values of 12.5 and 13.3 kWh/t, respectively, indicating that a reduction (in the BWI) was attributable to the HPGR processing.

SGS Lakefield recommended further HPGR work at Polysius with the REGRO unit accompanied by ATWAL abrasion testing. The testwork by Polysius judged that a specific energy input in a range of 1.5-2.0 kWh/t was the optimum for HPGR comminution of the test samples provided. A specific throughput rate of approximately 250 ts/m^{3}h was achieved with the grinding force selected.

The abrasion testwork indicated an ATWAL wear index of 8.3 g/t for sedimentary rock material and 24.8 g/t for intrusive rock material. This testwork was done at the standard feed moisture of 1%. This wear is medium abrasive compared with Polysius' database at the time.

13.2.2.3 2006 HQ Core Testwork

SGS Lakefield conducted an extensive test program to determine grinding parameters for the Donlin mineralization. The samples used were HQ drill core taken from the 1999 and 2002 drilling campaigns, which had been stored at the exploration site. Parameters were obtained from the following tests:

- Minnovex SAG power index (SPI), crusher index (Ci), and modified Bond ball mill work index (Modified Bond test)
- SMC drop-weight index test
- Bond low-energy impact, rod mill work index, ball mill work index, and abrasion index
- HPGR energy test.

13.2.2.4 2006 Whole PQ Core Testwork

HQ drill core was too small in diameter for full drop weight tests for JKSimMet modeling. Larger diameter PQ holes were drilled in a 2006 drilling campaign targeting bulk mineralized areas of the deposit and covering the full range of lithologies. The samples from these drill holes were processed by SGS Lakefield to develop JK grinding parameters, in addition to conventional Bond ball mill and rod mill work index numbers. This information was important for use in checking the grinding parameters developed from the HQ testwork. Comparing the work index properties from the PQ core results with the 2006 HQ core results, showed that the freshly drilled PQ core was consistently harder than the HQ core samples that had been stored for several years. It was recommended that a second set of HQ samples be selected from freshly drilled core in 2006.

13.2.2.5 2007 HQ Core Testwork

A total of 149 additional samples from fresh drill core were tested by SGS Lakefield in early 2007.

In parallel with this testwork, AMEC undertook grinding circuit design trade-off studies, which indicated that a semi autogenous mill, ball mill and pebble crushing (SABC) circuit design was the preferred option. Therefore, the test program was designed to maximize generation of comminution properties relating to the required parameters for the SABC circuit. The parameters tested in the program were then restricted to:

- Minnovex SAG power index
- Minnovex crusher index
- Minnovex modified Bond ball mill work index
- Rod mill work index
- Ball mill work index.

Results indicated consistently harder comminution properties than the 2006 test program using older core and were regarded as being a more reliable predictor of the hardness in the deposit.

Therefore, it was recommended that comminution data obtained from core predating 2006 be normalized to fresh rock conditions for design calculations when results are based on core that had been exposed to the Alaskan climate for an extended period on time. Normalization was carried out on the 1999 and 2001 HQ core data. Results were preserved to provide more test samples to augment the variance analysis and subsequent population of the geological model (see Section 13.2.2.8).

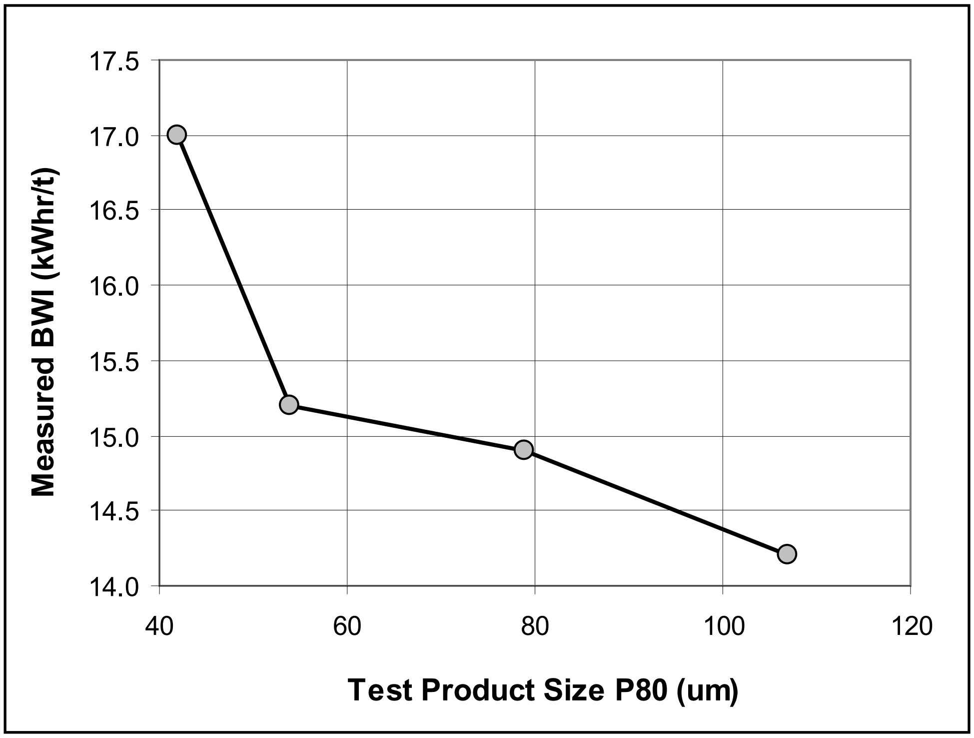
13.2.2.6 Effect of Grind Size on BWI

The potential adoption of a “mill chemical-float, mill chemical-float” (MCF2) flowsheet would result in two different ball mill product size distributions. The products from the primary ball milling circuit would target P_{80} 120-150 µm, and the secondary ball mill would target P_{80} 50 µm. It is known that in some circumstances the measured BWI of a test sample will vary according to the target P_{80} of the BWI test. To determine if this is the case for the Project, and to quantify that effect, a number of additional BWI tests at different final product sizes were undertaken by SGS Lakefield. For the feasibility grinding circuit modeling and design, an overall adjustment model was applied to the BWI value used, based on these results.

Using the blended pilot-plant feed sample, an additional set of BWI tests were undertaken at varying product sizes.

Results show that as the target product P_{80} size decreases, the measured BWI of the test sample increases. On the blended pilot-plant sample a significant increase in BWI occurs between P_{80} 54 µm and 42 µm. Figure 13-2 is a graphical representation of the pilot-plant blend feed sample results.

Figure 13-2: Test P_{80} vs. Measured BWI Results on Blend Composite Sample



Source: AMEC, 2011

13.2.2.7 Feasibility Comminution Circuit Selection

In 2006, Barrick contracted Orway Mineral Consultants (OMC), Australia, and SGS Lakefield (Minnovex), to perform appraisals of the various comminution options. Capital and operating cost alternatives for four options were investigated:

- Option 1 – autogenous milling with ball milling and crushing of pebble reject (ABC)
- Option 2 – semi-autogenous milling with ball milling and pebble crushing (SABC)
- Option 3 – coarse crushing followed by HPGR with ball milling
- Option 4 – fine crushing followed by ball milling.

AMEC performed a trade-off study based on the OMC recommendations to investigate the economics of various options, as well as non-economic factors.

Option 2 (SABC circuit) was selected as the comminution circuit for the following reasons:

- Lowest capital cost
- Ability to cope with the clay fraction in the ore

- Ability to cope with the climatic conditions
- General ease of operation and maintenance
- Flexibility in throughput rates
- Widely applied technology in the milling industry
- Barrick's extensive experience in SABC circuit application.

13.2.2.8 Geostatistical Assessment

AMEC developed a geometallurgical model using the comminution data (Ci, SPI, and BWI) to investigate relationships that may exist between the mineralized sample hardness, rock lithology, spatial location, grades (Au, S, As, Mg, Sb), and rock quality designation. The key results of the geotechnical assessment are summarized below:

- No correlations could be determined between these comminution properties, and therefore each parameter had to be assessed individually.
- Variability of the crushing index data is quite high within and between lithologies, and is dominated by lithology as indicated in Table 13-6.
- The SPI results are dominated by lithology, with the differences between lithologies being significant. Based on analysis of the variance of the data, four separate categories were defined and used to populate the block model. These are summarized in Table 13-7.
- Lithology was determined to be the significant variable influencing ball work index (Table 13-8). With the exception of the SHL and MD lithologies, the variance in results within a lithology was quite low due to a small number of test results.

Table 13-6: Estimation of Crushing Index (Ci)

Domain	Ci Mean	Standard Deviation	Sample Count
RDXL	21.9	8.2	33
Non-RDXL West (<541,000)	14.2	6.7	158
Non-RDXL East (>541,000)	14.8	6.9	81
Total/Average	**15.3**	**7.4**	**272**

Table 13-7: Estimation of SAG Power Index (SPI)

Domain	SPI Mean	Standard Deviation	Sample Count
SHL	58	17	9
RDXL	71	14	33
RDX	85	22	79
Remaining Lithologies	95	21	152
Total/Average	**88**	**23**	**273**

Table 13-8: Estimation of Bond Ball Work Index (BWI)

Domain	BWI Mean	Standard Deviation	Sample Count
RDX + RDF + MD	15.5	1.3	107
GWK + SHL	14.6	1.5	48
RDA	13.8	1.3	46
RDXL	14.4	1.1	33
RDXB	16.4	1.2	39
Total/Average	**15.0**	**1.5**	**273**

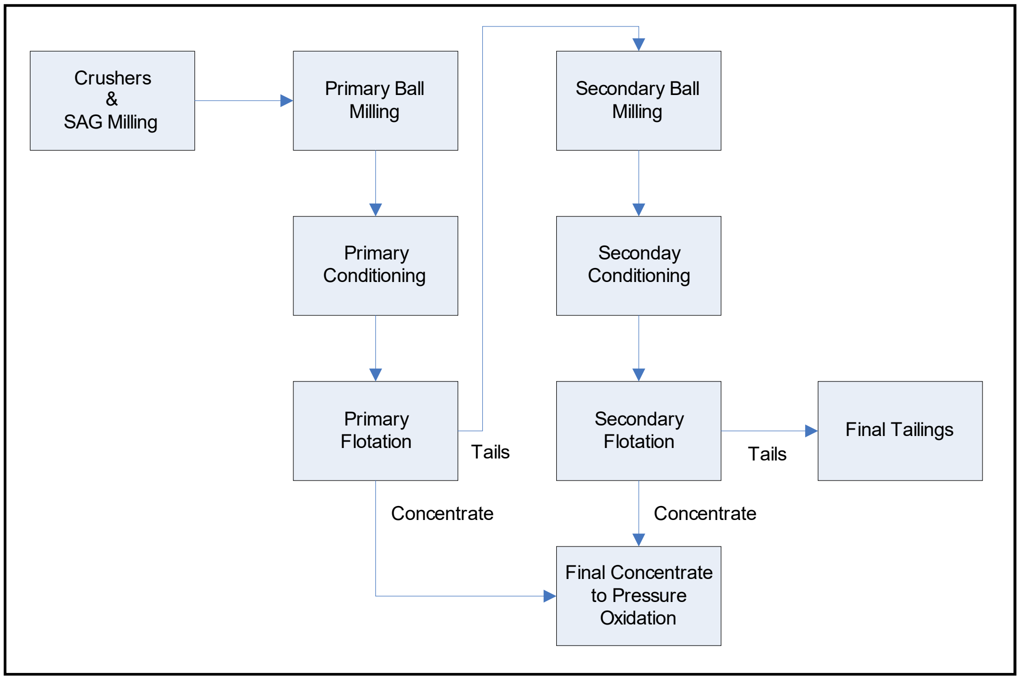
13.2.2.9 MCF2 Grinding Circuit Design Method and Capacity Definition

During the first quarter of 2007, pilot flotation testing was carried out at SGS Lakefield to assess the potential of a MCF2 flowsheet to improve confidence in the overall gold recovery and economic value of the Project. The MCF2 flowsheet shown in Figure 13-3 incorporates two separate stages of grinding and flotation utilizing the SABC configuration as the primary grinding step.

Mineralogical assessment, flotation bench-scale, and flotation pilot scale testwork have indicated that the primary rougher feed P_{80} should be in the range 120-150 µm, and that the secondary rougher feed P_{80} should be in the range 50-60 µm, to achieve high flotation gold recoveries. These assumptions were applied to the grinding circuit modeling and design.

Through various studies, the optimal mill throughput was determined to be 53,500 t/d.

Figure 13-3: Illustration of MCF2 Generic Flowsheet



Source: AMEC, 2011

13.2.3 Flotation

13.2.3.1 Summary

Extensive bench and pilot flotation testwork has been carried out on mineralized material from 1995-2007.

The objective for the flotation circuit is to provide high gold recovery (+90%) to a sulfur concentrate (greater than 6.5% total sulfur content) for pressure oxidation feed. Once the feed contains sufficient grade of sulfide sulfur as fuel to generate heat and achieve the required operating temperatures within the autoclave, there is little overall net benefit in increasing the sulfur grade of the concentrate any further. This is advantageous as overall gold recovery in flotation concentrate decreases significantly as concentrate grade is increased.

A general target of 7% sulfur grade has been selected for the final flotation concentrate, noting that the autogenous grade is determined to be approximately 6.5% sulfur, varying slightly with changes in the solids/liquid ratio, the carbon, and arsenic grade of the autoclave feed. The gap between 6.5% autogenous grade and the target 7% grade is provided to allow for variability in feed and to minimize any requirement to add external heating to the autoclaves.

13.2.3.2 1995–2006 Testwork

Wood reviewed the flotation testwork and made the following observations.

Bench Testing

- The testwork results indicated that producing a bulk concentrate was the optimum route to maximize gold recovery.
- A variety of reagent schemes were attempted. The best reagent system was using plant acid, copper sulfate, xanthate, and dispersant.
- Nitrogen-based flotation technology was tested extensively throughout the Project but provided no overall benefit.
- Adequate retention time was found to be very important. To achieve maximum recovery, a long flotation retention time, of 114 minutes, was found to be necessary.
- The required particle grind size reflects the presence of the two different mineralization types in the feed. While the intrusive mineralization can tolerate coarser sizes in the range of 75–110 µm, the sedimentary mineralization performs best in the range of 60–80 µm for the conventional flotation flowsheet.
- Mass pull from the rougher and scavenger circuits was dictated by entrainment of clays during the long residence time of the flotation. With the use of dispersants, lower flotation feed pulp densities and cleaning, the overall mass pull to final concentrate could be decreased to approximately 15%.
- The process development testwork indicated that froth recovery was a critical factor. Froth recovery can be enhanced by using crowding cones in the flotation machine and through launder design.
- Because of the different flotation responses of the samples, testwork was performed to assess the outcome of blending the two main ore types. Given adequate reagent dosages and residence times, it proved possible to produce high flotation recoveries with the LOM test blends provided for the testwork.

Flotation Pilot-Plant Testing

- Extensive testwork was performed in 2004 and 2006, demonstrating that a recovery of 91-92% on a LOM lithology blend was possible, confirming the performance of the conventional rougher/scavenger flowsheet under continuous operation.

13.2.3.3 Mineralogy Summary

Numerous mineralogy studies completed on many different flotation test samples, present relatively consistent themes with regards to achieving high gold recovery (+90%) to concentrate:

- Fine arsenopyrite must be recovered to final concentrate.
- Mineralized material (particularly from Lewis) must be ground finer than a P_{80} of 75 µm to improve the liberation characteristics.
- Pyrite hosts a significant portion of the gold in a solid solution and therefore must also be recovered to concentrate.
- Over-grinding of the liberated sulfides to a particle size less than 10 µm is detrimental to flotation recovery.
- The liberation properties of the sulfide mineralization are variable.
- The expected concentrate sulfur grade from a flotation circuit producing high gold recovery (+90%) is going to be less than 10% sulfur, because of the presence of low-grade composite particles of sulfide with gangue, floatable gangue (such as carbon and carbon/clay binaries), and non-floatable gangue as entrainment.

13.2.3.4 2006 Pilot-Plant Testing

G&T

Testwork was performed at G&T to confirm process parameters on a blended sample of 50% Lewis Intrusive, 25% ACMA Intrusive, and 25% Sedimentary. This pilot flotation testing did not produce flotation gold recoveries (at required grade) that were comparable to those achieved from a bench test undertaken on the same sample. Pilot results showed recovery in the range of 83-85% in the blended sample, compared to the bench flotation test at 91-93%.

Subsequent testing was undertaken to explain this discrepancy through exhaustive re-testing of both the bench and pilot-plant flotation cells under different operating and test conditions. This work led to the conclusion that the froth conditions generated by the G&T pilot plant were hindering recovery of sulfide composite particles to the concentrate. Consequently, the importance of froth recovery was emphasized in design.

SGS Lakefield

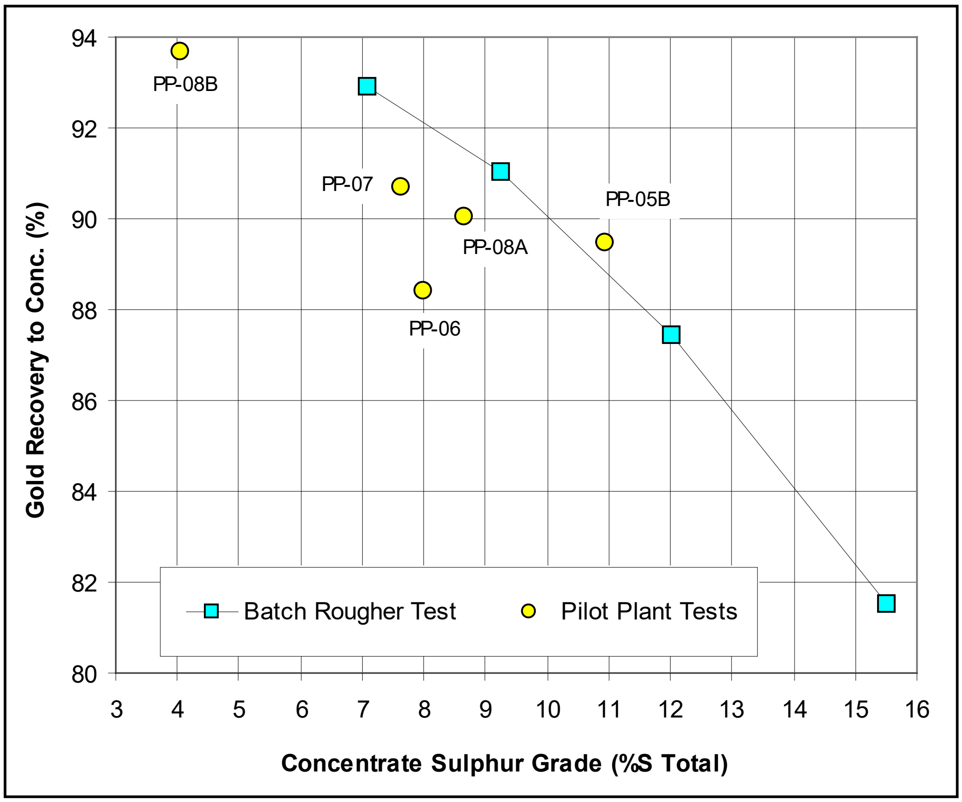
The SGS Lakefield pilot run was initiated to understand which key items were affecting gold recovery under pilot conditions.

The testwork was performed on available material and later followed up by another pilot run using freshly drilled core to eliminate any issues related to sample weathering.

It was found that the poorer flotation results were attributed to partially geologically oxidized ore in some of the upper areas of the deposit as evident from the 2007 variability testing.

The pilot-plant run showed approximately 91% gold recovery to a 7% sulfur concentrate was achieved compared to a bench test result of 93% recovery on a LOM lithology blend (Figure 13-4). This was a significant improvement from the earlier pilot work in 2006.

Figure 13-4: Comparison of SGS Lakefield Dec 2006 Key Bench and Pilot-Plant Results



Source: AMEC, 2011

13.2.3.5 Bench Testing of MCF2 Flowsheet

To further improve the overall performance of the flotation circuit design, an alternative MCF2 flotation configuration was considered, and some comparative bench testing of this option commenced.

Initial tests were conducted by SGS Lakefield at a nominal grind of P_{80} 40 µm for the second stage product and with varying primary stage grind sizes. The results showed that the alternative MCF2 grind/flotation configuration was realizing a measurable improvement in gold recovery of approximately 2% at the same final concentrate grade as the conventional grind/float configuration.

These tests also suggested that the primary grind size selection was not a key parameter and that there could be some flexibility in the final size selection for the stage of grinding.

Subsequent bench tests were undertaken to attempt to quantify the effect of the secondary grind size on gold recovery. The results did not clearly define an optimal secondary grind target but a nominal P_{80} of 50 µm was selected for the subsequent MCF2 pilot run based on trends identified through previous mineralogical work.

13.2.3.6 2007 Pilot-Plant Testing

A second SGS Lakefield pilot-plant campaign was undertaken with the aim to confirm pilot-plant recovery of the conventional flotation circuit on a LOM lithological blend using freshly drilled core.

The pilot runs produced on average 92.8% recovery to a 7% sulfur concentrate grade compared to the bench tests, which indicated approximately 94.0% recovery. A second series of pilot runs confirmed that a mild steel primary mill with high chrome media could be used in lieu of the original stainless steel mill with high chrome media. A third series of pilot tests were undertaken where the scavenger concentrate was reground before cleaning. A slight improvement in the grade/recovery profile was evident. A fourth series of pilot tests evaluating the MCF2 configuration showed a clear trend of improved recovery. At a concentrate of 7% sulfur grade, the recovery difference between MCF2 and conventional flotation is approximately 1.8%. This compares reasonably well to the approximately 2% recovery improvement indicated by the initial MCF2 screening bench test results.

Based on these results the MCF2 option was selected as a potential base case for the feasibility grinding/flotation circuit design.

13.2.3.7 2007 Variability Testing

The flotation variability testwork program consisted of 149 flotation tests using 102 test samples covering a range of lithologies and geological domains using core drilled throughout 2006. The number of samples selected for each lithology was based on the proportion of the orebody represented by that lithology.

Of the 102 samples, 22 were characterized as having some form of partial geological oxidation, and the remaining 80 were considered unoxidized fresh rock. Two types of variability bench scale flotation tests were carried out, a modified Minnovex Flotation Test (MFT), and a conventional bench flotation test (CFT).

Analysis of test data indicated:

- RDX, RDA, RDXL, and MD lithological domains appear to behave similarly in terms of average and standard deviation, and together have an average flotation recovery of ~96%.
- RDXB lithology is the worst-performing intrusive, with an average recovery of 94.8%. This is characteristic for RDXB, which has relatively high graphitic carbon content compared to the other intrusive rocks.
- RDF is the best-performing intrusive rock, with an average gold recovery of 97.7% and relatively low variance in performance.
- GWK and SHL recoveries are, on average, lower (91.5% and 89.8%, respectively), with a relatively large variation in performance.
- The GWK domain dataset does exhibit a weak correlation of flotation recovery with both arsenic and gold grade, noting that there is a natural strong relationship between gold and arsenic head grades. Given the relatively poor correlation with the GWK dataset this type of relationship is not recommended to predict recovery for the GWK ores. Instead, a non-weighted average of the test data should be used.
- For the SHL test results, no obvious correlations are evident to improve recovery predictions, and a non-weighted recovery average is recommended.

Test samples were also characterized via geological domain, which is mainly based on physical location rather than rock type. Examining the statistical representation of recovery data based on geological domain suggests:

- Samples from 400, ACMA, Akivik, and Aurora behave fairly similarly, with an overall average recovery of approximately 96.5%
- Samples from Lewis have an average recovery of 94.3%
- Samples from Vortex have an average recovery of 95.2%.

The Vortex geological domain is adjacent to the Lewis domain; the Lewis geological domain has a high content of RDXB intrusive where it is mainly concentrated.

13.2.3.8 Effect of Geological Oxidation on Flotation Performance

The upper portion of the deposit contains mineralization that shows some sign of geological oxidation or weathering. To quantify the potential impact of this oxidation on flotation, a series of 22 samples were selected. Results indicated that the presence of some form of geological oxidation significantly affects the flotation performance. The average gold recovery is 72%, with a relatively high standard deviation of 22%.

Where there was insufficient sulfur content in the test sample to generate a 7% concentrate, gold recovery was determined at the 15% mass-to-concentrate point instead.

13.2.3.9 Flotation Circuit Scale-up and Modeling for Feasibility Design

The conventional approach to designing flotation circuits focuses on the use of scale-up factors from bench-scale testwork to determine the residence time required for a full-scale plant.

JKSimFloat was used to model the flotation circuit design. Simulated results for a 53,500 t/d flotation circuit using the flotation parameters as tested suggested a gold recovery of about 94% at a concentrate grade of 7.2% sulfur, for a two row circuit configuration with head grade of 1.12% sulfur and 2.37 g/t gold. Flotation cell sizes of 300 m^3 were selected based on lip loading data.

13.2.3.10 2018 Bench Flotation Testwork

AuTec completed metallurgical testing to investigate optimization opportunities recommended in previous studies. Samples were selected from the Lewis pit representing the first 10 years of operation, and from the ACMA pit representing the first five years of operation. Composites consisted of intrusive, sedimentary and mixed (intrusive/sedimentary blends) samples.

Lewis and ACMA Pit

Intrusive and mixed composites were used as flotation testwork feed. Testwork investigations included grind size, pulp density, pH, flash flotation, MF1/MCF1/MF2/MCF2, oxide material flotation and CIL.

Oxide mineralization flotation: Intrusive samples with lower levels of oxidation indicated faster gold kinetics and recoveries (80-85% for Lewis and 90% for ACMA) compared to higher oxidation level samples. Results of the RDX and GWK Vortex composite samples indicate low recovery at 50-55% for Lewis and for the GWK Vortex at ACMA, a 38% recovery.

The results from the rougher flotation and CIL tests indicated that samples with a lower degree of oxidation were suitable for flotation but resistant to cyanidation. In contrast, the more oxidized samples did not perform well in flotation but responded favorably to direct CIL processing.

ACMA Pit Sulfides

This investigation was mainly focused on the effects of grind size, pulp density, pH and reagent optimization for the ACMA sulfide samples. Results are as follows:

- Reducing the flotation pH from pH 5.5-4.5 improved flotation kinetics
- For intrusives, a primary grind size P_{80} of 75 µm appeared to be optimum for gold recovery without a second stage grind.
- A coarser primary grind size (P_{80} of 100 µm) is possible as it yielded similar rougher flotation gold kinetics as compared to the grind size P_{80} of 75 µm.
- While maintaining gold recovery at 94-95% rougher flotation time could be reduced to:
    10. 4 minutes with a mass pull of 7% for intrusive composite. At 15 minutes retention time the Intrusive had a 97% gold recovery and a 13% mass pull.
    11. 15 minutes with a mass pull of 17% for sedimentary composite
    12. 15 minutes with a mass pull of 15% for mixed composite.
- Collector PAX dosage was 55 and 100 g/t for the intrusive and mixed composites, respectively. Other reagents including promoter A7249, dispersant or gangue depressants (sodium meta-silicate nonahydrate, corn starch, PEO and CMC) and sulfide activator (copper sulfate pentahydrate) did not improve gold recoveries for the intrusive and mixed composites. Reagent optimization was not investigated for the sedimentary composite.
- Use of stainless steel or mild steel grinding media resulted in similar final flotation gold recoveries for the intrusive and mixed composites.
- MCF2 or MF2 flowsheets with first and second stage grinds at P_{80} of 150 µm and 45 µm respectively, did not yield a better gold recovery than a single grind (MF1 and MCF1) at P_{80} of 75 µm for the intrusive composite.

Although the ACMA test report noted that a single grind to P_{80} of 75 µm could achieve the required recoveries for the intrusive composites and that a second grind might not be beneficial, it is noted that the mixed composites indicated that the MCF2 circuit configuration for both Lewis and ACMA performed better in terms of recovery and kinetics.

Further testwork was initiated to test the improved flotation scheme using a pH of 4.5 with a grind to a P_{80} of 75 µm. No locked cycle or pilot tests were completed for these samples.

13.2.4 Pressure Oxidation

13.2.4.1 Chemistry of Pressure Oxidation and Hot Cure

Pressure oxidation in gold processing generally refers to the oxidation of gold bearing sulfide minerals to metal sulfates using a combination of heat (typically 200-230°C), acid, and oxygen sparging in a specifically designed pressure vessel. The breakdown of the sulfide particles effectively releases the gold locked within the mineral matrix, rendering it amenable to leaching by cyanidation.

13.2.4.2 2004 Batch Autoclave Testing

Dynatec Scientific Laboratories, El Paso, Texas carried out bench-scale autoclave testing of four composite samples including ACMA Intrusive, ACMA Sediment, Lewis Intrusive, and Lewis Sediment. The scope of the test program included kinetic and locked-cycle mass balance pressure oxidation tests on the concentrates, followed by neutralization tests on the POX discharge liquors and (CIL) cyanidation tests.

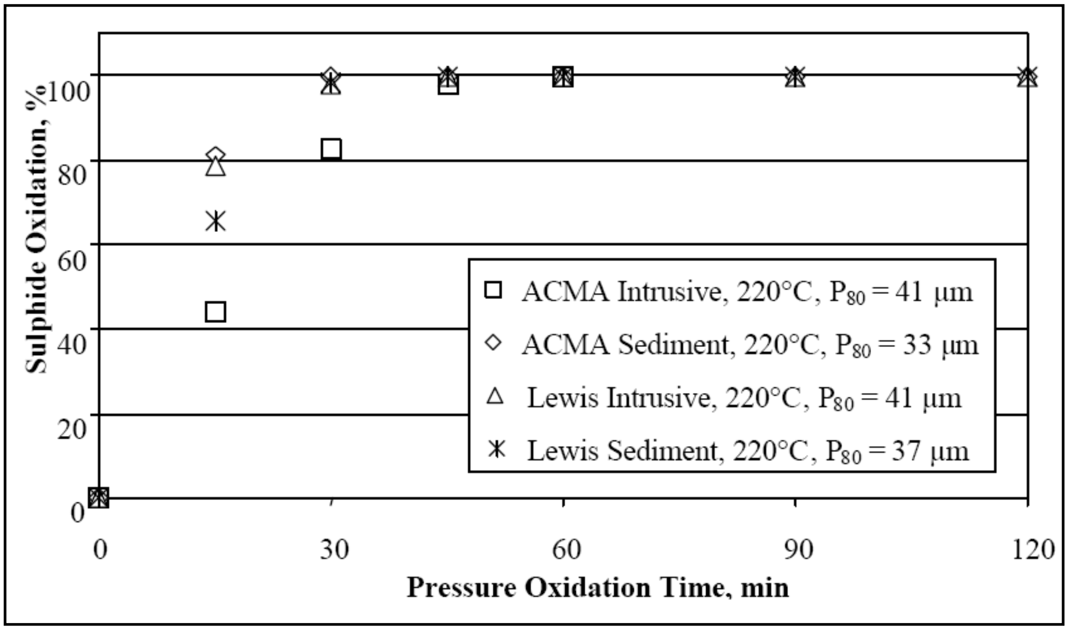
The concentrates were relatively fine, with P_{80} levels of 33-41 µm (82-89% minus 44 µm), and were tested without further size reduction for comparison purposes. Direct CIL cyanide leaching of the unoxidized feeds yielded gold extractions between 3% (ACMA Sedimentary rock) and 11% (Lewis Intrusive rock).

Higher autoclave oxidation kinetics were observed at 210°C and 220°C than at 200°C. Gold extractions were highest from the solids oxidized at 220°C. All subsequent pressure oxidation testwork on all four concentrates were, therefore, conducted at 220°C.

As shown in Figure 13-5, the sulfide sulfur oxidation kinetics was rapid, with more than 98% oxidation achieved within 30 minutes.

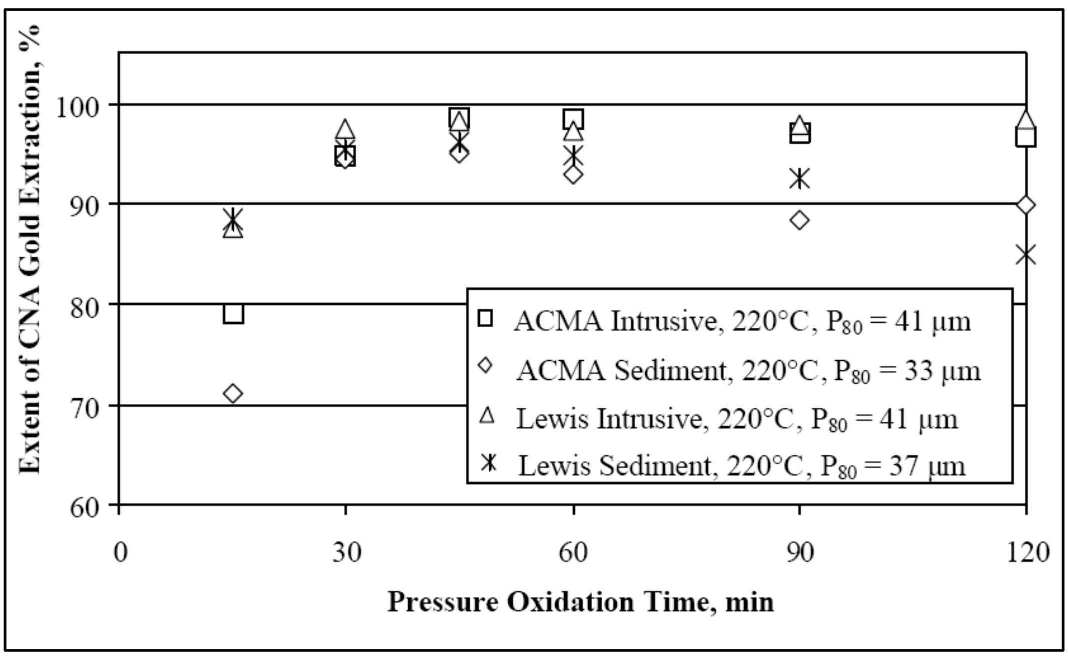
Gold extractions from the oxidized concentrates, as shown in Figure 13-6, were correspondingly high after 30 minutes of pressure oxidation and improved marginally to their maximum values of between 95.1% (ACMA Sedimentary rock) and 98.5% (ACMA Intrusive rock) after 45 minutes of oxidation. With extended pressure oxidation time, however, the gold extractions declined, most markedly for the sedimentary rock concentrates which had relatively high organic carbon content.

Figure 13-5: Sulfide Oxidation Pressure Oxidation Kinetics at 220°C



Source: AMEC, 2011

Figure 13-6: Gold Recovery Profiles from Pressure Oxidation at 220°C



Source: AMEC, 2011

A retention time of 45 minutes was selected for the POX in the subsequent material balance and locked-cycle testwork.

In the locked-cycle testwork, the POX tests were conducted at pulp densities approaching those anticipated for the discharge slurries in commercial autoclave operation. The extents of sulfide sulfur oxidation with the 45-minute POX retention time at 220°C exceeded 98%, with more than half over 99%. There was no systematic change in the oxidation extent with increasing cycle number, indicating that the recycled solution did not affect the sulfide oxidation.

Analysis of selected solution samples for gold and silver indicated that these were below their respective detection limits of 0.2 mg/L and 1 mg/L, respectively in the POX discharge solutions.

Stirred tank CIL cyanide leach gold extractions varied from 90.3-98.8% for the four oxidized concentrates, with median extractions of 93.5-97.4%.

Subsequent testing has indicated that oxidation rates and performance of the batch tests are strongly affected by the decision to pre-acidify the concentrate sample charge, or not, both at pilot scale and bench scale.

13.2.4.3 2006 Batch Autoclave Testing

A summary of the key conclusions of the various tests undertaken on composite samples by BTC are mainly limited to gold recovery performance in the following sections.

Baseline Tests

A series of batch tests were conducted to understand the potential impact of autoclave temperature, oxidation time, pre-acidification, oxygen concentration, and thiocyanate concentration on autoclave performance.

The results of the bench autoclave tests (BTAC) indicate that oxidation rate increases with POX temperature. The optimum temperature for the gold recovery profile is 230°C, which was marginally better in performance than at 220°C.

In conclusion, autoclave temperatures of more than 192°C are required to achieve >92% gold recovery within a nominal 1 hour autoclave residence time, and temperatures of 220-230°C provide maximum recovery values.

Some preliminary BTAC tests were undertaken to evaluate the potential impact of thiocyanate (SCN) dosed into the feed slurry on the CIL gold recovery of the autoclave products. No detrimental impact on recovery was indicated.

From the autoclave test data, it was noted that gold recovery improved with increasing POX residence time. Therefore, experiments were undertaken to test the potential application of higher temperature POX (230–240°C). It was seen that gold recovery does improve with extended autoclave time and that this improvement is indeed accelerated at 240°C.

2007 BTAC Testing Phase 1 Composite Concentrate Sample

The BTC carried out a series of BTAC tests on a sub-sample of the concentrate generated from the SGS Lakefield pilot flotation test program. The test program aimed to investigate the pressure oxidation characteristics of the new composite concentrate sample against previous testwork, and to investigate the effect of autoclave parameters on pressure oxidation performance. The best result obtained from the testwork was at 220°C with 45 minutes of residence time.

13.2.4.4 2006 Pilot-Plant Testing

Two phases of autoclaving pilot tests were conducted. Discussion of results on the gold recovery aspects of the pilot-plant results are presented as a whole.

A series of 220°C and one 225°C pilot campaign were conducted to determine what recovery could be achieved with more complete oxidation levels. This run successfully demonstrated that the CIL gold recoveries of the autoclave residue of more than 96% were possible.

A pilot test run at 240°C achieved high gold recoveries quickly, albeit with faster oxidation kinetics than at 220°C. An improvement in recovery is evident; however, it is not practical to run at the higher temperature.

13.2.4.5 2007 Phase 1 Pilot-Plant Testing

An additional autoclave pilot campaign was undertaken at the BTC to verify the autoclave design parameters and potential gold recovery results for the 2007 FS. This pilot run attempted to explore the three different operating temperatures, 200°C, 210°C, and 220°C utilizing pre-acidification of the concentrate.

Higher than design residence times were tested to try to achieve greater sample oxidation at the autoclave discharge, than had been achieved in previous campaigns. Each of the pilot-plant autoclave compartments were sampled and CIL tests undertaken on each sample.

Results showed a maximum gold recovery of approximately 96.9% was reached in compartments 3 and 4 for the 225°C test run, representing a residence time of 50-55 minutes. For the 220°C run, a maximum recovery of 95-96% was achieved in compartment 5 at a residence time of approximately 55 minutes. CIL gold recovery dropped significantly towards the discharge of the autoclave, after peaking at 96.0-96.4% near compartments 3 and 4. The results concluded that as the operating temperature of the autoclave was decreased, the required residence time for reaching maximum gold recovery increased.

13.2.4.6 Phase 2 Pilot-Plant Testing

The aim of the Phase 2 testwork was several-fold:

- To operate the pilot unit more closely to the design optimum autoclave residence time as determined from Phase 1
- To demonstrate that target CIL gold recoveries can be achieved on final products from the autoclave, not just on compartmental sub-samples
- To confirm potential downstream CIL gold recoveries
- To confirm the selected design criteria for the pressure oxidation circuit for the 2007 FS
- To provide more autoclave profile data for oxidation rates and gold recoveries.

The concentrate feed sample was obtained from material generated from the SGS Lakefield pilot flotation test program undertaken in 2007.

Concentrate pre-acidification reduced the carbonate content (inorganic carbon) of the concentrate from 0.35-0.05%, representing approximate 86% dissolution of the contained carbonates.

The pilot-run investigated two operating temperatures, 220°C and 225°C.

Oxidation rates were reasonably fast and consistent, and measurable sulfide oxidation was completed between 37 and 42 minutes residence time.

Sampling commenced when the autoclave was at steady state and continued for four to five hours.

It was found that if the autoclave residence time was permitted to exceed 50 minutes, then CIL gold recovery of the autoclave discharge drops to 93-94%. Similarly, if autoclave residence time was too short, then recovery was less than optimum at 96-97%. Optimum gold recovery, exceeding 97%, was recorded at around 45-49 minutes.

During operation of the pilot campaign four profile samples were collected approximately every 30 minutes. Results from these samples confirmed the optimum residence time to be in the range 37-47 minutes. Optimum gold recovery from the autoclave appears to correspond with the "just completed" extent of sulfide sulfur oxidation. Excess oxidation after that point is detrimental to gold recovery.

Test results showed the organic carbon content (by assay) had decreased from 0.78% to an average of 0.57%, with approximately 25% of the organic carbon being oxidized in the autoclave. The graphitic carbon (by assay) showed negligible reduction in assay grade in the autoclave. Inorganic carbon in the feed was reduced to below assay detection limits by the pre-acidification process and no apparent trends were discernible.

Standard gas testing was performed on the main vent gases from the autoclave, ahead of the water scrubber, with the purpose of quantifying the oxygen, carbon dioxide, carbon monoxide, and total hydrocarbon (THC) generation rates from the autoclave that would be fed to the gas scrubbing system.

Hot Cure

A testwork series on hot curing optimization was undertaken to confirm the proposed hot cure section design and to provide the latest information for future optimization.

The analyses of the hot cured solids and liquor components show that sulfur in the solids is dissolving, with a corresponding decrease in solids mass and liquor sulfuric acid concentration and a consequential increase in liquor iron content. A decrease in lime consumption is required for pH adjustment to 11 and CIL gold recovery improves slightly at the six-hour residence time point. The hot cure circuit was designed with a residence time of six hours. Lime consumption of well-washed hot cure solids to pH 9 was relatively constant at around 3-4 kg/t of hydrated lime.

Washing tests of hot-cured autoclave product followed by neutralization reduces the consumption of lime by up to 98%. This demonstrated that operating a CIL circuit at a pH of 9 is optimal for limiting lime consumption.

The benefit of operating a CIL circuit at pH 9, in terms of lime consumption, is also demonstrated from this work.

Summary

The main results of the 2007 phase 2 pilot autoclave testing program were as follows:

- Product CIL gold recoveries of 96.6% can readily be achieved and recoveries of 97% are possible under optimum operating conditions, as indicated by the tests on the discharge samples as well as the autoclave profile samples.
- CIL gold recovery from the pilot autoclave is sensitive to the autoclave residence time. Gold recovery is slightly lower than optimum when autoclave residence time is too short because of incomplete sulfide sulfur oxidation. Recovery can also be lower than optimum if autoclave residence time is too long and oxidation is excessive.
- Autoclave operating temperatures of 220–225°C provided good results with optimum residence times of 45-49 minutes for CIL gold recovery (exceeding 97%), based on the autoclave discharge samples.
- Measurable sulfide sulfur oxidation is essentially completed by 37-42 minutes residence time, as indicated by analysis of the autoclave profile samples.
- Organic carbon content (by assay) decreased from 0.78% to an average of 0.57%, with approximately 25% of the organic carbon being oxidized in the autoclave.
- The selected hot curing time of six hours as per the feasibility design provides good lime consumption results and CIL gold recovery performance, but dissolution of arsenic is evident, subsequently requiring precipitation in the following neutralization stage.

Autoclave Mercury Emission Testwork

During a pilot run, two gas streams were sampled for mercury content in the flash pot vent stream and the autoclave vent stream.

The amounts of mercury vented through the flash system and the autoclave vent were 0.21% and 0.026%, respectively, of the calculated mercury head in the concentrate.

The results from a second series of emissions testing indicated the presence of very little mercury emission in the combined gas streams (0.003% of feed mercury content), with the gas and scrubber mercury contents being close to assay detection limit.

Despite the low measurements, a mercury abatement system has been designed to comply with the December 2010 US EPA National Emissions Standard for Hazardous Air Pollutants for gold ore processing and production facilities. Future studies should consider any updates to legislation.

13.2.5 Neutralization

13.2.5.1 Summary

The oxidation of pyrite and other naturally occurring sulfides generates sulfuric acid and other metal sulfates within the autoclave and hot curing circuits. These species need to be neutralized and precipitated into a stable form to ensure that the final tails from the plant have a low soluble metals content and is also at approximately neutral pH.

Typically, limestone (calcium carbonate) and hydrated lime (calcium hydroxide) are utilized for this neutralization duty. Limestone can be used for the first part of the pH adjustment while the final pH adjustment to 7 (for acidic liquor neutralization) and 9-11 (for CIL feed) is carried out with hydrated lime as $Ca(OH)_2$, which is a more reactive neutralizing reagent.

Metallurgical studies were undertaken to determine the potential neutralization capacity of the flotation tails stream and also of calcareous sandstone (CSS), a local natural source of low grade carbonates. This local material is composed of ferroan dolomite, ankerite, calcite and siderite in decreasing levels.

The deposit has a relatively high carbonate content of 2.33% reported as CO_2 (or 3.18% reported as CO_3) over the LOM compared to a sulfur content of 1.13%. This represents a stoichiometric ratio of carbonate to sulfur of 1:1.49. Therefore, there is sufficient, or rather, excess alkali in the ore available to neutralize the sulfates generated from the oxidation of all the contained sulfur. However, this requires effective utilization (reactivity) of the measured neutralization content of the ore for the excess to be valid.

Testwork has shown that with the provision of sufficient neutralization residence time, and the elevation of the slurry temperature in neutralization, utilization of the contained carbonates in the ore is possible, resulting in a minimal amount of lime required.

Further, it has been determined that the use of CSS is not required for neutralization of the acidic autoclave acid, as flotation tails have been found to be more effective. CSS has a high pH (>5.0), and does not compete economically with imported lime. The local CSS resource can serve as a back-up alkali source as required.

13.2.5.2 2004 Neutralization Testwork

Dynatec tested the neutralization properties of the acidic liquors generated from bench autoclave tests using flotation tails, limestone, and lime. The neutralization tests conducted with limestone and lime performed well. At a pH of 8.0 all metals, with the exception of Mn and Mg, precipitated virtually completely from solution at generally below detection limit grade. The results were not favorable from a lime consumption cost perspective.

13.2.5.3 2005 Neutralization Testwork

PDTS investigated the neutralization capacities of CSS and flotation tails. Results showed that CSS with high CaO/MgO ratios provided higher carbonate utilization compared to low ratio composites. The grind size did not have a large effect on the neutralization capacity of the CSS composites.

Testwork on flotation tailings showed higher carbonate usage from the intrusive materials compared to the sedimentary materials. Overall, the utilization of the carbonate within the tailings was very low.

13.2.5.4 2006 Preliminary Batch Neutralization Testing

A limited batch neutralization testwork program was initiated in mid-2006, aiming to improve quantification of the neutralization options for the Project. The results indicated that the flotation tails/lime neutralization option, with extended neutralization residence time, was the most economic with the lowest total cost.

13.2.5.5 2006 Pilot Neutralization

A pilot neutralization testwork program was undertaken using acid solution produced during the pressure oxidation pilot run conducted with the autoclave operating at 220°C followed by hot curing of the slurry for at least 12 hours at 95°C.

Flotation tailings produced from G&T were blended in the ratio of 25% ACMA Intrusive, 50% Lewis Intrusive and 25% Sedimentary.

Profile samples were routinely collected from the pilot-plant run with lime added to achieve a final pH of 7.0. At the given ratio of flotation tailings to concentrate (5.135 kg tailings per kg concentrate), neutralization of the dilute acidic pressure oxidation solution using flotation tailings with a carbonate grade of about 1.8% CO_2 reached a pH of 4.0-4.5 after 12 hours. There was no significant increase in pH at longer retention times. Under the conditions tested, lime consumption after flotation tailings neutralization was approximately 4.5 g quicklime per liter of dilute acid solution, or 24 g of quicklime per kilogram of concentrate. It was also possible to use CSS at the rate of 1.6 kg of CSS per kg of concentrate.

13.2.5.6 2007 Neutralization Phase 1 Bench Testing

The acidic solution used for this testwork was generated from concentrate during the continuous pressure oxidation pilot run in 2007. The concentrate was pre-acidified to a pH of about 2 with sulfuric acid prior to the oxidation process. The campaign run was at 225°C with a 70-minute retention time. The discharge slurry was then hot cured for about 24 hours immediately following pressure oxidation.

The hot cure slurry was washed with gypsum saturated water in a pilot counter-current decant (CCD) circuit at a ratio of 2:1. The overflow acidic filtrate was collected and used for batch and continuous tests. Initially, the pH of the diluted pressure oxidation solution was approximately 1.0.

The flotation tailings used was a blend from a flotation piloting campaign in December 2006. The flotation tails used had a carbonate grade of 2.0% CO_3 and was sourced from the same pilot float feed sample that provided the concentrate used to generate the acidic liquor, via the pilot autoclave.

Bench tests at varying temperatures were undertaken to determine the effect of temperature on neutralization rate and utilization. Significant improvement in performance occurs as temperature increases. Temperatures of 55°C or greater allowed a pH of 6.0-6.5 to be reached, consequently resulting in the reduction of lime consumption to about 1-2 g/L of diluted acidic liquor.

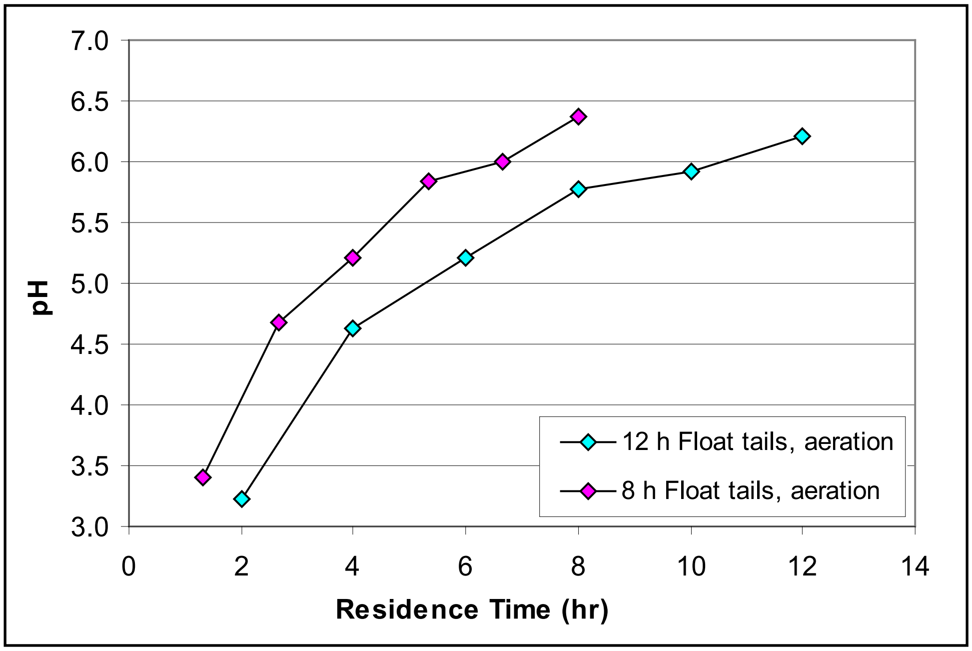
13.2.5.7 2007 Neutralization Phase 1 Pilot Testing

Based on the bench tests, a pilot neutralization campaign was initiated, using 70°C and two residence time selections of 8 and 12 hours, and the same acidic liquor and flotation tails as used for the bench testing.

The average pH profiles of both residence time test campaigns are shown in Figure 13-7. Final pH levels of greater than 6.0 were reached with the flotation tails for both runs.

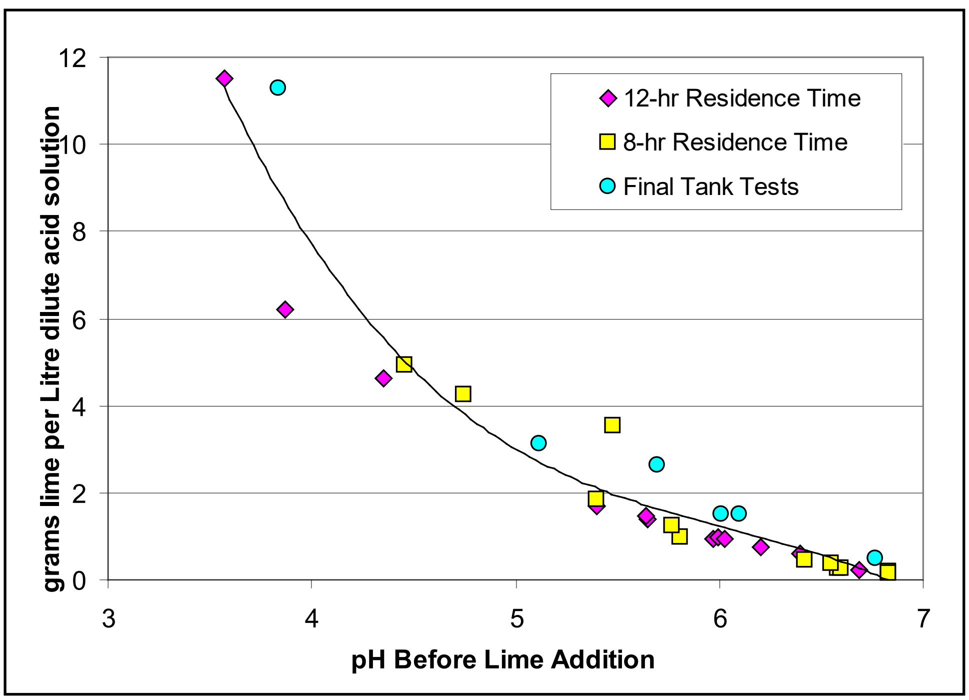
Figure 13-8 shows that lime consumption tests of profile samples taken from the pilot-plant runs required lime addition of less than 1 g/L of diluted acidic liquor, at the end of the neutralization circuit.

Figure 13-7: 2007 Phase 1 Neutralization Pilot pH Profiles



Source: AMEC, 2011

Figure 13-8: Phase 1 Neutralization Pilot Samples Lime Demand Test Results



Source: AMEC, 2011

13.2.5.8 2007 Neutralization Phase 2 Pilot Testing

During Phase 2 of the pilot autoclave test program the neutralization performance of the MCF2 pilot flotation tails and the FS design of the neutralization pilot circuit were investigated with the following key parameters:

- Four float tails neutralization tanks
- One lime neutralization tank
- Six hours residence time (based on testwork)
- Operating temperature of 55°C (based on heat balance)
- Addition of CIL tails into tank one
- Mixing ratios of diluted acidic liquor and flotation as per the feasibility mass balance.

In addition to the six-hour residence time run, a second three-hour residence time campaign was undertaken to investigate the potential to reduce the size of the circuit for the detailed design phase.

The source of the dilute acidic liquor was the overflow stream from the operation of the pilot CCD plant, washing of hot cured product. To be conservative, no calcium carbonate was added to this acidic liquor to correct for the pre-acidification process.

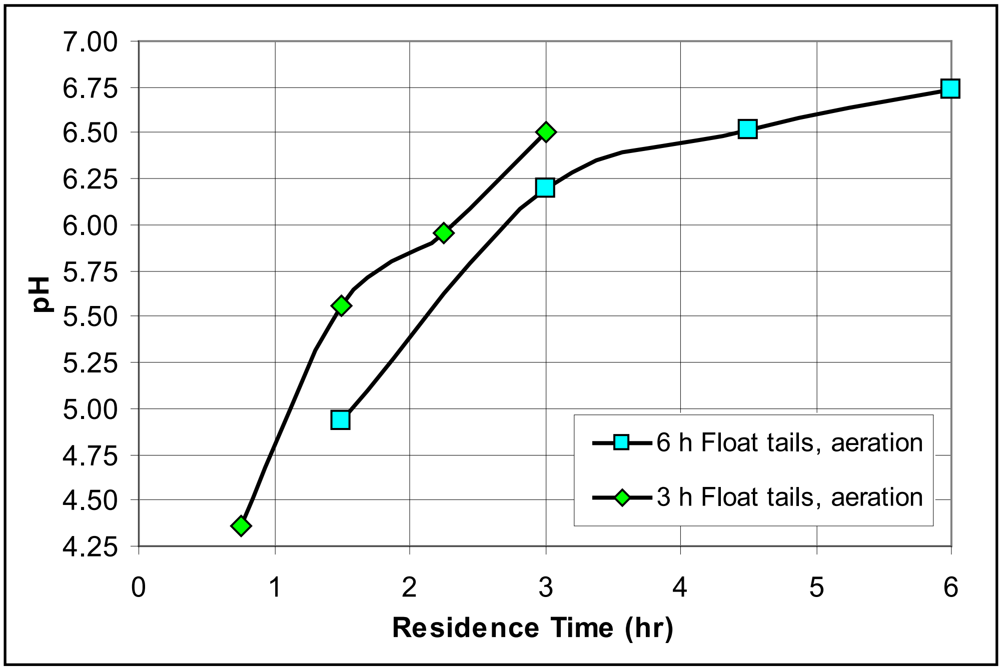
The source of the flotation tails was the 2007 SGS Lakefield flotation pilot, which incorporated the MCF2 grinding/flotation flowsheet. The sample used for the pilot test had a carbonate grade of 1.65% (as CO_3^{2+}), compared to the LOM predicted carbonate grade of 2.33% (as CO_3^{2+}).

The average pH profiles for the two test campaigns are shown in Figure 13-9. The six-hour test campaign (i.e., matching the feasibility circuit design) reached a final pH of 6.75 prior to the lime addition step. The three-hours residence time run resulted in a final pH of 6.50.

Figure 13-10 shows the results of the lime demand tests undertaken on profiles from the pilot plant. Lime consumption at the end of the neutralization circuit is less than 0.2 g/L of diluted autoclave acid solution, with wash water flows increased in the feasibility design (more dilution of the acidic liquor) to improve washing efficiency of the autoclave discharge hot cure product.

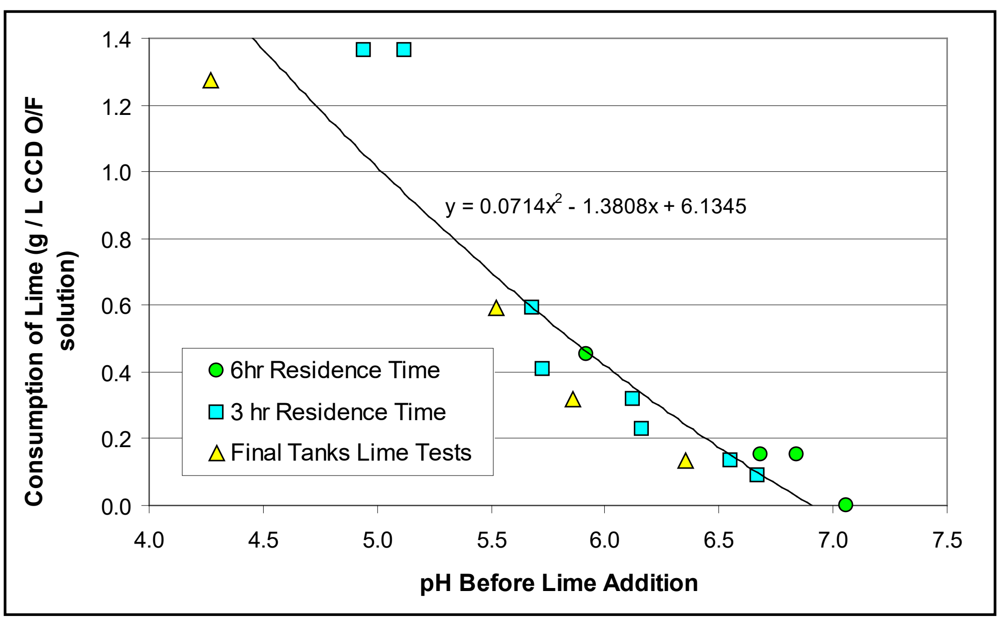
The results of the three-hour test campaign are encouraging, suggesting the potential to decrease the size of the neutralization circuit further. However, for the purposes of managing the potential variability of carbonate content in the ore, and also to provide time for the operations personnel to respond to unplanned grinding or flotation circuit shutdowns, the longer six-hour residence time circuit continues to be the recommended design.

Figure 13-9: 2007 Phase 2 Neutralization Pilot pH Profiles



Source: AMEC, 2011

Figure 13-10: Lime Demand Test Results of 2007 Phase 2 Pilot Samples, Plotted against Initial pH



Source: AMEC, 2011

13.2.5.9 2007 Neutralization Variability Testing

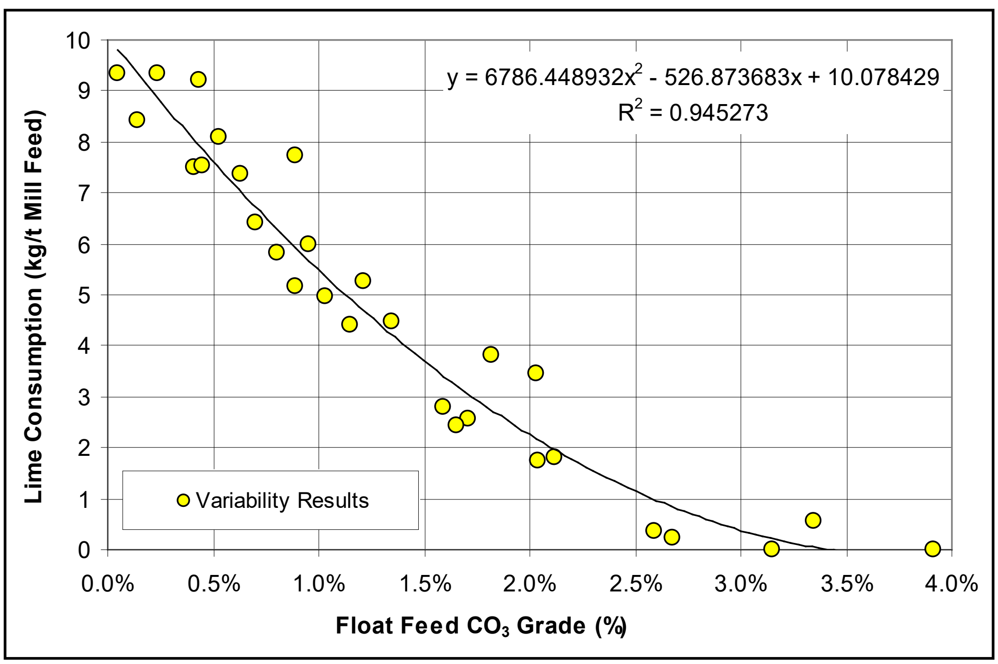
A variability neutralization testwork program was initiated at SGS Lakefield. The program had the dual aim of confirming the potential differences in neutralization performance of the varying lithologies and of developing a confident relationship between lime consumption (for final pH trim to 7) and the feed samples carbonate grade.

SGS Lakefield generated a synthetic dilute autoclave acid based on the prediction of key species content from the near-final MetSim model's prediction of that stream. Flotation tails samples from the recently completed flotation variability program were used as the source of neutralizing solids.

Figure 13-11 is a summary chart of the results of the tests completed.

Prediction of lime consumption for acidic liquor neutralization for the operating cost estimate is based upon the relationship between lime demand and flotation feed carbonate grade developed from this test data.

Figure 13-11: Plot of Neutralization Variability Testing Lime Demand Results at Six Hour Residence Time



Source: AMEC, 2011

13.2.6 Carbon-in-Leach

13.2.6.1 Summary

Extensive cyanidation testing has been undertaken on samples at various points in the flowsheet since 1995.

Cyanidation of unoxidized ore, with or without the presence of activated carbon, consistently yields very low gold recoveries of 5-30%, either as flotation feed, flotation tails, or concentrate. This is characteristic of ore where gold is predominantly associated with arsenopyrite or pyrite in solid solution form.

The bulk of the cyanidation tests carried out to date have largely been on autoclave compartmental and discharge samples, where large numbers of relatively small samples are leached with high concentrations of carbon and cyanide. This is a diagnostic tool that enables the performance of various autoclave tests to be established without the added complication of the constraints that could be imposed by attempting to optimize leaching kinetics.

CIL gold recovery has generally shown to be more sensitive to the autoclave operating conditions (residence time, temperature) than to the operating conditions and methods applied in the CIL circuit. The target of the metallurgical design of the CIL circuit is rather to ensure that good CIL recovery performance is achieved on the material presented to it from the autoclave, with optimum reagent (lime and cyanide) usage.

13.2.6.2 Key Metallurgical Aspects of Planned Donlin CIL Circuit

The aim of the testwork programs was to define the operating characteristics and circuit design of the CIL circuit for treatment of CCD-washed autoclave product.

Leach Circuit pH

The MetSim modeling and metallurgical testing have shown that the CIL circuit could operate well at a relatively low pH of 9.

Increasing CIL pH above 9 in the CIL circuit will consume additional lime through the precipitation of magnesium hydroxide from magnesium sulfate in solution. The magnesium hydroxide are then dissolves when combined with tailings and returns to the plant through tailings water recycle. To achieve the conventional CIL circuit pH levels of 10-11, all of the magnesium in the feed solution would need to precipitate completely.

Assuming a CIL pH of 9, lime addition is estimated to be in the order of 5-7 kg/t of concentrate. The key component affecting both is the washing efficiency of the autoclave product CCD wash circuit. To maximize washing efficiency, a four stage CCD circuit with a high wash ratio of 4:1 is used.

13.2.6.3 2006 CIL of Flotation Tails

BTC undertook CIL tests on the flotation tails from the G&T pilot-plant campaigns. Results showed low gold recovery, concluding that leach of flotation tails is not economically viable.

13.2.6.4 2006 CIL Optimization Testwork

A composite of autoclave discharge material from the 2006 autoclave pilot test program was collected and leached under varying conditions. The gold recoveries achieved from this work were limited due to the nature of the product from the particular pilot run. The following conclusions are drawn from the CIL optimization work undertaken on the pilot autoclave product:

- Cyanidation in the absence of carbon is detrimental to final leach gold recovery.
- Using higher carbon loadings (pre-loaded with gold) does not adversely affect gold recovery.
- Leaching at pH 9.2 does not negatively affect CIL gold recovery.
- Increasing the wash efficiency of the CIL feed slurry can significantly reduce reagents consumption rates.
- Leaching at too high a slurry density can negatively affect CIL gold recovery through unsuitable rheological properties of the slurry.

Samples of the detoxified CIL products were sent to the University of British Columbia for rheological testing, where it was seen that the slurry solids content (% solids) had a significant impact on the viscosity of the slurry. Densities above 35% are considered potentially problematic.

13.2.6.5 2007 CIL Pilot-Plant Testing

BTC carried out a pilot CIL test run using CCD-washed pilot autoclave product from the 2007 Phase 1 autoclave pilot testwork program.

Gold and Silver Recoveries

The carbon in the pilot CIL circuit was successfully loaded up to 4,000 g/t gold, and two carbon transfers were undertaken.

A gold balance for the period of the pilot-plant run returned a calculated gold recovery of 93.6% with a tailings grade of 1.39 g/t.

A bottle roll test of the CIL feed conducted at pH 11 yielded a comparative gold recovery of 93.9%, indicating that the pilot operation provides equivalent recovery performance to that achieved in a batch bottle roll test, even with the pilot plant operating at a relatively low pH of 9 and using profile-loaded carbon.

Silver recovery from the pilot plant was low, at 29.7%, which is typical of the leaching characteristics of silver from autoclave solids product due to its dissolution and subsequent precipitation as cyanide insoluble silver-jarosite within the autoclave.

Leaching residence time of 20-24 hours over six tanks is an appropriate design for the continuous CIL circuit.

Cyanide Consumption and Addition

The CIL feed is relatively free of cyanide consumers. This is characteristic of concentrate that has been subject to pressure oxidation, where sulfur is oxidized completely to sulfate and base metals are dissolved into solution. This is followed by CCD washing, which removes the dissolved metals from the autoclave product ahead of CIL.

Cyanide addition to the pilot circuit was 1.5–1.6 kg/t, with a consumption of 1.1–1.3 kg/t. Most of the consumption is likely to be from losses through hydrogen cyanide from the pilot CIL circuit, rather than consumption by species within the ore. Hydrogen cyanide losses of this magnitude will not be experienced at full scale, and the hydrogen cyanide that evolves will be recovered via the ventilation and scrubbing system and returned to the CIL circuit feed. Cyanide addition to the pH 11 bottle roll tests on the pilot-plant feed was 1.2 kg/t, with a consumption of 0.05 kg/t.

Assuming a CIL pH of 9, a cyanide addition of 0.7–0.9 kg/t of concentrate is estimated. Consumption of cyanide will be lower at a higher CIL pH of 11.

13.2.6.6 2007 CIL Optimization Testwork

A series of bench scale tests were undertaken on products from the 2007 Phase 2 pilot autoclave test program with focus on optimizing the CIL process.

Cyanide Addition Optimization

A series of 24-hour bottle roll CIL tests were conducted at varying cyanide concentrations to determine the relationship between cyanide addition rate and gold recovery. Due to the limitations associated with undertaking low pH CIL tests at laboratory scale, a higher pH of 11 was used. Results showed that CIL gold recovery was not significantly affected at low cyanide concentration, and that the process is more economically favorable at low CIL cyanide concentration levels.

Rheology

Additional rheology testing was undertaken on a detoxified CIL product from the 2007 Phase 2 pilot-plant work. Beyond a level of 35% solids the viscosity of the material was found to climb rapidly.

13.2.6.7 Cyanide Detoxification Testwork

Cyanide detoxification testwork was completed on CIL tails slurry generated from the 2006 pilot autoclave test program. Three types of cyanide detoxification methods were tested and found effective:

- *Prussian blue (iron sulfate precipitation)* – consists of adding autoclave discharge acid to detoxify the cyanide complexes
- *AVR (Acidification, Volatilization, Recycle)* – consists of acid addition to drive the cyanide off as hydrogen cyanide and capturing the hydrogen cyanide for reuse in the circuit
- *SO_2/air testing* – uses a combination of SO_2 and air to detoxify the cyanide.

The SO_2/air method was selected. The CIL tailings slurry was effectively treated in a single stage operating with approximately 60 minutes of retention and an SO_2 dosage of 4 g/g CNWAD. A pH of 8.5 was used for all tests as acid addition was required to lower pH levels. The addition of copper sulfate at 10 mg/L Cu^{2+} was required for effective removal of cyanide present in the feed.

The content of arsenic in the liquor phase was found to increase after SO_2/air cyanide detoxification. This solubilized arsenic will be re-precipitated upon mixing the CIL tails into the neutralization circuit as a result of the presence of high levels of dissolved iron in this circuit.

13.2.7 Thickening and Counter-Current Decantation Washing

The feasibility study flowsheet includes the following thickening/solids settling operations:

- Concentrate thickening after flotation
- CCD washing of pre-acidified concentrate with fresh water to provide optimal oxidation conditions
- CCD washing of hot cured autoclave product slurry with process water to reduce lime consumption ahead of CIL cyanide leaching
- Clarification of the portion of hot cure CCD overflow not reporting to pre-acidification or flotation conditioning to recover entrained gold values
- Thickening of flotation tailings prior to neutralization to minimize dilution during neutralization and reclaim of process water.

Earlier flowsheets included a final tailings thickener to dewater the combined CIL tailings and neutralization residue prior to discharge to the TSF. This thickener was removed from the feasibility study flowsheet.

The flotation tailings, which are a combination of the secondary rougher and the cleaner scavenger tailings, are de-watered before being directed to the pressure oxidation to provide cooling.

13.2.8 Tailings Neutralization Capacity

To provide samples that are reasonably representative of both the complete metallurgical processes, and also the ore, the testing of combined pilot-plant tailings was selected as the preferred testing method.

The final tailings consist of a blend of detoxified CIL tails (cyanide leached autoclave and hot cure product) and neutralized autoclave acidic liquor using the flotation tails stream.

The Project flowsheet is favorable for tailings that are not acid producing as a result of near complete sulfide sulfur oxidation.

Mineralogy undertaken by SGS Lakefield indicates that up to 23% of the sulfate sulfur in the 2006 pilot final tails sample is in the form of jarosite, with 7% in the 2007 Phase 1 pilot plant final tails and 8% in the 2007 Phase 2 pilot-plant final tails. Modifying the calculated ABA parameters, assuming that jarosite is an acid-forming component of the sulfate, indicates that the tailings will still contain an excess of neutralization capacity.

13.3 Recovery Estimates

There are two components to defining the final recovery of gold to bullion including:

- Gold recovered from the flotation circuit to the flotation concentrate
- Gold recovered through leaching/adsorption (CIL) of the pressure oxidized flotation concentrate.

Due to the refractory nature of the ore and the relatively low grade of the flotation tails stream, it is not economically viable to recover gold from the flotation tails stream.

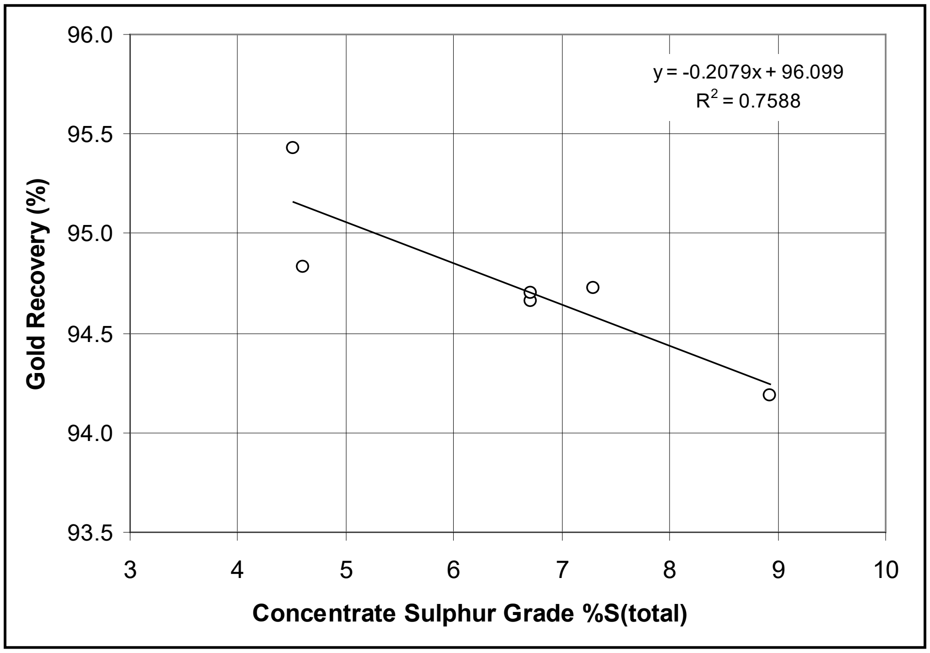
13.3.1 Flotation

The MCF2 flowsheet is assumed to be the basis of the flotation circuit design and for all recovery figures referenced in this section, both as bench testing and pilot testing.

Flotation pilot and variability testing was conducted on fresh ore while bench scale variability flotation testwork was also undertaken on oxidized (or partially) samples. While the extent of the geological oxidation is relatively low, there is still potential for significant effects on flotation recovery. The overall deposit flotation recovery was corrected to account for the small oxidized component of the deposit.

Gold recovery from the MCF2 pilot program is achieved by means of fitting a linear regression line through all the MCF2 pilot survey results (Figure 13-12). These gold recovery survey calculations incorporate both the primary rougher concentrate and the secondary rougher cleaner concentrate. Cleaner scavenger concentrate was recirculated to the feed of the secondary rougher. At the design concentrate target of 7% (total) sulfur, the gold recovery of the blended composite samples tested is 94.64%. This recovery forms the basis of the flotation gold recovery estimate and must be adjusted to account for effects of geological domain and alteration (oxidation extent).

Figure 13-12: MCF2 Pilot-Plant Campaign Survey Results



Source: AMEC, 2011

Variability Testing – Unoxidized Ores

Recoveries determined during the variability testwork were grouped by geological domain and sedimentary rocks as summarized in Table 13-9. The gold and sulfur head grades of the MCF2 pilot-plant composite are very close to the orebody average.

Table 13-9: Flotation Recovery from Variability Testwork Program

Geological Domain	Average Flotation Recovery (%)
Akivik	97.61
400	96.97
ACMA	96.45
Aurora	96.22
Vortex	95.19
Lewis	94.87
GWK	91.45
SHL	89.99
OXIDE	81.45

Variability Testing – Oxidized Ores

There is a large variation in flotation test results (i.e., gold recoveries to target concentrate grades) of the ores affected by oxidation. As a result, the relationship between sulfur grade and flotation recovery was used as the basis of recovery estimation.

An oxidation wireframe was used to flag the block model allowing the allocation of tonnes and grade of the oxidized ore in the production plan.

Oxidized ore will be stockpiled and subsequently reclaimed for mill feed during the course of the mine life. A sulfur degradation and flotation recovery factor of -5% was applied to reclaimed material that was stockpiled for longer than one year.

Final Flotation Recovery Model Definition

As the pilot-plant sample was originally composited on the basis of lithological domain, rather than geological domain, the recoveries have been adjusted slightly to account for the variation in content between the pilot-plant sample and the latest estimate for the orebody. Adjusting the MCF2 pilot-plant recovery based on the geological domain, variability flotation performance results in a minor upward adjustment of 0.16% for the unoxidized ore.

The proportionally adjusted flotation recoveries by geological domain are summarized in Table 13-10. Wood recommends that these recovery values be used within the mine plan where the geological domains are separately defined on a period-by-period basis.

No clear relationships between gold, arsenic, or sulfur head grades in flotation recovery were identified in the variability testwork.

Table 13-10: Summary of Flotation Recovery in Variability Testwork Program by Geological Domain and Adjusted to MCF2 Pilot Result

Geological Domain	Adjusted Recovery to MCF2 Pilot Result (%)
Akivik	97.77
400	97.13
ACMA	96.61
Aurora	96.38
Vortex	95.34
Lewis	95.03
GWK	91.61
SHL	89.99
OXIDE	81.45

13.3.2 Pressure Oxidation

Under the continuous pilot plant testwork and based on the proposed plant design, an overall gold recovery of 96.6% of concentrate for all ore types can be achieved through the pressure oxidation/CIL circuits on a continuous and long term basis. This considers sedimentary rock hosted ore as containing deleterious components that will be a consistent contributor to the blended ore received at the mill.

13.3.3 Overall Plant Gold Recovery

To determine the overall plant recovery, both pressure oxidation and flotation need to be considered together. The LOM plant recovery averages 90.0% based on mill feed from the mine plan.

13.4 Deleterious Elements

Arsenic is the predominant deleterious element, present as the arsenopyrite which is the major gold hosting mineral, as well as native arsenic and realgar in minor to trace amounts. Pressure oxidation of arsenopyrite in the presence of excess iron is generally considered a best practice process to generate stable scorodite ($FeAsO_4.2H_2O$) for tailings disposal. Promoting the formation of stable precipitates is favored when the molecular ratio of iron to arsenic ratio in process solutions exceeds 4:1. Within the plant feed there is sufficient iron to provide the recommended molar ratio of 4:1 of iron to arsenic. The actual assay grade of iron is typically double the iron content that is accounted for by arsenopyrite and pyrite alone and is more typically at grades of 15,000–40,000 ppm.

Mercury is present in the sulfide mineralization as a mercury sulfide. The mercury is released from the sulfide mineralization through pressure oxidation. The cyanide in the CIL circuit dissolves a portion of the mercury in the solids feed to the circuit. A portion of this dissolved mercury in the CIL circuit is adsorbed onto the activated carbon and is then recovered from the carbon via stripping and carbon regeneration. However, the capacity of the carbon to completely adsorb the mercury is limited and therefore a component of the soluble mercury remains in the CIL tails solution. This remaining soluble mercury will then be blended with the detoxified CIL tails into the neutralization circuit, which then reports to the TSF.

Reductions in soluble mercury content in recirculating plant waters can be achieved by addition of mercury precipitation reagents, which convert soluble mercury to a stable mercury sulfide product. This is currently practiced using the Cherokee Chemical UNR reagent suite at operating mine sites in the US.

Based on the testwork completed, it is recommended that the process plant design includes a dosage facility for Cherokee reagent UNR 829 to permit addition to a recirculating water stream for precipitation of mercury in solution into a stable mercury sulfide solid. Doing so will eliminate potential build-up of mercury in the process water circuit.

Based on the process design which includes mercury removal and abatement measures, the presence of the deleterious elements will not significantly affect potential economic extraction.

14.0 MINERAL RESOURCE ESTIMATES

14.1 Summary

This section describes the updated Mineral Resource estimate for the Project. QP Kim reviewed and validated the geologic model and mineral resource model prepared by NOVAGOLD and based on that review prepared a Mineral Resource statement.

14.2 Database

The Mineral Resource model is based on 1,737 core holes (456,450 m) and 387 RC holes (37,457 m) drilled between 1995 and 2022. Trenches, auger holes, and water wells were not used for estimation. A total of 47 holes (18,056 m) were completed in 2025 within the resource area; however, they were not used in the development of the geologic or Mineral Resource model. All 2025 drilling results were checked against the geologic and Mineral Resource model with good correlation.

14.3 Geologic Models

Donlin Gold LLC generated three-dimensional (3D) solids for the geological model using tools available in Leapfrog Geo, a commercial mine design software. The data used to construct the models include core holes, trenches, RC holes and field mapping conducted up to 2023. The 2024 drilling campaigns were not within the block model limits. The following 3D solids were used to incorporate geologic control into the grade model for the intrusive rocks:

- The five main intrusive types of RDA, RDF, RDX, RDXB, and RDXL
- Undefined intrusive rhyodacite (Isolated Intrusive)
- Mafic dike (MD)
- Sediment rocks (Mixed sediments – combined greywacke (GWK), shale (SHL), siltstone (SLT), and other sediments)
- Overburden.

A separate wireframe of oxide material was created based on logged oxidation in the drilling. The wireframe was used to flag the block model and used in final recovery calculations.

The Donlin deposits have been divided into nine geological domains as described in Table 14-1.

Table 14-1: Donlin Deposits Geological Domains

Domain	Domain Name	Description
1	ACMA	Dominated by a wide and compact sill sequence within a synclinal shale unit. Domain 1 is limited to the north by the Lo Fault and to the west by the ACMA Fault. The eastern and southern margins correspond to the edge of the Lower Vortex domain.
2	Lower Vortex	Dominated by the lower extension of the RDA dikes and the far eastern extension of the ACMA RDX and RDA sills. It is limited to the north by the Lo Fault. Arbitrary lateral boundaries were digitized on either side of the dike system to separate it from the Lewis domain on the south and east, and from the ACMA domain on the west.
3	Lewis	Dominated by a series of wide dikes that intersected the shale sequence and produced wide masses of sills. Domain 3 is limited to the west by the Lower Vortex and Vortex domains.
4	Vortex	Corresponds to the upper portion of the Vortex RDA dikes. It is limited to the south by the Lo Fault. Arbitrary lateral boundaries were digitized on either side of the dike system to separate it from the Akivik and Lewis domains.
5	Akivik	The upper faulted extension of the ACMA sill sequence and includes the RDXL dike. Domain 5 is limited to the south by the Lo Fault and to the west by the ACMA Fault. The eastern margin corresponds to the edge of the Vortex domain.
6	Tortured Block	Dominated by the ACMA sills that have been faulted into an indistinguishable mass of intrusive. It is limited to the north by the extension of the Lo Fault beyond the ACMA Fault (currently named the Hello Fault), to the west by the American Creek Fault, to the south by the Upper-Lo Fault, and to the east by the ACMA Fault.
7	Wedge Block	Dominated by the eastern extension of the ACMA domain RDA and RDXB sills. It is immediately south of the Tortured block below the Upper-Lo Fault. It is bound to the west by the American Creek Fault and to the east by the ACMA Fault.
8	400	Characterized by a series of sub-parallel sills that are the western extension of the ACMA/Tortured domains. Domain 8 is located above the Lo Fault between the ACMA Fault and American Creek Fault.
9	Aurora	Characterized by a series of sub-parallel sills that are the western extension of the ACMA/Tortured domains. Domain 9 is located below the Lo Fault and west of the ACMA and American Creek faults.

14.4 Exploratory Data Analysis

The assay database contains over 270,000 samples of generally 2 m in length. The geological model wireframes were used to define exploratory data analysis (EDA) envelope.

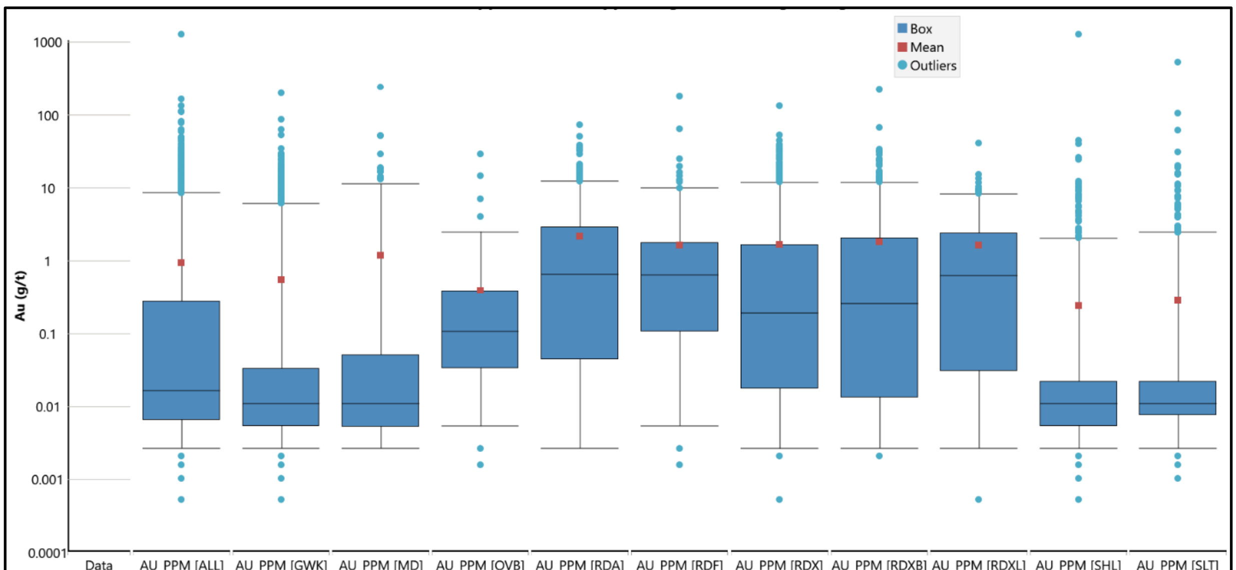
The box plot of raw gold assays shows that the average grade within the intrusive units is higher than the sedimentary units (Figure 14-1). Within the sedimentary units, the greywacke unit shows a higher average gold grade than the shale or silt units.

Gold estimation domains are defined by two major geological units defined by the geologic wireframes:

- Intrusive units including the mafic dike (RDX, RDXB, RDA, RDXL, RDF, MD)
- Mixed sediments (GWK, SHL, SLT).

Each gold estimation unit is further constrained by a grade shell generated by a gold probability model using a gold grade threshold of 0.25 g/t.

Figure 14-1: Box Plot Raw Gold Assay by Rock Type



Number of samples	259,934	104,793	8,120	1,527	18,977	4,135	47,378	16,244	6,956	25,994	24,638
Max	963.00	152.90	186.52	22.90	56.75	137.00	101.73	170.10	32.53	963.00	405.00
97.5% quantile	6.94	5.00	9.16	2.05	9.94	8.02	9.60	9.60	6.68	1.68	2.04
Upper quartile	0.24	0.03	0.05	0.33	2.45	1.50	1.41	1.72	2.00	0.02	0.02
Median	0.01	0.01	0.01	0.09	0.56	0.55	0.16	0.22	0.53	0.01	0.01
Lower quartile	0.01	0.00	0.00	0.03	0.04	0.09	0.02	0.01	0.03	0.00	0.01
2.5% quantile	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00
Min	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00
Mean	0.79	0.47	1.01	0.34	1.82	1.37	1.40	1.52	1.37	0.21	0.24
CV	3.74	5.34	6.40	3.23	1.69	2.19	2.33	2.25	1.55	12.38	9.79

Source: Wood 2025

14.5 Density Assignment

Specific gravity (SG) data were gathered from the 1996–1997 and 2006–2019 drilling programs. As of December 31, 2024, a total of 15,780 SG measurements had been collected. After removing samples with obvious outliers, uncommon rock types, mafic dike and overburden samples, and samples outside the resource area, 13,195 samples remained. Table 14-2 summarizes the average SG values by rock type grouped by geologic domain.

A total 1,045 SG samples were measured for the mafic dike with an average SG of 2.73; however, a more conservative value of 2.65 was used to account for erratic units that were observed.

Blocks that were flagged as overburden in the block model were assigned an SG value of 2.14.

Table 14-2: Specific Gravity by Rock Type

Rock Type/Domain	No. of Samples	SG
Intrusive Rocks		
RDA	1,111	2.64
RDF	457	2.65
RDX	2,059	2.66
RDXB	961	2.64
RDXL	384	2.64
Total/Average	**4,972**	**2.65**
Mixed Sediments		
Greywacke	5,292	2.71
Siltstone	1,429	2.72
Shale	1,217	2.71
Argillite	285	2.75
Total/Average	**8,223**	**2.71**

14.6 Grade Capping/Outlier Restrictions

Raw assays in the database were examined for the presence of local high-grade outliers, and overall grade distributions were used to establish capping values. The raw assay data were grouped by rock type, and capping values for gold were determined for each major rock type (Table 14-3). Individual frequency distribution plots were generated to determine the appropriate grade cap for each rock type. Capping was applied to all raw gold assays prior to compositing. Total sulfur, arsenic, mercury, and antimony assays were not capped.

Table 14-3: Capping Analysis for Gold

Rock Type	Capped Grade (g/t)	No. of Capped Samples	Uncapped		Capped		Metal Loss (%)
			Mean (g/t)	CV	Mean (g/t)	CV	
Greywacke	25	299	0.47	5.33	0.45	4.39	5.31
Shale/argillite	30	19	0.21	11.84	0.19	6.04	8.32
Siltstone	20	59	0.24	9.75	0.21	5.17	13.35
Mafic dike	30	58	1.01	6.40	0.78	3.30	22.54
RDA	20	76	1.82	1.69	1.80	1.57	1.39
RDF	16	36	1.37	2.19	1.32	1.65	4.19
RDX	30	74	1.40	2.33	1.38	2.12	1.36
RDXB	28	46	1.52	2.25	1.49	1.97	1.75
RDXL	10	69	1.37	1.55	1.33	1.37	2.75

14.7 Composites

Assay intervals were composited to a length of 6 m for gold, sulfur, arsenic, antimony, mercury, magnesium, calcium, carbonate and neutralization potential (NP). The composites were not broken at intrusive or sedimentary boundaries. Only core holes and RC holes were composited for estimation. Composites were back flagged with lithology using the geologic model (see Section 14.2).

14.8 Block Model Set Up

The block model parent size is 6 m (X) x 6 m (Y) x 6 m (Z). The block model was not rotated. The model extends beyond mineralization in the east, west, and south directions. Although mineralization is known to extend to the north at depth, the block model is terminated in the north and at depth to define a reasonable area for potential extraction. Block model definition parameters are summarized in Table 14-4.

Table 14-4: Block Model Set Up

Parmeter	Easting	Northing	Elevation
Minimum	539,000	6,877,000	-716
Offset	4,320	4,008	1,158
Block size	6	6	6
Number of blocks	720	668	193

14.9 Indicator Estimation

Separate indicator models were estimated for gold and sulfur using inverse distance squared (ID2) to identify areas of higher grade. An indicator value of 0 or 1 was assigned to the composites using a cut-off value of 0.25 g/t for gold and 0.50% for sulfur. Intrusive and sedimentary blocks were modeled separately. Composites flagged as intrusives were estimated for intrusive blocks and composites flagged as sediments were estimated for sediment blocks using two estimation passes and the estimation parameters summarized in Table 14-5. Search ellipsoids follow an orientation previously identified through structural orientations of veins in core holes and confirmed with directional gold correlograms. The only sample criteria that changes with each pass is the number of holes required to estimate blocks.

A probability threshold of 0.5 was selected to separate blocks that have a high confidence of containing grades greater than 0.25 g/t Au and 0.5% S, creating a mineralized envelope for grade estimation within the intrusives and sediments.

Table 14-5: Indicator Estimation Parameters

	Search Distance (m)			Search Rotation			Sample Selection		
Pass	**Major**	**Semi-Major**	**Minor**	**Bearing (Z)**	**Plunge (X)**	**Dip (Y)**	**Min**	**Max**	**Max/Hole**
1	175	175	100	24	-	-68	6	13	2
2	175	175	100	24	-	-68	4	13	2

14.10 Estimation Methods

Gold grades were estimated using inverse distance to the third power (ID3) methodology for blocks inside and outside the mineralized envelopes defined by the indicator model for the intrusives and the sediments. Interpolation was conducted with five passes with increasing search distances with each pass. The first pass used a "box search" with a search range equal to the dimensions of a single block. The range was increased for each successive pass, requiring at least two holes to estimate a block in passes 2 to 4. Search ellipse distances and sample weights were adjusted based on an anisotropic correlogram model developed for the indicator estimation. Pass 5 only required a single composite to be estimated with up to three from holes within close proximity to the block (Table 14-6).

Sulfur grades were estimated using the same method and parameters as for the gold grade estimation. Sulfur data are less extensive than gold data; therefore, sulfur was not estimated for a number of blocks during estimation runs due to a lack of support. Regression formulae were used to assign sulfur values to unestimated blocks based on the estimated gold grade. Where gold was not estimated, a value of 0.001 g/t Au was assumed for the calculation. The formulae used to assign unestimated sulfur blocks are summarized in Table 14-7.

Arsenic, mercury, and antimony grades were estimated using methods and parameters similar to those for the gold grade estimation. Data available for arsenic, mercury, and antimony are much less extensive than the data availability for gold and sulfur. Regression curves were derived from the relationship between gold and each of these elements for each of the major rock types. The regression formulae were then used to assign arsenic, mercury, and antimony values to non-estimated blocks based on the estimated gold grade. Where gold grade was not estimated, a value of 0.001 g/t Au was assumed for the calculation.

Table 14-6: Grade Estimation Parameters

	Search Distance (m)			Search Rotation			Sample Selection		
Pass	Major	Semi-Major	Minor	Bearing (Z)	Plunge (X)	Dip (Y)	Min	Max	Max/Hole
1	3	3	3	90	-	-	1	99	99
2	75	75	15	24	-	-68	2	3	1
3	75	75	30	24	-	-68	2	3	1
4	125	125	55	24	-	-68	2	3	1
5	30	30	10	24	-	-68	1	3	1

Table 14-7: Sulfur Regression Formulae

Rock Type	Regression Formula
RDA	S = 0.7382 x Au^0.2293
RDF	S = 1.0686 x Au^0.2568
RDX	S = 0.8616 x Au^0.3594
RDXB	S = 0.7571 x Au^0.3502
RDXL	S = 0.7744 x Au^0.2762
Isolated intrusives	S = 0.9738 x Au^0.3623
Mafic dike	S = 0.9489 x Au^0.3278
Mixed sediments	S = 0.9663 x Au^0.4015

14.11 Classification of Waste Rock Management Categories

Additional variables were included in the block model to aid with the geochemical classification of waste rock, based on ongoing waste rock characterization studies.

Acid potential (AP) was calculated from the estimated total sulfur concentration (S) where:

$$AP = 31.25 \times S\ (\%)$$

Neutralization potential (NP) from carbonate minerals (NP_{CO3}) was calculated from:

$$NP_{CO3} = 0.85 \times NP + 3.4$$

where by:

If NP ≤ 22.7 kg $CaCO_3$/t: $NP_{CO3} = NP$
If NP > 22.7 kg $CaCO_3$/t: $NP_{CO3} = 0.85 \times NP + 3.4$

The calculated variables NP_{CO3} and AP were used to calculate acid rock drainage (ARD) potential using the ratio:

$$ARD = NP_{CO3}/AP$$

Blocks were classified into seven waste rock management categories (Table 14-8) subdivided into potentially acid generating (PAG) and non-acid generating (NAG) groups, based on their ARD potential.

The estimated arsenic and sulfur values were used to calculate the ratio of arsenic to sulfur (As/S) for each block. During the process, arsenic leaching was recognized as being ubiquitous throughout the deposit.

Table 14-8: Waste Rock Management Categories

Category	PAG/NAG	Category Description	NP_{CO3}/AP Range
1	N/A	N/A	-
2	NAG	Very unlikely to generate ARD and potentially significant arsenic leaching	$NP_{CO3}/AP > 2$
3	N/A	N/A	-
4	NAG	Unlikely to generate ARD and potentially significant arsenic leaching	$1.3 < NP_{CO3}/AP \leq 2$
5	PAG	Very long delays (several decades) to the onset of ARD	$1.0 < NP_{CO3}/AP \leq 1.3$
6	PAG	During LOM (possibly less than a decade)	$0.2 < NP_{CO3}/AP \leq 1.0$
7	PAG	Shorter delays to the onset of ARD (less than a few years)	$NP_{CO3}/AP \leq 0.2$

14.12 Validation

A nearest neighbor (NN) model was generated during the estimation process and used for validation purposes.

The estimated block model gold grades were compared visually against drill holes and composites in section and plan view. Figure 14-2 and Figure 14-3 show section views and plan views of the estimated block grade for the Measured and Indicated blocks with the input 6 m gold composite grades. Overall, the estimated gold grades match well with the composite database gold values.

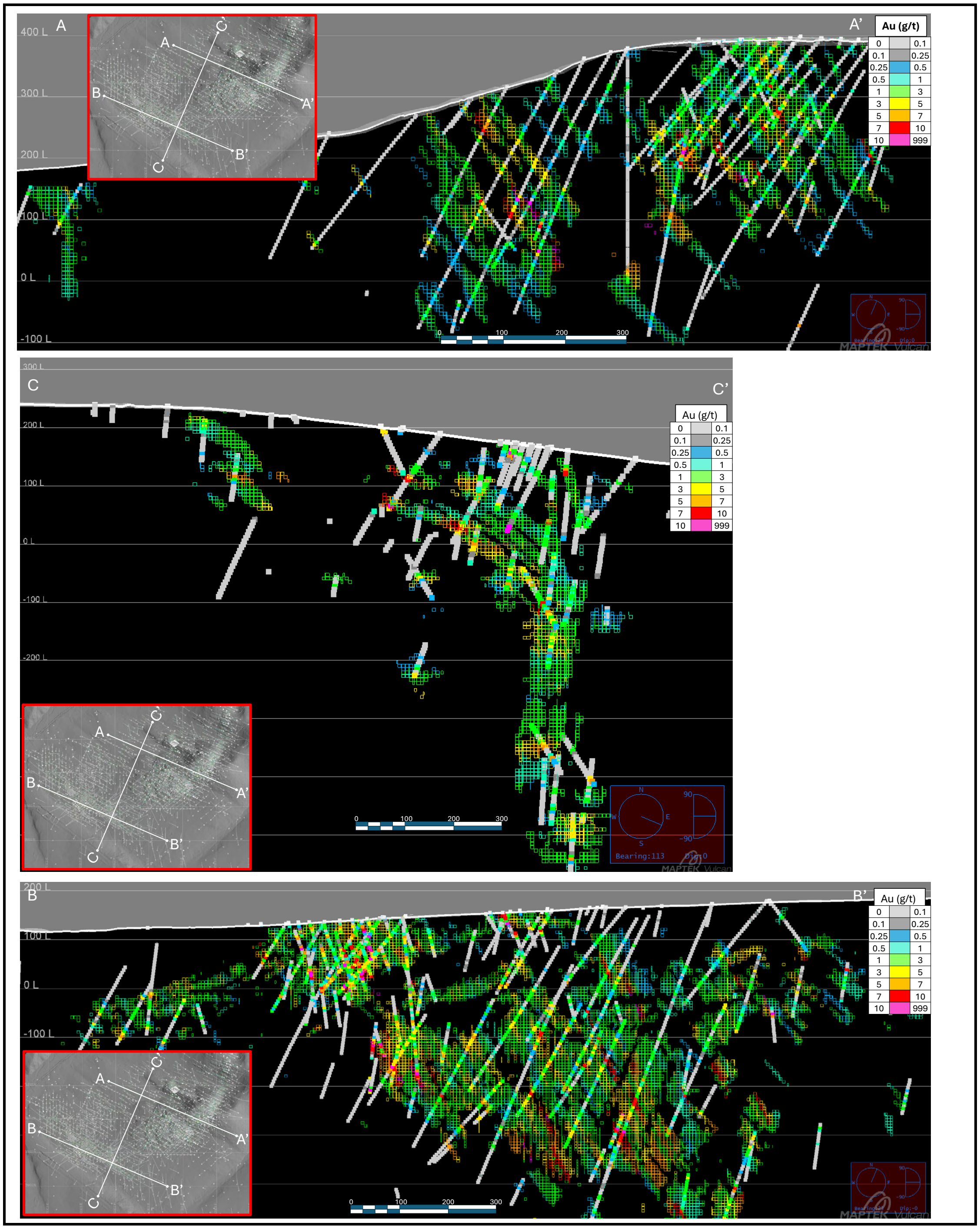
Box plots and summary statistics comparing the estimated gold block grades (Au ID3) and the declustered composites represented by the NN gold grades (Au NN) by estimation domains are shown in Figure 14-4 and Table 14-9. Mean values of the estimated gold and the declustered gold by estimation domains show good agreement.

Figure 14-5 shows swath plots of the gold grade profiles of the Measured and Indicated estimated blocks and the NN model along the easting, northing, and elevation directions. Swath plots show good grade profile agreements between the estimated gold grades and the declustered composite (NN) grades with no local estimation bias observed.

The relative degree of smoothing in the block estimates was evaluated using the Hermitian Polynomial Change of Support (HERCO) method, also known as the Discrete Gaussian Correction. A set of well-supported blocks (Measured and Indicated) inside the Mineral Resource pit was selected for the HERCO calculation using the 6 x 6 x 6 m and 12 x 12 x 6 m block models. Figure 14-6 shows HERCO grade-tonnage curves illustrating that the estimated model is under-smoothed at 0.5 g/t Au cut-off, slightly above the breakeven cut-off of 0.47 g/t Au for the 6 x 6 x 6 m block model. Figure 14-7 shows HERCO grade-tonnage curves illustrating that the estimated model is under-smoothed at 0.5 g/t Au, slightly above the breakeven cut-off of 0.47 g/t Au, for the 12 x 12 x 6 m model within an acceptable limit for the tonnes and above the acceptable limit for the grade

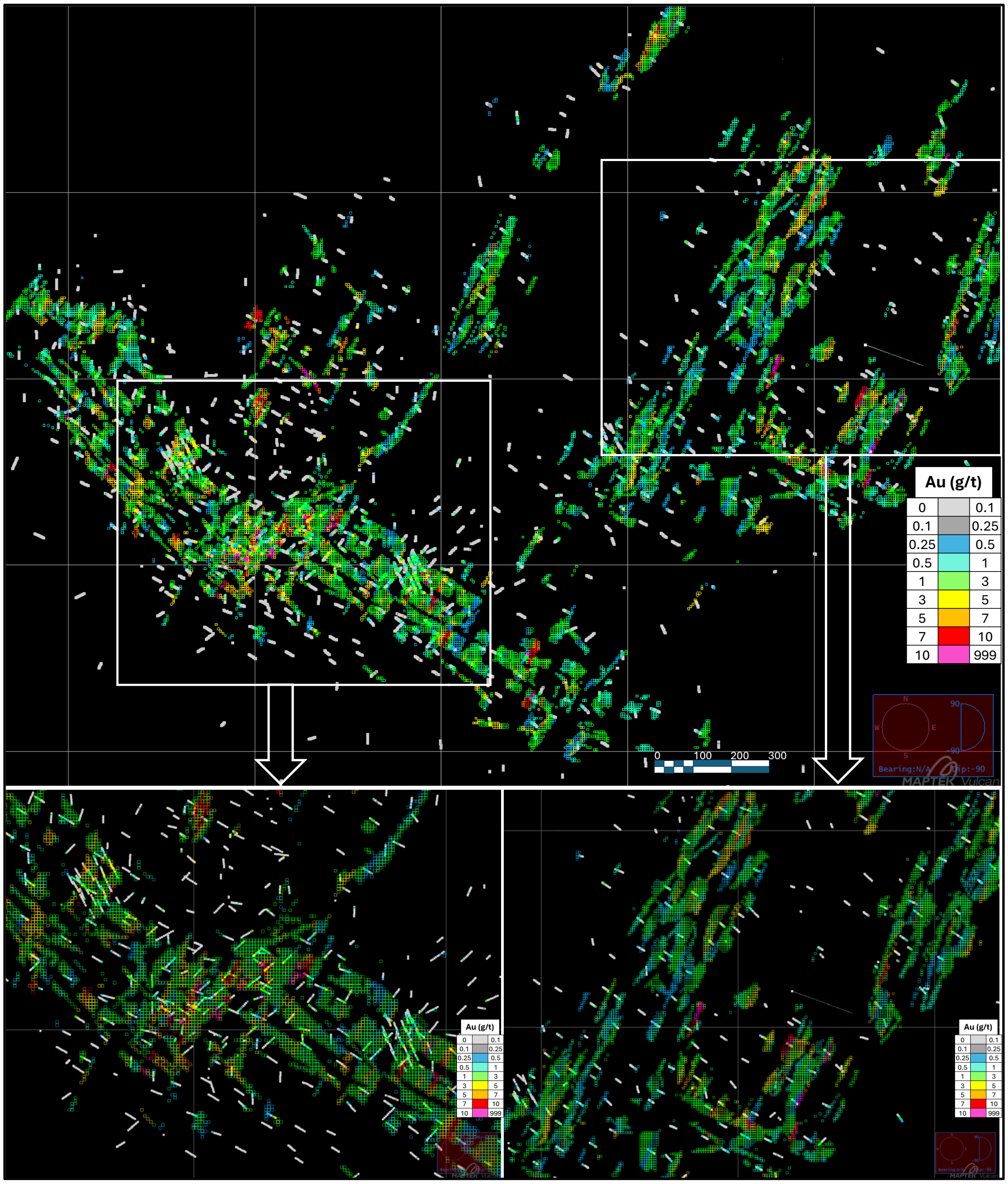
Additional dilution was applied to the 12 x 12 x 6 m block model for mine planning and Mineral Reserve estimation that accounts for a mix of in-situ and contact dilution, and mineralization loss. This process adds additional dilution, which offsets the under-smoothed re-blocked 12 x 12 x 6 m Mineral Resource model.

Figure 14-2: Section View Maps Showing the Estimated Gold Grades and Composite Au Grades for Measured and Indicated Blocks, ±20 m



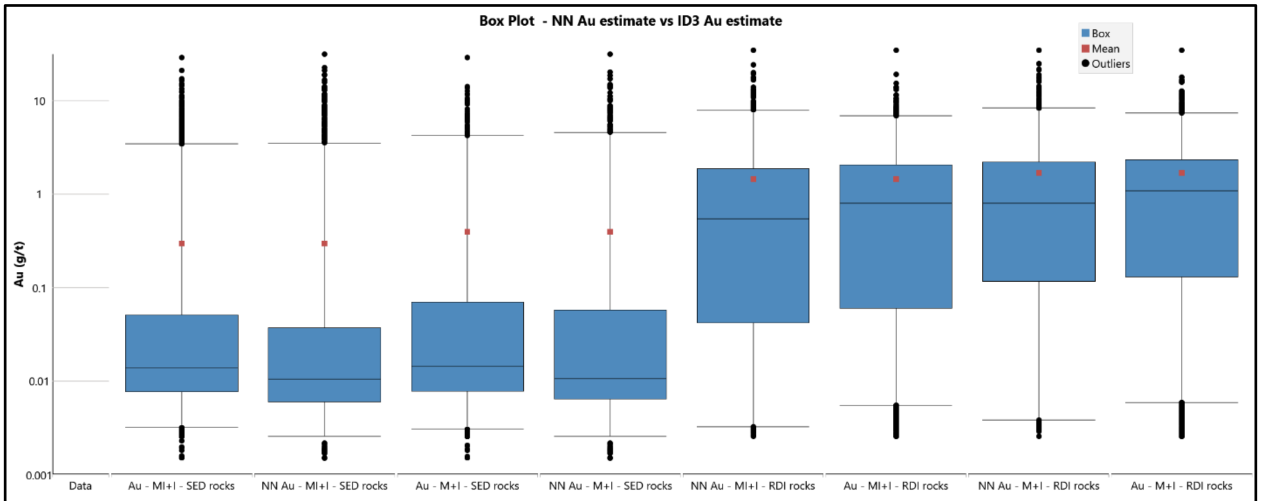
Source: Wood, 2025

Figure 14-3: Plan View Maps Showing the Estimated Gold Grades and Composite Au Grades for Measured and Indicated Blocks, 0m Elevation ±20 m



Source: Wood, 2025

Figure 14-4: Box Plots of Au ID3 and Au NN by Estimation Domain



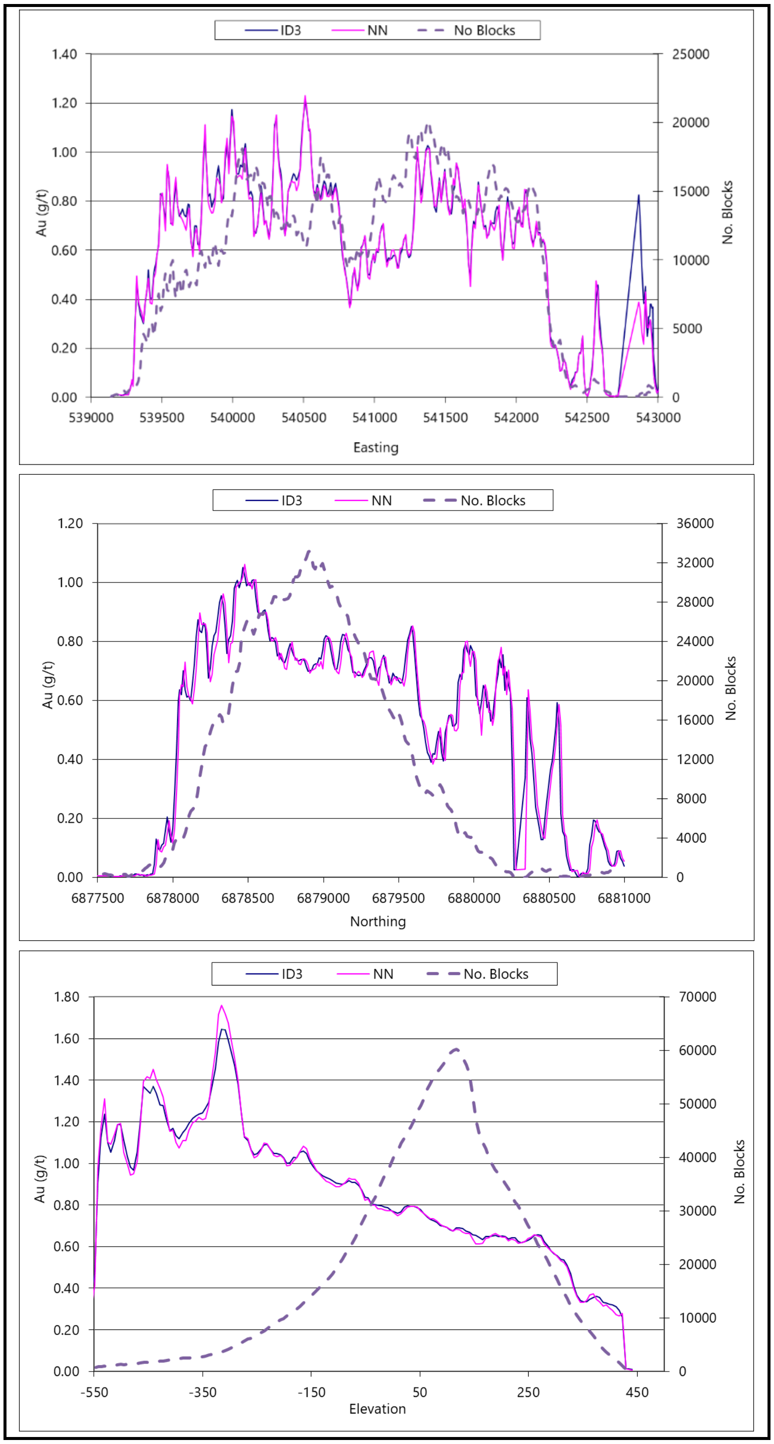
Data	Au - MI+I - SED rocks	NN Au - MI+I - SED rocks	Au - M+I - SED rocks	NN Au - M+I - SED rocks	NN Au - MI+I - RDI rocks	Au - MI+I - RDI rocks	NN Au - M+I - RDI rocks	Au - M+I - RDI rocks
Number of samples	3,633,168	3,631,710	2,054,567	2,054,019	1,482,493	1,482,804	1,147,388	1,147,611
Max	24.07	26.00	24.07	26.00	28.59	28.59	28.59	28.59
97.5% quantile	2.95	3.01	3.63	3.90	6.67	5.84	7.04	6.22
Upper quartile	0.05	0.04	0.06	0.05	1.64	1.79	1.92	2.03
Median	0.01	0.01	0.01	0.01	0.49	0.70	0.70	0.95
Lower quartile	0.01	0.01	0.01	0.01	0.04	0.06	0.10	0.12
2.5% quantile	0.00	0.00	0.00	0.00	0.00	0.01	0.00	0.01
Min	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00
Mean	0.27	0.27	0.35	0.36	1.26	1.27	1.46	1.46
CV	3.66	4.11	3.24	3.60	1.62	1.38	1.46	1.24

Source: Wood, 2025

Table 14-9: Summary Statistics of Gold ID3 and NN

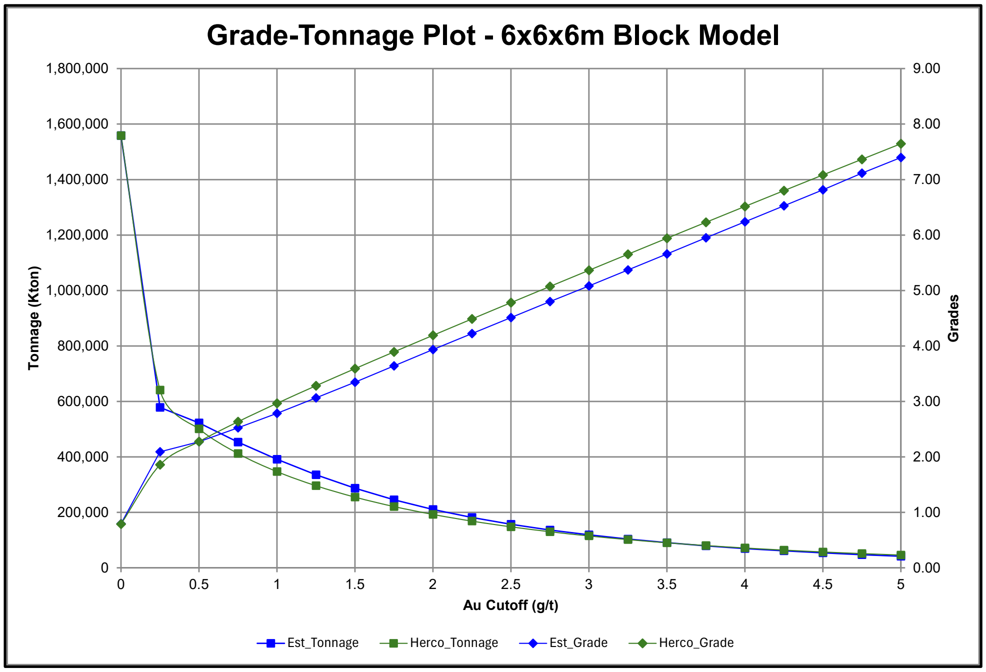
Estimation Type	Classification Category	No. Samples	Min (g/t)	Max (g/t)	Mean (g/t)	CV
Sedimentary Rock						
Au ID3	MI+I	3633168	0.00	24.07	0.27	3.66
Au NN	MI+I	3631710	0.00	26.00	0.27	4.11
Au ID3	M+I	2054567	0.00	24.07	0.35	3.60
Au NN	M+I	2054019	0.00	26.00	0.36	3.62
RDI Intrusive Rock						
Au ID3	MI+I	1482804	0.00	28.59	1.27	1.38
Au NN	MI+I	1482493	0.00	28.59	1.26	1.62
Au ID3	M+I	1147611	0.00	28.59	1.46	1.24
Au NN	M+I	1147388	0.00	28.59	1.46	1.46

Figure 14-5: Gold Swath Plots



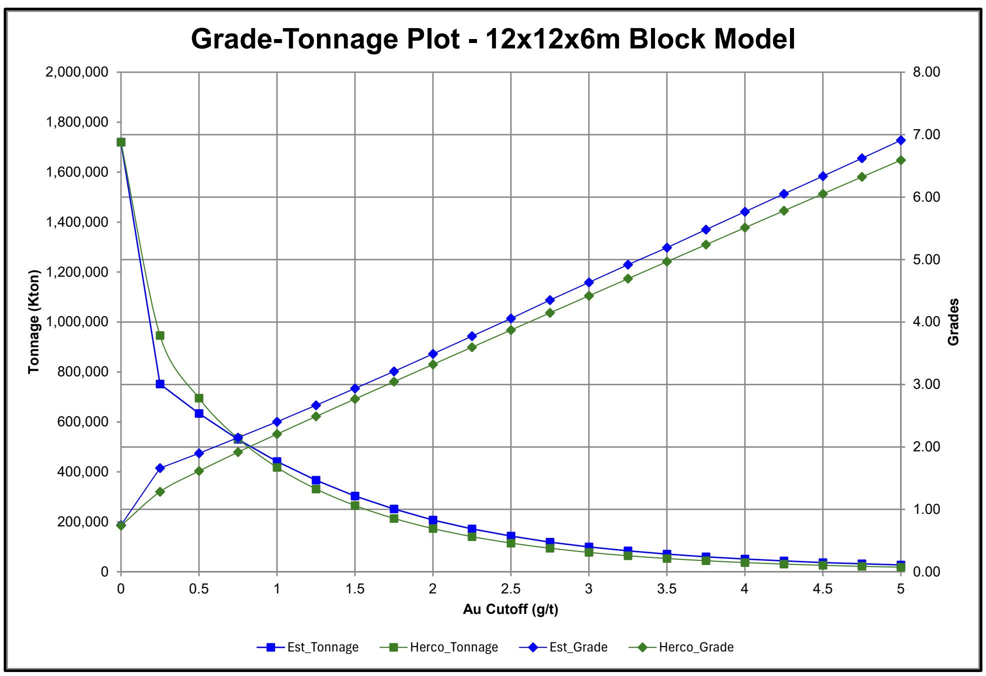
Source: Wood, 2025

Figure 14-6: HERCO Change of Support Grade-Tonnage Plots



Source: Wood, 2025

Figure 14-7: 12 x 12 x 6 m Block-HERCO Change of Support Grade-Tonnage Plots



Source: Wood (2025)

14.13 Classification

Mineral Resource classification criteria include the distance to the nearest composite as well as the number of drill holes used to estimate a block and the block indicator values (see Table 14-10).

Measured Mineral Resources were classified in four areas of closely spaced drilled zones, conducted in 2021 and 2022. The areas were drilled at grid distances of 20 x 20 m with some infill grids of 10 x 10 m. Wireframe solids were created around the grids and were used to flag the block model in the Measured category.

Indicated Mineral Resources were classified for blocks where the distance to the closest composite equates to the range of approximately 85% of the variance of the omni-directional indicator correlogram model, or 30 m.

Inferred Mineral Resources were classified for blocks where the distance to the closest composite equates to the range of approximately 90% of the variance of the omni-directional indicator correlogram model, or 60 m.

Additional drilling since 2010 confirmed longer ranges within the intrusive rocks and has confirmed more consistent sediment mineralization which is reflected by the indicator block conditions.

Table 14-10: Mineral Resource Classification

Classification Category	Min. Distance to Composite (m)	Max. Distance to Nearest Composite (m)	Min. No. of Drill Holes	Indicator Block Condition Criteria
All				
Measured	Wireframe solids around four 10-20 m grids			
Intrusive or Mafic Dike				
Indicated	-	30	≥2	≥0.0
Indicated	30	50	≥2	≥0.5
Inferred	30	50	≥2	≥0.0 & <0.5
Inferred	50	65	≥2	≥0.5
Sediments				
Indicated	-	30	≥2	≥0.0
Indicated	30	45	≥2	≥0.6
Inferred	30	45	≥2	≥0.0 & <0.6
Inferred	45	60	≥2	≥0.6

14.14 Reasonable Prospects for Eventual Economic Extraction

To determine reasonable prospects for eventual economic extraction, a Mineral Resource pit optimization analysis was performed using the Lerchs-Grossman (LG) algorithm provided in GEOVIA Whittle software. The analysis used the economic parameters summarized in Table 14-11. Only gold was used for the optimization. All costs were based on the 2021 feasibility study and escalated to 2025 pricing.

Gold recovery values are based on work completed for the Project. Metallurgical recoveries for unoxidized ores are quoted as a constant for each intrusive rock type, whereas recoveries for sediments used a weighted average of mixed sediments recovery for greywacke (88.22%) and shale (86.66%). Oxide ore recoveries vary with sulfur grade. The recoveries applied in pit optimization are listed in Table 15–4.

Table 14-11: Mineral Resource Pit Optimization Parameters

Economic Parameters	Unit	Value
Gold price forecast	$/oz	2,400
Reference mining elevation	masl	220.0
Base mining operating cost	$/t mined	2.68
Uphill incremental mining cost	$/t mined/m	0.0041
Mining sustaining cost	$/t mined	0.41
Range of process recoveries	%	29.4-94.17 (see Table 15-4)
Process operating cost	$/t processed	20.01
Process sustaining cost	$/t processed	2.14
G&A cost	$/t processed	4.57
Stockpile reclaim cost	$/t processed	0.30
Reclaim percentage	%	45.0
Refining recovery	%	99.9
Selling costs[1]	$/oz recovered	1.71
NSR royalty	%NSR	4.5
Production royalty	$/t processed	0.50
Pit slope angle	degrees	22–47 (see Table 15-5 to Table 15-7)

Note: (1) Selling costs include refining, freight and marketing costs

The long-term gold price forecast was applied to the 24-year expected mine life over which those Mineral Resources would be produced. This long-term forecast price is based on a combination of gold price information derived from financial institution reports, from pricing used in technical reports filed on SEDAR+ with Canadian regulatory authorities over the previous

12-month period, from pricing reported by major mining companies in public filings such as their annual reports in the previous 12-month period, spot pricing, and three-year trailing average pricing. From this assessment QP Kim considers industry consensus on a long-term price forecast on Mineral Reserves and cash flow of $2,100/oz gold is reasonable. In accordance with industry-accepted practice, a higher gold price of $2,400/oz is used for Mineral Resources than what is used for Mineral Reserves. This gold price forecast represents a 15% increase on the gold price used for Mineral Reserves, which attempts to ensure that the Mineral Reserves are a subset of the Mineral Resources.

14.14.1 Cut-off Determination

The net smelter return (NSR) and block value per tonne (VPT) are calculated using the following expressions:

NSR ($/t) = [(Au grade) x (processing recovery) x (refining recovery) x (Au price – selling costs) x (100% – NSR royalty)] – (production royalty)

VPT ($/t) = NSR – (mining costs[1] + processing costs[2] + G&A cost + stockpile reclaim cost[3])

Note: (1) Mining costs include base, incremental, and sustaining costs
(2) Processing costs include operating and sustaining costs
(3) Stockpile reclaim cost is affected by the reclaim percentage assumed

Blocks contained inside the Mineral Resource pit shell with a marginal block value (NSR > processing costs + G&A cost + stockpile reclaim cost) were considered as having reasonable prospects for eventual economic extraction assuming open pit mining methods. The marginal NSR cut-off value of $26.86/t is applied at the pit rim to determine whether the material is sent to the process plant or the WRF. The mining costs have already been considered in the pit optimization process.

14.15 Mineral Resource Statement

Mineral Resources inclusive of Mineral Reserves are summarized in Table 14-12 and Mineral Resources exclusive of Mineral Reserves are summarized in Table 14-13. Mineral Resources are defined using a marginal NSR cut-off value of $26.86/t assuming open pit mining methods and are classified in accordance with the CIM Definition Standards. The point of reference for the Mineral Resource estimate is in situ.

Table 14-12: Mineral Resources Statement Inclusive of Mineral Reserves

Category	Tonnage (kt)	Au Grade (g/t)	Contained Au (koz)	S Grade (%)
Measured	9,243	2.67	793	1.25
Indicated	550,727	2.21	39,195	1.12
Total Measured and Indicated	**559,970**	**2.22**	**39,988**	**1.12**
Inferred	88,886	2.03	5,812	1.09

Note: (1) The effective date of the Mineral Resource estimate is 30 November 2025. The QP for the Mineral Resource estimate is Mr. Henry Kim, P.Geo., an employee of Wood.

(2) Mineral Resources are prepared in accordance with CIM Definition Standards and the CIM MRMR Best Practice Guidelines.

(3) Mineral Resources are inclusive of Mineral Reserves.

(4) The cut-off date for the sample database used in the Mineral Resource estimate is 31 December 2024. However, more recent drilling data up to 30 November 2025 was used to validate the Mineral Resource model as remaining current.

(5) Mineral Resources are constrained within a pit shell using the following assumptions: gold price of $2,400/oz; reference mining cost of $2.68/t mined incremented $0.0041/t mined/m with depth from the 220 m elevation (equates to an average mining cost of $3.23/t mined); mining sustaining cost of $0.41/t mined; variable metallurgical recoveries by rock type and geological domain, ranging from 29.4% in oxide to 94.17% in intrusive rocks in the Akivik domain; process operating cost of $20.01/t processed; process sustaining cost of $2.14/t processed; G&A cost of $4.57/t processed; stockpile reclaim cost of $0.30/t processed assuming a reclaim percentage of 45%; refining recovery of 99.9%; selling cost of $1.71/oz gold; royalty considerations of 4.5% NSR and $0.50/t processed; and variable pit slope angles, ranging from 22 to 47°.

(6) The NSR value for each block is determined using the gold grade, processing and refining recoveries, gold price, selling costs, and royalties.

(7) Mineral Resources are reported using a marginal NSR cut-off value of $26.86/t based on a total process cost of $22.15/t processed, G&A operating cost of $4.57/t processed, and a stockpile reclaim cost of $0.30/t processed assuming a reclaim percentage of 45%.

(8) The average LOM process recovery for Mineral Resources is 89.8%.

(9) Sulfur is not an economic contributor to the Project; however, it does impact the POX process.

(10) Tonnage and grade measurements are in metric units. Contained gold ounces are reported as troy ounces.

(11) Rounding may result in apparent summation differences between tonnes, grade, and contained metal content.

Table 14-13: Mineral Resources Statement Exclusive of Mineral Reserves

Category	Tonnage (kt)	Au Grade (g/t)	Contained Au (koz)	S Grade (%)
Measured	1,432	1.18	54	1.05
Indicated	175,224	1.32	7,439	1.00
Total Measured and Indicated	**176,656**	**1.32**	**7,493**	**1.00**
Inferred	74,426	1.87	4,483	1.06

Note: (1) The effective date of the Mineral Resource estimate is 30 November 2025. The QP for the Mineral Resource estimate is Mr. Henry Kim, P.Geo., an employee of Wood.

(2) Mineral Resources are prepared in accordance with CIM Definition Standards and the CIM MRMR Best Practice Guidelines.

(3) Mineral Resources are exclusive of Mineral Reserves.

(4) The cut-off date for the sample database used in the Mineral Resource estimate is 31 December 2024. However, more recent drilling data up to 30 November 2025 was used to validate the Mineral Resource model as remaining current.

(5) Mineral Resources are constrained within a pit shell using the following assumptions: gold price of $2,400/oz; reference mining cost of $2.68/t mined incremented $0.0041/t mined/m with depth from the 220 m elevation (equates to an average mining cost of $3.23/t mined); mining sustaining cost of $0.41/t mined; variable metallurgical recoveries by rock type and geological domain, ranging from 29.4% in oxide to 94.17% in intrusive rocks in the Akivik domain; process operating cost of $20.01/t processed; process sustaining cost of $2.14/t processed; G&A cost of $4.57/t processed; stockpile reclaim cost of $0.30/t processed assuming a reclaim percentage of 45%; refining recovery of 99.9%; selling cost of $1.71/oz gold; royalty considerations of 4.5% NSR and $0.50/t processed; and variable pit slope angles, ranging from 22 to 47°.

(6) The NSR value for each block is determined using the gold grade, processing and refining recoveries, gold price, selling costs, and royalties.

(7) Mineral Resources are reported using a marginal NSR cut-off value of $26.86/t based on a total process cost of $22.15/t processed, G&A operating cost of $4.57/t processed, and a stockpile reclaim cost of $0.30/t processed assuming a reclaim percentage of 45%.

(8) The average LOM process recovery for Mineral Resources is 89.8%.

(9) Sulfur is not an economic contributor to the Project; however, it does impact the POX process.

(10) Tonnage and grade measurements are in metric units. Contained gold ounces are reported as troy ounces.

(11) Rounding may result in apparent summation differences between tonnes, grade, and contained metal content.

14.16 Factors that May Affect the Mineral Resource Estimate

Factors that may affect the Mineral Resource estimates include:

- Changes to the assumed gold price
- Changes to the assumed metallurgical recoveries
- Changes to the pit slope angles and geotechnical characteristics of the rock mass
- Changes to the assumptions used to generate the resource cut-off
- Changes to the gold threshold for defining the indicator mineralized shells
- Changes in interpretations of fault geometry, in particular the Vortex and Lo faults
- Changes to the search orientations used for grade estimation
- Changes to the geological model; in general, all geological models are a simplified presentation of what occurs in nature. With more data, the geological models reflect more of the actual complexity of the deposit
- Changes to the Mineral Resource classification criteria.

QP Kim is not aware of any other known risks that could materially affect the Mineral Resource estimate.

15.0 MINERAL RESERVE ESTIMATES

15.1 Modifying Factors

15.1.1 Summary

Mineral Reserves are optimized for all Measured and Indicated blocks assuming a gold price forecast of $2,100/oz. Mineral Reserves include dilution based on the block model, which identified blocks amenable to bulk mining (12 m high benches) and selective mining (6 m high benches). Economic parameters for Mineral Reserves are summarized in Table 15-1. All costs were based on the feasibility study and escalated to 2025 pricing.

The Mineral Reserve pit shell is constrained in the northwestern part of the ACMA mining area to prevent it from encroaching on Crooked Creek, which is a salmon-bearing stream.

Table 15-1: Mineral Reserve Pit Optimization Parameters

Economic Parameters	Unit	Value
Gold price forecast	$/oz	2,100
Reference mining elevation	masl	220.0
Base mining operating cost	$/t mined	2.68
Uphill incremental mining cost	$/t mined/m	0.0041
Mining sustaining cost	$/t mined	0.41
Range of process recoveries	%	29.4-94.17 (see Table 15-4)
Process operating cost	$/t processed	20.01
Process sustaining cost	$/t processed	2.14
G&A cost	$/t processed	4.57
Stockpile reclaim cost	$/t processed	0.30
Reclaim percentage	%	45.0
Refining recovery	%	99.9
Selling costs[1]	$/oz recovered	1.71
NSR royalty	%NSR	4.5
Production royalty	$/t processed	0.50
Pit slope angle	degrees	22–47 (see Table 15-5 to Table 15-7)

Note: (1) Selling costs include refining, freight and marketing costs.

15.1.2 Dilution and Mine Loss

The Mineral Resources for the Project were reported on an undiluted basis. To determine Mineral Reserves, dilution was applied to the Mineral Resource model through re-blocking. The original 6 m x 6 m x 6 m Mineral Resource model was re-blocked to a 12 m x 12 m x 6 m model on a regular grid, blending all material. Measured and Indicated material retained its grade, while Inferred material was assigned a gold grade of 0 g/t.

An elevated cut-off grade was applied to limit the impacts of dilution and prevent exceeding the capacity of the TSF.

Further re-blocking was completed on the 12 m x 12 m x 6 m model to a hybrid 6 m/12 m bench height model following an evaluation on bulk mining (12 m bench) versus selective mining (6 m bench). Table 15-2 shows the distribution of mined material of the two mining methods.

Table 15-2: Bulk vs. Selective Mining

Mining Method	Bench Height (m)	Ore Tonnes (kt)	Waste Tonnes (kt)
Bulk	12	446,766	3,192,442
Selective	6	58,942	105,340

Bulk mining reduces average operating costs throughout the LOM; however, it also adds further dilution and mine losses.

A summary of dilution and mine losses is shown in Table 15-3. Mine losses include:

- isolated blocks that are above cut-off undiluted but below cut-off when diluted
- material remaining in the stockpile (due to TSF capacity constraints).

Mine dilution includes:

- Measured and Indicated material below cut-off that is mined with the blocks above cut-off. This diluting material retains its grade
- Inferred material that is mined with blocks above cut-off. This diluting material is taken at zero grade

This total mine loss accounts for 18.5% of potential mill feed tonnes, or 10.5% of potential mill feed ounces. Total dilution is 29% at 0.15 g/t gold of total mill feed tonnes. Figure 15-1 and

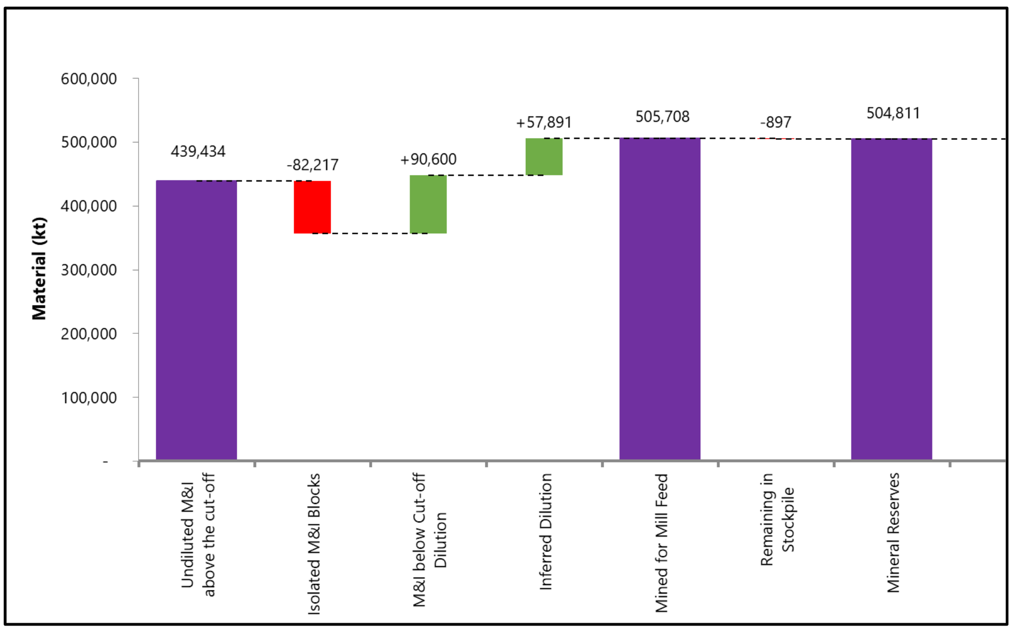
Figure 15-2 shows the same information graphically. Estimating dilution and mine loss through re-blocking is a conservative approach. Alternative approaches may result in reduced dilution.

Table 15-3: Summary of Dilution and Mine Losses

Item	Material (kt)	Au (g/t)	Au (koz)	S (%)
Undiluted M&I above cut-off	439,434	2.54	35,820	1.16
Isolated M&I blocks (loss)	(82,217)	1.41	(3,735)	1.03
M&I below cut-off dilution	90,600	0.25	737	0.82
Inferred dilution	57,891	0.00	0	0.88
Total dilution	148,492	0.15	737	0.84
Mined for mill feed	505,708	2.02	32,822	1.09
Remaining in stockpile (loss)	(897)	0.85	(25)	0.88
Mineral Reserves	**504,811**	**2.02**	**32,797**	**1.09**

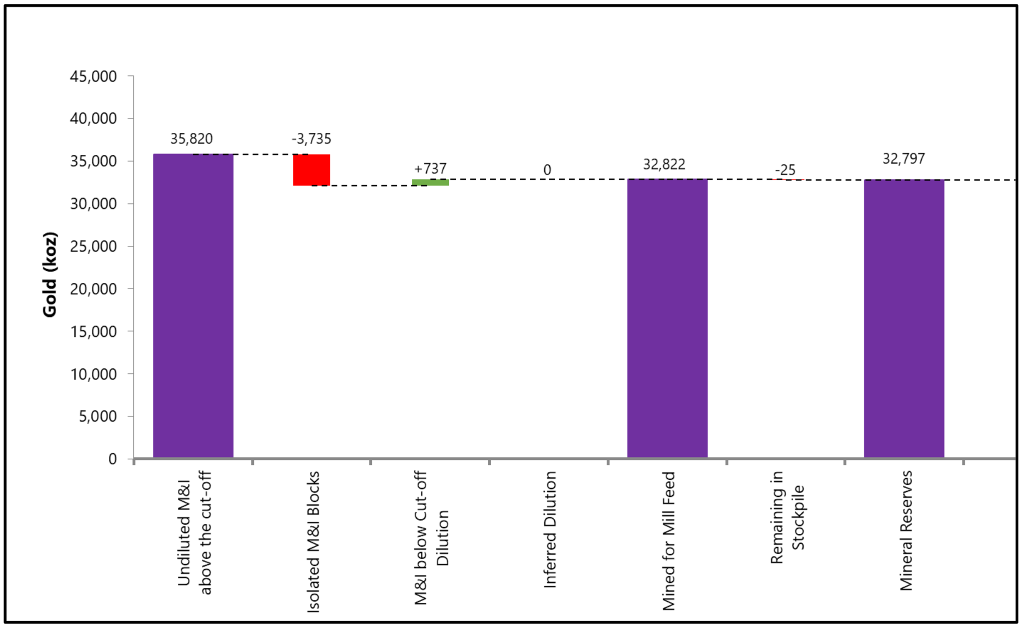
Note: M&I = Measured and Indicated blocks. Figures may not sum due to rounding.

Figure 15-1: Dilution and Mine Loss (Material Tonnage)



Source: Wood, 2025

Figure 15-2: Dilution and Mine Loss (Metal Content)



Source: Wood, 2025

15.1.3 Metallurgical Recovery

Gold recovery values are based on metallurgical testwork completed for the Project. Metallurgical recoveries for unoxidized ores are quoted as a constant for each intrusive rock type, whereas recoveries for sediments used a weighted average of mixed sediments recovery for greywacke (88.22%) and shale (86.66%). Oxide ore recoveries vary with sulfur grade. The recoveries applied in pit optimization are listed in Table 15-4.

15.1.4 Gold Price Forecast

The long-term gold price forecast was applied to the 27-year expected mine life over which those Mineral Reserves would be produced. This long-term forecast price is based on a combination of gold price information derived from financial institution reports, from pricing used in technical reports filed on SEDAR+ with Canadian regulatory authorities over the previous 12-month period, from pricing reported by major mining companies in public filings such as their annual reports in the previous 12-month period, spot pricing, and three-year trailing average pricing. From this assessment QP Peralta considers industry consensus on a long-term price forecast on Mineral Reserves and cash flow of $2,100/oz gold is reasonable.

Table 15-4: Pit Optimization Process Recoveries

Rock Type	Geological Domain	Au Recovery (%)
Unoxidized Ores		
Intrusive	Akivik	94.17
Intrusive	400	93.55
Intrusive	ACMA	93.05
Intrusive	Aurora	93.61
Intrusive	Vortex	91.82
Intrusive	Lewis	91.52
Intrusive	Tortured[1]	93.05
Intrusive	Wedge[1]	93.05
Mixed sediments[2]	All	(0.80 x 88.22) + (0.20 x 86.66)
Oxide/Weathered Ores		
Oxide/weathered rocks – S >1.8%		87.90
Oxide/weathered rocks – S ≤1.8%		$[(8.7361 \times S^3) - (49.806 \times S^2) + (95.233 \times S + 30.004)] \times 0.966$

Note: (1) Recoveries for Tortured and Wedge intrusives adopt the ACMA recovery
(2) Mixed sediments recovery estimated assuming 80% greywacke and 20% shale

15.1.5 Pit Slopes

Geotechnical domains, design sectors, slope angles, and associated assumptions were provided by BGC Engineering Inc. (BGC) (2023c). BGC's inter-ramp slope angles were reduced for each design sector in each of the geotechnical domains to flatten the pit shell allowing for haulage ramps to be included in the mine design. Slope angle reductions were based on the haulage ramp width, the number of times a haulage ramp traversed a design sector, geotechnical berms, and the overall slope height of the sector.

Certain slope angles were further adjusted to smooth the transition to an adjacent design sector. This enabled the LG software to generate structural arcs in cases where the slope angles contrasted sharply in "narrow" design sectors. The slope angles were either increased or decreased to enable the generation of arcs while attempting to preserve slope steepness. The slope angles used in the pit optimizations are shown in Table 15-5 to Table 15-7.

Table 15-5: Pit Optimization Slopes – Weathered/Oxidized

Domain	BGC Design Sector Start	BGC Design Sector End	Whittle™ Bearing	BGC Slope Angle	Reduced Slope Angle	Whittle™ Transition Adjusted
Lewis	348	57	22.5	27	27	27
	57	110	83.5	33	33	33
	110	171	140.5	39	39	39
	171	244	207.5	35	35	35
	244	305	274.5	41	41	41
	305	348	326.5	35	35	35
Lewis Steep	333	68	20.5	34	34	34
	68	180	124.0	37	37	37
	180	241	210.5	40	40	40
	241	285	263.0	40	40	40
	285	333	309.0	37	37	37
North Limb	337	42	9.5	33	33	33
	42	98	70.0	35	35	35
	98	150	124.0	36	28	28
	150	184	167.0	38	38	35
	184	242	213.0	40	40	40
	242	302	272.0	38	38	38
	302	337	319.5	37	37	37
North Dipping Syncline	0	60	30.0	40	40	40
	60	107	83.5	39	39	30
	107	154	130.5	27	21	22
	154	187	170.5	27	27	27
	187	247	217.0	26	26	26
	247	305	276.0	35	35	35
	305	0	332.5	40	40	40
South Dipping	359	29	14.0	33	33	31
	29	77	53.0	28	28	28
	77	153	115.0	37	34	34
	153	214	183.5	27	27	27
	214	317	265.5	40	31	31
	317	359	338.0	28	26	26
Steep Limb	10	71	40.5	34	34	34
	71	185	128.0	35	35	35
	185	251	218.0	37	37	37
	251	310	280.5	37	37	37
	310	10	340.0	39	39	39
Runway	3	83	43.0	32	32	32
	83	150	116.5	36	36	36
	150	274	212.0	39	39	39
	274	319	296.5	40	40	40
	319	3	341.0	35	35	35

Table 15-6: Pit Optimization Slopes – Fresh/Unoxidized

Domain	BGC Design Sector Start	BGC Design Sector End	Whittle™ Bearing	BGC Slope Angle	Reduced Slope Angle	Whittle™ Transition Adjusted
Lewis	348	57	22.5	27	25	25
	57	110	83.5	42	40	40
	110	171	140.5	46	42	42
	171	244	207.5	35	28	28
	244	305	274.5	48	40	40
	305	348	326.5	43	36	36
Lewis Steep	333	68	20.5	34	24	24
	68	180	124.0	45	40	40
	180	241	210.5	40	29	29
	241	285	263.0	47	35	35
	285	333	309.0	45	39	39
North Limb	337	42	9.5	33	32	32
	42	98	70.0	35	31	31
	98	150	124.0	36	34	34
	150	184	167.0	44	41	40
	184	242	213.0	40	31	31
	242	302	272.0	46	35	35
	302	337	319.5	44	36	36
North Dipping Syncline	0	60	30.0	42	42	42
	60	107	83.5	46	46	36
	107	154	130.5	27	28	27
	154	187	170.5	27	27	27
	187	247	217.0	26	26	26
	247	305	276.0	43	43	43
	305	0	332.5	47	40	40
South Dipping	359	29	14.0	33	30	32
	29	77	53.0	28	28	28
	77	153	115.0	45	39	39
	153	214	183.5	27	27	27
	214	317	265.5	47	40	40
	317	359	338.0	28	27	27
Steep Limb	10	71	40.5	34	32	32
	71	185	128.0	43	37	37
	185	251	218.0	37	31	31
	251	310	280.5	45	36	36
	310	10	340.0	46	37	37
Runway	3	83	43.0	32	32	32
	83	150	116.5	44	44	44
	150	274	212.0	47	36	36
	274	319	296.5	47	39	39
	319	3	341.0	40	36	36

Table 15-7: Pit Optimization Slopes – Fresh Divide Fault

Domain	BGC Design Sector Start	BGC Design Sector End	Whittle™ Bearing	BGC Slope Angle	Reduced Slope Angle	Whittle™ Transition Adjusted
Lewis Steep	333	68	20.5	34	34	34
	68	180	124.0	29	29	29
	180	241	210.5	27	27	27
	241	285	263.0	47	37	37
	285	333	309.0	45	45	45
North Limb	337	42	9.5	33	33	33
	42	98	70.0	35	35	35
	98	150	124.0	36	36	34
	150	184	167.0	27	27	27
	184	242	213.0	40	40	36
	242	302	272.0	46	46	46
	302	337	319.5	44	44	44
North Dipping Syncline	0	60	30.0	29	29	29
	60	107	83.5	46	46	36
	107	154	130.5	27	27	27
	154	187	170.5	27	27	27
	187	247	217.0	26	26	26
	247	305	276.0	43	43	43
	305	0	332.5	47	45	45
South Dipping	359	29	14.0	27	27	27
	29	77	53.0	28	28	28
	77	153	115.0	45	45	45
	153	214	183.5	27	27	27
	214	317	265.5	47	47	47
	317	359	338.0	28	29	29

15.2 Cut-off Determination

Due to the capacity constraints of the TSF, which was designed to allow processing a maximum of 504.8 Mt of ore, a gold cut-off grade sensitivity analysis was conducted during pit optimization to determine an elevated cut-off grade that will limit ore sent to the process plant.

An elevated gold cut-off grade of 0.74 g/t was initially used to determine the Mineral Reserve blocks within the Mineral Reserve pit. However, for the definition of the ore within the pit design and for the mine plan, a slightly increased elevated gold cut-off grade of 0.75 g/t was used.

The net smelter return (NSR) and block value per tonne (VPT) are calculated using the following expressions:

*NSR ($/t) = [(Au grade) x (processing recovery) x (refining recovery)
x (Au price – selling costs) x (100% – NSR royalty)] – (production royalty)*

VPT ($/t) = NSR – (mining costs[1] + processing costs[2] + G&A cost + stockpile reclaim cost[3])

Note: (1) Mining costs include base, incremental, and sustaining costs
(2) Processing costs include operating and sustaining costs
(30 Stockpile reclaim cost is affected by the reclaim percentage assumed

When determining Mineral Reserves, a two-pass approach was taken:

- Blocks contained inside the pit design with an economic value greater than the operating costs which range between $29.95 and 32.36/t
- Blocks above 0.75 g/t gold.

15.3 Mineral Reserve Statement

Mineral Reserves are summarized in Table 15-8. Using the proposed open pit mining method, modifying factors identified and discussed in Section 15.1 have been applied to the Measured and Indicated Mineral Resources to determine Proven and Probable Mineral Reserves, in accordance with the CIM Definition Standards. Inferred Mineral Resources contained within the pit design are classified as waste. The point of reference for the Mineral Reserve estimate is at the point of delivery to the process plant.

Table 15-8: Mineral Reserve Statement

Category	Tonnage (kt)	Au Grade (g/t)	Contained Au (koz)	S Grade (%)
Proven	9,487	2.29	698	1.15
Probable	495,324	2.02	32,099	1.09
Total Proven and Probable	**504,811**	**2.02**	**32,797**	**1.09**

Note: (1) The effective date of the Mineral Reserve estimate is 30 November 2025. The QP for the Mineral Reserve estimate is Mr. Edwin Peralta, PE, an employee of Wood.

(2) Mineral Reserves are prepared in accordance with CIM Definition Standards and the CIM MRMR Best Practice Guidelines.

(3) Mineral Reserves are constrained within an engineered pit design using the following assumptions: gold price of $2,100/oz; reference mining cost of $2.68/t mined incremented $0.0041/t mined/m with depth from the 220 m elevation (equates to an average mining cost of $3.23/t mined); mining sustaining cost of $0.41/t mined; variable metallurgical recoveries by rock type and geological domain, ranging from 29.4% in oxide to 94.17% in intrusive rocks in the Akivik domain; process operating cost of $20.01/t processed; process sustaining cost of $2.14/t processed; G&A cost of $4.57/t processed; stockpile reclaim cost of $0.30/t processed assuming a reclaim percentage of 45%; refining recovery of 99.9%; selling cost of $1.71/oz gold; royalty considerations of 4.5% NSR and $0.50/t processed; and variable pit slope angles, ranging from 22 to 47°.

(4) Mineral Reserves are reported using an economic NSR cut-off value of $29.95–32.36/t followed by an elevated gold cut-off grade of 0.75 g/t. The NSR value for each block is determined using the gold grade, processing and refining recoveries, gold price, selling costs, and royalties.

(5) The average LOM processing recovery for the Mineral Reserves is 90.0%.

(6) Sulfur is not an economic contributor to the Project; however, it does impact the POX process.

(7) Tonnage and grade measurements are in metric units. Contained gold ounces are reported as troy ounces.

(8) Rounding may result in apparent summation differences between tonnes, grade, and contained metal content.

15.4 Factors that May Affect the Mineral Reserve Estimate

Factors which may affect the Mineral Reserve estimates include:

- Changes to the Mineral Resource estimate
- Gold price
- Metallurgical recoveries
- Pit slope angles and geotechnical characteristics of the rock mass
- Assumptions used to determine costs
- Assumed mining dilution and mine loss
- Ability of the mining operation to meet the planned annual throughput rate assumptions for the process plant
- Effectiveness of surface and ground water management, and timing and ability to obtain required permits.

16.0 MINING METHODS

16.1 Throughput Considerations

Based on previous throughput rationalization studies, the mining rate and production schedule must be able to consistently deliver the process rate of 53,500 t/d and meet the feed requirements and constraints for the concentrator and the POX circuit.

16.2 Geotechnical Considerations

BGC (2023c) provided feasibility-level slope criteria for the proposed open pit. The design incorporates a database of subsurface investigation. The state of Alaska does not have specific regulatory requirements for pit slope design. This review is based on accepted industry practice described in Read and Stacey (2009). The pit slope criteria integrates models for geology, structure, rock mass, alteration, hydrogeology, and geotechnical data into a design framework to establish slope configurations that meet acceptance criteria for safety and operational requirements.

16.2.1 Site Investigations

Site investigations have been conducted from 2004 to present and are summarized on Table 16-1. The database used by BGC (2023c) consists of 52 geotechnical drill holes and 233 exploration drill holes completed for Mineral Resource definition.

Table 16-1: Summary of Site Investigations

Type	Description
Drilling Programs	Diamond core drilling and geotechnical logging
Geologic Mapping	Lithologies, faults, joints, bedding planes
Laboratory Testing	Shear strength, deformability, index properties
Hydrogeological Data	Pumping tests, piezometric monitoring
Seismic Hazard	Regional seismicity incorporated into stability

16.2.2 Model Development

BGC used or developed five integrated models that underpin the design:

- *Geologic Model*: Defines lithology and alteration zones (developed by Donlin Gold)
- *Structural Model*: Captures fault systems and bedding orientations
- *Rock Mass Model*: Classifies material using rock mass rating (RMR), Q-system, and Hoek-Brown parameters
- *Hydrogeologic Model*: Simulates groundwater flow and pore pressure
- *Geotechnical Model*: Provides engineering properties for slope stability analysis.

16.2.2.1 Geologic Model

A geological model was developed for pit slope design that is consistent with the geology described in Section 7.

The geologic units used in the geological model for pit slope designs are:

- Greywacke
- Siltstone
- Shale
- Mixed Sediments
- Intrusives
- Ash Beds
- Fault Zones.

16.2.2.2 Structural Model

Seven structural domains summarized in Table 16-2 are defined by major faults and bedding orientation, informing geotechnical unit assignment. Variations in bedding orientation and joint density informed sub-domain delineation. Bedding (the primary sedimentary layering of greywacke, siltstone, and shale) is the dominant structural fabric in the pit area. Bedding surfaces are laterally more continuous than other discontinuities, with an estimated mean persistence of approximately 26 m (versus approximately 5 m for joints and minor faults).

Table 16-2: Structural Domains – Donlin Gold Feasibility Pit Slope Design

Domain	Location	Bedding Orientation	Boundaries	Data Sources
Lewis (LWS)	Northeast pit	Moderately dips SW	South: Lewis Steep, Divide Fault	Televiewer, oriented core
Lewis Steep (LST)	Between LWS & Divide Fault	Steep to moderate dip SW	North: LWS, South: Divide Fault	Televiewer, oriented core, surface mapping (2 loc.)
North Limb (NL)	Between Divide Fault & North Dipping Syncline	Moderately dips SW	South: Synclinal fold hinge	Televiewer, limited surface mapping
North Dipping Syncline (ND)	Between NL & South Dipping	Shallow dip NE	South: Transition to south dipping bedding	Televiewer, limited oriented core
South Dipping (SD)	Between ND & Steep Limb	Moderately dips SW	South: Transition to steep/overturned bedding	Televiewer, oriented core
Steep Limb (STP)	Between SD & Runway Fault	Steeply dips, may be overturned	North: SD, South: Runway Fault	Televiewer, oriented core
Runway (RWY)	Southwest end of ACMA pit	Shallow to moderate dip SW, rotated east	NE boundary: Runway Fault	Oriented core, limited televiewer & surface mapping

16.2.2.3 Rock Mass and Geotechnical Properties

Strength parameters are derived from laboratory testing and field data. Laboratory testing included unconfined compressive strength tests, triaxial compressive tests, Brazilian tensile strength tests, direct shear tests and point load tests. Rock mass quality is classified using RMR76, Q-system, and Hoek–Marinos flysch criteria, with friction angles differentiated by lithology and discontinuity type. Fault zones and ash beds are assigned lower friction angles based on direct shear testing.

The rock mass model has been developed to a level consistent with feasibility study standards and associated cost estimates include reasonable contingency allowances.

One ash sample has been tested. Shale discontinuities with slickensides have been tested using triaxial compression.

16.2.2.4 Hydrogeologic Model

BGC (2023c) characterizes groundwater flow as fracture-dominated and mirroring topography. They indicate that some compartmentalization of groundwater flow is suggested by high hydraulic heads at elevated topography and find no strong evidence that major faults significantly control groundwater movement. Dewatering requirements and pore pressure control are modeled using MODFLOW-SURFACT, with seasonal fluctuations and compartmentalized fracture networks considered. Updated hydrogeologic models inform depressurization and drainage design.

16.2.2.5 Slope Stability Analysis

Slope stability analysis are as follows:

- *Kinematic Analysis*: Sectors are grouped by instability mechanisms (planar, wedge, toppling). Bedding dips into the pit are most susceptible to planar sliding.
- *Bench Scale Analysis*: Bench face angles and catch bench widths are recommended based on discontinuity characteristics and sector orientation.
- *Limit Equilibrium Analysis*: Inter-ramp and overall slope stability are evaluated using deterministic models, incorporating isotropic and anisotropic properties, fault structures, and pore pressure scenarios. Depressurization is required for bedding-parallel and fault-adjacent sectors.

Design Acceptance Criteria (DAC) define the minimum safety and reliability standards that pit slope designs must meet to be considered acceptable for mine planning and operation. These criteria are aligned with industry standards. They are applied at different scales and depend on the consequence of failure for each pit wall.

16.2.2.6 Pit Slope Design Criteria

Based on the results of the pit slope analyses, the following criteria were developed as input to develop the mine plan.

- Inter-ramp angles range from 47–54° (fresh bedrock), 43–47° (weathered), and 27–39° (fault-adjacent).
- Bench height: 6 and 12 m
- Height between catch benches: 12 or 24 m
- Berm width: 8–38 m

- Road width allowance: 40–45 m
- Maximum inter-ramp height: 120 m
- Minimum geotechnical berm width: 20 m (wider if equipment access is needed)
- Controlled blasting and in some cases depressurization systems are required to maintain slope stability.

16.3 Pit Phases

The ACMA ultimate pit has been divided into seven phases based on optimized nested pit shell guidance, gold grade, strip ratio, the ability to access the pit, and locations for waste backfill. The planned ACMA pit has a top elevation of 232 masl and a bottom elevation of 368 m below sea level (bsl). The seven phases are delineated in Figure 15-1.

The Lewis pit will be on a hill directly above and to the northeast of the ACMA pit, at an elevation ranging from 424 masl to 80 mbsl. The Lewis ultimate pit has been divided into four phases based on optimized pit shell guidance, gold grade, and ramp access. The four phases are delineated in Figure 16-2.

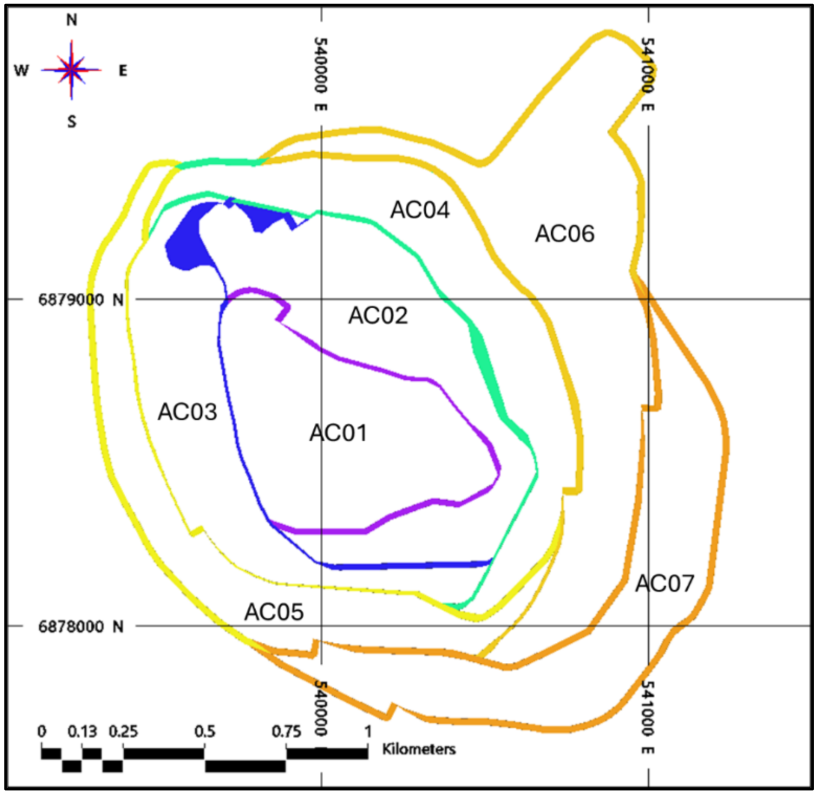
16.4 Pit Design Parameters

The general design parameters used in the detailed pit design are summarized in Table 16-3.

Haul roads are required between the pit phases and the ore crusher, WRFs, overburden stockpiles, construction areas, and truckshop. The ex-pit roads have generally been laid out with a cut-and-fill balance. Roads within the ultimate WRF are all fill construction.

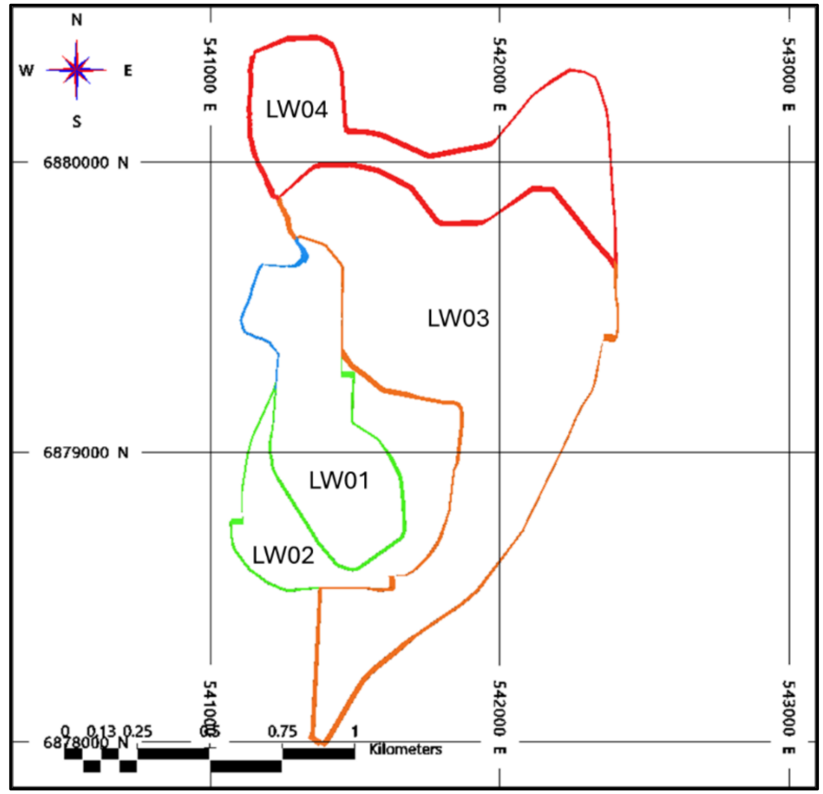
The initial phases of the two pits are independent, but they partially merge later in the mine life. The final overall pit layout plan is included as Figure 16-3.

Figure 16-1: ACMA Phases in Plan at 100 m Elevation



Source: Wood, 2025

Figure 16-2: Lewis Phases in Plan at 196 m Elevation

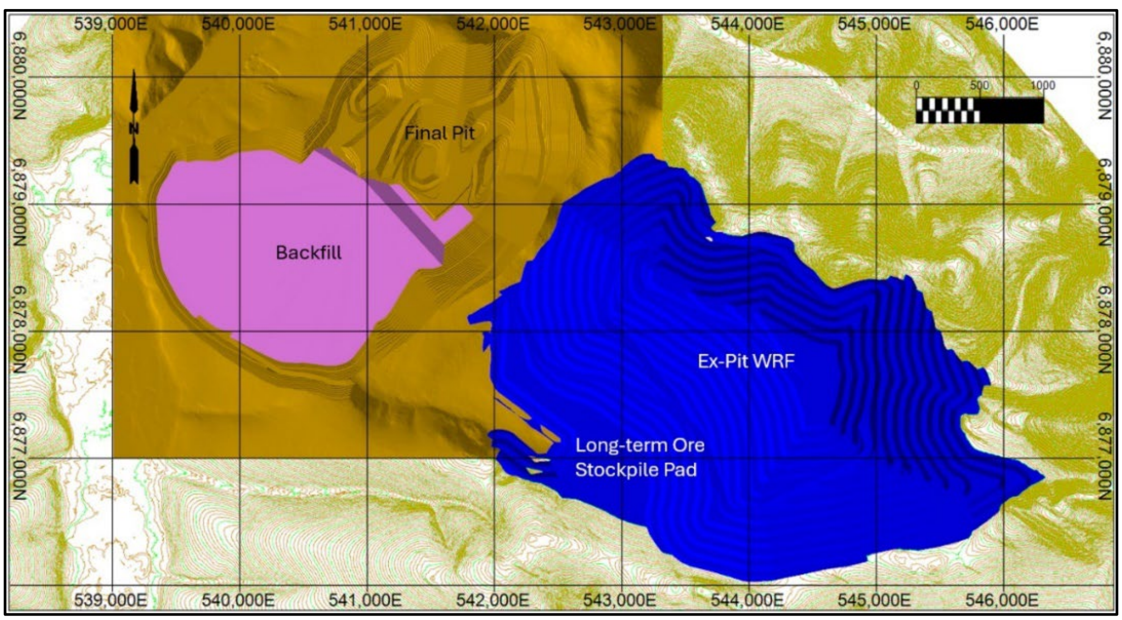


Source: Wood, 2025

Table 16-3: Pit Design Parameters

Design Parameter	Unit	Value
Bench height, single-bench mining	m	6 and 12
Height between catch benches	m	12 or 24
Bench face angle (variable)	degrees	43–65
Berm width (variable)	m	8–38
Total width allowance, final roads	m	40
Running surface on final two-way roads	m	29
Minimum road inside radius on corners	m	20
Berms and ditches	m	9–11
Maximum elevation between geotechnical berms	m	120
Maximum grade uphill loaded	%	10
Maximum grade downhill loaded	%	8

Figure 16-3: End-of-Mine Plan Layout of Open Pit



Source: Wood, 2025

16.5 Production Schedule

16.5.1 Planned Production Schedule

Pre-production mining has been defined as starting in July of Year -1 and finishing in June of Year 1, when the plant is commissioned. Process production starts in July of Year 1. The operating mine life is estimated at 24 years followed by three years of stockpile processing based on a nominal processing rate of 53,500 t/d.

Given the geometry of the orebody and the distribution of the ore, almost all of the waste material, and those ore zones that can be mined without significant loss or dilution, will be mined on 12 m benches. There are select ore zones that are planned for mining on 6 m benches.

The long-term stockpile will hold all ore produced at the mine in excess of process feed, separated into three sections according to sulfur grade for blending purposes, as follows:

High S grade (HS) ....S ≥1.4%

Medium S grade (MS) ....S ≥0.9% and S <1.4%

Low S grade (LS)...S <0.9%

The short-term stockpile was established to accommodate fluctuations in the average daily process feed.

16.5.2 Pit-Phase Mining Rates

The key features of the planned mine schedule are as follows:

- The mine will operate 355 d/a, with 10 days allowed for delays due to winter conditions.
- The plant is scheduled to operate 365 d/a.
- The average productivity of the main loading units (hydraulic shovel) is 4,500 t/h.
- Pre-stripping in Year-1 to Q2 Year 1 from inside the pits totals 64.7 Mt.
- The mine can sustain maximum material movement of 550,000 t/d based on 355 d/a.
- ACMA phase 7 is mined out in Year 18 providing most of the space for the in-pit WRF.
- Pit floor elevations (and therefore vertical advance rates) show the rate at which the mine must be dewatered to allow pit development. Vertical advance rates were limited to 10 benches per year.
- The initial phases are generally mined at a lower rate with the intent of keeping as many alternative areas open as possible in each phase.

- In the production years, three to four phases will be active in any given period, with three to four active phases per year from Years 2 to 16. This is driven by the requirements for ore blending and to make ACMA available for the in-pit WRF.
- The maximum mining rate of 545,000 t/d is achieved in Years 18 and 19 when in-pit dumping becomes available. The average mining rate increases progressively 296,000 t/d in Year 1, 370,000 t/d in Year 2, to an average of 450,000 t/d from Years 3-17. The mining rate decreases gradually from Year 20 to end of Year 24 when it completes mining at 320,000 t/d.
- The phase mining rate peaks at approximately 530,000 t/d in Lewis phase 3 in Year 19 with the use of in-pit dumping in the mined out ACMA pit, and at 310,000 t/d in ACMA phase 6 in Year 11. The permanent double-access strategy and wider phases allow for high mining rates. During initial production, the peak phase mining rate peaks at 232,000 t/d in ACMA Phase 2 in Year 2.

Table 16-4 summarizes the proposed LOM production schedule. Figure 16-4 graphically summarizes the planned mining rate per pit phase and Figure 16-5 provides the breakdown between ore and waste on an annual basis.

16.5.3 Mine Production Plan

To define areas to be mined as selective 6 m benches, an evaluation was conducted on polygons within ACMA pit phases. The polygons were created to compare the economics of bulk vs selective mining, and the mining cost of 6 m benches was compared against the increased dilution cost in processing. Where the benefit of 6 m benches was >5% compared to bulk mining, selective mining was utilized.

The mine plan, in addition to ensuring sustainable ore production, also considers consistent use of the selected material movement fleet, minimizing peaks in equipment acquisition requirements throughout the life of mine.

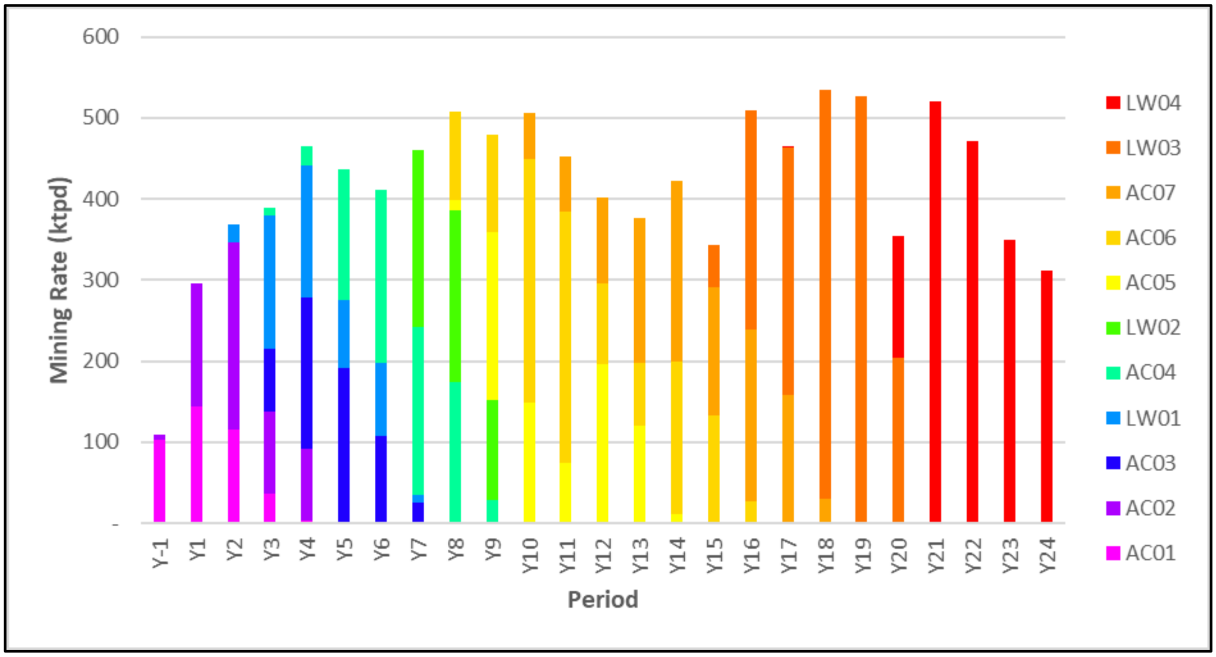
Ore is scheduled for stockpiling or direct feed and the combined mill feed is outlined in Table 16-5.

Table 16-4: Mine Production Summary by Year

Period	Mined for Mill Feed Tonnes (kt)	Au (g/t)	S (%)	Waste NAG (kt)	Waste PAG 5 (kt)	Waste PAG 6 (kt)	Waste PAG 7 (kt)	Waste OVB (kt)	Total Rock (kt)
Y-1	1,215	1.89	0.81	12,691	775	1,732	103	3,484	**20,000**
Y1	15,474	2.17	0.94	72,969	2,832	4,655	41	11,904	**107,874**
Y2	23,654	2.20	1.03	102,003	4,117	4,547	5	121	**134,446**
Y3	27,767	2.09	1.05	102,915	5,017	4,916	20	1,534	**142,169**
Y4	35,136	2.01	1.06	124,428	4,912	5,151	7	367	**170,000**
Y5	30,999	1.92	0.97	113,807	4,847	5,287	18	4,661	**159,619**
Y6	37,249	2.19	1.04	105,476	3,640	3,384	-	251	**150,000**
Y7	27,150	2.10	1.07	132,740	3,834	3,731	7	694	**168,155**
Y8	34,050	1.95	1.13	138,717	5,267	6,206	2	1,728	**185,969**
Y9	21,665	2.30	1.15	143,116	2,897	2,563	9	4,749	**175,000**
Y10	11,152	1.79	0.92	164,034	2,504	3,808	0	3,059	**184,558**
Y11	16,503	1.77	1.00	141,704	3,014	3,339	-	440	**165,000**
Y12	14,247	1.86	0.95	127,659	2,626	2,506	18	129	**147,185**
Y13	13,387	2.20	1.00	119,926	1,839	2,004	73	-	**137,230**
Y14	10,632	1.91	1.01	137,628	3,139	2,547	8	-	**153,953**
Y15	13,669	2.13	0.96	107,902	1,762	1,751	-	468	**125,552**
Y16	10,939	2.17	0.98	170,352	2,090	1,974	-	798	**186,153**
Y17	15,197	2.14	1.11	149,841	2,203	2,325	-	434	**170,000**
Y18	20,902	2.08	1.17	166,274	3,264	3,532	3	1,025	**195,000**
Y19	39,479	1.86	1.24	145,866	3,731	3,094	6	3	**192,178**
Y20	20,573	2.05	1.39	105,740	1,535	1,390	-	165	**129,404**
Y21	8,394	1.65	1.15	173,954	2,566	4,328	578	180	**190,000**
Y22	16,744	1.95	1.16	148,548	3,382	3,446	162	-	**172,282**
Y23	19,324	1.94	1.16	102,601	3,342	2,322	-	-	**127,590**
Y24	20,208	1.81	1.20	90,292	2,432	1,239	-	-	**114,171**
Total/Average	**505,708**[1]	**2.02**	**1.09**	**3,101,183**	**77,567**	**81,777**	**1,062**	**36,194**	**3,803,490**

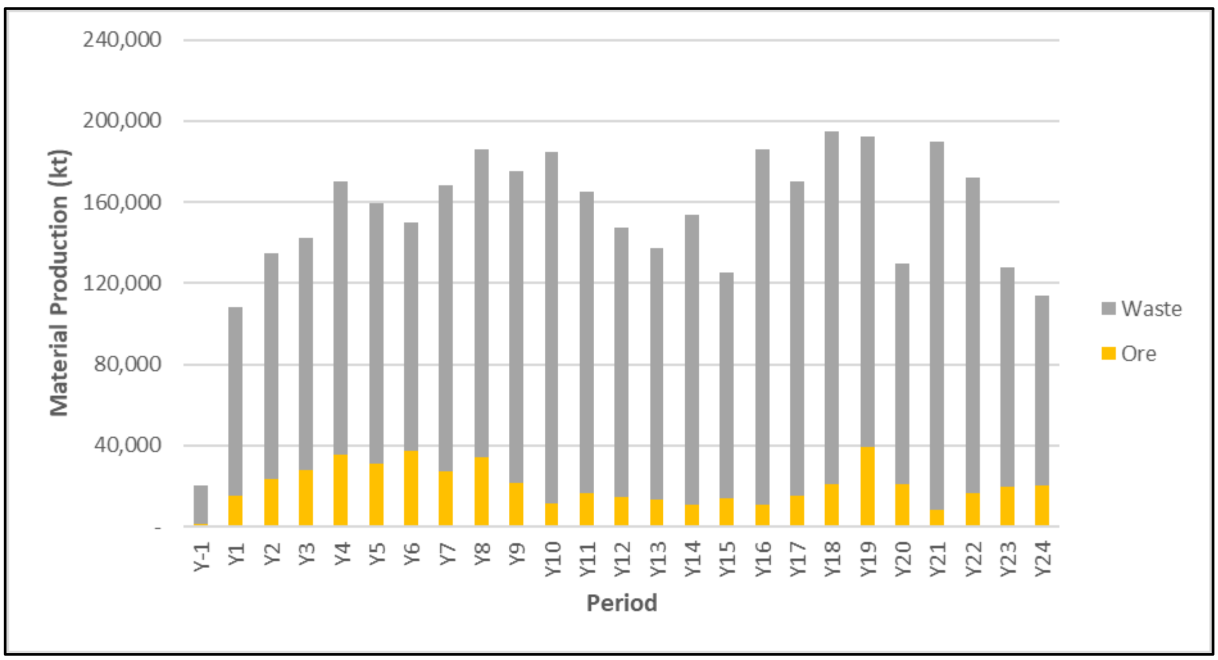
Note: (1) Includes 897 kt material that remains in the stockpile (not processed)

Figure 16-4: Mining Rate per Pit Phase



Source: Wood, 2025

Figure 16-5: Mine Production Schedule



Source: Wood, 2025

Table 16-5: Mill Feed Schedule

	Direct Feed				Stockpile Reclaim Feed				Total Feed			
Period	Tonnes (kt)	Au (g/t)	S (%)	Au Recovery (%)	Tonnes (kt)	Au (g/t)	S (%)	Au Recovery (%)	Tonnes (kt)	Au (g/t)	S (%)	Au Recovery (%)
Y-1	-	-	-	-	-	-	-	-	-	-	-	-
Y1	3,977	2.32	0.96	89.43	3,562	2.68	1.06	86.97	7,539	2.49	1.01	88.18
Y2	15,452	2.51	1.09	91.89	4,075	2.36	1.06	90.18	19,528	2.48	1.09	91.55
Y3	10,557	2.60	1.13	87.85	8,972	2.25	0.95	89.07	19,529	2.44	1.05	88.37
Y4	11,225	2.72	1.18	88.59	8,356	1.85	0.96	88.36	19,581	2.35	1.08	88.51
Y5	16,576	2.30	1.01	88.37	2,951	2.38	1.11	89.37	19,528	2.31	1.03	88.52
Y6	11,787	3.08	1.25	91.28	7,741	2.12	1.06	88.84	19,527	2.70	1.18	90.52
Y7	13,443	2.61	1.16	89.70	6,085	1.94	1.05	89.94	19,528	2.40	1.12	89.76
Y8	11,081	2.64	1.33	90.38	8,500	2.09	1.03	90.66	19,581	2.40	1.20	90.49
Y9	11,500	2.98	1.24	90.32	8,027	2.18	1.17	91.03	19,528	2.65	1.21	90.56
Y10	11,028	1.80	0.92	89.68	8,500	2.27	1.08	89.78	19,528	2.00	0.99	89.73
Y11	11,028	2.12	1.06	91.66	8,500	2.08	0.93	89.46	19,528	2.10	1.01	90.71
Y12	11,081	2.12	0.98	92.37	8,500	1.89	0.99	90.36	19,581	2.02	0.99	91.55
Y13	11,028	2.45	1.04	91.69	8,500	1.65	0.92	90.31	19,528	2.10	0.99	91.22
Y14	10,528	1.92	1.01	91.15	9,000	1.66	0.89	88.87	19,528	1.80	0.96	90.18
Y15	12,363	2.27	0.98	91.59	7,164	1.27	0.91	88.85	19,528	1.90	0.95	90.92
Y16	9,581	2.35	0.99	89.36	10,000	1.02	0.94	89.40	19,581	1.67	0.97	89.37
Y17	12,745	2.38	1.13	89.85	6,782	0.99	0.88	89.14	19,528	1.90	1.04	89.72
Y18	16,696	2.38	1.20	90.38	2,831	0.99	0.87	88.92	19,528	2.18	1.15	90.28
Y19	19,528	2.45	1.44	89.69	-	-	-	-	19,528	2.45	1.44	89.69
Y20	17,032	2.27	1.41	89.89	2,549	1.74	0.98	90.93	19,581	2.20	1.35	90.00
Y21	7,528	1.68	1.15	88.03	12,000	1.00	0.95	90.38	19,528	1.26	1.03	89.17
Y22	15,362	2.04	1.19	90.31	4,165	1.40	1.13	89.31	19,528	1.90	1.17	90.15
Y23	16,742	2.08	1.19	90.24	2,785	1.54	1.00	90.55	19,527	2.00	1.16	90.27
Y24	17,891	1.92	1.22	90.57	1,690	1.64	0.90	91.38	19,581	1.90	1.19	90.63
Y25	-	-	-	-	19,527	0.90	1.00	89.76	19,527	0.90	1.00	89.76
Y26	-	-	-	-	19,527	0.86	0.97	90.07	19,527	0.86	0.97	90.07
Y27	-	-	-	-	8,763	0.84	0.96	88.76	8,763	0.84	0.96	88.76
Total/Average	**305,758**	**2.33**	**1.16**	**90.21**	**199,053**	**1.54**	**0.99**	**89.63**	**504,811**	**2.02**	**1.09**	**90.04**

16.5.4 Process Feed Plan

The process feed is based on meeting the minimum sulfur in the concentrate feed to the POX plant through a blending strategy that combines ore feed directly from the mine with ore from stockpiles. The variable metallurgical recovery by domain was considered in creating the feed plan.

After plant ramp-up, process feed averages 53,500 t/d. Contained gold in the process feed averages approximately 1.21 Moz/a over the LOM, while contained gold in the process feed averages 1.55 Moz/a from Year 2 to Year 9, with a maximum of 1.70 Moz in Year 6.

Mill recovered gold averages approximately 1.09Moz/a, while the recovered gold averages 1.39 Moz/a from Year 2 to Year 9, with a maximum of 1.53 Moz in Year 6.

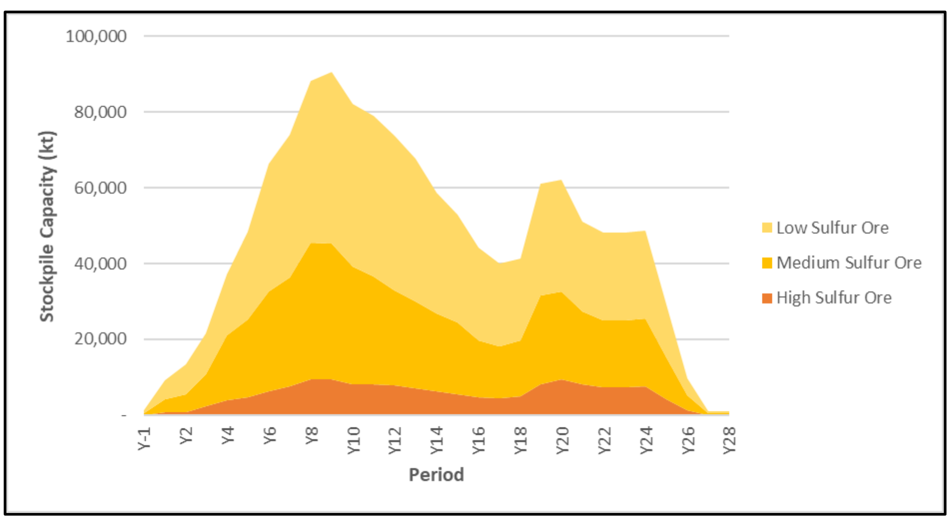
A small portion of the mine stockpile (897 kt) will not be processed due to capacity limitations of the TSF facility.

16.6 Ore Stockpiles

The maximum long-term stockpile capacity is 90.5 Mt at the end of Year 9. This includes 9.3 Mt of high sulfur grade material, 35.9 Mt of medium sulfur grade material, and 45.3 Mt of low sulfur grade material.

Planned relative sulfur grades in the ore stockpile are shown in Figure 16-6.

Figure 16-6: Relative Sulfur Grades in Ore Stockpile

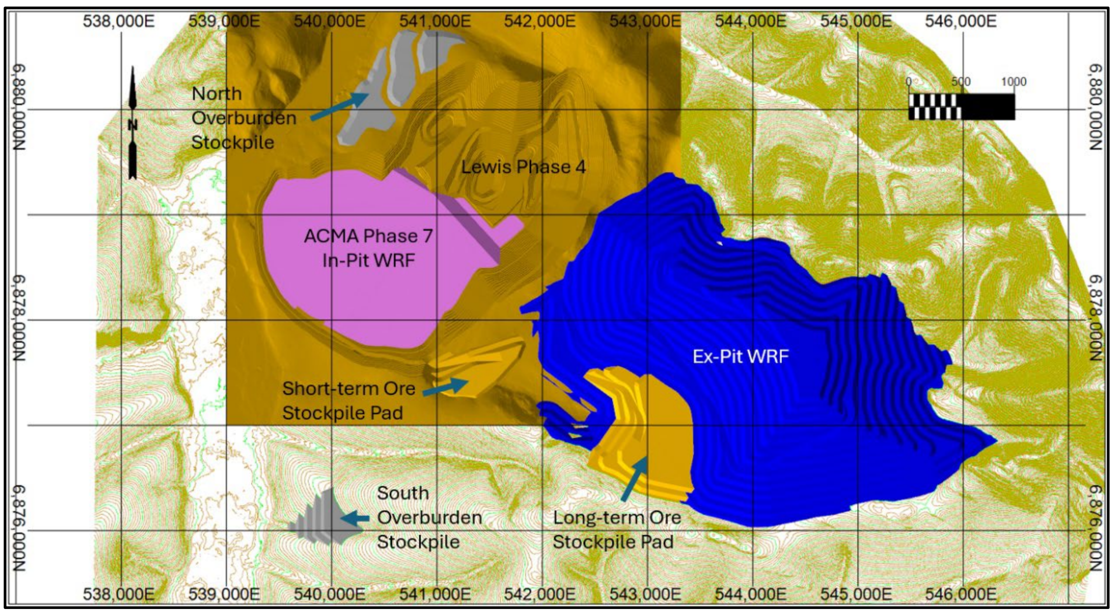


Source: Wood, 2025

16.7 Waste Rock Facilities

Waste material will consist of overburden, NAG rock, and PAG rock. The waste will be stored in several WRFs (Figure 16-7). The initial stripped materials will be stockpiled for construction and reclamation purposes or placed into the WRF. NAG and most PAG rock from the ACMA and Lewis pits will be routed to the ex-pit WRF during the first part of the mine life and to the in-pit WRF once mining the ACMA pit is completed. In certain periods, suitable material will be sent to the tailings dam location where it will be used for dam lift construction. Table 16-6 summarizes the capacities of the various WRFs.

Figure 16-7: Waste Rock Storage Locations



Source: Wood, 2025

Table 16-6: Waste Stockpiling Capacity and Requirements

Item	Waste (Mt)
Ex-Pit WRF Capacity	2,458
In-Pit WRF Capacity	940
Overburden Stockpiles	17
Construction Material Requirement	91
Maximum Waste Disposal Capacity	**3,506**
Total Waste Mined	**3,298**

Rock type influences ARD and metal leaching (ML), but sulfide mineralization introduces variability, making segregation based on geochemical characteristics feasible. SRK (2016) developed a site-specific classification system for waste rock based on $NPCO_3$/AP ratios, which correlate strongly with acid generation potential and metal leaching behavior. For operational simplicity, $NPCO_3$/AP and the ratio of arsenic/sulfur (As/S) were selected as the primary criteria for classification, resulting in four categories (see Section 14): NAG (1–4), PAG 5, PAG 6, and PAG 7, corresponding to increasing ARD risk and shorter onset times.

The mine plan anticipates generating approximately 3.3 billion tonnes of waste rock over the life of the Project. Most of this material (~ 95%) will consist of NAG waste rock and overburden (NAG 1–4 and OVB). PAG waste rock represents a small fraction of the total. PAG 5 extraction is relatively consistent throughout the mine life and never exceeds 5% of annual waste rock production, making blending or encapsulation with NAG material a viable strategy to prevent ARD.

PAG 6, which can become acid-generating within a decade, will be segregated in isolated cells at Rob's Gulch and Unnamed Gulch. A NAG foundation will be placed beneath PAG 6 to limit water contact and act as a drainage layer. PAG 6 will be placed in lifts up to 30 m high and capped with low-permeability material (e.g., terrace gravel) to minimize infiltration. NAG 1–4 will surround and cover PAG 6 cells for further isolation. Initially, PAG 6 will go into these cells; later, it will be placed in the ACMA pit backfill once available.

PAG 7, highly mineralized and most reactive, will be temporarily stored in a low-grade stockpile near the WRF toe. After ACMA pit limits are reached, PAG 7 will be relocated to the pit bottom.

16.8 Water Management

Water management will play an important role in mine development. The ACMA pit area is bisected by American Creek, and the west wall of the ultimate pit will be close to Crooked Creek. Surface ditches, a contact water pond immediately upstream of the pit and downstream of the WRF, plus diversion systems further upstream, will control surface waters in the pit and WRF areas.

Mine pit dewatering system will consist a combination of depressurization techniques, including vertical perimeter wells, vertical in-pit wells, and horizontal drains drilled into bench faces, to manage groundwater inflows. Tests indicate vertical wells are effective for dewatering most of the bedrock, but horizontal drains will be necessary to handle localized groundwater and deeper pit sections where bedrock hydraulic conductivity is lower, making vertical wells less effective.

Pre-stripping begins in ACMA 12 months prior to commencement of processing plant operation. Pre-stripping at Lewis begins in Q4 of YR2. The perimeter dewatering wells will begin pumping approximately six months before the start of pre-stripping so that pit depressurization targets can be achieved. The dewatering system includes up to 115 wells (35 perimeter and 80 in-pit), with depths averaging 215 m for perimeter wells and 188 m for in-pit wells. Horizontal drains totaling 268 km will assist in depressurizing pit slopes, lowering the water table to -335 masl by Year 20.

Mining operations are expected to lower groundwater levels near the open pit, resulting in the formation of a cone of depression extending outward from the pit and dewatering wells. This drawdown is anticipated to reduce groundwater discharge to Crooked Creek and its tributaries, potentially decreasing stream flow as the hydraulic gradient reverses and surface water from the creek begins to recharge the aquifer. Specifically, the segment of Crooked Creek adjacent to the pit is projected to transition from a gaining reach (where groundwater contributes to stream flow) to a losing reach (where stream water infiltrates into the aquifer) during active mining. However, operational water balance modeling indicates that, despite these changes, the overall flow in Crooked Creek will generally be maintained, as water management systems, including collection, treatment, and controlled discharge of mine water, are designed to offset potential reductions in natural base flow.

Once the mining activities are done, pit dewatering activities will end, which will allow groundwater levels to gradually recover. Over time, the open pit will refill with water, forming a pit lake, and groundwater and surface water interactions are expected to re-establish more natural flow conditions as the system equilibrates. The pit design assumes general depressurization zones to extend 60–100 m into pit slopes and targeted fault depressurization ranging from 100–300 m behind select pit faces (BGC, 2023c). Critical areas include bedding parallel sectors on the north wall of the Lewis pit and north/northeast walls of the ACMA pit, as well as major faults like Lo1 and Divide. Field verification with pore pressure monitoring and systematic flow monitoring for wells and drains will be required to confirm effectiveness of the dewatering system.

16.9 Blasting and Explosives

A blend of 70% emulsion phase and 30% ammonium nitrate/fuel oil (ANFO) will be used for blasting, based on anticipated groundwater conditions.

An explosives supplier will be contracted to provide a "down-the-hole" blasting service. The supplier will provide the ammonium nitrate, emulsion phase components, and blasting accessories. The supplier will also supply the emulsion plant, explosives magazines, mixing equipment, and delivery trucks. The Donlin operator will provide fuel oil and accommodation. Supplier personnel will charge the holes, place the detonators and boosters, and tie in the patterns.

16.10 Mining Equipment

To determine the number of equipment units required for each major fleet, productivities were calculated based on estimated annual operating hours, mechanical availability, and utilization of availability. Annual operating hours varied by fleet due to associated availabilities. A value of 50 net operating minutes per gross operating hour (GOH) was applied to all equipment to account for time spent on non-primary production tasks. The vendor-estimated mechanical availability of the equipment decreases with hours worked. An average mechanical availability based on the life of the fleet was assigned to replicate the availability for a fleet containing units of mixed ages. Table 16-7 provides equipment selection for maximum mining equipment fleet for pre-production and peak production years.

The mining department will use GPS machine guidance and a fleet management system to guide and control the mining operation on a near real-time basis. Fleet management is the assignment of equipment to mining tasks, while GPS machine guidance assists the operator with respect to spatial positioning of the ground-engaging tools (GETs).

Table 16-7: Mine Equipment Requirements

Equipment Unit	Pre-Production (#)	Production (#)
Shovel Electric (38 m^3)	1	5
Shovel Diesel (38 m^3)	1	1
Front-End Loader (40 m^3)	0	2
Front-End Loader (19 m^3)	1	3
Haul Truck (360 t)	7	90
Haul Truck (135 t)	2	13
Rotary Drill (250 mm)	2	7
Rotary Drill (200 mm)	2	9
Top Hammer Drill (140 mm)	3	4
Track Dozer (850 hp)	4	7
Track Dozer (580 hp)	3	4
Wheel Dozer (800 hp)	4	7
Grader (533 hp)	1	2
Grader (297 hp)	3	7
Water Truck	1	3
Excavator (5 m^3)	1	2
Excavator (12 m^3)	1	2

16.10.1 Drilling

Donlin will undertake five different types of drilling:

- RC drilling to provide samples for ore control and geological modeling
- Blast pattern drilling to fragment the rock for mining
- Frost drilling to deal with previously blasted material that has become frozen
- Horizontal drain hole drilling to prevent water pressure from building up behind the pit walls. The horizontal drain hole drilling is a specialized activity and will be performed by a contractor.
- Vertical dewatering wells. The vertical dewatering well development is a specialized activity and will be performed by a contractor.

Three different types of drill will be used: a rotary drill for bulk waste with 251 mm diameter holes for 12 m benches; a rotary drill with 200 mm diameter holes for ore and waste in 12 m benches; and a hammer drill with 140 mm diameter holes for ore and waste in the 6 m benches and for pre-split and RC drilling.

16.10.2 Loading

Primary loading for bulk mining areas will be performed by electric-hydraulic shovels with a 37 m^3 bucket. One 40 m^3 front-end loader (FEL) will be used for secondary production, and another for backup production, cleanup, and stockpile rehandling.

16.10.3 Hauling

Large 363 t payload haul trucks will be used for primary mine production. A maximum of ninety 363 t payload trucks is required.

16.10.4 Secondary Fleet

A second fleet of mining equipment will be required mainly for selective mining and stockpile rehandling. The fleet may be used for overburden management, concurrent reclamation, snow removal, road maintenance, and special projects. This fleet will consist of smaller, more agile 18.1 m^3 FELs and 135 t trucks and will allow the primary fleet to focus on production ore and waste mining.

16.10.5 Support Equipment

The major tasks to be completed by the support equipment include the following:

- Bench and road maintenance
- Reclamation support
- Stockpile construction
- General maintenance
- Ditch preparation and maintenance
- Tailings dam support
- Shovel support/cleanup.

Additional auxiliary equipment will serve and support the mine operations and maintenance groups. Equipment selection for auxiliary equipment is shown by function and is included in Table 16-8.

Table 16-8: Mine Auxiliary Equipment

Equipment Unit	Number (#)
Horizontal Drain Hole Drill	1
Excavator with Hydraulic Hammer	1
200 t Class Crane	1
150 t Class Crane	1
60 t Class Crane	1
23.5 t Forklift	1
16.3 t Forklift	1
4.5 t Telehandler	1
4.5 t Forklift	1
Fuel/Lube Truck	3
Medium Mech Truck	4
Large Mech Truck	1
Tire Handler	2
Lift Truck	1
Loader	1
Lowboy / Tow	1
Operations Field Truck	1
Soil Compactor	1
Backhoe Loader	1
Light Plant	20
Skid Steer Trailer	3
Light Vehicle	35
Crew Bus	5
Cable Reeler	1
Shovel Motivator	1

16.11 Work Schedule

The work schedule assumes mine production will operate 24 h/d, 7 d/wk, 365 d/a. Operations personnel will work on two 12 h/d shifts.

Delays are included in operator time to account for blasting, fueling, pre-shift meetings, shift change, and an additional 10 d/a of delays related to weather.

Hourly (non-exempt) personnel assigned to the mine will work a 2-weeks-in/1-week-out rotation.

Salaried (exempt) personnel will work a 12 h/d shift on either an 8-days-in/ 6-days-out rotation, a 2-weeks-in/ 1-week-out rotation, or a 2-weeks-in/2-weeks-out rotation.

16.11.1 Manpower Requirements

Workforce estimates are based on mine schedules with the equipment fleets selected for this project. Personnel categories include salaried staff (management and technical roles), hourly operators for drilling, loading and hauling, support services, and maintenance employees. For the first two years, all mobile fleets will be maintained by contractors under Maintenance and Repair Contract (MARC) agreements. After two years, the operation will assume more risk but will reduce costs by phasing out the MARC contracts and taking responsibility for all mobile equipment maintenance.

Peak workforce requirements are summarized in Table 16-9.

Table 16-9: Manpower Requirement Summary

Role	Pre-Production (#)	Production (#)
Salaried Employees	57	105
Drill Operators	24	39
Loading Operators	12	33
Hauling Operators	32	339
Support Services	37	87
Maintenance	0	244
Total Hourly Personnel	105	742
Total Mine Employees	162	847

17.0 RECOVERY METHODS

17.1 Plant Design

The process design is supported by the testwork and results presented in Section 13. The plant description and design basis summarized in Table 17-1 indicate conditions of a plant operating at full capacity. They are appropriate for plant design, but they do not consider variation introduced during facility start-up or tapering-off of production near the end of mine life.

The process design summarized in this Report is unchanged from the Donlin 2021 Technical Report.

Table 17-1: Process Design Basis

Parameter	Unit	Value
Annual Throughput	t/a	19,527,500
Average Daily Throughput[1]	t/d	53,500
Overall Plant Availability	%	93
Average Nominal Feed Grade		
Au	g/t	2.02
As	ppm	2.3
S (total)	%	1.03
Gold Recovery		
Over LOM	%	90.0
Average Gold per Year[2]	oz	1,141,400

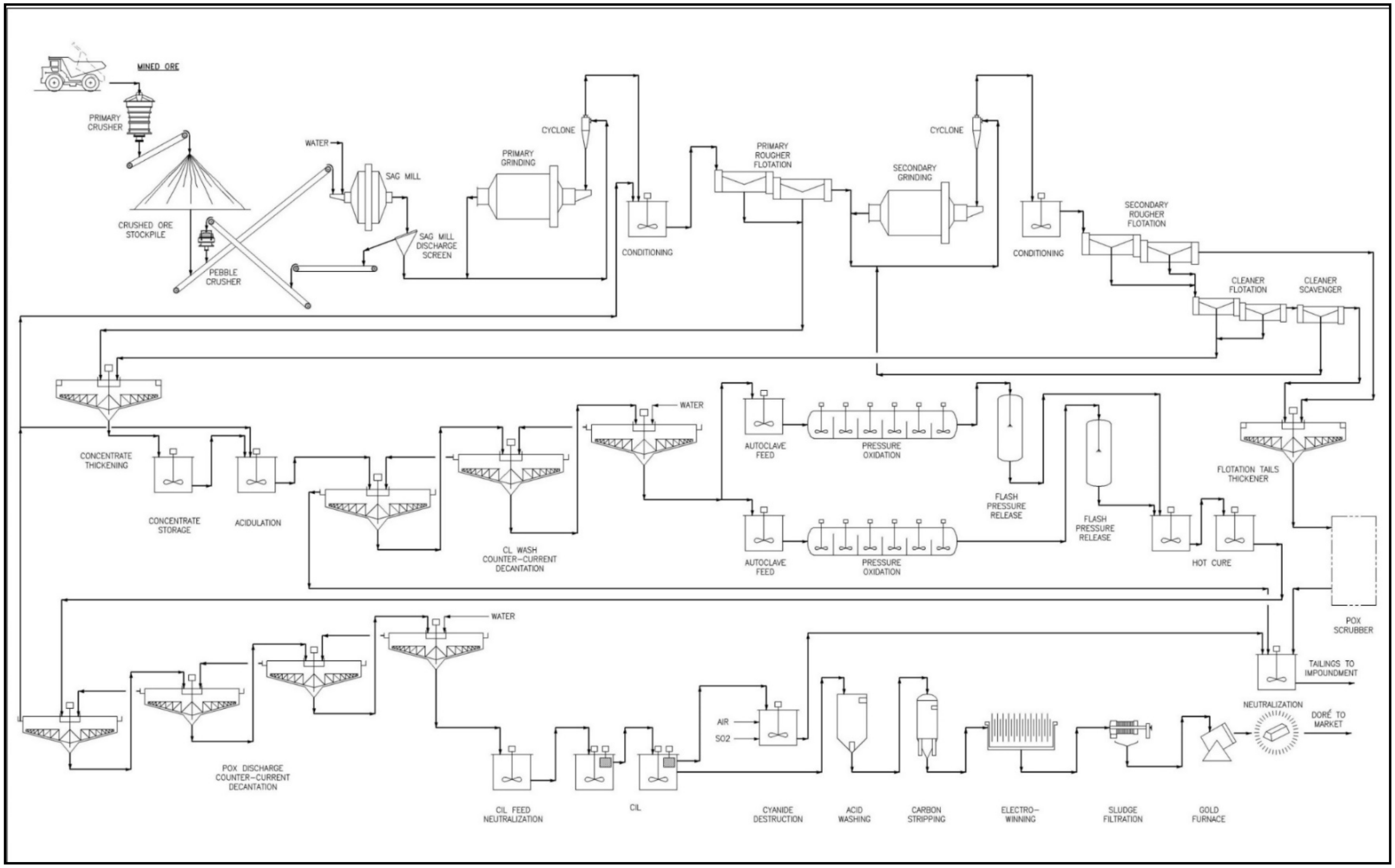
Note: (1) 93% plant availability has been applied to the average daily throughput
(2) Based on Annual throughput, average grade and recovery.

17.1.1 General

The process is based on conventional and proven technology for the concentrator, flotation, POX, and cyanidation facilities for large, modern gold processing plants. A simplified flow diagram of the overall process is shown in Figure 17-1.

All process equipment, except for thickening, neutralization tanks, and concentrate storage tanks, will be enclosed in buildings. Installed standby pump spares are provided for all critical process streams.

Figure 17-1: Donlin Gold Simplified Flow Diagram



Source: Wood, 2025

17.1.2 Crushing and Coarse Ore Stockpile

Mine haul trucks (with capacities to 363 t) will dump ROM open-pit ore directly into a dump hopper ahead of a 60" x 89" gyratory crusher. The maximum design crushing capacity of the crusher is 4,630 t/h producing a crusher product at P_{80} 125 mm. The crusher will discharge to a covered coarse ore stockpile with a live capacity of 38,000 t, representing 16 hours of process plant operation, and a total capacity of approximately 174,000 t.

The reclaim tunnel with feeders will be used to reclaim material from the stockpile, discharging onto the SAG mill feed conveyor. The steady-state SAG mill feed rate will be 2,397 t/h. SAG mill critical size material will report to the pebble crusher. The discharge from the pebble crusher will join the new feed from the coarse ore stockpile.

17.1.3 Grinding and Pebble Crushing

The overall grinding configuration will consist of an open-circuit SAG mill followed by the MCF2 circuit. The MCF2 circuit will entail a primary ball mill followed by primary rougher flotation; the tailings produced from primary flotation will be sent to a secondary ball mill, followed by a secondary rougher flotation. The two individual ball mills will operate in a closed circuit with their respective classification cyclones.

SAG mill discharge will be screened and oversized pebbles will be conveyed to two large pebble cone crushers. Crushed pebbles will normally be returned to the SAG mill. Typical SAG mill discharge will have a P_{80} of 1,700 µm. After primary ball mill grinding, the P_{80} is anticipated to be 121 µm; after secondary grinding, the P_{80} is anticipated to be 50 µm. The total system throughput is expected to average 53,500 t/d at 93% availability.

The SAG mill feed conveyor will discharge into the SAG mill feed chute and then into the SAG mill. Process solution will be added at this point to flush the ore into the mill and provide the correct dilution for grinding. Copper sulfate will be added to the feed end of the SAG mill to activate sulfide mineralization. The SAG mill discharge will be screened with undersize from the trommel screen and vibrating discharge screen prior to flowing into the primary cyclone feed pumpbox. The 11.6 m diameter X 7.6 m (effective grinding length (EGL)) long SAG mill will be powered by a 20 MW wrap-around variable-speed drive.

Optimum economics are expected to be achieved by using the fewest, largest equipment units available. The 7.9 m diameter x 13 m (EGL) long ball mills will be trunnion-supported units with 18 MW wrap-around drive configurations. Discharge from the primary ball mill will exit the discharge trunnion into a trommel screen attached to the ball mill. Oversize material will drop from the end of the trommel screen into a rejects hopper. Undersize material will pass through the trommel screen into the primary cyclone feed pumpbox along with the SAG mill screen underflow. The primary cyclone feed pump will be variable-speed and will transport slurry to the cyclone cluster, which will classify particles by size to return coarse particles to the ball mill for further size reduction.

The primary cyclone overflow will be designed to operate at 40% solids, with an anticipated average 80% passing particle size of 121 µm; the cyclone circulating load is estimated to be 210%. The fresh feed for the secondary ball mill will be a combination of the slurry from the rougher tailings pumpbox and the cleaner scavenger concentrate. These streams will flow into the secondary grinding cyclone feed pumpbox, where they will join the secondary ball mill discharge.

Discharge from the secondary ball mill will exit in the same manner as for the primary mill. Oversize material will be dropped from the end of the trommel screen into a rejects hopper. Undersize material will pass through the trommel screen and into the secondary cyclone feed pumpbox, together with the rougher tailings and the cleaner scavenger concentrate. The secondary cyclone feed pump (variable-speed) will transport slurry to the secondary cyclone clusters. The secondary cyclone overflow is anticipated to be 27.2% solids with an average 80% passing particle size of 50 µm; the cyclone circulating load is estimated at 210%.

17.1.4 Flotation

Donlin ore contains a mixture of intrusive and sedimentary rock hosted sulfide mineralization. The optimum gold recovery will be achieved by maximizing sulfide recovery. Producing a bulk flotation concentrate in two rougher flotation steps with relatively low selectivity and high mass pulls has been determined to provide the best results. The secondary rougher concentrate will be sent to a cleaner flotation circuit; the concentrate obtained from cleaner flotation will be combined with the primary rougher concentrate. The tails obtained from the cleaner flotation will be sent to a cleaner scavenger flotation train. The cleaner scavenger concentrate will be sent to the secondary grinding cyclone feed pumpbox, and the tails will be mixed with rougher tailings to flow by gravity to the flotation tailings thickener. The primary rougher concentrate and the concentrate produced from the cleaner flotation will report to the concentrate thickener.

Overflow slurry from the primary grinding circuit cyclone will be discharged over a flotation safety screen before entering conditioning tank No. 1. Oversized material will be sent to a hopper.

Acidic solution from the POX counter-current decantation (CCD) washing circuit, as well as flotation process water and copper sulfate, will be added to the first conditioning tank. The slurry will then pass to conditioning tank No. 2 where potassium amyl xanthate (PAX), dispersant

Cytec E-40, and methyl isobutyl carbinol (MIBC) frother will be added. The discharge from the second conditioning tank will be pumped to a distributor box, which will split the feed to two parallel trains of 300 m^3 primary rougher cells. The rougher flotation feed pump system will include an installed spare. Each train of cells will have 11 individual units and provide 57 minutes of residence time.

Primary rougher concentrate from the rougher flotation will be sent directly to the concentrate thickener. Primary rougher tailings will be sent to the secondary grinding cyclone feed pumpbox as part of the MCF2 circuit.

The secondary rougher flotation circuit will be fed from the secondary rougher conditioning tank where copper sulfate and MIBC will be added. The flow will be divided into two trains; each train will have 11 individual 300 m^3 secondary rougher cells providing a total residence time of 57 minutes. Additional MIBC, PAX, Cytec E-40, soda ash, and a second frother, F-549, will be added throughout the secondary rougher as long.

Secondary rougher concentrate will be sent to the cleaner flotation circuit. Secondary rougher tailings will be sent directly to the flotation tailings thickener. The secondary flotation concentrate will be cleaned in a bank of six 300 m^3 tank-type flotation cells, with a residence time of 100 minutes. The concentrate will be sent to the cleaner concentrate pumpbox, from where it will be pumped to the concentrate thickener. The cleaner tailings will flow by gravity to the cleaner scavenger flotation circuit.

The cleaner scavenger circuit will have four 300 m^3 cells with a combined residence time of 150 minutes. The cleaner scavenger concentrate will be sent to the secondary grinding cyclone feed pumpbox. The tailings will be sent to the flotation tails thickener by gravity. Flotation streams will be sampled automatically for metallurgical accounting and control purposes.

An on-stream x-ray fluorescent analyzer (analyzing Fe and As) will provide continuous data to enable operators and supervisory control systems to optimize flotation and to respond to upset conditions.

A slip stream from each of the two cyclone overflow streams will pass through a particle size analyzer to provide information for grinding control.

17.1.5 Thickening, Concentrate Storage, Acidulation, and CCD Washing

Flotation concentrate will discharge into a 3.5 x 5 m concentrate thickener de-aeration tank and then to a 45 m diameter concentrate thickener. Thickener overflow will be returned to the grinding and flotation areas as process water while underflow slurry at an estimated 46% solids will be pumped to the concentrate storage tanks. A total of 36 hours of concentrate storage will be provided. In normal operation material will be withdrawn from the storage tanks and sent to the acidulation circuit. A bypass line will be provided to pump slurry directly from thickening to acidulation. Acidic solution recovered from the POX CCD wash circuit will be mixed with the concentrate with the aim of consuming 85-100% of the carbonate gangue component of the concentrate.

The acidulated material will be washed with raw water in a three-stage chloride wash CCD circuit to reduce the overall levels of chlorides and other mineral species in solution reporting to the POX circuit.

17.1.6 Autoclave Plant

Acidulated feed slurry will be stored in an agitated autoclave feed storage tank adjacent to the POX area. This tank will provide the autoclave plant with a continuous feed unaffected by short-term upstream throughput variations. When full, it will allow the autoclave circuit to continue operating for four hours when upstream equipment is not operating. Slurry will be pumped with a slurry heater feed pump from the feed tank through two parallel lines to the heater vessels that will pre-heat the incoming slurry to varying temperatures, depending on the sulfide sulfur grade of the feed material. Pre-heat temperature will be optimized based on maintaining autogenous conditions in the autoclave while minimizing cooling water addition.

Slurry pre-heating will be accomplished with the use of flash steam produced in the pressure letdown flash vessels on the autoclave discharge. Each autoclave train will discharge to two flash vessels and two vent gas cyclones via parallel slurry discharge lines. Slurry heater discharge temperature control will be achieved by bypassing a portion of the feed to each heater to the heater sump. From each heater discharge, a single feed line will feed each autoclave, each with a charge pump to create sufficient pressure for the suction side of the autoclave slurry feed pump; a slurry strainer to remove any scale or oversize particles that could damage the autoclave slurry feed pumps; and a high-pressure piston diaphragm feed pump with suction accumulator and discharge dampener.

The autoclave will be supplied with high-pressure oxygen gas, high-pressure cooling water, and high-pressure steam. Oxygen will be produced at an on-site air separation plant. Cooling water will be distributed to the autoclave from a cooling water tank located in the POX area. High-pressure, horizontal multi-stage centrifugal pumps will supply the cooling water to all compartments of the autoclave. A common piping system (and spargers) will be utilized for both the oxygen and high-pressure steam to the autoclave.

High-pressure steam (produced from de-mineralized water) will not be required for normal operation, but is required for autoclave heat-up. De-mineralized water will also be used for the agitator seal water system and the oxygen plant boiler system. The autoclave building will be serviced by an overhead crane, allowing any of the agitators to be removed without need for disc disassembly (impeller blades require removal).

Each autoclave is approximately 5 m diameter x 33 m long and will discharge into two flash vessels in parallel. The flash vessels are approximately 5 m diameter x 5 m long. Autoclave discharge slurry will be depressurized to near-atmospheric pressure, generating flash steam in the process. Flash vessel underflows will be directed by gravity to an oxidized slurry seal tank. Slurry from this tank will be transferred by gravity to the downstream hot cure tanks. Steam generated in the hot cure tanks will be condensed using a spray tree condenser vessel. Vent-gas from the autoclave will be passed into the vent gas quench vessel. Flotation tails will be used as quenching medium, as the steam will be condensed across a baffle arrangement inside the vessel. The quench vessel will reduce the temperature of the vent-gas and the quantity of steam (through condensation) that will be fed to downstream equipment. The quench vessel will also facilitate additional removal of carryover from the autoclave and slurry heater, and pre-heat the flotation tails ahead of the downstream neutralization process, thereby improving neutralization reaction kinetics. Vent gas from the quench vessel will be piped to a secondary spray tree condenser vessel where raw water will further cool the gas and condense the steam. The gas will then pass through a venturi scrubber where the gas will be further cleaned of particulates by pressure drop and the addition of fresh water.

Gas will exit the venturi scrubber saturated with water vapor and at a temperature of 40°C. The process off-gas temperature will be reduced to a target of 4°C. The gas volume will also be reduced by water vapor condensation. The wet-gas condenser will promote the condensation of elemental mercury during periods of upset conditions with higher than normal gaseous mercury levels. The overall mercury loading on the downstream carbon adsorption process will be reduced as a result. Gas will enter a wet gas coalescer to remove any remaining entrained mercury. The gas will then be subjected to cyclonic separation and then enter a coalescer prior to the carbon pre-cleaning section.

The combined gas will enter a pre-cleaning carbon bed. The relatively inexpensive activated carbon contained in the pre-cleaning bed will be used to remove volatile organic compounds (VOCs). Exiting gases will immediately enter the mercury removal carbon bed, which will contain sulfur-impregnated carbon, specifically designed to adsorb vapor-phase mercury. Oxidized and particulate forms of mercury will also be collected. The carbon bed vessels are 4.5 m diameter x 2.5 m high. Cleaned gas will be discharged to atmosphere. The mercury-loaded carbon will be removed periodically in an environmentally safe manner and sent off site for disposal. A standby series of carbon beds will be available so that neither production nor mercury removal will be interrupted when a bed is taken offline for carbon change-out or maintenance.

A common mercury collection tank will receive all the condensate and scrubber water from the POX gas handling systems. The tank will be designed to settle mercury and separated solids removed from the gas, clarifying the water. The clarified water will be passed through coalescing filters to remove any remaining elemental mercury prior to being recycled to the chloride wash circuit. Coalesced elemental mercury will be returned to the mercury collection tank for settling. The resulting sludge will then be drained from the mercury collection tank as contaminated waste.

Arsenic in the processed ore is managed using the POX process. The ore has sufficient iron content to permit precipitation of arsenic with iron to form stable forms of arsenic precipitation products, suitable for long term storage in the TSF.

17.1.7 Counter Current Decantation Pressure Oxidization Thickening and Washing

Slurry flow from the POX circuit will be washed in a four-stage CCD circuit. The thickeners are 50 m in diameter. Reclaim water will be added to the last thickener in a flow direction counter to the solids in order to decrease the acidity of the pulp.

Washed slurry in the underflow from the final thickener will be pumped to the CIL solids neutralization circuit. Thickener overflow will be treated in a clarifier and used within the plant to provide acidification of the concentrate fed to the POX circuit, and to the flotation feed to assist in promotion of the sulfide mineral floatability. The balance of the clarified process water will report to neutralization. Clarifier sludge will be intermittently returned to the first thickener in the circuit.

17.1.8 Flotation Tailings Neutralization

In this circuit, flotation tailings will be pre-heated to 55°C through the autoclave quench vessel and then combined with the excess diluted acidic wash liquor from the chloride CCD wash circuit in a series of large aerated and agitated tanks. The flotation tailings will act as neutralizing material (source of natural carbonates) for reaction with the acidic liquor. All neutralization tanks will be insulated for heat conservation.

Flotation tailings will be collected, sampled, and passed to the flotation tailings thickener. Thickener overflow will be pumped to the flotation process water tank. Thickener underflow will be pumped to the POX circuit autoclave scrubber.

Acidic solution from the POX CCD wash and spent acid from the elution circuit will be combined with autoclave quench tails in the solution neutralization circuit. The circuit will consist of five mechanically-agitated and aerated tanks in series. Enough reaction time will be provided in the front-end portion of the circuit to bring the pH of the solution up to pH 5, utilizing the quench tails. Tailings from the cyanide destruction circuit will be introduced into a lime neutralization tank where lime will be added in the presence of air to bring the pH to 7. This material will then flow by gravity to the final tailings pumpbox. Owing to their size, the flotation tailings neutralization tanks will be installed outdoors.

17.1.9 Solids Carbon-in-Leach Neutralization

Underflow from the final POX CCD wash circuit thickener will be neutralized in the solids CIL neutralization circuit. The circuit will consist of two mechanically-agitated tanks where lime will be added to the slurry in the presence of oxygen to bring the pH of the slurry to approximately pH 9. This material will then be pumped to the CIL circuit.

17.1.10 Carbon-in-Leach Cyanidation Circuit

A nominal mass flowrate of 365 t/h at 35% solids will be pumped to the first of six CIL tanks in series, each with a residence time of four hours. The slurry will flow by gravity through each of the six tanks, ultimately reporting to the cyanide destruction reactor tank.

The slurry will flow by gravity from the CIL tank No. 6 to the safety screen feed distribution box. A distribution box will direct the slurry to either one or both of two carbon safety screens. The safety screens will prevent carbon that passes through the screen in the last CIL tank from leaving the circuit to the tailings storage facility. Screen undersize will flow by gravity to the cyanide destruction reactor tank and will then be pumped to flotation tailings neutralization tank No. 5 by centrifugal slurry pumps. Sodium cyanide solution will be pumped to the CIL circuit for cyanide leaching of gold. A lime loop will allow for lime addition to each of the six CIL tanks. The pH will be monitored, and lime added as needed to maintain a pH set point of approximately pH 11.

Oxygen required for cyanide leaching will be supplied as pure oxygen from the oxygen plant. The CIL tanks and cyanide destruction reactor tank will be covered to contain any hydrogen cyanide (HCN) gas that evolves during cyanide leaching. The tanks will be ventilated, and the gas is passed to an HCN scrubber caustic solution to recycle HCN.

A carbon concentration of 15 g/L will be maintained in each of the CIL tanks. Each tank will be equipped with a carbon retention screen. A vertical spindle pump with a recessed impeller will be installed in each tank to transfer the carbon upstream, counter-current to the flow of slurry, at a rate of 23 t/d. Regenerated carbon from the carbon transfer tank in the carbon stripping and regeneration circuit, will be transferred to CIL tank No. 6 to replace the carbon pumped upstream. Loaded carbon will be pumped from CIL tank No. 1 over the loaded carbon screen to retain and wash the carbon. The washed carbon will report to one of the two carbon acid-wash vessels.

17.1.11 Cyanide Destruction System

Slurry from CIL tank No. 6 will flow by gravity through the carbon safety screens, and the screen undersize will report to the cyanide destruction reactor tank. This will be a covered, agitated tank where the residual weakly acid-dissociable (WAD) cyanide concentration will be reduced from nominally 100 ppm to the cyanide levels required by permit. Air and SO_2 will combine to oxidize the cyanide to carbon dioxide and ammonia. Copper sulfate solution will be added to top of the tank and will serve as a reaction catalyst to maintain the reaction kinetics. Lime will be added as necessary to maintain an appropriate pH level to ensure adequate reaction kinetics. The destruction reactor tank will be sized for one hour of retention time.

17.1.12 Carbon Elution, Electrowinning, Reactivation, and Gold Refining

Loaded carbon, at a nominal gold loading of 4,800 g/t, will report to one of two 12 t capacity carbon acid-wash columns by gravity from the loaded carbon screen. The two acid wash columns will process 36 t/d of carbon. After the acid wash and neutralization processes are complete, the carbon will be transferred from the acid wash vessel to one of two elution columns. A carbon elution will begin as soon as the carbon is transferred to the elution column and the transport water has completely drained out of the vessel. For ease of operation, the two elution columns will be the same size as the two acid wash columns. Barren solution with NaOH and NaCN added to a concentration of 1% NaOH and 0.1% NaCN, respectively, will be pumped upflow through the bottom of the elution column. The pregnant solution will exit the elution column and flow through the heat exchanger before reporting to the pregnant solution tank. The barren solution will be circulated through the elution column for a nominal eight hours to complete each elution. When the strip is complete, one bed volume of raw water will be pumped through the elution column. This solution will rinse the residual solution from the carbon and cool the carbon in preparation for transfer.

After the carbon is rinsed, it will be transferred to the carbon dewatering screen before the kiln. Carbon will be processed through the kiln at the rate of 1.5 t/h for reactivation. The kiln will be sized to process 100% of the eluted carbon to maintain high carbon activity levels throughout the carbon circuit.

The pregnant solution will be pumped through two parallel banks of two electrowinning cells at a nominal flow rate of 48 m^3/h. On exiting the cells, the solution will report to the barren solution discharge tank and is pumped to the barren tank.

The electrowinning cells will be taken out of service for cleaning three times each week. One cell will be shut down and cleaned at a time, allowing the electrowinning circuit to function normally while the cell is cleaned. The precious-metal-bearing sludge will be washed from the bottom of the cell. The cathodes will either be washed in place or removed to a wash tank and power-washed to release the sludge. The sludge from the electrowinning cell and the cathode wash tank will report to the electrowinning sludge tank by gravity and be pumped through one of two sludge filter presses. The solution discharged from the sludge press reports to the barren solution discharge tank for return to the barren tank.

The sludge filter presses will be taken down and cleaned after the electrowinning cells are cleaned. The sludge will be placed in pans, loaded into a mercury retort, and heated to remove mercury. Most of the mercury will report as elemental mercury and be condensed and collected in 34.5 kg flasks, which will be shipped off site. The residual mercury vapor in the retort off-gas will adsorb onto activated carbon within the retort. Over time, the activated carbon will become saturated with mercury and will be replaced periodically with new carbon. The mercury loaded carbon will be shipped off site.

Smelting fluxes will be mixed with the sludge after the retort, and the mixture will be charged into the induction smelting furnace. Doré bars will be poured from the smelting furnace and shipped off site for further refining.

17.1.13 Mercury Abatement Systems

The mercury abatement circuits in gas handling have been improved and designed in more detail. Mercury abatement systems will be required at the following locations:

- Carbon reactivation – kiln feed and discharge
- Electrowinning cell fume hoods
- Gold refinery area
- POX vent gas (as described in Section 17.1.6).

In each area, mercury will be expected to volatilize into the gas stream exiting the circuit because of the elevated temperatures. Fume hoods and ducting will be used to transport the gas to mercury scrubbing systems and other mercury removal equipment. Mercury will be collected and disposed of in two forms: condensed liquid, which will be collected in specialized flasks, and mercury-loaded carbon. Both will be shipped off site.

The mercury recovery system for the regeneration kiln will consist of a spray wet scrubber, a venturi scrubber, a wet gas condenser, a coalescer, and carbon columns. The resulting clean gas will be exhausted through a stack to atmosphere. The spent carbon from the carbon filters will be periodically replenished with fresh carbon. Spent carbon transferred out of the columns will be securely packaged and is transported off site to a certified hazardous waste facility.

Raw water will be used as the quenching and scrubbing solution. In the process, various condensate streams will be collected from the scrubbing circuit. In the regeneration abatement system, the excess condensates can be sent directly to the POX blowdown area or via a coalescer to recover remaining amounts of mercury down to regulatory limits before being pumped to the POX area for reuse in the process.

In the electrowinning area, vapor and air from the electrowinning cells and other equipment in the gold recovery area will be discharged at a rate of approximately 227 m^3/min and at temperatures up to 80°C. The mercury in this stream will be recovered using two wet spray scrubbers, a venturi scrubber, a wet gas condenser, and a coalescer. Carbon columns set up in a lead-lag arrangement will serve as the final capture for mercury, to reduce emissions to regulatory limits in this area.

A booster fan will funnel the final air stream out through an exhaust stack to the atmosphere. The carbon from the carbon filters will be periodically disposed of to a certified hazardous waste facility, offsite.

Raw water will be used throughout the scrubber system. Overflow water will be sent to a coalescer and settling tank combination to capture elemental mercury before proceeding to cyanide destruction. The underflow discharge of these systems will be sent to a solution collection tank in the cyanide area mercury treatment facilities. The heavier mercury that settles to the bottom of the tank will be withdrawn for disposal. The discharge water closer to the top of the tank will be pumped to a recirculating coalescer, from which treated scrubber solution will be pumped to the cyanide destruction circuit.

In the refinery, gas discharge from the induction furnace will leave the unit at an elevated temperature of 80°C and low mercury content, but with potential for dusting. Therefore, the mercury entrainment system in this area will consist of a dust capture cyclone and carbon columns. The cyclone underflow will capture gold particulate matter that has drifted out in the gas stream. The cyclone overflow will pass through two sets of carbon columns, both filled with sulfur-impregnated carbon. These will adsorb any remaining mercury before a booster fan releases the gas to atmosphere through an exhaust stack.

17.2 Reagents

All reagent mixing will take place in a designated area within the mill building. The design of this area will incorporate such features as section bunding with dedicated sump pumps for individual reagent types, and segregated ventilation and dust control for areas with potential for dust or fume release. The design of the reagent preparation area aims to be consistent with the general intent of the International Cyanide Management Code.

Reagents will be received in the following forms:

- *Xanthate* – will be received as pellets in 850 kg bulk bag.
- *Frother 1 (MIBC)* – will be received in 20 t bulk isotainers as a high-strength solution.
- *Frother 2 (F549)* – will be received in 1 t reagent totes as a high-strength solution.
- *Dispersant* – will be received in 23 t bulk isotainers as a high-strength solution.
- *Soda Ash* – will be received as granules in bulk in 18.1 t lined sea containers.
- *Flocculant* – will be received as powder in 6 m lined containers.
- *Cyanide* – will be received as briquettes in bulk 22 t isotainers. To minimize personnel exposure to the reagent mixing process, each isotainer will be pre-piped to serve as a cyanide mix tank.
- *Carbon* – will be received in 454 kg bulk bags.
- *Nitric acid* – will be received in 25 t bulk isotainers as a high-strength solution.
- *Caustic soda* – will be received as dry beads in 1 t bulk bags.
- *Copper sulfate* – will be received in crystal form in 1.25 t bulk bags.
- *Antiscalant (Millsperse 813)* – will be received in 1 t tote tanks as a high-strength solution.
- *Mercury suppressant (UNR 829)* – will be received in 1 t tote tanks as a high-strength solution.

17.2.1 Lime

Lime is received as pebble lime in 6 m lined containers and will be transferred to a lime silo beside the reagent area. The pebble lime will be slaked in a grinding-mill-type slaker and will then be held in two holding tanks in the reagent area inside the building. The lime will be used for pH control at various points. Lime addition will be by means of a pressurized lime loop distribution system.

17.2.2 Sulfur Dioxide

Sulfur dioxide will be added to the cyanide destruction process. Bulk sulfur will be delivered to site in 1 t bulk bags. The sulfur will be transferred to the molten sulfur tank, where it will be melted by heat from the carbon elution circuit. The molten sulfur will be pumped to a furnace where it will combust with air to yield a discharge gas containing 17% SO_2 by volume. Discharge SO_2 gas will pass through the induced draft fan and blower for dilution to between 2% and 3% SO_2 and will be pressurized to feed the cyanide destruction tanks.

17.3 Process Services

17.3.1 Air and Gaseous Oxygen

The air separation unit (ASU) will produce ~99.5% purity high-pressure oxygen. Its design production capacity is 1,750 t/d of contained oxygen gas. By design, the ASU will have a production turndown of 50%, or 875 t/d, by shutting down one of the main air compressors. With low-pressure liquid oxygen tanks completely full, the low pressure liquid oxygen (LP-LOX) facility will be able to supply another 24 hours, or 1,710 t, of stored liquid oxygen before having to shut down.

Waste nitrogen from the ASU will be pressurized with a compressor and will be used in the autoclave gas handling circuit to reduce the oxygen content of the off-gas prior to mercury scrubbing. When completely full, the low pressure liquid nitrogen (LP-LIN) tanks will be able to supply another 24 hours, or 660 t, of stored liquid nitrogen.

Plant air and instrument air will be provided by three sets of compressors and an air receiver. Plant air will be delivered directly from this receiver, while instrument air will be further filtered and dried before distribution.

Low-pressure process air compressors will be provided for solution neutralization, cyanidation and cyanide destruction service. Low-pressure air blowers will be supplied for flotation, where air will be supplied directly through manifolds to each flotation cell.

17.3.2 Plant Water Distribution

Water for the plant distribution system will come from the following sources: contact water from the contact water pond, raw water from peripheral pit dewatering, reclaim water from the TSF, and fresh water from interception ponds as required to make up shortfalls.

Contact water will come from the mine facilities and waste rock storage facility runoff that collects in the contact water pond. Typically, it will have low levels of suspended and dissolved solids, and would be used for:

- Elution
- Electrowinning and refining
- Autoclave process water system (quench and gland)
- Concentrate CCD wash, gland service and wash water
- Cyanide, caustic, and other reagent systems make-up
- Primary flocculant mixing
- Make-up water to cooling systems including the oxygen plant.

The distribution system will consist of the contact water tank and a pump distribution system.

During periods of high runoff into the contact pond, when quality degrades and quantities are excessive, contact water will substitute for reclaim water in flotation and throughout the plant. In turn, raw water and fresh water can be substituted for normal contact water uses if the quality of the contact water suffers from high suspended solids.

The highest-quality water for use in the plant will be raw water which comes from the dewatering wells. As long as contact water is of sufficient quality and quantity, the raw water will be treated in the water treatment plant and discharged to the environment. When required to replace contact water, it will be suitable for all contact water usages. Raw water will also be important as the source of water for charging mill cooling and heat transfer systems. The raw water distribution system will consist of a single raw water tank and two distribution pumps.

When the quantity of pit dewatering water is insufficient, runoff water recovered from the diversion system around the TSF will be pumped from the diversion dams to a fresh/firewater tank and from there to the raw water tank.

The reclaim water system will supply water to processes that do not need high-quality water. Water will be reclaimed from the TSF and pumped to a reclaim water head tank. The water will be passed through a double-pipe heat exchanger (the heat being recovered from plant tailings during winter) and will then flow by gravity for distribution to the following areas:

- POX CCD
- CIL feed neutralization and CIL
- Carbon regeneration
- Cyanide destruction tails safety screens.

Reclaim water will also be supplied as the feed to the gland water system and flotation process water system.

Reclaim water used in CIL will first be treated for magnesium removal by an ion exchange vendor package.

POX blowdown water has been adopted as the terminology for raw water used in the POX process, but is still relatively clean and, more importantly, has not been contaminated by chlorides (above levels in the original raw water). Within the POX area, raw water will be supplied to the gas handling systems—secondary condenser, venturi scrubber, mist eliminator—where it will be used to cool and clean particulates from the off-gas. Raw water will also be used as feed to the demineralized water system and in turn as feed to the boiler.

Demineralized water will be produced as a utility in both the POX and concentrator areas serving the POX plant and the combined requirements of the carbon elution circuit and power plant, respectively. Demineralized water will be produced from plant raw water using multimedia filters and reverse osmosis system (concentrator/power plant) or ion exchange (POX).

17.3.3 Power Requirements

The total connected electric load is estimated to be 227 MW and includes mine electric distribution. The average running load is approximately 153 MW of which the process plant contributes 134 MW. Motors constitute most of the electrical load with the grinding area and pressure oxidation areas contributing more than 50% of this load.

18.0 PROJECT INFRASTRUCTURE

18.1 Summary

The planned infrastructure for the Project site includes a process plant, ancillary facilities, a power plant, TSF and water diversion and retention structures, and accommodation camp as well as extensive off-site facilities designed to support the mine, including a buried steel natural gas pipeline that originates from the Beluga gas pipeline near Cook Inlet. There is currently no road access to the Project site. Supplies will be shipped on ocean barges to a port at or near Bethel where cargo will be transferred to river barges that will transport supplies and fuel to the Jungjuk Port. An all-season access road will connect Jungjuk Port to the mine site. The site also features a gravel airstrip for personnel transport and emergency response. Water supply for processing and potable use is sourced from wells and reservoirs, with modular water treatment plants for both construction and operations.

18.2 Access and Logistics

18.2.1 Port-to-Mine Access Road

The port-to-mine access road (Jungjuk route) will traverse varied terrain from the Kuskokwim River port site near the mouth of Jungjuk Creek to the mine site (Figure 18-1).

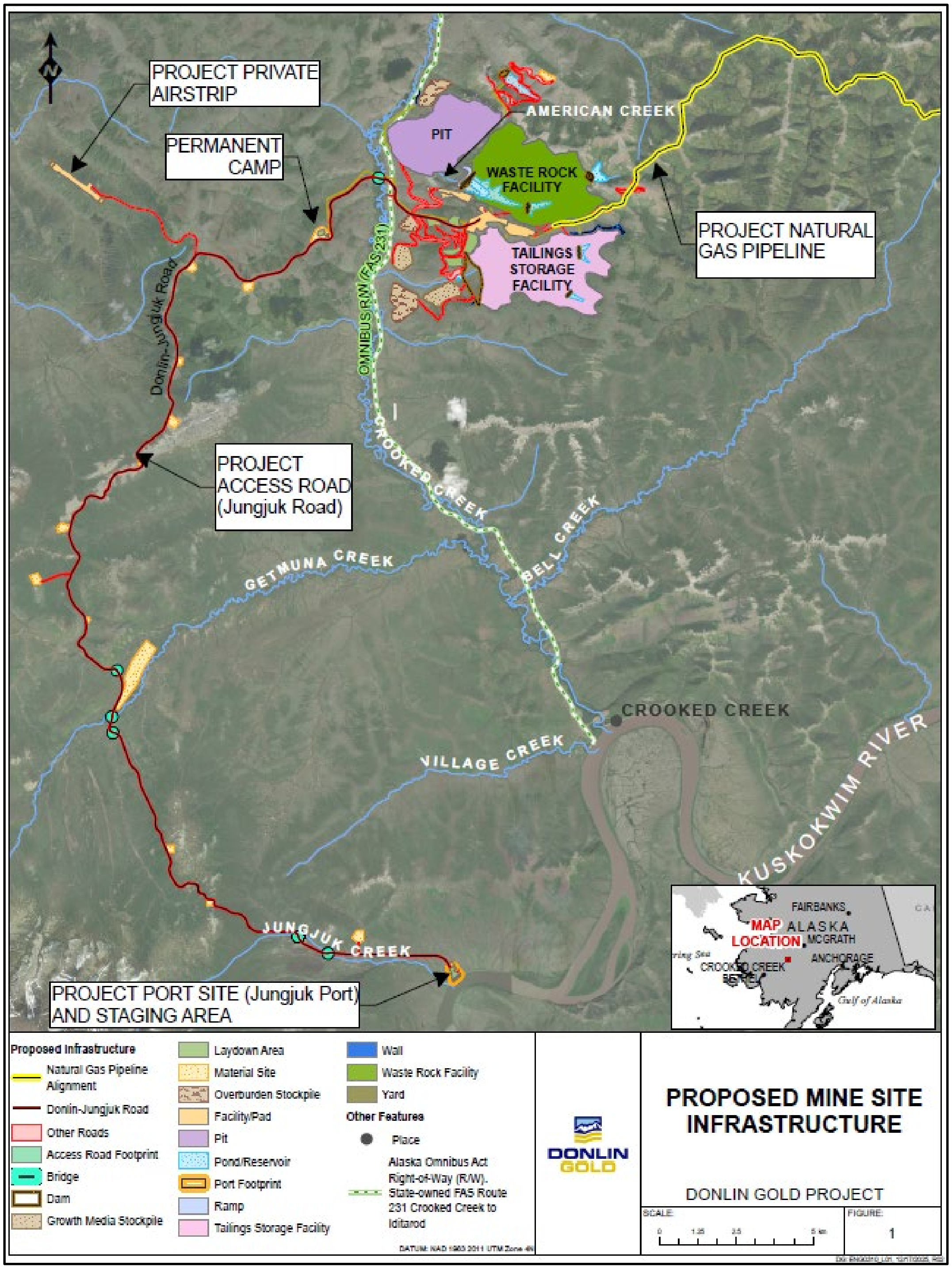
The port site is located on the north bank of the Kuskokwim River, 13 km downriver of the village of Crooked Creek; there is currently no road connection between the two locations. To the mine site battery limits, the road will be 44 km long.

The entire road will be new construction in an untracked region, with no passage through or near any settlements or communities, and no junctions with any existing road system. The road route will traverse mostly upland terrain. A 4.8 km long spur road, beginning at route km 8.7 from the mine site, will serve the Project airstrip. The mine permanent camp facilities will be located at 3.9 km from the mine site.

There are 50 identified stream or drainage crossings along the road route, but only six of them are significant and will require bridging. Bridge lengths vary from 7.5-25 m.

The access road enables the transport of cargo, fuel, equipment, and personnel throughout both construction and operational phases. The road traverses remote upland terrain where no prior road connections or settlements exist. Short spur roads branch off to serve the airstrip, material borrow sites, and the permanent accommodation camp.

Figure 18-1: Basic Route of Mine Access Road



Source: Donlin Gold LLC, 2025

To provide operational flexibility, the road will be engineered for year-round use, the access road features a two-lane, gravel surface. The design addresses seasonal drainage and spring runoff, with about 50 stream or drainage crossings along the route. Six crossings are equipped with bridges designed for durability and load capacity, accounting for ice, snow, and flood events. The remaining crossings use culverts sized to local hydrological conditions, with erosion control and armoring measures implemented as needed. Construction materials, including gravel and fill, are sourced from roughly 20 borrow sites situated along the road and within the mine footprint that will support both initial construction and ongoing maintenance. This locally sourced gravel is compacted to form a stable running surface, suitable for heavy mine traffic and the region's seasonal weather variations.

18.2.2 Road Construction

Road construction activities will be divided among three distinct areas: the site access roads to ancillary facilities (explosives storage, airstrip, haul roads); the port site permanent access road; and the Crooked Creek winter construction access road. In general, roads will be constructed using conventional cut-and-fill techniques, except for the winter construction access road, which will be developed as a winter ice road.

Simple fill-type embankments will be constructed on most of the ridge-line sections to preclude surface disturbance and reduce snow-drifting on the roadway. In general, material sites have been located at regular intervals along the alignment, precluding the need for extended haulage of construction rock.

A geotextile underlay will be provided in areas identified as having significant amounts of permafrost or wet soils. To mitigate the road deterioration due to frost heave, the roads will be built using techniques and standards used in cold regions.

18.2.3 Airstrip

The airstrip will be approximately 14 km by road west of the mine site. The airstrip design is based on U.S. Department of Transportation, Federal Aviation Administration (FAA) standards. The specified aircraft are the DHC Dash 8 and the Hercules C-130. The design was governed by the needs of the Hercules C-130. A gravel runway is suitable for both types of aircraft. A single airstrip was considered sufficient to accommodate the predominant wind directions.

18.2.4 Cargoes

General cargoes sourced globally will be shipped in containers or as break-bulk to marine terminals in Seattle and Vancouver where they will be loaded onto ocean barges for transport to Bethel. Because the Kuskokwim River begins to freeze in October and the mouth of the river does not usually clear of ice until late May, the shipping season for both ocean and river barges will be limited to 1 June to 1 October each year. The first general cargo barge of each shipping season will load and leave Seattle or Vancouver in early-May to be at the mouth of the river by the beginning of June.

Because of draft restrictions on the river downriver of Bethel later in the season, fully-laden ocean barges will discharge part of their cargo direct to river barges at Oscarville Crossing about 10 km downriver from Bethel. Those cargoes unloaded at Bethel will be placed into temporary storage or transferred directly to river barges for shipment to Jungjuk, about 312 km upriver. At Jungjuk general cargo will be off-loaded and either placed in temporary storage to await transport or loaded directly onto trucks for transport to the mine site.

Empty container storage yards, local transport, marine terminals in Seattle and Vancouver, and the ocean and river general cargo barge fleets will be operated by third parties.

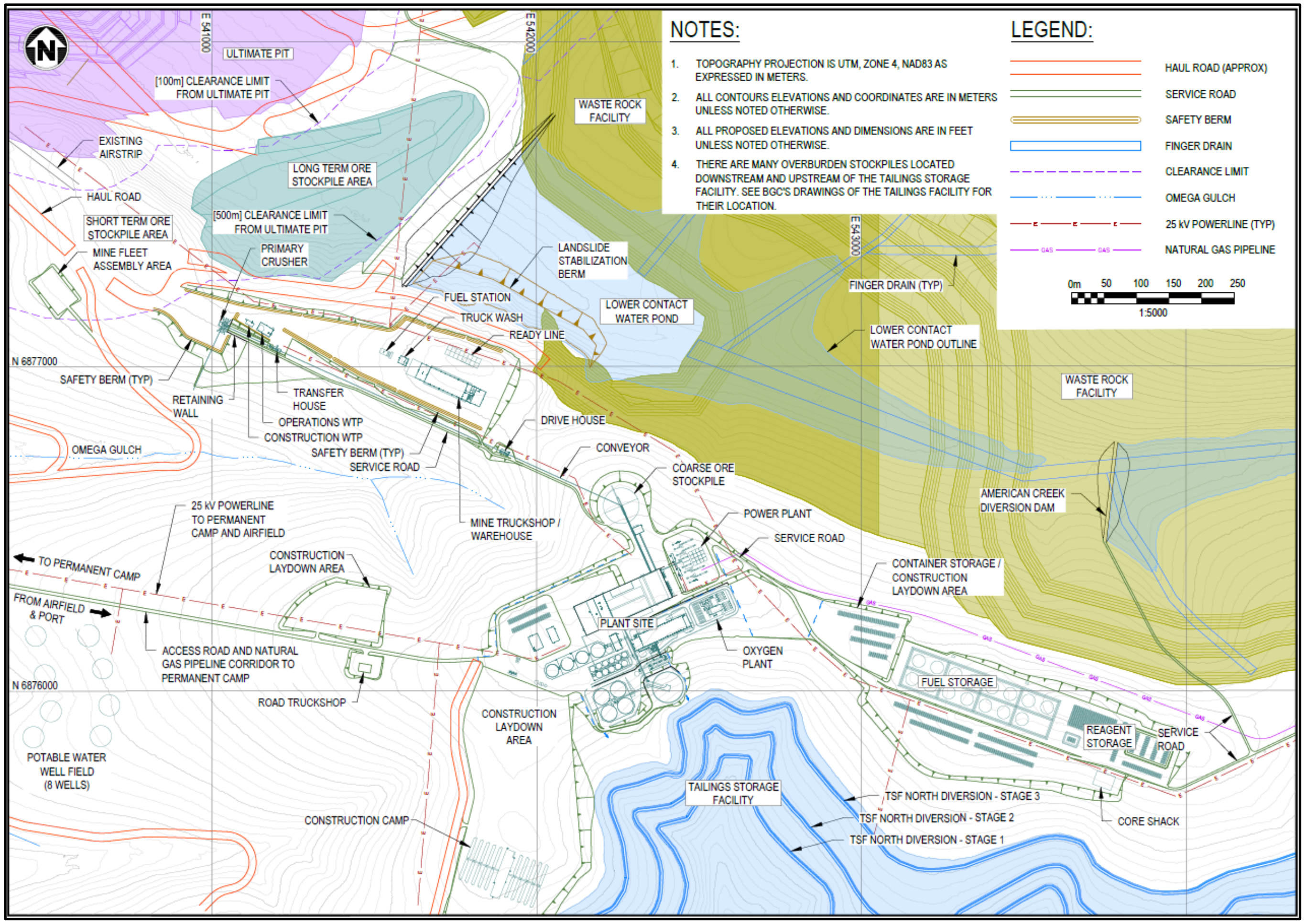
18.2.5 Fuel

Fuel sourced at refineries in the Pacific Northwest will be delivered by ocean barge to Dutch Harbor, where it will be stored in 49 ML fuel tanks to await onward transport to Bethel. The first shipments should leave the refineries in early May. Fuel will be shipped from Dutch Harbor to Bethel using an ocean barge with an 11 ML capacity. The barge will be loaded in a way to avoid the need to reduce draft before navigating the shallower sections downstream of Bethel. At Bethel, fuel will either be transferred to 38 ML storage tanks or loaded directly to a river barge for transport to Jungjuk. At Jungjuk, fuel will be offloaded into 9.5 ML storage tanks. From there, a fleet of tanker trucks with total capacity of 0.51 ML will deliver fuel to the mine site where it will be stored in fuel storage tanks with combined capacity of 142 ML. The fuel terminals at Dutch Harbor and Bethel, as well as the ocean and river barge fleets will be operated by third-party contractors.

18.3 Plant Site Facilities

Plant site facilities as shown in Figure 18-2. The layout of the plant site was designed to take maximum advantage of the topography and provides for efficient movement of equipment and material products around the site.

Figure 18-2: Plant Site Layout



Source: Wood, 2025

For the design of plant site buildings and other structures, the seismic design provisions of the 2006 International Building Code have been adopted and will need to be verified with the current 2021 edition of International Building Code.

The crusher structure will be a concrete tower founded on a raft-type foundation with a reinforced earth wall around three sides. Crushed ore will be fed to a ~1.3 km long coarse ore conveyor running to a coarse ore stockpile at the process plant.

The truckshop will house 10 heavy vehicle repair bays, various specialty service bays, a warehouse, changerooms, and offices. The mine rescue truck, fire truck, and first aid facility will also be housed within this building. The layout has been designed to allow the building to be extended in the future by up to four bay-lines, for a total of eight additional maintenance bays.

The explosives plant will be located north of the pit, behind a ridgeline to protect the explosives plant from potential flyrock. The location and characteristics of the explosives magazine will be designed to meet federal safety standards.

Exploration, environmental, geotechnical, and engineering studies have been conducted at the plant site location. These studies have typically included geotechnical drilling and assessments of soil, permafrost, and bedrock conditions. The results of the studies inform the understanding of ground stability and requirements and constraints for construction in the Arctic environment of the Project.

18.4 Camps and Accommodation

The construction camp will be built on a bench near the process plant, at a safe distance from the process plant pad so that this facility is not affected by flying rock during blasting operations for the facility rough grading, foundations excavation and buried utilities construction. The camp will include 14 stand-alone, three-story dormitories designed to accommodate 2,560 people and a stand-alone, single-story core services facility.

The permanent accommodation complex will be located just off the main access road, approximately 6 km west of the plant site. The complex will house an estimated 434 people in Year 2 and be expanded in Years 1 and 6 to accommodate a maximum of 638 people during operations.

Two modular sewage treatment plants (STPs) will be provided: one for the permanent accommodations facility, and one for the construction camp southwest of the plant site. The sewage treatment plant for the construction camp will later be reduced in size to accommodate the operational requirements of the plant site area.

18.5 Waste Rock Facilities

18.5.1 Location

The ex-pit WRF will be located in the American Creek valley, east of the open pit. The ultimate footprint of the ex-pit WRF covers an area of approximately 9 km^2. The ex-pit WRF has a maximum capacity of 2,458 Mt. Approximately 2,318 Mt of waste rock is scheduled to be placed in the ex-pit WRF. The top lift of the ex-pit WRF will be at an elevation of approximately 610 m amsl, resulting in a maximum WRF height of about 430 m. The ex-pit WRF will be constructed in 30 m lifts. The toe of each subsequent lift will be set back 47 m from the crest of the previous lift, resulting in an overall dump slope of 3H:1V.

18.5.2 Acid-base Accounting

The waste rock assessments evaluated acid generation potential and metal/metalloid leaching (ML) using mineralogical studies, bulk geochemistry, acid-base accounting (ABA), meteoric water mobility procedure (MWMP) leach tests, and kinetic experiments employing humidity cells (HCTs) and field barrels. Predictions of source water chemistry, derived from laboratory and field kinetic tests initiated in 2006 and 2008, have informed the comprehensive site water quality model.

The results indicated that most waste rock has low ARD potential (NP/AP >2), though some samples fall into uncertain or PAG ranges. Arsenic, antimony, and mercury concentrations exceed global crustal averages due to mineralization, with arsenic showing potential for leaching under both acidic and non-acidic conditions. It was established that sulfide oxidation rates correlate strongly with sulfur content, and arsenic release aligns with arsenic content, indicating that bulk rock chemistry predicts leaching behavior. The delay to ARD onset is linked to NP/AP: rock near 1.3 may take decades to generate acidity, whereas NP/AP <1 could produce ARD within several years. Arsenic leaching remains a concern across most waste rock due to elevated concentrations and demonstrated leachability.

In 2022, SRK implemented twelve additional field barrels (plus one duplicate) and commenced further humidity cell analyses to address data limitations for Waste Rock Management Category 2 (WRMC 2) and enhance the definition of source terms, with particular attention to selenium (SRK, 2023). Analysis of 20 weeks of HCT data for WRMC 2 composite samples indicates several key trends.

Lower detection limits for selenium in solids and leachates have provided improved insight into selenium leaching behavior. Early results show an initial flush followed by declining concentrations, with sulfate leaching stabilizing, enabling calculation of average release rates.

All leachates remain non-acidic, consistent with bulk ABA results. Selenium concentrations are comparable to early data from historical Phase 1 and Phase 2 programs, with similar declining trends expected to continue as concentrations fall below previous detection limits. Observed declines suggest selenium leach rates may decrease by an order of magnitude over several years, reducing source term inputs; however, caution is warranted due to particle breakdown and exposure of new surfaces over time. Bulk selenium content in 2022 samples ranges from <1-3 mg/kg, implying depletion timeframes of decades to centuries. Current data are insufficient to revise arsenic or antimony source terms without parallel on-site test results.

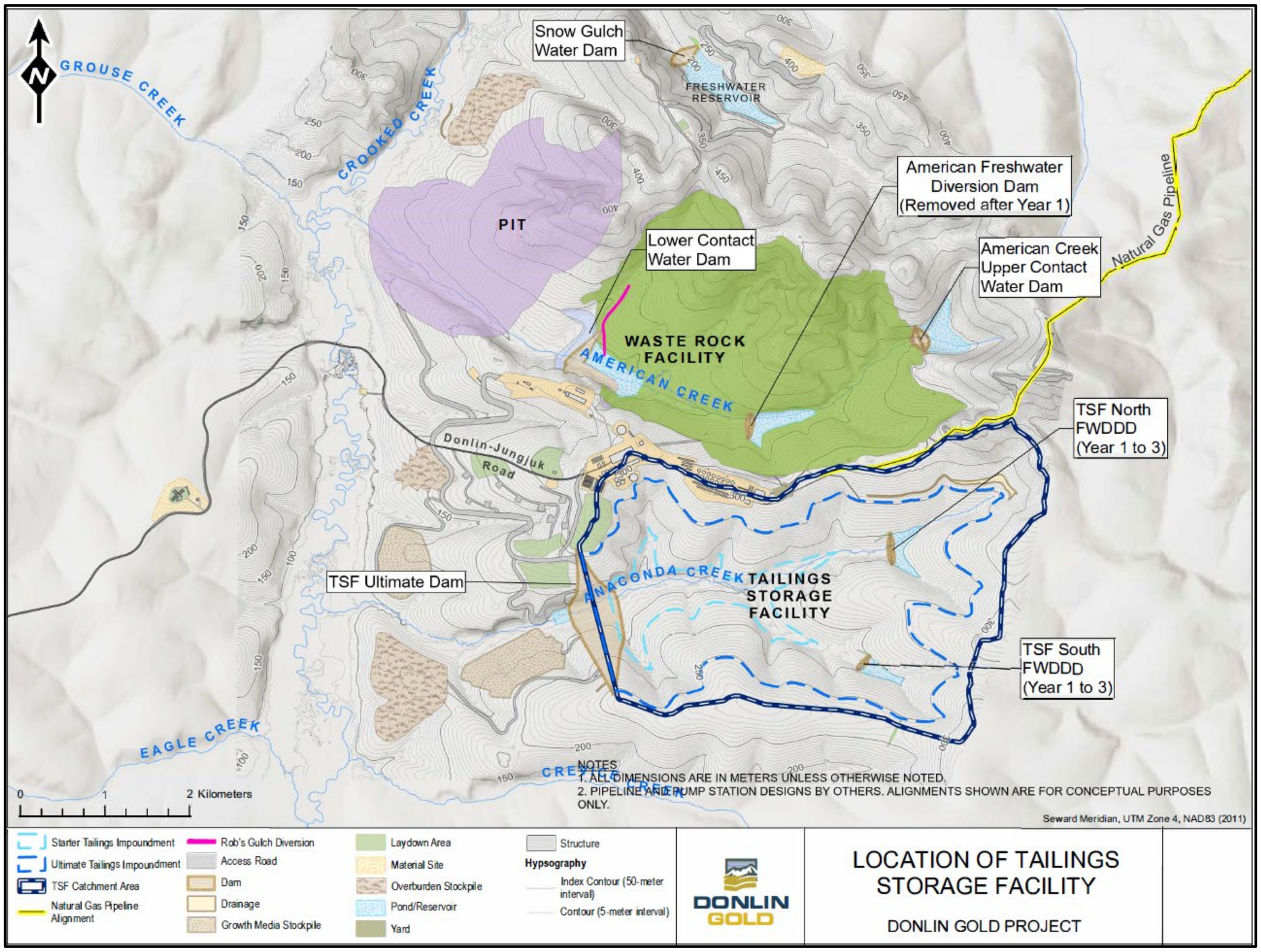
18.5.3 Geotechnical Design

Drilling investigations were conducted for the WRF between 2004 and 2016. Seismic refraction and resistivity geophysical studies were performed to delineate subsurface stratigraphy. These investigations confirmed overburden consists of peat, loess, colluvium, and terrace gravels over bedrock comprised of greywacke, siltstone, and shale. The foundation preparation plan includes stripping organic and ice-rich overburden under a perimeter shell for the facility and replacement with coarse waste rock to enhance stability and reduce settlement risk. The planned construction method is end-dumping with dozer spreading. The ultimate height with a slope of 3H:1V is achieved through 30 m lifts with 47 m horizontal setbacks between lifts. Slope stability analyses were completed for the ultimate configuration under static and seismic loading (BGC, 2011) using factor of safety targets consistent with feasibility-level standards and regulatory guidance.

18.6 Tailings Storage Facilities

The TSF will be constructed in the Anaconda Creek valley, 3.5 km south of the open pit and 3 km east of the confluence of Anaconda and Crooked creeks (Figure 18-3). Anaconda Creek flows west through the gently sloping valley bottom before joining Crooked Creek. The dam for the TSF spans the valley bottom with abutments extending up the valley sides. Two freshwater diversion dams described further in Section 18.7 are required to limit surface water reporting to the tailings dam for the first three years of operation.

Figure 18-3: Location of Tailings Storage Facility



Source: Donlin Gold LLC, 2025

18.6.1 Site Geology

Site geology is described in Section 7.2. Ash and shale that have been identified are potential weak foundation units within the TSF footprint and surrounding geotechnical domains. Ash layers, when preserved, are typically bedding-parallel. These materials occur in low-energy depositional environments and often overlap spatially. Slickensides that are observed on discontinuities in the shale also represent potential weaknesses in the foundation.

18.6.2 Seismicity

The site is seismically active due to Pacific–North American plate collision. Deformation has produced east-west folds and multiple fault orientations. Modern faulting is dominated by north-northeast striking, right lateral, strike-slip faults. The nearest identified active faults are as follows:

- Iditarod-Nixon Fork Fault: ~20 km north of the site
- Denali-Farewell Fault: ~70 km southeast of the site.

The following are the largest recorded events reported in the region:

- Magnitude 5.5 earthquake in 1959, ~90 km northwest of the site
- Magnitude 6.9 earthquake in 1903, ~55 km southwest of the site.

18.6.3 Geotechnical Site Investigation and Laboratory Testing

BGC performed a geotechnical investigation program for the TSF to characterize subsurface conditions, evaluate material properties, and provide design parameters for dam construction and tailings management. The program addressed foundation stability, permafrost behavior, and material suitability for engineered structures.

18.6.3.1 Site Investigation Program

Drill holes to investigate overburden, bedrock, and potential geohazards were summarized in Section 10. In addition, seismic refraction and resistivity studies were performed for subsurface profiling and anomaly detection. Key findings of the site investigation program include:

- Highly weathered bedrock, consisting of a soil-like, very poor-quality material was between 0-7 m thick.
- Competent weathered bedrock, consisting of poor to fair quality material was between 2-44 m thick.
- Permafrost distribution was mapped to inform thaw settlement analysis with these findings:
  - It is sporadic and typically confined to valley bottom and mid-slope
  - Thickness ranging from 1.5 m to over 15 m (average ~4 m)
  - Ice-rich soils (>15% visible ice) are generally encountered in loess or silty alluvium near valley bottoms

  - Coarse-grained colluvium on slopes and ridge tops typically has low visible ice content
  - Ice and frozen infill within bedrock discontinuities are infrequently observed.
- Fault gouge zones, ash beds, and slickensided shale joints were identified in bedrock
- The water table generally follows surface topography, with shallow groundwater in valley bottoms and deeper levels beneath ridges
- Groundwater gradients trend toward Crooked Creek and American Creek, which act as regional discharge zones
- Seasonal fluctuations occur due to snowmelt and precipitation
- Overall groundwater movement is slow because of low hydraulic conductivity.

18.6.3.2 Laboratory Testing Program

Samples of soils, rockfill, gravels, and tailings were tested per ASTM standards to define geotechnical properties, assess material suitability, and evaluate thaw settlement risks as summarized in Table 18-1.

Laboratory tests (sieve and hydrometer) were also performed on tailings and show that the material consist mainly of fine particles, with gradations typical of silt and minor clay and sand fractions (BGC, 2024). Atterberg Limits testing (ASTM D4318) indicates the tailings are nonplastic. The specific gravity of tailings particles generally ranges from about 2.6 to 2.8. X-ray diffraction performed to characterize mineralogy indicates that tailings are mainly composed of silicate minerals, with minor clay minerals. No significant concentrations of acid-generating minerals (e.g., pyrite) were noted. A tailings column for the grind produced during the feasibility study settled to a steady-state condition with an average solids content of 54% after deposition.

Table 18-1: Laboratory Results Summary of Soil Samples

Material/Sample	Test Type	ASTM Standard	Key Result / Observation	Estimated Strength Parameters	Design Implication
Soil (Overburden, Terrace Gravels)	Standard Proctor Compaction	D698	Max dry density ≈ 2080 kg/m	N/A	Excellent compaction properties for dam construction
Soil (Fine-Grained)	Atterberg Limits	D4318	Plasticity Index moderate in silt/clay zones	Cohesion: ~15–25 kPa; Friction Angle: ~28–32°	Indicates potential frost susceptibility
Soil/Permafrost	Thaw Strain & Ice Content	Custom (Frost Tests)	Excess ice in isolated zones	N/A	Requires removal or mitigation to prevent settlement
Rock (Intrusive Units)	UCS/Point Load	ISRM / ASTM D5731	High strength intrusive rocks	UCS: 80–120 MPa; Friction Angle: ~40°	Very stable foundation material
Shale (Sedimentary)	Direct Shear	D3080	Slickensided shale shows reduced shear strength	Cohesion: ~5–15 kPa; Friction Angle: ~18–25°	Bedding-parallel weaknesses
Ash Layers	Direct Shear/Preliminary Tests	D3080	Limited data; strain-softening potential noted	Cohesion: ~0–10 kPa; Friction Angle: ~15–20°	High uncertainty; conservative slope design needed
Foundation Rock	Packer Permeability	Constant Head Method	Low hydraulic conductivity	N/A	Favorable for seepage control under TSF

18.6.4 TSF Design

The TSF design follows the ADNR dam safety guidelines.

18.6.4.1 Design Criteria

Initial design criteria used a proposed hazard classification of the tailings dam in Anaconda Creek of Class I, or High. Considering this classification, the following criteria were developed:

- Minimum factors of safety will be determined by the engineer, based on hazard classification and engineering detail.
- The inflow design flood (IDF) should be based on the probable maximum flood (PMF).
- The design basis earthquake is the maximum credible event (MCE).

18.6.4.2 Inflow Design Flood

The IDF was determined on the basis that the probable maximum precipitation (PMP) is assumed to occur at the end of the 200-year snowmelt with the ground fully saturated; therefore, the entire PMP runs off the catchment area. The TSF will store the full volume of the IDF without discharge.

18.6.4.3 Foundation Stability

The dam foundation will be prepared by stripping all overburden to bedrock. No stability analyses were reported in the preliminary design report.

18.6.4.4 Configuration

The starter and ultimate dam will be constructed in a downstream direction with heights of 60 m and 144 m high, respectively, as measured from the crest to the downstream toe. The dam slopes and geometry have been established based on civil design and functional requirements. Upstream slopes are 2H:1V and downstream slopes are 1.7H:1V. No stability analyses were provided for these slopes in the preliminary design report.

18.6.4.5 Seepage and Pore Pressure Control

The TSF includes a liner as the primary barrier to seepage, preventing contaminated water from migrating into the underlying soils and groundwater control seepage. The TSF uses a geomembrane liner system installed over a prepared bedding layer (BGC, 2024). The liner is installed over the upstream face and is tied into the liner within the Anaconda Creek Valley to provide continuity and reduce the likelihood of leakage paths. The liner is placed on top of the excavated foundation, which has been stripped of unsuitable soils (such as peat, loess, fine-grained alluvium, and colluvium) to expose competent weathered bedrock.

Filter zones are required between the tailings and the rockfill that constitutes the body of the dam for protection in the event of a puncture in the liner. A factor of safety of 10 is generally considered acceptable for the underdrain design flow to account for uncertainties in the flows and drain construction and has been applied to the TSF impoundment drains. The filter materials are sourced from local quarries and waste rock and selected for their durability, gradation, and resistance to internal erosion. Laboratory testing confirms their suitability. A geotextile has been proposed for filter protection of the TSF underdrain system that extends under the TSF dam.

A seepage recovery system (SRS), consisting of a pond excavated into bedrock, lined rockfill berms, diversion ditches, and monitoring/collection wells, will be installed during initial construction, located immediately downstream of the closure footprint of the tailings dam. The seepage recovery pond and collection wells will provide secondary and tertiary containment, respectively, while the TSF liner provides primary containment.

18.6.4.6 Seismic Hazard Analyses

The TSF seismic design is deterministic, meaning the primary design earthquake is based on the MCE for the site. TSF deterministic seismic hazard analysis was completed including a near-field fault study (identifying potentially rupturing faults within 100 km of the site) and a site-specific hazard analysis based on the modified version of the US Geological Survey (USGS) National Seismic Hazard Model (NSHM) for Alaska. The recommended MCE for the tailings dam design is characterized by a peak horizontal ground acceleration on rock of 0.315 g for dam foundation and 0.372 g for rock fill, from a magnitude 7.9 earthquake (Mw).

As part of the design process and regulatory compliance, a probabilistic hazard assessment was also performed. This is not the basis for the main design, but it serves for comparison and risk evaluation to confirm the facility can withstand extremely rare events. The 1-in-10,000-year event was estimated as 0.44 g peak ground acceleration.

To define site conditions for ground motion estimation and response spectra BGC (2024) selected a VS30 of 900 m/s to represent rock fill and the VS30 of 1,380 m/s to represent the dam foundation.

18.6.5 TSF Construction

The starter dam will be constructed prior to the start of the mine operation, and then the raises will be completed approximately every four years throughout the mine life. Surface water management will be required during construction and will comprise conventional best-management practices such as cofferdams, sumps, sediment traps, and pumped diversions.

18.6.6 TSF Operation

The tailings will be transported to the TSF at a slurry density of 36% solids by weight and will be discharged via piping and spigots. Tailings beach slopes of 0.5% above the water surface and 1% below the water surface have been assumed for capacity and deposition planning purposes. Multiple subaerial spigotting deposition locations would form embankment and "force" the supernatant pool, via selected or cyclical point deposition, to initially flow towards reclaim collection (i.e., decant) barge located in the center on the north side of the impoundment, and ultimately towards the southeast end of the impoundment at the end of process operations (SRK, 2016a).

18.7 Water Management

18.7.1 Water Balance

Water management strategies for the construction, operations and closure phases were evaluated using both deterministic and stochastic water balance models. The water balance models incorporate multiple inputs, including precipitation, groundwater flows, process parameters, and surface water dynamics:

- Average annual precipitation at the Project site is about 499 mm, split between rainfall (69%) and snowfall (31%), with snow accumulation starting mid-October and melting by early May.
- Groundwater inflows to pit dewatering systems (pit dewatering wells and horizontal drains) are derived from the groundwater model. The groundwater model calibration was done by comparing simulated hydraulic heads against observed heads measured at monitoring wells within the identified HSU's.
- Stream flow measurements used for calibration of the numerical hydrogeologic model (BGC, 2014) include periods of continuous flow measurements available for five surface water monitoring stations between 1996 and 2013. This data is primarily limited to the summer period when creeks were not frozen.

- Operational parameters include a 27-year operation life, and variable throughput averaging 53,500 t/d, with tailings slurry at 35.9% solids and consolidation continuing for 52 years post-closure.

Surface runoff is calculated using the Vandewiele model for undisturbed ground and adjusted for disturbed areas, while lake levels and evaporation are tracked weekly. Stochastic modeling for the 15-month period before plant start-up shows that 3.1 Mm^3 of water from the lower contact water dam (LCWD) will be needed to meet initial process demands until TSF reclaim water becomes reliable. To prevent shortages during potential droughts, the American freshwater diversion dam (FWDD) along with Snow Gulch will provide backup supply at the start of operations.

The operational water balance model results indicate that average annual flows within the operations water balance system vary significantly across weekly, monthly, and annual scales under different precipitation scenarios. Wetter sequences lead to greater water accumulation in the TSF impoundment and increased diversion, while drier conditions result in reduced treatment and discharge volumes as more water is retained for plant use. The buildup of water in the TSF is influenced more by the frequency of extremely wet or dry years than by long-term average precipitation trends.

18.7.2 Water Retention Structures

Water retention structures and water storage are summarized in Table 18-2. A detailed water retention design will occur during the next phase.

Table 18-2: Water Retention Structures and Water Storage

Dam Name	Purpose
American FWDD	Retain surface runoff to provide fresh water to the process plant during start-up and limit inflows to the lower contact water dam (LCWD) site during construction
Upper Contact Water Dam	Stores contact water from mine facilities
Lower Contact Water Dam	Collects and manages contact water downstream
TSF North and TSF South Freshwater Diversion Dams	Retain surface runoff from entering the TSF impoundment and facilitate construction of the TSF Starter Dam and initial lined facility
Snow Gulch Freshwater Dam	Provide contingency fresh water to the process plant
Ore stockpile berm and sump	Limit surface water from entering pit

The two contact water dams (CWDs) are intended to manage and retain water that has been in contact with mine operations, including runoff, seepage, and pumped flows from the open pit and waste rock facilities. During construction, the LCWD operating capacity (8.82 Mm^3) accounts for pre-stripping runoff and extreme precipitation, with contingency measures to pump excess water to other facilities or treat it before release. No waste rock would be placed within the footprint of the LCWD during construction. The design incorporates locally sourced rockfill and terrace gravels for embankment and filter construction, with foundation preparation involving the removal of overburden and treatment of weathered bedrock to provide a stable base. A geomembrane liner tied into a concrete plinth cut-off that extends to competent bedrock provides seepage control. The dams are designed to remain in service for the duration of mine operations and will be decommissioned upon closure, with provisions for reclamation using stockpiled overburden materials.

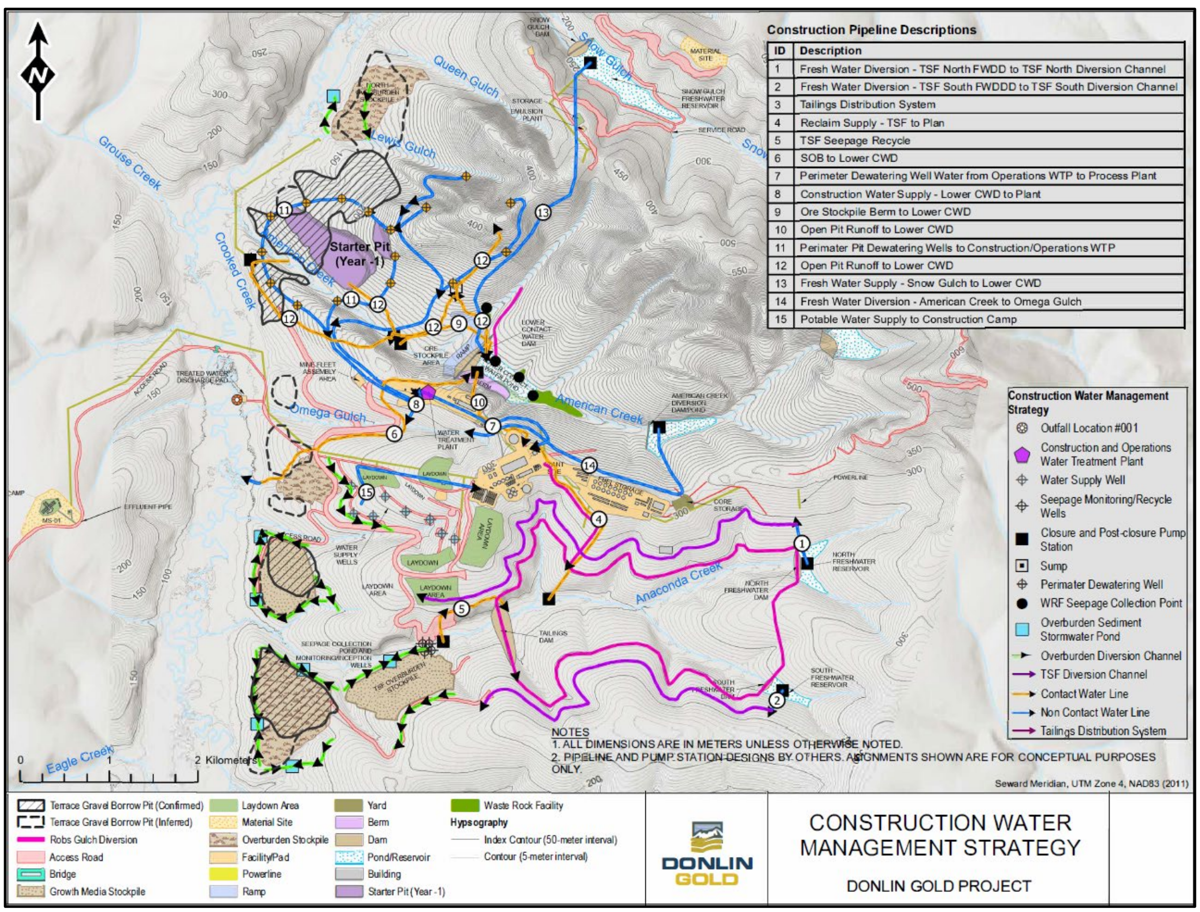
The American FWDD is intended to divert water from American Creek into the TSF and process plant, and to limit inflows to the LCWD during construction. Excess fresh water (non-contact) accumulating in the American Creek FWDD would be stored up to a maximum capacity of 1.07 Mm^3, with the excess discharged to Crooked Creek at Omega Gulch. The dam will be constructed on Kuskokwim Group bedrock, primarily siltstone and greywacke. Foundation preparation for the dam will involve removing highly weathered bedrock and overburden and then treating the competent bedrock. The structure will include a concrete face slab on the upstream slope to provide an impermeable barrier, supported by well-graded rockfill zones and transition zones for seepage control and stability. A concrete plinth anchored into bedrock will form a watertight connection at the toe of the upstream face. Beneath the plinth will be a grout curtain to minimize seepage through the foundation. Hydraulic features include a spillway and diversion structures designed to manage flood events and operational flows, with energy dissipation measures downstream. Construction will use locally sourced rockfill and be staged to accommodate seasonal constraints and material availability, with special consideration for permafrost and frost heave in the foundation design. Instruments, such as piezometers and settlement gauges, will be installed to monitor dam performance throughout its operational life.

The Snow Gulch freshwater dam (FWD) is designed to provide contingency fresh water to the process plant during mine start-up and operations. The dam will be constructed within Snow Gulch, north of the American Creek watershed, and is intended to be in service for the life of the mine before decommissioning at closure. Except when water is being withdrawn from the pond for use in the process plant, the dam will be kept at its maximum storage capacity of 4 Mm^3. The dam embankment will consist of a rockfill shell with an internal geomembrane liner. Locally sourced terrace gravels and quarried rock will be used for construction materials. Foundation preparation will involve stripping overburden to weathered bedrock and excavating rippable near-surface bedrock beneath the dam footprint. Seepage will be controlled by installing a geomembrane liner tied into a concrete plinth cut-off extending to bedrock. Detailed seepage analyses will inform the design.

18.7.3 Construction Water Management Strategy

Construction focuses on treating and discharging pit dewatering groundwater to prevent excessive buildup in the contact water dams, minimizing contact water treatment, ensuring adequate water supply for plant commissioning, and providing pit depressurization without storing water in the TSF until just before start-up. Key components during construction include the American Creek FWDD, temporary TSF diversion dams, Snow Gulch Reservoir, pit dewatering systems, ore stockpile berms, stormwater runoff controls, potable water wells, and domestic WTP discharge. Water infrastructure required for the construction phase is illustrated in Figure 18-4.

Figure 18-4: Construction Water Management Layout



Source: Donlin Gold LLC, 2025

Runoff from mine facilities in the American Creek drainage, including the pre-stripping excavations for the open pits, the WRF, and other mine facilities would be managed as contact water. Alaska Pollutant Discharge Elimination System (APDES) general permit allows treatment of contact water.

18.7.3.1 Initial Pit Dewatering

Pre-stripping begins in ACMA and Lewis pits, approximately 15 months prior to commencement of process plant operation, and runoff from this initial pit footprint will be managed as contact water. To achieve pit depressurization targets, pumping from perimeter dewatering wells will begin six months before the start of pre-stripping. Construction of the operations WTP will be completed prior to pre-stripping so that pit depressurization targets can be achieved. Seventeen perimeter dewatering wells will be operational by Year -1 (SRK, 2017). Pit dewatering water collected during construction would be treated in the WTP and discharged to Crooked Creek near the confluence of Omega Gulch under an APDES permit.

18.7.3.2 Tailings Storage Facility

Two temporary FWDDs will be constructed upstream of the TSF in Anaconda Creek and completed in Year -2. The diversion dams will minimize runoff to the TSF from undisturbed ground and also divert fresh water (surface water and noncontact stormwater) during construction of the TSF starter dam and placement of the impoundment liner. The dams would be in use until Year 3 of operation, at which time both the TSF North and South FWDD would be decommissioned and the area would be regraded and incorporated into the ultimate TSF impoundment to allow for additional tailings storage.

A rock fill underdrain capable of handling the base flow through the Anaconda Creek valley will be placed beneath the liner system to prevent the build-up of pore pressures beneath the TSF. The underdrain will be placed prior to installing the impoundment liner. During construction, discharge from the SRS may require treatment prior to discharge and if needed would be treated in the WTP and discharged to Crooked Creek near the confluence of Omega Gulch under an APDES permit. An average annual discharge of 340 m^3/h from the SRS is anticipated during late construction (SRK, 2017).

18.7.3.3 Overburden Stockpiles

A number of overburden stockpiles are required to store material that will be used to reclaim the TSF and WRF. All of these stockpiles lie beyond areas that drain into proposed dams and will need sediment control structures. Overburden stockpiles requiring sediment control include:

- The north overburden (NOB) stockpile north of American Creek on the east side of Crooked Creek
- The south overburden (SOB) stockpile between the American Creek and Anaconda Creek valleys on the east side of Crooked Creek
- Three stockpiles downstream of the TSF dam. (TSF Overburden Stockpile, Growth Media Stockpile 1, Growth Media Stockpile 2).

Runoff from these stockpiles will be managed by intercepting and directing surface runoff toward sediment ponds sized to contain the 10-year return period, 24-hour duration storm.

The diversion channels will be sized for the 100-year rainfall event. Two sets of diversion channels are proposed. Upslope diversions will limit runoff to the overburden dumps, while channels on the downslope side will direct surface runoff to the sediment ponds.

18.7.3.4 Construction Camp Potable Water Supply and Domestic Wastewater

The source of water supply for the construction camp and, later, the plant site potable water systems, will be an array of eight wells south of Omega Gulch, near Crooked Creek.

Two modular sanitary treatment plants (STPs) will handle domestic wastewater for camp facilities. The permanent accommodation STP, situated about 10 km west of the main plant, is designed for 638 residents, while the construction camp STP, located next to the plant site, will serve up to 2,560 people. At its peak, the construction camp STP is expected to discharge up to 533 m^3/d into Crooked Creek under an APDES general permit. Once construction winds down, the construction camp's STP will be downsized to suit the smaller wastewater volumes generated at the plant site. During operations, discharges from both the plant-site and permanent accommodation STPs will be rerouted to the process plant or TSF. Biosolids will undergo filter pressing to remove extra water before incineration.

18.7.3.5 Fire Water

Fire protection water supplies for the construction camp and permanent accommodation facility will be provided from the freshwater storage tanks. The tanks will have a reserve supply for fire protection capable of providing 1,900 L/min for one hour, in accordance with NFPA requirements for ordinary hazard occupancies.

18.7.3.6 Plant Start-Up Water Supply

Approximately 0.23 Mm^3 of non-turbid water will be required for process plant commissioning, and 3.1 Mm^3 will be required at process plant start-up (SRK, 2017). This startup volume is based on the ability to meet the process water requirements until reclaim water from the TSF can be relied on (i.e., sediment content/clarity is suitable for process use).

The deterministic and stochastic water balance model results indicate this volume will be met by the LCWD, even in dry years. In extreme dry years, the remaining water requirement could be supplied from the Snow Gulch Reservoir and/or the American FWDD.

18.7.4 Operations Water Management Strategy

The water supply and management plan during operations is designed to ensure sufficient fresh water for processing, maintain treated water discharge within environmental standards, and minimize the TSF pond volume throughout operations and closure. Water for the process plant will come from multiple sources with priority ranking as listed:

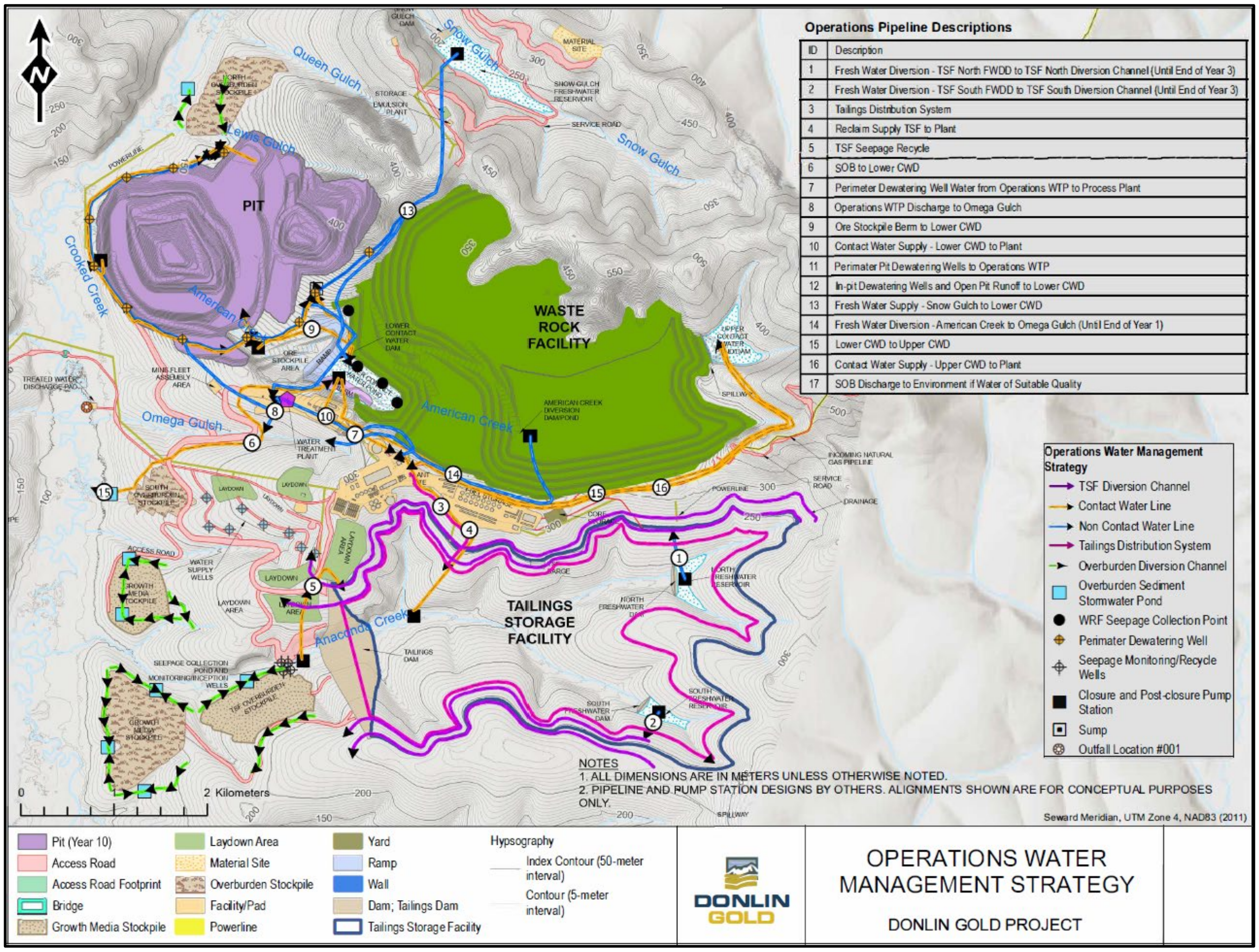
- Dewatering wells
- TSF reclaim water
- TSF SRS water
- Runoff collected in the LCWD and UCWD
- Retentate (brine) from the WTP
- Fresh water from Snow Gulch Reservoir.

A number of structures and operating rules have been designed to meet these objectives. Operations water management components and structures are shown in Figure 18-5.

18.7.4.1 Mine Pit Dewatering

Mining operations conducted below the water table necessitate dewatering. The depressurization system consists of surface water diversions, vertical perimeter wells, horizontal drains, and in-pit water collection and water removal infrastructure. The dewatering system includes up to 115 wells (35 perimeter and 80 in-pit), with depths averaging 215 m for perimeter wells and 188 m for in-pit wells. Pumping rates will range from 242–630 m^3/h, with peak operation involving 52 wells during Years 14–15. Groundwater extraction will start at about 386 m^3/h in Year 2, peak at 540 m^3/h in Year 12 and decline to 250 m^3/h after Year 20 as perimeter wells are shut down during backfilling. Horizontal drains totaling 268 km will assist in depressurizing pit slopes, lowering the water table to -335 masl by Year 20.

Figure 18-5: Operations Water Management Layout



Source: Donlin Gold LLC, 2025

18.7.4.2 Tailings Storage Facility

The TSF is designed to manage tailings from flotation, wash-thickener overflow, and detoxified cyanide slurry. It includes a main lined embankment, two temporary FWDDs, a reclaim water system, and an underdrain connected to the SRS. The TSF will provide storage for tailings, operating pond, floodwater, and a 2 m emergency freeboard, meeting the Federal Emergency Management Agency (FEMA) requirements for the PMF and 24-hour PMP. The starter dam height is 60 m, with an ultimate height of 143.5 m and length of 1,787 m, impounding about 515 Mt of tailings and over 451.7 Mm³ of water. No spillway is needed during operations, but one will be added at closure. Tailings slurry will flow by gravity through a high-density polyethylene (HDPE) pipeline with a 120 cm diameter in a lined corridor 16 m wide, with an average flow of 3,960 m³/h. Reclaim water will be pumped back to the plant at about 3,213 m³/h via a 2.5 km pipeline. The SRS includes a seepage collection pond (20,000 m^3 capacity), monitoring wells, and interceptor wells to manage seepage and prevent contamination. Interceptor wells can pump up to 400 m^3/h if needed. Diversion channels lined with 1 mm LLDPE will divert runoff from catchments up to 380 ha, designed for 200-year peak flows of 0.9–3.7 m^3/s. It is expected that ice will form on the operating pond in winter, so beach slopes of 0.5% above water and 1% below water will prevent liner damage.

The stochastic water balance model was used to predict the likely range of the TSF impoundment volume over the LOM and at the end of operations. While the stochastic water balance model results show a steady increase in the TSF impoundment volume over the LOM, impoundment volumes are expected to fluctuate from year to year.

18.7.4.3 Waste Rock Facility

The LCWD and UCWD in American Creek are designed to manage runoff from the WRF and pit. The UCWD, includes a spillway to handle the PMF, while the LCWD has no spillway because its storage capacity accommodates the 24-hour PMP plus an operating pond. The UCWD has a drainage area of 206 ha and storage of 4 Mm^3, while the LCWD covers 1,390 ha with 8.82 Mm^3 storage.

The WRF layout has been designed such that the LCWD can store 0.5 Mm^3 of contact water without inundating waste rock. There are three concerns with the waste rock becoming excessively inundated by the pond storage:

- Potential siltation of the underdrain voids over time
- Geotechnical stability issues with the potential siltation of the underdrains
- Geochemical interactions.

Constant wetting and drying of the waste rock could degrade the water quality of runoff to the LCWD such that it could no longer be considered as a water source for the process plant. Water quality issues would be addressed during operations by limiting pond volumes in the LCWD such that a volume of 0.5 Mm^3 is not exceeded more than 5% of the time. A pumping plan ensures that when volumes exceed 0.35 Mm^3 in the LCWD, water is transferred to the UCWD.

18.7.4.4 Runoff Controls

The plant site, located on the ridge between American Creek and Anaconda Creek, will have controlled drainage directed to the CWDs or TSF, ensuring noncontact stormwater is diverted through TSF diversion channels without contamination. Runoff originating from upslope areas near active mining sites will be intercepted by a constructed ditch and diverted around the perimeter of the pit. Within the open pit, both runoff and snowmelt will be systematically managed through an integrated network of ditches, sumps, pumps, and booster stations. This system comprises two main components: a series of pumping stations and gravity sumps placed around the pit crest, along with a network of in-pit pipelines, pumps, and ditches designed to transport water out of the pit. The system is engineered with a peak pumping capacity of 1,885 m^3/h, which allows it to remove runoff from a 30 mm storm event within three days and from a 76 mm storm event within seven days. Mobile pumps are included to ensure operational flexibility during more extreme weather conditions. Additionally, roadside and bench ditches will capture runoff and direct it to sumps where it can be efficiently pumped into the perimeter system, thereby minimizing pumping expenses and supporting continuous operational effectiveness.

18.7.4.5 Process Water Requirements

Water requirements depend on process plant feed rates, which vary annually (SRK, 2017). The process plant requires an average water supply of 4,088 m^3/h during active operations, with water losses of about 62 m^3/h. A minimum of 568 m^3/h of makeup water is needed, sourced from contact water ponds, pit dewatering, TSF SRS, and, if necessary, Snow Gulch Reservoir. Reclaim water from the TSF is the primary source of process water, averaging 3,213 m^3/h and peaking at 3,520 m^3/h when the TSF pond volumes exceed 6.0 Mm^3.

The WTP should operate at a maximum capacity of 1,080 m^3/h, treating water from dewatering wells (maximum 540 m^3/h, average 181 m^3/h), CWDs (maximum 250 m^3/h, average 101 m^3/h), TSF pond water (maximum 10 m^3/h, average 5 m^3/h), and SRS water (maximum 235 m^3/h, average 95 m^3/h). In low-precipitation years, Snow Gulch Reservoir will supplement the water supply. Groundwater from dewatering will primarily supply process water unless combined contact dam storage exceeds 1.8 Mm^3, in which case water will be treated and discharged to Crooked Creek.

The Snow Gulch Reservoir will serve as a contingency source of fresh water for the process plant during operations. It will have an operating capacity of approximately 4 Mm^3, with a dam height of 46 m and a spillway designed to pass peak runoff from a 100-year storm.

18.7.4.6 Operations Water Treatment Plant

The Operations WTP will be required to treat dewatering and other process water flows that are expected to require treatment to meet AWQS prior to discharge to Crooked Creek. The Operations WTP is expected to operate for approximately 29 years (two years during pre-production followed by 27 years during mine operation). The original WTP design was based on a high-density sludge chemical precipitation process for removal of metals, including arsenic and manganese, and included reactor tanks for lime and iron coagulant addition, clarifiers, pH adjustment and filtration. The current design is a membrane-based reverse osmosis (RO) system that includes clarifiers and filters for pretreatment.

The Operations WTP will have a combined maximum design capacity of approximately 1,080 m^3/h, with an anticipated maximum treatment rate of approximately 1,009 m^3/h, and will treat water from the following sources:

- TSF reclaim: annual volume not to exceed net precipitation falling over TSF impoundment and ore stockpile area less evaporation from ponds
- CWD flows: contact water from the UCWD and LCWD
- SRS flows
- Pit dewatering flow from dewatering wells.

During periods of high runoff, TSF pond volumes are predicted to rise even with treatment of the other sources of contact water. Therefore, when excess TSF pond volumes develop, TSF water will be sent to the WTP and mixed for treatment along with the other sources of water. The objective of treating water from the CWDs is to build flexibility into the water management system such that TSF pond volume is minimized to the extent practical during operations, while maintaining a sufficient supply of process makeup water. Seepage from the lined TSF is expected to be minimal and the SRS water is considered a cleaner source for treatment and will typically be sent directly to the WTP during mine operations and construction as needed. Open-pit dewatering well water will be the cleanest available water source and prioritized for treatment and release.

Water quality estimates for the sources that supply the Operations WTP are derived from a combination of baseline environmental surface water and groundwater characterization data, results from humidity cell tests, and modeling outputs using process and geochemistry models.

18.8 Bethel Marine Terminal

Bethel is about 105 km upriver from the mouth of the Kuskokwim River where it empties into Kuskokwim Bay. It is the only port on the Kuskokwim River accessible to ocean barges.

The Knik Construction Yard site, which is downstream from the Port of Bethel, has been identified as the site for a general cargo barge terminal dedicated to the Project. It has road access and could be connected to the local power grid.

Ocean barges will be unloaded of their cargoes, which will either be transshipped directly to river barges or placed in storage to wait for an available river barge. The salient features of the terminal will include:

- A berth for ocean barges
- A berth for river barges
- A roll-on/roll-off berth
- A storage area for containers and break bulk cargoes
- Buildings and ancillary infrastructure and services.

To provide adequate space for five ocean barge loads, the terminal yard will need to be about 5.1 ha in area. With a further 1.4 ha required for the wharf, buildings, access roads, and ancillary services and facilities, the terminal will have a total area of 6.5 ha.

Two years prior to operations, during the peak construction year, up to 30 ocean barges will call at Bethel and unload construction cargoes.

The wharf structures, fender systems, and mooring systems will be designed for a minimum service life of 25 years. Other conditions factored into the design of the wharf include site conditions, ice, environmental conditions, a corrosion allowance, and the vessels that will use the terminal.

Electrical power for yard lighting and office needs at the port will be provided from the local grid. Potable water will be obtained from a well, and sewage will be sent to a septic drain field. Water for fire protection will be pumped from the river through a temporarily deployed pump and hose into a heated, insulated, above-ground 900 m^3 dedicated firewater storage tank.

To accommodate the mining facility consumption of about 152 ML of diesel fuel annually, storage capacity at the existing tank farms in Bethel will be increased by 22.71 ML.

18.9 Jungjuk Port Site

The Jungjuk port site is located on the north bank of the Kuskokwim River, 13 km downriver of the village of Crooked Creek and 312 km upriver of Bethel. A viable site has been identified approximately 300 m upriver of the mouth of Jungjuk Creek. A staging area of 8.2 ha will be developed on adjacent uplands immediately north of the dock site.

The port will be developed on a south-facing slope rising from the Kuskokwim River. Elevation at the port site is roughly 40 masl, on the high ground between the Jungjuk Creek drainage to the west and a minor stream that discharges into the Kuskokwim River about 500 m upriver of the mouth of Jungjuk Creek.

Facilities at Jungjuk will include two river barge berths, a barge ramp, container-handling equipment, seasonal storage for containers, break-bulk cargo, and fuel, and barge-season office/lunchroom facilities.

The wharf structures, fender systems, and mooring systems will be designed for a minimum service life of 25 years. The river barges will arrive in tows of four (two-by-two configuration). River barges will be unloaded of their cargoes, which will be placed in storage to await onward transport to the mine site by truck.

The terminal yard will be capable of storing about 1,000 TEU containers.

Containers and other cargo will be trucked to the mine throughout the summer barging season. Fuel will be off-loaded from barges and temporarily held in a storage tank before being pumped to B-train trucks for transport to the main fuel storage facility at the mine site.

Electrical power for yard lighting and office needs at the port will be provided by diesel generators. Potable water will be obtained from a well, and sewage will be sent to a septic drain field. Water for fire protection will be pumped from the river through a temporarily deployed pump and hose into a heated, insulated, above-ground 900 m^3 dedicated firewater storage tank.

18.10 Power and Electrical

Electric power for the Project site will be generated from a dual-fueled, (natural gas and diesel) reciprocating engine power plant with a steam turbine utilizing waste heat recovery from the engines. The conceptual design of the generating station does not provide thermal energy for building or process heating; the waste heat is used to generate electricity. The power plant consists of two equal halves, each consisting of six reciprocating engines, and a separate steam turbine. Ancillary equipment will include engine halls, electrical room, control room, station service transformers, black-start gensets, exhaust stacks, radiators, lube systems, waste oil handling, waste heat recovery, turbine halls, electrical distribution equipment, and control and protection. The two parts of the plant will be separated by a blast wall such that either half can provide emergency power to the site in the event of a catastrophic failure.

The total generation facility is nominally rated at 182 MW initially. This will increase to 215 MW after four years with the addition of two more gensets (one in each half) to allow for N+2 redundancy, thus permitting planned maintenance and predicted outages without cutting back production.

To minimize electrical distribution costs and load losses, the power plant will be strategically located adjacent to the two major process electrical loads: the oxygen plant and the grinding mills.

The total connected electric load is estimated to be 227 MW, the average running load approximately 153 MW, and the peak load 182 MW. Motors constitute most of the electrical loading. The largest are the grinding mill motors, which are gearless (wrap around) type that use cyclo converter variable-speed drives with soft-start features. The oxygen plant will have three large synchronous motors that use a load-commutated inverter (LCI) controller to provide motor soft-start to minimize the stress on the power supply during starting and to reduce voltage flicker.

Power will be distributed to the main process areas by metal-clad cable feeders in trays mounted on racks routed to the local electrical rooms through utilidors, process buildings, and conveyor galleries. This will include the grinding, POX, CIL leach, refinery, and coarse ore reclaim areas. Overhead power lines will be run to the more remote areas, including the primary crusher, the water system, pumping, tailings, mine open pit electric shovels, and pit dewatering sites. Each process area load center will have associated electrical rooms, distribution transformer(s) to bring the voltage down to utilization levels, secondary distribution centers and switchgear, and motor/feeder distribution equipment. To minimize costs the electrical rooms will be prefabricated off-site with all electrical equipment installed, pre-wired, and tested prior to shipping to site.

Power will be provided to the permanent accommodation facility on a 34.5 kV power line. In addition, a local emergency power diesel genset will be installed to provide backup power in the event of a failure of the pole line. There are no major facilities located at the airport. The airstrip will have a 100 kW genset to run fuel pumps and lights as needed and will not be connected to the site power supply.

The Jungjuk port site will have an independent, stand-alone power generation facility equipped with two 600 kW (one backup) gensets. The power station will be supplied completely self-contained with all controls and ancillary equipment, ready for installation on site. The unit will include all accessories, radiators, cooling fans, exhaust systems, air and fuel filters, engine control panels, alternators, controls, starting batteries, and chargers within a modular enclosure suitable for shipping and travel on standard highways. The module will be complete with mounting skids, access (maintenance, equipment, and man-doors), ventilation, interior and exterior lighting, general power receptacles, provision for grounding, fuel day tank, supply and return fuel lines to engines, and circuit breaker with control, protection, indication, and alarms.

The port site will also include a control module unit to house the electrical switchgear, synchronization controls, and operator interface equipment. The three modular units will be arranged together and will be installed on concrete foundations.

18.11 Natural Gas Pipeline

18.11.1 Design Overview

The 14-inch natural gas pipeline will be approximately 507 km long. It commences at the west end of the Beluga Gas Field, approximately 53 km west of Anchorage at a tie-in near Beluga. The pipeline route crosses an area with no significant pre-existing infrastructure and does not follow any existing utility corridors.

The pipeline will receive booster compression supplied by one compressor station located at approximately mile post 0.4. No additional compression along the pipeline route will be required. The pipeline will transport approximately 1.56 Mm^3/d.

The pipeline will be regulated by the US Department of Transportation (DOT) under Title 49 of the Code of Federal Regulations, Part 192 – Transportation of Natural Gas and Other Gas by Pipeline: Minimum Federal Safety Standards (49 CFR 192). The pipeline will be designed, constructed, and operated in accordance with the applicable requirements of 49 CFR 192 and will incorporate pig launching and receiving facilities (receipt, midpoint, and delivery), approximately 16 mainline block valves spaced at 32 km intervals along the line, cathodic protection, leak detection, a supervisory control and data acquisition (SCADA) system, and a fiber optics cable along the full length of the route, from origin to the mine site.

The design life of the pipeline is 30 years; however, design life can be extended with additional maintenance and repair.

18.11.2 Geotechnical Review of Pipeline Route

The Donlin Gold natural gas pipeline will transport natural gas from the existing ENSTAR Beluga pipeline near Beluga, Alaska, to the Donlin Gold mine. The route traverses varied terrain, including plains, hills, mountains, and river crossings, and is designed for a 30-year engineering life. The geotechnical design aims to provide a stable foundation for all pipeline elements, appropriately managing risks to public safety and the environment. Key aspects include material selection for bedding, padding, and backfill.

The pipeline-related geohazards that have been assessed include frost heave, thaw settlement, seismicity/tectonics, slope stability, erosion, and buoyancy, as well as geochemical issues and unique soil structures.

The pipeline geothermal design focuses on the interaction between soil mechanics and heat transfer, especially thaw settlement in ice-rich soils. Mitigation options for thaw settlement include replacing unstable soils, applying insulation, or controlling pipeline temperature. Hydraulic and geothermal modeling are used to predict soil temperature changes and thaw progression, informing settlement and strain calculations for pipeline integrity.

The pipeline route crosses regions of high seismic activity due to the Pacific and North America plate convergence. The pipeline route crosses the Denali Fault, which is a significant crossing, and the pipeline design includes parameters for lateral displacement (6.4 m) and a fault zone width of 30.5 m. Due to the pipeline route crossing high seismic areas, site-specific investigations must be conducted to assess liquefaction potential for saturated soils where risks are identified.

Slopes along the route are assessed for risks such as landslides, debris flows, avalanches, and rock falls. It is always preferred to avoid risks, but where avoidance is not possible, mitigation is used. Mitigation includes slope flattening, drainage improvements, buttressing, and burial beneath shear planes.

Geotechnical characterization of the pipeline route includes about 450 probes/holes and 50 test pits, mapping of terrain units and soil and groundwater conditions, index property tests, and ground temperature collection for permafrost assessment.

18.12 Fuel

18.12.1 Diesel

The maximum diesel fuel storage capacity at the plant site is estimated to be 142 ML. Storage for 9.5 ML of fuel will be provided at the Jungjuk port site, but this is only intended for short term use while the fuel barges are being unloaded during the summer and is not included in the design of the overall site storage capacity.

The plant site fuel storage facilities are sized for a ten-month supply plus one month of contingency for the mine fleet and site mobile equipment. Fifteen fuel tanks, each having a capacity of 9.5 ML will be installed within a high density polyethylene (HDPE)-lined and bunded tank farm approximately 600 m east of the plant site and at an elevation of approximately 299 m.

During the summer shipping season, fuel tanker trucks will transfer diesel fuel from the 9.5 ML temporary storage tanks at the Jungjuk port to the plant site fuel farm. From there, the fuel will be distributed to various day tanks around the site.

18.12.2 Natural Gas

Natural gas will be supplied to the various buildings at the plant site for heating through an underground network of pipes. The main distribution line will extend 7.2 km along the main access road to the permanent accommodation complex and supply fuel for the forced-air heating system and for the cooking appliances in the camp kitchen.

The supply of construction diesel fuel and propane will be delivered during the summer shipping season. The diesel fuel will be stored in temporary 500 m^3 tanks until the first two 9.5 ML storage tanks are completed and can be filled with fuel. Propane will be stored in fourteen 38,000 L mobile tanks.

19.0 MARKET STUDIES AND CONTRACTS

19.1 Marketing Partnership Agreement

The Limited Liability Company Agreement of Donlin Creek LLC dated 1 December 2007 as amended (the LLC Agreement) provides for the members of DCLLC (the Members) to take in kind their respective shares of the gold production, which they can then sell for their own benefit.

The LLC agreement further provides that neither Member shall have any obligation to account to the other Member for, nor have any interest or right of participation in, any profits or proceeds, nor have any obligation to share in any losses from futures contracts, forward sales, trading in-puts, calls, options or any similar hedging, price protection or marketing mechanism employed by the other Member with respect to its proportionate share of the production.

19.2 Gold Marketing

NOVAGOLD's portion of gold production is likely to be sold on the spot market, by marketing experts retained by or on behalf of NOVAGOLD. Gold can be readily sold on numerous markets throughout the world and it is not difficult to ascertain its market price at any time. Since there are many available gold purchasers, NOVAGOLD would not be dependent upon the sale of gold to any one customer. Gold could be sold to various gold bullion dealers or refineries on a competitive basis at spot prices.

There are no contracts material to NOVAGOLD required for refining, transportation, handling, sales and hedging, and forward sales contracts or arrangements in place. NOVAGOLD expects that terms contained within any sales contract that could be entered into would be typical of, and consistent with, standard industry practices, and be similar to contracts for the supply of gold elsewhere in the world.

19.3 Gold Price

A long-term forecast gold price of $2,100/oz was applied over the time period the Mineral Reserves will be produced and the 27-years of cash flow. This long-term forecast price reflects the combination of information derived from a number of reputable banks as well as cash flow prices used in technical reports filed in Canada over the previous 12-month period and historical price averaging.

In accordance with industry-accepted practice, a higher gold price for the Mineral Resource estimates is used than what was used for Mineral Reserves. This helps ensure that Mineral Reserves are a subset of Mineral Resources. The long-term forecast gold price was increased by 15% to provide the Mineral Resource estimate gold price of $2,400/oz which was fixed over the 24-year LOM.

20.0 ENVIRONMENTAL STUDIES, PERMITTING, AND SOCIAL AND COMMUNITY IMPACT

The Project has undergone environmental baseline studies, permitting, and community engagement for 30 years in support of the development of the large-scale open-pit gold mining operation.

20.1 Baseline Studies

Comprehensive baseline data collection began in 1995 and continues into 2025, encompassing water quality (surface and groundwater), aquatic and terrestrial habitats, air quality, wetlands, wildlife, fisheries, cultural resources, subsistence, socioeconomics, health, mercury, noise, and land use. These studies have informed Project design, environmental controls, and mitigation measures, and provide a reference for ongoing monitoring during operations and closure. Baseline studies completed in support of Project development and ongoing monitoring are identified in Table 20-1.

Baseline data were collected to support Project design, to determine and implement environmental controls to mitigate impacts, and sufficiently characterize the environment in support of permit applications and environmental impact assessments. The environmental baseline data also provides a reference point against which environmental conditions can be evaluated during operations to facilitate early detection of potential changes during Project development and future operation.

Some programs have continued into 2025 for parameters such as ambient air and precipitation, while others, including deep well pump tests, were limited to 2023 due to specific technical and regulatory planning considerations. Baseline fish spawning surveys are generally conducted every other year, with the most recent survey conducted in 2024. A site visit by QP Dockweiler occurred between 18 and 22 August 2025, during which updated data were reviewed to verify ongoing baseline studies and assess current site conditions in alignment with regulatory requirements. While some permits associated with the Project remain under review for extension; none have expired.

Table 20-1: Environmental Baseline Studies (1995–2020)

Baseline Study	1995	1996	1997	1998	1999	2000	2001	2002	2003	2004	2005	2006	2007	2008	2009	2010	2011	2012	2013	2014	2015	2016	2017	2018	2019	2020	2021	2022	2023	2024	2025
Water Quality																															
Surface Water Quality	X	X	X	X	X	X		X	X	X	X	X	X	X	X	X									X	X	X	X		X	
Groundwater Quality									X	X	X	X	X	X	X	X															
Deep Well Pump Testing to evaluate streambed hydraulic conductivity																			X										X		
Air Quality																															
Meteorological Data	X	X	X	X	X	X			X	X	X	X	X	X	X	X									X	X	X	X	X	X	X
Precipitation				X	X	X				X	X	X	X	X	X	X											X				
Ambient Air									X	X		X	X	X	X	X											X				X
SODAR														X	X																
Aquatic Studies																															
Biomonitoring/Tissue Sampling			X	X						X	X	X	X	X	X	X										X	X	X		X	
Spawning Surveys	X	X	X	X						X	X	X	X	X	X	X											X	X		X	
Resident Fish Surveys		X	X	X						X	X	X	X	X	X	X				X	X					X	X	X		X	
Terrestrial Wildlife																															
Habitat Mapping										X	X		X			X															
Wildlife Surveys										X		X	X	X	X	X															
Avian Studies																															
Avian Surveys										X		X	X	X	X	X															
Wetlands Program																															
Wetlands Delineation		X	X	X					X	X	X	X	X	X	X	X	X	X	X	X		X	X								
Waste Rock Characterization																															
Acid Rock Drainage/Metal Leaching Studies			X	X	X					X	X	X	X	X	X	X	X	X	X	X	X	X	X	X	X	X	X	X	X	X	X
Cultural Studies																															
Cultural Site Surveys										X		X	X	X	X	X															
Socioeconomics									X			X	X	X	X														X		
River Studies																															
River Use/Fishing Activity Surveys												X	X	X	X	X				X	X	X	X		X	X					
Marine/River Wildlife Surveys												X	X	X	X	X															
Erosion Studies												X	X															X	X	X	X
Barge Studies													X																X		
Rainbow Smelt Studies																															
Other Programs																															
Mercury Baseline												X	X	X	X	X											X				
Noise Surveys											X																				

20.2 Permitting

The Project is subject to a rigorous permitting regime at federal, state, and local levels. Major permits obtained include the USACE Section 404/10 permit (with a JROD issued in 2018), the ADEC Integrated Waste Management Permit, Air Quality Control Permit, and APDES permit. Additional permits for water rights, dam safety, and fish habitat are in place or in process.

All permits remain current, with renewals processed as required and no expirations to date. Several permits have been administratively extended, or are under review for extensions consistent with regulatory timelines and Project contingencies. The permitting process has included agency and stakeholder consultation, and the Project is designed to comply with the NEPA Clean Water Act, Alaska Dam Safety Guidelines, and other applicable regulations.

Collecting baseline data in parallel with design and prior to filing permit applications has resulted in a Project that mitigates environmental impacts where practicable and addresses known environmental concerns. Most permits needed for the start of operations have been obtained and are listed in Table 20-2, which also identifies outstanding permits, plans, and approvals. Donlin Gold LLC has maintained communication with regulatory agencies, stakeholder representatives, and recognized technical experts to verify that the required information is collected appropriately, and that baseline data meet permitting requirements while allowing regulators to become familiar with the Project. The comprehensive permitting process for the Project can be divided into three phases, each essential to the establishment of a successful future mining operation:

- *Exploration Stage Permitting*: This process secures approval for exploration drilling, environmental baseline studies, and FS investigations.
- *Pre-application Phase*: This phase is conducted alongside FSs and involves collecting environmental baseline data and engaging in consultations with stakeholders and regulatory agencies.
- *NEPA Process and Formal Permit applications*: These applications initiate formal agency review and analysis of the Project, resulting in the issuance or denial of required permits.

To date, Donlin Gold LLC has completed nearly all permitting phases and is in the process of finalizing remaining state permits and supplementing the environmental analysis under NEPA. As of December 2025, permit requirements were being revisited and extended to align with updated federal, state, and local guidelines. The JROD identified several mitigation measures required under federal land management approval, most of which remain outstanding, as these measures include actions required prior to the start of operations, during operations, and at closure.

Table 20-2: Status of Federal, State, and Local Permits

Agency	Application Type	Facility or Activity	Application Number/ Submittal Date	Approval Date	Status
ADEC	Integrated Waste Management Permit	Waste Management Permit for TSF, WRF, inert landfills, and overburden stockpiles	2017DB0001/ 9/16/2015	1/18/2019; Reissued 6/25/2019	Approved with Modifications Expired 1/17/2024; Application renewal 10/17/2024 (administratively extended while renewal pending)
ADEC	APDES	Treated Water Discharge from Mine Dewatering Activities and Water Management	AK0055867/ 4/4/2017	5/24/2018	Approved. Expired 6/20/2023 (administratively extended on 12/13/2022 while renewal pending)
ADEC	Air Quality Control Construction Permit Application	Air Emissions from Mine Operations	AQ0934CPT02/ 0/15/2015	67/1/2023 Extension of Construction Commencement Deadline Granted 512/5/2024	Approved. Permit has indefinite term as long as construction begins within a fixed timeframe for the permit to remain in place.
ADEC	401 Certification	Donlin Gold Mine	POA-1995-120/ July 2012, Revised in December 2014, August 2015, December 2017	Issued 4/4/2019 Reissued 8/10/2019 reaffirmed 5/13/2022	Approved[1]. Expires 8/31/2038
ADEC	Septic System	-	-	-	-

Agency	Application Type	Facility or Activity	Application Number/ Submittal Date	Approval Date	Status
ADEC	Multisector Stormwater General Permit	-	-	-	-
ADF&G	Fish Habitat Permit Application	American Creek Area Facilities	FH18-III-0191/ 12/29/2017	08/30/2018	Approved. Expires upon removal of Dam
ADF&G	Fish Habitat Permit Application	Anaconda Creek Area Facilities	FH18-III-0190/ 12/29/2017	08/30/2018	Approved. Expires upon Mine Closure and Rehabilitation of the Site
ADF&G	Fish Habitat Permit Application	Snow Gulch Area Facilities	FH18-III-0189/ 12/29/2017	08/30/2018	Approved. Expires upon removal of Dam
ADF&G	Fish Habitat Permit Application	Ruby Creek/Queen Gulch Area Aquatic Mitigation	FH18-III-0192/ 12/29/2017	08/30/2018	Approved. Expires upon Closure
ADF&G	Fish Habitat Permit Application	Alaska Pollution Discharge Elimination System Discharge Point Construction (Crooked Creek)	FH18-III-0188/ 12/29/2017	08/30/2018	Approved. Expires upon removal
ADF&G	Fish Habitat Permit Application	Crooked Creek Bridge Construction	FH18-III-0187/ 10/8/2015	08/30/2018	Approved. Expires upon removal of bridge and rehabilitation of the stream crossing
ADF&G	Fish Habitat Permit Application	Getmuna North Fork Bridge Construction	FH18-III-0186/ 10/8/2015	08/30/2018	Approved. Expires upon removal of

Agency	Application Type	Facility or Activity	Application Number/ Submittal Date	Approval Date	Status
					bridge and rehabilitation of the stream crossing
ADF&G	Fish Habitat Permit Application	Getmuna South Fork Bridge Construction	FH18-III-0185/ 10/8/2015	08/30/2018	Approved. Expires upon removal of bridge and rehabilitation of the stream crossing
ADF&G	Fish Habitat Permit Application	Unnamed Getmuna South Tributary Bridge Construction	FH18-III-0184/ 10/8/2015	08/30/2018	Approved. Expires upon removal of bridge and rehabilitation of the stream crossing
ADF&G	Fish Habitat Permit Application	Fish Habitat Permit – Lower Jungjuk Creek Bridge Construction	FH18-III-0182/ 10/8/2015	08/30/2018	Approved. Expires upon removal of bridge and rehabilitation of the stream crossing
ADF&G	Fish Habitat Permit Application	Fish Habitat Permit – Upper Jungjuk Creek Bridge Construction	FH18-III-0183/ 10/8/2015	08/30/2018	Approved. Expires upon removal of bridge and rehabilitation of the stream crossing
ADF&G	Fish Habitat Permit Application	Jungjuk (Anyaruaq) Port Wharf Construction	FH18-III-0181/ 10/8/2015	08/30/2018	Approved.
ADF&G	Title 16 Permits	Pipeline	-	-	Not complete.
ADF&G	Special Use Permits	Pipeline Infrastructure in Susitna Flats State Game Refuge	2016 (initial Submittals)	-	In Process. Preparing Revised Applications
ADNR	Certificate of Approval to Construct a Dam	7 dams within the Mine Area	4/12/2013	-	In Process. Initial applications submitted

Agency	Application Type	Facility or Activity	Application Number/ Submittal Date	Approval Date	Status
ADNR	Water Rights Application	TSF Interceptor and Seepage Collection Wells	LAS 29175/ 5/16/2013	Permit to Appropriate Water issued 4/26/2021	Approved. Statement of Beneficial Use or extension request must be submitted by 4/25/2031.
ADNR	Water Rights Application	Upper CWD	LAS 29168/ 5/16/2013	Permit to Appropriate Water issued 4/26/2021	Approved. Statement of Beneficial Use or extension request must be submitted by 4/25/2031.
ADNR	Water Rights Application	Snow Gulch Fresh Water Dam	LAS 29169/ 5/16/2013	Permit to Appropriate Water issued 4/26/2021	Approved. Statement of Beneficial Use or extension request must be submitted by 4/25/2031.
ADNR	Water Rights Application	Lower CWD	LAS 29170/ 5/16/2013	Permit to Appropriate Water issued 4/26/2021	In Process. Water rights were released for public comment on 11/20/2020.
ADNR	Water Rights Application	Jungjuk (Anyaruaq) Port Site Surface Water-Kuskokwim River	LAS 29171/ 5/16/2013	Permit to Appropriate Water issued 4/26/2021	Approved. Statement of Beneficial Use or extension request must be submitted by 4/25/2031.
ADNR	Water Rights Application	Construction Camp/Shop, Office, Warehouse, & Process	LAS 29172/ 5/16/2013	Permit to Appropriate Water issued 4/26/2021	Approved. Statement of Beneficial Use or extension request must

Agency	Application Type	Facility or Activity	Application Number/ Submittal Date	Approval Date	Status
					be submitted by 4/25/2031.
ADNR	Water Rights Application	Pit Perimeter & In-Pit Dewatering Wells & Drains	LAS 29173/ 5/16/2013	Permit to Appropriate Water issued 4/26/2021	Approved. Statement of Beneficial Use or extension request must be submitted by 4/25/2031.
ADNR	Water Rights Application	Getmuna Creek Surface Water	LAS 29174/ 5/16/2013	Permit to Appropriate Water issued 4/26/2021	Approved. Statement of Beneficial Use or extension request must be submitted by 4/25/2031.
ADNR	Water Rights Application	Jungjuk (Anyaruaq) Port Site Well	LAS 29176/ 5/16/2013	Permit to Appropriate Water issued 4/26/2021	Approved. Statement of Beneficial Use or extension request must be submitted by 4/25/2031.
ADNR	Water Rights Application	TSF	LAS 29177/ 5/16/2013	Permit to Appropriate Water issued 4/26/2021	Approved. Statement of Beneficial Use or extension request must be submitted by 4/25/2031.
ADNR	Water Rights Application	Permanent Camp Potable Water Well Field	LAS 29178/ 5/16/2013	Permit to Appropriate Water issued 4/26/2021	Approved. Statement of Beneficial Use or extension request must

Agency	Application Type	Facility or Activity	Application Number/ Submittal Date	Approval Date	Status
					be submitted by 4/25/2031.
ADNR	Water Rights Application	Crooked Creek Surface Flows and Associated Diversion Structures	LAS 31477/ 9/23/2016	Permit to Appropriate Water issued 4/26/2021	Approved. Statement of Beneficial Use or extension request must be submitted by 4/25/2031.
ADNR	Application for Pipeline Right-of-Way Lease	Natural Gas Pipeline ROW (State of Alaska Lands)	ADL 231908/ 4/9/2014	1/17/20, reaffirmed 7/19/2021.	Approved. Expires 2/11/2050 (may be extended or up to an additional 30 years).
ADNR	Application for Easement	Fiber Optic ROW (State of Alaska Lands)	ADL 232368/ 11/16/2015	1/02/2020	Approved. Construction start deadline extended to February 2, 2030
ADNR	Application for Lease or Purchase of State Land	Airstrip Land Lease (on State Land)	ADL 232199/ 10/9/2015	1/02/2020	Approved. Construction start deadline extended to February 2, 2030
ADNR	Application for Lease or Purchase of State Land	Land Lease for Submerged State of Alaska Lands at Jungjuk (Anyaruaq)	ADL 232200/ 10/9/2015	1/02/2020	Approved. Construction start deadline extended to February 2, 2030
ADNR	Land Use Permit Application	Temporary Access Road to Donlin-Jungjuk Road Material Site-8	ADL 232361/ 5/19/2016	1/02/2020	Approved.

Agency	Application Type	Facility or Activity	Application Number/ Submittal Date	Approval Date	Status
ADNR	Land Use Permit Application	Temporary Access Road to Donlin-Jungjuk Road Material Site-16	ADL 232366/ 5/19/2016	1/02/2020	Approved.
ADNR	Application for Easement	Donlin-Jungjuk Road (State Land Portions)	ADL 232346/ 10/9/2015	1/02/2020	Approved. Construction start deadline extended to February 2, 2030
ADNR	Material Sites and Reclamation Plan	Donlin-Jungjuk Road Material Site-04	ADL 232334/ 10/9/2015	1/02/2020	Approved. Expires 1/02/2050
ADNR	Material Sites and Reclamation Plan	Donlin-Jungjuk Road Material Site-08	ADL 232335/ 10/9/2015	1/02/2020	Approved. Expires 1/02/2050
ADNR	Material Sites and Reclamation Plan	Donlin-Jungjuk Road Material Site-09	ADL 232336/ 10/9/2015	1/02/2020	Approved. Expires 1/02/2050
ADNR	Material Sites and Reclamation Plan	Donlin-Jungjuk Road Material Site-10	ADL 232337/ 10/9/2015	1/02/2020	Approved. Expires 1/02/2050
ADNR	Material Sites and Reclamation Plan	Donlin-Jungjuk Road Material Site-12	ADL 232338/ 10/9/2015	1/02/2020	Approved. Expires 1/02/2050
ADNR	Material Sites and Reclamation Plan	Donlin-Jungjuk Road Material Site-13	ADL 232339/ 10/9/2015	1/02/2020	Approved. Expires 1/02/2050
ADNR	Material Sites and Reclamation Plan	Donlin-Jungjuk Road Material Site-16	ADL 232340/ 10/9/2015	1/02/2020	Approved. Expires 1/02/2050
ADNR	Reclamation and Closure Plan Approval	Donlin Gold Project	A20196226	1/18/2019 re-affirmed 06/25/2019	Approved. Expired 1/17/2024 (administratively extended while renewal pending)

Agency	Application Type	Facility or Activity	Application Number/ Submittal Date	Approval Date	Status
ADNR	Reclamation Plan Approval	Donlin-Jungjuk Road Material Site-01	LAS 30533/ 10/9/2015	1/18/2019 Re-affirmed 06/25/2019	Approved.
ADNR	Reclamation Plan Approval	Donlin-Jungjuk Road Material Site-02	LAS 30534/ 10/9/2015	1/18/2019 Re-affirmed 06/25/2019	Approved.
ADNR	Reclamation Plan Approval	Donlin-Jungjuk Road Material Site-03	LAS 30535/ 10/9/2015	1/18/2019 Re-affirmed 06/25/2019	Approved
ADNR	Reclamation Plan Approval	Donlin-Jungjuk Road Material Site-05	LAS 30536/ 10/9/2015	1/18/2019 Re-affirmed 06/25/2019	Approved
ADNR	Reclamation Plan Approval	Donlin-Jungjuk Road Material Site-06	LAS 30537/ 10/9/2015	1/18/2019 Re-affirmed 06/25/2019	Approved.
ADNR	Reclamation Plan Approval	Donlin-Jungjuk Road Material Site-07	LAS 30538/ 10/9/2015	1/18/2019 Re-affirmed 06/25/2019	Approved.
ADNR	Public Easement Re-location	Donlin Mine. Involves relocation of existing public ROWs to account for the need to restrict public access in certain areas.	Draft Application Summer 2020	-	In Process. Permit obtained on 1/20/2023 Alternative access not constructed yet.
BLM	Application for Transportation and Utility Systems and Facilities on Federal Lands	Natural Gas Pipeline ROW (Bureau of Land Management Lands)	AA-92403/ 3/11/2010	8/13/2018	Approved[2]. Expires 12/31/2048 (may be renewed)
BLM	Application for Transportation and Utility Systems and	Fiber Optic ROW (Bureau of Land Management Lands)	AA-92403/ 1/8/2014	01/21/2025-	Approved. Expires 01/01/2032

Agency	Application Type	Facility or Activity	Application Number/ Submittal Date	Approval Date	Status
	Facilities on Federal Lands				
BLM	Port ANCSA 17(b) Easement Modification	Mine and Transportation Area	-	-	Completed.
DOT Pipeline and Hazardous Materials Safety Administration (PHMSA)	Special Permit	Natural Gas Pipeline Special Permit Strain-based Design	PHMSA-2016-0149/ 11/11/2016	6/5/2018	Approved. No Expiration Date
USACE	Individual 404	Mine Site, Transportation, and Pipeline Facilities	-	-	Approved[1]. Expires 8/31/2038
Cook Inlet Region, Inc	Pipeline Authorization on Cook Inlet Region, Inc. (CIRI) Lands	Construction Easement for Pipeline	2020	-	In Process

Source: Wood, Internal Permit Matrix Table, prepared by Geosyntec Consultants, Inc., December 31, 2025.

Note: ANCSA: Alaska Native Claims Settlement Act

(1) In *Orutsaramiut Native Counsel v. Alaska Department of Environmental Conservation*, Case No. 3AN-21-060502CI, plaintiff challenged ADEC's issuance of Water Quality Certification in the Superior Court for the State of Alaska, Third Judicial District Anchorage. In May 2025, the Court affirmed ADEC's issuance of the Water Quality Certification. Plaintiff appealed this decision to the Alaska Supreme Court on May 28, 2025, and briefing is in progress.

(2) In *Orutsararmuit Native Counsel, et al., v. USACE, et al.,* Case No. 3 : 23-cv-00071-SLG, Plaintiffs challenged the issuance of this permit in Federal District Court for the District of Alaska. On June 10, 2025, the Court remanded, without vacating the permits, to the federal agencies to supplement the analysis conducted under NEPA to evaluate the environmental effects of a larger hypothetical release from the TSF. On October 24, 2025, The Federal Permitting Improvement Steering Council included the project in the Fast-41 Program. Resulting from litigation from challenging the issuance of the BLM's and USACE's JROD authorizing issuance of the ROW and Section 404 permits for the Project on June 10, 2025, the U.S. Federal District Court for the District of Alaska, remanded to the BLM and USACE to supplement the EIS for the project and reaffirm, modify, or rescind the prior record of decision. The permits remain in good standing during the pendency of the SEIS.

In addition, the EIS remains under appeal, and a supplemental EIS has been mandated to assess potential impacts of a hypothetical tailings release larger than the one already evaluated in the EIS. BLM and ADNR have issued right-of-way (ROW) authorizations for the natural gas pipeline. Furthermore, the Alaska Department of Fish and Game (ADF&G) Title 16 Permit for a natural gas pipeline and Native Land authorizations of the pipeline are not complete. These in-progress permits are specific to energy and natural gas, posing a potential risk for the Project. Supply of natural gas in Alaska is limited, and there are concerns regarding how the Project will solve the issue of placing additional demand on the limited natural gas supply.

The Project is working to secure a land agreement from Cook Inlet Region, Inc., (CIRI) for the remaining piece of the ROW for the natural gas pipeline. CIRI and Donlin Gold LLC have agreed on the terms of this lease and are working on finalizing the agreement. Other operational permits will be required immediately prior to construction, such as stream crossing from ADF&G, water sources from ADNR, and development from the Matanuska-Susitna borough. Legal challenges to ADNR's issuance of the ROW are also complete with the Alaska Supreme Court issuing a ruling in November upholding ADNR's issuance of the ROW.

There are several projects proposed and/or in the development phase to address the natural gas supply limitation for the rail belt, including the Alaska liquified natural gas project, which would bring natural gas from the North Slope to the rail belt as well as various proposed liquid natural gas import developments. As a large industrial off-taker, Donlin Gold LLC has the potential to positively affect the price of natural gas for consumers in the rail belt under any of these supply scenarios. Donlin Gold LLC is actively engaged with all these potential projects.

20.2.1 Exploration Stage Permitting

Through advanced exploration and feasibility engineering, numerous permits or authorizations have been issued by federal agencies, state agencies, and other organizations to support ongoing operations. These include the following:

- Permits issued or modified by USACE
- Three permits/authorizations issued or reviewed by the United States Environmental Protection Agency (EPA)
- Six land use or ROW permits from BLM
- 22 mineral exploration or temporary water use permits issued or modified by the ADNR
- Eight permits certified, issued, or modified by the ADEC
- Eight Fish Habitat (Title 16) Stream Crossing Permits issued or modified by ADF&G.

20.2.2 Pre-application Phase

Donlin Gold LLC has conducted a comprehensive stakeholder interaction and consultation process (see Section 20.5.1). This process was an important component of the pre-permit application phase of the Project and has been crucial to successful completion of the permitting and NEPA processes.

An equally important component of this phase has been ongoing interaction with permitting agencies and individual regulators who have been responsible for reviewing the Project's permit applications. This process began in 1995 when Placer Dome Inc., then the Project operator, first met with State of Alaska regulators and industry personnel to develop an understanding of the regulatory process in the state.

The concept of the ADNR Large Mine Permitting Team (LMPT) was developed in the process of permitting the Fort Knox gold mine near Fairbanks in the 1990s, and this model has been used successfully in the permitting of all major mining projects in the state since that time. Under this concept, a company with a major project can enter into a reimbursable services agreement (RSA) with ADNR (and separately with ADEC) to compensate the state for expenses incurred in permitting and consultation activities related to the Project. ADNR is responsible for administering the RSA, and the LMPT coordinator assigned to the Project is responsible for coordinating all state regulatory input and for coordinating with federal regulatory personnel. This process is specifically structured to be implemented early in the Project life so that agencies can provide input into project design and baseline data collection ahead of the permitting and NEPA process.

Baseline data collection and management were conducted under documented protocols consistent with regulatory standards. Environmental and technical datasets were compiled into a centralized digital archive with version control and standardized metadata to maintain traceability. This archive served as the repository for all project-related information and was accessible to authorized personnel for regulatory review Quality assurance and quality control procedures were applied throughout, and all data handling steps were recorded to provide an auditable record of compliance with applicable requirements.

The LMPT coordinator works with the company to organize interagency project update meetings and identify required expertise in regulatory agencies. The coordinator also works with stakeholders to provide mining education and to address questions about the state regulatory process. Formal Project consultation with the LMPT began in 2003, and the first RSA was implemented in 2004. Since that time, state regulators have been continuously involved in reviewing Project data collection, participating in Project update meetings, providing input to

Project design decisions, participating in stakeholder meetings related to the Project throughout the Kuskokwim region, and providing interagency coordination of state permitting activities.

In addition to the LMPT interaction, Project personnel have regularly consulted with federal agency personnel, principally with the EPA, USACE, United States Fish and Wildlife Service (USFWS), and BLM. Meetings have also been held with the National Marine Fisheries Service, and the United States Coast Guard.

20.2.3 The NEPA Process and Permit Applications

In July 2012, Donlin Gold LLC submitted an application to the USACE for a permit pursuant to Section 10 of the Rivers and Harbors Act of 1899 and Section 404 of the Clean Water Act. Permits issued by federal agencies constitute "federal actions," which triggers review under NEPA. Following the initiation of the permitting process, USACE issued a Notice of Intent (NOI) to prepare an EIS.

The NEPA review evaluates all elements of the Project, their cumulative impacts, and alternatives to the proposed action. The review also identifies mitigation measures. The federal agency with the primary permit is designated as the lead for the NEPA process; for this Project, USACE served as the lead agency, while the JROD was issued as a joint effort between USACE and BLM.

The Notice of Availability for the Final EIS was published in the Federal Register on 27 April 2018, and the JROD was issued 13 August 2018. The JROD supports issuance of the permit for the North Route Pipeline Option (natural gas pipeline) referred to as Alternative 2 in the EIS. The EIS and JROD describe the conditions of the decision to issue the permit and explain the basis for the decision.

Each federal and state permit will have compliance stipulations or mitigation requirements requiring review and possibly negotiation by the applicant and appropriate agency. A list of some of the mitigation requirements identified through the NEPA process is provided in Table 20-3. The full list of mitigation requirements is found in the JROD and associated state permits.

Table 20-3: Key Environmental Issues and Proposed Mitigation Measures

Issue	Mitigation Measures
Mercury	• Collect detailed mercury baseline information; Kuskokwim region has naturally elevated mercury levels • Complete modeling studies using local, regional, and global mercury measurements • Implement state-of-the-art engineered mercury abatement controls for processing operations (emission controls on autoclave, carbon regeneration kiln, electrowinning cells, mercury retort) • Engage in stakeholder discussions in ongoing community outreach efforts.
Cyanide	• Adhere to International Cyanide Management Code (ICMC) for transport and use • Use purpose-built stainless-steel International Standards Organization (ISO) containers for solid sodium cyanide briquettes • Incorporate cyanide detoxification system and containment areas • Implement monitoring plans • Begin ICMC adherence to the start of operations.
Water Quantity and Quality	• Address specific water management areas (detailed water balance model, containment structures, freshwater diversion structures, wastewater treatment systems, lined TSF, and flexibility water management designs) • Conduct temperature monitoring of Crooked Creek surface waters and adjust stream temperature to protect fish spawning as needed using an adaptive management approach • Provide wastewater treatment in perpetuity following pit lake formation and development.
Waste Rock and Stormwater Management	• Develop a sampling and analysis plan to ensure PAG rock and other sources of contaminants are not used for construction at the mine or for road surfacing where such construction could lead to surface water quality impacts • Classify and segregate waste rock; backfill high acid rock drainage potential material below water table • Construct dedicated cells within the WRF for waste that cannot be blended or backfilled • Develop a comprehensive plan for managing arsenic leaching from waste rock, including capturing and treatment of arsenic impacted waters. • Practice concurrent reclamation over the WRF • Capture all mine drainage and all stormwater runoff in a contact water pond or treat prior to discharge • At closure, collect and manage all runoff and seepage in the pit.

Issue	Mitigation Measures
Hazardous Waste	• Identify and map potentially contaminated soil prior to pipeline construction; avoid contaminated soil in final grading.
Cultural Resources	• Conduct preconstruction cultural surveys to locate cultural sites, and avert cultural sites, or reduce the Project footprint to avoid or minimize impacts to cultural sites as practicable • Mitigate impacts to known cultural sites, for instance by implementing a narrower construction ROW and using horizontal directional drilling under a sensitive site • Follow the National Historic Preservation Act Section 106 Programmatic Agreement, which includes but is not limited to the Cultural Resource Management Plan, the Treatment of Historic Properties, and procedures for the Treatment of Human Remains.
Transportation	• Incorporate a natural gas pipeline for site power generation • Use shipping containers that meet or exceed industry standards to transport hazardous materials • Use custom double-hull barges to transport fuel • Implement modern scheduling and navigation aids on the Kuskokwim River • Develop detailed ocean, river, and land spill response plans and train personnel • Develop communication strategies to provide timely advance warning of barge passage and recreational/subsistence users and to establish a method for conflict resolution on the river • Provide clear signage distinguishing trails from the pipeline ROW.
Flora and Fauna	• Use HDPE or synthetic netting material, regular inspection, and sampling protocols • Avoid vegetation clearance during bird nesting seasons • Employ seasonal timing restrictions on blasting to reduce noise during sensitive subsistence hunting activities • Implement an aquatic resource monitoring plan and the Rainbow Smelt Monitoring Program.

Issue	Mitigation Measures
Water Resources, Wetlands. & Riparian Areas	• Avoid wetlands and riparian areas where practicable • Follow construction best management practices, such as storing wetland topsoil to be used in temporarily impacted areas • Implement a compensatory mitigation plan and associated requirements, such as deed restrictions, financial assurances, and performance standards • Restore, re-establish, enhance, or preserve more than 80 ha of wetlands, riparian areas, stream channels, and upland buffers in the upper reaches of the Crooked Creek watershed, which will enhance aquatic habitat, remove barriers to fish passage, and improve anadromous and resident fish-rearing habitat in some reaches affected by historical placer mining • Preserve restored wetlands and aquatic habitat by creating riparian buffers around the restoration areas • Preserve a parcel of land in exchange for pipeline construction impacts, which will protect 2,376 ha, including 1,323 ha of wetlands and ponds, and 169 ha of streams and rivers, totaling 1,492 ha of Waters of the United States • Preserve 883 ha of upland riparian area and buffers, and 78,655 linear meters of streams in the Chuitna Watershed • Purchase 3.6 Riverine released credits and 6.2 Slope wetland release credits from Great Land Trust In-Lieu Fee Program for the loss of 0.7 ha of riverine and 1.1 ha of slope wetlands from the pipeline construction in the Matanuska-Susitna Borough.

The NEPA application process was completed, and the federal permits for the BLM ROW and the USACE 404 permit were issued in 2018. All permits have been maintained since 2021 with extensions and renewal applications. On 5 April 2023, Earthjustice, representing Orutsararmiut Traditional Native Council (ONC), Kwethluk, and Tuluksak filed suit in the United States Federal Court for the District of Alaska, challenging BLM's and the USACE's issuance of the JROD. The court order required the agencies to conduct a supplemental analysis of the environmental impacts of a larger hypothetical release from the TSF; however, these federal permits were not vacated and remain valid while the agencies conduct this analysis. This supplemental analysis has been included in the FAST-41 process, with the agencies submitting their timeline for completion to the Federal Permitting Council by 23 December 2025. This process should be completed within 18 months (Donlin Gold Project/Fast-41). Donlin Gold LLC has started working on the supplemental briefing while a Notice of Intent is expected from the Army Corps of Engineers in early 2026.

20.2.4 Laws, Regulations, and Permit Requirements

The federal and state permits and authorizations of Donlin Gold LLC have been updated for 2025, and the status of each permit can be viewed in Table 20-2.

20.3 Key Environmental Issues and Mitigation

Environmental concerns include tailings decant water from the process plant as they will likely contain elevated levels (at or above the drinking water or aquatic life standards) of water-soluble antimony, arsenic, manganese, mercury, molybdenum, and selenium. Sulfates at levels greater than 10,000 mg/L are most likely from the presence of magnesium, which increases the solubility of sulfate. This results in principal environmental issues that would include mercury management, cyanide use, water resources, waste rock and stormwater management, hazardous waste, cultural resources, and impacts on flora and fauna.

To mitigate these hazards, closure and reclamation plans have been developed for all major facilities, with financial assurance and a trust fund established to cover long-term obligations. An updated monitoring plan was developed in 2022 that incorporates surface water monitoring and sampling and a focused monitoring strategy for mercury for the Project and temperature monitoring of Crooked Creek (Donlin Gold LLC, 2022b). The Reclamation Plan received approval from the ADNR and ADEC in January 2019 and will be updated throughout the mine life. Additionally, both BLM and State of Alaska have authorized the natural gas pipeline ROW for portions of federal and state lands.

Waste will be stored in a single ex-pit WRF, located in the American Creek Valley. Surface ditches, a contact water dam, and diversion systems will control the surface water in the pit and WRF areas. Groundwater will be managed with dewatering systems in vertical dewatering wells, both in the pit and around the WRF areas, horizontal drains, and sump pumps. The Project incorporates engineered controls, such as a fully lined TSF, WTPs for both operations and closure, robust mercury abatement systems, and water and waste management practices. The post-closure WTP will result in the planned treatment of post-closure water to meet applicable water quality standards (Section 20.6.1).

The key environmental issues of concern and a portion of the mitigation measures are summarized in Table 20-3. These issues have been analyzed and addressed during the development of the EIS and the issuance of the JROD, which includes several mitigation measures to address potential environmental impacts. Potential issues are managed and mitigated through a combination of baseline data collection, rigorous engineering and project design, construction practices, targeted mitigation measures, and comprehensive public consultation.

20.4 Waste and Tailings Disposal and Water Management

Refer to Section 18 for plans on waste and tailings disposal, site monitoring, and water management during operations. Refer to Section 20.6 for plans on waste and tailings disposal, site monitoring, and water management post mine closure.

20.5 Considerations of Social and Community Impacts

Donlin Gold LLC has engaged in sustained, culturally sensitive consultation with Alaska Native Regional Corporation (Calista), Alaska Native Village Corporation (The Kuskokwim Corporation), tribal councils, and other regional stakeholders.

The Project has conducted baseline data studies to understand and gain respect for traditional subsistence activities, local workforce development, business opportunities, and community development project. Stakeholder mapping and ongoing engagement are utilized for Project planning, permitting, and operations. The NEPA process includes a comprehensive social baseline and impact assessment, and the Project is committed to sustainable development and positive legacy outcomes for the Yukon-Kuskokwim region. Donlin Gold LLC is focusing on sustainable development to benefit local communities over the long term by providing opportunities for direct employment, local procurement, and community development projects. Associated with these examples are efforts to develop lasting capacities that will continue after mine closure. The following stated principles, which underpin Donlin Gold LLC's approach to community engagement activities, have been actively applied since early exploration and will reportedly continue throughout the life cycle of the mine:

- Engage with communities in a respectful and culturally sensitive manner
- Develop long-term mutually beneficial relationships
- Be responsive to stakeholders' concerns and questions
- Build trust and confidence through accountability and transparency
- Understand the complex interests among diverse communities
- Adapt Project activities to fit with local needs and contexts
- Plan activities with closure in mind
- Monitor results and impacts.

Donlin Gold LLC continues to strengthen its relationship with the Yukon-Kuskokwim region communities by actively involving local communities in multiple aspects of the Project. Over the years, many direct hires for field programs, as well as full-time employees, have been Alaska Native. Donlin Gold LLC has also addressed community concerns by including Project design changes that avoided or minimized concerns regarding subsistence and barging issues, through the following:

- Incorporation of the natural gas pipeline into the mine plan to reduce the amount of barging in the Kuskokwim River
- Development of a barge communication plan to prevent and minimize potential conflicts between subsistence users and barge operators
- Establishment of a Subsistence Advisory Committee with residents to advise the Project on subsistence issues
- Initiation of steps with landowners to develop a subsistence plan to address access to subsistence resources around the mine site as well as a subsistence policy for the Project.

Donlin Gold LLC and the Alaska Native Corporation landowners participate in engagement meetings with individual stakeholders, local community members and in ongoing community engagement in environmental management, safety, training, education, health, and cultural initiatives. Donlin Gold LLC maintains a positive relationship with the Yukon-Kuskokwim regional communities and includes local communities regarding the Project. Donlin Gold LLC has a community relations team focused on concerns and issues facing the local Tribal communities.

Donlin Gold LLC recognizes cultural awareness is essential to help identify all stakeholders, including potentially vulnerable minority groups. Donlin Gold LLC is led by the principles in the International Council on Mining & Metals Position Statement on Mining and Indigenous Peoples

(ICMM) promoting constructive relationships between the mining and metals industry and indigenous peoples based on respect, meaningful engagement, and mutual benefit.

The Project's approach has been to build trust and mutually beneficial relationships to guide the development of mitigation plans and to manage risks in a responsible manner. Since the completion of the EIS and issuance of the JROD, mitigation measures related to stakeholders have been defined. Donlin Gold LLC plans to continue engagement with stakeholders throughout the development of the mine and during operations, closure, and post-closure.

As the Project progresses, Donlin Gold LLC will continue to focus on developing programs that benefit local communities to include improved infrastructure, support for education and health services, cultural heritage preservation, employment and business opportunities, increased income flows through royalty streams and compensation payments, and environmental restoration and protection.

20.5.1 Stakeholders

The region has a complex political and social structure, represented by a diverse group of social, business, and governmental entities. Relationships between these various entities are often complex and are influenced by competing political and economic interests.

Entities within the region can be split into two primary categories: nonprofit organizations (tribal/cultural/social) and for-profit corporations. Tribal organizations played an important role in the NEPA process, as the NEPA process included government-to-government consultation with the federal agency leading the permitting. Calista and TKC, the two primary Native business entities of the Yukon-Kuskokwim region, each hold a financial interest in the Project. Several other Native business entities and associations are also engaged, given the Project's potential regional impacts.

The Project is anticipated to deliver distinct benefits to both for-profit Native corporations and nonprofit tribal organizations. For-profit entities, such as Calista and TKC, are positioned to gain through economic participation, revenue streams, and expanded business opportunities tied to the Project. Nonprofit organizations, which focus on cultural preservation, social services, and community well-being, are expected to benefit indirectly through increased employment opportunities, enhanced community and regional economic growth. Collectively, these outcomes support long-term economic development, community health, and cultural resilience across the Yukon-Kuskokwim region.

With the addition of the proposed natural gas pipeline, the area of influence expanded beyond the region, bringing a wider group of stakeholders into the process. A Project-wide stakeholder database is maintained to help manage Project communications and continue ongoing consultation efforts.

To successfully acquire the support and "social license" required to develop and operate the Project, Donlin Gold LLC will continue the process of ongoing engagement and consultation with stakeholders throughout the remaining permitting process, construction, operation, closure, and post-closure of the Project.

20.6 Closure Plan Considerations

Reclamation will begin during construction with the stabilization of topsoil and overburden stockpiles and cut-and-fill slopes. Major reclamation activities following construction will focus on material borrow sites and areas where disturbed wetlands can be restored and habitats enhanced. Additional reclamation will occur concurrently with mining and continue through the cessation of mining and processing operations. Area and component specific reclamation plans will be refined throughout the life of the Project and as it approaches mine closure. Near the end of the mine life, alternatives will be evaluated and incorporated into the final reclamation plan where appropriate.

The closure water management strategy focuses on safely handling runoff, seepage, and consolidation of water while ensuring compliance with Alaskan water quality standards (AWQS) and preventing uncontrolled discharges.

The lake to be formed in the ultimate pit will serve as the central feature of post-closure water management for the foreseeable future. The ACMA and Lewis pits will partially backfill during operations, then form a pit lake during closure. Backfill elevations will reach approximately 64 m amsl. An emergency spillway will be constructed before the lake nears capacity with an invert elevation of 109.5 m amsl and designed for a PMF of 320 m^3/s. The spillway will prevent fish passage and uncontrolled flow with Crooked Creek.

Surface water and groundwater monitoring systems for process components will remain in place for up to 30 years or longer, depending on compliance history, and until each facility has been stabilized both physically and chemically to the satisfaction of the applicable state and federal regulatory agencies.

Once physical reclamation begins, temporary diversions and sedimentation control systems will be installed and routinely monitored. These systems will be cleaned, repaired, and modified as needed. Long-term or permanent diversions, water treatment, physical barriers, and signage will be monitored and maintained as needed until all closure standards are met, reclamation surety has been released, and property management reverts to the landowners.

Final closure determinations and the release of any remaining financial assurances require agreement from the ADNR and the Alaska Department of Environmental Conservation (ADEC). The BLM is only responsible for approving the pipeline component (SRK, 2018).

20.6.1 Water Treatment Plant

A post-closure WTP will be built at the site. The proposed post-closure WTP is based on the projected pit lake water quality and design water flows. Current information regarding forecasted water quality and water quantity has resulted in the planned treatment of post-closure water to meet applicable water quality standards. The design is based on a high-density sludge chemical precipitation process for removal of metals including arsenic, manganese, and selenium. The process includes reactor tanks for lime pH adjustment/precipitation, iron coagulant addition, clarifiers, pH adjustment, and filtration. The WTP would have a design treatment rate of 1,500 m^3/h and a maximum capacity of approximately 1,700 m^3/h and operate approximately 6 months per year with an average annual flow to Crooked Creek of approximately 662 m^3/h. The final WTP configuration will be updated closer to the end of the mine life to incorporate advances in treatment technologies (SRK, 2017).

The influent flow and water quality entering the closure WTP are determined using a hydrodynamic-geochemical pit lake model created by Lorax in their technical memorandum for the revised water management plan (Lorax, 2012). This model predicts both the rate of dewatering and the water quality of the pit lake, which serves as the source for the WTP. At the time of pit lake overflow, modeled concentrations of antimony, arsenic, manganese, mercury, and selenium are projected to exceed the most stringent AWQS.

The sludge from the post-closure WTP will be chemically stable and will be sent to the bottom of the pit lake for final disposal. More testwork will be conducted as the Project approaches the end of operations to support the final design, flowsheet selection, and permitting of the post-closure WTP. The results of ongoing bench and pilot testing, in addition to operational water treatment data, will be used to update the water treatment process design for the post-closure WTP.

20.6.2 Tailings Storage Facility

Before the end of operations, the tailings deposition procedure will be modified so that the operating pond will be directed to the southeast corner of the TSF. The TSF will be reclaimed with a multilayer cover system: 0.35 m of peat/organic growth medium over 0.3 m of colluvium or terrace gravel, and 1.0 m of competent rockfill to provide a capillary break. The TSF surface will be contoured to direct runoff toward the southeast corner, where a lined collection pond (capacity ~200,000 m^3) will store runoff for testing before discharge. A spillway will be excavated in the ridge between Anaconda Creek and Crevice Creek catchments, with an invert elevation of 254 m amsl and designed for a PMF of 380 m^3/s. Discharge to Crevice Creek will occur only after water meets applicable standards, which is anticipated to be around Year 10 post-closure (SRK, 2016).

During the first year of closure, TSF pond water will be pumped to the pit via the reclaim pipeline. Over 5 years, the tailings surface will be progressively covered, while pumping continues to prevent pond redevelopment. From Years 6 to 52, water from tailings consolidation and infiltration will be collected via drain sumps in manholes and pumped to the pit. Approximately 16.28 mm^3 of void water is expected during consolidation. The SRS downstream of the TSF will remain active, pumping seepage (estimated at 102 m^3/h) to the pit lake until water quality meets standards.

20.6.3 Waste Rock Facility

The WRF will be in the American Creek Valley, east of the open pit (Hanson et al., 2021a). The ultimate footprint of the WRF covers an area of approximately 9 km^2. Approximately 2,318 Mt of mine waste will be placed in the WRF. The top lift of the WRF will be at an elevation of approximately 610 m amsl, resulting in a maximum WRF height of about 430 m. Most of the WRF will be constructed in 30 m lifts. The toe of each subsequent lift will be set back 47 m from the crest of the previous lift, resulting in an overall dump slope of 3H:1V (SRK, 2016).

The WRF has been designed to maximize concurrent reclamation, minimize the effects of PAG materials, add flexibility to the site's water balance, and optimize the cost of closure. The WRF will be constructed entirely from the bottom up. During its development, organic materials, loess, and ice-rich overburden will be stripped from the footprint as the dump expands. These stripped materials will either be placed in temporary overburden stockpiles or mixed with waste rock within the WRF. The stripped areas will be replaced with coarse waste rock, ensuring a stable foundation and minimizing the risk of instability during early construction and throughout the WRF’s operational life (SRK, 2016).

The facility will be constructed in lifts and will be reclaimed as soon as practical after each lift or portion of the facility is complete. After active dumping ceases on each lift, the slopes will be regraded to an overall 3H:1V slope or flatter (Hanson et al., 2021). The WRF will be progressively reclaimed during operations, covering approximately 970 hectares. The reclamation cover will be designed to minimize water infiltration and will include a minimum of 0.35 m of peat-mineral mix as a growth medium placed over at least 0.3 m of terrace gravel or colluvium. The growth medium will be vegetated to enhance stability and reduce infiltration.

A series of channels is required to collect and convey runoff from the surface of the reclaimed WRF to the pit lake during the closure period. The LCWD and UCWD will be removed, and surface runoff and seepage from the WRF will be directed to the ACMA pit. The seepage flows from the WRF will be isolated by constructing concrete containment structures at the outlet of the rock drains for American Creek and Rob's Gulch and piped to the pit bottom to promote stratification. Before the cover is placed, the waste rock surface will be contoured to promote natural drainage toward the southern margin of the WRF. This contouring will create swales that will develop naturally and self-armor over time. Maintenance activities, such as placing riprap or cobble, will be carried out as needed to ensure the integrity of the swales and prevent erosion. Progressive reclamation during operations is expected to complete most of the WRF reclamation before mine closure. At a minimum, haul roads and ramps will be reclaimed during the closure phase (SRK, 2016).

20.6.4 Roads and Airstrip

Under both Donlin Gold LLC's corporate standards and regulatory standards, the mine site roads will need to be reclaimed. The Jungjuk port-to-mine access road and on-site roads needed for monitoring will remain for the foreseeable future following mine closure. The airstrip will remain a long-term asset; accordingly, it is not proposed for reclamation and is excluded from the reclamation cost estimate.

20.6.5 Foundations and Buildings

With the exception of the post-closure WTP, buildings must be removed from their foundations, and the debris must be either buried on-site or transported elsewhere, according to regulatory requirements. Once the buildings are demolished, the foundations will either be broken up and removed or broken up and buried to prevent them from being an impermeable impediment to natural percolation of meteoric waters. A minimum thickness of cover will be established over the buried debris to ensure that it remains below surface for the foreseeable future. The Project may also elect to maintain certain camp facilities after closure to support ongoing site monitoring or other post-closure activities.

20.6.6 Waste Disposal

Nonhazardous construction debris will either be placed in the inert solid waste landfills located in the WRF or used to fill subsurface voids exposed during the demolition of facilities. Hazardous and toxic materials, such as reagents, petroleum products, acids, and solvents, will be transferred off-site by licensed transporters and either returned to the vendor or disposed of at licensed treatment, storage, and disposal facilities. Hydrocarbon-contaminated soil will be treated on-site or removed from the site for off-site treatment and/or disposal.

20.6.7 Port and Mine Support Facilities

The Jungjuk port facilities will be reclaimed, leaving only a small barge landing area and the access road to the mine site. All mine support facilities except the airstrip and a small camp to support post-closure activities will be removed and reclaimed.

20.6.8 Mobile Equipment

Logistical constraints (access road and barge) preclude the decommissioning and removal of the mobile equipment fleet from the site. Therefore, this equipment is also planned to be buried in the WRF at site closure. Prior to burial of the equipment, all fluids will be removed and properly disposed of.

20.6.9 Trust Fund and Financial Assurance

The Report expects Donlin Gold LLC to use a mix of traditional bond or letter of credit and a trust fund to meet Alaska’s financial assurance requirements for mine closure. The bond or letter of credit will be issued when construction begins, while the trust fund will accumulate from Year 1 of operations. As the trust fund grows, the bond or letter of credit can be reduced accordingly. This fund is not intended to cover reclamation costs but rather ensures adequate cash flow for all obligations following mine closure, including (but not limited to) construction and maintenance of the spillway from the TSF, employee severance payments, and capital for the construction of the post-closure WTP, as well as ongoing operating costs for perpetual water treatment and associated facility and access maintenance. Notably, current cost estimates are based on data from 2019 and are scheduled for an update in 2026 to reflect any changes in market conditions or regulatory requirements. A model was developed for the trust fund calculation. The funding amount is estimated at $7.8 million provided annually over the construction and operating period, for a total of $412 million accrued to the trust fund at the start of closure. The following assumptions were made in determining this annual funding requirement:

- An income of 5% per annum is estimated to be earned on the cumulative trust fund.
- All operating expenditures for monitoring, water treatment, etc., and all capital expenditures are adjusted for an estimated annual inflation of 2%.
- The post-closure WTP will operate for about six months of the year.
- Annual costs are estimated for approximately 220 years after closure of the mine.

The original approval permit issued by ADNR was effective from 18 January 2019, through 17 January 2024. A formal request for extension was submitted on 18 September 2023, followed by a second extension requested on 17 November 2024. In addition to the trust fund, financial assurance will also be provided in the form of letters of credit and/or surety bonds to cover the physical aspects of mine construction and operation. In accordance with the Reclamation Plan Approval, financial assurance in the amount of approximately $322 million must be submitted in a form and substance approved by ADNR. The annual cost to maintain this assurance is estimated at 0.40% of the total assured amount, or about $1.3 M/a, commencing at the start of construction through the end of operations. These stated financial assurance costs are projected to increase in 2026 to reflect updated closure costs requirements.

20.6.10 Closure Cost Estimate

An independent cost estimate was developed, which was within 1% of the ADNR-approved reclamation and closure cost estimate. The final reclamation cost estimate for this update is shown in Table 20-4.

Table 20-4: Estimated Reclamation Costs

Item	Total ($M)
Reclamation, Closure, and Post-Closure Standard Reclamation Cost Estimate	532.6
Indirect Costs	364.9
Post-Closure Monitoring	10.0
Abandonment of NG Pipeline	12.1
Post-Closure Water Treatment	801.3
Total	**1,720.9**

Note: 25.6% was added to the costs from 2021 to account for inflation in 2025.

21.0 CAPITAL AND OPERATING COSTS

21.1 Capital Cost Estimates

21.1.1 Summary

The total initial capital cost is $9,233.4 million, including an Owner provided mining fleet and Owner performed mining pre-development. The level of accuracy for the capital cost estimate is ±25% and includes a contingency of 13.8%.

Table 21-1 shows the initial capital costs broken out by major area.

The capital cost estimate is based on updated costs escalated to fourth quarter 2025 pricing applied to the engineering designs and material take-offs (MTOs) from the feasibility study.

For pricing obtained in Canadian dollars, an exchange rate of US$0.71:C$1 was used.

Table 21-1: Summary of Initial Capital Costs by Major Area

Description	Estimate ($M)
Direct Costs	
Mining	429.4
Site Preparation and Roads	334.1
Process Facilities	1,817.8
Tailings Storage Facility and Reclaim Systems	169.4
Utilities	1,875.4
Ancillary Buildings and Facilities	437.9
Off Site Facilities	346.4
Subtotal Direct Cost	5,410.4
Owner's Costs	665.7
Indirect Costs	2,038.1
Contingency	1,119.1
Total Initial Capital Cost	**9,233.4**

Note: Indirect costs exclude contingency. Figures may not sum due to rounding.

With the exception of the following three design changes, no other changes to engineering or MTOs were made:

- Mine plan quantities and sequencing, including fleet purchase schedules

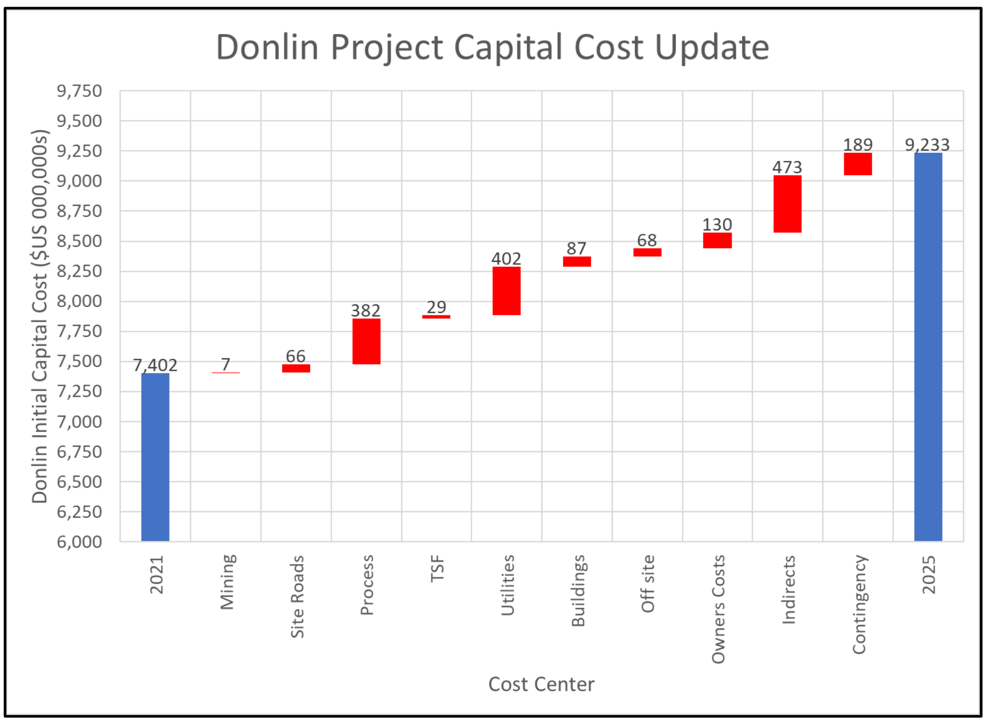
- The natural gas pipeline was updated for an increase in pipe diameter from 12" to 14" and for modifications made to the route (i.e., the North Route Alignment) between mile post (MP) 85 and 112.

Initial and sustaining capital estimates were updated for all areas. Closure and reclamation costs were based on Donlin Gold LLC's filed closure plan with the ADNR. Warehouse inventory is excluded from the capital cost estimate but is included in the financial model as part of the working capital provision.

The total initial capital cost estimate is $9,233 million, which is an increase of $1,831 million, a 24.7% increase, compared to the 2021 initial capital estimate. Figure 21-1 shows the step changes by major area for the 2025 capital cost estimate.

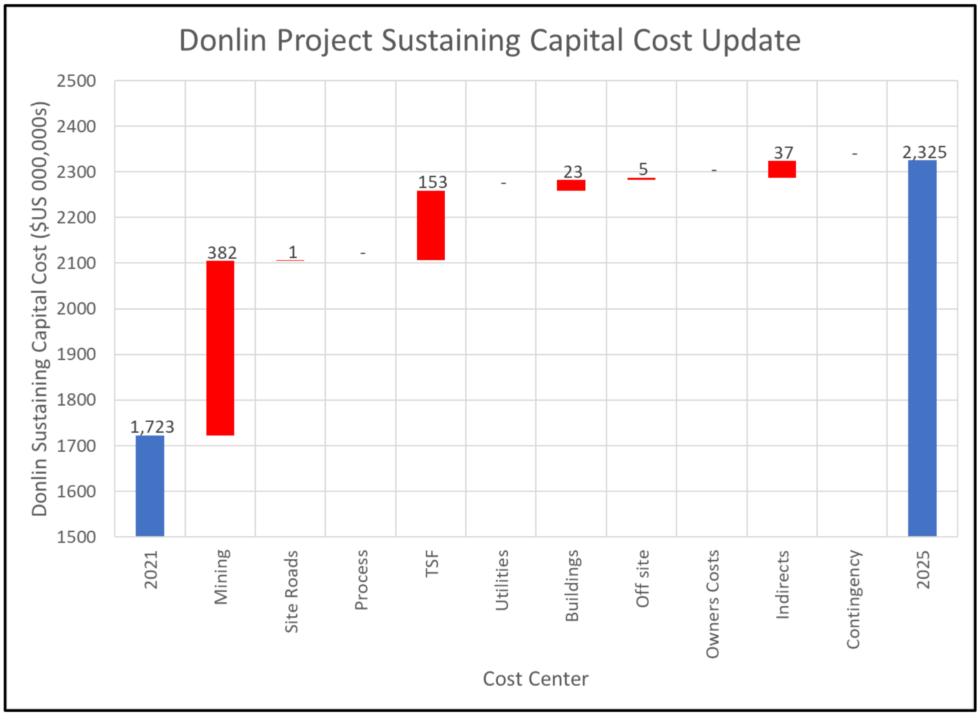
The total sustaining capital estimate is $2,325 million, which is an increase of $602 million, a 34.9% increase, compared to the 2021 sustaining capital cost estimate. Figure 21-2 shows the step changes by major area for the 2025 sustaining capital cost estimate.

Figure 21-1: 2021 to 2025 Initial Capital Cost Changes by Major Area



Source: Wood, 2025

Figure 21-2: 2021 to 2025 Sustaining Capital Cost Changes by Major Area



Source: Wood, 2025

21.1.1.1 Basis of Estimate

Installation costs are estimated using a rate x manhours approach to develop the overall installation cost which formed the basis for all discipline cost adjustments. Craft labor rates have been escalated from the 2021 estimate as well as benchmarked against current union rates. Burdened labor rates typically include:

- Base labor rate, payroll burdens and benefits, completion bonus, incentives and overtime premiums
- Workers compensation
- Fringe, industry funds
- Government fees (pension, employment insurance)
- Safety clothing and safety supplies
- Small tools and consumables including welding rods, sealant, adhesives and lubricants
- Contractors' supervision and administration.
- Construction equipment.

A 50-60% allowance of the installation cost has been included to cover the contractors' distributable costs.

Some of the major process and infrastructure equipment costs were based on recent firm quotations. Recent budget quotes were also received for civil lining and the natural gas pipeline. Updated pricing was obtained for major mobile equipment for the open pit operation. The mine plan and mining costs have been updated which has resulted in some costs being moved from initial capital to sustaining capital.

Bulk material and minor equipment costs are based on a factored approach using indices.

The Federal Reserve Economic Database (FRED) indices and Consumer Price Index (CPI) were used as the normalization factor to escalate the estimate. FRED indices were used where information is available and is consistent with past escalation efforts. CPI is used where FRED indices do not cover past escalation or the category of work being escalated. When used, the CPI index is an average factor for all disciplines. Both methods were benchmarked against the Information Handling Services (IHS) indices. IHS indices are applied at a discipline level. With the IHS index used as the normalization factor, the average escalation was 22%, which is within reasonable range of the 24.7% escalation from the combined FRED-CPI method used.

The escalation indices for indirect costs were calculated by determining the average escalation of the direct costs.

Engineering procurement and construction management (EPCM) costs were reviewed by an expert in EPCM contracting. The EPCM costs were adjusted to reflect current contracting strategies and updated prior to applying the escalation index.

21.1.1.2 Scope of Responsibilities

Input was provided by QPs Peralta, Drake, Baluch, Pretorius, and Sisson. QP Peralta prepared the mining estimate, QP Drake contributed to the process facilities estimate, and QP Baluch contributed to the estimates for the site preparation, utilities, ancillary buildings, and off-site facilities. QPs Pretorius and Sisson contributed to the estimates for water management and TSF. Reviews were completed with independent estimating experts on the estimation methodology of the Owner's costs and indirects. Wood QPs coordinated all the data being assembled into the estimate.

21.1.1.3 Mining

Mining initial capital costs are estimated at $429.4 million and distributed as detailed in Table 21-2.

Mining capital costs include pre-production stripping and development of waste stockpiles, mobile equipment, water management, and site preparation for the contractor owned explosives facility.

Mining costs were estimated with current rates for labor, fuel, consumables, and budgetary quotations for mining equipment. Approximately 76% of the mobile equipment is based on recent pricing, with the remainder escalated from previous prices and benchmarked against database costs.

Mine pre-production and development includes costs related to 20 Mt of pre-stripping completed in Year -1, as well as capital spares and mine operations overhead.

A first principles cost model was developed that incorporates the Donlin mine production plan to determine equipment requirements, including initial purchases and replacements. Capital replacements are assumed to be purchased in the year they are needed in sustaining capital.

The water management costs included in the mining costs cover the WTP for the operation, perimeter wells, overland drainage, piping and culverts related to the pit, WRFs and ore stockpile.

Table 21-2: Breakdown of Mining Capital Costs

Area Description	Estimate ($M)
Pre-production and Development	129.8
Mobile Equipment	180.6
Water Management	118.0
Explosives Handling and Storage	1.0
Total	**429.4**

Note: Figures may not sum due to rounding.

21.1.1.4 Site Preparation and Roads

Site preparation and roads have been estimated at $334.1 million as detailed in Table 21-3.

Site preparation and roads initial capital costs have been updated using current construction labor rates, commodity unit rates, and index factors applied against the same scope as the 2021 technical report.

Access road and airstrip road includes the access from Jungjuk port to site and the connecting road to the airstrip.

General site development includes diversion dams and ditching, as well as general site earthworks related to prepping the site for construction of the processing facilities.

Plant site roads and construction roads include various site roads' construction for construction vehicle access related to construction of site facilities.

Air strip includes capital for preparation of the airfield, as well as provision of two 50 passenger planes for flights to and from Anchorage.

Table 21-3: Breakdown of Site Preparation and Roads Capital Costs

Area Description	Estimate ($M)
Access Road and Airstrip Road	27.0
General Site Development	242.6
Plant Site Roads and Construction Roads	26.9
Air Strip	37.6
Total	**334.1**

Note: Figures may not sum due to rounding.

21.1.1.5 Process Facilities

Process facilities have been estimated at $1,817.8 million, as detailed in Table 21-4.

The process plant initial capital was updated by applying current labor rates, commodity unit rates, index factors, and updated budgetary equipment quotations to the existing feasibility design criteria, flowsheets, mechanical equipment list and MTOs. Twenty-five percent of the value of mechanical equipment from the 2021 technical report was updated with current budgetary pricing with the remainder being factored based on the project indices outlined in Section 21.1.1.1.

Table 21-4: Breakdown of Process Facilities Capital Costs

Area Description	Estimate ($M)
Crushing	125.3
Grinding	289.3
Floatation	292.9
Thickening	116.2
Pressure Oxidation	807.5
Carbon-in-Leach	54.9
Gold Recovery	61.3
Reagents and Process Control	70.4
Total	**1,817.8**

Note: Figures may not sum due to rounding.

The capital cost estimate for the process plant includes provision for all mechanical and electrical equipment, buildings, and quantities for bulks such as earthworks, concrete, steel, piping, electrical, and instrumentation.

Major processing equipment were sized based on the process design criteria. Mechanical scopes were sent to the market for budgetary pricing by local and international equipment suppliers.

The pressure oxidation estimate includes $455.1 million for the pressure oxidation circuit, $192.1 million for the oxygen plant, and $159.7 million for the associated thickeners and cyanide neutralization.

21.1.1.6 Tailings Management and Reclaim Systems

Tailings management and reclaim systems have been estimated at $169.4 million, as detailed in Table 21-5.

The TSF initial capital cost was updated by applying current construction labor rates for updated construction unit costs. Updated budget quotations account for 32.7% of total material costs. Equipment costs and material costs, where recent budget quotations were not received, have been escalated using the project indices outlined in Section 21.1.1.1.

The capital cost estimate for the TSF includes provision for construction of the initial starter dam. An allowance has been included for additional haulage of waste material from the open pit and process site construction.

Table 21-5: Breakdown of the Tailings Management and Reclaim Systems Capital Costs

Area Description	Estimate ($M)
Tailings Storage Facility	122.6
Tailings Line	30.7
Reclaim System	16.2
Total	**169.4**

Note: Figures may not sum due to rounding.

An allowance has been made for grubbing and stripping overburden from the footprint of the dam. Supply and installation of the liner was considered for placement up to the starter dam elevation.

21.1.1.7 Utilities

Site utilities costs are estimated at $1,875.4 million, as detailed in Table 21-6.

Site utilities initial capital costs have been updated using current construction labor rates, commodity rates, and index factors applied against the same scope as outlined in the 2021 technical report. Included within the power generation are costs for the natural gas pipeline construction.

The current natural gas pipeline cost estimate of $1,090.8 million is based on 2025 material pricing for piping and a budgetary construction estimate for construction costs prepared by personnel with experience in Alaska's North Slope and Kenai Peninsula.

Table 21-6: Breakdown of the Utilities Capital Costs

Area Description	Estimate ($M)
Power Generation	1,516.8
Water Systems	200.6
Compressed Air	4.0
Fuel Storage and Distribution	90.1
Communications	63.9
Total	**1,875.4**

Note: Figures may not sum due to rounding.

Other major utilities include provision for construction of the on-site natural gas power plant, site water management, compressors, fuel farm, and communications systems.

21.1.1.8 Ancillary Buildings and Facilities

Ancillary buildings and facilities capital costs are estimated at $437.9 million, as detailed in Table 21-7.

On-site infrastructure initial capital costs have been updated using current construction labor rates, commodity unit rates, and index factors applied against the same scope as the 2021 technical report.

Costs for offices and storage are outfitting costs only as space has been allocated within the processing facility complex.

Plant mobile equipment includes long-term site maintenance fleet, as well as equipment utilized in construction of the Project.

Table 21-7: Ancillary Buildings and Facilities Capital Cost Breakdown

Area Description	Estimate ($M)
Administration/Dry	1.4
Assay Laboratory	3.7
First Aid and Emergency Vehicle Storage	0.9
Accommodation Complex	72.0
Exterior Utilidors	15.0
Cold Storage	9.3
Mine Truck Shop and Warehouse	110.6
Plant Mobile Equipment	224.9
Total	**437.9**

Note: Figures may not sum due to rounding.

21.1.1.9 Off-Site Facilities

Off-site infrastructure initial capital costs have been updated using current construction labor rates, commodity unit rates, and index factors applied against the same scope as the 2021 technical report, as outlined in Table 21-8.

Table 21-8: Breakdown of the Off-Site Facilities Capital Costs

Area Description	Estimate ($M)
Crooked Creek Facilities	35.6
Jungjuk Port Site	81.9
Marine Equipment and Facilities	228.9
Total	**346.4**

Note: Figures may not sum due to rounding.

Crooked Creek facilities include early construction ice roads and storage for initial access of construction prior to Jungjuk port site being constructed.

Jungjuk port includes allowances for construction of the port site and offloading equipment for river barges at the Jungjuk port. Semi-trailer tractors for fuel transport and general cargo are also included.

Marine equipment and facilities are costs for ocean fleet and facilities upgrades required to support the Project.

21.1.1.10 Owner's Costs

Owner's costs are based on an escalation of the 2021 technical report Owner's costs estimate. Total Owner's costs are $665.7 million, primarily related to construction camp costs.

Owner's reserves of $226.4 million are included in the estimate. The Owner's reserves are the difference between the P_{85} estimate and the P_{40} estimate contingency.

Table 21-9: Owner's Costs Capital Cost Breakdown

Area Description	Estimate ($M)
Site Overhead Costs	433.0
Alaska Office Costs	6.3
Owner's Reserves	226.4
Total Owner's Costs Capital Cost	**665.7**

Note: Figures may not sum due to rounding.

21.1.1.11 Indirect Costs

The indirect costs are based on an escalation of the 2021 technical report indirect estimate. Indirect costs for the natural gas pipeline were calculated separately and added to produce a total indirect cost of $2,038.1 million.

Indirects amount to approximately 37.7% of the direct initial capital cost. Table 21-10 outlines the distribution of indirect costs.

Table 21-10: Breakdown of the Indirect Costs

Area Description	Estimate ($M)	Percentage of Direct Capital (%)
Temporary Facilities and Services	594.7	11.0
EPCM	677.0	12.5
Construction Camp	237.6	4.4
First Fills and Capital Spares	126.2	2.3
Freight	331.7	6.1
Vendor Costs	31.0	0.6
Start-up and Commissioning	40.0	0.7
Total	**2,038.1**	**37.7**

Note: Figures may not sum due to rounding.

21.1.1.12 Contingency

The blended project contingency is 13.8%, as outlined in Table 21-11.

Table 21-11: Contingency Breakdown

Contingency Area	Total Project Cost Including Contingency ($M)	Contingency ($M)	Contingency (%)
Mining, Processing, Site Infrastructure, Off-site Facilities	6,730.1	802.6	13.5
Pressure Oxidation and Oxygen Plant	1,115.6	137.1	14.0
Natural Gas Pipeline	1,387.7	179.4	14.9
Total	**9,233.4**	**1,119.1**	**13.8**

Note: Figures may not sum due to rounding.

21.1.2 Sustaining Capital

The 2021 technical report sustaining capital estimate was escalated to 2025 pricing. Changes to the mine plan adjusted the mobile equipment purchasing schedule and fleet totals. No changes were made to project engineering.

Mining sustaining capital is for mobile equipment fleet replacement as required within the operating hours model.

The storage capacity of the TSF will be increased through additional raises of the dam in Years 1, 5, 9, 13, 17, and 21, and 25. These additional costs for tailings are included in sustaining capital related to ongoing construction of the facility as the operation progresses.

Owner's costs are not included in the sustaining capital estimate because in the operating phase, camp related expenses are captured under G&A operating costs rather than capital.

Contingency is embedded within the unit rates for sustaining capital activities. These rates incorporate an allowance of less than 15%, which provides coverage for potential cost without requiring an additional contingency allocation.

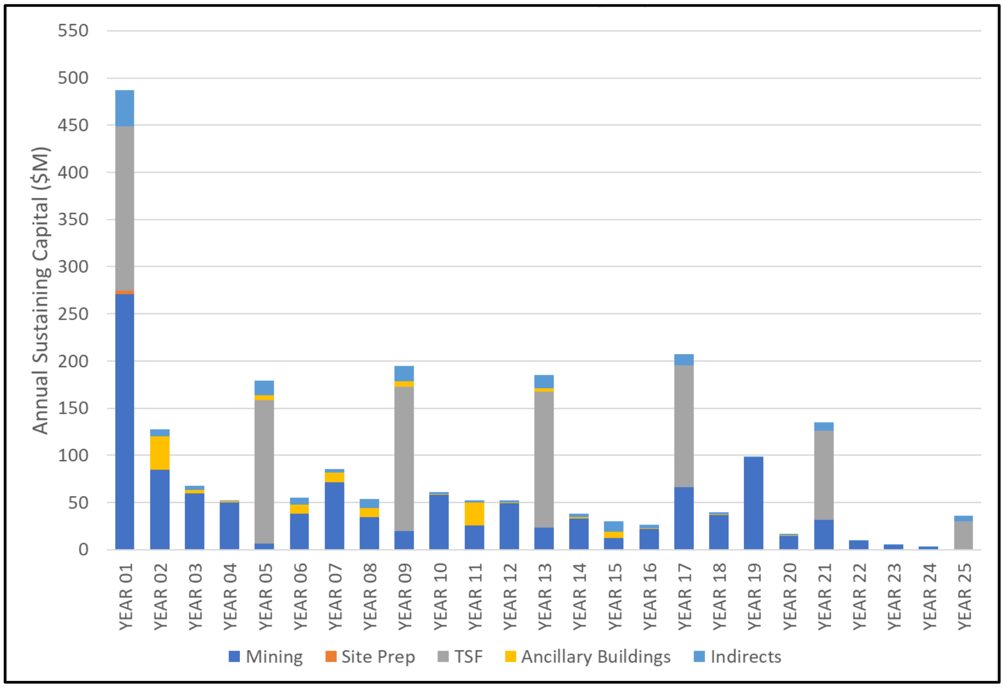
The total sustaining capital estimate is $2,325 million, as outlined in Table 21-12. Figure 21-2 shows the step changes by major area for the 2025 update. Figure 21-3 shows the annual distribution of sustaining capital.

Table 21-12: Breakdown of Sustaining Capital Costs

Description	Estimate ($M)
Mining	1,125.0
Site Preparation and Roads	4.0
Process Facilities	-
Tailings Storage Facility and Reclaim Systems	876.9
Utilities	-
Ancillary Buildings and Facilities	119.5
Off Site Facilities	24.9
Subtotal Direct Cost	2,150.3
Owner's Costs	-
Indirect Costs	174.7
Contingency	-
Total	**2,325.1**

Note: Indirect costs exclude contingency. Figures may not sum due to rounding.

Figure 21-3: Distribution of Sustaining Capital



Source: Wood, 2025

21.2 Operating Cost Estimates

21.2.1 Summary

Total operating cost average $48.54/t of ore processed as shown in Table 21-13. Operating cost accuracy is within ±25%. Contingency is included and is less than 15%. The operating cost estimates were updated to fourth quarter 2025 pricing by updating key cost drivers including energy, labor, consumables, and freight.

Changes to the mine plan have affected the manning schedules and consumables for mining, but the total mill feed processed and annual processing schedule remained unchanged.

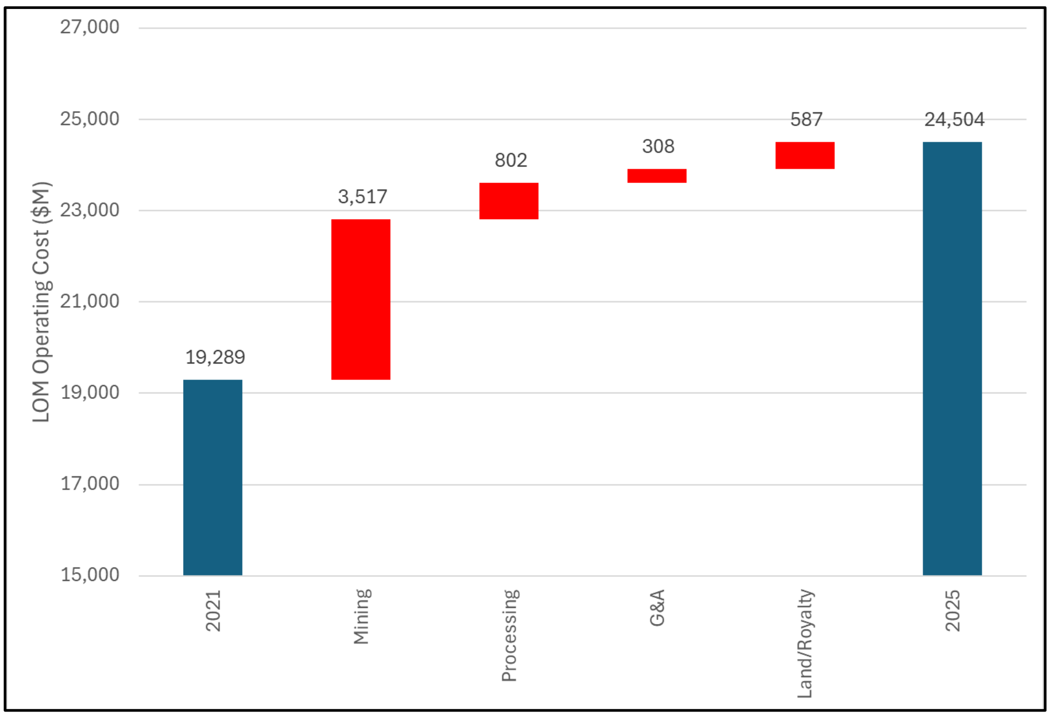
Figure 21-4 shows the step changes by major area for the 2025 update. For the purposes of the mine plan, costs were based on the feasibility study and escalated to 2025 pricing. With the mine plan finalized, the costs were updated to the fourth quarter 2025 which are not materially different from those used in the mine plan.

Table 21-13: LOM Operating Costs

Area	Total LOM ($M)	$/t Ore Processed
Mine	11,946.7	23.67
Processing	7,718.3	15.29
G&A	2,070.4	4.10
Land and Royalty Payments	2,768.8	5.48
Total	**24,504.3**	**48.54**

Note: Figures may not sum due to rounding.

Figure 21-4: 2021 to 2025 Operating Cost Changes by Major Area



Source: Wood, 2025

21.2.2 Basis of Estimate

The labor rates are consensus rates based on 2025 CostMine labor rates for Alaska and Alaska Department of labor and Workforce Development research and analysis of Alaska statewide wages.

The work schedules assume that production will operate 24 h/d, 7 d/wk, 365 d/a.

An estimated fuel cost of $0.94/L was used, based on the US Energy Information Administration forecast retail price, less state and federal taxes, plus freight to transport to Donlin. This assumes a $0.747/L purchase price and a $0.198/L freight cost.

The natural gas fuel price of $12.51/MMBtu used considers the following liquid natural gas (LNG) costs:

- export price of $7.51/MMBtu delivered to Cook Inlet
- $1.60/MMBtu transport to Kenai
- $2.63/MMBtu re-gasification
- $0.20/MMBtu storage charges
- $0.57/MMBtu energy and pipeline costs to deliver to Donlin.

The estimated cost of electricity from a gas-fired dual-fueled (natural gas and diesel back-up) reciprocating engine power plant is $0.119/kWh, not including capital costs.

The total unit operating cost of the general cargo supply chain was estimated to be equivalent to $284.67/t freight. This has been used to calculate freight costs for mine and mill consumables as outlined in Section 21.2.5.

21.2.3 Mine Operating Costs

The mine operating cost estimate reflects updates to the mine schedule and includes costs for operating and maintenance labor, staff, and supplies. Operating and maintenance supplies are based on North American supply and include an allowance for freight and delivery to marshalling yards at the port of Seattle. Logistics costs to deliver freight to site have been reallocated from G&A based on the proportion of freight attributable to mining.

Consumables (fuel, explosives, supplies) were calculated from expected use, unit consumptions, and allowances for minor items. All mining costs are based on production Years 1 to 27. Pre-production costs have been capitalized and included in the initial capital cost estimate.

Mining costs include stockpile rehandling costs for mill feed.

The mining cost averages $3.16/t mined (ex-pit) during mine operations (excluding tonnes and costs from the pre-production period). Table 21-14 provides a breakdown of mining costs.

Table 21-14: LOM Mining Operating Cost

Item	Total ($M)	$/t Mined Ex-pit
Labor	2,281.2	0.60
Electricity	104.9	0.03
Diesel	3,329.4	0.88
Lube	154.3	0.04
Drilling Supplies	55.1	0.01
GET	118.3	0.03
Blasting Accessories	387.3	0.10
Emulsion	1,536.4	0.41
Tires	426.9	0.11
Maintenance Parts & Supplies	2,566.2	0.68
Other	986.8	0.26
Total	**11,946.7**	**3.16**

Note: Figures may not sum due to rounding. GET = ground engaging tools

21.2.4 Process Operating Costs

The processing costs cover operation and maintenance of the processing facilities, from the coarse ore dump pocket at the primary crusher through to the bullion room, as well as process and reclaim water pumping. The processing costs account for the expenses associated with purchasing consumables, equipment maintenance, personnel, and power consumption. Costs are summarized in Table 21-15. Process consumables costs are based on recent vendor quotations.

Equipment maintenance costs were developed for all areas from first principles for major equipment and factored for minor equipment. The costs were then expressed as a percentage of the capital cost of equipment for all areas, except for the POX and the oxygen plant. Mill liner costs are based on vendor estimates.

Annual maintenance costs for the pressure oxidation and the oxygen plant were developed from first principles and escalated to 2025 pricing. The maintenance costs are also allocated by area.

Power consumption was derived from the estimated load of the equipment combined with power requirements for the crushing and grinding circuits. Power consumption was estimated to average 1,050 GWh/a. The calculation of primary crusher power consumption was based on the crusher running at 80% of full power, available 65% of the time.

Table 21-15: Process Operating Costs

Item	Annual Total ($M)	LOM Total ($M)	$/t Processed
Labor	37.6	972.6	1.93
Reagents and Consumables	118.4	3,061.7	6.07
Power	92.4	2,389.7	4.73
Maintenance Supplies	50.0	1,294.3	2.56
Total	**298.6**	**7,718.3**	**15.29**

Note: Figures may not sum due to rounding.

21.2.5 General and Administrative Operating Costs

A portion of G&A expenses are allocated to the mining and process operating cost structures. Logistics costs are classified as follows:

- Logistics costs for mining are allocated to the mining cost structure from G&A
- Logistics costs for processing are included in consumables pricing
- Logistics costs for diesel are incorporated into diesel unit price
- Remaining logistics costs are included in G&A.

Other G&A functions are presented as standalone operating cost categories.

The LOM G&A operating costs reflect updated pricing and cover cost centers not directly associated with mining and process activities. These include management, safety, security, environmental, information services, warehousing and other overhead functions. Estimates for each cost center were developed from first principles or escalated.

Approximately $440.2 million in LOM G&A operating costs are allocated back to mining. Table 21-16 shows the LOM G&A operating costs by cost item. The LOM G&A operating costs averages $4.10/t processed after the G&A allocation.

Table 21-16: LOM G&A Operating Cost Estimate

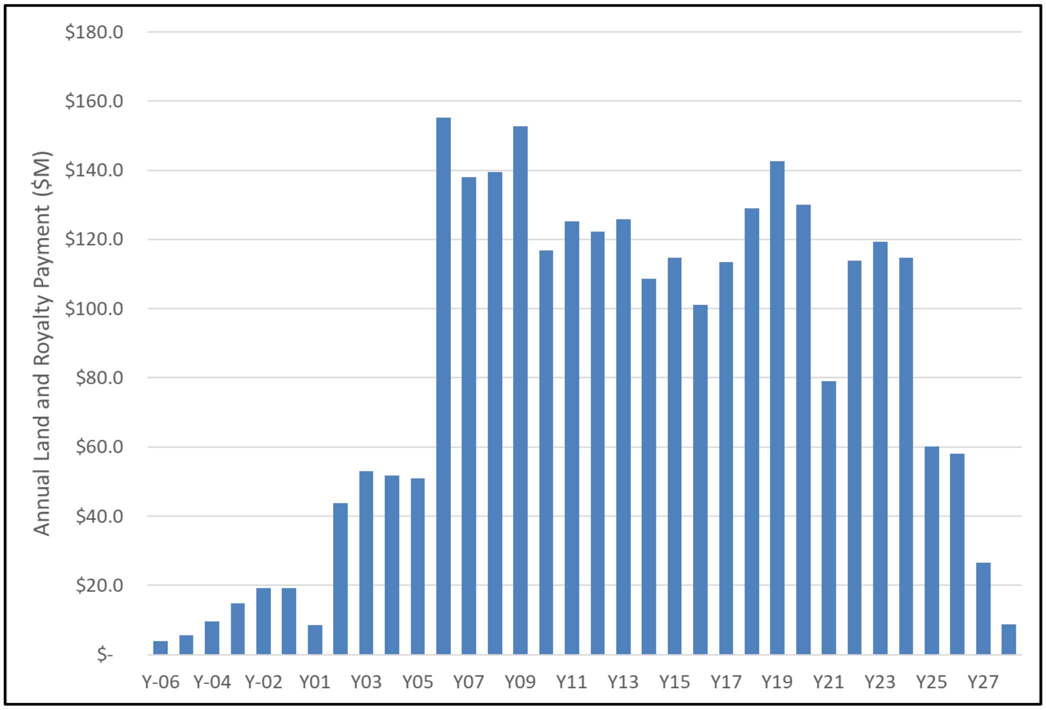
Item	Total ($M)	$/t Processed
Aviation	98.0	0.20
Site Maintenance & Mobile Equipment	207.0	0.41
Camp and Catering	481.1	0.95
Clinic	35.8	0.07
Community Relations	192.1	0.38
Emergency Response	13.6	0.03
Environmental	60.6	0.12
Finance and Accounting	37.3	0.07
Health and Safety	58.9	0.12
Insurance	229.4	0.45
Information Technology	282.2	0.56
Legal	49.6	0.10
Logistics	1,649.9	3.25
Management	53.6	0.11
Land	19.4	0.04
Security	50.3	0.10
Training and Recruiting	27.5	0.05
Waste Management	30.9	0.06
Power	120.3	0.24
Subtotal	3,697.2	7.32
Logistics included in diesel price	(766.7)	
Logistics included in mill consumables	(419.7)	
Logistics allocated to mine	(440.2)	
Total	**2,070.4**	**4.10**

Note: Figures may not sum due to rounding.

21.2.6 Land and Royalty Payments

Calista, TKC, and the Lyman family receive payments for land access and use. Over the LOM, the land and royalty payments amount to $2,768.8 million, distributed as shown in Figure 21-5.

Figure 21-5: Land and Royalty Annual Payments



Source: Wood, 2025

22.0 ECONOMIC ANALYSIS

Certain information and statements contained in this section of the Report are forward-looking in nature and are subject to known and unknown risks, and that actual results of the economic analysis may vary from what is forecast. Examples of forward-looking information include gold price assumptions, cash flow forecasts, projected capital and operating costs, mine and metallurgical recoveries, mine life and production rates, and other assumptions used in this feasibility study identified in the relevant sections of this Report. Material risk factors are identified in Section 25.

22.1 Valuation Methodology

The overall economic viability of the Project has been assessed using both undiscounted and discounted cash flow techniques and on a pre-tax and after-tax basis. Net annual cash flows were estimated by projecting yearly cash inflows (or revenues) and subtracting projected yearly cash outflows (such as capital and operating costs, royalties, and taxes). Undiscounted techniques include total net cash flow, IRR and payback period (measured from start of production in third quarter of Year 1). Discounted cash flow techniques include NPV. Discounted values are calculated using a 5% discount rate and a discrete end-of-year convention relative to Year -6, the start of basic and detailed engineering.

Estimates have been prepared for all the individual elements of cash receipts and cash expenditures for ongoing operations. Capital cost estimates have been prepared for initial development and construction of the Project (initial capital) and for ongoing operations (sustaining capital). Cost estimates have also been prepared for reclamation and closure of the mine and for post-closure obligations. These form the basis for the annual trust funding requirements over the LOM required to meet these obligations.

22.2 Financial Model Parameters

The financial analysis was based on royalty agreements described in Section 4, the Mineral Reserves tabulated in Section 15, the forecast mine plan and production schedule presented in Section 16, the process plan and assumptions detailed in Section 17, the projected infrastructure requirements outlined in Section 18, the gold price assumption in Section 19, the permitting, social and environmental regime discussions as well as closure obligations in Section 20, the capital and operating cost estimates detailed in Section 21, and the tax considerations in Section 22.2.7.

22.2.1 Metallurgical Recoveries

Recovery is estimated to average 90.0% over the LOM.

22.2.2 Transportation, Smelting and Refining Terms

Doré refining and shipping charges were estimated at $1.56/oz of doré sold based on escalating to 2025 refining charges for large U.S.-based mining operations and a quotation for transportation and insurance costs from the planned Donlin mine site to a U.S.-based refinery utilized in 2011. A doré gold content of 90% was considered. In addition, 0.1% of gold produced at the mine is deducted as a cost of refining.

22.2.3 Metal Prices

Estimated cash flows from revenue are based on a gold price of $2,100/oz.

22.2.4 Capital Costs

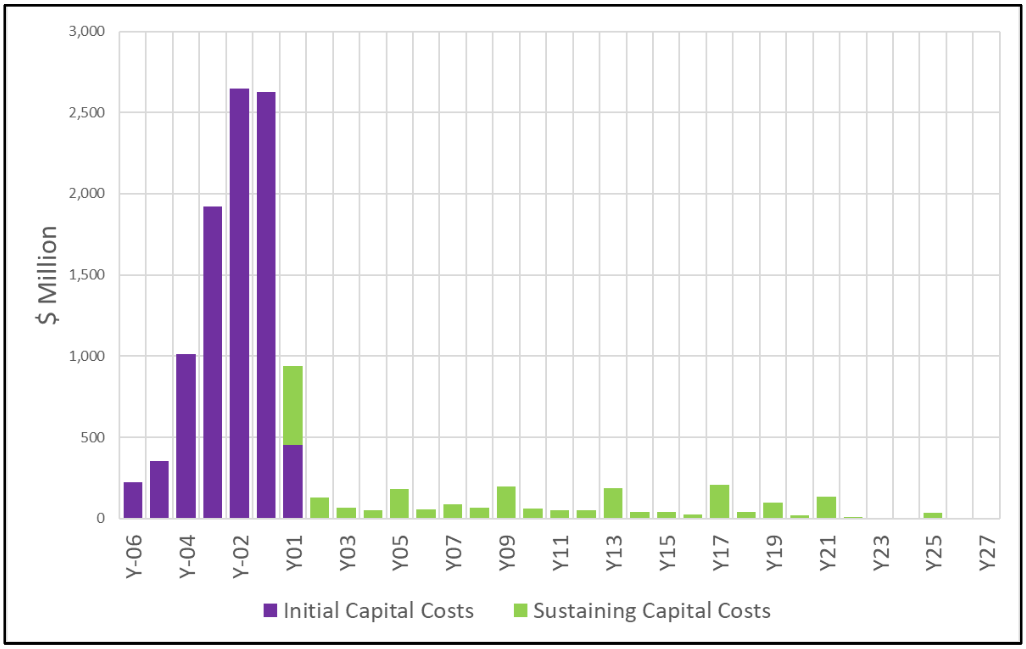
The initial capital costs for the Project are estimated at $9,233 million. Sunk costs (all costs incurred prior to Year -6) are excluded from the cash flow analysis. Sustaining capital costs over the LOM are estimated at $2,325 million. Total capital is estimated at $11,558 million. Figure 22-1 shows the distribution of initial and sustaining capital included within the financial model.

22.2.5 Operating Costs

LOM operating costs (including mining, processing, G&A and royalties) are estimated at $24,504 million, equivalent to $48.54/t of ore processed. Figure 22-2 provides an annual distribution of operating costs.

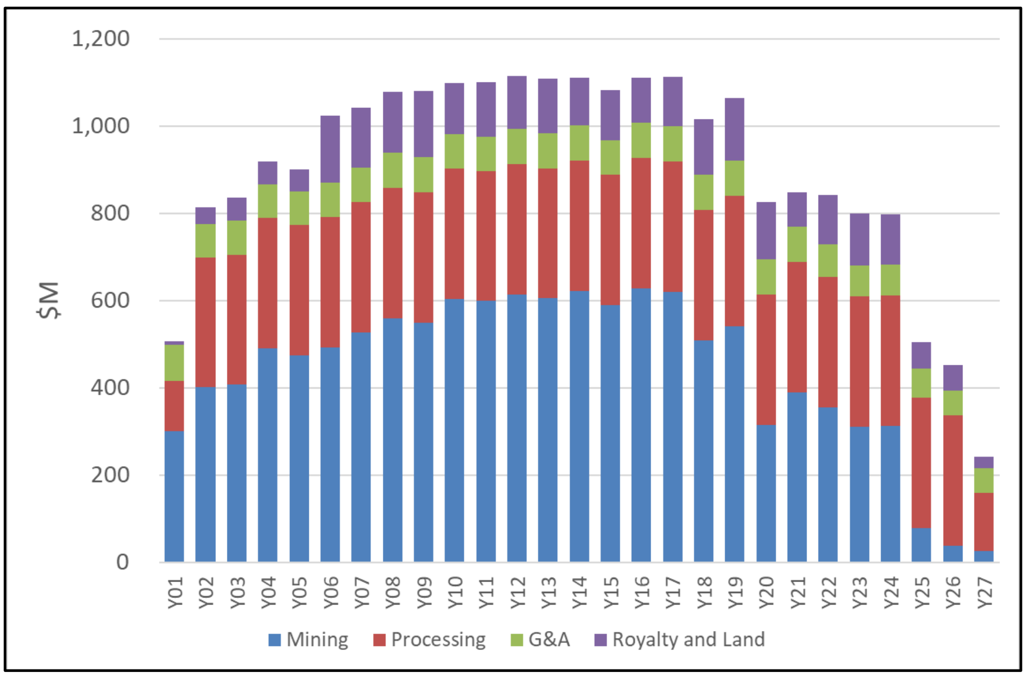
Calista, TKC, and the Lyman family receive payments for land access and use. Over the LOM the land and royalty payments amount to $2,768.8 million. Annual royalties are included in Figure 22-2.

Figure 22-1: Distribution of Initial and Sustaining Capital



Source: Wood, 2025

Figure 22-2: Operating Costs



Source: Wood, 2025

22.2.6 Working Capital

Inventory of consumables plus working capital are included in the cash flow. They are expenditures in the early years and are recovered in the final years. Thus, the sum of all working capital over mine life is zero. The following terms were assumed:

- First fills inventory = included in capital costs
- Initial inventory = 100 days of non-labor operating costs
- In-process inventory = 3 days of revenue
- Finished products inventory = 10 days of revenue
- Accounts payable = 30 days of non-labor operating costs
- Accounts receivable = 10 days of revenue.

22.2.7 Taxes

The following taxation summary was provided by NOVAGOLD for the Project.

U.S. Federal Corporate Income Tax

In accordance with the Tax Cuts and Jobs Act (TCJA) enacted in December 2017 and largely effective January 1, 2018 and the One Big, Beautiful Bill Act (OBBBA) enacted in 2025, which extended or made permanent many TCJA provisions:

- Corporate income tax rate is 21%.
- Post-2017 net operating loss (NOL) carryovers are limited to 80% of taxable income, while there is no limitation on pre-2018 NOL carryovers.
  - The TCJA changed the NOL rules by limiting NOL deductions to 80% of taxable income, disallowing NOL carrybacks, and lifting the 20-year limit on NOL carryovers for post-2017 NOLs. The pre-TCJA NOL rules (80% limitation, disallowed carrybacks, 20-year carryover period) still apply to pre-2018 NOLs. The NOL rules were not meaningfully modified by the OBBBA.
- The alternative minimum tax (AMT) was eliminated, but the corporate alternative minimum tax (CAMT) replaced it with the enactment of the Inflation Reduction Act (IRA) in 2022. The CAMT generally only applies to corporations with average annual adjusted financial statement income in excess of $1 billion over a three-taxable-year period.

- A percentage depletion deduction is applied upon the commencement of production and sale of gold. Cost depreciation is not applicable as there is no Project tax basis for the mineral property.
- Bonus depreciation is not considered in the tax depreciation calculation.

Alaska Corporate Incomed Tax

- Alaska corporate income tax rate is 9.4%.
- Alaska conforms to U.S. federal tax treatment, and it imposes a state CAMT equal to 18% of the federal CAMT.

Alaska Mining License Tax

- Alaska mining license tax rate is 7%.
- Alaska mining license tax holiday is applied for the first 3½years from the start of production.

22.2.8 Closure Costs

The closure cost funding cost totals $423.0 million, comprised of:

- $371.2 million based on $11.6 million/year accruals from Year -5 to the end of the LOM
- $51.8 million to maintain a financial guarantee of $404 million (escalated from the $322 million presented in Section 20) considering an annual cost of 0.4% per year from Year -5 to the end of the LOM

The trust fund will be sufficient to cover the closure costs outlined in this Report at end of mine life.

22.2.9 Salvage

No salvage is assumed at the end of operations.

22.2.10 Financing

Financing has been assumed on a 100%, all equity, stand-alone basis.

22.2.11 Inflation

Escalation/inflation has been excluded from the forecast cash flow. Escalation has been included in the determination of the funding requirements for the trust fund, but the trust fund values in the cash flow are expressed on an un-escalated (real) basis.

22.3 Economic Results

Based on the economic evaluation, the Project generates positive before and after-tax economic results. After tax NPV is $5,058 million at a 5% discount rate, an IRR of 10.3% and payback of 6.5 years from the start of production. Table 22-1 provides a summary of key evaluation metrics, Table 22-2 shows the cash flow, and Figure 22-3 shows the distribution of after-tax cash flows and NPV.

Table 22-1: Summary of Key Evaluation Metrics

Item	Unit	Value
Total Mined	Mt	3,803
Ore Treated	Mt	504.8
Strip Ratio	W/O	6.5
Gold Recovered	Moz	29.5
Gold Recovery	%	90.0
Gold Payable	%	99.9
Gold Price	$/oz	2,100
Total Before Tax Cash Flow	$M	25,415
Total Before Tax $NPV_{5\%}$	$M	7,516
Before Tax IRR	%	12.5
Before Tax Payback Period	years	4.9
Total After Tax Cash Flow	$M	19,614
Total After Tax $NPV_{5\%}$	$M	5,058
After Tax IRR	%	10.3
After Tax Payback Period	years	6.5
Gross Revenue	$M	61,952
Selling Costs	$M	51
Operating Costs (Inc. Royalties)	$M	24,504
Initial Capital	$M	9,233
Sustaining Capital	$M	2,325
Total LOM Capital	$M	11,558
Closure Costs	$M	423
Taxes	$M	5,801

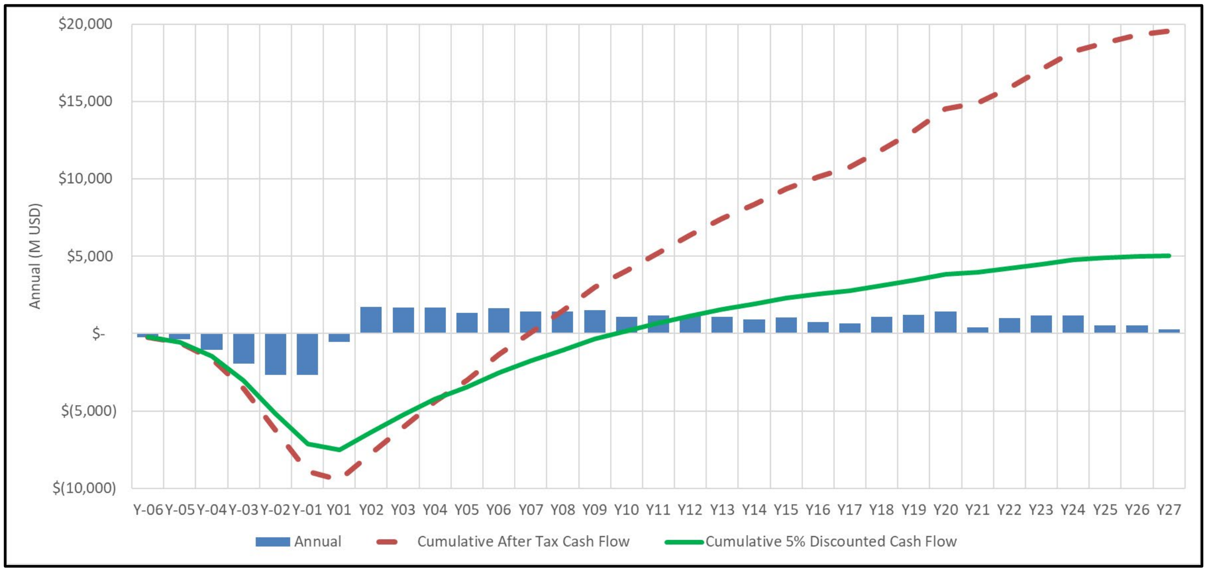
Table 22-2: Cash Flow Analysis

			Years																	
Item	**Unit**	**LOM Total**	**PP -6**	**PP -5**	**PP -4**	**PP -3**	**PP -2**	**PP -1**	**Yr 1**	**Yr 2**	**Yr 3**	**Yr 4**	**Yr 5**	**Yr 6**	**Yr 7**	**Yr 8**	**Yr 9**	**Yr 10**	**Yr 11**	**Yr 12**
PRODUCTION																				
Mill Feed Tonnes	kt	504,811	-	-	-	-	-	-	7,539	19,528	19,529	19,581	19,528	19,527	19,528	19,581	19,528	19,528	19,528	19,581
Mill Feed Grade	g/t	2.02	-	-	-	-	-	-	2.49	2.48	2.44	2.35	2.31	2.70	2.40	2.40	2.65	2.00	2.10	2.02
Mill Recovery	%	90.0	-	-	-	-	-	-	88.18	91.55	88.37	88.51	88.52	90.52	89.76	90.49	90.56	89.73	90.71	91.55
Gold Payable	Koz	29,501	-	-	-	-	-	-	533	1,421	1,351	1,308	1,282	1,533	1,351	1,366	1,505	1,126	1,195	1,164
Selling Costs	$M	(51)	-	-	-	-	-	-	(1)	(2)	(2)	(2)	(2)	(3)	(2)	(2)	(3)	(2)	(2)	(2)
Net Smelter Revenue	**$M**	**61,901**	**-**	**-**	**-**	**-**	**-**	**-**	**1,117**	**2,983**	**2,835**	**2,745**	**2,691**	**3,216**	**2,835**	**2,866**	**3,158**	**2,362**	**2,507**	**2,442**
OPERATING COSTS	**$M**	**(24,504)**	**(4)**	**(6)**	**(10)**	**(15)**	**(19)**	**(19)**	**(507)**	**(813)**	**(835)**	**(918)**	**(901)**	**(1,025)**	**(1,042)**	**(1,078)**	**(1,081)**	**(1,098)**	**(1,101)**	**(1,115)**
Mining	$M	(11,947)	-	-	-	-	-	-	(300)	(400)	(406)	(490)	(474)	(492)	(527)	(559)	(549)	(604)	(598)	(614)
Processing	$M	(7,718)	-	-	-	-	-	-	(115)	(299)	(299)	(299)	(299)	(299)	(299)	(299)	(299)	(299)	(299)	(299)
G&A	$M	(2,070)	-	-	-	-	-	-	(83)	(77)	(77)	(77)	(78)	(78)	(79)	(80)	(80)	(79)	(79)	(79)
Land & Royalty Payments	$M	(2,769)	(4)	(6)	(10)	(15)	(19)	(19)	(8)	(37)	(53)	(52)	(51)	(155)	(138)	(139)	(153)	(117)	(125)	(122)
Operating Cash Flow Before Tax	**$M**	**37,396**	**(4)**	**(6)**	**(10)**	**(15)**	**(19)**	**(19)**	**611**	**2,170**	**2,000**	**1,826**	**1,790**	**2,192**	**1,793**	**1,788**	**2,077**	**1,264**	**1,406**	**1,327**
Taxes	**$M**	**(5,801)**	**-**	**-**	**-**	**-**	**-**	**-**	**(68)**	**(136)**	**(233)**	**(56)**	**(268)**	**(429)**	**(293)**	**(285)**	**(340)**	**(152)**	**(155)**	**(142)**
Alaska State Income Tax	$M	(1,633)	-	-	-	-	-	-	(23)	(45)	(77)	(18)	(74)	(119)	(81)	(79)	(94)	(42)	(43)	(39)
Alaska Mining License Tax	$M	(1,094)	-	-	-	-	-	-	-	-	-	-	(55)	(88)	(60)	(59)	(70)	(31)	(32)	(29)
Federal Income Tax	$M	(3,074)	-	-	-	-	-	-	(46)	(91)	(156)	(37)	(139)	(222)	(152)	(147)	(176)	(78)	(80)	(74)
Operating Cash Flow After Tax	**$M**	**31,595**	**(4)**	**(6)**	**(10)**	**(15)**	**(19)**	**(19)**	**543**	**2,034**	**1,767**	**1,771**	**1,522**	**1,763**	**1,500**	**1,503**	**1,737**	**1,112**	**1,251**	**1,185**
CAPITAL COST	**$M**	**(11,981)**	**(222)**	**(368)**	**(1,024)**	**(1,934)**	**(2,659)**	**(2,641)**	**(951)**	**(141)**	**(81)**	**(66)**	**(192)**	**(69)**	**(99)**	**(79)**	**(208)**	**(74)**	**(66)**	**(65)**
Initial Capital	$M	(9,233)	(222)	(354)	(1,010)	(1,921)	(2,646)	(2,628)	(451)	-	-	-	-	-	-	-	-	-	-	-
Sustaining Capital	$M	(2,325)	-	-	-	-	-	-	(487)	(128)	(68)	(53)	(179)	(56)	(85)	(66)	(195)	(61)	(53)	(52)
Closure Costs (Inc. Trust Fund)	$M	(423)	-	(13)	(13)	(13)	(13)	(13)	(13)	(13)	(13)	(13)	(13)	(13)	(13)	(13)	(13)	(13)	(13)	(13)
Working Capital																				
Change in Working Capital	$M		-	-	-	-	-	-	(149)	(162)	8	(9)	6	(36)	18	(7)	(17)	41	(8)	2
Net Cash Flow																				
Before Tax	$M	25,415	(226)	(373)	(1,033)	(1,949)	(2,679)	(2,661)	(340)	2,029	1,919	1,761	1,598	2,123	1,694	1,709	1,870	1,189	1,340	1,261
Net After Tax	$M	19,614	(226)	(373)	(1,033)	(1,949)	(2,679)	(2,661)	(408)	1,893	1,686	1,705	1,330	1,694	1,401	1,424	1,529	1,037	1,185	1,119

Table 22-2 Cont'd

Item	Unit	LOM Total	Years															
			Yr 13	Yr 14	Yr 15	Yr 16	Yr 17	Yr 18	Yr 19	Yr 20	Yr 21	Yr 22	Yr 23	Yr 24	Yr 25	Yr 26	Yr 27	Yr 28-247
PRODUCTION																		
Mill Feed Tonnes	kt	504,811	19,528	19,528	19,528	19,581	19,528	19,528	19,528	19,581	19,528	19,528	19,527	19,581	19,527	19,527	8,763	-
Mill Feed Grade	g/t	2.02	2.10	1.80	1.90	1.67	1.90	2.18	2.45	2.20	1.26	1.90	2.00	1.90	0.90	0.86	0.84	-
Mill Recovery	%	90.0	91.22	90.18	90.92	89.37	89.72	90.28	89.69	90.00	89.17	90.15	90.27	90.63	89.76	90.07	88.76	-
Gold Payable	koz	29,501	1,201	1,018	1,083	939	1,069	1,234	1,378	1,245	705	1,074	1,132	1,083	507	486	211	-
Selling Costs	$M	(51)	(2)	(2)	(2)	(2)	(2)	(2)	(2)	(2)	(1)	(2)	(2)	(2)	(1)	(1)	(0)	-
Net Smelter Revenue	**$M**	**61,901**	**2,521**	**2,136**	**2,273**	**1,970**	**2,243**	**2,590**	**2,892**	**2,613**	**1,478**	**2,254**	**2,376**	**2,272**	**1,063**	**1,019**	**442**	**-**
OPERATING COSTS	**$M**	**(24,504)**	**(1,109)**	**(1,110)**	**(1,083)**	**(1,110)**	**(1,112)**	**(1,016)**	**(1,064)**	**(825)**	**(848)**	**(842)**	**(800)**	**(797)**	**(503)**	**(451)**	**(242)**	**(9)**
Mining	$M	(11,947)	(604)	(622)	(589)	(628)	(619)	(508)	(542)	(314)	(390)	(355)	(310)	(312)	(78)	(37)	(25)	-
Processing	$M	(7,718)	(299)	(299)	(299)	(299)	(299)	(299)	(299)	(299)	(299)	(299)	(299)	(299)	(299)	(299)	(134)	-
G&A	$M	(2,070)	(80)	(80)	(80)	(81)	(81)	(81)	(81)	(81)	(81)	(75)	(71)	(71)	(67)	(57)	(57)	-
Land & Royalty Payments	$M	(2,769)	(126)	(109)	(115)	(101)	(113)	(129)	(143)	(130)	(79)	(114)	(119)	(115)	(60)	(58)	(27)	(9)
Operating Cash Flow Before Tax	**$M**	**37,396**	**1,412**	**1,027**	**1,191**	**860**	**1,131**	**1,574**	**1,828**	**1,788**	**630**	**1,412**	**1,577**	**1,476**	**560**	**568**	**200**	**(9)**
Taxes	**$M**	**(5,801)**	**(142)**	**(79)**	**(103)**	**(95)**	**(229)**	**(424)**	**(480)**	**(379)**	**(131)**	**(363)**	**(390)**	**(282)**	**(99)**	**-**	**-**	**-**
Alaska State Income Tax	$M	(1,633)	(1,633)	(39)	(22)	(29)	(26)	(64)	(117)	(133)	(105)	(36)	(101)	(108)	(78)	(28)	-	-
Alaska Mining License Tax	$M	(1,094)	(1,094)	(29)	(16)	(21)	(19)	(47)	(87)	(99)	(78)	(27)	(75)	(80)	(58)	(21)	-	-
Federal Income Tax	$M	(3,074)	(3,074)	(73)	(41)	(53)	(49)	(119)	(219)	(248)	(196)	(67)	(188)	(202)	(146)	(51)	-	-
Operating Cash Flow After Tax	**$M**	**31,580**	**1,270**	**947**	**1,087**	**765**	**901**	**1,149**	**1,347**	**1,408**	**499**	**1,048**	**1,186**	**1,193**	**460**	**522**	**200**	**(9)**
CAPITAL COSTS	**$M**	**(11,981)**	**(198)**	**(51)**	**(55)**	**(40)**	**(220)**	**(53)**	**(112)**	**(30)**	**(148)**	**(23)**	**(18)**	**(16)**	**(49)**	**(13)**	**(13)**	**-**
Initial Capital	$M	(9,233)	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
Sustaining Capital	$M	(2,325)	(185)	(38)	(42)	(27)	(207)	(40)	(99)	(17)	(135)	(10)	(5)	(3)	(36)	-	-	-
Closure Costs - Trust Fund	$M	(423)	(13)	(13)	(13)	(13)	(13)	(13)	(13)	(13)	(13)	(13)	(13)	(13)	(13)	(13)	(13)	-
Working Capital																		
Change in Working Capital	$M		(3)	21	(3)	13	(16)	(3)	(25)	55	60	(43)	(0)	6	116	10	66	62
Net Cash Flow																		
Before Tax	$M	25,415	1,214	975	1,136	820	911	1,521	1,716	1,758	482	1,389	1,558	1,460	511	555	187	(9)
After Tax	$M	19,614	1,072	896	1,033	725	682	1,096	1,236	1,378	352	1,025	1,168	1,177	411	510	187	(9)

Figure 22-3: After-Tax Cash Flow and NPV5% ($2,100/oz Gold Price)

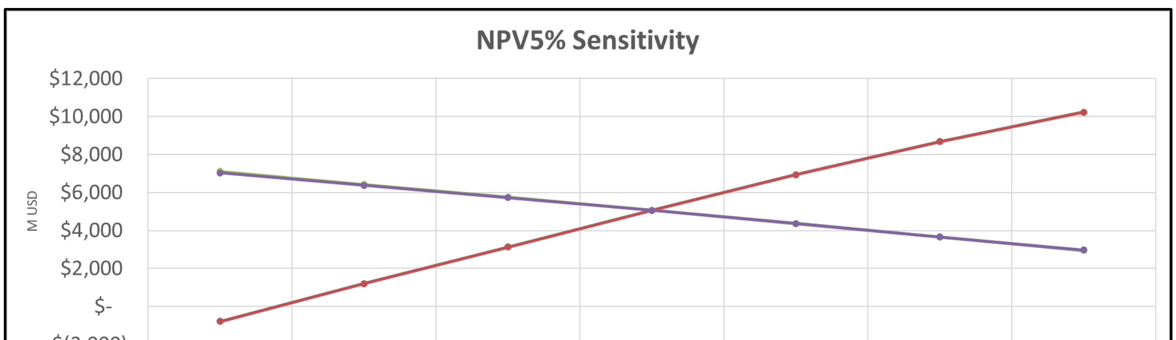


Source: Wood, 2025

22.4 Sensitivity Analysis

Sensitivity analyses have been performed to illustrate the impact on the after-tax and IRR from variations on gold price, gold grade, operating costs, and capital costs. Sensitivities are illustrated in a spider graph shown in Figure 22-4 and Figure 22-5.

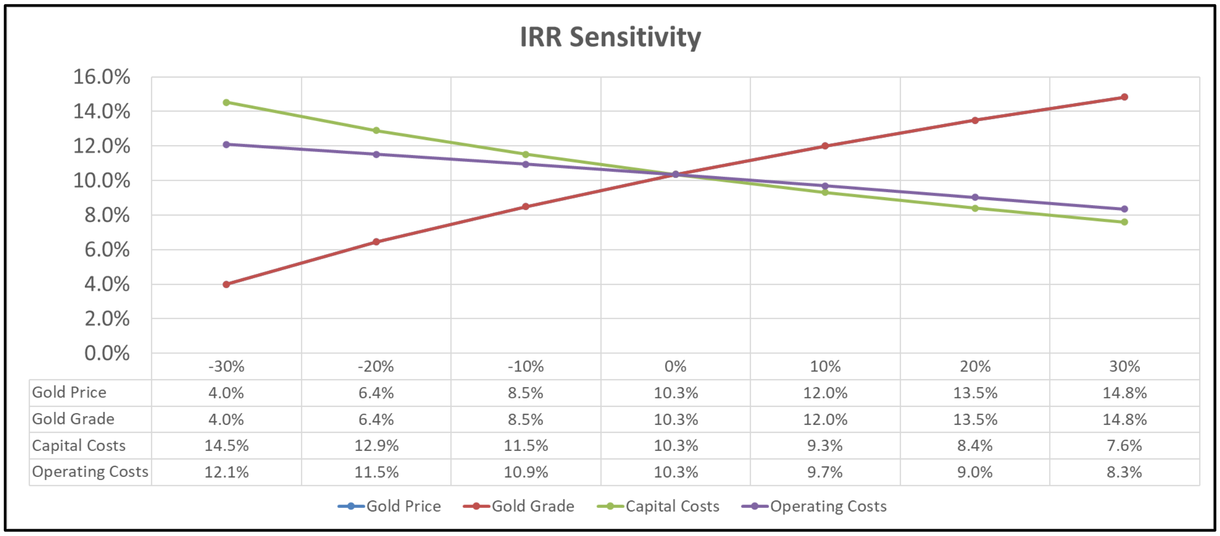
Figure 22-4: After-Tax NPV5% – Sensitivity



	-30%	-20%	-10%	0%	10%	20%	30%
Gold Price	$(787)	$1,206	$3,127	$5,058	$6,940	$8,681	$10,235
Gold Grade	$(782)	$1,210	$3,128	$5,058	$6,938	$8,679	$10,231
Capital Costs	$7,109	$6,428	$5,744	$5,058	$4,356	$3,650	$2,944
Operating Costs	$7,026	$6,381	$5,728	$5,058	$4,362	$3,666	$2,978

Source: Wood, 2025

Figure 22-5: After-Tax IRR – Sensitivity



Source: Wood, 2025

The Project is most sensitive to fluctuations in the gold price and feed grade, and less sensitive to changes in capital and operating costs.

Table 22-3 lists the sensitivities of after-tax cash flow, NPV, IRR, and payback to variations in gold price from a range of $1,470-$2,730/oz.

Table 22-3: Base Case Project Sensitivity to Gold Price

Gold Price ($/oz)	After Tax Cash Flow ($M)	After Tax NPV_5 ($M)	After Tax IRR (%)	Payback (years)
1,470	5,977	(787)	4.0	13.0
1,680	10,599	1,206	6.4	9.2
1,890	15,096	3,128	8.5	7.5
2,100	19,614	5,058	10.3	6.5
2,310	24,021	6,940	12.0	5.6
2,520	27,856	8,681	13.5	5.1
2,730	31,160	10,235	14.8	4.6

Note: Base case is highlighted.

23.0 ADJACENT PROPERTIES

There are no adjacent properties that are relevant to this Report.

24.0 OTHER RELEVANT DATA AND INFORMATION

The QP authors are not aware of any additional information or explanation necessary to make the Report understandable and not misleading.

25.0 INTERPRETATION AND CONCLUSIONS

25.1 Summary

The following is a summary of the relevant results and interpretations in the feasibility study. This Report updates the technical and economic information in the feasibility study on the Project, and provides a current Report.

25.2 Agreements, Mineral Tenure, Surface Rights, and Royalties

Information from NOVAGOLD and their legal experts supports that the mining tenure held and Property agreements are valid and support declaration of Mineral Resources, Mineral Reserves and demonstration of economic viability of the Project.

25.3 Geology, Mineralization and Exploration

Knowledge of the deposit settings, lithologies, and structural and alteration controls on mineralization is sufficient to support Mineral Resource and Mineral Reserve estimation.

The deposit models as used in the exploration programs have been appropriate to the style and setting of the mineralization. Additional exploration potential remains in the Project area.

25.4 Drilling and Data Analysis

The type of drilling, drill spacing and quality of sampling and analysis are considered reliable to support the purposes used in the Report.

The quantity and quality of the lithological, geotechnical, collar and downhole survey data collected in the exploration and infill drill programs are sufficient to support Mineral Resource and Mineral Reserve estimation and the confidence categories assigned. Appropriate data verification procedures have been conducted by the QP authors.

25.5 Metallurgical Testwork

The quantity and quality of the metallurgical testwork are sufficient to support Mineral Resource and Mineral Reserve estimation and the confidence categories assigned.

Metallurgical testwork and associated analytical procedures were performed by recognized testing facilities, and the tests performed were appropriate to the mineralization type.

Samples selected for testing were representative of the various types and styles of mineralization at the Project. Samples were selected from a range of depths within the deposit. Sufficient samples were taken so that tests were performed on sufficient sample mass.

Testwork completed to date has shown that the ore requires pre-treatment prior to cyanidation to recover the gold. Process development work has determined that POX is the preferred method of pretreatment. Extensive testwork on composites has shown that acceptable gold recoveries can be produced through a combination of flotation pre-concentration, POX, and CIL cyanidation.

A small portion of oxidized (altered) ore exists in the deposit and has been accounted in the production schedule through stockpiling. Degradation of the sulfide ore by oxidation in the stockpile will also affect the flotation recovery and has been accounted for.

The flowsheet includes sulfide flotation pre-concentration with POX followed by CIL, providing an estimated total LOM gold recovery of 90.0%.

25.6 Mineral Resource Estimates

The Mineral Resources for the Project, which have been estimated using core and RC drill data, have been prepared in accordance with industry best practices as described in CIM MRMR Best Practice Guidelines and CIM Definition Standards.

25.7 Mineral Reserve Estimates

The Mineral Reserves for the Project appropriately consider modifying factors and have been estimated using industry best practices as described in CIM MRMR Best Practice Guidelines and are in accordance with CIM Definition Standards.

25.8 Mine Methods

The proposed Project will be a conventional, large-tonnage, open-pit operation designed to provide for a nominal process throughput of 19.5 Mt/a, or 53,500 t/d. The operating mine life is estimated at 24 years based on the planned processing rate.

The mine plan developed for the Project envisages mining seven pit phases at ACMA, and four pit phases at Lewis.

The mine design, complete with haulage access, includes 504.8 Mt of ore containing 32.8 Moz of contained gold and has a strip ratio of 6.52:1.

Donlin will be mined by a combination of bulk and selective mining. The SMU block size of selective mine areas is 12 m x 12 m x 6 m. The SMU block size of bulk mine areas is 12 m x 12 m x 12 m. The bench height will be either 6 m or 12 m, depending on mining selectivity requirements.

The mine design incorporates geotechnical and hydrogeological considerations.

25.9 Mine Geotechnical

The Report includes recommendations in Section 26 to support the upcoming feasibility study to confirm assumptions and mitigate potential risks in the pit slope design and provide reasonable contingency allowances.

25.10 Recovery Methods

The process equipment and current designs for the Project are appropriate to the mine plan, and to the Project setting.

The process flowsheet consists of sulfide flotation pre-concentration with POX followed by CIL to produce gold at acceptable recoveries through the application of autoclave sulfide oxidation technology.

A semi-autogenous grinding circuit followed by pebble crushing and in-series ball milling has been selected for the comminution requirements based on the ability to handle the variable hardness and quality of the ore.

The flotation flowsheet will provide a circuit design that will maximize sulfide recovery, as demonstrated during numerous bench and pilot plant metallurgical tests.

POX of the flotation concentrate will allow higher recoveries in CIL cyanidation than direct cyanidation of the flotation concentrate, as demonstrated through extensive pilot-plant testing.

Effective neutralization of acidic solutions can be carried out through the efficient use of the natural carbonate content of the flotation tailings, thereby reducing the amount of lime or limestone that must be transported to site.

Effective management of arsenic in the processed ore is possible through using the POX circuit. The ore has sufficient iron content to permit precipitation of arsenic with iron to form stable forms of arsenic precipitation products, suitable for long term storage in the TSF.

Mercury emissions can be effectively controlled with the proposed mercury abatement systems on the POX, carbon regeneration, electrowinning, retort, and bullion smelting processing equipment. The proposed mercury recovery technology for the autoclave off gas systems achieves current known best practices for this application.

25.11 Project Infrastructure

The infrastructure requirements and current designs for such Project infrastructure are appropriate to the mine plan, and to the Project setting.

Supplies will be shipped on ocean barges where cargo will be transferred to river barges at Bethel and transported to the Jungjuk Port where an access road connects to the Project site.

The Project involves several development sites considerable distances apart, incurring high infrastructure costs to provide interconnecting roads, pipelines, services, and utilities. The decision to use material from the plant site excavation as a borrow source for constructing the starter tailings dam is an effective way to reduce the site preparation costs.

Off-site infrastructure will be arranged, designed, and constructed using techniques that are proven to result in functional and durable facilities suited to their remote location and cold environment.

Although the terrain is diverse and severe through the Alaskan Range, the pipeline route has been field verified by construction and survey personnel.

The electrical power generation and distribution system has been designed to meet the requirements of the proposed loads, the site location, and facility layout. Electric power for the Project site will be generated from a dual-fueled (natural gas and diesel) reciprocating engine power plant with a steam turbine utilizing waste heat recovery from the engines.

25.12 Water Management

The overall water management strategy is appropriate to the mine plan, and to the Project setting.

The water management strategy incorporates a range of engineered solutions and adaptive approaches to address both contact and noncontact water throughout all phases of the mine lifecycle.

The water balance modeling approach is appropriate for the Project, as it represents both surface and subsurface flow contributions in alignment with site hydrologic processes and supports sound design and closure planning.

The overall WTP process flow appears technically sound. Considering the wide range of metals and constituents requiring treatment (including challenging species, such as selenium, cyanide, and sulfate), the selected approach is appropriate.

Results of geochemical characterization studies indicate that most waste rock has low ARD potential but some samples are uncertain or PAG. Arsenic, antimony, and mercury concentrations are elevated due to mineralization, with arsenic showing potential for leaching under both acidic and non-acidic conditions.

The proposed waste rock handling and segregation strategy is well-suited for the current project phase and reflects an understanding of ARD and metal leaching risks. Blending PAG materials with NAG mitigates long-term acid generation and groundwater seepage and aligns with industry best practices.

The proposed depressurization and dewatering strategy reflects an approach to managing groundwater inflows and maintaining pit slope stability.

25.13 Tailings Storage Facility

The TSF design is sufficient to address requirements for regulatory approval of dam safety and demonstrate familiarity and experience with good practice for investigation and design of tailings facilities.

The gridded seismicity source is an important contributor to the seismic hazard and is largely based on historical seismicity. The record of historical seismicity in the vicinity of the site is brief and incomplete due to the sparse seismic network in the region.

The Report includes recommendations in Section 26 to support the upcoming feasibility study to confirm assumptions and mitigate potential risks in the tailings storage design and provide reasonable contingency allowance.

25.14 Market Studies and Contracts

NOVAGOLD's portion of gold production is likely to be sold on the spot market, by marketing experts retained by or on behalf of NOVAGOLD. Gold can be readily sold on numerous markets throughout the world and it is not difficult to ascertain its market price at any time.

The long-term forecast gold price of $2,100/oz used for mine planning and cash flow analysis was based on a combination of information derived from a number of reputable banks as well as cash flow prices used in technical reports filed in Canada over the previous 12-month period and historical price averaging.

25.15 Environmental, Permitting and Social Community Impact

Donlin Gold LLC has obtained nearly all permits required for the start of operations, has completed nearly all permitting phases and is in the process of finalizing remaining state permits and supplementing the environmental analysis under NEPA. Supplements to the current EIS have been mandated to assess potential impacts of a larger hypothetical dam breach. Authorizations for the natural gas pipeline ROW have been received.

A trust fund combined with financial assurance is expected to meet Alaska's financial assurance requirements for mine closure. This includes the funding required to generate sufficient cash flow to cover the spillway construction from the TSF to Crevice Creek, the WTP including perpetual water treatment and monitoring; and associated facility and access maintenance.

Donlin Gold LLC is committed to sustainable development to benefit local communities over the long term by providing opportunities for direct employment, local procurement, and community development projects.

25.16 Capital and Operating Cost Estimates

Capital and operating costs have been adequately accounted for using the assumptions in this Report. The level of accuracy for the estimate is ±25% with a contingency of 13.8%. The total estimated initial capital cost to design and build the Project is $9,233 million and sustaining capital costs are estimated at $2,325 million.

The operating costs over the LOM are estimated at $24,504 million.

25.17 Economic Analysis

The economic analysis in the feasibility study demonstrates the economic viability of the Mineral Reserves. Under the assumptions presented in this Report, the after-tax Project NPV at a discount rate of 5% is $5,058 million and the IRR is 10.3% and a payback period of 6.5 years.

The Project is most sensitive to variations in the gold price and gold grade and is less sensitive to variations in operating costs and capital costs.

25.18 Risks

25.18.1 Summary

The Project is subject to risks that are commonly expected to exist with large, undeveloped mines and risks that are specific to Arctic conditions. These include the following:

25.18.2 Mineral Resource Estimates

Risks identified for Mineral Resources are uncertainties in, or changes from, what was assumed in the estimate for:

- Gold price
- Unrecognized variability in the metallurgical recoveries
- Uncertainties in the geotechnical characteristics of the rock mass and the impact on the pit slope angles
- Uncertainties to the inputs to the resource cut-off
- Gold threshold for defining the indicator mineralized shells
- Uncertainties in the interpretations of fault geometry, in particular the Vortex and Lo faults
- Search orientations used for grade estimation
- Uncertainty in the geological model
- Mineral Resource confidence classification criteria.

25.18.3 Mine Geotechnical

Risk factors and uncertainties associated with pit design are:

- The orientation and location of faults: faults with unfavorable orientation (e.g., dipping out of the slope face) may act as failure planes for planar or wedge-type collapses.

- Strength, spacing, and persistence of weak joints and strata (e.g. ash layers and shale slickensides) may cause pit slope failures.
- Unrecognized structural complications, low strength of joint with slickensides, faults, or ash layers, or unfavorable groundwater conditions (especially at faults) could introduce unfavorable pit slope stability conditions.
- Limited information available on how major faults affect the groundwater systems.
- Unexpected differences from the assumed water encountered in the pit walls producing extensive seepage into the pit requiring additional horizontal drains and pumping wells.

The risk associated with these uncertainties is that slope designs may overestimate stability, leading to unexpected failure that can damage haul roads, equipment, and infrastructure, and may require temporary or permanent cessation of mining in affected areas. In severe cases, failures have resulted in injury or loss of life. Conservative slope designs and hazard zoning are often implemented in structurally complex areas. These approaches reduce risk but may constrain pit expansion and limit ultimate pit depth and can significantly affect both safety and economic outcomes.

25.18.4 Waste Rock Facility

Risks and uncertainties associated with the WRF are:

- A risk associated with the stability of the WRF design is the presence of organic and ice-rich overburden deposits in the interior foundation of the WRF that could result in instability. This could limit overburden stripping due to lack of available dump space with possible impacts on ore stripping, or could pose operational safety issues.
- The presence of organic and ice-rich overburden deposits in the interior foundation of the WRF could result in instability that limit overburden stripping due to lack of available dump space with possible impacts on ore stripping, or could pose operational safety issues.

25.18.5 Tailings and Geotechnical

Risks and uncertainties associated with the TSF are:

- The proposed use of a geotextile for filter protection of the TSF underdrain system could delay approval for construction of the tailings dam.

- Weak ash layers or shale joints with slickensides could exist in the foundation of the TSF, and in conjunction with undrained or strain-softening behavior could require flatter slopes than the current design, increasing required fill volume and cost.
- Insufficient or inadequate borrow material could delay TSF embankment construction resulting in delays or limitations for production.
- A failure of a temporary water diversion dam could create dam safety or operational issues for the TSF with a resultant impact on production.
- The presence of organic and ice-rich overburden deposits in the interior foundation of the WRF could result in instability that limit overburden stripping due to lack of available dump space with possible impacts on ore stripping, or could pose operational safety issues.

25.18.6 Recovery Methods

Risks and uncertainties associated with the process design are:

- Sulfur content variability in mill feed poses a risk to production. Years with lower sulfur content may result in reduced processing efficiency, making it challenging to achieve planned production levels
- Stockpiling sulfide ore may lead to oxidation that could negatively impact flotation performance and gold recovery.

25.18.7 Project Infrastructure

Risks and uncertainties associated with the infrastructure design are:

- There are known to be intermittent areas of permafrost and poor ground conditions at the various facility sites that could affect foundation design and site preparation.
- Concrete retaining walls are used within the plant site footprint to separate tiers of the facility. Over the 27 year operation of the facility, these retaining walls pose a risk if they begin to fail.
- The design of plant site buildings and other structures adopted the seismic provisions of the 2006 International Building Code which may not be in accordance with the current code 2021 edition.

- The overall number of proposed Horizontal Directional Drill crossings (HDDs) on the natural gas pipeline is currently eight. Additional HDDs may be required based on the number of streams along the proposed route. Further construction evaluation and regulatory agency consultations will be necessary to determine the total number of HDDs required.

25.18.8 Water Management

Risks and uncertainties associated with water management are:

- Groundwater data near major faults remains sparse, increasing uncertainty in hydrogeologic interpretations
- Use of outdated MODFLOW versions limits accuracy for complex pit simulations and future pit lake modeling.
- Without appropriate controls, widespread arsenic contamination of water resources could occur.

25.18.9 Environmental, Permitting, and Social Community Impact

The Project reliance on natural gas introduces a strategic risk due to Alaska's limited supply and potential permitting delays. These factors could affect energy availability, impacting construction timelines, and operational continuity.

26.0 RECOMMENDATIONS

26.1 Summary

The QPs have made the following recommendations with a total budget of $1.64 million. This total does not include the planned feasibility study update.

26.2 Mineral Resource Estimates

Incorporate the 2025 drilling data information into both the geological model and Mineral Resource block model in the next feasibility update.

Estimated cost is $40,000.

26.3 Mine Planning

The following mining studies are recommended:

- Evaluate the opportunities to reduce dilution to increase the gold grade of the mill feed
  - Past studies have indicated a range of dilution, including 7.1% in 2021 and 29% in this Report
  - Dilution estimation approach utilizing selective shape generation such as MSO or alternative solutions
  - Fleet sizing and selective mining unit optimization used in Mineral Resource estimation
- Review the blending strategy in the mine plan to manage sulfur content in the concentrate feed to the POX plant to ensure minimum sulfur content levels are achieved through the LOM.
- Evaluate technological advancements in mining equipment that have occurred since the previous feasibility study was completed; automation, alternative fuels/electrification.

Estimated cost is $150,000.

26.4 Mine Geotechnical

The upcoming feasibility study update should include a pit slope design report with the following scope:

- Confirm assumptions and mitigate potential risks associated with estimates of strength parameters for fault gouge, shale slickensides and ash beds by performing index testing, mineralogy determination, and advanced strength testing with a preference for oriented direct shear test on shale slickensides and direct simple shear on ash beds
- Update the geologic model with recent reinterpretation of the Runway fault as folding rather than faulting
- Drill four holes totaling 370 m in the southeast ACMA pit area to better characterize Lo1 fault and three holes totaling 275 m to further delineate and characterize the Divide fault. With an all in drilling cost of $350/m, the cost to drill these holes will be approximately $225,000.
- Evaluate alternative distributions (e.g., log-normal) for joint spacing and persistence
- Perform targeted piezometer installation, pumping and airlift testing, especially at the Lo1 fault to confirm assumptions and mitigate potential risks by associated with the assumed 100–300 m extent of the depressurized envelope for faults.

The estimated cost of $625,000.

26.5 Waste Rock Facility

An operational placement plan should be developed for the WRF facility that considers potential internal instability of the WRF.

Estimated cost is $75,000.

26.6 Tailings and Geotechnical

The upcoming feasibility study should include a geotechnical design with the following scope:

- Provide additional support for the assumed consequence classification of the TSF and the resulting selection of design criteria, or alternatively, a dam breach and inundation study
- Perform additional testing on ash layers and shale slickenside, with development of strength parameters and confidence intervals for use in stability analyses

- Evaluate the potential for brittle behavior in weak foundation units
- Use appropriate methodologies that account for potential brittle or undrained behavior for embankment design analyses
- Complete a tailings placement plan for early operations (2–5 years post start-up), including sensitivity studies based on a range of defensible beach slopes
- Complete a fill placement plan to confirm suitable borrow availability during early operations, with appropriate contingency volumes
- Complete a closure plan for the TSF detailed enough to estimate potential post-operation liabilities and include a risk assessment that considers tailings settlement and seismic loading
- Complete an outline for a tailings management plan (TMP) to define tailings governance and a plan for TMP development
- Perform further evaluation to justify whether the gridded seismicity source from the USGS 2023 NSHM for Alaska is appropriate for the site-specific SHA considering the sparse seismic network, near the site
- Reassess Project risks associated with the 1:100-year IDF criteria for temporary water diversion dams and consider:
    - The consequences of diversion dam failure on TSF safety and operational capacity
    - The potential benefits of adopting more robust criteria to strengthen public support for project approval.

The estimated cost for this work is $500,000.

26.7 Project Infrastructure

The following recommendations are made:

- On the natural gas line one set of pig traps (launcher/receivers) is currently proposed near the mid-point of the pipeline. Further analysis is recommended to find one to two additional trap locations along the route.
- Review the design of plant site buildings and other structures in accordance with the 2021 International Building Code.

Estimated cost is $50,000.

26.8 Water Management

The water management strategy for the Project integrates a combination of engineered solutions and adaptive approaches to effectively manage both contact and non-contact water throughout all phases of the mining lifecycle. The following recommendations aim to further improve system performance and resilience as the Project progresses to the next phases of development:

- Data collected during the 2025 drilling campaign to validate the geology structural model should be interpreted and integrated with the hydrogeological conceptualization. This data should include results from targeted hydraulic testing and head measurements conducted near major faults, to improve understanding of their influence on groundwater flow.
- Future updates to the hydrogeologic model should prioritize expanding the model domain to include adjacent watersheds near the TSF and WRF, as current drawdown predictions approach these boundaries. Incorporating newer versions of MODFLOW software is recommended to enable flexible gridding, improved discretization around the open pit, and enhanced representation of geologic details.
- Conduct a geochemical characterization to determine whether arsenic leaching represents a significant concern to long-term water quality. If it is a concern, then develop a comprehensive plan for managing arsenic leaching from waste rock.
- Confirm underlying water quality assumptions used in the design of the Operations WTP by incorporating updated data and evaluating any new information.

Estimated cost for this work is $100,000.

26.9 Environmental and Permitting

The following are recommended:

- Further address supply and demand concerns for the Project and for Anchorage-based communities for the natural gas pipeline, including considerations for new natural gas storage and pipeline facilities. It is understood this plan is currently under development.
- Complete an evaluation to prevent unregulated water discharge and to comply with current ADPES permit water quality standards. Review the WTP design to confirm sufficient flexibility to manage excess water during exceptional precipitation events.

- Complete a reconciliation of the Project as described in this Report with the existing Project permits as there are differences between components of the Project in the Report from those used as the basis for permitting. The results of this reconciliation should be used to inform the upcoming feasibility study update for the Project.

The estimated cost is $100,000.

26.10 Summary of Costs

Costs recommended work are summarized in Table 26-1.

Table 26-1: Costs for Recommended Work

Description	**Cost ($000s)**
Mineral Resource	40
Mine Plan	150
Mine Geotechnical	625
Waste Rock Storage	75
Tailings and Geotechnical	500
Project Infrastructure	50
Water Management	100
Environmental and Permitting	100
Total	**1,640**

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