ARTEMIS
GOLD INC

Blackwater Gold Mine
British Columbia
NI 43-101 Technical Report on 2024 Expansion Study

Prepared for: Artemis Gold Inc.

Prepared by:
Ms. Sue Bird, P.Eng., Moose Mountain Technical Services.
Mr. Marc Schulte, P.Eng., Moose Mountain Technical Services.
Dr. John A. Thomas, P.Eng., JAT MetConsult Ltd.
Mr. Daniel Fontaine, P.Eng., Knight Piésold Ltd.
Mr. Rolf Schmitt, P.Geo., ERM Consultants Canada Ltd.
Mr. John Dockrey, P.Geo., Lorax Environmental Services Ltd.
Mr. Olav Mejia, P.Eng., Lycopodium Minerals Canada Limited
Mr. Sohail Samdani, P.Eng., Lycopodium Minerals Canada Limited

Report effective date: 21 February, 2024.

CERTIFICATE OF QUALIFIED PERSON

I, Sue Bird, P.Eng., am employed as the Senior Vice President of Geology and Underground, with Moose Mountain Technical Services, with an office address of #210 1510 2nd Street North Cranbrook, BC V1C 3L2.

This certificate applies to the technical report titled “Blackwater Gold Mine, British Columbia, NI 43-101 Technical Report on 2024 Expansion Study” that has an effective date of 21 February, 2024 (the “technical report”).

I am a member of the Association of Professional Engineers and Geoscientists of British Columbia (#25007). I graduated with a Geologic Engineering degree (B.Sc.) from Queen’s University in 1989 and a M.Sc. in Mining from Queen’s University in 1993.

I have worked as an engineering geologist for over 25 years since my graduation from university. I have worked on precious metals, base metals and coal mining projects, including mine operations and evaluations. Similar resource estimate projects specifically include those done for Summit, NM, Spanish Mountain, BC, Marban, QB as well as numerous due diligence gold projects in the southern US done confidentially for various clients.

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) for those sections of the technical report that I am responsible for preparing.

I visited the Blackwater Gold Mine on July 14, 2020, a duration of one day.

I am responsible for Sections 1.1, 1.2, 1.3, 1.4, 1.5, 1.6, 1.7, 1.8, 1.9, 1.10, 1.11, 1.12, 1.14, 1.15, 1.27.2; Sections 2.1, 2.2, 2.3, 2.4.1, 2.5, 2.6, 2.7; Section 3; Sections 4.1, 4.2, 4.3, 4.4, 4.5, 4.6, 4.7, 4.11; Section 5.1, 5.3, 5.6; Section 6; Section 7; Section 8, Section 9, Section 10.1, 10.2, 10.3, 10.4, 10.5, 10.6, 10.8, 10.9, 10.10, 10.11, 10.12, 10.13; Section 11; Section 12.1.1; Section 14; Section 23; Section 24; Sections 25.1, 25.2, 25.3, 25.4, 25.6, 25.16.2; and Section 27 of the technical report.

I am independent of Artemis Gold as independence is described by Section 1.5 of NI 43–101.

I have previously co-authored technical reports on the Blackwater Gold Mine:

- Kalanchey, R., Bird, S., Dermer, G., Fontaine, D., Garner, J., Schulte, M., Lee, P., Dockrey, J., and Thomas J.A., 2021: NI 43-101 Technical Report on Updated Feasibility Study: report prepared for Artemis Gold Inc. by Ausenco Engineering Canada, Moose Mountain Technical Services, Knight Piésold Ltd., Allnorth, ERM, LORAX, and JAT Met Consult Ltd., effective date 10 September, 2021;
- Bird, S., Fontaine, D., Meintjes, T., Schulte, M., Thomas, J., 2020: Technical Report, Blackwater Gold Project British Columbia NI 43-101 Technical Report on Pre-Feasibility Study: report prepared for Artemis Gold Inc. by Moose Mountain Technical Services, Knight Piésold Ltd, JAT Met Consult Ltd, effective date 26 August, 2020.

I have read NI 43–101 and the sections of the technical report for which I am responsible have been prepared in compliance with that Instrument.

As of the effective date of the technical report, to the best of my knowledge, information and belief, the sections of the technical report for which I am responsible contain all scientific and technical information that is required to be disclosed to make those sections of the technical report not misleading.

Dated: 8 April, 2024

“Signed and sealed”

Sue Bird, P.Eng.

CERTIFICATE OF QUALIFIED PERSON

I, Marc Schulte, P.Eng., am employed as the Vice President of Engineering and Operations, with Moose Mountain Technical Services, with an office address of #210 1510 2nd Street North Cranbrook, BC V1C 3L2.

This certificate applies to the technical report titled "Blackwater Gold Mine, British Columbia, NI 43-101 Technical Report on 2024 Expansion Study" that has an effective date of 21 February, 2024 (the "technical report").

I am a member of the self-regulated association Engineers and Geoscientists BC (#54035). I graduated with a Bachelor of Science in Mining Engineering from the University of Alberta in 2002.

I have worked as a mining engineer for over 20 years since my graduation from university. Throughout my career I have worked on numerous open pit precious metals projects, within project engineering studies and within mining operations, on mineral reserve estimates, mine planning, and mine cost estimates.

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) for those sections of the technical report that I am responsible for preparing.

I visited the Blackwater Gold Mine on July 14, 2020, a duration of one day.

I am responsible for Sections 1.1, 1.2, 1.3, 1.12, 1.16, 1.17, 1.18, 1.22, 1.23 (mining costs), 1.24 (mining costs) 1.25, 1.26, 1.27, 1.28, 1.29; Sections 2.1, 2.2, 2.3, 2.4.2, 2.5, 2.6, 2.7; Section 3; Section 12.1.2; Section 15, Section 16, Section 19; Sections 21.1, 21.2.1, 21.2.2, 21.2.7, 21.3.1, 21.3.2, 21.3.6, Section 22, Section 24, Sections 25.1, 25.7, 25.8, 25.12, 25.13 (mining costs), 25.14 (mining costs), 25.15, 25.16, 25.17, and Sections 26.1, 26.2, and Section 27 of the technical report.

I am independent of Artemis Gold as independence is described by Section 1.5 of NI 43–101.

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- Kalanchey, R., Bird, S., Dermer, G., Fontaine, D., Garner, J., Schulte, M., Lee, P., Dockrey, J., and Thomas J.A., 2021: NI 43-101 Technical Report on Updated Feasibility Study: report prepared for Artemis Gold Inc. by Ausenco Engineering Canada, Moose Mountain Technical Services, Knight Piésold Ltd., Allnorth, ERM, LORAX, and JAT Met Consult Ltd., effective date 10 September, 2021;
- Bird, S., Fontaine, D., Meintjes, T., Schulte, M., Thomas, J., 2020: Technical Report, Blackwater Gold Project British Columbia NI 43-101 Technical Report on Pre-Feasibility Study: report prepared for Artemis Gold Inc. by Moose Mountain Technical Services, Knight Piésold Ltd, JAT Met Consult Ltd, effective date 26 August, 2020.

I have read NI 43–101 and the sections of the technical report for which I am responsible have been prepared in compliance with that Instrument.

As of the effective date of the technical report, to the best of my knowledge, information and belief, the sections of the technical report for which I am responsible contain all scientific and technical information that is required to be disclosed to make those sections of the technical report not misleading.

Dated: 8 April, 2024

“Signed and sealed”

Marc Schulte, P.Eng.

CERTIFICATE OF QUALIFIED PERSON

I, John Alan Thomas, P.Eng., am employed as the President of JAT Met Consult Ltd. of 5766 Goldenrod Crescent, Delta, BC V4L 2G6.

This certificate applies to the technical report titled "Blackwater Gold Mine, British Columbia, NI 43-101 Technical Report on 2024 Expansion Study" that has an effective date of 21 February, 2024 (the "technical report").

I am a member of the Association of Professional Engineers and Geoscientists of British Columbia (#125986).

I graduated with Chemical Engineering degrees (B.Sc. M.Sc. and Ph.D.) from the University of Manchester in 1969, 1971 and 1973 respectively.

I have worked as a process engineer for over 50 years since my graduation from university. I have worked on precious metals and base metals, including process development, engineering, project management and mine operation. Similar projects specifically include those done for Atlantic Gold, Infinito Gold, Bolivar Gold Corp., Bolivar Goldfields and Star Mining.

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) for those sections of the technical report that I am responsible for preparing.

I visited the Blackwater Gold Mine on May 7, 2020, a duration of one day.

I am responsible for Sections 1.1, 1.2, 1.3, 1.12, 1.13, 1.20.1 (except process plant), 1.23 (Owner costs), 1.24 (G&A costs), 1.27.1, 1.29; Sections 2.1, 2.2, 2.3, 2.4.3, 2.6; Section 12.1.3; Section 13; Section 18.1, 18.2, 18.3, 18.9, 18.10, 18.11, 18.12, 18.13; Sections 21.1, 21.2.1, 21.2.5, 21.2.6; 21.2.7, 21.3.1, 21.3.4, 21.3.5, 21.3.6; Sections 25.1, 25.5, 25.13 (Owner costs), 25.14 (G&A costs); 25.16.1; Sections 26.1, 26.3; and Section 27 of the technical report.

I am independent of Artemis Gold as independence is described by Section 1.5 of NI 43–101.

I have previously co-authored technical reports on the Blackwater Gold Mine:

- Kalanchey, R., Bird, S., Dermer, G., Fontaine, D., Garner, J., Schulte, M., Lee, P., Dockrey, J., and Thomas J.A., 2021: NI 43-101 Technical Report on Updated Feasibility Study: report prepared for Artemis Gold Inc. by Ausenco Engineering Canada, Moose Mountain Technical Services, Knight Piésold Ltd., Allnorth, ERM, LORAX, and JAT Met Consult Ltd., effective date 10 September, 2021;
- Bird, S., Fontaine, D., Meintjes, T., Schulte, M., Thomas, J., 2020: Technical Report, Blackwater Gold Project British Columbia NI 43-101 Technical Report on Pre-

Feasibility Study: report prepared for Artemis Gold Inc. by Moose Mountain Technical Services, Knight Piésold Ltd, JAT Met Consult Ltd, effective date 26 August, 2020.

I have read NI 43–101 and the sections of the technical report for which I am responsible have been prepared in compliance with that Instrument.

As of the effective date of the technical report, to the best of my knowledge, information and belief, the sections of the technical report for which I am responsible contain all scientific and technical information that is required to be disclosed to make those sections of the technical report not misleading.

Dated: 8 April, 2024

“Signed and sealed”

John Alan Thomas, P.Eng.

CERTIFICATE OF QUALIFIED PERSON

I, Daniel Fontaine, P.Eng., am employed as a Specialist Engineer and Associate, with Knight Piésold Ltd., with an office address at 1400 – 750 West Pender Street, Vancouver, British Columbia, V6C 2T8.

This certificate applies to the technical report titled “Blackwater Gold Mine, British Columbia, NI 43-101 Technical Report on 2024 Expansion Study” that has an effective date of 21 February, 2024 (the “technical report”).

I am a registered Professional Engineer (License #36208) with Engineers and Geoscientists of British Columbia. I graduated from McGill University in 2006 with a bachelor’s degree in civil engineering.

I have practiced my profession for 18 years since graduation from university. I have been directly involved in performing and overseeing geotechnical engineering design, tailings management and water management studies, environmental assessments, and monitoring construction activities for mining projects during this time.

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) for those sections of the technical report that I am responsible for preparing.

I most recently visited the Blackwater Gold Mine from February 7–9, 2024, a duration of three days.

I am responsible for Sections 1.1, 1.3, 1.12, 1.20.2, 1.20.3, 1.23 (tailings and water management costs), 1.27.1, 1.29; Sections 2.1, 2.2, 2.3, 2.4.4, 2.6; Sections 5.2, 5.4, 5.5; Section 10.7; Section 12.1.4; Sections 18.4, 18.5, 18.6, 18.7, 18.8; Sections 21.1, 21.2.1, 21.2.4, 21.2.7; Sections 25.1, 25.10, 25.13 (tailings and water management costs); Sections 26.1, 26.4; and Section 27 of the technical report.

I am independent of Artemis Gold as independence is described by Section 1.5 of NI 43–101.

I have previously co-authored technical reports on the Blackwater Gold Mine:

- Kalanchey, R., Bird, S., Dermer, G., Fontaine, D., Garner, J., Schulte, M., Lee, P., Dockrey, J., and Thomas, J.A., 2021: NI 43-101 Technical Report on Updated Feasibility Study: report prepared for Artemis Gold Inc. by Ausenco Engineering Canada, Moose Mountain Technical Services, Knight Piésold Ltd., Allnorth, ERM, LORAX, and JAT Met Consult Ltd., effective date 10 September, 2021;

- Bird, S., Fontaine, D., Meintjes, T., Schulte, M., Thomas, J., 2020: Technical Report, Blackwater Gold Project British Columbia NI 43-101 Technical Report on Pre-Feasibility Study: report prepared for Artemis Gold Inc. by Moose Mountain Technical Services, Knight Piésold Ltd, JAT Met Consult Ltd, effective date 26 August, 2020.

I have read NI 43–101 and the sections of the technical report for which I am responsible have been prepared in compliance with that Instrument.

As of the effective date of the technical report, to the best of my knowledge, information and belief, the sections of the technical report for which I am responsible contain all scientific and technical information that is required to be disclosed to make those sections of the technical report not misleading.

Dated: 8 April, 2024

“Signed and sealed”

Daniel Fontaine, P.Eng.

Sustainability is our business

CERTIFICATE OF QUALIFIED PERSON

I, Rolf Schmitt, P.Geo., am employed as the Technical Director, ERM Consultants Canada Ltd., with an office address #1000 – 1100 Melville Street, Vancouver, British Columbia, Canada V6E 4A6.

This certificate applies to the technical report titled “Blackwater Gold Mine, British Columbia, NI 43-101 Technical Report on 2024 Expansion Study” that has an effective date of 21 February, 2024 (the “technical report”).

I am a member in good standing of the Nunavut/Northwest Territories Association of Professional Engineers and Geoscientists (NAPEG), License # L4706, (ERM Permit No. P388) and Engineers and Geoscientists of British Columbia, License #19824 (ERM Permit No. 1001271).

I graduated from the University of British Columbia – Honours Bachelor of Science (B.Sc.) Geology (1977), and a Master of Science (M.Sc.) Regional Planning (1985), and University of Ottawa - Master of Science (M.Sc.) Exploration Geochemistry (1993).

I have practiced my profession for over 40 years since graduation; six years in mineral exploration, 20 years in government mining regulation and geochemical research, and 18 years (since 2005) as a senior mining and natural resource regulatory consultant (since 2005). I have been directly involved in directing and managing mine project Environmental Assessments, permitting and due diligence assignments for 18 years throughout Canada and internationally.

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) for those sections of the technical report that I am responsible for preparing.

I have not visited the Blackwater Gold Mine.

I am responsible for Sections 1.1, 1.3, 1.12, 1.21; Sections 2.1, 2.2, 2.3, 2.6; Sections 4.8, 4.9, 4.10; Section 12.1.5; Section 20; Sections 25.1, 25.11; and Section 27 of the technical report.

I am independent of Artemis Gold as independence is described by Section 1.5 of NI 43–101.

I have no previous involvement with the Blackwater Gold Mine.

I have read NI 43–101 and the sections of the technical report for which I am responsible have been prepared in compliance with that Instrument.

As of the effective date of the technical report, to the best of my knowledge, information and belief, the sections of the technical report for which I am responsible contain all scientific and technical information that is required to be disclosed to make those sections of the technical report not misleading.

Dated: 8 April, 2024

“Signed and sealed”

Rolf Schmitt, P.Geo.

CERTIFICATE OF QUALIFIED PERSON

I, John Dockrey, P.Geo., am employed as an Environmental Geochemist, with Lorax Environmental Services Ltd., with an office address at 2289 Burrard Street, Vancouver, BC, Canada, V6J 3H9.

This certificate applies to the technical report titled "Blackwater Gold Mine, British Columbia, NI 43-101 Technical Report on 2024 Expansion Study" that has an effective date of 21 February, 2024 (the "technical report").

I am a registered member with Engineers and Geoscientists BC (Permit to Practice #1001840).

I graduated from the University of Wisconsin, Madison in 2007 with a B.S. in Geology and the University of British Columbia in Vancouver, B.C. in 2010 with an M.Sc. in Geoscience.

I have practiced my profession for 14 years. I have been directly involved in geochemical characterization programs and assessment of acid rock drainage and metal leaching potential at mine sites over this time period.

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) for those sections of the technical report that I am responsible for preparing.

I have not visited the Blackwater Gold Mine.

I am responsible for Sections 1.1, 1.3, 1.12, Sections 2.1, 2.2, 2.3, 2.6; Section 12.1.6; Section 16.10.1; and Section 27 of the technical report.

I am independent of Artemis Gold as independence is described by Section 1.5 of NI 43–101.

I have previously co-authored a technical report on the Blackwater Gold Mine:

- Kalanchey, R., Bird, S., Dermer, G., Fontaine, D., Garner, J., Schulte, M., Lee, P., Dockrey, J., and Thomas J.A., 2021: NI 43-101 Technical Report on Updated Feasibility Study: report prepared for Artemis Gold Inc. by Ausenco Engineering Canada, Moose Mountain Technical Services, Knight Piésold Ltd., Allnorth, ERM, LORAX, and JAT Met Consult Ltd., effective date 10 September, 2021.

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Dated: 8 April, 2024

"Signed and sealed"

John Dockrey, P.Geo.

CERTIFICATE OF QUALIFIED PERSON

I, Olav Mejia, P.Eng., am employed as the Manager, Process Engineering, Lycopodium Minerals Canada Limited, with an office address at Suite 700, 5090 Explorer Drive, Mississauga, Ontario, L4W 4T9, Canada.

This certificate applies to the technical report titled "Blackwater Gold Mine, British Columbia, NI 43-101 Technical Report on 2024 Expansion Study" that has an effective date of 21 February, 2024 (the "technical report").

I am a Professional Engineer of Ontario in Canada with Registration Number 100602612. I graduated from the San Marcos University in California, with a B.Eng. in Chemical Engineering in 1992 and Masters degree in Mineral Processing from the University of British Columbia in 2018.

I have practiced my profession for over 25 years since graduation. I have been directly involved in a gold plant operation as a senior metallurgist and in gold plant process engineering development as a lead/principal engineer.

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) for those sections of the technical report that I am responsible for preparing.

I visited the Blackwater Gold Mine on 14 November 2023, a duration of one day.

I am responsible for Sections 1.1, 1.2, 1.3, 1.12, 1.19, 1.24 (process costs); Sections 2.1, 2.2, 2.3, 2.4.5, 2.6; Section 12.1.7; Section 17, Sections 21.1, 21.3.1, 21.3.3, 21.3.6; Sections 25.1, 25.9, 25.14 (process costs), Sections 26.1, 26.3; and Section 27 of the technical report.

I am independent of Artemis Gold as independence is described by Section 1.5 of NI 43–101.

I have no previous involvement with the Blackwater Gold Mine.

I have read NI 43–101 and the sections of the technical report for which I am responsible have been prepared in compliance with that Instrument.

As of the effective date of the technical report, to the best of my knowledge, information and belief, the sections of the technical report for which I am responsible contain all scientific and technical information that is required to be disclosed to make those sections of the technical report not misleading.

Dated: 8 April, 2024

“signed and sealed”

Olav Mejia, P.Eng.

CERTIFICATE OF QUALIFIED PERSON

I, Sohail Samdani, P.Eng., am employed as the President, Lycopodium Minerals Canada Limited, with an office address at Suite 700, 5090 Explorer Drive, Mississauga, Ontario, L4W 4T9, Canada.

This certificate applies to the technical report titled “Blackwater Gold Mine, British Columbia, NI 43-101 Technical Report on 2024 Expansion Study” that has an effective date of 21 February, 2024 (the “technical report”).

I am a registered Professional Engineer (License # 100089323) with Professional Engineers Ontario (PEO).

I graduated from the University of Toronto in 2003 with a Master of Applied Science degree in Structural Engineering and from the NED university of Engineering & Technology in 2000 with a Bachelor of Engineering degree in Civil Engineering.

I have practiced my profession for over 20 years since graduation. I have been directly involved in performing and overseeing studies and project delivery of mineral processing plants as Study Manager, Project Manager and Study/Project Sponsor.

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) for those sections of the technical report that I am responsible for preparing.

I have not visited the Blackwater Gold Mine.

I am responsible for process plant-related content in Sections 1.1, 1.2, 1.3, 1.12, , 1.20.1, 1.23, 1.27, 1.29; Sections 2.1, 2.2, 2.3, 2.6; Section 12.1.8; Sections 18.1, 18.2, 18.12; Sections 21.1, 21.2.1, 21.2.3, 21.2.5, 21.2.7; Sections 25.1, 25.10, 25.13, 25.16; and Section 27 of the technical report.

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Dated: 8 April, 2024

“Signed and sealed”

Sohail Samdani, P.Eng.

IMPORTANT NOTICE

CONTENTS

TABLES

FIGURES

1.0 SUMMARY

1.1 Introduction

Ms. Sue Bird, P.Eng., Mr. Marc Schulte, P.Eng., Dr. John A. Thomas, P.Eng., Mr. Daniel Fontaine, P.Eng., Mr. Rolf Schmitt, P.Geo., Mr. John Dockrey, P.Geo., Mr. Olav Mejia, P.Eng., and Mr. Sohail Samdani, P.Eng., collectively the Qualified Persons (QPs), prepared an NI 43-101 Technical Report (the Report) on phased operations (Phase 1, Phase 2, and Phase 3; collectively the 2024 Expansion Study) for the Blackwater Gold Mine for Artemis Gold Inc. (Artemis Gold).

The Blackwater Gold Mine is located in central British Columbia, approximately 112 km southwest of Vanderhoof and 446 km northeast of Vancouver.

The construction of the Phase 1 processing plant of 6 Mt/a is well advanced, and this Report assumes that Phase 1 has been completed. The 2024 Expansion Study purpose is to optimize the timing of mine expansion. The expansions are expected to be funded from operating cash flows based on the input assumptions of the study. The expansion study is based on Blackwater's existing Proven and Probable Mineral Reserves. No changes were made to the Mineral Reserve and Mineral Resource estimates in the study. All cost estimates are at a minimum pre-feasibility study level. Phase 1 costs are assumed to be sunk costs; however, debt servicing of the Project loan facility, which partly funded Phase 1, is recognized in the cashflows modelled, as are the various royalties and streams related to the Blackwater Gold Mine.

1.2 Key Outcomes

The 2024 Expansion Study has the following outcomes:

- Proven and Probable Mineral Reserves totalling 334.3 Mt grading 0.75 g/t Au and 5.8 g/t Ag;
- Milling operations are planned in three phases: Phase 1 of 6 Mt/a, Phase 2, initiated in Year 2, consists of a 9 Mt/a expansion to 15 Mt/a, and Phase 3, initiated in Year 7, consists of a 9.5 Mt/a expansion plus optimization of Phase 1 for an additional 0.5 Mt/a, resulting in a total throughput of 25 Mt/a;
- Open pit mine life of 17 years, 15 years of active open pit mining and two years of stockpile treatment;
- Life-of-mine (LOM) capital cost estimate (growth + sustaining, Phases 2 and 3) of C$2,619 M; Phase 1 capital costs, excluding the Project loan facility, are considered to be sunk costs;
- LOM operating cost estimate of C$20.03/t milled;
- After-tax net present value (NPV) of C$3.25 B, using a 5% discount rate;
- Average annual production of 469,000 oz of gold equivalent (AuEq);

- Average annual all-in sustaining costs of US$781/gold ounce (based on selling costs, royalty payments, operating costs, sustaining capital and closure costs, less silver by-product credits and adjustments to stockpile inventory, divided by payable gold ounces);
- Measured and Indicated Mineral Resources (inclusive of Mineral Reserves) totalling 596.8 Mt grading 0.61 g/t Au and 6.4 g/t Ag;
- Inferred Mineral Resources totalling 16.9 Mt grading 0.45 g/t Au and 12.8 g/t Ag.

1.3 Terms of Reference

The Report was prepared to support disclosures in the Artemis Gold news release entitled "Artemis Gold Announces Results of Expansion Study for Blackwater Mine", dated 21 February 2024.

The term "Project" is used to refer to the overall area within the mineral tenure holdings that form the Blackwater Project. The term "Blackwater Gold Mine" is used to refer interchangeably to the mine site area, the existing Blackwater operation, and the expansion of the Blackwater Gold Mine.

This Report provides expansion details planned for the Blackwater Gold Mine:

- Phase 1: assumed to be completed for the purposes of this Report; process plant operating at 6 Mt/a;
- Phase 2: process plant capacity increase of 9 Mt/a in Q4 of Year 2, for an overall 15 Mt/a production rate starting in Year 3;
- Phase 3: process plant capacity increase in Year 7 of 9.5 Mt/a and optimization of Phase 1 plant for an additional 0.5 Mt/a achieving an overall plant capacity of 25.0 Mt/a.

For the purposes of the capital cost estimates, "growth capital" is the sum of the deferred + expansion capital cost estimates.

Mineral Resources and Mineral Reserves are classified using the 2014 edition of the Canadian Institute of Mining and Metallurgy (CIM) Definition Standards for Mineral Resources and Mineral Reserves (the 2014 CIM Definition Standards).

Units used in the Report are metric units unless otherwise noted. Monetary units are in Canadian dollars (C$) unless otherwise stated.

1.4 Project Setting

The Blackwater Gold Mine is readily accessible by vehicle from Vanderhoof using the Kluskus Forest Service Road and the Kluskus–Ootsa Forest Service Road. The Kluskus Forest Service Road joins Highway 16 about 10 km west of Vanderhoof. The mine site can be accessed from the Kluskus–Ootsa Forest Service Road at km 146, using an 18 km-long exploration access road.

A new 13.8 km long mine access road will be built to replace the exploration access road, which will be subsequently decommissioned.

Helicopter access is from bases in Vanderhoof, Prince George, or Quesnel.

The climate is sub-continental, characterized by brief warm summers and long cold winters. Mining operations will be conducted on a year-round basis.

The Project area is very sparsely inhabited; the closest Indian Reserve to the mine site is Tatelkus Lake 28, approximately 15 km away, and three ranches are found within a 20 km radius of the Project site. Some services are available in Vanderhoof, but Prince George is the regional hub with air service from major centres.

The elevations within the Project area range from just over 1,000 m (above sea level) in low-lying areas northeast of the proposed mine site to 1,800 m on the southwest side of the Project area at the summit of Mt. Davidson, which is the highest peak in the Fawnie Range. Bedrock outcrops are limited, and most of the area is covered by 2 m or greater thicknesses of glacial deposits, except for the upper 150 m of Mt. Davidson and a few localized areas at lower elevations. Low-elevation valley bottoms are dominated by stands of lodgepole pine. Hybrid white spruce tends to dominate on moist to wet sites below 1,500 m, while subalpine fir and Englemann spruce are dominant above 1,500 m.

1.5 Mineral Tenure, Surface Rights, Water Rights, Royalties and Agreements

Artemis Gold, through BW Gold, holds a 100% recorded interest in 320 mineral claims and one mineral lease (the Blackwater mining lease), covering a total area of 149,037.3 ha.

The grant of Mines Act permit M-246 provides Artemis Gold with the necessary surface rights to support construction of the required mine infrastructure, including the open pit, access roads, stockpiles, waste rock storage facilities (WRSFs) and tailings storage facility (TSF). Artemis Gold was granted a license of occupation for the power transmission line on 25 April, 2023.

Artemis Gold has obtained all the water rights required to support mine construction and operations.

Artemis Gold's 100% interest in the Blackwater mining lease is subject to three net smelter return (NSR) agreements:

- A 1.5% NSR royalty is payable on former mineral claim 515809 (Dave Claim). The claim covers a portion of the Blackwater deposit;
- A 1% NSR royalty is payable on former mineral claim 515810 (Jarrit Claim). The claim covers a portion of the Blackwater deposit;
- The current agreement would allow Artemis Gold to purchase two-thirds of three Blackwater claims (637203, 637205, and former 637206) NSR royalty for C$1,000,000 at any time, such that a 1% NSR royalty would remain.

Only the royalties with respect to the Dave Option and the Jarrit Option affect the Mineral Resource and Mineral Reserve estimates.

The purchase agreement between Artemis Gold and New Gold Inc. (New Gold) included a gold stream agreement. New Gold maintained a security interest over the Blackwater Gold Mine in connection with the gold stream agreement. On 13 December, 2021, New Gold announced the sale of the gold stream agreement to Wheaton Precious Metals Corp (Wheaton).

1.6 History

Limited exploration activity, on what is now the Project site was first recorded in 1973. Granges Inc. completed geophysical and geochemical surveys and limited drilling between 1973 and 1994. Following some further drilling from 2005 to 2007, the Project was acquired by Richfield Ventures Corp. (Richfield) in early 2009. During the second half of 2009, throughout 2010 and the first five months of 2011, Richfield continued its exploration drilling program at Blackwater.

New Gold purchased Richfield in May 2011 and thereby acquired a 75% interest in the Davidson claims and 100% interests in each of the Dave and Jarrit claims and subsequently acquired Geo Minerals Ltd. and Silver Quest Resources Ltd.

New Gold undertook a major exploration drilling, metallurgical testwork, and feasibility-level engineering program, including completion of a feasibility study in 2013 and subsequent technical report in 2014. Artemis Gold completed the Project acquisition on 21 August 2020. Artemis Gold has acquired all of New Gold's mineral tenures, assets, and rights related to the Project and now hold a 100% interest in the Project.

No production has occurred from the Project area.

1.7 Geological Setting

The Blackwater deposit is an example of an intermediate sulphidation epithermal-style gold–silver deposit.

Mineralization is hosted within felsic to intermediate composition volcanic rocks that have undergone extensive silicification and hydrofracturing in association with pervasive stockwork veined and disseminated sulphide mineralization.

Mineralization is strongly controlled by northwest–southeast-trending structures characterized by zones of tectonic brecciation and chloritic gouge. A major north–south-trending fault dissects the along UTM easting 375,600E, and east–northeast-trending faults were also observed. The major fault represents a well-defined disruption in lithology, alteration, and mineralization patterns and was used to subdivide the resource block model into two structural domains, one to the east of it and one to the west.

The alteration minerals most commonly identified included muscovite, high- and low-temperature illite, ammonium-bearing illite, smectite, silica, biotite, and chlorite.

Gold–silver mineralization is associated with a variable assemblage of pyrite–sphalerite–marcasite–pyrrhotite ± chalcopyrite ± galena ± arsenopyrite (± stibnite ± tetrahedrite ± bismuthite).

1.8 Exploration

Given the lack of bedrock exposures in the immediate Blackwater deposit area, geologic information was obtained primarily by exploration drilling. New Gold mapping of pits and road-cut exposures over the deposit supported the geological interpretation of the deposit in the subsurface.

Soil and stream geochemical surveys over parts of the Property area were undertaken in 2012. A total of 4,517 samples were collected. The results of the soil survey indicated numerous areas displaying multi-element anomalies including gold, zinc, silver, copper, bismuth, and molybdenum, many of which merit follow-up investigation. Results of a restricted stream silt sampling program of 43 samples indicated anomalous copper and zinc values from streams to the northwest and southeast of the Blackwater deposit.

During 2010, Richfield contracted Quantec Geoscience Ltd. of Toronto to conduct a Titan 24 direct current resistivity and induced polarization (IP) chargeability geophysical survey. The results of the survey indicate good correspondence between known mineralization and the Titan IP-resistivity results. In general, zones of significant gold mineralization correlate positively to zones of moderate resistivity and moderate IP chargeability.

Polished section petrographic analysis, X-ray diffraction analysis and whole-rock lithogeochemical analyses were conducted on selected drill samples. A two-phase alteration study was completed to develop the alteration model for the deposit.

1.9 Mineralization

Disseminated gold-silver mineralization is defined by an east–west-trending tabular–conical-shaped deposit with a lateral extent of up to 1,300 m east–west x 950 m north–south. Mineralization remains open at depth in the southwestern part of the deposit as well as to the north and northwest. The centre of the deposit has an average thickness of 350 m and, where open, a vertical extension of up to 600 m. The mineralized zone plunges shallowly to the north and northwest with inferred steep, north-plunging higher-grade mineralized shoots, measuring tens of metres thick, likely influenced by near-vertical structural intersections.

1.10 Drilling

A total of 1,053 core holes (324,839 m) were drilled in the Project area between 2009 and January 2013. Of this total, 134 core holes were completed by Richfield, and 919 core holes by New Gold.

A total of 1,002 core holes are included in the resource database used for estimation purposes. The drilling of 109 condemnation holes has confirmed no economic mineralization beneath the proposed mine infrastructure. Artemis Gold has not conducted any core drilling since acquiring the Project, but completed a pre-production grade control program between November 2020 and March 2021.

The exploration drilling carried out between 2009–2013 consisted predominantly of HQ diameter (63.5 mm) drill core except where a reduction to NQ diameter (47.6 mm) was required to attain target depths.

Geological logging included geotechnical, magnetic susceptibility, and specific gravity measurements taken at regular intervals. Lithology was logged and the core was prepared for systematic sampling at regular 1 m intervals. Magnetic susceptibility and conductivity data were measured at 10 cm increments along the core with a hand-held conductivity and magnetic susceptibility meter. Recovery and rock quality designation (RQD) data were also measured and recorded.

Core recovery for the 2009, 2010, 2011, and 2012 drilling programs averaged 92%, and the median core recovery was 96%.

Planned drill hole collar locations were measured in the field using hand-held global positioning system (GPS) instruments. Locations were subsequently confirmed by Trimble differential GPS. Of the 1,053 drill holes, 1,037 were then professionally surveyed by All North Consulting using a real time kinematic (RTK) technique to enhance the precision of the location data. Elevations for the drill collars were determined by draping collar coordinates over the topography measured by an aerial light detection and ranging (LiDAR) survey.

Down-hole surveys were performed using Reflex survey equipment, and dip angle and azimuth were recorded. A +18.8° magnetic declination correction factor was applied to the magnetic azimuth record.

Thirteen specific geotechnical HQ holes were drilled; in addition, 10 hydrological pilot holes (also at HQ size) were drilled to serve as monitoring stations, where a piezometer was installed to measure the level of the aquifer in the deposit area. Twenty-seven specific metallurgical holes were drilled, four of which were HQ in size; the remaining 23 holes were drilled at PQ. Fourteen waste rock characterization holes (HQ size) were drilled, and 91 RC holes and 18 core holes comprised the condemnation drill program.

Artemis Gold drilled 561 reverse circulation (RC) holes for a total of 33,216 m during a pre-production grade control program during the winter of 2020–2021. Its purpose was to de-risk the mill start up and establish more detailed continuities of the mineralization.

1.11 Sampling and Analysis

Previous owners Richfield and New Gold personnel conducted the drill core handling and sampling.

Certified reference standards (CRMs), blanks, and duplicates were inserted into the sample stream. The drill hole database is supported by over 43,000 quality assurance and quality control (QA/QC) check assays.

Eco Tech Stewart Group Laboratories (Eco Tech) in Kamloops, British Columbia (BC) and ALS Mineral Laboratories (ALS) in Vancouver, Vanderhoof, Terrace, Reno, and Elko were used for sample preparation. Eco Tech was used as the primary assayer beginning with Richfield exploration in 2009. Assays continued in Kamloops through October 2011, then moved to the ALS laboratory in North Vancouver. External duplicate analysis was performed at the SGS Canada Inc. laboratory, located in Burnaby, British Columbia (SGS). All laboratories were accredited and are independent of New Gold and Artemis Gold. Pre-production grade control sample preparation and analysis were performed by SGS. SGS holds ISO/IEC accreditation for selected sample preparation and analytical techniques and is independent of Artemis Gold.

Drill core samples were prepared using standard crush, split, and pulverise sample preparation procedures. Pulverized samples were analysed for gold by fire assay (FA) atomic absorption spectrometry (AAS). Preparation and FA AAS procedures varied between laboratories but were generally similar. The Eco Tech samples were initially assayed for silver by aqua-regia digestion (AR) and AAS finish, and later by AR and induction-coupled plasma spectrometry atomic emission spectrometry (ICP AES) finish. Eco Tech overlimit results (>30 g/t Ag) were re-assayed by an AR/AAS method. The ALS samples were analyzed for silver by four acid digestion ICP AES finish until July 2012, after which time silver was analyzed by a four-acid digestion AAS. ALS overlimit results (>100 g/t) were re-assayed by a four-acid digestion with AAS finish with a higher detection limit. Assay procedures also include a multi-element package (28 elements at Eco Tech, 33 elements at ALS) by AR digestion and ICP AES finish. Overlimit analysis was completed on samples returning >1% Cu, Pb, or Zn.

QA/QC protocols included “blind” insertion of certified reference materials (CRMs), blanks, field duplicates, and pulp duplicates. Quality control procedures implemented from 2009–2013 were reviewed and assay and drill data from that period is of sufficient quality to support Mineral Resource estimation.

The current drill hole and assay database for the Project is stored in an Access database administered from Artemis Gold’s Vancouver office.

Chain-of-custody procedures consisted of completion of sample submittal forms that were sent to the laboratory with sample shipments to ensure that all samples shipped were received by the laboratory.

Remaining half cores were archived in core racks in the vicinity of the existing camp infrastructure with metal roofing protecting the core boxes from the elements.

1.12 Data Verification

The QPs individually reviewed the information in their areas of expertise, and concluded that the information supported Mineral Resource and Mineral Reserve estimation, and could be used in mine planning and in the economic analysis that supports the Mineral Reserve estimates.

1.13 Metallurgical Testwork

Testwork was completed in support of a number of mining studies from 2008 to 2019, by Inspectorate, G and T Laboratories, SGS, Dawson Metallurgical Laboratories, McClellan Laboratories, and Pocock and MetSolve. Testwork included sample characterization, comminution, gravity concentration, leaching, flotation, oxygen uptake, cyanide destruction, carbon loading, and settling/viscosity tests. Testwork results led to the elimination of heap leaching and flotation methods, and the conclusion that whole ore leaching was the most appropriate method for recovering gold and silver. All recent work has focused on whole ore leaching and incorporated gravity separation as an integral part of the recovery process for gold and silver.

Results of work index comminution testwork, indicating variable results for work index measurements, led to the adoption of three-stage crushing and a single ball mill for plant design purposes. Gravity concentration was effective on the tested ore and was incorporated in the process flow sheet. Leach tests indicated that gravity concentration increased overall recovery. A total of 48 samples were selected for variability testing, distributed throughout the deposit area. Six variability samples gave overall extractions (gravity +leach) of <90%; the remaining other samples gave results significantly >90%.

The average extractions for the three composites representing the first five years of mining were 94.4% for gold and 61.5% for silver. The average gold extraction for the 48 variability composites was 93.1% for gold and 69.8% for silver. Considering these results, the use of a recovery of 93% for gold and 65% for silver is recommended, which would include solution losses assuming a dissolved gold concentration of 0.008 mg/L in the final solution for gold, and 0.1 mg/L for silver.

Variability testwork focused on the material to be treated in the earlier phases of the LOM plan. Samples selected for metallurgical testing were representative of the various types and styles of mineralization within the earlier phases of the mine plan. Samples were selected from a range of locations within the deposit zones. Enough samples were taken so that tests were performed on sufficient sample mass. Additional variability testwork is recommended for mineralization in the later periods of the mine plan.

No deleterious elements are known from the processing perspective.

1.14 Mineral Resource Estimation

The Mineral Resource estimate is based upon a block model that incorporates 288,738 individual assays from 309,293 m of core from 1,002 drill holes. The drill hole database is supported by analysis of over 43,000 QA/QC samples.

The block model is created using block dimensions of 10 x 10 x 10 m.

Gold interpolation has been done using multiple indicator kriging (MIK) with silver grades interpolated by ordinary kriging (OK). MIK has been used for Au estimation due to the significant value and non-linear distribution of the Au mineralization at higher grades. This is evident by the cumulative probability plots (CPPs) and coefficients of variation (C.V.s) of the Au grades by domain, as discussed in Section 14. Ordinary kriging has been used for Ag because the C.V.s are generally lower, and the Ag is generally lognormally distributed at higher grades. The interpolated grades were validated through comparison of the de-clustered composite data by global bias checks, grade- tonnage curves for smoothing checks, and visual validation in section and plan.

The interpolations were limited by the domain boundaries and were clipped to the overburden surface. Blocks were assigned a preliminary classification based on the variography and drill hole spacing by domain, with Measured and Indicated classifications then adjusted for continuity of blocks.

To assess reasonable prospects for eventual economic extraction, a Lerchs–Grossmann (LG) pit was used to constrain the Mineral Resource. The economic assumptions used in the LG shell are almost identical to the economic assumptions used for the Mineral Reserve pit optimization with the notable exception of metal prices, which are higher for the Mineral Resource estimate, and pit slopes which are constant at 40º.

1.15 Mineral Resource Statement

The Qualified Person for the resource estimate is Sue Bird, P. Eng. of Moose Mountain. The Mineral Resource is classified in accordance with the 2014 CIM Definition Standards and was estimated using the 2019 CIM Best Practice Guidelines. Mineral Resources in Table 1-1 are reported inclusive of Mineral Reserves. Mineral Resources that are not Mineral Reserves do not have demonstrated economic viability.

Table 1-1 includes a range of gold equivalent (AuEq) cut-off grades to show the sensitivity of the resource estimate to variations in cut-off grade. The base case cut-off grade within the reasonable prospects of eventual economic extraction conceptual pit is 0.20 g/t AuEq, as highlighted in Table 1-1.

Table 1-1: Blackwater Mineral Resource Estimate, Effective Date 5 May, 2020 (base case is highlighted)

Classification	Cut-off (g/t AuEq)	Tonnage (kt)	In-situ Grades			In-situ Contained Metal		
			AuEq	Au	Ag	AuEq	Au	Ag
			(g/t)	(g/t)	(g/t)	(koz)	(koz)	(koz)
Measured	***0.20***	***427,123***	***0.68***	***0.65***	***5.5***	***9,360***	***8,905***	***75,802***
	0.30	313,739	0.84	0.80	5.9	8,463	8,109	59,009
	0.40	238,649	0.99	0.96	6.1	7,627	7,347	46,727
	0.50	186,687	1.15	1.11	6.2	6,881	6,656	37,333
	0.60	149,261	1.30	1.26	6.4	6,223	6,039	30,521
	0.70	120,916	1.45	1.41	6.6	5,633	5,479	25,619
Indicated	***0.20***	***169,642***	***0.56***	***0.51***	***8.5***	***3,046***	***2,766***	***46,578***
	0.30	123,309	0.68	0.61	10.4	2,677	2,431	41,112
	0.40	86,473	0.81	0.74	12.4	2,264	2,057	34,419
	0.50	64,305	0.94	0.85	14.8	1,947	1,763	30,681
	0.60	50,527	1.05	0.95	17.2	1,705	1,537	27,957
	0.70	40,317	1.15	1.03	19.6	1,493	1,340	25,458
Measured + Indicated	***0.20***	***596,765***	***0.65***	***0.61***	***6.4***	***12,406***	***11,672***	***122,381***
	0.30	437,048	0.79	0.75	7.1	11,140	10,540	100,120
	0.40	325,122	0.95	0.90	7.8	9,890	9,404	81,146
	0.50	250,992	1.09	1.04	8.4	8,828	8,419	68,014
	0.60	199,788	1.23	1.18	9.1	7,928	7,577	58,478
	0.70	161,233	1.37	1.32	9.9	7,125	6,819	51,077
Inferred	***0.20***	***16,935***	***0.53***	***0.45***	***12.8***	***288***	***246***	***6,953***
	0.30	11,485	0.66	0.57	16.2	245	210	5,971
	0.40	8,690	0.77	0.65	19.2	214	182	5,373
	0.50	5,552	0.95	0.79	26.0	169	142	4,648
	0.60	4,065	1.10	0.90	32.7	143	118	4,279

Notes:

1. The Mineral Resource estimate was prepared by Sue Bird, P.Eng., the Qualified Person for the estimate and employee of Moose Mountain Technical Services. The estimate has an effective date of May 5, 2020. There have been material changes since this data.
2. Mineral Resources are reported using the 2014 CIM Definition Standards and are estimated in accordance with the 2019 CIM Best Practices Guidelines.
3. Mineral Resources are reported inclusive of Mineral Reserves.
4. Mineral Resources that are not Mineral Reserves do not have demonstrated economic viability.
5. The Mineral Resource has been confined by a conceptual pit shell to meet “reasonable prospects of eventual economic extraction” using the following assumptions: the 143% price case with a Base Case of US$1,400/oz Au and US$15/oz Ag

at a currency exchange rate of 0.75 US$ per C$; 99.9% payable Au; 95.0% payable Ag; US$8.50/oz Au and US$0.25/oz Ag offsite costs (refining, transport, and insurance); a 1.5% NSR royalty; and uses a 93% metallurgical recovery for gold and 55% recovery for silver.

6. The AuEq values were calculated using US$1,400/oz Au, US$15/oz Ag, a gold metallurgical recovery of 93%, silver metallurgical recovery of 55%, and mining smelter terms for the following equation: AuEq = Au g/t + (Ag g/t x 0.006).
7. The specific gravity of the deposit has been determined by lithology as being between 2.6 and 2.74.
8. Numbers may not add due to rounding.

As part of the model validation process, a comparison of the gold content in the 2020 model (which used MIK for the gold estimate) to that in the 2014 resource model (which used OK) was completed. The comparison used the 2014 resource pit, the AuEq calculation from 2014 and a cut-off of 0.3 g/t AuEq (as used for the 2014 resource statement) in order to compare a similar volume and grade distribution. The comparison shows that the respective resource tonnage and Au grade are within 5%, and the total contained gold content is within 2% for the Measured and Indicated categories.

The following factors, among others, could affect the Mineral Resource estimate: commodity price and exchange rate assumptions; pit slope angles and other geotechnical factors; assumptions used in generating the LG pit shell, including metal recoveries, and mining and process cost assumptions.

Mineral Resources are reported inclusive of those Mineral Resources converted to Mineral Reserves. Mineral Resources that are not Mineral Reserves do not have demonstrated economic viability. Mineral Resources were estimated using the 2019 CIM Best Practice Guidelines and are reported in situ, using the 2014 CIM Definition Standards.

1.16 Mineral Reserves Estimation

Proven and Probable Mineral Reserves are modified from Measured and Indicated Mineral Resources. Inferred Mineral Resources are set to waste.

The open pit is based on the results of Pseudoflow sensitivity analysis, and then designed into detailed pit phases for production scheduling purposes.

1.17 Mineral Reserves Statement

Mineral Reserves are reported at the point of delivery to the primary crusher using the 2014 CIM Definition Standards, and have an effective date of 10 September, 2021.

The Qualified Person for the estimate is Mr. Marc Schulte, P.Eng., a member of Moose Mountain.

Mineral Reserves are summarized in Table 1-2.

Table 1-2: Mineral Reserves Statement

Confidence Category	Tonnage (Mt)	Gold Grade (g/t Au)	Contained Gold Metal (Moz Au)	Silver Grade (g/t Ag)	Contained Silver Metal (Moz Ag)	AuEq Grade (g/t)
Proven	325.1	0.74	7.8	5.8	60.4	0.78
Probable	9.2	0.80	0.2	5.8	1.7	0.83
Total Proven and Probable	***334.3***	***0.75***	***8.0***	***5.8***	***62.2***	***0.78***

Notes to accompany Mineral Reserves table:

1. Mineral Reserves are reported at the point of delivery to the primary crusher, inclusive of mining loss and dilution, using the 2014 CIM Definition Standards, and have an effective date of 10 September, 2021.
2. Mineral Reserves are supported by the 2024 Expansion Study life of mine plan.
3. The Qualified Person for the estimate is Mr. Marc Schulte, P.Eng., a member of Moose Mountain Technical Services.
4. Mineral Reserves are reported at a net smelter return (NSR) cut-off of C$13.00/t. The NSR cut-off covers processing costs of C$9.00/t, administrative (G&A) costs of C$2.50/t and stockpile rehandle costs of C$1.50/t. The NSR cut-off assumes US$1,400/oz Au and US$15/oz Ag at a currency exchange rate of 0.75 US$ per C$; 99.9% payable gold; 95.0% payable silver; US$8.50/oz Au and US$0.25/oz Ag offsite costs (refining, transport, and insurance); a 1.5% NSR royalty; and uses a 93% metallurgical recovery for gold and 55% recovery for silver.
5. Gold equivalent (AuEq) values are calculated using the same parameters as NSR listed above, resulting in the following equation: AuEq = Au g/t + (Ag g/t x 0.006).
6. Numbers have been rounded as required by reporting guidelines

Changes in the following factors and assumptions may affect the Mineral Reserve estimate: metal prices and foreign exchange rates; interpretations of mineralization geometry and continuity of mineralization zones; geotechnical and hydrogeological assumptions; changes to pit designs from those currently envisaged; ability of the mining operation to meet the annual production rate; changes to operating and capital cost assumptions; mining and process plant recoveries; and the ability to meet and maintain permitting and environmental license conditions and the ability to maintain the social license to operate.

1.18 Mining Methods

Mining is based on conventional open pit methods suited for the deposit location and local site requirements. Open pit operations will commence 3–6 months prior to mill start-up and are anticipated to run for 15 years. Following pit mining operations, stockpiled low-grade material will be processed for an additional two years, resulting in a total LOM of 17 years.

Ultimate pit limits are split into phases or pushbacks to target higher economic margin material earlier in the mine life. The pit is split into nine phases, with initial phases containing higher gold grade and lower strip ratio. The first phase targets higher-grade, lower-strip-ratio ore, providing mill feed over the initial years of the operation. The remaining phases will expand the pit to the east, west, north, and south targeting progressively deeper, lower economic margin ore.

The production is planned on 10 m bench heights in both ore and waste.

Mill feed targets are 6.0 Mt/a in the first year of operation, increasing to 15 Mt/a for the next five years of operation, and finally to 25 Mt/a until the end of the planned mine life.

Before the final mill expansion, all ore with NSRs between C$13/t and C$27/t will be stockpiled. Cut-off grade optimization on the mine production schedule will also send ore above an NSR of C$27/t to an ore stockpile in the initial year of operations. The stockpiled ore is planned to be re-handled back to the crusher during the mine life. The higher-grade material will be re-handled in advance of the lower-grade material.

Owner-managed mining and fleet maintenance operations are planned for 365 days/year, with two 12-hour shifts planned per day. An allowance of 10 days of no mine production has been built into the mine schedule to allow for adverse weather conditions. Contractor drill and blast services are planned for the first three years of operations, with drill operations converting to an owner operated function in Year 4, and contractor blasting services continuing throughout the remaining life of operations.

Mining will be undertaken using 600 t class hydraulic shovels, 400 t class hydraulic excavators, and 240 t payload class haul trucks. The initial drill and loading fleets are planned to be diesel-drive, with expansion fleet requirements being electric-drive. The initial mine equipment fleet is paid back through a lease arrangement with the supplier.

In-pit and perimeter pumping dewatering systems will be established. All surface water and precipitation in the pits will be handled by skid-mounted mobile diesel pumps.

Ore will be hauled to a crusher that will be located 1 km northeast of the open pit limit, which will feed the process plant. Waste rock will generally be used as fill for construction of the tailings storage facility (TSF) that will be located 2.5–5 km north of the open pit limits, or in the case of potentially acid generating (PAG) waste rock, placed within the TSF itself for subaqueous storage. Additional storage facilities, to be constructed within 1.5 km northwest of the pit, will be used to store excess overburden and non-acid generating (NAG) waste rock. Ore stockpiles, to be located within 1 km to the west of the open pit, will be used as temporary storage for re-handle back to the crusher over the planned mine life.

Maintenance on mine equipment will be performed in the field with major repairs to mobile equipment conducted in the workshops that will be located within 1 km north of the open pit.

Annual mine operating costs per tonne mined will range from C$2.37–C$3.66/t with a LOM average of C$2.97/t mined. Mine operations will include ore control, production drilling, blasting, loading, hauling, and pit, haul road and stockpile maintenance functions. The largest component of the estimated mine operating costs is for the hauling function, and a significant portion of the planned hauls for the Blackwater mine are downhill loaded hauls. Mobile equipment maintenance operations will also be managed by the Owner and are included in the mine planning and costs.

1.19 Recovery Methods

The process plant design is based on a metallurgical flowsheet designed for recovery with minimum operating costs. The flowsheet is based upon unit operations that are well proven in industry.

Plant design comprises:

- Phase 1 (existing): 6 Mt/a;
- Phase 2: expansion by 9 Mt/a to 15 Mt/a;
- Phase 3: expansion by 9.5 Mt/a to 24.5 Mt/a.

The optimization of Phase 1 will add additional 0.5 Mt/a, bringing the overall plant capacity to 25 Mt/a (Phase1 + Phase 2 + Phase 3).

The process plant consists of a crushing circuit, milling, gravity, pre-oxidation, leach, and carbon-in-leach (CIL) circuit, cyanide detox, and an elution plant. A second plant gold room will be included to house additional electrowinning cells as well as new gold pouring facilities. The existing plant reagents, oxygen, air, and water services will be re-used and or expanded as required. Un-thickened tailings flow by gravity to the TSF. A dedicated decant return line is used to make-up process water as required. The process flow is summarized in Table 1-3.

Table 1-3: Process Flow

<table>
<tr><th>Item</th><th>Phase 1</th><th>Phase 2</th><th>Phase 3</th></tr>
<tr><td>Primary crushing</td><td>Primary gyratory crushing of ROM material</td><td>Primary gyratory crushing of ROM material, shared with the Phase 1 circuit, to a distribution bin splitting material back to the Phase 1 circuit with the remainder to the Phase 2 circuit.</td><td>Primary gyratory crushing of ROM material with a portion of the material transferred to the Phase 1 circuit for increased throughput via a distribution bin.</td></tr>
<tr><td>Secondary crushing</td><td>Secondary and tertiary cone crushing circuit with classification screens, to produce a fine product for storage on a fine ore stockpile ahead of the milling plant.</td><td colspan="2">Secondary cone crushing circuit with classification screen, to produce a crushed product for storage on a stockpile ahead of the milling plant.</td></tr>
<tr><td>Grinding</td><td>Single-stage ball mill operating in closed circuit with classification cyclones.</td><td colspan="2">SABC circuit with the SAG mill operating in closed circuit with a pebble crusher and the ball mill operating in closed circuit with classification cyclones.</td></tr>
<tr><td>Gravity recovery</td><td colspan="3">Gravity recovery of mill discharge material by semi-batch centrifugal gravity concentrators, followed by intensive cyanidation of the gravity concentrate and electrowinning of the pregnant leach solution in a dedicated electrowinning cell located in the goldroom.</td></tr>
</table>

Item	Phase 1	Phase 2	Phase 3
Trash screening	Trash screening of cyclone overflow before leaching.		
Oxygen	Pre-aeration with pure oxygen ahead of the leach and CIL circuit with pure oxygen addition throughout.		
Washing, elution, regeneration	Acid washing of loaded carbon and AARL elution circuit followed by electrowinning and smelting to produce doré. Carbon regeneration by rotary kiln.		
Cyanide destruction	SO_2/oxygen process.		
Tailings	Tailings slurry transfer to the TSF via gravity.	Tailings slurry transfer to the tailings disposal facility via pumping.	

Note: ROM = run-of-mine; SAG = semi-autogenous grind, SABC = SAG, ball mill, and crusher circuit; CIL = carbon in leach; AARL = Anglo American Research Laboratory; TSF = tailings storage facility.

The following reagent systems are required for the process: quick lime, sodium cyanide, sodium hydroxide, hydrochloric acid, copper sulphate, sulphur dioxide, flocculant, activated carbon, anti-scalant, and smelting fluxes. High pressure air and oxygen supplies will be required. Grinding media are needed for the mills.

Raw water will be supplied from the water management pond and depressurisation wells into a raw water storage tank. Tailings return water and mill cooling water return will meet most of the process water requirements. Potable water for plant use will be supplied from a potable water treatment plant.

1.20 Project Infrastructure

1.20.1 Major Infrastructure

The overall Blackwater Gold Mine facilities and major infrastructure cover the mine site area, TSF, WRSF, camp site, main access road, and site wide water management systems.

The following Phase 1 infrastructure is assumed to be in place: power supply; roads (access and haul roads); process plant (Phase 1); tailings storage facility (stage 1) and associated water management structures; mine services (heavy mine equipment (HME) workshop, wash bay, tire service area, explosive storage, bulk fuel and lube facilities); pit development (pit clearing, bench access, low-grade stockpiles, WRSFs); accommodations (operations and construction, 537 beds); ancillary building (mining, process, laboratory, warehousing and security); water treatment (metals and lime neutralization); and communications (fibre optic cable). The 2024 Expansion Study assumes that the existing facilities will be built out as necessary to meet the Phase 2 and Phase 3 expansion requirements.

On site roads will provide access to the plant site, mine services area, accommodation, and explosives storage facility. These roads will support two-way, light vehicle traffic and will be wider as required for the passage of mine trucks.

1.20.2 Tailings Storage Facility

The tailings storage facility (TSF) will comprise two adjacent valley-fill style impoundments, TSF C and TSF D, and was designed to permanently store tailings, PAG waste rock, and potentially metals leaching (ML) NAG waste rock generated during mine operation. The facility was designed to hold 470 Mm^3 of tailings and waste rock material, and up to 12 Mm^3 of pond storage under normal operating conditions. Additional freeboard allowances are included in the design to manage seasonal inflows and provide protection from severe natural flooding.

TSF C will be formed by the construction of three embankments: Main Dam C, Saddle Dam, and West Dam, in the upper reaches of the Davidson Creek drainage area. TSF D will be formed by construction of one embankment, the Main Dam D, which will be adjacent to and downstream of TSF C within the Davidson Creek drainage area. The TSF embankments will be engineered, water retaining, zoned earth–rockfill structures.

The construction of TSF C is in progress with the ongoing construction of Stage 1 of Main Dam C underway. TSF C was designed to receive PAG/ML NAG waste rock through Year 5 and tailings for 15 years from Year 1 to the end of Year 15.

TSF D construction will start in approximately Year 3 during the proposed Phase 2 expansion to provide additional storage capacity for PAG/ML NAG waste rock and tailings. TSF D is designed to receive waste rock for 12 years starting in Year 4 to the end of Year 15, then tailings for up to two years starting in Year 16 when TSF C reaches design capacity, the timing of which will depend on mining rates and tailings consolidation processes.

A tailings distribution system will transport tailings slurry to TSF C, and later to TSF D. There will be two expansion phases of tailings distribution infrastructure, one each for Phase 2 and Phase 3.

1.20.3 Water Management

The water management structures are designed to collect and divert non-contact surface water not needed for mine operations and to collect and control mine affected contact surface water. Collected contact runoff will be recycled for use as process water and/or treated using a metals removal water treatment plant or lime neutralization circuit, as required depending on where the water will be directed. Surplus water not required to support mine operations will be sampled and analyzed, compared to applicable water quality criteria, and if compliant, will be used to augment flow in lower Davidson Creek.

Key water management facilities include: the central diversion system, northern diversion system, freshwater reservoir, water management ponds, reclaim water system, stockpile water management structures, Lake 16 diversion berm and Lake 15–16 connector channel, and water treatment plants.

1.21 Environmental, Permitting and Social Considerations

1.21.1 Environmental Considerations

The Blackwater Gold Mine is supported by a suite of environmental, social, economic, and cultural heritage baseline studies and potential effects of the operations were fully assessed. Comprehensive biophysical studies were completed to support the Environmental Impact Statement and permit applications, and Artemis Gold continues to collect data in conformance with conditions in provincial and federal authorizations.

The Blackwater Gold Mine was granted an Environmental Assessment Certificate #M19-01 on 21 June, 2019 under the 2002 *Environmental Assessment Act* and an Environmental Assessment Decision Statement on 15 April, 2019 under the *Canadian Environmental Assessment Act, 2012*.

Assessment of components to address updates in the Blackwater Gold Mine design were considered in recent permits.

To manage potential effects of the Blackwater Gold Mine, an Environmental Management System is supported by a comprehensive set of management plans. The individual management plans and supporting documentation that form the basis of the Environmental Management System are required by the *Mines Act*, *Environmental Management Act*, or the Environmental Assessment Certificate Conditions and were developed by qualified professionals and subject matter experts. The management plans were developed and revised to incorporate comments/reviews from Indigenous groups provided through review of monitoring results and/or previous iterations of the plan.

Offsetting plans were prepared to mitigate potential impacts to fish and fish habitat, wetlands, and southern mountain caribou.

1.21.2 Closure and Reclamation Planning

Reclamation of the Blackwater Gold Mine area will conform to the requirements of the Health, Safety, and Reclamation Code for Mines in BC. The Reclamation and Closure Plan is approved through *Mines Act* Permit M-246 and describes how end land use and land capability objectives will be achieved.

1.21.3 Permitting Considerations

All major provincial and federal permits, licenses, and authorizations for construction and operation of the Blackwater Gold Mine were issued. The Blackwater Gold Mine is currently authorized for a milling capacity of 55,000 t/day (or 20 Mt/a). Prior to implementing the Phase 3 mill capacity increase, amendments to several permits may be required including *Mines Act* Permit M-246 and *Environmental Management Act* Permits #110650 and #110652.

Key federal approvals include those associated with impacts to fish habitat (*Fisheries Act*) and deposition of mine waste in waters frequented by fish (*Metal and Diamond Mining Effluent Regulations*, SOR/2002-222).

Key provincial approvals include the *Mines Act* permit approving the mine plan and reclamation program and various discharge permits (*Environmental Management Act*), water use license (*Water Sustainability Act*), and occupancy of Crown land for the mine access road and transmission line (*Lands Act*).

The Environmental Assessment Certificate #M19-01 contains 43 binding conditions, which identify requirements for environmental and social management plans, consultation requirements related to management plans, and requirements for an Environmental Monitoring Committee, Community Liaison Committee, Independent Environmental and Aboriginal Monitor(s). The federal Decision Statement includes 102 binding conditions, which identify requirements for plans to offset impacts, consultation requirements for offset plans and follow-up programs, and specific mitigation measures. Artemis Gold is addressing these conditions in accordance with the timelines specified by the Environmental Assessment Certificate and Decision Statement.

1.21.4 First Nations

The Blackwater Gold Mine is located within the traditional territories of Ulkatcho First Nation, Lhoosk'uz Dené Nation. The power transmission line and other infrastructure pass through the traditional territories of Nadleh Whut'en First Nation, Saik'uz First Nation, and Stellat'en First Nation (collectively, the Nechako First Nations).

1.22 Markets and Contracts

No formal marketing studies were completed. There are many markets in the world where gold and silver are bought and sold, and it is not difficult to obtain a market price at any time. The gold and silver market is highly liquid with many well-informed potential buyers and sellers active at any given time.

Mineral Resources use commodity pricing of US$2,000/oz Au and $16/oz Ag. Mineral Reserves use commodity prices of US$1,400/oz Au and $15/oz Ag.

The cashflow analysis uses a reverting price curve for gold, based on consensus estimates from a consortium of banks. The average January 2024 spot price is approximately US$2,000/oz Au and this reverts to long-term pricing of US$1,800/oz Au from Year 5 onwards. The long-term guidance silver price used in the economic analysis is US$23/oz.

Artemis Gold has a modest hedging program in place to secure the returns on capital invested in the early years of operations and de-risk the servicing loan facility during the pay-back period.

Artemis Golds 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 doré elsewhere in the world. Other than for product sales, the largest contracts will cover items such as mine equipment, drill-and-blast operations, bulk commodities, and technical services. Artemis Gold is currently negotiating contract terms. Contracts will be re-negotiated and renewed as needed during the LOM.

1.23 Capital Cost Estimates

Capital cost estimates are at a minimum pre-feasibility study level. Phase 1 costs are assumed to be sunk costs.

Data supporting the estimate were obtained during Q4 2023 and Q1 2024.

The estimate was based on inputs developed by Artemis Gold, Knight Piésold, Moose Mountain, Lycopodium, JAT MetConsult, and other third parties, using budget quotes and historical data, and is an amalgamation of engineering, material take offs and in-house benchmarks.

The estimate assumes that the process plant will be executed using an engineering and design, procurement, and construction management (EPCM) approach.

Capital costs include mining, infrastructure (TSF, water management, mining infrastructure), process, and Owner costs.

Capital costs over the LOM total C$2,619 M, excluding closure costs, and are summarized in Table 1-4. Closure costs (net of salvage value) are estimated at an additional C$250 M.

Table 1-4: Capital Cost Estimate

Major Area	Discipline Area	C$ M
Growth capital cost estimate	Mining	285
	Process	981
	TSF and water management	97
	Infrastructure	17
	Owners	118
	Subtotal	*1,497*
Sustaining capital cost estimate	Mining	724
	Process	—
	TSF and water management	281
	Infrastructure	—
	Owners	117
	Subtotal	*1,122*

Major Area	Discipline Area	C$ M
Growth + sustaining capital cost estimates	***Total***	***2,619***
Closure costs (net of salvage value)		250

Notes: The growth capital cost estimate includes deferred initial capital costs. Numbers have been rounded.

1.24 Operating Cost Estimates

Operating cost estimates are at a minimum pre-feasibility study level.

Operating cost estimates were derived using first principal estimates based on typical operating data/standard industry practices, and inputs compiled from a variety of sources. Data supporting the estimate were obtained during Q4 2023 and Q1 2024.

Operating costs include mining, process, and general and administrative costs.

Operating costs over the LOM total $20.03/t milled, and are summarized in Table 1-5.

Table 1-5: Operating Cost Estimate

Area	Units	Phase 1 (Years 1–2)	Phase 2 (Years 3–6)	Phase 3 (Years 7–15)	Stockpile Phase (Years 16–17)	LOM (17 years)
Mining	$/t mined	2.46	2.15	2.78	n/a	2.57
Process	$/t milled	10.51	10.06	9.80	9.83	9.88
G&A	$/t milled	5.30	3.43	2.41	1.90	2.67

Note: Mining costs exclude the cost of major component replacements, which are reported as sustaining capital, and include low-grade ore stockpile rehandle. Numbers have been rounded.

1.25 Economic Analysis

1.25.1 Forward-Looking Information Note

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

Information that is forward-looking includes:

- Mineral Resource and Mineral Reserve estimates;
- Assumed commodity prices and exchange rates;
- Mine production plans;

- Projected recovery rates;
- Sustaining and operating cost estimates;
- Assumptions as to closure costs and closure requirements;
- Assumptions as to environmental, permitting, and social risks.

Additional risks to the forward-looking information include:

- Changes to costs of production from what is assumed;
- Changes to construction execution strategy from what is assumed;
- Unrecognized environmental risks;
- Unanticipated reclamation expenses;
- Unexpected variations in quantity of mineralized material, grade, or recovery rates;
- Geotechnical and hydrogeological considerations during mining being different from what was assumed;
- Failure of plant, equipment, or processes to operate as anticipated;
- Accidents, labour disputes and other risks of the mining industry.

1.25.2 Economic Analysis

All dollar amounts are expressed in Q1 2024 Canadian dollars, unless otherwise noted.

The expansion study assumes that construction of the 6 Mt/a Phase 1 process plant is completed. The purpose of the expansion study is to optimize the timing of mine expansion through the advancement of Phase 2 to Q4 of Year 2 of operations at an increased production capacity of 15 Mt/a, and Phase 3 to Year 7 of operations at an increased production capacity of 25 Mt/a. Phase 1 capital costs of C$730–C$750 M are considered sunk costs for the purposes of the economic analysis in this Report. The net present value (NPV) is reported net of the scheduled repayment of the C$385 million loan facility associated with Phase 1, with all gold and silver stream participations included. The NPV at a 5% discount rate is discounted to the commencement of Phase 2 construction.

The economic analysis is based on 100% equity financing and is reported on a 100% ownership basis. The economic analysis assumes constant prices with no inflationary adjustments, and uses a reverting gold price curve. The cashflow includes provision for two private NSR royalties at 1.0% and 1.5% over portions of the Mineral Reserve, which were applied to the economic cash flow model based upon the mine plan. Estimated payments to Indigenous groups are included in the economic cash flow model for the Blackwater Gold Mine. Closure costs are estimated at approximately C$250 M, which includes a salvage value of C$37 M. Closure costs are assumed to be applied in Year 18. Bonding of the reclamation and closure costs has been applied throughout the model, and are based on progressive disturbance.

The cashflow assumes the repayment of the loan facility associated with Phase 1 as follows:

- The loan facility of C$385 million will be repaid in quarterly instalments over a six-year term commencing 31 May, 2025, with a repayment holiday during Years 4 and 5;
- An annual interest rate of Canadian Dollar Offered Rate (assumed at 4.0%) plus a margin of 4.75% up to 31 May, 2025 with the margin reducing to 4.25% thereafter;
- Commitment fees of 1.75% associated with the unused C$40 M cost overrun facility;
- Phase 2 and Phase 3 growth capital is assumed to be funded through operating cashflow.

Key provincial and federal tax considerations in the economic analysis include:

- BC mining tax: 2% provincial minimum tax payable on net current proceeds which is creditable against the 13% effective mining tax rate which is calculated based on operating profit less applicable capital cost deductions. The mining tax is deductible in computing provincial and federal income tax;
- BC provincial income tax: 12.0%, payable after applicable deductions are used;
- Canadian federal income tax: 15.0%, payable after applicable deductions are used.

The after-tax NPV is C$3.25 B, using a 5% discount rate. The Blackwater Gold Mine is cashflow positive in Phase 1, and so there is no internal rate of return or payback period that is relevant to this Report. A cashflow summary table is provided in Table 1-6.

Table 1-6: Summary Cashflow Analysis

Item	Unit	First 5 Years	First 10 Years	LOM (17 years)
Average throughput capacity	Mtpa	12	18	20
Gold grade	g/t	1.29	0.91	0.75
Silver grade	g/t	7.75	5.92	5.78
Gold equivalent grade	AuEq g/t [1]	1.36	0.96	0.79
Gold recoveries	%	93	93	93
Average annual gold production	Au oz	463,000	478,000	438,000
Average annual silver production	Ag oz	1,944,000	2,165,000	2,376,000
Average annual gold equivalent production	AuEq oz [2]	488,000	506,000	469,000
Strip ratio	Waste:ore	1.99	2.13	2.01
Growth capital [3,4]	C$ M	1,174	1,497	1,497
Sustaining capital[4]	C$ M	499	874	1,122
Operating costs	C$/tonne milled	26.86	23.00	20.03
Cash costs [5]	US$/oz	456	577	645
All-in sustaining costs [6]	US$/oz	615	712	781
Average annual free cash flow [7]	C$ M	552	489	413
After-tax NPV5% [8]	C$ B	3.25		

Notes: Numbers have been rounded.

1. Gold equivalent grades have been determined using a gold price of US$1,800/oz, a silver price of US$23/oz, a gold metallurgical recovery of 93%, a silver metallurgical recovery of 65%, and mining smelter terms for the following equation: AuEq = Au g/t + (Ag g/t x 0.0085).
2. Gold equivalent ounces have been determined using a gold-to-silver ratio of 78:1 (US$1,800:US$23).
3. Includes deferred initial capital costs.
4. Excludes closure costs and salvage value.
5. Cash costs include selling costs, royalty payments, operating costs, less silver by-product credits and adjustments to stockpile inventory, divided by payable gold ounces.
6. All-in sustaining costs include cash costs as defined above, sustaining capital and closure costs, divided by payable gold ounces.
7. Free cash flow = operating cash flow less sustaining capex, closure costs and taxes.
8. After-tax NPV represents the net present value of after-tax cash flows, discounted at a rate of 5%. The after-tax cash flows take into account the repayment of the Project loan facility of $385 million, as well as the effect of the gold stream and silver stream arrangements.

1.26 Sensitivity Analysis

A sensitivity analysis was performed examining capital costs, operating costs, foreign exchange rate, gold grade and gold price. The NPV is most sensitive to fluctuations in gold price (gold grade) and foreign exchange rate assumptions, and less sensitive to variations in capital and operating costs. The impacts of changes in the gold grade mirror the impact of changes in the gold price.

1.27 Risks and Opportunities

1.27.1 Risks

The major risks include:

- Changes to metal prices and exchange rate assumptions;
- Capital cost growth;
- Increases in operating costs;
- Changes to productivity assumptions;
- Changes to mining grade and dilution control assumptions;
- Presence of high-grade silver in the mill feed;
- Geotechnical and hydrogeological uncertainty;
- Climate uncertainty and associated water management needs;
- Changes to the planned integration of mining operations and the TSF construction.

1.27.2 Opportunities

Opportunities include:

- Mineral Resources exclusive of Mineral Reserves: a portion of the estimated Measured and Indicated Mineral Resources were not converted to Mineral Reserves. This material represents upside potential for extending the mine life once studies have been completed that support conversion to Mineral Reserves;
- Gold price: Mineral Reserves are based on a US$1,400/oz gold price. Using a higher gold price for pit design and a higher cut off represents upside potential for extending the mine life;
- Inferred Mineral Resources: there is upside potential for the estimates if mineralization that is currently classified as Inferred can be upgraded to higher-confidence Mineral Resource categories;

- Mineralization remains open at depth: particularly in the northwest of the deposit where an increasing trend in gold grade is noted. Additional drilling and evaluation may support estimation of Mineral Resources in this area;
- Exploration potential: the Blackwater land package remains largely under-explored, and warrants additional exploration efforts;
- Evaluation of alternative methods for transportation of waste: the ex-pit haul route for waste material is expected to be relatively fixed for the LOM, opening up the possibility of using alternative methods to haul of waste material, which may have the potential to significantly reduce operating costs and lower greenhouse gas emissions;
- Electrification of the hauling fleet: the deployment of battery electric vehicles has the potential to significantly reduce operating costs and lower greenhouse gas emissions;
- Automation of hauling operations: the potential to automate hauling operations presents an opportunity to optimize production efficiencies and reduce operating costs;
- Process engineering initiatives: Evaluation of alternative processing methodologies that may result in lower capital and operating costs for Phase 3.

1.28 Interpretation and Conclusions

Under the assumptions described in this Report, the proposed LOM plan is achievable, and the economic analysis supports declaration of Mineral Reserves.

1.29 Recommendations

A single phase work program is proposed. All elements of the program can be conducted concurrently. The estimated budget to complete the program is about C$2.7 M, comprising the following key elements:

- Mining: recommended work programs include additional geotechnical drilling; evaluating opportunities to mine on larger benches in waste zones; trade-off studies to investigate the merits of implementing technologies; and other technologies to optimize material movement from the open pit to various mine infrastructure. The budget estimate to complete is approximately C$1.5 M.
- Process: recommended work programs include additional testwork on ore samples from areas in the latter half of the production cycle, and completion of flotation, ultrafine grinding, and leach testwork in support of the Phase 3 circuit. The budget estimate to complete is approximately C$1.1 M;
- Geotechnical and water management: recommended work programs include updating the existing site investigation plan, and updating of the LOM water balance model. The costs to

complete the site investigation work are included in the Phase 2 growth capital costs, and an additional budget of approximately C$100,000 will be required in addition to the allocated costs to update the LOM water balance model.

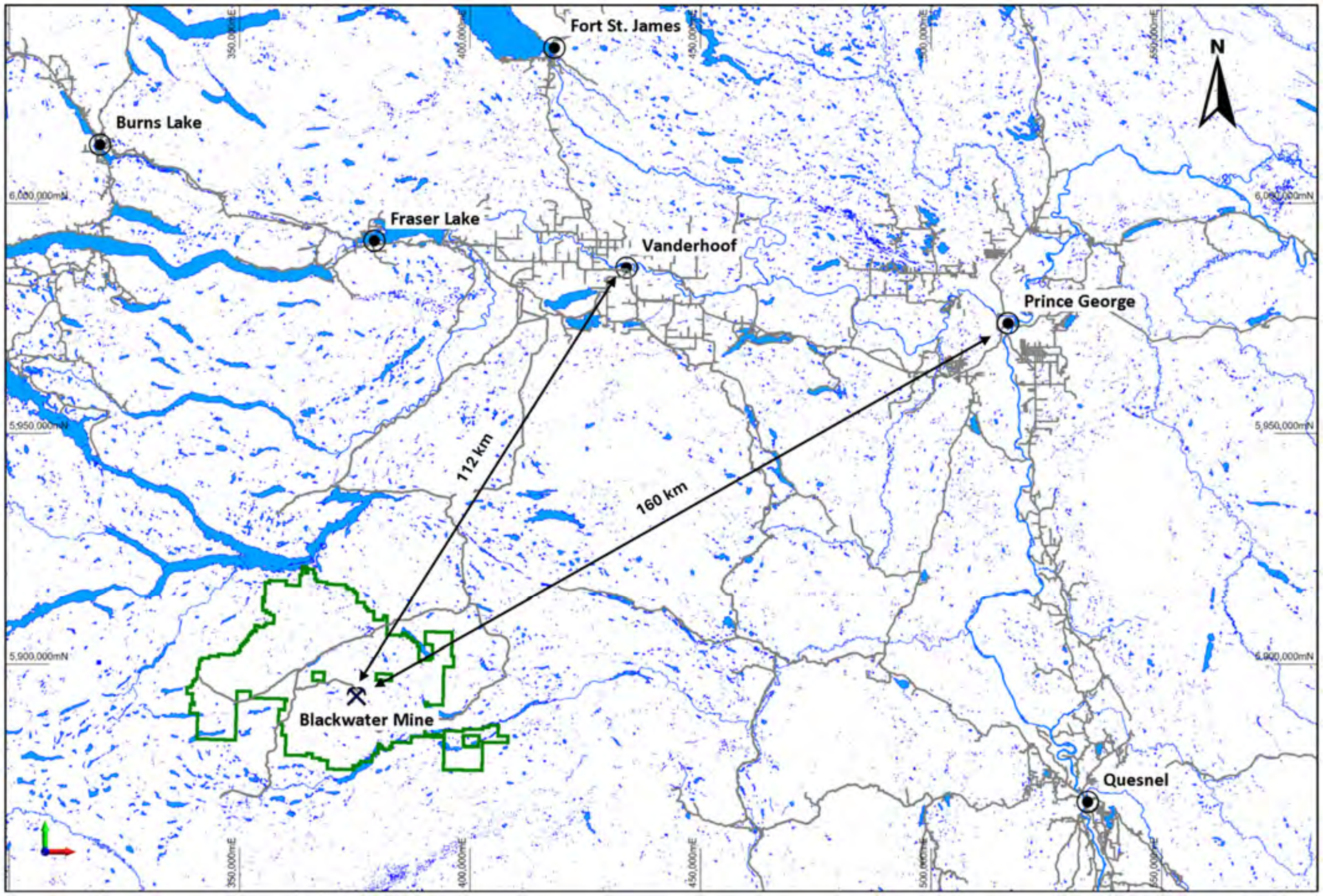
2.0 INTRODUCTION

2.1 Introduction

Ms. Sue Bird, P.Eng., Mr. Marc Schulte, P.Eng., Dr. John A. Thomas, P.Eng., Mr. Daniel Fontaine, P.Eng., Mr. Rolf Schmitt, P.Geo., Mr. John Dockrey, P.Geo., Mr. Olav Mejia, P.Eng., and Mr. Sohail Samdani, P.Eng., collectively the Qualified Persons (QPs), prepared an NI 43-101 Technical Report (the Report) on phased operations (Phase 1, Phase 2, and Phase 3; collectively the 2024 Expansion Study) for the Blackwater Gold Mine for Artemis Gold Inc. (Artemis Gold).

The Blackwater Gold Mine is located in central British Columbia, approximately 112 km southwest of Vanderhoof and 446 km northeast of Vancouver (Figure 2-1).

Figure 2-1: Project Location Plan



Note: Figure prepared by Artemis Gold, 2024. Green outline is the outline of the Project mineral tenure.

The construction of the Phase 1 processing plant of 6 Mt/a is well advanced, and this Report assumes that Phase 1 has been completed. The 2024 Expansion Study purpose is to optimize the timing of mine expansion. The expansions are expected to be funded from operating cash flows based on the input assumptions of the study. The expansion study is based on Blackwater's existing Proven and Probable Mineral Reserves. No changes were made to the Mineral Reserve and Mineral Resource estimates in the study. All cost estimates are at a minimum pre-feasibility study level. Phase 1 costs are assumed to be sunk costs; however, debt servicing of the Project loan facility, which partly funded Phase 1, is recognized in the cashflows modelled, as are the various royalties and streams related to the Blackwater Gold Mine.

2.2 Terms of Reference

The Report was prepared to support disclosures in the Artemis Gold news release entitled "Artemis Gold Announces Results of Expansion Study for Blackwater Mine", dated 21 February 2024.

The term "Project" is used to refer to the overall area within the mineral tenure holdings that form the Blackwater Project. The term "Blackwater Gold Mine" is used to refer interchangeably to the mine site area, the existing Blackwater operation, and the expansion of the Blackwater Gold Mine.

This Report provides expansion details planned for the Blackwater Gold Mine:

- Phase 1: assumed to be completed for the purposes of this Report; process plant operating at 6 Mt/a;
- Phase 2: process plant capacity increase of 9 Mt/a in Q4 of Year 2, for an overall 15 Mt/a production rate starting in Year 3;
- Phase 3: process plant capacity increase in Year 7 of 9.5 Mt/a and optimization of Phase 1 plant for an additional 0.5 Mt/a achieving an overall plant capacity of 25.0 Mt/a.

For the purposes of the capital cost estimates, "growth capital" is the sum of the deferred + expansion capital cost estimates.

Mineral Resources and Mineral Reserves are classified using the 2014 edition of the Canadian Institute of Mining and Metallurgy (CIM) Definition Standards for Mineral Resources and Mineral Reserves (the 2014 CIM Definition Standards).

Units used in the Report are metric units unless otherwise noted. Monetary units are in Canadian dollars (C$) unless otherwise stated.

2.3 Qualified Persons

The following serve as the qualified persons (QPs) for this Report as defined in National Instrument 43-101, Standards of Disclosure for Mineral Projects, and in compliance with Form 43-101F1:

- Ms. Sue Bird, P.Eng., Senior Vice President of Geology and Underground, Moose Mountain Technical Services (Moose Mountain);
- Mr. Marc Schulte, P.Eng., Vice President of Engineering and Operations, Moose Mountain;
- Dr. John A. Thomas, P.Eng., Director, JAT MetConsult Ltd. (JAT MetConsult);
- Mr. Daniel Fontaine, P.Eng., Specialist Engineer and Associate, Knight Piésold Ltd. (Knight Piésold);
- Mr. Rolf Schmitt, P.Geo., Technical Director, ERM Consultants Canada Ltd. (ERM);
- Mr. John Dockrey, P.Geo., Environmental Geochemist, Lorax Environmental Services Ltd. (Lorax);
- Mr. Olav Mejia, P.Eng., Manager, Process Engineering, Lycopodium Minerals Canada Limited (Lycopodium);
- Mr. Sohail Samdani, P.Eng., President, Lycopodium.

2.4 Site Visits and Scope of Personal Inspection

2.4.1 Ms. Sue Bird

Ms. Bird visited the site on 14 July 2020. She reviewed the drill hole locations and layout, the core storage, the camp site, the sampling protocols followed during drilling, and the quality assurance/quality control (QA/QC) procedures. The geology and mineralization within pertinent drill holes were also inspected and reviewed.

2.4.2 Mr. Marc Schulte

Mr. Schulte visited the site on 14 July 2020. During his visit, Marc viewed the general topography, inspected proposed pit and stockpile locations, and the locations of existing and proposed infrastructure. Verification that topography has not been materially altered from the date of the site visit to 2024 was done via site photos and videos taken of the open pit and stockpile areas in late 2023, and an updated site visit is considered unnecessary.

2.4.3 Dr. John Thomas

Dr. Thomas visited the site on 7 May, 2020. During his visit, Dr. Thomas viewed the proposed location of the processing plant, pits, WRSFs, haul roads, TSF, reviewed selected drill core samples, and reviewed the on-site and off-site infrastructure options.

2.4.4 Mr. Daniel Fontaine

Mr. Daniel Fontaine visited the site most recently from 7–9 February, 2024, as well as on several previous occasions between 2011 and 2023. During his visits, Mr. Fontaine viewed the proposed locations of the process plant, open pit, low-grade ore, and waste rock storage facilities, select haul roads, tailings storage facility (TSF) and associated dam sites, and select locations associated with water management components.

2.4.5 Mr. Olav Mejia

Mr. Mejia visited the site on 14 November, 2023. During his visit, he viewed the ongoing Phase 1 construction. Mr. Mejia viewed the warehouse, the laydown area for main equipment such as motors and mill shells, construction of the crushing and grinding area, and the installation progress of secondary and tertiary crushing equipment.

2.5 Effective Dates

The Report has several effective dates including:

- Date of the Mineral Resource estimate: 5 May, 2020;
- Date of the Mineral Reserve estimate: 10 September, 2021;
- Date of the economic analysis that supports the Mineral Reserves: 21 February, 2024.

The overall effective date of the Report is the date of the economic analysis that supports the Mineral Reserves, and is 21 February, 2024.

2.6 Information Sources and References

The reports and documents listed in Section 2.7 and Section 27 of this Report were used to support the preparation of the Report.

Additional information was sought from Artemis Gold personnel where required.

2.7 Previous Technical Reports

Artemis Gold has filed the following technical reports on the Project:

- Kalanchey, R., Bird, S., Dermer, G., Fontaine, D., Garner, J., Schulte, M., Lee, P., Dockrey, J., and Thomas J.A., 2021: NI 43-101 Technical Report on Updated Feasibility Study: report prepared for Artemis Gold Inc. by Ausenco Engineering Canada, Moose Mountain Technical Services, Knight Piésold Ltd., Allnorth, ERM, LORAX, and JAT Met Consult Ltd., effective date 10 September, 2021;
- Bird, S., Fontaine, D., Meintjes, T., Schulte, M., Thomas, J., 2020: Technical Report, Blackwater Gold Project British Columbia NI 43-101 Technical Report on Pre-Feasibility Study: report prepared for Artemis Gold Inc. by Moose Mountain Technical Services, Knight Piésold Ltd, JAT Met Consult Ltd, effective date 26 August, 2020.

New Gold, the previous Project owner, filed the following technical reports:

- Christie, G., Lipiec T., Simpson, R.G., Horton, J., and Borntraeger, B., 2014: Blackwater Gold Project, British Columbia, NI 43-101 Technical Report on Feasibility Study: report prepared for New Gold Inc. by AMEC Americas Ltd., GeoSim Services Inc, Norwest Corporation, and Knight Piésold Ltd., effective date 14 January 2014;
- Simpson, R.G., Welhener, H.E., Borntraeger, B., Lipiec T., and Mendoza, R., 2012: Blackwater Project British Columbia, Canada NI 43-101 Technical Report on Preliminary Economic Assessment: report prepared for New Gold Inc. by GeoSim Services Inc, Independent Mining Consultants Inc, Knight Piésold Ltd. and AMEC Americas Ltd., effective date 28 August, 2012;
- Simpson, R., 2012: Technical Report, Blackwater Gold Project, Omineca Mining Division, British Columbia, Canada: report prepared for New Gold Inc. and Silver Quest Resources Ltd., effective date 7 March, 2012;
- Simpson, R., 2011a: Technical Report, Blackwater Gold Project, Omineca Mining Division, British Columbia, Canada: report prepared for New Gold Inc. and Silver Quest Resources Ltd., effective date 2 March, 2011, re-addressed 6 June, 2011;
- Simpson, R., 2011b: Technical Report, Blackwater Gold Project, Omineca Mining Division, British Columbia, Canada: report prepared for New Gold Inc. and Silver Quest Resources Ltd., effective date 19 September, 2011.

3.0 RELIANCE ON OTHER EXPERTS

3.1 Introduction

The QPs have relied upon the following other expert reports, which provided information on mineral tenure, taxation, and marketing assumptions.

3.2 Mineral Tenure, Surface Rights, Royalties and Agreements

The QPs have not reviewed the mineral tenure, nor independently verified the legal status, ownership of the Project area or underlying property agreements. The QPs refer to and fully rely upon, and disclaim responsibility for, information supplied by Artemis Gold experts and experts retained by Artemis Gold for this information through the following document:

- Artemis Gold, 2024: RE: Blackwater Gold Mine, British Columbia, NI 43-101 Technical Report: mineral title opinion letter prepared for Moose Mountain Technical Services, 2 April, 2024.

This information is used in Section 4 of the Report, and in support of the Mineral Resource estimate in Section 14, the Mineral Reserve estimate in Section 15, and the economic analysis in Section 22.

4.0 PROPERTY DESCRIPTION AND LOCATION

4.1 Introduction

The Project is located in central British Columbia, approximately 112 km southwest of Vanderhoof and 446 km northeast of Vancouver (refer to Figure 2-1). The Project is within NTS map sheet 93F/02 and is centred at 5893000 N and 375400 E (UTM NAD83).

4.2 Project Ownership

4.2.1 Ownership History

The general Project area was initially explored by Granges Inc. (Granges) from 1973. In 2005, Silver Quest Resources Ltd. (Silver Quest) acquired an interest in the area and entered into a joint venture with Richfield Ventures Corp. (Richfield) in 2009.

In 2011, New Gold Inc. (New Gold) acquired Richfield and Silver Quest, and a third company, Geo Minerals Limited (Geo), to consolidate the ground holdings that currently comprise the Project.

In 2012, New Gold signed an option agreement to earn a 100% interest in a single Capoose area mineral claim from a private corporation.

In 2012, New Gold acquired the Auro properties from Gold Reach Resources Ltd. (Gold Reach) and added several additional mineral claims. New Gold acquired the Key claims in 2013 from Troymet Exploration Corporation (Troymet).

In January 2014 New Gold recorded four additional mineral claims, which were added to the Blackwater Project.

In June 2017, New Gold acquired the BW East, BW West, and BW South claims from RJK Explorations Ltd. (RJK) as well as the Big Bear property from Parlane Resource Corp. (Parlane).

4.2.2 Current Ownership

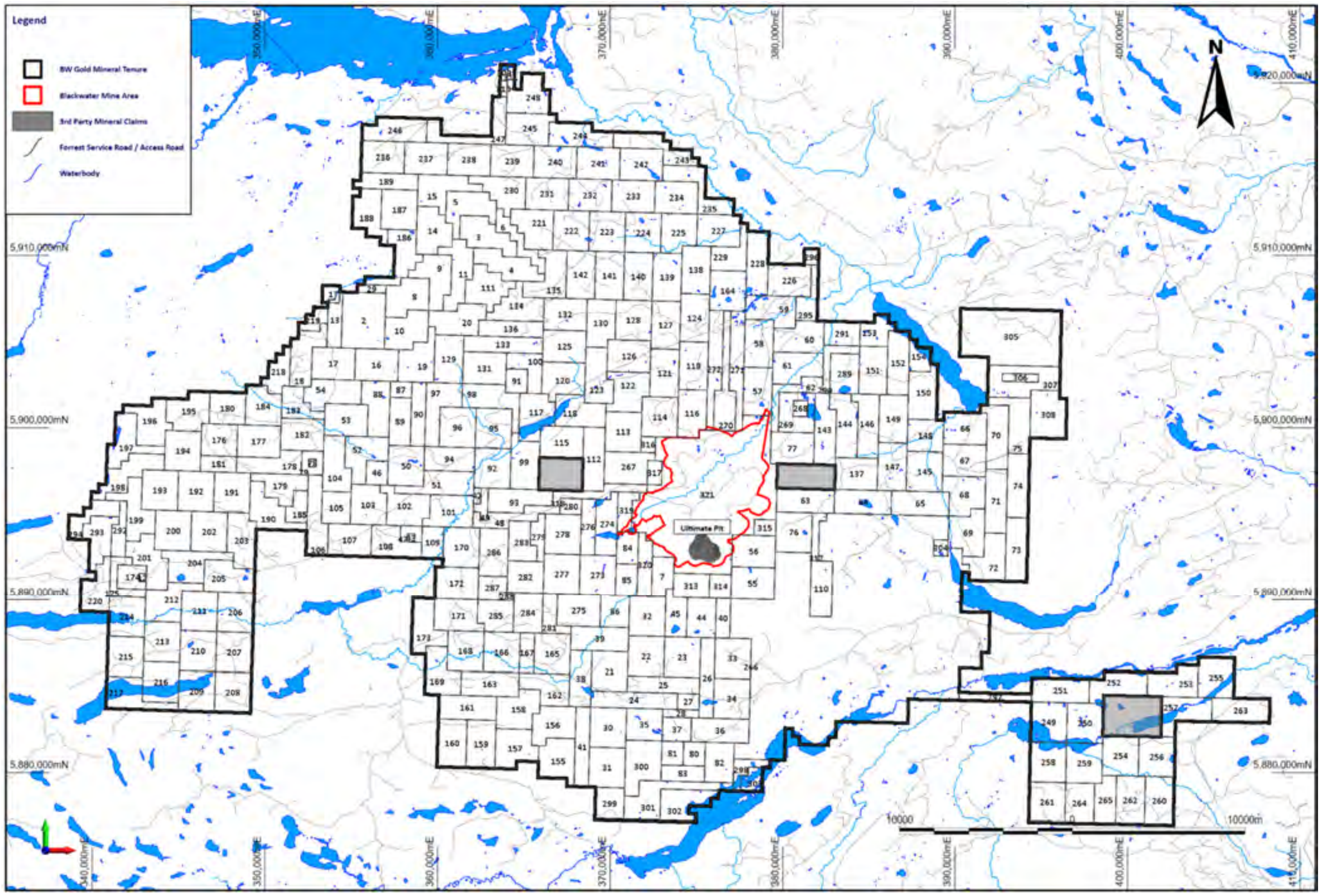
On 21 August, 2020, Artemis Gold acquired all of New Gold's mineral tenure, assets and rights related to the Project and now indirectly holds a 100% interest in the Project. Artemis Gold uses its wholly owned subsidiary, BW Gold Ltd. as the operating entity for the Project and the Blackwater Gold Mine.

4.3 Mineral Tenure

Artemis Gold, through BW Gold, holds a 100% recorded interest in 320 mineral claims and one mineral lease (the Blackwater mining lease), covering a total area of 149,037.3 ha.

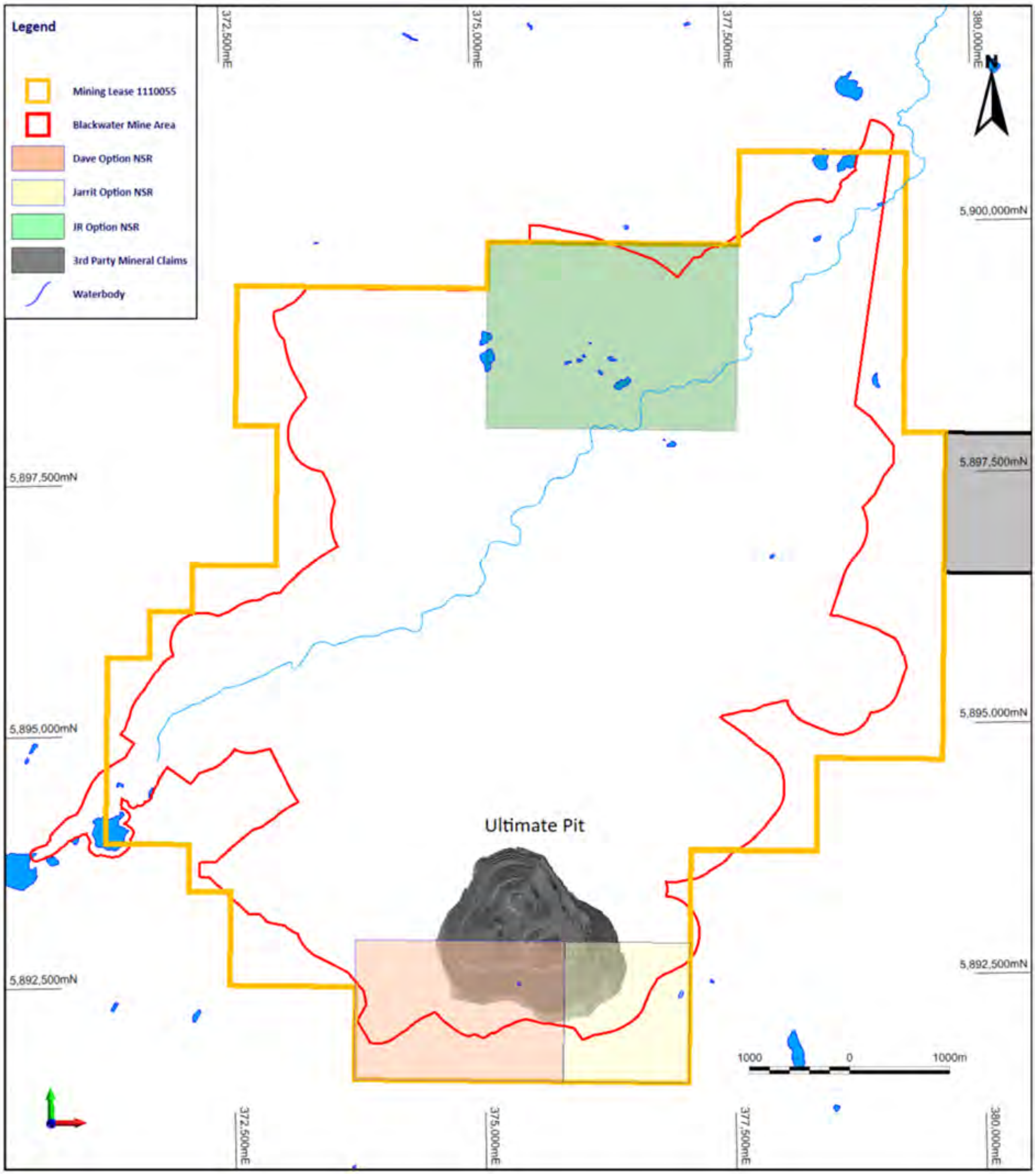
A mineral tenure location map is provided in Figure 4-1. The location of the planned open pit within the Blackwater mining lease is shown in Figure 4-2. A list of the mineral claims is included as Table 4-1. The claim reference numbers shown on Figure 4-1 are provided in the first column of the table. The notes/comments column of Table 4-1 refers to the royalty agreements summarized in Table 4-2. The areas of the key royalty agreements that pertain to the Blackwater mining lease were shown on Figure 4-2. Table 4-2 also refers to two claims that have partial claim overlaps.

Figure 4-1: Mineral Tenure Map



Note: Figure prepared by Artemis Gold, 2024. Blackwater Mine Area is the outline of the permitted mine area. Black lines show claim outlines. Claims shown as greyed-out are held by parties other than Artemis Gold.

Figure 4-2: Location Plan, Blackwater Mining Lease



Note: Figure prepared by Artemis Gold, 2024.

Table 4-1: Mineral Tenure Summary Table

Map No.	Title Number	Claim Name	Title Sub Type	Issue Date	Good To Date	Area (ha)	Notes/ Comments
1	238045	Cap	Claim	1978/Sep/18	2028/Oct/16	100.0	Four Post Claim
2	512838		Claim	2005/May/17	2029/Jan/30	811.9	b
3	534364	Jag-1	Claim	2006/May/24	2029/Jan/30	482.8	c
4	534365	Jag-2	Claim	2006/May/24	2029/Jan/30	482.9	c
5	534366	Jag-3	Claim	2006/May/24	2029/Jan/30	482.6	c
6	534367	Jag-4	Claim	2006/May/24	2029/Jan/30	289.7	c
7	536650	Night Flight	Claim	2006/Jul/06	2028/Oct/16	271.4	
8	552493	NE Capoose	Claim	2007/Feb/22	2029/Jan/30	483.1	b
9	552494	NE Capoose 2	Claim	2007/Feb/22	2029/Jan/30	483.0	b
10	552495	E Capoose	Claim	2007/Feb/22	2028/Oct/16	483.3	b
11	552497	NE Capoose3	Claim	2007/Feb/22	2029/Jan/30	483.0	c
12	553489	Paw	Claim	2007/Mar/03	2028/Oct/16	19.4	d
13	555053	Cap	Claim	2007/Mar/26	2028/Oct/16	251.3	
14	557495	Jag-5	Claim	2007/Apr/23	2029/Jan/30	482.7	c
15	557496	Jag-6	Claim	2007/Apr/23	2029/Jan/30	482.5	c
16	564372	Capoose S	Claim	2007/Aug/09	2028/Oct/16	464.2	b
17	564373	Capoose SW	Claim	2007/Aug/09	2028/Oct/16	464.2	b
18	564375	Capoose SW2	Claim	2007/Aug/09	2028/Oct/16	483.5	b
19	564376	Capoose E2	Claim	2007/Aug/09	2028/Oct/16	483.5	b
20	564377	Capoose E3	Claim	2007/Aug/09	2029/Jan/30	483.2	b
21	564994	Key 1	Claim	2007/Aug/24	2028/Oct/16	485.2	e
22	564995	Key 2	Claim	2007/Aug/24	2028/Oct/16	485.2	e

Map No.	Title Number	Claim Name	Title Sub Type	Issue Date	Good To Date	Area (ha)	Notes/ Comments
23	564996	Key 3	Claim	2007/Aug/24	2028/Oct/16	485.2	e
24	564997	Key 4	Claim	2007/Aug/24	2028/Oct/16	466.0	e
25	564998	Key 5	Claim	2007/Aug/24	2028/Oct/16	388.3	e
26	564999	Key 6	Claim	2007/Aug/24	2028/Oct/16	388.2	e
27	565000	Key 7	Claim	2007/Aug/24	2028/Oct/16	116.5	e
28	565001	Key 8	Claim	2007/Aug/24	2028/Oct/16	97.1	e
29	580086	Capoose North	Claim	2008/Apr/01	2029/Jan/30	77.3	
30	589167	Lock 1	Claim	2008/Jul/30	2028/Oct/16	485.6	e
31	589177	Lock 2	Claim	2008/Jul/30	2028/Oct/16	485.8	e
32	589183	Lock 3	Claim	2008/Jul/30	2028/Oct/16	484.9	e
33	589231	Lock 4	Claim	2008/Jul/30	2029/Jan/30	485.2	e
34	589232	Lock 5	Claim	2008/Jul/30	2029/Jan/30	485.4	e
35	589234	Lock 6	Claim	2008/Jul/30	2028/Oct/16	388.4	e
36	589236	Lock 7	Claim	2008/Jul/30	2029/Jan/30	466.2	e
37	589238	Lock 8	Claim	2008/Jul/30	2028/Oct/16	233.1	e
38	589241	Lock 9	Claim	2008/Jul/30	2028/Oct/16	407.6	e
39	589242	Lock 10	Claim	2008/Jul/30	2028/Oct/16	465.7	e
40	589243	Lock 11	Claim	2008/Jul/30	2028/Oct/16	194.0	e
41	589244	Lock 12	Claim	2008/Jul/30	2028/Oct/16	388.6	e
42	598000	Buck	Claim	2009/Jan/26	2028/Oct/16	38.7	
43	601527	Fawn	Claim	2009/Mar/23	2028/Oct/16	19.4	
44	602167	BWD	Claim	2009/Apr/05	2028/Oct/16	387.9	
45	602168	BWD2	Claim	2009/Apr/05	2028/Oct/16	310.3	
46	606724	Fawn	Claim	2009/Jun/27	2028/Oct/16	174.3	
47	606728	Malaput E-W	Claim	2009/Jun/27	2028/Oct/16	96.9	

Map No.	Title Number	Claim Name	Title Sub Type	Issue Date	Good To Date	Area (ha)	Notes/ Comments
48	607195	Blackwater 1	Claim	2009/Jul/08	2028/Oct/16	348.7	
49	617183	Buck 2	Claim	2009/Aug/10	2028/Oct/16	96.9	
50	625583	M-1	Claim	2009/Aug/29	2028/Oct/16	484.1	
51	625603	M-2	Claim	2009/Aug/29	2028/Oct/16	484.2	
52	625623	M-3	Claim	2009/Aug/29	2028/Oct/16	484.0	
53	625624	M-4	Claim	2009/Aug/29	2028/Oct/16	464.5	
54	625625		Claim	2009/Aug/29	2028/Oct/16	483.7	
55	630903	BW1	Claim	2009/Sep/09	2029/Jan/30	465.3	
56	630963	BW3	Claim	2009/Sep/09	2029/Jan/30	465.2	
57	636603	Kassy 2	Claim	2009/Sep/18	2029/Jan/30	464.3	
58	636604	Kassy 3	Claim	2009/Sep/18	2029/Jan/30	464.0	
59	636623	Kassy 4	Claim	2009/Sep/18	2029/Jan/30	463.8	
60	636643	Kassy 5	Claim	2009/Sep/18	2029/Jan/30	483.3	
61	636644	Kassy 6	Claim	2009/Sep/18	2029/Jan/30	483.5	
62	636663	Kassy 7	Claim	2009/Sep/18	2029/Jan/30	290.2	
63	636683	Right Stuff 1	Claim	2009/Sep/18	2029/Jan/30	464.9	
64	636684	Right Stuff 2	Claim	2009/Sep/18	2029/Jan/30	464.9	
65	636703	Right Stuff 3	Claim	2009/Sep/18	2029/Jan/30	464.9	
66	636723	Right Stuff 4	Claim	2009/Sep/18	2029/Jan/30	464.4	
67	636724	Right Stuff	Claim	2009/Sep/18	2029/Jan/30	464.6	
68	636725	Right Stuff 6	Claim	2009/Sep/18	2029/Jan/30	484.2	
69	636727	Right Stuff 7	Claim	2009/Sep/18	2029/Jan/30	484.4	
70	636743	Right Stuff 8	Claim	2009/Sep/18	2029/Jan/30	483.8	
71	636763	Right Stuff 9	Claim	2009/Sep/18	2029/Jan/30	464.8	
72	636764	Right Stuff 10	Claim	2009/Sep/18	2029/Jan/30	484.6	

Map No.	Title Number	Claim Name	Title Sub Type	Issue Date	Good To Date	Area (ha)	Notes/ Comments
73	636765	Right Stuff 11	Claim	2009/Sep/18	2029/Jan/30	465.1	
74	636766	Right Stuff 12	Claim	2009/Sep/18	2029/Jan/30	464.8	
75	636767	Right Stuff 13	Claim	2009/Sep/18	2029/Jan/30	464.4	
76	637203	Ozzy	Claim	2009/Sep/19	2029/Jan/30	484.4	f
77	637205	Baby Jane	Claim	2009/Sep/19	2029/Jan/30	464.6	f
78	641983	Fawn	Claim	2009/Sep/27	2028/Oct/16	19.4	d
79	641984	Fawn 2	Claim	2009/Sep/27	2028/Oct/16	154.9	d
80	642003	Yellow & Black	Claim	2009/Sep/27	2028/Oct/16	174.9	g
81	642004	Black & Yellow	Claim	2009/Sep/27	2028/Oct/16	174.9	g
82	642023	Black	Claim	2009/Sep/27	2028/Oct/16	388.6	g
83	642024	Yellow	Claim	2009/Sep/27	2028/Oct/16	233.2	g
84	642043	BW	Claim	2009/Sep/27	2028/Oct/16	232.6	
85	642063	BW 2	Claim	2009/Sep/27	2028/Oct/16	232.7	
86	642064	BW3	Claim	2009/Sep/27	2028/Oct/16	310.3	
87	642544	Fawnie Dome	Claim	2009/Sep/28	2028/Oct/16	116.1	d
88	642564	FD 2	Claim	2009/Sep/28	2028/Oct/16	464.4	d
89	642565	FD 3	Claim	2009/Sep/28	2028/Oct/16	348.4	d
90	642583	FD 4	Claim	2009/Sep/28	2028/Oct/16	309.6	d
91	642603	Top Lake	Claim	2009/Sep/28	2028/Oct/16	174.1	
92	643103	Buck 1	Claim	2009/Sep/29	2028/Oct/16	484.1	h
93	643104	Buck 2	Claim	2009/Sep/29	2028/Oct/16	445.5	h
94	643106	Buck 3	Claim	2009/Sep/29	2028/Oct/16	406.6	h
95	643107	Buck 4	Claim	2009/Sep/29	2028/Oct/16	483.9	h
96	643108	Buck 5	Claim	2009/Sep/29	2028/Oct/16	483.9	h
97	643109	Buck 6	Claim	2009/Sep/29	2028/Oct/16	483.7	h

Map No.	Title Number	Claim Name	Title Sub Type	Issue Date	Good To Date	Area (ha)	Notes/ Comments
98	643110	Buck 7	Claim	2009/Sep/29	2028/Oct/16	483.7	h
99	643123	Buck 8	Claim	2009/Sep/29	2029/Jan/30	484.1	h
100	643323	Top	Claim	2009/Sep/29	2028/Oct/16	309.4	
101	644244	Capoose M6	Claim	2009/Sep/30	2028/Oct/16	484.3	
102	644283	Capoose M7	Claim	2009/Sep/30	2028/Oct/16	484.3	
103	644285	Capoose M8	Claim	2009/Sep/30	2028/Oct/16	465.0	
104	644323	Capoose M9	Claim	2009/Sep/30	2028/Oct/16	464.7	
105	644363	Capoose M10	Claim	2009/Sep/30	2028/Oct/16	310.0	
106	645063	Capoose M11	Claim	2009/Sep/30	2028/Oct/16	465.0	q
107	645064	Capoose M12	Claim	2009/Sep/30	2028/Oct/16	465.1	
108	645065	Capoose M13	Claim	2009/Sep/30	2028/Oct/16	426.4	
109	645066	Capoose M14	Claim	2009/Sep/30	2028/Oct/16	232.6	
110	646683	Princess	Claim	2009/Oct/03	2029/Jan/30	407.2	i
111	649243	Jag-8	Claim	2009/Oct/08	2029/Jan/30	483.1	c
112	694043		Claim	2010/Jan/04	2029/Jan/30	464.7	j
113	694044		Claim	2010/Jan/04	2029/Jan/30	483.9	j
114	694045		Claim	2010/Jan/04	2029/Jan/30	483.8	j
115	694046		Claim	2010/Jan/04	2029/Jan/30	464.6	j
116	694063		Claim	2010/Jan/04	2029/Jan/30	445.0	j
117	694064		Claim	2010/Jan/04	2029/Jan/30	483.8	j
118	694065		Claim	2010/Jan/04	2029/Jan/30	483.7	j
119	694066		Claim	2010/Jan/04	2029/Jan/30	464.1	j
120	694083		Claim	2010/Jan/04	2029/Jan/30	483.6	j
121	694084		Claim	2010/Jan/04	2029/Jan/30	464.2	j
122	694085		Claim	2010/Jan/04	2029/Jan/30	483.6	j

Map No.	Title Number	Claim Name	Title Sub Type	Issue Date	Good To Date	Area (ha)	Notes/ Comments
123	694086		Claim	2010/Jan/04	2029/Jan/30	464.2	j
124	694087		Claim	2010/Jan/04	2029/Jan/30	463.9	j
125	694088		Claim	2010/Jan/04	2029/Jan/30	464.0	j
126	694089		Claim	2010/Jan/04	2029/Jan/30	464.1	j
127	694090		Claim	2010/Jan/04	2029/Jan/30	463.9	j
128	694103		Claim	2010/Jan/04	2029/Jan/30	483.2	j
129	694123		Claim	2010/Jan/04	2028/Oct/16	464.1	j
130	694143		Claim	2010/Jan/04	2029/Jan/30	444.6	j
131	694144		Claim	2010/Jan/04	2028/Oct/16	464.2	j
132	694145		Claim	2010/Jan/04	2029/Jan/30	463.9	j
133	694146		Claim	2010/Jan/04	2028/Oct/16	425.4	j
134	694147		Claim	2010/Jan/04	2029/Jan/30	463.8	j
135	694148		Claim	2010/Jan/04	2029/Jan/30	483.0	j
136	694163		Claim	2010/Jan/04	2028/Oct/16	348.0	j
137	694164		Claim	2010/Jan/04	2029/Jan/30	464.7	k
138	694183		Claim	2010/Jan/04	2029/Jan/30	463.6	j
139	694184		Claim	2010/Jan/04	2029/Jan/30	463.6	j
140	694185		Claim	2010/Jan/04	2029/Jan/30	463.6	j
141	694186		Claim	2010/Jan/04	2029/Jan/30	463.6	j
142	694187		Claim	2010/Jan/04	2029/Jan/30	463.6	j
143	694188		Claim	2010/Jan/04	2029/Jan/30	483.8	k
144	694203		Claim	2010/Jan/04	2029/Jan/30	483.8	k
145	694204		Claim	2010/Jan/04	2029/Jan/30	484.1	k
146	694205		Claim	2010/Jan/04	2029/Jan/30	483.8	k
147	694206		Claim	2010/Jan/04	2029/Jan/30	464.7	k

Map No.	Title Number	Claim Name	Title Sub Type	Issue Date	Good To Date	Area (ha)	Notes/ Comments
148	694207		Claim	2010/Jan/04	2029/Jan/30	387.1	k
149	694208		Claim	2010/Jan/04	2029/Jan/30	464.4	k
150	694209		Claim	2010/Jan/04	2029/Jan/30	483.6	k
151	694210		Claim	2010/Jan/04	2029/Jan/30	464.1	k
152	694223		Claim	2010/Jan/04	2029/Jan/30	464.1	k
153	694224		Claim	2010/Jan/04	2029/Jan/30	290.0	k
154	694225		Claim	2010/Jan/04	2029/Jan/30	193.4	k
155	694243		Claim	2010/Jan/04	2028/Oct/16	485.8	l
156	694245		Claim	2010/Jan/04	2028/Oct/16	485.6	l
157	694263		Claim	2010/Jan/04	2028/Oct/16	485.7	l
158	694264		Claim	2010/Jan/04	2028/Oct/16	485.5	l
159	694265		Claim	2010/Jan/04	2028/Oct/16	466.3	l
160	694283		Claim	2010/Jan/04	2028/Oct/16	466.3	l
161	694284		Claim	2010/Jan/04	2028/Oct/16	466.1	l
162	694285		Claim	2010/Jan/04	2028/Oct/16	427.1	l
163	694286		Claim	2010/Jan/04	2028/Oct/16	465.9	l
164	694287		Claim	2010/Jan/04	2029/Jan/30	483.0	j
165	694288		Claim	2010/Jan/04	2028/Oct/16	485.2	l
166	694289		Claim	2010/Jan/04	2028/Oct/16	485.2	l
167	694290		Claim	2010/Jan/04	2028/Oct/16	194.1	l
168	694291		Claim	2010/Jan/04	2028/Oct/16	485.2	l
169	694292		Claim	2010/Jan/04	2028/Oct/16	465.9	l
170	694293		Claim	2010/Jan/04	2028/Oct/16	484.6	l
171	694294		Claim	2010/Jan/04	2028/Oct/16	484.9	l
172	694295		Claim	2010/Jan/04	2028/Oct/16	465.4	l

Map No.	Title Number	Claim Name	Title Sub Type	Issue Date	Good To Date	Area (ha)	Notes/ Comments
173	694296		Claim	2010/Jan/04	2028/Oct/16	465.6	l
174	704807	Pawing	Claim	2010/Jan/26	2028/Oct/16	213.3	d
175	704817	Paws	Claim	2010/Jan/26	2028/Oct/16	213.3	d
176	704825	Fawn West	Claim	2010/Jan/26	2028/Oct/16	387.2	d
177	704826	FW 2	Claim	2010/Jan/26	2028/Oct/16	464.6	d
178	704827	FW 3	Claim	2010/Jan/26	2028/Oct/16	406.6	d
179	704828	FW 4	Claim	2010/Jan/26	2028/Oct/16	387.4	d
180	704829	FW 5	Claim	2010/Jan/26	2028/Oct/16	387.0	d
181	704830	FW 6	Claim	2010/Jan/26	2028/Oct/16	193.6	d
182	704854	FW 7	Claim	2010/Jan/27	2028/Oct/16	464.6	d
183	704855	FW 8	Claim	2010/Jan/27	2028/Oct/16	309.6	d
184	704863	FW 9	Claim	2010/Jan/27	2028/Oct/16	445.1	d
185	706011	FW 10	Claim	2010/Feb/11	2028/Oct/16	193.7	d
186	706593	CPN1	Claim	2010/Feb/19	2029/Jan/30	482.9	
187	706594	CPN2	Claim	2010/Feb/19	2029/Jan/30	482.6	
188	706595	CPN3	Claim	2010/Feb/19	2029/Jan/30	444.0	
189	706596	CPN4	Claim	2010/Feb/19	2029/Jan/30	328.1	
190	706597	CPW1	Claim	2010/Feb/19	2028/Oct/16	484.4	q
191	706598	CPW2	Claim	2010/Feb/19	2028/Oct/16	484.3	
192	706599	CPW3	Claim	2010/Feb/19	2028/Oct/16	484.3	
193	706600	CPW4	Claim	2010/Feb/19	2028/Oct/16	484.3	
194	706602	CPW5	Claim	2010/Feb/19	2028/Oct/16	484.0	
195	706603	CPW6	Claim	2010/Feb/19	2028/Oct/16	483.8	
196	706605	CPW7	Claim	2010/Feb/19	2028/Oct/16	483.9	
197	706606	CPW8	Claim	2010/Feb/19	2028/Oct/16	484.0	

Map No.	Title Number	Claim Name	Title Sub Type	Issue Date	Good To Date	Area (ha)	Notes/ Comments
198	706607	CPW9	Claim	2010/Feb/19	2028/Oct/16	484.2	
199	706608	CPW9	Claim	2010/Feb/19	2028/Oct/16	484.4	
200	706609	CPW10	Claim	2010/Feb/19	2028/Oct/16	484.5	
201	706610	CPW11	Claim	2010/Feb/19	2028/Oct/16	484.7	
202	706612	CPW12	Claim	2010/Feb/19	2028/Oct/16	484.5	
203	706613	CPW13	Claim	2010/Feb/19	2028/Oct/16	484.5	
204	706614	CPW14	Claim	2010/Feb/19	2028/Oct/16	484.7	
205	706615	CPW15	Claim	2010/Feb/19	2028/Oct/16	484.8	
206	706616	CPW16	Claim	2010/Feb/19	2028/Oct/16	485.0	
207	706617	CPW17	Claim	2010/Feb/19	2028/Oct/16	485.2	
208	706618	CPW18	Claim	2010/Feb/19	2028/Oct/16	485.4	
209	706619	CPW19	Claim	2010/Feb/19	2028/Oct/16	485.4	
210	706620	CPW20	Claim	2010/Feb/19	2028/Oct/16	485.2	
211	706621	CPW21	Claim	2010/Feb/19	2028/Oct/16	485.0	
212	706622	CPW22	Claim	2010/Feb/19	2028/Oct/16	484.9	
213	706623	CPW23	Claim	2010/Feb/19	2028/Oct/16	485.2	
214	706625	CPW24	Claim	2010/Feb/19	2028/Oct/16	485.0	
215	706626	CPW25	Claim	2010/Feb/19	2028/Oct/16	485.2	
216	706627	CPW26	Claim	2010/Feb/19	2028/Oct/16	485.4	
217	706628	CPW27	Claim	2010/Feb/19	2028/Oct/16	485.5	
218	706629	CPNW1	Claim	2010/Feb/19	2028/Oct/16	290.1	
219	706630	CPNW2	Claim	2010/Feb/19	2028/Oct/16	154.6	
220	706638	Paws 2	Claim	2010/Feb/19	2028/Oct/16	407.3	d
221	713362	KL1	Claim	2010/Mar/04	2029/Jan/30	482.7	m
222	713382	KL2	Claim	2010/Mar/04	2029/Jan/30	482.7	m

Map No.	Title Number	Claim Name	Title Sub Type	Issue Date	Good To Date	Area (ha)	Notes/ Comments
223	713402	KL3	Claim	2010/Mar/04	2029/Jan/30	482.7	m
224	713422	KL4	Claim	2010/Mar/04	2029/Jan/30	482.7	m
225	713442	KL6	Claim	2010/Mar/04	2029/Jan/30	444.1	m
226	713462	KL7	Claim	2010/Mar/04	2029/Jan/30	463.6	m
227	713482	KL8	Claim	2010/Mar/04	2029/Jan/30	463.4	m
228	713502	KL9	Claim	2010/Mar/04	2029/Jan/30	463.6	m
229	713522	KL10	Claim	2010/Mar/04	2029/Jan/30	347.6	m
230	713542	KL11	Claim	2010/Mar/04	2029/Jan/30	463.2	m
231	713562	KL12	Claim	2010/Mar/04	2029/Jan/30	463.2	m
232	713582	KL13	Claim	2010/Mar/04	2029/Jan/30	463.2	m
233	713602	KL14	Claim	2010/Mar/04	2029/Jan/30	463.2	m
234	713622	KL15	Claim	2010/Mar/04	2029/Jan/30	463.2	m
235	713642	KL16	Claim	2010/Mar/04	2029/Jan/30	463.2	m
236	713662	KL17	Claim	2010/Mar/04	2029/Jan/30	463.0	m
237	713682	KL18	Claim	2010/Mar/04	2029/Jan/30	463.0	m
238	713702	KL19	Claim	2010/Mar/04	2029/Jan/30	463.0	m
239	713722	KL20	Claim	2010/Mar/04	2029/Jan/30	463.0	m
240	713742	KL21	Claim	2010/Mar/04	2029/Jan/30	463.0	m
241	713782	KL22	Claim	2010/Mar/04	2029/Jan/30	463.0	m
242	713802	KL22	Claim	2010/Mar/04	2029/Jan/30	463.0	m
243	713822	KL23	Claim	2010/Mar/04	2029/Jan/30	462.9	m
244	713842	KL24	Claim	2010/Mar/04	2029/Jan/30	462.9	m
245	713862	KL25	Claim	2010/Mar/04	2029/Jan/30	482.1	m
246	713882	KL26	Claim	2010/Mar/04	2029/Jan/30	482.2	m
247	713902	KL27	Claim	2010/Mar/04	2029/Jan/30	482.1	m

Map No.	Title Number	Claim Name	Title Sub Type	Issue Date	Good To Date	Area (ha)	Notes/ Comments
248	713922	KL28	Claim	2010/Mar/04	2029/Jan/30	462.7	m
249	745822	NG1	Claim	2010/Apr/12	2029/Jan/30	485.5	i
250	745842	NG2	Claim	2010/Apr/12	2029/Jan/30	485.5	i
251	745862	NG3	Claim	2010/Apr/12	2029/Jan/30	465.9	i
252	745882	NG4	Claim	2010/Apr/12	2029/Jan/30	465.9	i
253	745902	NG5	Claim	2010/Apr/12	2029/Jan/30	465.8	i
254	745922	NG6	Claim	2010/Apr/12	2029/Jan/30	485.7	i
255	745942	NG7	Claim	2010/Apr/12	2029/Jan/30	485.2	i
256	745962	NG8	Claim	2010/Apr/12	2029/Jan/30	485.7	i
257	745982	NG9	Claim	2010/Apr/12	2029/Jan/30	485.4	i
258	746002	NG10	Claim	2010/Apr/12	2029/Jan/30	485.7	i
259	746022	NG11	Claim	2010/Apr/12	2029/Jan/30	485.7	i
260	746042	NG12	Claim	2010/Apr/12	2029/Jan/30	466.5	i
261	746062	NG13	Claim	2010/Apr/12	2029/Jan/30	486.0	i
262	746082	NG14	Claim	2010/Apr/12	2029/Jan/30	466.5	i
263	746102	NG15	Claim	2010/Apr/12	2029/Jan/30	466.0	i
264	746182	NG15	Claim	2010/Apr/12	2029/Jan/30	388.8	i
265	746202	NG16	Claim	2010/Apr/12	2029/Jan/30	330.4	i
266	831124	Auro Property	Claim	2010/Aug/05	2029/Jan/30	14,026.0	i, r
267	834367	Rich 1	Claim	2010/Sep/27	2029/Jan/30	484.1	
268	834533	Davidson 1	Claim	2010/Sep/29	2029/Jan/30	77.4	
269	834534	Davidson 2	Claim	2010/Sep/29	2029/Jan/30	406.3	
270	834923	Davidson 3	Claim	2010/Oct/02	2029/Jan/30	483.6	
271	834924	Davidson 4	Claim	2010/Oct/02	2029/Jan/30	483.5	
272	834926	Davidson 5	Claim	2010/Oct/02	2029/Jan/30	483.5	

Map No.	Title Number	Claim Name	Title Sub Type	Issue Date	Good To Date	Area (ha)	Notes/ Comments
273	834948		Claim	2010/Oct/03	2028/Oct/16	484.7	
274	834998	Rich 2	Claim	2010/Oct/04	2028/Oct/16	426.3	
275	835005		Claim	2010/Oct/04	2028/Oct/16	465.5	
276	835009		Claim	2010/Oct/04	2028/Oct/16	271.3	
277	835011		Claim	2010/Oct/04	2028/Oct/16	484.7	
278	835012		Claim	2010/Oct/04	2028/Oct/16	484.5	
279	835013		Claim	2010/Oct/04	2028/Oct/16	174.4	
280	835014	Dave	Claim	2010/Oct/04	2028/Oct/16	116.2	
281	835016		Claim	2010/Oct/04	2028/Oct/16	232.8	
282	835019		Claim	2010/Oct/04	2028/Oct/16	387.8	
283	835020		Claim	2010/Oct/04	2028/Oct/16	329.5	
284	835021	BW West	Claim	2010/Oct/04	2028/Oct/16	387.9	
285	835022	BW West2	Claim	2010/Oct/04	2028/Oct/16	368.6	
286	835023		Claim	2010/Oct/04	2028/Oct/16	465.2	
287	835024	Dave2	Claim	2010/Oct/04	2028/Oct/16	213.3	
288	835025	BW West2	Claim	2010/Oct/04	2028/Oct/16	38.8	
289	835434	Jonechako1	Claim	2010/Oct/08	2029/Jan/30	386.8	n
290	835436	Jonechako2	Claim	2010/Oct/08	2029/Jan/30	193.4	n
291	835527	Jonechako3	Claim	2010/Oct/09	2029/Jan/30	290.0	n
292	843656	Johnny North	Claim	2011/Jan/20	2028/Oct/16	213.2	
293	843657	Johnny NW	Claim	2011/Jan/20	2028/Oct/16	407.0	
294	843658	Johnny W	Claim	2011/Jan/20	2028/Oct/16	310.0	
295	920729	Jonblk	Claim	2011/Oct/21	2029/Jan/30	251.2	
296	940115	Noreadd	Claim	2012/Jan/06	2029/Jan/30	77.3	
297	982702	BW Bridge	Claim	2012/Apr/26	2029/Jan/30	194.1	

Map No.	Title Number	Claim Name	Title Sub Type	Issue Date	Good To Date	Area (ha)	Notes/ Comments
298	1015566	RJ1	Claim	2012/Dec/31	2029/Jan/30	174.9	
299	1015573	RJK4	Claim	2012/Dec/31	2028/Oct/16	311.0	
300	1015575	RJK5	Claim	2012/Dec/31	2028/Oct/16	563.5	
301	1015577	RJK6	Claim	2012/Dec/31	2028/Oct/16	602.6	
302	1015578	RJK8	Claim	2012/Dec/31	2028/Oct/16	466.6	
303	1015579	RJK9	Claim	2012/Dec/31	2029/Jan/30	155.5	
304	1018105		Claim	2013/Mar/27	2029/Jan/30	77.5	
305	1024945	BW NE	Claim	2014/Jan/09	2029/Jan/30	1,817.2	
306	1024947	BW-N 1	Claim	2014/Jan/09	2029/Jan/30	96.7	
307	1024948	BW-N 2	Claim	2014/Jan/09	2029/Jan/30	599.6	
308	1024956	BW-N 3	Claim	2014/Jan/09	2029/Jan/30	561.1	
309	1046035	The Cub	Claim	2016/Aug/18	2029/Jan/30	38.5	
310	1046802	The Cub 2	Claim	2016/Sep/19	2029/Jan/30	38.6	
311	1046869	The Cub 3	Claim	2016/Sep/22	2029/Jan/30	57.8	
312	1091359	Blackwater	Claim	2022/Jan/26	2033/Jan/26	135.6	
313	1107142		Claim	2005/Jul/01	2028/Oct/16	290.8	o
314	1107144		Claim	2005/Jul/01	2028/Oct/16	174.5	p
315	1107147		Claim	2009/Sep/09	2029/Jan/30	116.3	
316	1107149		Claim	2010/Sep/27	2029/Jan/30	77.4	
317	1107151		Claim	2009/Sep/09	2029/Jan/30	116.2	
318	1107152		Claim	2009/Jul/08	2029/Jan/30	290.5	
319	1107154		Claim	2009/Sep/26	2028/Oct/16	213.1	
320	1107156		Claim	2009/Sep/25	2028/Oct/16	290.8	
321	1110055		Lease	2024/Jan/08	2025/Jan/08	5,326.4	a

Table 4-2: Notes to Accompany Table 4-1

Note/Comment Number	Description
a	Mining lease
b	2.25% NSR payable to individual. Capoose Option Joint Venture, option to purchase four-ninths of NSR for C$1.5 M.
c	2.0% NSR payable to Individual. JAG Option, option to purchase half of NSR for C$1 M. Artemis Gold to pay annual advance royalty of C$30,000, to be credited against NSR royalty.
d	1.5% NSR payable to Individual. Capoose Property Option, option to purchase two thirds for C$2 M.
e	2.0% NSR payable to Troymet. Key Agreement, option to purchase half of NSR for C$2 M. Additional 3% NSR to individual. option to purchase two-thirds of NSR for C$1 M.
f	3.0% NSR payable to Silver Quest. JR Option, option to purchase two-thirds of NSR for C$1 M.
g	2.0% NSR payable to Troymet. Key Agreement, option to purchase half for C$2 M. Additional 2.0% NSR payable to individual. Option to purchase three-quarters of NSR for C$750,000.
h	1.5% NSR payable to Paget Minerals. Buck Option, two-thirds of NSR can be purchased for C$2 M.
i	2.0% NSR payable to Gold Reach. Auro Claims Block agreements and encumbrances.
j	2.0% NSR payable to individual. Parlane Claim Block Agreement Nov. 2010, option to purchase half of NSR for C$1 M.
k	2.0% NSR payable to individual. RJK Claim Block Agreement dated Dec. 3, 2010, option to purchase half of NSR for C$1 M.
l	2.0% NSR payable to individual. RJK Claim Block Agreement dated Dec. 1, 2010, option to purchase half of NSR for C$1 M.
m	2.0% NSR payable to individual. Parlane Claim Option Agreement Oct. 2010, option to purchase half for C$1 M.

	Additional 1.0% NSR payable to Greencastle Resources Ltd.
n	3.0% NSR payable to Individual. RJK Claim Block Agreement dated August 2011, option to purchase half of NSR for C$2 M.
o	1.5% NSR payable to individual. Dave Option.
p	1.0% NSR payable to individual Jarrit Option.
q	Partially overlap portion of a legacy claim held by third party
r	Partially overlaps a no-staking reserve

4.4 Surface Rights

Artemis Gold, as the holder of mineral claims and the mining lease for the lands upon which the mine site is located, does not have exclusive possession of the surface or exclusive right to use the surface of those lands. However, the holder of a mineral claim or a mining lease does have the right to access those lands for the purpose of exploring for minerals and to use the surface for mining activities (exploration, development, and production) and there is no legal requirement to obtain a surface lease (issued pursuant to the Land Act) or other surface tenure to undertake such activities within the area covered by the permitted mine area. No further surface rights are required.

The grant of Mines Act permit M-246 (refer to discussion in Section 20) provides Artemis Gold with the necessary authority to support construction of the required infrastructure for the life-of-mine (LOM), including the open pit, access roads, stockpiles, waste rock storage facilities (WRSFs) and tailings storage facility (TSF).

Artemis Gold was granted a license of occupation for the power transmission line on 25 April, 2023.

4.5 Water Rights

Artemis Gold has obtained all the water rights required to support mine construction and operations.

4.6 Royalties and Encumbrances

Table 4-2 summarized the royalty obligations for the Project area. All royalties are net smelter return royalties. The royalty areas that affect the mining lease were shown on Figure 4-2.

4.7 Property and Metals Streaming Agreements

The purchase agreement between Artemis Gold and New Gold included a gold stream agreement. New Gold maintained a security interest over the Blackwater Gold Mine in connection with the gold stream agreement. On 13 December, 2021, New Gold announced the sale of the gold stream agreement to Wheaton Precious Metals Corp (Wheaton).

4.8 Permitting Considerations

Permitting considerations are discussed in Section 20.

4.9 Environmental Considerations

Environmental considerations are discussed in Section 20.

4.10 Social License Considerations

Social licence considerations are discussed in Section 20.

4.11 QP Comments on Section 4

The QP notes:

- Information from legal and Artemis Gold's experts indicated that Artemis Gold holds 100% of the mineral claims and the Blackwater mining lease;
- Information from legal experts supports that the mineral tenure comprising the Project is valid and is sufficient to support declaration of Mineral Resources and Mineral Reserves;
- Information from legal experts noted that most of the Project is located on Crown lands;
- Based on the information provided by Artemis Gold, there are no other significant factors and risks known to the QP that may affect access, title, or the ability to perform work on the Project.

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

5.1 Accessibility

The Blackwater Gold Mine is readily accessible by vehicle from Vanderhoof using the Kluskus Forest Service Road and the Kluskus–Ootsa Forest Service Road. The Kluskus Forest Service Road joins Highway 16 about 10 km west of Vanderhoof.

The site can be accessed from the Kluskus–Ootsa Forest Service Road at km 146, using an 18 km-long exploration access road that was built in 1986 by Granges and improved by Richfield. This forest service road provides direct access to the Project area and camp location. Driving time from Vanderhoof to the Blackwater Gold Mine is about 2.5 hours.

A new 13.8 km long mine access road will be built to replace the exploration access road, which will be subsequently decommissioned.

Helicopter access is from bases in Vanderhoof, Prince George, or Quesnel.

5.2 Climate

The climate is sub-continental, characterized by brief warm summers and long cold winters resulting from the influence of cold arctic air. The climate is also influenced by moisture-laden weather systems moving east by way of the low Kitimat Ranges.

Hydrometeorological data have been collected at the Project site since early 2011. The data collected through the end of 2020 were correlated with long-term regional data to characterize the prevalent hydrometeorological conditions at the site (Knight Piésold, 2021b). Key findings are as follows:

- Temperatures range from a minimum of approximately -40°C in winter to a maximum of approximately 32°C in summer;
- The long-term mean annual temperature is approximately 2°C, with minimum and maximum mean monthly temperatures estimated to be -7°C in December and 11°C in July, respectively;
- The long-term mean annual precipitation for the site is estimated to be 595 mm with approximately 60% falling as rain and 40% as snow;
- Long-term mean annual actual evapotranspiration is estimated to be in the range of 330–440 mm.

Mining operations will be conducted on a year-round basis.

5.3 Local Resources and Infrastructure

Infrastructure required for the planned mining operations is discussed in Section 18.

The Project area is very sparsely inhabited; the closest Indian Reserve to the mine site is Tatelkus Lake 28, approximately 15 km away, and three ranches are found within a 20 km radius of the Blackwater Gold Mine.

Some services are available in Vanderhoof, but Prince George is the regional hub with air service from major centres.

The mine will have direct access to BC Hydro's power grid through the approximately 140 km transmission line currently being built by Artemis Gold.

Personnel to support mine development and operations can be drawn from British Columbia's well-developed mining industry.

The deposit is located on the north slope of Mt. Davidson, and the proposed infrastructure including the mill, waste, and ore stockpiles, and TSF will be predominantly sited in the Davidson Creek watershed.

5.4 Physiography

5.4.1 Elevations

The elevations within the Project area range from just over 1,000 m (above sea level) in low-lying areas northeast of the proposed mine site to 1,800 m on the southwest side of the Project area at the summit of Mt. Davidson, which is the highest peak in the Fawnie Range. The Blackwater deposit is located on the northern flanks of Mount Davidson.

5.4.2 Outcrops

The geomorphic landforms of the region are largely the product of the last glaciation period (the Late Wisconsinan Fraser Glaciation). The ice sheet extended across the full extent of the site and developed a thick succession of glacial (lodgement till and ablation till) deposits which partially overlay the claims area. Bedrock outcrops are limited, and most of the area is covered by 2 m or greater thicknesses of glacial deposits, except for the upper 150 m of Mt. Davidson and a few localized areas at lower elevations.

5.4.3 Vegetation

The Project area falls within the Fraser Plateau biogeoclimatic region and more specifically within the Nazsko Upland subregion. Low-elevation valley bottoms are dominated by stands of lodgepole pine. Hybrid white spruce tends to dominate on moist to wet sites below 1,500 masl, while subalpine fir and Englemann spruce are dominant above 1,500 masl. Lodgepole pine is a major species on dry, fire-prone sites at most elevations. The pine beetle epidemic infested almost all of the lodgepole pine forests within this subregion.

5.4.4 Waterbodies

The Davidson Creek valley is incised locally and flows northeast from the site toward Chedakuz Creek downstream of Tatelkuz Lake. The Blackwater deposit lies within the upper reaches of the Davidson Creek catchment area. The terrain within this footprint is predominantly gently inclined, except along the incised portions of Davidson Creek. Creek 661 flows northeast from the Project site into Chedakuz Creek upstream of Tatelkuz Lake. Chedakuz Creek drains Tatelkuz Lake before its confluence with Davidson Creek approximately 800 m downstream of the lake. Chedakuz Creek flows northwest passing under a bridge at the Kluskus FSR approximately 2 km downstream from the lake. Chedakuz Creek flows northwest from this point for approximately 25 km to the Nechako Reservoir.

Matthews Creek and Creek 705 both flow west of the Project area and combine with westward flowing Fawnie Creek to form a second predominant surface water flow pattern in the region. Fawnie Creek flows towards Laidman Lake and Johnny Lake, into Entiako Provincial Park, and ultimately forms a portion of the flow of the Entiako River into the Nechako Reservoir.

5.5 Seismicity

The Blackwater Gold Mine is situated within a region of British Columbia (BC) where the level of recorded historical seismic activity has been low. Higher seismicity is associated with the Queen Charlotte–Fairweather fault system located offshore of the west coast of BC and the Alaskan panhandle; however, the level of seismicity in the interior of BC and the Rocky Mountains region drops off rapidly with distance from the west coast and to the north. The seismicity of southwestern BC associated with the Cascadia and Explorer subduction zones has the potential for large magnitude 8 to 9+ earthquakes, but these zones are too distant to make a significant contribution to the seismic hazard at the site.

A seismic hazard assessment was carried out for the Blackwater Gold Mine area in 2021 (Knight Piésold, 2021a) to provide seismic design parameters for the TSF and other facilities. Design ground motion parameters provided by the seismic hazard analysis include peak ground acceleration, spectral acceleration (defining the uniform hazard spectrum), and earthquake magnitude.

5.6 QP Comments on Section 5

The QP notes:

- Mining activities should be capable of being conducted year-round;

- There is sufficient suitable land available for future tailings disposal, mine waste disposal, and related mine infrastructure within the mineral claims and the mining lease;
- Surface rights in relation to the proposed operation are discussed in Section 4.4.

6.0 HISTORY

6.1 Exploration History

The Project ownership history was discussed in Section 4.2.1.

Table 6-1 summarizes the work completed in the Project area to the Report effective date. No production has occurred from the Project area.

Table 6-1: Project History

Operator	Year	Work Completed
Granges	1973	A regional silt survey located anomalous silver, zinc, and lead in the Mt. Davidson area. This was followed by a wide-spaced soil survey northeast of Mt. Davidson.
	1976	Soil sample and ground magnetometer surveys to follow up 1973 soil results.
	1977	Pem claim staked covering most of the presently defined mineral deposit. Pulse electromagnetic (EM) survey on the Pem claim (12.5 km).
	1979	Vector Pulse electromagnetic (EM) survey on the Pem claim (7 km).
	1981	Helicopter EM and magnetometer survey. Horizontal Loop EM survey on the Deb #1 claim. Reconnaissance mapping of the Mt. Davidson area.
	1982	Soil geochemistry (220 samples) and ground magnetometer survey (20.8 line km) on the Pem claim.
	1983	Hammer seismic survey.
	1984	Hand-trenching (30 trenches; 66 m) and VLF survey (4.8 line km) on the Pem claim. Only one trench intersected bedrock.
	1985	Winkie drilling (8 drill holes; 507 m) on the Pem claim. Construction of access road from km 146.5 on the Kluskus Haulage road, east 18 km to the Pem grid.
	1986	Percussion drilling (34 drill holes; 1,524 m) on the Pem claim.
	1987	Core drilling (23 drill holes; 2,617 m) on the Pem claim.
	1992	Line cutting (58.8 km), soil samples (955), stream silt samples (35), geological mapping (6,000 ha), geophysical surveys (50km induced polarization (IP), magnetics, very low frequency (VLF)), core drilling (5 drill holes; 785 m).
	1994	Line Cutting (48.2 km), rock samples (29), soil samples (1,598), silt samples (23), lake sediment samples (4) Dighem airborne geophysical survey (881 line km of EM, magnetics, radiometrics), Dave claim IP survey (20 km), core drilling (5 drill holes; 761.68 m).
Kennecott Canada	1997	Line cutting (4 km) and IP survey; Dave claim.
Silver Quest	2005-2006	Core drilling (7 drill holes; 1,333 m)
	2007	Soil samples (335)

Operator	Year	Work Completed
Richfield	2009–2011	Core drilling (134 drill holes; 44,684 m).
	2011	Initial Mineral Resource estimate and two subsequent resource estimate updates.
New Gold	2011	Core drilling (125 drill holes; 49,316 m), metallurgical test holes (7 drill holes; 2,282 m).
	2012	Core drilling (716 holes totalling 207,333 m), geotechnical holes (13 totalling 5,003 m), metallurgical test holes (20 totalling 1,816 m), waste rock characterization (14 holes totalling 2,952 m) hydrological monitoring or pilot holes (7 holes totalling 2,265 m). Three resource estimate updates. Completion of preliminary economic assessment.
	2013	Resource estimate update. Draft AIR submitted. Feasibility Study (2013 FS); estimation of Mineral Reserves based on 2013 resource estimate. Core drilling (13 holes totalling 7,521 m).
	2014	Final Application Information Requirements (AIR) submitted.
	2014–2019	Permitting activities, acquisition of additional mineral claims, additional metallurgical testwork. Assessment of the approved Application for an EA Certificate was conducted by the BC Environmental Assessment Office from 12 January, 2016 to 17 May, 2019. EA Certificate # M19-01, which included the Certified Project Description and a Table of Conditions, issued on 21 June, 2019. CEAA commenced the environmental assessment on 21 December, 2012, and the Decision Statement was issued on 15 April, 2019.
Artemis Gold	2020	Acquires Project. Completes pre-feasibility study, estimation of Mineral Resources and Mineral Reserves. Commenced pre-production grade control program of 561 RC holes and 33,216 m, completed in March 2021.
	2021	Completes feasibility study, estimation of Mineral Resources and Mineral Reserves
	2023	Major Mines Act Permit received and Fisheries Act authorization. Amendment to Schedule 2. Start of mine construction.

Note: EA = environmental assessment ; CEAA = Canadian Environmental Assessment Agency.

6.2 Production

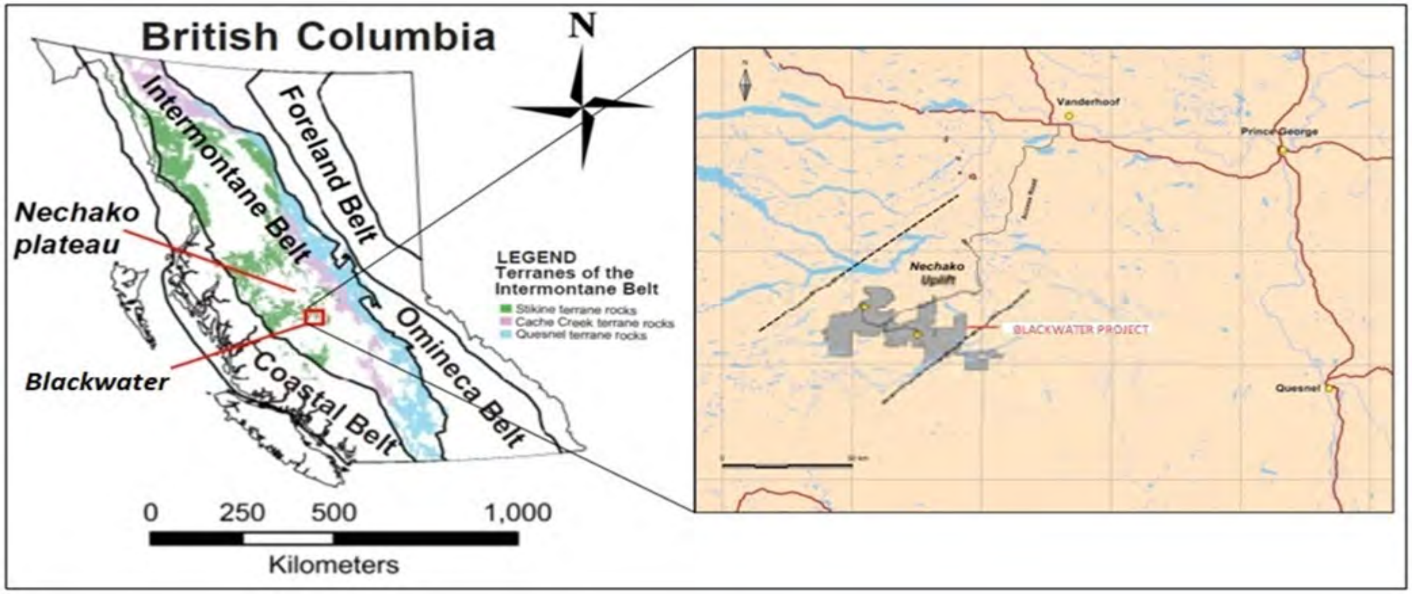
There is no known modern production from within the Project area.

7.0 GEOLOGICAL SETTING AND MINERALIZATION

7.1 Regional Geology

The Project is located on the Nechako Plateau near the geographic centre of British Columbia. The plateau is part of the Intermontane Belt superterrane situated between the Coast Belt to the west and the Omineca Belt to the east (Figure 7-1).

Figure 7-1: Regional Geology Map and Project Tectonic Setting



Note: Figure prepared by New Gold, 2014.

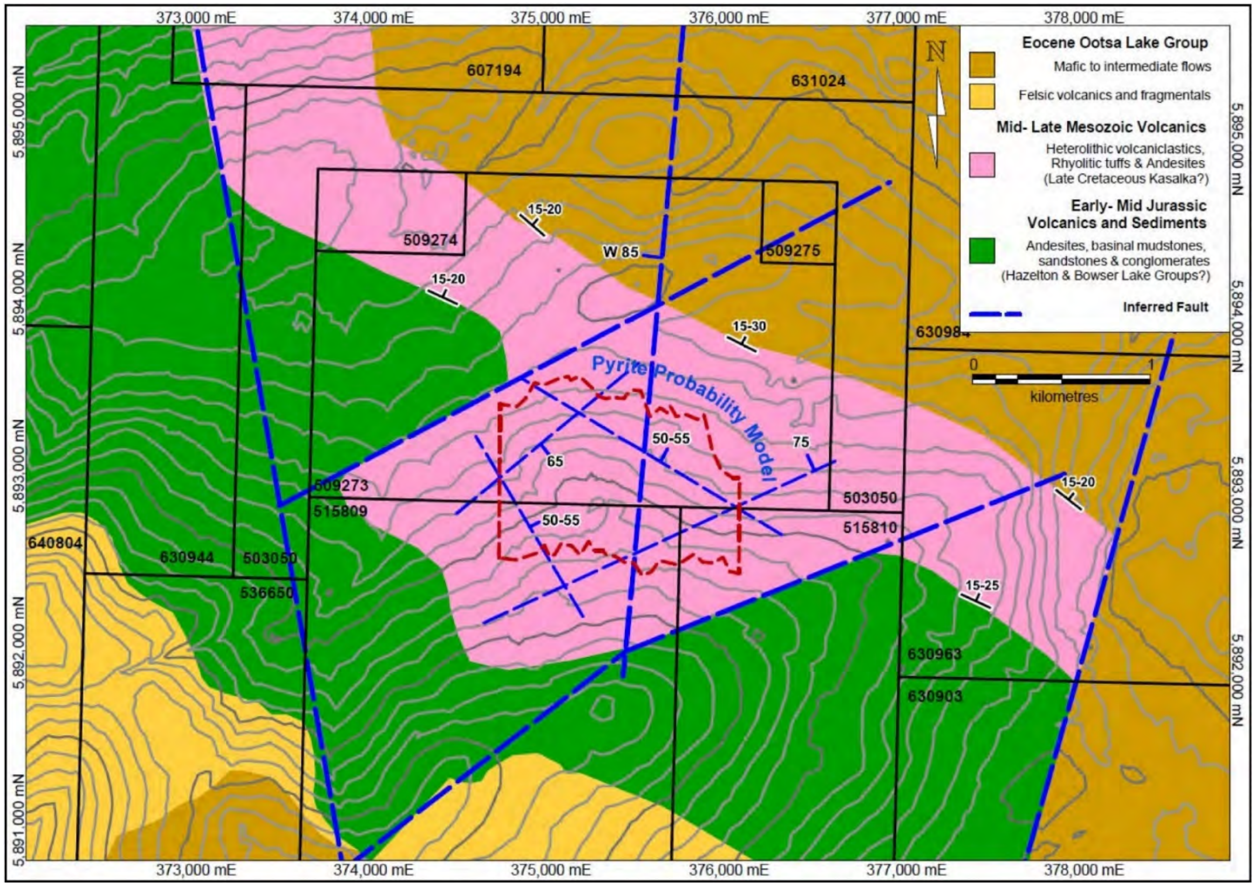
Topographic relief for the plateau is moderate with elevations ranging from 1,000 to 1,800 m above sea level. The Intermontane Belt consists of an assemblage of three accreted tectonostratigraphic terranes: Stikine, Cache Creek, and Quesnel (Riddell, 2011). The Project area is underlain by rocks of the Stikine terrane, comprising an assemblage of magmatic arc and related sedimentary rocks that span from the Jurassic to the early Tertiary. These rocks were exposed within an easterly-trending structural high termed the Nechako uplift.

The Nechako uplift is bounded to the north and south by the northeast-striking Natalkuz and Blackwater faults, respectively (Diakow and Levson, 1997; Diakow et al., 1997). The latest extensional displacement along these faults juxtaposes older Mesozoic and Tertiary rocks in the central part of the uplift against younger Cretaceous and Tertiary volcanic rocks to the north and south (Diakow et al., 1994; Diakow and Levson, 1997; Friedman et al., 2001). Though the Natalkuz and Blackwater faults are poorly defined due to scarce bedrock exposures, a feature characteristic of the Nechako Plateau in general, strong linear trends marking the traces of these structures are evident in the available gravity and airborne magnetics data for the region. The eastern and western limits of the uplift are not clearly defined by current geologic mapping coverage. The northwesterly-trending Chedakuz fault and adjacent Nechako range transect the uplift and mark the eastern limit of the Project area. To the west the Nechako uplift extends into a provincial park that is well beyond the area currently being explored.

7.2 Project Geology

Quaternary glacial overburden, colluvial, and fluvial deposits mask the majority of bedrock within the Project area. Project geology is based on interpretations derived from observations and interpretation of geological field mapping in conjunction with core and reverse circulation drilling data collected between 2009 and 2013. Figure 7-2 is a sketch map of the top-of-bedrock geology for the proposed open pit area. The red dashed line shown in Figure 7-2 delineates the outer limits of the pyrite probability shell.

Figure 7-2: Geology Map, Blackwater Deposit Area



Note: Figure prepared by New Gold, 2014.

The Project is underlain by a sequence of volcanic units consisting of heterolithic breccias, rhyolitic tuff, and andesite. The local volcanic section is further subdivided as follows: a lower sequence of andesite, felsic volcaniclastic rock, heterolithic breccias, and tuff, which host the Blackwater deposit, and an upper sequence of post-mineral Eocene-age felsic volcanic and fragmental rocks and mafic to intermediate flows belonging to the Ootsa Lake Group.

The felsic volcaniclastic rocks and tuff of the lower sequence are late Cretaceous in age based on U-Pb geochronologic dating of zircons which yielded ages ranging from 72.4 ± 1.0 Ma to 74.1 ± 2.2 Ma (Mortensen, 2011). The adjacent andesites were interpreted to conformably underlie the felsic volcaniclastic rocks and thus belong to the late Cretaceous Kasalka Group. Additional work is required to fully constrain the age of the andesite in the lower volcanic sequence.

Together the lower and upper volcanic sequences comprise a gently northeasterly-dipping section underlain by basinal mudstones, fine sandstones, and conglomerates interpreted as belonging to the late Jurassic Bowser Lake Group. These units are cross-cut by well-developed systems of northeast-, northwest-, and northerly-striking faults that define a polygonal structural fracture pattern at all scales.

Outcrops of massive felsic lapilli tuff assigned to the Ootsa Lake Group are found along the uppermost elevations of Mt. Davidson to the south of the Blackwater deposit. The Ootsa rocks comprise felsic and andesitic units that are distinguished from those hosting the Blackwater deposit by their darker gray colour, larger lithic clasts, plagioclase phyric content and the presence of fresh, black, stubby euhedral doubly-terminated quartz crystals up to 1 mm across, which commonly make up a few percent of the rock.

7.3 Deposit Geology

7.3.1 Lithologies

Given the lack of outcrop, geological interpretation has been based primarily on drill information plotted on section and plan views. The current Blackwater geological model is based on three principal components: lithology and structure, alteration, and mineralization. The lithological and structural component includes andesite, volcanic fragmental, and laminated volcanic rocks.

Host rocks within the Blackwater deposit area are pervasively hydrofractured, pyritized, and altered to a mixture of silica and sericite. Locally the amount of silica introduced through hydrofracturing and silicification may affect 25% or more of the total volume of altered host rocks.

At the deposit scale, brittle-style tectonic deformation affects all rock units. Interpretation and correlation of clearly recognizable faults are made difficult by the intense hydrofracturing and multiple fault sets. Instead, extensive zones of broken rocks cross-cut the mineralized zone and grade laterally into unbroken rock with no obvious bounding fault surfaces.

Within the Blackwater deposit and surrounding area, the Kasalka volcanic units commonly contain dark reddish–brown garnet crystal fragments up to a centimetre in diameter as an accessory in the heterolithic breccias, locally making up 1% to 2% of the rock. X-ray fluorescence (XRF) data on the garnets indicate they are manganese-rich spessartine.

The lithological codes used in the Blackwater drill hole database were defined according to observed descriptive criteria only. The codes do not include assignment of individual rock units to formally defined regional stratigraphic units. The lithological codes are summarized in Table 7-1.

Table 7-1: Drill Database Lithological Codes

Code	Description
OB	Overburden
AND	Andesite
FT	Felsic tuff
FLPT	Felsic lapilli tuff
VC	Volcaniclastic
EC	Epiclastic
SED	Argillite, sandstone, and conglomerate

7.3.2 Structure

Mineralization is strongly controlled by northwest–southeast-trending structures characterized by zones of tectonic brecciation and chloritic gouge. Northeast-trending structural discontinuities also appear to have a major control on alteration and mineralization, but do not appear to be affected by recent movement. A set of east–northeast-trending graben-forming faults bound the mineralization and fragmental package to the southeast.

A major north–south trending fault dissects the deposit along UTM easting 375,600E, and east–northeast-trending faults were also noted. The major fault represents a well-defined disruption in lithology, alteration, and mineralization pattern and was used to subdivide the block model, as described in Section 14 into two structural domains, one to the east of it and one to the west.

7.3.3 Alteration

The alteration minerals most commonly identified included muscovite, high and low-temperature illite, ammonium bearing illite, smectite, silica, biotite, and chlorite. Six alteration assemblages were defined as shown in Table 7-2.

Table 7-2: Alteration Assemblages

Type	Description
Potassic hornfels	Biotite ± K-feldspar “flooding” or replacement by biotite and/or garnet with pyrrhotite ± actinolite alkali feldspar (albite, orthoclase).
Sericite–chlorite	Illite, Fe-chlorite ± interlayered illite–smectite, carbonate (commonly siderite).
Quartz–sericite	Fine-grained, sugary, greenish–grey to buff-coloured quartz, muscovite, or highly crystalline illite + pyrite, black sphalerite, dendritic black sulphide (DBS) and lesser pyrrhotite, rare tourmaline.
Silica–sericite	Silica, illite ± pyrite, red sphalerite, pyrrhotite.
Massive silica	Grey, glassy, massive, finely-crystalline silica, sulphide-destructive.
Ammonium	Ammonium-bearing micas and rare buddingtonite (NH_4-bearing feldspar).

Those six alteration assemblages were subsequently consolidated into three principal categories: ammonium-bearing illite overprint, texture-destructive quartz–mica ‘sericitic’, and potassic.

The alteration model indicates the presence of two centres of texture-destructive sericitic alteration cored by the ammonium-bearing overprint and haloed by early potassic alteration and hornfelsed andesite.

7.3.4 Mineralization

Core drilling has defined a zone of continuous gold mineralization that extends at least 1,300 m along its longest dimension east-west and at least 950 m north–south. The vertical thickness of the zone ranges up to 600 m, remaining open at depth in the southwestern part of the deposit, as well as to the northwest and west. The centre of the deposit has an average thickness of 350 m and, where open, a vertical extension of up to 600 m.

The mineralized zone plunges shallowly to the north and northwest with inferred steep, north-plunging higher-grade mineralized shoots, measuring tens of metres thick, likely influenced by near-vertical structural intersections.

Mineralized rocks within the main Blackwater resource area can be broadly divided into a thick succession of felsic to intermediate pyroclastic and volcaniclastic rocks, volcanic flows and breccias, and related volcanic and lithic-derived sedimentary units (fine to coarse epiclastic rocks). Whole-rock analysis indicates that these units range from rhyolite to dacite to andesite in composition. Detailed age relationships between the mineralized host rocks at Blackwater are not entirely understood, but the vertical succession and locally observed progressive inter-bedding of these units suggest the andesite to be oldest, followed by the felsic tuffs and subsequently the felsic volcaniclastic rocks.

In general, all rocks in the Blackwater deposit area are mineralized, with trace pyrite–pyrrhotite–sphalerite in outboard andesite flows and volcaniclastics, or as gold-bearing polymetallic sulphide mineralization within the fragmental felsic unit of the deposit. The only exceptions are Eocene (?) dacite porphyry dykes intersected along the southern and northwestern part of the drilling grid, and amygdaloid mafic intermediate flows in the northern part of the grid, possibly related to the Eocene Ootsa Group.

Gold–silver mineralization is associated with a variable assemblage of pyrite–sphalerite–marcasite–pyrrhotite ± chalcopyrite ± galena ± arsenopyrite (± stibnite ± tetrahedrite ± bismuthite).

Sulphide mineralization at Blackwater can be divided into three types, see Table 7-3.

Table 7-3: Sulphide Mineralization Types

Type	Description
Disseminated	Pinhead to coarse blebby sulphide grains and aggregates typically ranging from 1% to 5% total volume of the rock, but locally exceeding this volume. Disseminations may be uniform or irregular, with sulphides displaying an anhedral to euhedral crystal form
	Disseminations of a dark-grey, very fine grained sulphide material (DBS) is common at Blackwater, and may form as fine disseminations to coarse clusters, as thicker coatings to fractures, or as an irregular network of “dendritic” micro cracks within the rock mass
Porosity infill	Sulphides that fill, rim, or replace devitrified pyroclasts, tephra, and juvenile pumiceous material. Sulphides also commonly form parallel to compositional layering and laminations within felsic pyroclastic flows and laminated tuff units. Mineralized amygdules and altered feldspars are also observed in the andesite flow units
Vein	Polymetallic, anhedral to euhedral sulphide assemblages in sub-millimetre to centimetre-scale polymetallic veinlets–veins of quartz–sericite–chlorite–clay (illite) ± (iron) carbonate ± tourmaline ± vivianite
	Hydrothermal brecciation and related silicification– consisting of centimetre- to metre-scale zones of hydrothermal brecciation, alteration, and elevated sulphide content. These breccia zones are typically healed with silica-sericite-sulphide cement and cut by a micro stockwork of vitric quartz ± sulphide veinlets
	Structure-related (late?); sulphides crushed to comminuted in brittle fault breccia and gouge

Hydrothermal alteration (and possibly contact metamorphism) has produced several superimposed alteration assemblages, including pervasive silica–sericite–clay (illite) ± biotite alteration and veinlet/fracture-controlled silica–sericite–chlorite–clay ± iron carbonate ± tourmaline. An early (?) biotite–silica–albite ± chlorite/actinolite hornfelsing event may have been significant, although mineralization in these rocks appears to be lower than in units without evident hornfelsing. Visible native gold has been noted in some drill holes.

Five types of secondary quartz were identified:

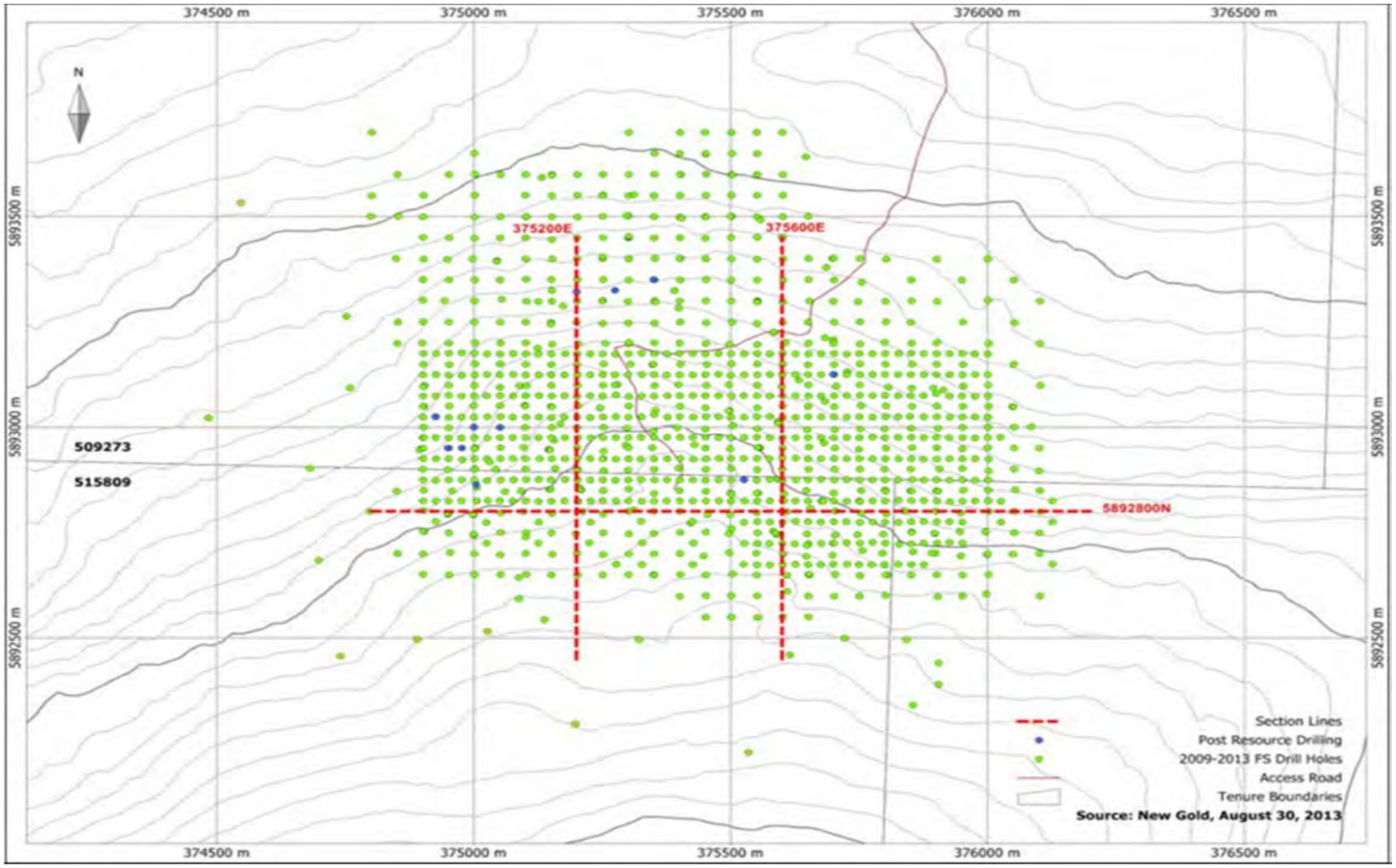
- Pervasive, amorphous to translucent silicification with associated illite ± sericite;
- Intense silicification of felsic units, epiclastic rocks, and more intermediate volcaniclastic rocks with biotite alteration of the matrix (hornfels) identified in drill holes;

- Cryptocrystalline silica replacements in felsic ash-tuff layering;
- Silica cement/matrix to local hydrothermal brecciation;
- Sub-millimetre vitric quartz veinlets in zones of intense silicification; commonly as a micro-stockwork.

Gold and silver mineralization is hosted predominantly within a central core of felsic tuffs and volcaniclastic breccias that are enveloped by a sequence of massive and more- cohesive andesitic flows and tuffs. The deposit is roughly rhombohedral in plan, bounded by near vertical northwest- and northeast trending faults. The fragmental package is funnel-shaped, elongated to the west–northwest–east–southeast, and open to the southwest at depth. The alteration component indicates the presence of two centres of texture-destructive sericitic alteration cored by an ammonium-bearing overprint and haloed by early potassic alteration and hornfelsed andesite. The mineralization component has been built through a combined “Pyrite + DBS” simulation, which identified the pyritic mineralization domain and independently confirmed the presence of key faults seen in the lithological and structural model.

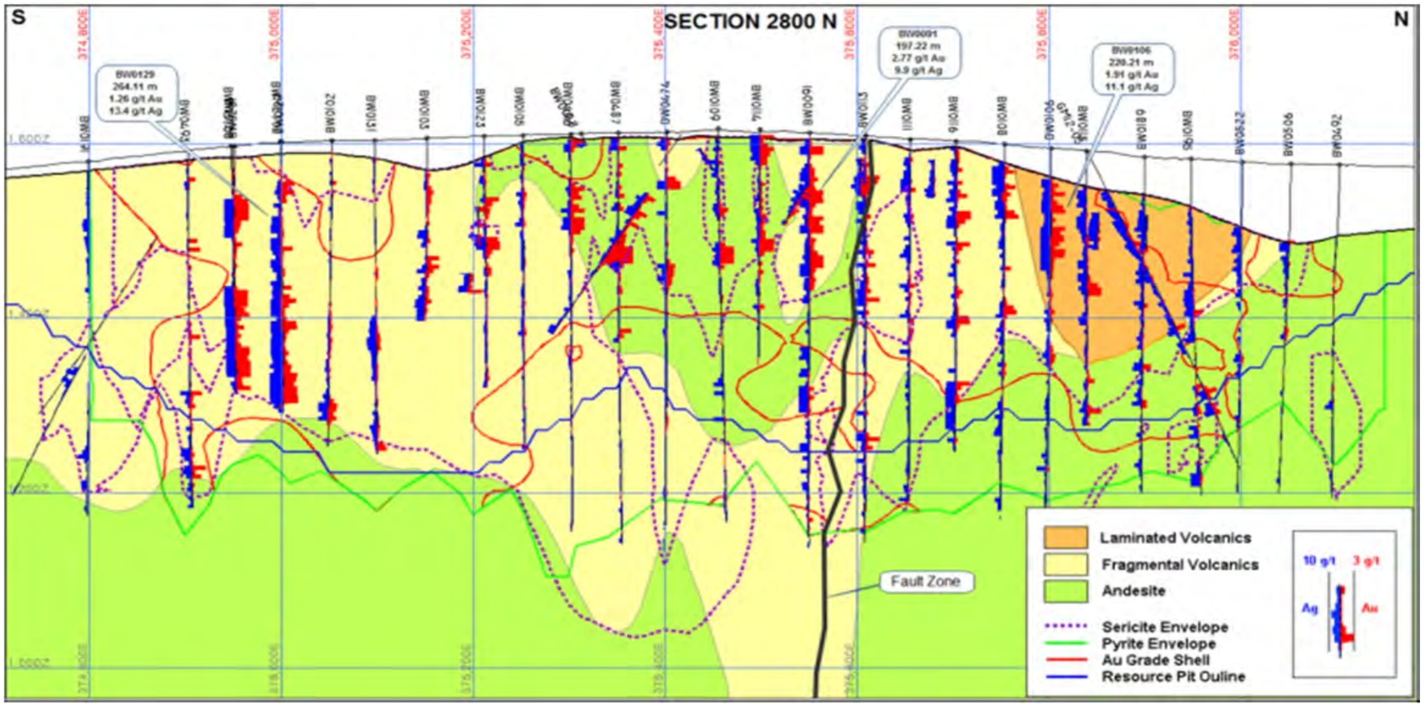
Figure 7-3 is a control plan for the drill sections included as Figure 7-4, Figure 7-5, and Figure 7-6. The sections present gold and silver grade as histogram bars representing 5 m down-hole composites. The gold composites are constrained by the pyritic mineralisation domain. Lengths and average grades of some representative intervals are shown on the sections.

Figure 7-3: Drill Hole Collar Map Showing Location of Referenced Cross-Sections



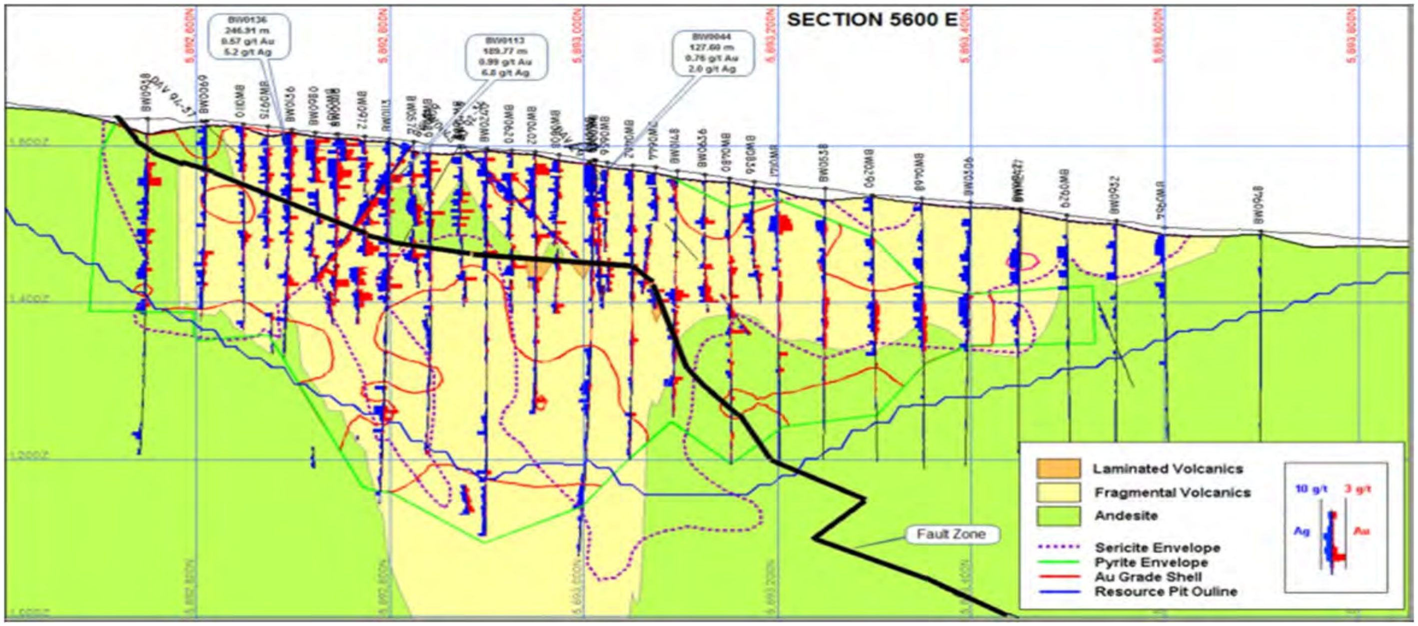
Note: Figure prepared by New Gold, 2014.

Figure 7-4: Cross-Section 2800 N



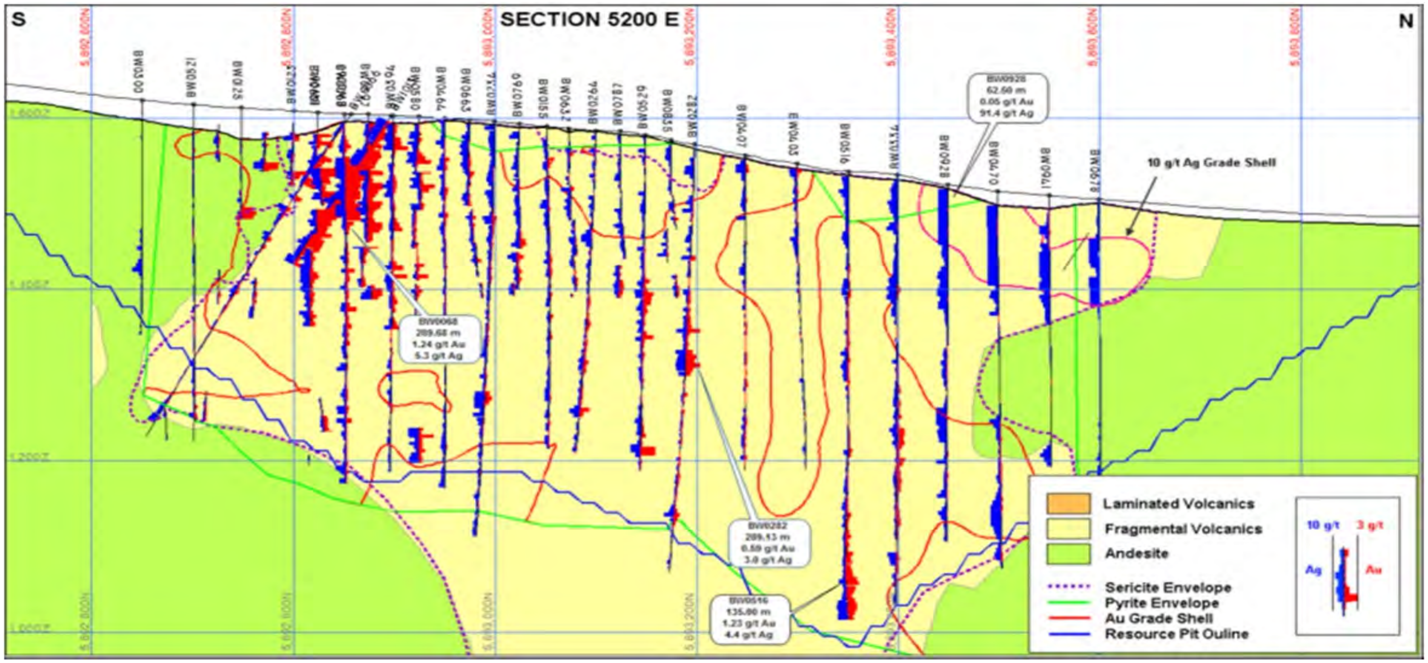
Note: Figure prepared by New Gold, 2014.

Figure 7-5: Cross-Section 5600 E



Note: Figure prepared by New Gold, 2014.

Figure 7-6: Cross-Section 5200 E



Note: Figure prepared by New Gold, 2014.

8.0 DEPOSIT TYPES

8.1 Overview

The Blackwater deposit is considered an example of a volcanic-hosted, epithermal-style gold–silver deposit.

Pervasive stockwork-veined and disseminated sulphide mineralization at Blackwater is hosted within felsic to intermediate volcanic rocks that have undergone extensive silicification and hydrofracturing.

The geological setting, style of gold-silver mineralization, and associated alteration assemblage for the Blackwater deposit share the characteristics of both low and intermediate sulphidation epithermal deposit types, according to the classification system of Sillitoe and Hedenquist (2003).

Gold–silver mineralization is associated with a variable assemblage of pyrite-sphalerite-marcasite-pyrrhotite ± chalcopyrite ± galena ± arsenopyrite (± stibnite ± tetrahedrite ± bismuthite). Sulphide and gangue mineralogy are reasonably characteristic of an intermediate sulphidation regime as defined by Sillitoe and Hedenquist (2003). However, the massive fine-grained silicification present at Blackwater is more typical of high-sulphidation deposits and minor carbonate gangue of a low-sulphidation environment.

8.2 QP Comments on Section 8

The QP considers that exploration programs that use an epithermal deposit model are appropriate to the Project area.

9.0 EXPLORATION

9.1 Introduction

Artemis Gold has performed no exploration activities since acquiring the Project in August 2020.

The following is a summary of the exploration carried out by New Gold as summarized from New Gold (2014).

9.2 Grids and Surveys

Eagle Mapping Ltd. (Eagle Mapping) generated detailed topography in August 2010 from an aerial survey flown on 7 July of the same year. Topography was generated at 2 m contour intervals over an area of 5 km^2 and at 5 m contours over an area of 56 km^2.

Eagle Mapping performed an aerial light detection and ranging (LiDAR) survey of the Project area from 8–9 August, 2011. Although the area of interest (AOI) for this survey was 412 km^2 in size, the survey actually covered approximately 500 km^2 to buffer the true AOI for quality assurance purposes.

The LiDAR topographic data were collected using a Riegl VQ-480 laser scanner and airborne global positioning system (GPS)/inertial measurement unit (IMU). Data were collected in one to two pulses/m^2 with a ±0.25 m vertical accuracy and ±0.35 m horizontal accuracy based on ground control points.

9.3 Geological Mapping

Given the lack of bedrock exposures in the immediate Blackwater deposit area, geological information has been obtained primarily by exploration drilling.

In 1992, Granges carried out 1:10,000 scale geologic mapping to the north of the deposit and, in an earlier 1984 program, excavated a total of 30 hand trenches. Only one trench in the northwestern part of the current Mineral Resource area reached bedrock and returned an anomalous silver grade from a grab sample. Mapping of trenches and road-cut exposures over the deposit confirmed the geological interpretations in the subsurface.

9.4 Geochemical Sampling

Soil and stream geochemical surveys were carried out over parts of the Project area between late May and mid-September 2012. The purpose of the geochemical surveys was two-fold: to conduct a soil orientation survey over the known Blackwater deposit; and to investigate the potential for additional areas of mineralization in the Blackwater area by testing surface soils and silts in streams draining regions of higher relief.

The soil samples were collected at 100 m stations along grid lines spaced 300 m apart. The results of the soil survey indicated numerous areas displaying multi-element anomalies including gold, zinc, silver, copper, bismuth, and molybdenum, many of which merit follow-up investigation.

A total of 43 stream sediment samples were collected in 2012 in key drainage areas around Blackwater. The samples were sent to SGS Laboratories in Vancouver, BC, for analysis. T he results indicated anomalous copper and zinc values from streams to the northwest and southeast of the Blackwater deposit.

Geochemical data were superceded by drill data in the deposit area.

9.5 Geophysical Surveys

During 2010, Richfield contracted Quantec Geoscience Ltd. (Quantec) of Toronto to conduct a Titan 24 DC resistivity and IP chargeability geophysical survey. The objective of the study was to determine the relationship between IP chargeability and resistivity and zones of known gold mineralization within the mineral resource area to aid in geologic interpretation and drill targeting. The survey was carried out along five 3.5 km long north–south-oriented lines spaced 400 m apart with dipole length of 100 m. In October 2011, Quantec carried out a second-phase survey, consisting of eleven 2 km north–south-oriented lines with a dipole length of 50 m.

The results of the survey indicate good correspondence between known mineralization and the Titan IP-resistivity results. In general, zones of significant gold mineralization positively correlate to zones of moderate resistivity and moderate IP chargeability.

9.6 Petrology, Mineralogy, and Research Studies

Polished section petrographic analysis was conducted on selected drill samples in 2009 and 2010. Sample descriptions were performed by Vancouver Petrographics Ltd.

In 2009, Eco Tech Laboratories performed whole-rock litho-geochemical analyses with the aim of constraining the geochemical fingerprint of the host volcanic rocks by providing insight into the tectonic affinity, geochemical classification, and petrological evolution.

The metallurgical division of Inspectorate Laboratories completed an analysis of a drill composite from drill hole BW0059. Opaque phases identified from X-ray diffraction (XRD) analysis included quartz, micas, orthoclase, clays, and minor calcium sulphates and carbonates. Pyrite, iron oxides (limonite, hematite, magnetite, goethite), and pyrrhotite were the main iron-bearing phases. Mineralization identified included sphalerite, chalcopyrite, cubanite, and traces of tetrahedrite, chalcocite, and dioptase. In some samples, the chalcopyrite and cubanite were observed to be tightly intergrown. Other minerals such as rutile, ilmenite, and traces of graphite were also observed.

A two-phase alteration study was completed to develop the alteration model for the deposit. For the first phase, 20 widely-spaced drill holes were re-logged in detail and analyzed by short-wave infrared (SWIR) spectrometer at approximate 10 m spacings down hole. In the second phase an additional 135 representative holes spaced at about 100 m centres, were selected for re-logging and spectral analysis at a nominal 20 m down hole sample spacing. The alteration minerals most commonly identified included muscovite, high-and low-temperature illite, ammonium-bearing illite, smectite, silica, biotite, and chlorite. Relative proportions of alteration mineral species were quantified by intensity, grouped into alteration assemblages, and plotted on down hole spectral strip logs.

9.7 Exploration Potential

The Project area offers excellent exploration potential as the deposit is open at depth, particularly in the northwest of the deposit where an increasing trend in gold grade is noted.

10.0 DRILLING

10.1 Introduction

A total of 1,053 core drill holes (324,839 m) were drilled in the Project area between 2009 and January 2013. A summary of this drilling is given in Table 10-1. Of this total, 134 drill holes were completed by Richfield, and 919 by New Gold. The drilling of 109 condemnation holes has confirmed no economic mineralization beneath the proposed mine infrastructure. Artemis Gold has not conducted any core drilling since acquiring the Project.

Table 10-1: 2021 FS Drill Hole Summary Table

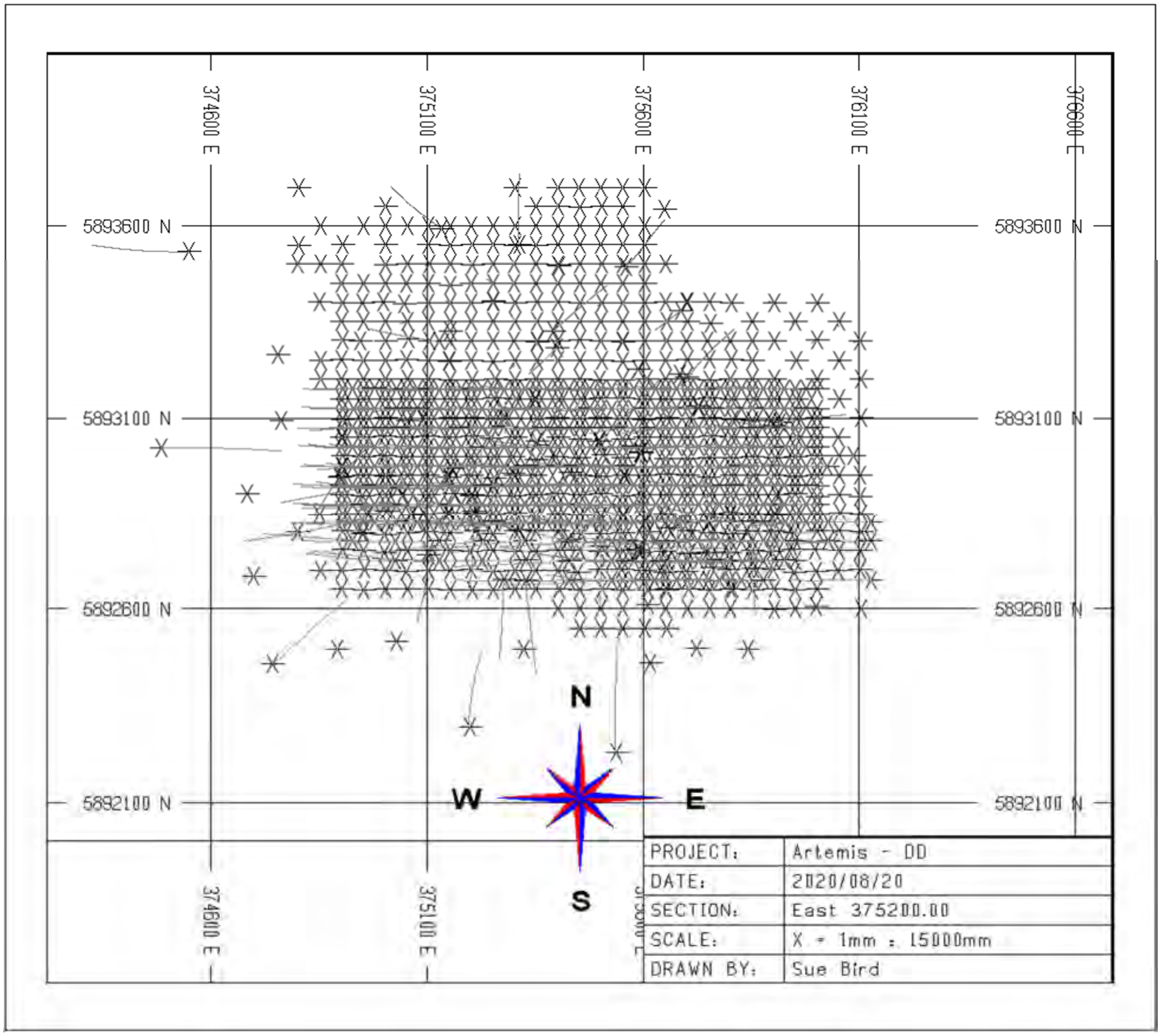
Series	Company	Year	Holes	Total Meters
BW0042 to BW0059	Richfield	2009	18	3,621.23
BW0060 to BW0116	Richfield	2010	57	21,335.92
BW117 to BW0175	Richfield	2011	59	19,727.37
Subtotal	*Richfield*		*134*	*44,684.52*
BW0176 to BW0298, BW0050R	New Gold	2011	125	49,315.78
BWMET01 to BWMET07	New Gold	2011	7	2,281.91
BW0296 to BW1013	New Gold	2012	716	207,333.20
BWMET08 to BWMET27	New Gold	2012	20	1,816.50
BWWR01 to BWWR14	New Gold	2012	14	2,952.50
PH12 series not PH12-2-3 (pilot holes)	New Gold	2012	7	2,265.27
GM12 series	New Gold	2012	13	5003
BW1014 to BW1026	New Gold	2013	13	7520.66
PH13 series and PH12-2-3 (pilot holes)	New Gold	2013	4	1,645.74
Subtotal	*New Gold*		*919*	*280,154.56*
Grand Total			***1,053***	***324,839.08***

An overall drill collar location plan was included as Figure 7-3. Representative cross sections showing drill holes and grades were included in Section 7 (refer to Figure 7-4, Figure 7-5, and Figure 7-6).

Drilling by parties other than Richfield and New Gold, referred to as legacy drilling, is summarized in Table 6-1.

The collar locations in the area of the proposed open pit are shown in Figure 10-1.

Figure 10-1: Project Drill Hole Location Plan



10.2 Drill Methods

The exploration drilling carried out from 2009–2013 consisted predominantly of HQ diameter (63.5 mm) diamond drill core except where a reduction to NQ diameter (47.6 mm) was required to attain target depths. Twenty-three metallurgical holes (BWMET05–BWMET27), and one deep hole (BW0364) were PQ diameter (85 mm) core. Ninety-one reverse circulation (RC) holes were drilled as part of a condemnation program. Contractors and rig types used on the Project for the Richfield and New Gold drill programs are summarized in Table 10-2.

Table 10-2: Drill Contractor and Rig Type Summary Table

Year	Company	Drill Rig Type
2009	Falcon Drilling	F-2000
2010	Falcon Drilling	F-2000
2011	Falcon Drilling Paycore Drilling	F-2000, F-5000, F-6000 TITAN
2012	Falcon Drilling Paycore Drilling Hy-Tech Drilling Boart Longyear	F-2000, F-5000, F-6000 TITAN, Discovery S-F Tech 5000 Ingersol Rand TH100
2013	Paycore Drilling Hy-Tech Drilling	TITAN, Discovery S-F Tech 5000

Drill core was transported from drill to camp by four-wheel drive vehicle for core logging.

10.3 Geological Logging

Drill core was logged in a specially built core handling facility at the Project site. Logging included geotechnical, magnetic susceptibility, and specific gravity (SG) measurements taken at regular intervals. Lithology was logged and the core prepared for systematic sampling at regular 1 m intervals. Core sawing and sampling were the last steps in core handling. Core was cut in half using a diamond blade rock saw, with one half of the sample interval submitted for assay and geochemical analysis and the other half returned to the core box and stored at the Project site for future reference.

Logged data were entered into LogChief tables by Project geologists.

Magnetic susceptibility and conductivity data were measured at 10 cm increments along the core with a hand-held conductivity and magnetic susceptibility meter (GDD MPP-EM2S+Probe) and stored internally for future use.

Recovery and rock quality designation (RQD) data were measured and recorded in LogChief. Recovery and RQD measurements were performed by New Gold geotechnical staff.

The lithological nomenclature at the Project has undergone revision on two occasions to facilitate consistency in logging, geologic interpretation, and ultimately resource modelling. As a result, the following six principal rock lithology types were defined: overburden (OB), felsic tuff (FT), felsic lapilli tuff (FLPT), volcaniclastic (VC), andesite (AND), and sediments (SED).

10.4 Recovery

Core recovery for the 2009, 2010, 2011, and 2012 drilling programs averaged 92%, and the median core recovery was 96%. Poor core recovery often occurred in zones of faulting and fracturing.

10.5 Collar Surveys

Planned drill hole collar locations were measured in the field using hand-held GPS instruments. Locations were subsequently confirmed by Trimble differential GPS. Of the 1,053 holes, 1,037 were then professionally surveyed by All North Consulting using a real time kinematic (RTK) technique to enhance the precision of the location data. Elevations for the drill collars were determined by draping collar coordinates over the topography measured by the LiDAR survey.

10.6 Downhole Surveys

Down-hole surveys were performed using Reflex survey equipment, and dip angle and azimuth are recorded. A +18.8° magnetic declination correction factor is applied to the magnetic azimuth record. Data are entered into LogChief in tables designed specifically for the Project.

10.7 Geotechnical and Hydrogeological Drilling

Geotechnical and hydrogeological site-wide investigation programs were completed by Knight Piésold in 2012, 2013, 2019, 2020–2021, and 2022–2023, and are summarized in Table 10-3.

Table 10-3: Geotechnical Site Investigation Programs

Year	Program
2012, 2013	Drilling of 28 geotechnical drill holes using overburden drilling eccentric (ODEX) drilling methods with standard penetration tests (SPTs) in the surficial materials and HQ3 coring (61.1 mm core diameter) with packer permeability testing in bedrock. Drilling of 68 geotechnical drill holes using sonic drilling methods. Drilling of 16 geomechanical drill holes in the open pit using HQ3 coring methods with core orientation, detailed geomechanical logging, and packer permeability testing. Drilling and installation of two pumping wells to assess hydrogeology in the open pit.

Year	Program
	Installation of 40 standpipe piezometers and five vibrating wire piezometers (VWPs) in select geotechnical drill holes. Installation of 25 VWPs in geomechanical drill holes in the open pit area. Installation of 47 VWPs in observation wells in the open pit area. Installation of 28 monitoring wells for long-term groundwater quality monitoring. Excavation of 305 test pits to investigate near-surface material characteristics and foundation conditions. Completion of response testing in the screened completion zone of 27 standpipe piezometers and monitoring wells. Completion of 35.3 km of seismic refraction surveys. Completion of 5.2 km of high-resolution resistivity and induced polarization lines. Completion of downhole seismic tests (DSTs) in seven geotechnical drill holes. Off-site laboratory index and strength testing of soil grab samples collected from drill holes and test pits. Off-site laboratory strength testing of rock samples collected from geotechnical and geomechanical drill holes.
2019	Drilling of eight geotechnical drill holes using ODEX drilling methods with SPTs in the surficial materials and HQ3 coring with packer permeability testing in bedrock. Drilling of 12 geotechnical drill holes using sonic methods with targeted airlift permeability testing in the surficial materials. Installation of 13 standpipe piezometers (including deep and shallow paired piezometers at one site) and 13 VWPs in select geotechnical drill holes. Installation of closed-bottom polyvinyl chloride (PVC) pipes in five geotechnical drill holes. Excavation of 33 test pits to investigate near-surface material characteristics and foundation conditions. Completion of response testing in the screened completion zone of eight standpipe piezometers. Completion of seismic cone penetration tests (SCPTs) at five locations to investigate the in-situ condition of glaciolacustrine deposits. Off-site laboratory index and strength testing of soil grab and undisturbed samples from drill holes and test pits.
2020–2021	Drilling of 23 geotechnical drill holes using sonic methods plus one additional twinned drill hole for targeted sampling and four additional twinned drill holes for in situ testing. Installation of 12 standpipe piezometers and 30 VWPs. Excavation of 133 test pits to investigate near-surface material characteristics and foundation conditions. Completion of DST in 12 geotechnical drill holes Completion of SCPT at three locations with pore pressure dissipation tests. Completion of pressuremeter tests at two locations. Off-site laboratory index testing of soil grab samples collected from drill holes and test pits.
2022–2023	Drilling of 70 geotechnical drill holes using sonic methods, including paired drill holes at six sites for monitoring wells, plus five additional twinned holes for targeted sampling. Installation of four standpipe piezometers (including deep and shallow paired piezometers at one site) and 40 VWPs. Installation of 18 monitoring wells for long-term groundwater quality monitoring. Installation of two slope inclinometers.

Year	Program
	Installation of closed-bottom PVCs in six geotechnical drill holes. Excavation of 44 test pits to investigate near-surface material characteristics and foundation conditions. Completion of 3.95 km of seismic refraction surveys. Completion of surface wave tests for multi channel analysis of surface waves at 11 locations in the plant site area. Completion of DSTs in 22 geotechnical drill holes. Completion of response testing in the screened completion zone of 12 standpipe piezometers and monitoring wells. Off-site laboratory index and strength testing of soil grab and undisturbed samples collected from drill holes and test pits. Off-site laboratory isotope analysis of soil grab samples collected from drill holes for groundwater source tracing.

The geological, geotechnical, and hydrogeological site data collected during the 2012–2013 and 2019–2021 periods was analyzed and compiled into a stand-alone Dam Site Characterization Report (Knight Piésold, 2021c). That report combined the site data with information available from other sources and presented the site geological model and site wide geotechnical and hydrogeological conditions. The interpreted foundation conditions at the TSF, water management structures, and stockpiles areas based on the data collected to 2021 were presented in that report and separate reports. The data used to develop the Dam Site Characterization Report were augmented with site data collected during a late 2021 test pitting program and the 2022–2023 site investigation programs. The collection of site data to date in proximity of natural water courses has been limited due to riparian zone buffer requirements (minimum 30 m from the wetted perimeter). Individual site characterization update letters are being prepared as needed for specific infrastructure to update the previous characterization and foundation condition interpretations considering the additional data.

Additional work is outlined in the three-year site investigation plan (Knight Piésold, 2024a) to progressively refine the site geological model, verify assumed foundation conditions, and collect supplemental information to support the detailed design of subsequent stages of the TSF. The recommended work program includes geotechnical drilling and in situ testing at locations within the proposed infrastructure footprints. Select locations were identified to investigate inferred buried glaciofluvial channels, continuous glaciolacustrine units, and clay-rich completely weathered bedrock horizons.

It is recommended the site investigation plan be updated for the earlier construction of Main Dam D if/when the decision is made to proceed with this expansion plan.

10.8 Metallurgical Drilling

Twenty-seven specific metallurgical holes were drilled, four of which were HQ-size holes, (BWMET01–04), and 23 were PQ-size holes, (BWMET05–27). The total length of these metallurgical holes was 4,098 m.

10.9 Waste Rock Characterisation Drilling

Fourteen specific HQ-size waste rock characterisation holes were drilled, (BWWR01–14). The total length of these holes was 2,952.5 m.

10.10 Condemnation Drilling

Eighteen diamond drill holes (HQ; 7,036.53 m) and 91 reverse circulation (RC; 33,252 m) holes were drilled to condemn potential site facility areas surrounding the Blackwater deposit.

10.11 Drilling Supporting Mineral Resource Estimation

Some drill holes were excluded from the assay database because they are either outside of the Blackwater deposit area or were specialty holes as described above. A total of 1,002 core drill holes are included in the resource database used for estimation purposes as shown in Table 10-4.

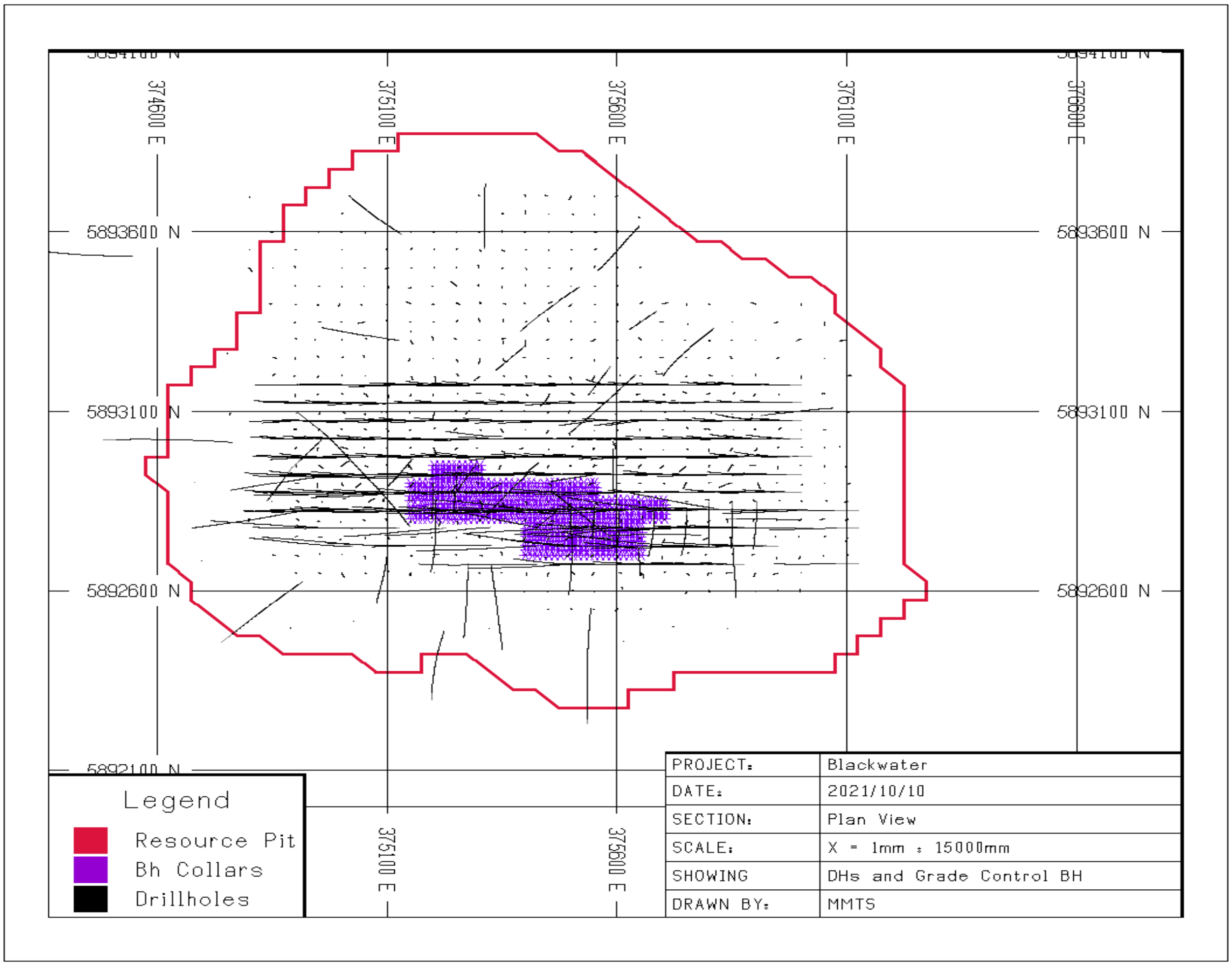
Table 10-4: Resource Database Used In Estimation

Year	Company	Number of Drill Holes	Metres Assayed	Number of Intervals Assayed
2009	Richfield	18	3,413.8	3,450
2010	Richfield	56	20,048.1	20,172
2011	Richfield	59	18,840.5	18,484
2011	New Gold	125	46,231.2	46,008
2012	New Gold	743	203,416.4	200,211
2013	New Gold	1	414	413
Total		***1,002***	***292,364***	***288,738***

10.12 Pre–Production Grade Control Drilling

Artemis Gold conducted a 561 hole, 33,216 m pre-production grade control program between November 2020 and March 2021. Drill hole depths ranged from 45–72 m. The program targeted a zone within the first 60 m from surface to improve the understanding of potential ore variability during production start-up. Information from the program is being used to confirm grade selectivity, investigate orebody continuity , optimize drill-and-blast designs and optimizing the sequence of ore feed to the plant during ramp-up and initial operations. The drill hole data were not incorporated in the resource estimate, as the drill holes were assayed only, not geologically logged. Figure 10-2 shows the location of the grade control RC holes.

Figure 10-2: Plan View of Grade Controls drill holes and drill holes Supporting the Resource Estimate



Note: BH = bore hole = grade control RC drill holes.

The grade control program illustrated conservatism in the modelled gold tonnage and grades. The potential borehole bias was accounted for in this analysis.

10.13 Comments on Section 10

In the QP's opinion the quantity and quality of the lithological, geotechnical, collar, and down- hole survey data collected in the exploration and infill drill programs from 2009 to 2013 are sufficient to support Mineral Resource and Mineral Reserve estimation. There are no known sampling or recovery factors with these programs that could materially impact the accuracy and reliability of the results.

11.0 SAMPLE PREPARATION, ANALYSES, AND SECURITY

The sample preparation, security, and analytical procedures used by the Project owners since 2009 have ensured the validity and integrity of samples taken. The procedures and results have been described in detail in Simpson (2011a, 2011b; 2012), Simpson et al., (2012) and Christie et al., (2014) and are summarized in this section.

New Gold reviewed the control sample results when received from the laboratory (New Gold, 2012a to 2012j). Moose Mountain reviewed the final QA/QC and grade control sample results for the current resource estimate.

Quality control procedures implemented from 2009 through 2013 have been reviewed and it has been determined that the drilling during these years is of sufficient quality to be used in the resource estimate.

Data from holes drilled between 1981 and 1994 have no documented QA/QC information and were not used in resource estimation.

11.1 Sample Methods

11.1.1 Core Sampling

Previous owners or New Gold personnel conducted the drill core handling and sampling. Samples were taken systematically on 1 m long sawn half-core sample intervals, then tagged and bagged. Four sample bags were placed into a larger rice bag labelled with the sample numbers and sealed with a numbered banker's security tag. Between preparation and shipment, a period of up to four days, the rice bags containing the samples were stored in a secure area behind the core cutting area.

The remaining half cores were archived in core sheds in the Project area and personnel drove trucks containing the samples to Prince George. From there the samples were delivered to the laboratories by bonded couriers.

11.1.2 Grade Control Sampling

The pre-production grade control program sampling consisted of RC drill cuttings being collected at a drill mounted Metzke rotating cone splitter by Artemis Gold personnel on 3 m intervals. Samples were placed into bags and further split at the Blackwater sampling facility to ~3 kg sample size using a Jones riffle splitter. Samples were tagged with barcodes, placed in a sample crate and a laboratory dispatch form was completed. Samples were stored in a secure location prior to shipping. Chain-of-custody procedures consisted of filling out sample submittal forms that were sent to the laboratory with sample shipments to make certain that all samples were received by the laboratory.

11.2 Analytical and Test Laboratories

Eco Tech Stewart Group Laboratories (Eco Tech) in Kamloops, BC, and ALS Mineral Laboratories (ALS) in Vancouver, Vanderhoof, Terrace, Reno, and Elko were used for sample preparation. Eco Tech was used as the primary assayer beginning with Richfield exploration in 2009. Assays continued in Kamloops through October 2011, then moved to the ALS laboratory in North Vancouver. External duplicate analysis was performed at the SGS laboratory in Vancouver. All laboratories are accredited and are independent of Artemis Gold.

Pre-production Grade Control sample preparation and analysis were performed by SGS Canada Inc. (SGS), located in Burnaby, British Columbia, Canada (SGS). SGS holds ISO/IEC17025 accreditation for selected sample preparation and analytical techniques and is independent of Artemis Gold.

11.3 Sample Preparation and Analysis

Drill core samples were prepared using standard crush, split, and pulverise sample preparation procedures. Pulverized samples were analysed for gold by fire assay (FA) atomic absorption spectrometry (ASS). Preparation and FA AAS procedures varied between laboratories but were generally similar.

The Eco Tech samples were initially assayed for silver by aqua-regia digestion (AR) and AAS finish, and later by AR and induction-coupled plasma spectrometry atomic emission spectrometry (ICP AES) finish. The ALS samples were analyzed for silver by four acid digestion ICP AES finish until July 2012, after which time silver was analyzed by a four-acid digestion AAS. Eco Tech overlimit results (>30 g/t Ag) were re-assayed by an AR/AAS method. ALS overlimit results (>100 g/t) were re-assayed by a four-acid digestion with AAS finish with a higher detection limit.

Assay procedures also include a multi-element package (28 elements at Eco Tech, 33 elements at ALS) by AR digestion and ICP AES finish. Overlimit analysis was completed on samples returning greater than 1% Cu, Pb, or Zn.

Pre-production grade control sample preparation consisted of drying, crushing and pulverizing to 75% passing 75 µm. Gold and silver analyses were performed using a five-hour 1,000 g LeachWELL method with an inductively coupled plasma mass spectrometry finish (ICP-MS).

11.4 Metallurgical Sampling

Metallurgical samples were selected from the designated metallurgical holes, and samples from numerous resource holes across the deposit. The samples were collected and despatched from site to laboratories under the supervision of the New Gold Exploration Manager. Sample security protocols used were the same as the exploration sample protocols described above.

11.5 Density Determinations

Specify gravity measurements were made the field for more than 32,000 samples using a water immersion method without a wax coating. ALS verified the field measurements by analyzing 154 samples using a water immersion method without a wax coating and 55 samples using a wax-coat water immersion method. The results showed no bias between the field and laboratory methods for all but overburden samples.

11.6 Quality Assurance and Quality Control

QA/QC protocols included “blind” insertion of certified reference materials (CRMs), blanks, field duplicates, and pulp duplicates. The drill hole database was verified by Moose Mountain, who performed an analysis of more than 43,000 QA/QC assays, approximating 15% of the assay database used for the resource estimate. The analysis summarized below shows the data are of sufficient quality for resource estimation and no significant problems were identified.

11.6.1 Standards

The assay QA/QC program involved the insertion of CRMs into the assay stream, which is at industry standard levels of insertion rates. Failed CRMs outside the ± 2 standard deviation (SD) range were routinely identified and the five assays before and after the failed samples were sent for re-assay. Several spot checks verified the replacement of these re-assays in the standards and assay databases. Concerns with CRMs not performing consistently are documented and the change of these materials is noted when appropriate. Overall, 48 different CRMs appear in the standards database of more than 22,000 insertions.

A subset of 16,309 gold assays of CRM insertions was checked to confirm accuracy. These were selected to include the larger instances of a single CRM insertion and include expected values across a wide range of assays.

The results of 29 gold CRMs are presented in Table 11-1. The results show few fails at the ± 3 SD from the expected value due to the previously described diligent identification and re-assay of failed samples. Of these fails, most appear to be likely mislabeled as the value is significantly different from the expected value. It appears in these cases the samples were not rerun or relabeled, but because the instance of them is low it does not present a problem. The mean of the assays compares closely to the expected value, in the cases where the error approaches 5%, the mean is lower than the expected value which is acceptable. The CV of the assays of the CRMs is reasonable and not indicative of any problems.

Table 11-1: Au CRM Checks

CRM (Au)	Samples	High Fail Au	Low Fail Au	Percent Fail	Expected Value Au (g/t)	Sample Average Au (g/t)	Error (%)	StdDev of Au (g/t)	CV (%)
GLG310-3	145	0	1	0.7	0.119	0.121	1.6	0.008	7.0
G911-6	239	0	1	0.4	0.17	0.161	-5.5	0.006	3.7
G303-8	3,067	3	0	0.1	0.26	0.247	-5.1	0.026	10.5
G308-7	245	0	0	0.0	0.27	0.257	-4.9	0.009	3.4
G310-4	3,172	1	0	0.0	0.43	0.414	-4.0	0.015	3.5
CGS-27	156	0	0	0.0	0.432	0.447	3.3	0.020	4.4
GS-P4A	263	0	0	0.0	0.438	0.446	1.9	0.014	3.1
PM449	311	0	0	0.0	0.45	0.452	0.4	0.012	2.7
G311-1	247	0	0	0.0	0.52	0.509	-2.2	0.018	3.6
CGS-22	295	0	1	0.3	0.64	0.641	0.2	0.038	5.9
G310-6	2,191	1	4	0.2	0.65	0.628	-3.6	0.024	3.9
GS-P7B	193	0	0	0.0	0.71	0.724	1.9	0.030	4.1
ME-1	61	0	0	0.0	0.87	0.876	0.7	0.021	2.5
G907-2	1,133	1	0	0.1	0.89	0.876	-1.6	0.066	7.5
PM452	246	0	0	0.0	0.952	0.976	2.4	0.028	2.9
GS-1H	148	0	0	0.0	0.972	0.991	1.9	0.044	4.4
GS-1G	133	0	0	0.0	1.14	1.159	1.7	0.038	3.3
G311-5	625	0	0	0.0	1.32	1.308	-0.9	0.035	2.7
GBMS911-3	654	1	1	0.3	1.33	1.329	-0.1	0.098	7.4
GS-1P5D	222	0	0	0.0	1.47	1.470	0.0	0.056	3.8
PM440	147	0	0	0.0	1.62	1.655	2.1	0.030	1.8
ME-2	523	0	0	0.0	2.1	2.102	0.1	0.053	2.5

CRM (Au)	Samples	High Fail Au	Low Fail Au	Percent Fail	Expected Value Au (g/t)	Sample Average Au (g/t)	Error (%)	StdDev of Au (g/t)	CV (%)
G308-8	328	0	0	0.0	2.45	2.428	-0.9	0.062	2.5
GS-3H	134	0	0	0.0	3.04	3.062	0.7	0.085	2.8
GS-3F	467	0	0	0.0	3.1	3.113	0.4	0.067	2.1
GBMS304-4	230	1	1	0.9	5.67	5.707	0.6	0.239	4.2
G996-7	234	0	0	0.0	5.99	5.950	-0.7	0.221	3.7
GS-7B	457	1	0	0.2	6.42	6.455	0.5	0.108	1.7
GS-11A	43	0	0	0.0	11.21	11.153	-0.5	0.309	2.8

A subset of 5,305 silver CRM insertions was analyzed and the results are presented in Table 11-2. It is observed that there are no failures at the ± 3 SD level for seven of the eight standards analyzed. It is also shown that for most of the CRMs, the mean of the assays is less than the expected values, which although a consistent issue, does not present a risk for resource modeling. In general, the CRM results indicate acceptable accuracy with respect to silver assays.

Table 11-2: Ag CRM Checks

CRM (Ag)	Samples	High Fail Ag	Low Fail Ag	Percent Fail	Expected Value Ag (g/t)	Sample Average Ag (g/t)	Error (%)	StdDev of Ag (g/t)	CV (%)
GBMS911-3	640	0	0	0%	1.70	1.56	-8.8	0.267	17.1
GBMS304-4	273	0	0	0%	3.40	3.17	-7.2	0.396	12.5
GBM910-6	1408	0	0	0%	3.60	3.33	-8.0	0.326	9.8
GBM908-3	1124	0	0	0%	4.80	4.64	-3.3	0.282	6.1
GBM900-3	1062	0	0	0%	7.50	7.37	-1.7	0.556	7.5
CDN Labs GS-P7B	192	0	0	0%	13.40	13.12	-2.1	0.737	5.6
CDN Labs ME-2	544	0	0	0%	14.00	14.08	0.6	0.388	2.8
CDN Labs ME-1	62	0	4	6.5%	39.30	38.50	-2.1	2.295	6.0

11.6.2 Blank Samples

The database of blank samples was reviewed to determine the percentage of assays for each laboratory that exceeded five times the detection limit. These results are shown in Table 11-3 and indicate little problem with contamination.

Table 11-3: Summary of Blank Results

	Eco Tech	ALS
5* DL Au (g/t)	0.15	0.025
% fail Au	0	0.3%
5* DL Ag (g/t)	1.0	2.5
% fail Ag	0.2	0.2
Blank samples	1850	3061

11.6.3 Duplicates

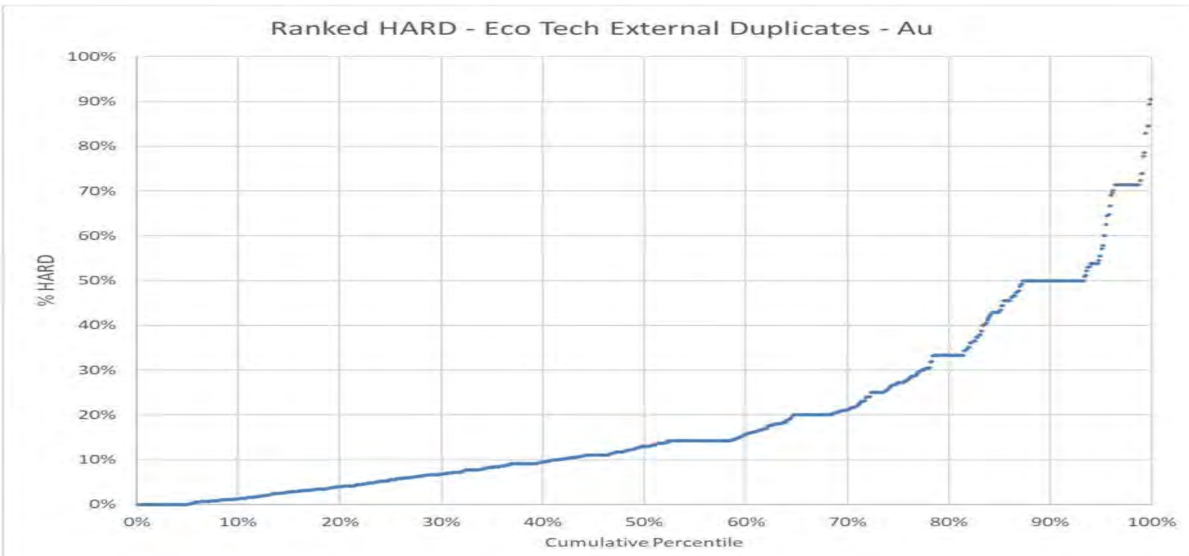
Four types of duplicates were run to assess the precision of the assay analyses; R1 = repeat, D1 = pulp duplicate, D2 = coarse duplicate, and E1 = external check. The insertion rates were 1/10, 1/20, 1/20, and 1/50 respectively. Of most interest are the external checks that are discussed here. The assay database contains approximately 17% assay samples by Eco Tech, the remainder by ALS, and 14% of the external checks were conducted by Eco Tech. Table 11-4 presents a summary of statistics of the external duplicates by laboratory and it is seen that the difference in both means and medians is very low, with SGS always slightly lower.

Table 11-4: Summary of External Gold Duplicate Pairs

Laboratory Comparison	Number of Duplicates	Au Grade	
		Mean (g/t)	Median (g/t)
Eco Tech	845	0.484	0.130
SGS		0.486	0.131
Difference (%)		-0.4	-0.8
ALS	5080	0.345	0.065
SGS		0.349	0.066
Difference (%)		-1.2	-1.5

Ranked half absolute relative difference (HARD) plots are typically used to evaluate duplicate pairs. A ranked HARD plot for the Eco Tech external duplicate gold assay is shown in Figure 11-1. This shows only 40% of the pairs at less than 10% HARD which is not particularly good. In this dataset, the differing laboratory lower detection limits (0.03 g/t Au at Eco Tech and 0.005 g/t Au at SGS) are responsible for the flat portions of the curve and also contribute to differences in assay values, in addition to the "nugget effect" seen in gold mineralization. The Eco Tech data do not appear to be significantly biased.

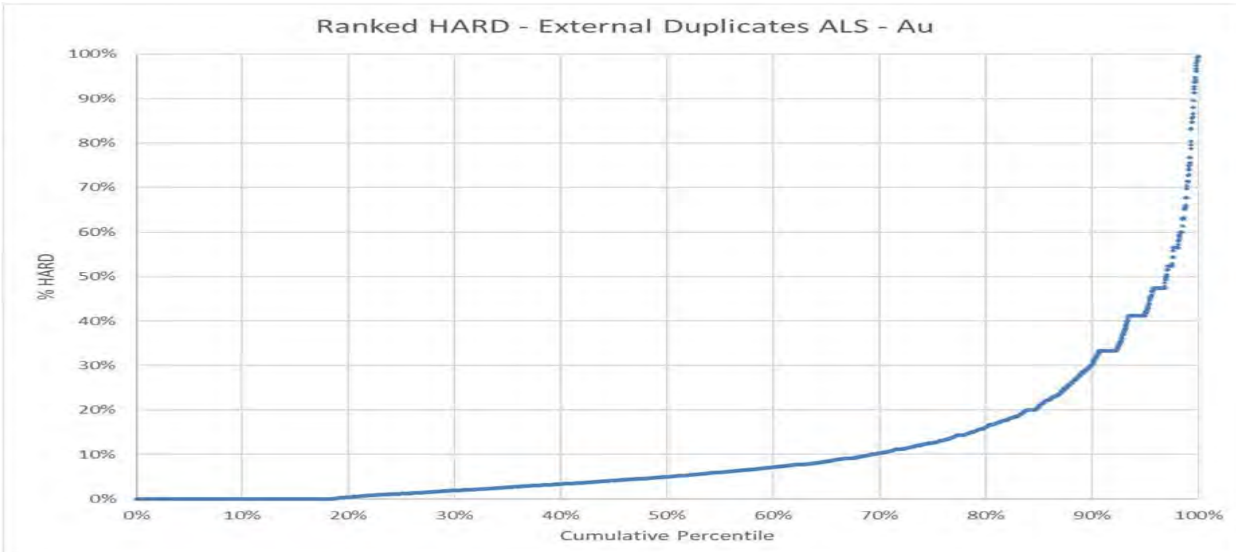
Figure 11-1: Ranked HARD Plot of Eco Tech External Duplicate Pairs – Au



Note: Figure prepared by Moose Mountain, 2020

The ranked HARD plot for the ALS gold data is shown in Figure 11-2. Here, 70% have less than 10% HARD which is reasonable for gold pulps and no significant bias is seen in the data.

Figure 11-2: Ranked HARD Plot of ALS External Duplicate Pairs – Au



Note: Figure prepared by Moose Mountain, 2020

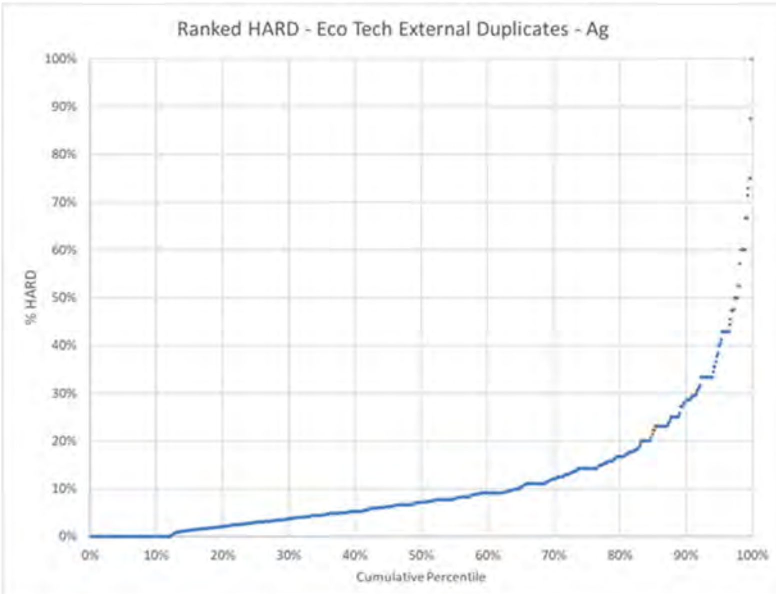
A summary of the external silver duplicate pairs is given in Table 11-5. For both Eco Tech and ALS, the means are slightly higher than the external assays and the medians compare well. The difference is not considered significant with respect to the resource model.

Table 11-5: Summary of External Ag Duplicate Pairs

Laboratory Comparison	Number of Duplicates	Au Grade	
		Mean (g/t)	Median (g/t)
Eco Tech	845	3.92	1.7
SGS		3.88	1.8
Difference (%)		1.0	-5.9
ALS	2938	3.7	1.5
SGS		3.56	1.5
Difference (%)		3.8	0.0

Figure 11-3 shows ranked HARD values for the external duplicate pairs for silver assays done at Eco Tech with pairs at and below detection limit excluded. This indicates that approximately 65% have less than 10% HARD which is not unreasonable.

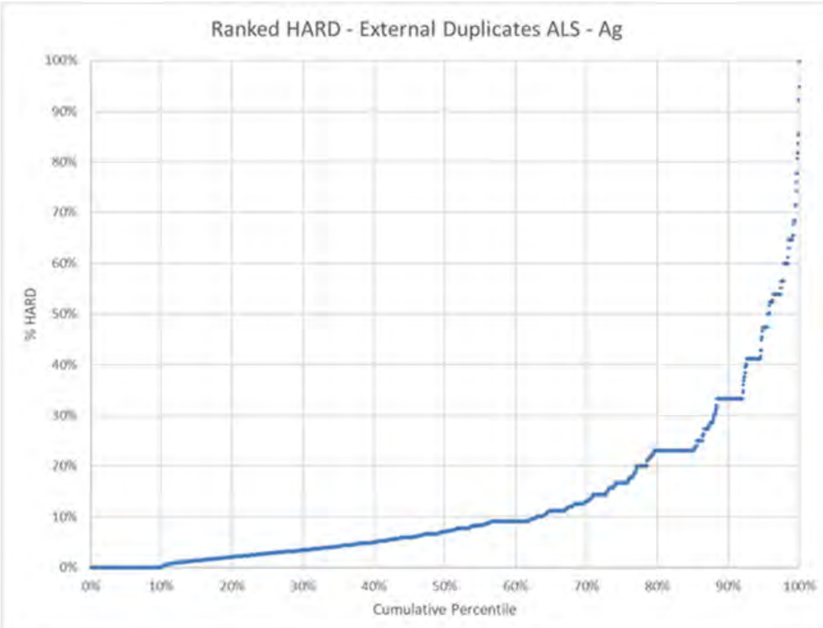
Figure 11-3: Ranked HARD Plot of Eco Tech External Duplicate Pairs – Ag



Note: Figure prepared by Moose Mountain, 2020

Figure 11-4 shows ranked HARD values for silver assays first done at ALS and gives approximately 63% less than 10% HARD, again considered acceptable.

Figure 11-4: Ranked HARD Plot of ALS External Duplicate Pairs – Ag



Note: Figure prepared by Moose Mountain, 2020

11.6.4 Field Duplicates

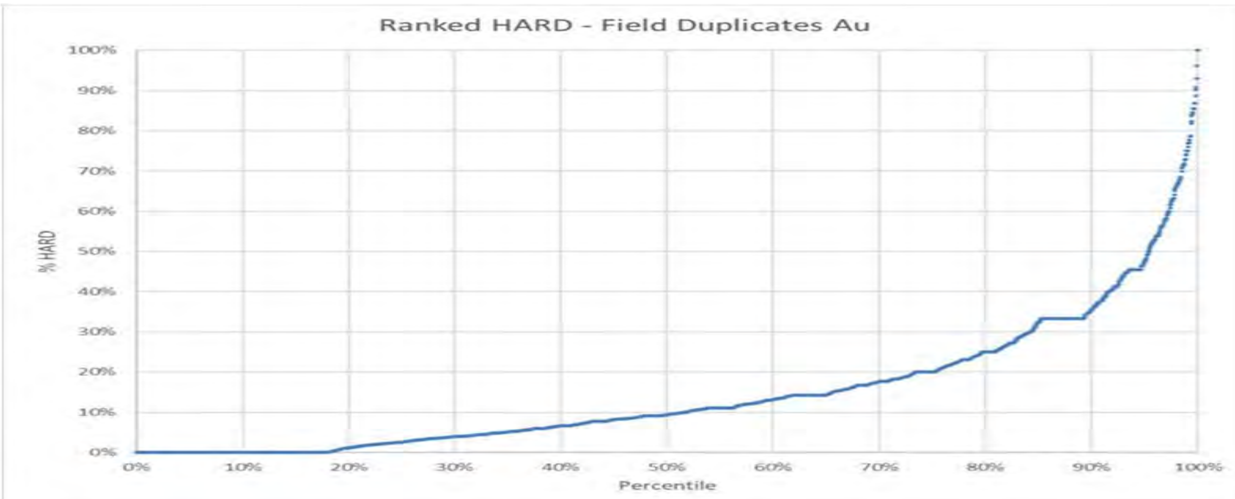
Assay results for 2,482 field duplicate pair results from 2010 to 2011 were analyzed. A summary of statistics is presented in Table 11-6 and shows agreement between means and medians, with the exception being the gold mean. When the nine samples with average gold assays above 10.0 g/t were excluded, the re-calculated means agreed well.

Table 11-6: Summary of Assay Results

Parameter	S1	S2	Difference (%)
Number of samples	2,482		
Au mean (g/t)	0.505	0.568	11.1
Au median (g/t)	0.128	0.128	0.0
Mean Au mean <10.0 g/t	0.412	0.417	1.2
Ag mean (g/t)	4.02	4.14	2.9
Ag median (g/t)	1.6	1.6	0.0

Figure 11-5 shows the ranked HARD plot of gold field duplicates and indicates that only 50% give less than 10% HARD which is not unreasonable given the typical “nugget effect” in gold deposits. Silver pairs showed approximately 57% less than 10% HARD. As of October 2011, quarter-core field duplicates were no longer inserted.

Figure 11-5: Field Duplicates Ranked HARD plot – Au



Note: Figure prepared by Moose Mountain, 2020

11.6.5 Sampling Procedure Optimization

During 2012, check programs were run by New Gold on different stages of the sampling procedure to try to optimize the level of precision achieved at ALS. The programs included drying the original sample for a longer time to remove extra moisture to see if this could improve the homogeneity achieved during milling; pulverizing samples to different particle size specifications to test for any impact on achievable precision; and assaying different sample aliquot sizes. All the programs undertaken confirmed the procedures already in place were the optimum specifications to prepare and analyze Blackwater samples.

11.6.6 Pre-production Grade Control Program

Precision checks for the pre-production grade control program consisted of comparisons to results from split samples using 50 g fire assay methods, comparison of data to existing diamond drill holes and review of SGS laboratory standards. Additionally, LeachWELL tails solids were consistently checked for non-leachable gold and silver using 50 g fire assay for gold and 2 g four-acid digestion with an ICP finish for silver. Artemis Gold randomly inserted blank and duplicate samples into the sample stream as part of the quality assurance and quality control (QA/QC) monitoring for the program at an insertion rate of ~14%.

The representativeness of the LeachWELL analytical technique to a mineral deposit is dependent on the leaching characteristics of the material submitted. As no certified standard reference material had been prepared from the same material that was to be leached, no standard reference materials were inserted in the grade control sample stream to directly monitor analytical precision. The QA/QC program indicated no material biases with the grade control analyses, and no sample quality issues were observed with the data.

11.7 Databases

The current drill hole and assay database for the Project is stored in an SQL database administered from the Artemis Gold Vancouver office.

Drillhole data logged in the field during the Richfield and New Gold exploration programs were entered into a LogChief database, which validated the data as they were entered. The assay certificates received from both Eco Tech and ALS were delivered in a format that allowed for instant import to the database.

Access permission for entering and editing data into the database is restricted to the Artemis Gold Database Administrator. The database is hosted on the Artemis Gold server, which routinely backs up every day for protection from data loss due to potential drive failures or other technical issues.

11.8 Sample Security

Samples were transported to Prince George by truck, where the driver waited with the samples in the truck until pick-up for onward shipment by a bonded courier. Before July 2011, the Richfield samples, including the standards, blanks, and duplicates, were shipped to Eco Tech; subsequently, samples were shipped to ALS.

11.9 Sample Storage

Remaining half cores were archived in core racks in the vicinity of the existing camp infrastructure with metal roofing protecting the core boxes from the elements. Core racks which were within the footprint of the mine infrastructure were relocated in 2022 to safe locations southeast of the processing plant. Minor portions of regional core and approximately 3% of the Blackwater core were destroyed during a wildfire event in July 2023.

All pulp materials are archived in sample storage containers at the Blackwater site.

11.10 Comments on Section 11

In the opinion of the QP, the sample preparation security and analysis are appropriate to support Mineral Resource estimation. Data from holes drilled between 1981 and 1994 have no documented QA/QC information, and they are not deemed acceptable for use in resource estimation.

12.0 DATA VERIFICATION

12.1 Data Verification Performed by the QPs

12.1.1 Ms. Sue Bird

The QP visited the Blackwater site on 14 July, 2020. Verification of drilling and site conditions included:

- Inspection and verification of the drill hole collar locations and layout;
- Fly-over to obtain and overview of the general site layout;
- Examination of the core for several mineralized intervals;
- Correlation of mineralization with logged intervals in the database;
- Discussion of sample preparation, handling, storage, and transportation with the site staff.

Ms. Bird oversaw review of 1% of the assay database (2,137 samples) against the original laboratory certificates for accuracy in which no errors were found. The collar, survey and assay data were also validated when imported to the modelling software (MineSight) to ensure no errors in the database. Validation of the drill hole data included:

- Checking for missing collars when survey and/or assays were available and vice-versa;
- Checking that data prior to the implementation of rigorous QAQC and without original Certificates available has not been used in the drill hole database used for modelling;
- Ensuring there are no overlaps of assay intervals;
- Checking of mis-matched downhole survey recordings;
- Ensuring collar elevation matched topography;
- Exporting drill hole data to ensure that exported data matched imported data;
- Comparing the composited weighted mean grade values of the assay data to ensure that there were no errors on importing.

12.1.2 Mr. Marc Schulte

The resource model data was loaded into mine planning software and data was confirmed by the QP, via independent re-creation of resource tonnages and grades in the mine planning software.

Topography data for the open pit and stockpile areas was verified during Mr. Schulte's site visit. Verification that topography has not been materially altered from the date of the site visit to 2024 was verified via on site photos and videos taken of the open pit and stockpile areas in late 2023.

Excavation and topography alteration as part of the planned Phase 1 works was not verified, as the work had not been completed as at the effective date of this Report.

Application of geotechnical and hydrogeotechnical data used to inform mine designs was independently verified via design reviews by the QP with Knight Piésold geotechnical engineers, who prepared the design criteria.

Application of all other mine design criteria data was verified via inhouse independent engineer reviews with the QP.

Mine design data used in the general arrangement was verified by the QP as current for this Report.

Mineral reserve tonnages and grades were verified via inhouse independent engineer reviews with the QP.

The QP has verified that the Mineral Reserves and mine production schedule are properly represented in the financial model.

Mining capital and operating cost input data were sourced from vendor quotations and were verified for reasonableness, by the QP, compared to competitive quotations received for other open pit mining operations in Canada. Input data for mine operation system productivities and consumable and labour usage were also verified by the QP as reasonable, when compared to rates measured at other large scale Canadian open pit mining operations.

The financial model data inputs were verified by the QP to be reasonable, with respect to industry standards for the sale of gold and silver doré. The various debt and streaming contracts in place with Artemis Gold were verified by the QP to be accurately represented in the model. The model was verified by the QP to ensure it accurately reflected the intended production plan and associated capital and operating cost estimates prepared by all parties.

12.1.3 Dr. John Thomas

Dr. Thomas performed a site visit in 2020, as outlined in Section 2.4.

Verification of the metallurgical test work included review of historical test work results, much of which was carried out in 2019 for the previous owners of the Project. Historic comminution data were also available and was reviewed by the QP and found to be acceptable. The data were incorporated into the dataset on comminution.

The QP undertook a new series of test work using carbon in leach with pre-aeration, cyanide destruction and carbon loading. Base Metallurgical Laboratories (BaseMet) was selected for the program. The QP has worked with BaseMet for many years and is familiar with the test procedures and has found their work to be reliable. The analysis techniques and results were reviewed and discussed with the principal at BaseMet.

12.1.4 Mr. Daniel Fontaine

Mr. Fontaine completed a site visit during 2024 as outlined in Section 2.4.

Verification of hydrometeorological data collection and site conditions by Mr. Fontaine included:

- Inspection and verification of key hydrological and climate monitoring locations and equipment;
- Regular review of hydrological and climate data collection site visit summaries, baseline data reports, and the associated engineering hydrometeorological site characterization reports;
- Discussion of data collection methods, data quality and analysis techniques with the relevant responsible professionals carrying out the work to be satisfied that the characterization of the site conditions is reasonable and suitable to support engineering design for project advancement.

Verification of geotechnical data and site conditions by Mr. Fontaine included:

- Planning and executing a series of phased geotechnical and hydrogeological site investigation programs between 2012 and 2023;
- Inspecting and verifying key geotechnical data collection locations, including conducting field reviews during site investigation execution;
- Review and seal of site investigation data reports prepared for site investigation programs conducted in 2019 and 2020–2021. Review and approval of the site investigation data report from 2022–2023;
- A desktop level review of the 2012–2013 geotechnical data, referring back to the original field notes and data, and updating interpretations in consideration of site investigation findings between 2019–2021 and the most recent geological model for the site;
- Discussion of data collection methods, data quality and analysis techniques with the other supporting registered professionals carrying out the work to be satisfied that the characterization of the site conditions is reasonable and suitable to support engineering design for project advancement;
- Review and seal of the geotechnical site characterization report and subsequent site characterization update letters that interpret the site conditions.

12.1.5 Mr. Rolf Schmitt

Mr. Schmitt has not conducted a site visit. He has examined in detail Google Earth images of the mine permitting area and examined photographic records of mine site locations included in baseline environmental survey reports prepared by qualified persons.

Mr. Schmitt's senior technical and QP responsibilities have involved the direction, and review of the technical studies and report content of environmental baseline and monitoring studies performed over nearly two decades under the direction of multiple qualified professionals in accordance with regulatory requirements and BC best practice guidelines for collection of environmental baseline data at the time of data collection.

Mr. Schmitt reviewed the summary of the scope of studies summarized in Section 20 for accuracy and completeness. Mr. Schmitt has directly reviewed and verified for accuracy, the alignment of management plans, with conditions in issued permits (Mines Act, Environmental Management Act, Land Act), and directly assisted the owner in responses to regulator, Indigenous community, and stakeholder engagement through the regulatory and permitting processes. Therefrom, a detailed understanding of the environmental and social setting, issues, risks, and mitigations was acquired.

In support of the preparation of this Report, Mr. Schmitt validated the receipt of the permits and authorizations received by the Owner, as maintained in the Owner's permit registry in relation to the confirmed list of requirements from the regulators, and the applications submitted for permits and authorizations, as presented in Table 20-2 and Table 20-3. Mr. Schmitt was able to further verify that there are no absences of major permits or authorizations required to construct and operate the mine.

Mr. Schmitt has personal knowledge of the Environmental Management System framework of management plans and directly participated in their development and acted as qualified professional sign-off of many of these management plans to fulfill the requirements of the Mines Act permit.

12.1.6 Mr. John Dockrey

Mr. Dockrey reviewed the geochemical baseline test procedures, results, and QA/QC procedures in preparation for the Joint Application for Mines Act Environmental Management Act Permit.

This included the acid base accounting testwork that was used to define ML/ARD management practices for waste rock and tailings, and the kinetic test results used in source term predictions used to inform water treatment requirements. This dataset included sorption testwork and saturated column tests that were directly supervised by Mr. Dockrey in the Lorax Laboratory, as well as historic analytical test completed prior to Mr. Dockrey becoming the lead geochemistry.

Historic geochemical characterization was summarized in the 2013 Geochemical Characterization Report (AMEC, 2014). This report was submitted in support of the Blackwater Mine Environmental Assessment but was not signed or sealed by qualified professionals. Original data files from this report were provided to Mr. Dockrey, and the methods and results were documented in sufficient detail to allow for verification of results. Mr. Dockrey is aware that documents of this type were not routinely signed and sealed by qualified professionals at the time of the EA submission.

As a result of his reviews, Mr. Dockrey considers the information acceptable to support the purposes used in the Report.

12.1.7 Mr. Olav Mejia

Mr. Mejia reviewed and validated the accuracy and reliability of the estimated process plant operating cost, as of Q4, 2023 and Q1 2024. The verification process encompasses a thorough review of the cost estimation methodology and data sources provided by Artemis Gold.

The data used for estimating operating costs was sourced from suppliers of consumables, Artemis Gold, equipment suppliers, internal databases, and third-party sources. Assumptions made in the cost estimation process were documented and reviewed for consistency and reasonableness.

The methodology employed for calculating operating costs followed industry practices and standards. It included consideration of consumption rates for consumables and reagents, and estimation of power consumption based on equipment specifications and load list. Where applicable, comparisons were made with industry benchmarks and historical data to validate the assumptions and results. The operating cost estimate represents the estimated expenses associated with the future plant operations (Phase 2 and Phase 3), excluding items such as sunk costs, contract mining costs, TSF operation, and general and administrative expenses.

The process plant operating costs can be used in the economic analysis in Section 22, and support the Mineral Reserves in this Report.

12.1.8 Mr. Sohail Samdani

Mr. Samdani reviewed and validated the accuracy and reliability of the estimated process plant capital cost estimate, as of Q4, 2023 and Q1 2024. The verification process encompassed a thorough review of the cost estimation methodology and data sources for the process plant, data provided by Artemis Gold and vendors through applicable budgetary costs requests.

The estimate was based on inputs provided by Artemis Gold, Knight Piésold, Moose Mountain, and Lycopodium for the process plant, with the basis of the capital cost estimate being budget quotes and historical data.

The data used for estimating capital costs were sourced from equipment suppliers, contractors, Artemis Gold, internal databases, and third-party sources. Assumptions made in the cost estimation for the process plant were documented and reviewed for consistency and reasonableness.

The methodology employed for estimating capital costs followed industry best practices and standards. Where applicable, comparisons were made with industry benchmarks and historical data to validate the assumptions and results. The process plant capital cost estimate represents the estimated expenses associated with the proposed Phase 2 and Phase 3 process plant operations.

Mr. Samdani, based on the review and verification process, confirmed the reliability and accuracy of the estimated process plant capital costs as of Q4, 2023. The exclusion of certain costs was noted and accounted for in the verification process.

13.0 MINERAL PROCESSING AND METALLURGICAL TESTING

13.1 Introduction

Testwork was completed in support of a number of mining studies from 2008 to 2019, by Inspectorate, G and T Laboratories, SGS, Dawson Metallurgical Laboratories, McClellan Laboratories, and Pocock and MetSolve.

Testwork included sample characterization, comminution, gravity concentration, leaching, flotation, oxygen uptake, cyanide destruction, carbon loading, and settling/viscosity tests.

Testwork results led to the elimination of heap leaching and flotation methods, and the conclusion that whole ore leaching was the most appropriate method for recovering gold and silver.

All recent work has focused on whole ore leaching and incorporated gravity separation as an integral part of the recovery process for gold and silver.

13.2 Metallurgical Testwork

13.2.1 Legacy Testwork

The average values from legacy comminution testwork are summarized in Table 13-1.

Table 13-1: Blackwater Comminution Parameters, Average of All Programs

Ore Type		SAG Mill Breakage Parameter (Axb)	Bond Work Index (kWh/t)	Abrasion Index
Oxide	Average	55.8	14.5	0.111
	Std. Dev.	13.0	2.16	0.042
Transition	Average	38.0	16.0	0.142
	Std. Dev.	8.7	2.71	0.099
Sulphide	Average	31.15.3	18.3	0.206
	Std. Dev.		1.8	0.114
All Types	Average	35.7	17.4	0.182
	Std. Dev.	11.1	2.45	0.110

Leaching testwork showed that a grind to a P_{80} of 150 µm was optimum and pre-aeration was effective in reducing cyanide consumption. The addition of lead nitrate had no effect on recovery and it was stated that all samples responded well to direct cyanidation with extractions of about 90%. Silver extractions of about 65% were obtained. Leach times of 48 hours were recommended to ensure maximum extraction. Extractions from ore with lower head grades did show some reduction.

Gravity concentration was not tested, as no coarse (200 µm) gold was found in the samples.

Extractions of gold and silver were expressed in the form of equations relating extraction to head grade.

At head grades of 1 g/t Au, and 5 g/t Ag the extractions for gold and silver used in the 2013 FS were as outlined in Table 13-2.

Table 13-2: Extraction of Gold and Silver Presented in 2013 Feasibility Study

Ore Type	Percent Extraction at 1 g/t Au	Percent Extraction at 5 g/t Ag
Oxide	90.3	64.7
Transition	84.5	59.9
Sulphide	87.9	44.8

13.2.2 Current Testwork

An extensive program of testwork was carried out in 2019 by BaseMet Laboratories, with some additional work in 2020.

Basic leach conditions were initially determined using composites comprising samples representing expected grades over the first 10 years of mining. Fifteen drill core intervals were used for the first composite, 19 for the second and nine for the third.

A P_{80} grind of 150 µm was confirmed, as were the requirements for pre-aeration and a somewhat long leach time of 48 hours. Initial cyanide addition was varied and for the composites it was found that 500 ppm was adequate, although higher concentrations resulted in faster leach kinetics. It was determined that gravity concentration prior to leaching recovered significant amounts of gold and increased the overall recovery. This was incorporated into the proposed flow sheet and all samples were first ground and subjected to gravity concentration using a centrifugal concentrator before being leached.

An additional 48 samples were taken from drill holes distributed throughout the deposit and treated using the proposed flow sheet. In addition to these tests, comminution testing was completed. Cyanide destruction was tested, using SO_2/air. Carbon loading characteristics and slurry rheological parameters were determined.

Comminution Testing

Table 13-3 shows comminution results, including Bond ball mill work index (BWi), Bond rod work index (RWi), abrasion index (Ai) and SMC breakage parameters (axb), from four twin holes drilled specifically to provide metallurgical samples.

Table 13-3: Grinding Tests on Twinned Holes

Sample ID	CSS (µm)	P80 (µm)	BWi (kWh/t)	WiRM kWh/t	Ai	SMC (axb)
BW91:108–122 m	212	137	14.4		0.066	27.2
BW624:85.7–108 m	212	159	17.7	16.3	0.295	37.1
BW832:45–65 m	212	151	11.8	15.0	0.072	36.1
BW832:79.5–95 m	212	153	13.8		0.103	52.0

Note: CSS = closed side setting

Table 13-4 shows the results obtained from metallurgical composites 1, 2 and 3.

Table 13-4: Grinding Tests on Composites

Sample	CSS (µm)	P80 (µm)	BWi (kWh/t)
Comp 1	212	157	21.1
	150	109	20.3
	106	78	19.3
Comp 2	212	157	19.4
	150	109	18.3
	106	78	17.4
Comp 3	212	157	19.8
	150	109	19.2
	106	78	18.9

Note: CSS = closed side setting

Results are similar to those obtained in the legacy testwork, but the BWi is somewhat higher for the composites than for the twinned holes, indicating ore hardness variability.

The variable work index results made the sizing of a semi-autogenous grind (SAG) mill/ball mill combination difficult; selecting the 75% quartile for the design would probably lead to periods when design throughput would not be reached. This has occurred in at least two mining projects in this region of British Columbia.

Gravity Concentration

A blend of the composites was used to provide sufficient sample mass to assess the effectiveness of gravity concentration. Results are summarized in Table 13-5.

Table 13-5: Gravity Concentration Test Results

Sample	Product	Weight %	Au Assay (g/t)	Au Distribution
Blend of three composites	Knelson Con 1	0.3	126.0	25.8
	Knelson Con 2	0.3	85.8	17.2
	Knelson Con 3	0.3	74.3	14.9
	Knelson Tail 3	99.1	0.65	42.1

Despite the high mass pull generally associated with the laboratory testing of gravity concentration overestimating the gravity recovery obtained in the plant, it is evident that gravity concentration is effective on this ore and should be incorporated in the process flow sheet.

Early tests on the composites were carried out to determine the effect of gravity concentration on the overall recovery. At a grind of P80 = 150 µµm, composite 1 without gravity concentration gave an overall extraction of 91.5% Au, which rose to 94.6% with gravity concentration. Composite 2 showed similar behaviour with a rise from 92.1% to 93.8%, while composite 3 was essentially unchanged, giving an extraction of 95.3% without gravity, and 95.4% with gravity. This, with similar results at finer grind sizes, led to the use of gravity concentration in all subsequent tests. This testwork also demonstrated that finer grinds than a P80 of 150 µm did not result in a significant increase in gold recovery.

Sample Characterization

Analysis of the three metallurgical composites is provided in Table 13-6.

Table 13-6: Composition of the Three Composites

Sample	Au (g/t)	Ag (g/t)	Cu (%)	Fe (%)	S(t) (%)	SO_4 (%)	S(s) (%)
Composite 1	1.05	4	0.03	3.5	1.4	0.05	1.35
Composite 2	1.15	5	0.02	3.3	1.65	0.07	1.51
Composite 3	1.41	6	0.05	1.9	1.49	0.07	1.39

Note: S(t) = total sulphur , S(s) = sulphur as sulphide

A quantitative evaluation of materials by scanning electron microscope (QEMSCAN) analysis was carried out to determine the minerals present and the sulfur distribution.

Major mineral species identified included copper sulphides, galena, sphalerite, pyrrhotite, and arsenopyrite. Mineralogical analysis indicated the presence of copper sulphides, sphalerite, pyrite, pyrrhotite, and trace quantities of chrysocolla, galena and arsenopyrite.

The major gangue minerals noted in the mineralogical examinations included quartz, K-feldspar, muscovite, biotite/phlogopite, and chlorite.

Lesser to trace quantities of other gangue minerals were identified, including iron oxides, plagioclase feldspar, epidote, kaolinite (clay), rutile/anatase, manganiferous limonite, and apatite.

Overall, significant levels of copper and zinc were found to be present. Leaching testwork showed that three samples from the 48 selected contained elevated copper (up to 2,640 ppm), two of which gave lower than expected leach extractions, but one gave >90% gold extraction. The other 45 samples had a mean copper content of 190 ppm and no effect of copper content could be identified. Generally high gold extractions in cyanide leaching indicated that although sphalerite was present, it had no identifiable effect on gold extraction.

Arsenic was present, 0.1% of arsenopyrite being reported in two of the composites, but appeared to have no effect on gold leaching except for one sample which was very high in arsenic (8,950 ppm) and which gave low gold extraction.

Mercury levels appeared low, and from leach solution, an average level of 0.07 ppm was calculated.

Oxygen Uptake Rate

The rate of oxygen uptake was measured for each of the composites. The results show some difference between the composites, but the main oxygen demand occurred in the first few hours and this should be accounted for when designing the oxygen addition system.

Leach Testing

The results of the leach test series were reviewed. A comparison between tests 2–6 and tests 7–11 showed that gravity concentration increases overall recovery. Comparison of tests 8, 12 and 13 showed no difference between initial cyanide concentrations of 500, 1,000, and 1,500 g/t. An initial cyanide concentration of 1,000 mg/L was used for tests 1–11. From test 14 on, pre-aeration was used and cyanide consumption dropped, in most case a little under 1 kg/t, compared with over 1.5 kg/t with no pre-aeration. The addition of lead nitrate appeared to have no effect.

All tests were carried out at 40% solids, apart from test 22 where 50% solids was used. The use of this higher solids percentage appeared to reduce the overall recovery by about 2%, but this requires confirmation.

In general, oxygen was sparged into the pulp during leaching. Leach kinetics were found to be significantly slower when air is used. The final extractions obtained were very close to those obtained using oxygen. Composite 1, upon which most of the tests were carried out, had the highest oxygen demand of the three composites.

The overall gold and silver extractions for each of the composites, using the mean for tests done under similar conditions as those used on composite 1, were for gold, 94.1%, 93.8% and 95.4% for composites 1, 2 and 3 respectively and for silver, 59.3%, 59.3% and 65.9%, for composites 1, 2, and 3 respectively.

Leach Variability Testing

To investigate the variability of the extraction over the orebody, 48 samples were selected, distributed throughout the deposit. The extraction testing was carried out using the standard: grind to a P80 of 150 µm-gravity concentration (pre-oxidation for two hours- leaching at 40% solids, 500 mg/L initial cyanide, and oxygen sparging for 48 hours).

Of the 48 drill composites tested, six gave overall extractions (gravity +leach) of <90%; the remaining other samples gave results significantly >90%. Of the samples which initially showed low extractions, VC7, VC37 and VC42 all had high copper contents, of 2,640, 1,940 and 1,340 ppm respectively. In the case of sample VC7, repeating the test with higher cyanide concentration raised the extraction to 99% but the other two samples remained with rather low extractions when cyanide concentration was raised. These compare with all the other samples which had an average of 190 ppm copper if these three samples were ignored.

Another of the low recoveries (VC25) occurred in drill hole BW 0979, which had an abnormally high arsenic content (8,950 ppm) and was low in gold grade, as was sample VC26.

It is presumed that in the other three cases, additional cyanide was needed, or the gold contained was at least partially encapsulated in the copper or arsenic sulfide minerals.

The low extractions obtained from samples VC16, VC33 (drill hole BW 839) and VC46 (drill hole BW 91) require further investigation.

Cyanide Destruction

The leach slurry from carrying out the standard extraction process on composites 1, 2 and 3 were tested for cyanide destruction, using the SO_2/air method. Sodium metabisulfite was used as the SO_2 source.

The results show that very low levels of weakly acid dissociable cyanide (CN_{WAD}) were obtained, using a 60 minute retention time, an initial copper catalyst concentration of 15 mg/L, and SO_2:CN_{WAD} ratios of 4.5, 3.5 and 4 for composites 1,2 and 3 respectively. Detailed water analysis was carried out on the final liquids.

Carbon Loading

The leach solution produced from composite 3 was used for carbon loading testwork. A sample of carbon was obtained from Quadra Chemicals (type PJ612G-SS240), the likely supplier of carbon for operations.

Cyanide Destruction Using Hydrogen Peroxide

The use of SO_2/air introduces a sulfate into the tailings stream and a limited number of tests were carried out using hydrogen peroxide were carried out as a possible way to eliminate sulfate build up. It is well documented that the use of peroxide on slurries, particularly those containing sulphides is not efficient and the tests were carried out on solution, which would be separated using a tailings thickener or possibly on tailings pond overflow.

The CN_{WAD} levels in the liquid phase of leach solution from composites 1 and 2 (tests 3 and 4) were similar at 244 and 237 mg/L respectively. This is similar to earlier work. Using a 2:1 ratio of peroxide to CN_{WAD} and 50 mg/L copper as catalyst, removal of CN_{WAD} down to <1 mg/L was obtained.

Slurry Viscosity

The sheared viscosity properties of the cyanide tailings from test 3 and 4 were determined using a Brookfield DV2T viscometer.

13.3 Recovery Estimates

The average extractions for the three composites representing the first five years of mining were 94.4% for gold and 61.5% for silver. The average gold extraction for the 48 variability composites was 93.1% for gold and 69.8% for silver.

Considering all of these results, the use of a recovery of 93% for gold and 65% for silver is recommended, which would include solution losses assuming a dissolved gold concentration of 0.008 mg/L in the final solution for gold, 0.1 mg/L for silver.

The average cyanide consumption in the variability testwork was 1 kg/t and the lime consumption 1.06 kg/t. However, in the previous, extensive testwork, cyanide consumption was only 0.4 kg/t and it is known that the half core samples used for the variability tests had been stored for some years and had undergone visible oxidation. Formation of metal-cyanide complexes and thiocyanate would certainly increase the apparent cyanide consumption. The test procedure used was to adjust the pH periodically and a pH of as low as 10.1 was recorded in some tests prior to addition of more lime to bring it back up to 10.5. Taking this into account and considering the values recorded in previous testwork carried out with fresh core and at a pH of 11, it is recommended that a cyanide consumption of 0.6 kg/t is used.

13.4 Metallurgical Variability

Variability testwork focused on the material to be treated in the earlier phases of the LOM plan. Samples selected for metallurgical testing were representative of the various types and styles of mineralization within the earlier phases of the mine plan. Samples were selected from a range of locations within the deposit zones. Enough samples were taken so that tests were performed on sufficient sample mass.

Additional variability testwork is recommended for mineralization in the later periods of the mine plan (see discussion in Section 26).

13.5 Deleterious Elements

No deleterious elements are known from the processing perspective.

14.0 MINERAL RESOURCE ESTIMATES

14.1 Introduction

The Mineral Resource estimate includes data for all drilling completed by Richfield and New Gold between August 1, 2009, and January 16, 2013. The resource estimate was prepared by Ms. Sue Bird, P.Eng.

14.2 Blackwater Mineral Resource

The Mineral Resource statement for the Blackwater deposit with an effective date of May 5, 2020 is listed in Table 14-1. Mineral Resources are reported inclusive of those Mineral Resources converted to Mineral Reserves. Mineral Resources that are not Mineral Reserves do not have demonstrated economic viability. Mineral Resources were estimated using the 2019 CIM Best Practice Guidelines and are reported using the 2014 CIM Definition Standards.

Table 14-1: Blackwater Mineral Resource Estimate, Effective Date 5 May, 2020 (base case is highlighted)

Classification	Cut-off (g/t AuEq)	Tonnage (kt)	In-situ Grades			In-situ Contained Metal		
			AuEq	Au	Ag	AuEq	Au	Ag
			(g/t)	(g/t)	(g/t)	(koz)	(koz)	(koz)
Measured	***0.20***	***427,123***	***0.68***	***0.65***	***5.5***	***9,360***	***8,905***	***75,802***
	0.30	313,739	0.84	0.80	5.9	8,463	8,109	59,009
	0.40	238,649	0.99	0.96	6.1	7,627	7,347	46,727
	0.50	186,687	1.15	1.11	6.2	6,881	6,656	37,333
	0.60	149,261	1.30	1.26	6.4	6,223	6,039	30,521
	0.70	120,916	1.45	1.41	6.6	5,633	5,479	25,619
Indicated	***0.20***	***169,642***	***0.56***	***0.51***	***8.5***	***3,046***	***2,766***	***46,578***
	0.30	123,309	0.68	0.61	10.4	2,677	2,431	41,112
	0.40	86,473	0.81	0.74	12.4	2,264	2,057	34,419
	0.50	64,305	0.94	0.85	14.8	1,947	1,763	30,681
	0.60	50,527	1.05	0.95	17.2	1,705	1,537	27,957
	0.70	40,317	1.15	1.03	19.6	1,493	1,340	25,458
Measured + Indicated	***0.20***	***596,765***	***0.65***	***0.61***	***6.4***	***12,406***	***11,672***	***122,381***
	0.30	437,048	0.79	0.75	7.1	11,140	10,540	100,120
	0.40	325,122	0.95	0.90	7.8	9,890	9,404	81,146
	0.50	250,992	1.09	1.04	8.4	8,828	8,419	68,014
	0.60	199,788	1.23	1.18	9.1	7,928	7,577	58,478
	0.70	161,233	1.37	1.32	9.9	7,125	6,819	51,077
Inferred	***0.20***	***16,935***	***0.53***	***0.45***	***12.8***	***288***	***246***	***6,953***
	0.30	11,485	0.66	0.57	16.2	245	210	5,971
	0.40	8,690	0.77	0.65	19.2	214	182	5,373
	0.50	5,552	0.95	0.79	26.0	169	142	4,648
	0.60	4,065	1.10	0.90	32.7	143	118	4,279

Notes:

1. The Mineral Resource estimate was prepared by Sue Bird, P.Eng., the Qualified Person for the estimate and employee of Moose Mountain Technical Services. The estimate has an effective date of May 5, 2020. There have been material changes since this data.
2. Mineral Resources are reported using the 2014 CIM Definition Standards and are estimated in accordance with the 2019 CIM Best Practices Guidelines.
3. Mineral Resources are reported inclusive of Mineral Reserves.
4. Mineral Resources that are not Mineral Reserves do not have demonstrated economic viability.
5. The Mineral Resource has been confined by a conceptual pit shell to meet “reasonable prospects of eventual economic extraction” using the following assumptions: the 143% price case with a Base Case of US$1,400/oz Au and US$15/oz Ag

at a currency exchange rate of 0.75 US$ per C$; 99.9% payable Au; 95.0% payable Ag; US$8.50/oz Au and US$0.25/oz Ag offsite costs (refining, transport, and insurance); a 1.5% NSR royalty; and uses a 93% metallurgical recovery for gold and 55% recovery for silver.

6. The AuEq values were calculated using US$1,400/oz Au, US$15/oz Ag, a gold metallurgical recovery of 93%, silver metallurgical recovery of 55%, and mining smelter terms for the following equation: AuEq = Au g/t + (Ag g/t x 0.006).
7. The specific gravity of the deposit has been determined by lithology as being between 2.6 and 2.74.
8. Numbers may not add due to rounding.

Multiple indicator kriging (MIK) was used for gold estimation due to the significant value and non-linear distribution of the gold mineralization at higher grades. This is evident by the cumulative probability plots and coefficients of variation (CVs) of the gold grades by domain, as discussed in Section 14.5. Ordinary kriging (OK) was used for silver estimation because the CVs are generally lower, and the silver is generally lognormally distributed at higher grades.

The base case cut-off grade within the "reasonable prospects of eventual economic extraction" constraining pit is 0.20 g/t gold equivalent (AuEq), as highlighted in Table 14-1. Table 14-1 includes a range of AuEq cut-off grades to show the sensitivity of the resource estimate to variations in cut-off grade. At a 0.20 g/t AuEq cut-off, the total Measured and Indicated Mineral Resource is estimated at 597 Mt at 0.65 g/t AuEq, 0.61 g/t Au, and 6.4 g/t Ag for a total of 12.4 M AuEq ounces. Of the total Measured and Indicated Mineral Resources, 75% are in the Measured category.

The difference in the contained metal from the 2013 resource estimate is 3% more gold ounces in the Measured and Indicated categories. The slight increase in metal content in 2020 is considered to be due primarily to the cut-off changing from 0.3 g/t AuEq in 2013 to 0.2 g/t AuEq in 2020.

Mineral Resources are reported inclusive of those Mineral Resources converted to Mineral Reserves. Mineral Resources that are not Mineral Reserves do not have demonstrated economic viability. Mineral Resources were estimated using the 2019 CIM Best Practice Guidelines and are reported in situ, using the 2014 CIM Definition Standards.

The following factors, among others, could affect the Mineral Resource estimate: commodity price and exchange rate assumptions; pit slope angles and other geotechnical factors; assumptions used in generating the constraining conceptual pit shell, including metal recoveries, and mining and process cost assumptions.

The QP is not aware of any environmental, permitting, legal, title, taxation, socioeconomic, marketing, political, or other relevant factors that could materially affect the Mineral Resource estimate.

14.3 Key Assumptions and Data used in the Estimate

The total sample database contains results from 1,041 core holes totalling 317,718 m. Due to lack of QA/QC and accurate survey information, holes drilled before 2009 were not used for statistical analysis, or grade estimation.

A summary of the drill holes within the Blackwater block model and used for interpolation is provided in Table 14-2.

Table 14-2: Summary of Drill Holes and Assays used in the Blackwater Resource Estimate

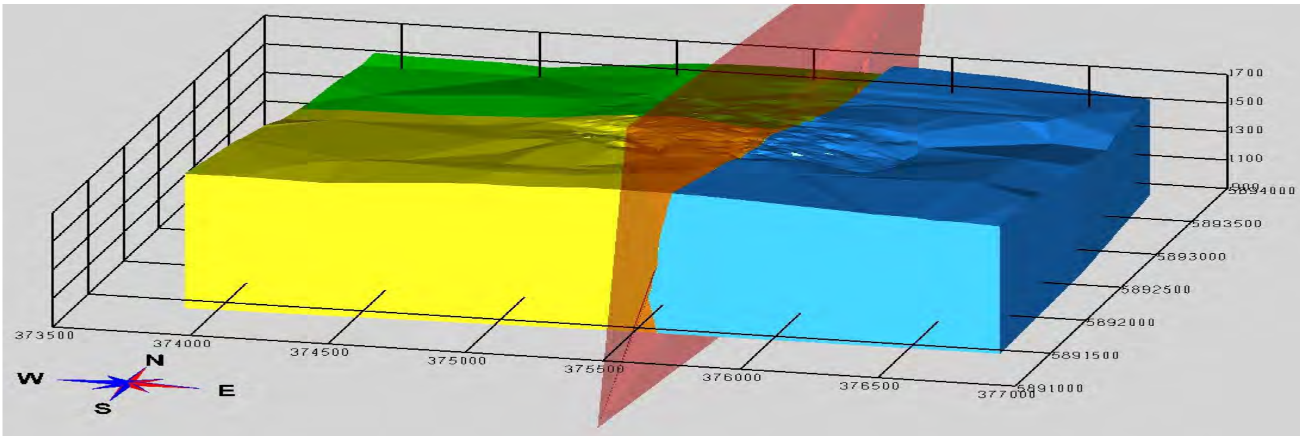
Year	Company	Holes	Metres	Intervals Assayed	Metres Assayed	% Assayed
2009	Richfield	18	3,621	3,450	3,414	94.3
2010	Richfield	56	20,920	20,172	20,048	95.8
2011	New Gold	125	49,316	46,008	46,231	93.7
2011	Richfield	59	19,727	18,484	18,841	95.5
2012	New Gold	743	215,289	200,211	203,416	94.5
2013	New Gold	1	420	413	414	98.6
Total	***1,002***	***309,293***	***288,738***	***292,364***	***94.5***	

An additional 12 drill holes within the deposit area were discovered subsequent to the resource modelling and the mineral resource estimate. They were determined to not be material to the resource estimate therefore the estimate remained unchanged.

14.4 Geological Modelling

The geological, alteration and structural models were created by New Gold (see Section 7). The domains used in the interpolation are based on this work, as well as noted changes in orientation of the mineralization. The overburden surface was created using logging provided by Artemis Gold. The domains and interpolation were clipped to the bottom of the overburden surface, and the resource estimate is based only on the percentage of the block below the overburden. Figure 14-1 illustrates the three domains used for interpolation. The major north–south-oriented fault bisecting the deposit divides the domains, with an additional domain splitting the deposit to the north due to a change in the mineralization orientation. Additional faulting recognized in the structural modelling is not used, as these faults do not represent hard boundaries to mineralization.

Figure 14-1: Domains Used for Interpolation

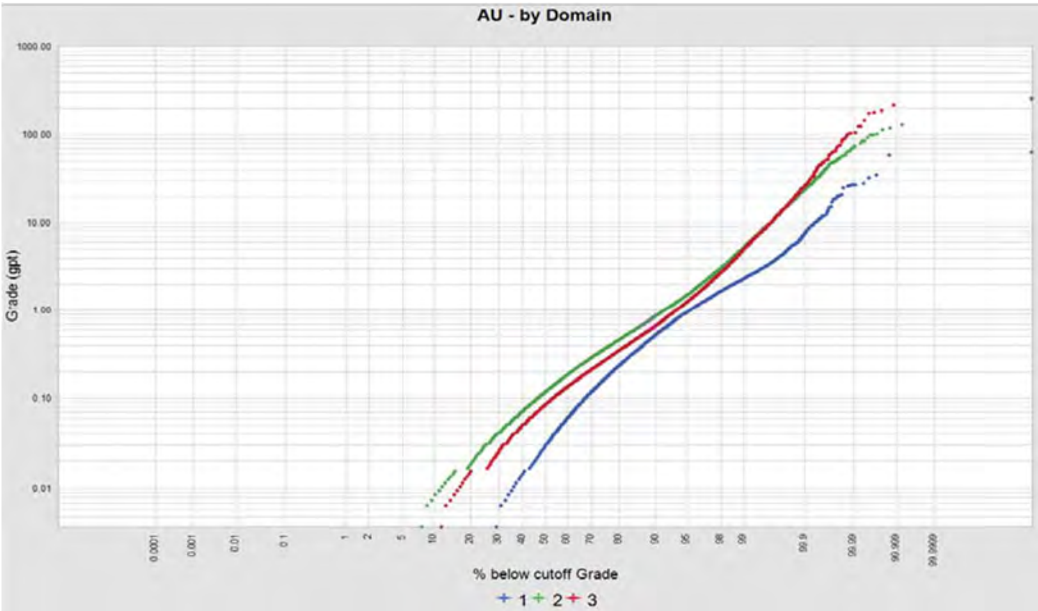


Note: Figure prepared by Moose Mountain, 2020. Green = 1, yellow = 2, blue = 3, red = major north–south fault.

14.5 Assay Statistics and Capping

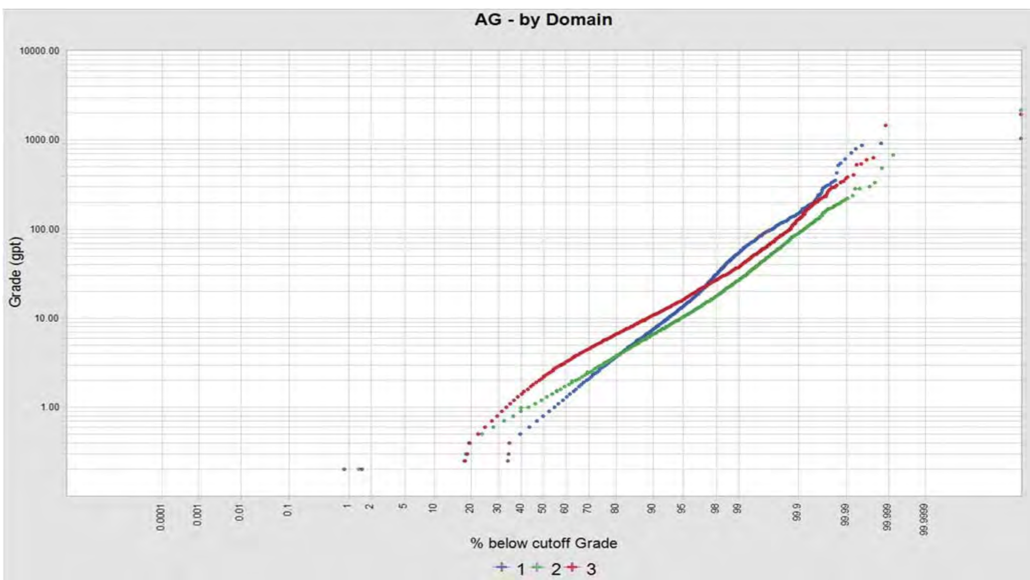
The assay statistics were examined using boxplots, histograms, and CPPs. The grade distribution for silver within the domains is generally lognormal. However, the distribution for gold contains inflection points above about the 90% of the data, high grades that are not lognormal, and contains a significant amount of gold metal. The interpolation method used for gold grade estimation was selected to be MIK, and the interpolation method for silver is OK. Figure 14-2 and Figure 14-3 illustrate the cumulative probability plots by domain for gold and silver respectively.

Figure 14-2: Cumulative Probability Plot of Au Assay Grades by Domain



Note: Figure prepared by Moose Mountain, 2020.

Figure 14-3: Cumulative Probability Plot of Ag Assay Grades by Domain



Note: Figure prepared by Moose Mountain, 2020.

Assay statistics for uncapped gold and silver grades are summarized in Table 14-3, illustrating that composited grades equal assay grades, and therefore compositing has not introduced a bias.

Table 14-3: Summary Statistics of Assays and Composites

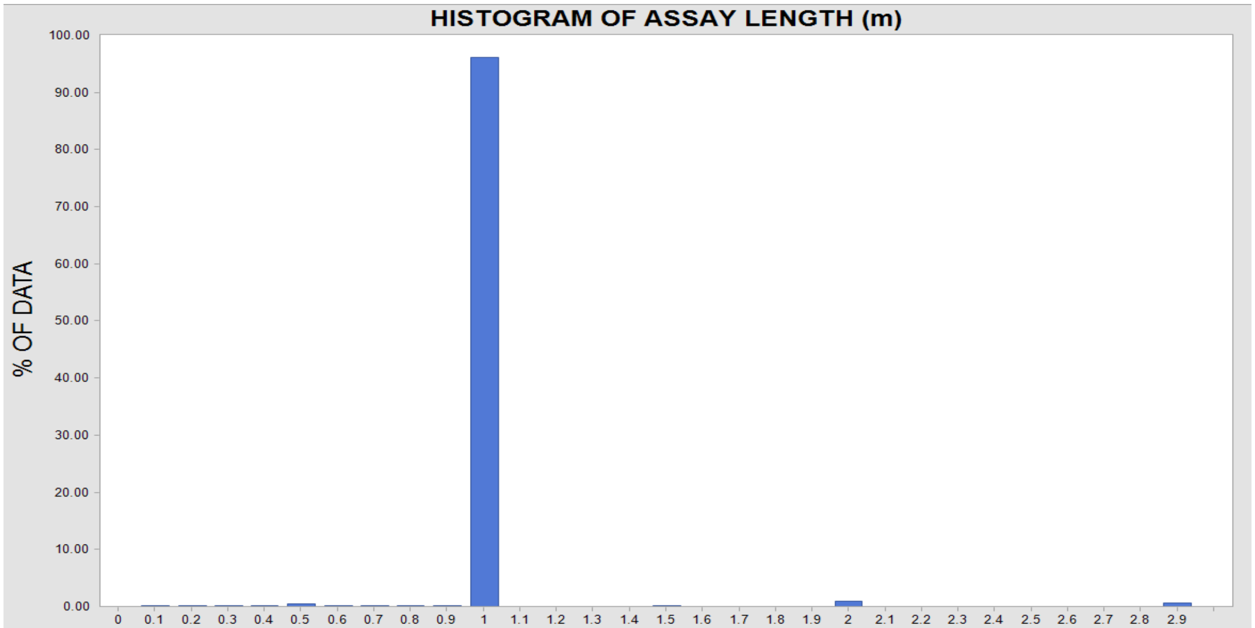
Source	Parameter	Au By Domain			Ag By Domain		
		1	**2**	**3**	**1**	**2**	**3**
Assays	Num samples	65,382	137,890	85,466	65,382	137,890	85,465
	Num missing samples	215	495	392	215	495	393
	Min (g/t)	0.003	0.003	0.003	0.10	0.10	0.10
	Max	63.70	252.00	262.00	1045.00	2170.00	1950.00
	Wtd mean (g/t)	0.208	0.446	0.401	3.94	3.04	4.90
	Weighted CV	3.609	4.480	6.538	3.96	3.09	2.85
Comps	Num samples	33,295	69,629	43,355	33,295	69,629	43,355
	Num missing samples	241	1350	1708	241	1350	1708
	Min (g/t)	0.003	0.003	0.003	0.10	0.10	0.10
	Max	45.900	132.450	221.000	615.50	1087.90	1705.00
	Wtd mean (g/t)	0.208	0.446	0.401	3.94	3.04	4.90
	Weighted CV	3.083	3.664	5.554	3.44	2.45	2.58
Difference (1-Assay/comp)		0.0%	0.0%	0.1%	0.0%	0.0%	0.0%

Note: Num = number, Wtd = weighted, CV = co-efficient of variation.

14.6 Compositing

Assay sample lengths varied across the drill programs but are generally between 1.0 and 2.0 m. A histogram of the assay intervals is shown in Figure 14-3, illustrating that virtually all assays are 1.0 m.

Figure 14-4: Histogram of Assay Lengths



Note: Figure prepared by Moose Mountain, 2020.

A base composite length of 2.0 m was used based on the fact that the planned bench height is 5 m and the assay length is 1.0 m. Assay data were coded with a domain value prior to compositing. The domain code was honoured during compositing. Any interval within a domain that was less than 1.0 m was composited with the interval above it, resulting in composite length ranging from 1.5 to 2.5 m.

Composite statistics, for the capped values are summarized in Table 14-4. The CV for gold remains rather high, further pointing to MIK as an appropriate interpolation method for gold interpolation. The capping for silver reduced the CV to a level at which OK estimation is appropriate.

Table 14-4: Summary of Capped Composite Statistics

Parameter	Au By Domain			Ag By Domain		
	1	2	3	1	2	3
Num samples	33,295	69,629	43,355	33,295	69,629	43,355
Num missing samples	241	1350	1708	241	1350	1708
Min (g/t)	0.003	0.003	0.003	0.10	0.10	0.10
Max	45.900	106.450	215.000	615.50	1002.90	1000.00

Parameter	Au By Domain			Ag By Domain		
	1	2	3	1	2	3
Wtd mean (g/t)	0.208	0.446	0.401	3.94	3.04	4.89
Weighted CV	3.070	3.605	5.522	3.44	2.39	2.21

Note: Num = number, Wtd = weighted, CV = co-efficient of variation.

14.7 Density Assignment

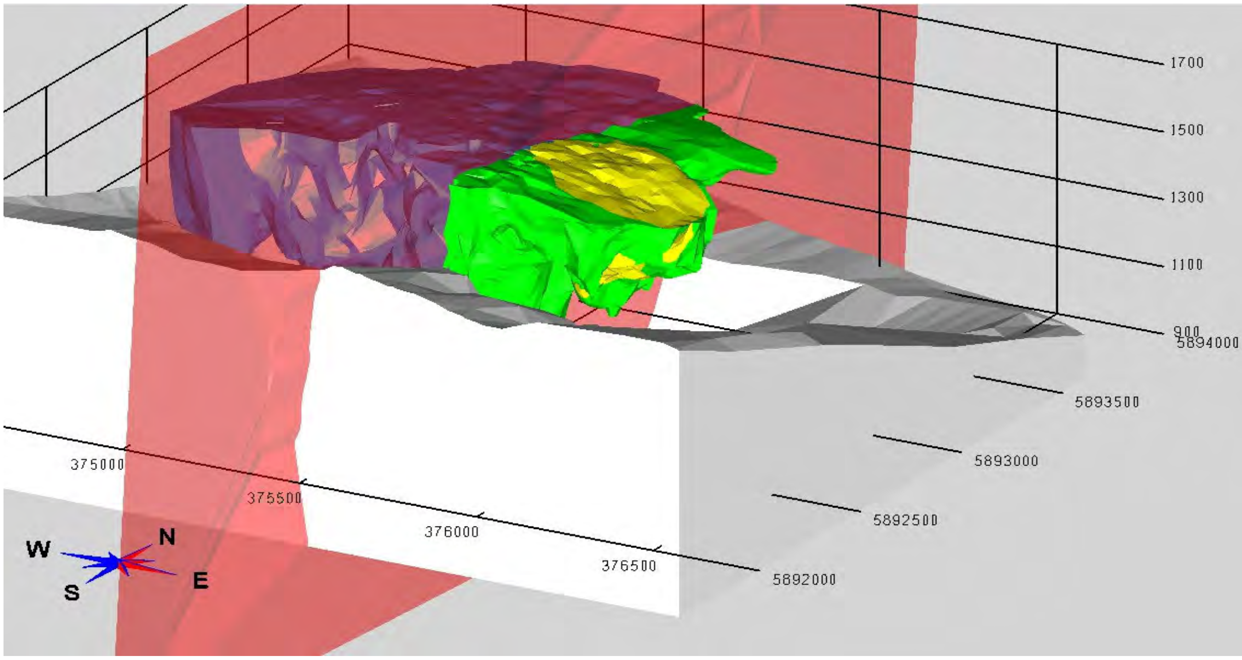
Model blocks were assigned the mean specific gravity value based on lithology and alteration as summarized in Table 14-5.

Table 14-5: Specific Gravity Assignment by Lithology

Lithology	SG
Overburden	2.0
Sediments	2.7
Laminated Volcanics	2.6
Fragmental Volcanics-West	2.7
Fragmental Volcanics-East	2.73
Unaltered Andesite	2.74

The solids used for SG estimation are shown in Figure 14-5.

Figure 14-5: Sediments and alteration Solids used for SG



Note: Figure prepared by Moose Mountain, 2020. White = sediments, yellow = central sericite, green = east sericite, blue = west sericite, red = fault.

14.8 Block Model Interpolations

The block model uses 10 x10 x 10 m blocks with the extents of the model summarized in Table 14-6.

Table 14-6: Summary of Block Model Extents

Direction	Minimum	Maximum	Block size	# Blocks
Easting	374,100	376,600	10	250
Northing	5,892,000	5,894,100	10	250
Elevation	800	1,850	10	105

MineSight software was used for geostatistical investigations and interpolations, as well as for the “reasonable prospects of eventual economic extraction” pit and to generate the resource statement.

14.9 Variography

Variograms were created for all Indicator bins and for each domain for gold and for each domain for silver. The orientation of the variography remains the same for each gold grade bin and for silver as summarized in Table 14-7.

Table 14-7: Summary of Orientations for Interpolation

Metal	Domain	Rot-Z	Rot-X	Rot-Y
Au	1	290	0	0
	2	-35	-10	-10
	3	0	-20	-20
Ag	1	290	0	0
	2	-35	0	0
	3	-35	0	0

Cut-off bins for gold were established so that each bin contains approximately the same gold metal content.

Correlogram parameters are summarized in Table 14-8 and Table 14-9 for gold and silver respectively.

Table 14-8: Summary of Correlogram Parameters for Au

Dom	Ind	Cut-off	C0	C1	Ranges – Spherical - 1				Ranges – Spherical - 2		
					C2	Y (Major)	X (Minor)	Z (Vert)	Y (Major)	X (Minor)	Z (Vert)
1	1	0.003	0.2	0.35	0.45	75	50	40	720	300	270
	2	0.146	0.2	0.35	0.45	75	50	40	720	300	270
	3	0.323	0.2	0.35	0.45	75	50	40	720	300	270
	4	0.522	0.25	0.35	0.4	60	50	40	650	280	270
	5	0.765	0.3	0.4	0.3	60	50	40	650	280	220
	6	1.075	0.3	0.4	0.3	40	30	30	500	250	220
	7	1.496	0.55	0.25	0.2	40	30	20	400	220	180
	8	2.133	0.7	0.2	0.1	20	20	10	250	180	150
	9	3.786	0.9	0.1	0	30	10	5			
2	1	0.003	0.3	0.4	0.3	50	60	40	200	230	250
	2	0.257	0.3	0.4	0.3	50	60	40	200	230	250
	3	0.478	0.5	0.3	0.2	50	60	40	200	230	250
	4	0.765	0.5	0.3	0.2	30	60	40	160	200	250
	5	1.185	0.5	0.3	0.2	20	20	25	120	130	150
	6	1.893	0.7	0.2	0.1	30	30	35	120	130	140
	7	3.350	0.8	0.15	0.05	50	30	50	90	100	110
	8	6.241	0.9	0.1	0	45	25	60			
	9	14.244	0.95	0.05	0	30	25	60			
3	1	0.003	0.3	0.4	0.3	50	60	40	200	230	250
	2	0.212	0.3	0.4	0.3	50	60	40	200	230	250
	3	0.433	0.5	0.3	0.2	50	60	40	200	230	250
	4	0.766	0.5	0.3	0.2	30	60	40	160	200	250
	5	1.296	0.5	0.3	0.2	20	20	25	120	130	150
	6	2.244	0.7	0.2	0.1	30	30	35	120	130	140
	7	4.272	0.8	0.15	0.05	50	30	50	90	100	110
	8	9.143	0.9	0.1	0	45	25	60			
	9	25.165	0.9	0.1	0	45	25	60			

Table 14-9: Summary of Correlogram Parameters for Ag

Dom	C0	C1	C2	C3	Ranges – Spherical - 1			Ranges – Spherical - 2			Ranges – Spherical - 3		
					Y (Major)	X (Minor)	Z (Vert)	Y (Major)	X (Minor)	Z (Vert)	Y (Major)	X (Minor)	Z (Vert)
1	0.2	0.5	0.2	0.1	50	30	30	170	80	80			
2	0.3	0.6	0.1		30	30	20	180	150	220	250	220	200
3	0.3	0.6	0.1	30	30	20	180	150	220				

Search distances for gold and silver are provided in Table 14-10.

Table 14-10: Search Parameters for Au and Ag

Dom	Pass 1			Pass 2			Pass 3			Pass 4		
	Y (Major)	X (Minor)	Z (Vert)	Y (Major)	X (Minor)	Z (Vert)	Y (Major)	X (Minor)	Z (Vert)	Y (Major)	X (Minor)	Z (Vert)
1	30	20	10	50	35	30	125	75	65	500	300	270
2	15	20	30	20	25	50	100	115	125	200	230	250
3	15	20	30	20	25	50	100	115	125	200	230	250

The searches allowed sharing of composite values between Domains 1 and 2 (soft boundary), with a hard boundary between Domain 3, east of the major north–south-trending fault.

14.10 Classification of Mineral Resources

Blocks were assigned preliminary classifications based on the average distances to at least two drill holes as summarized in Table 14-11.

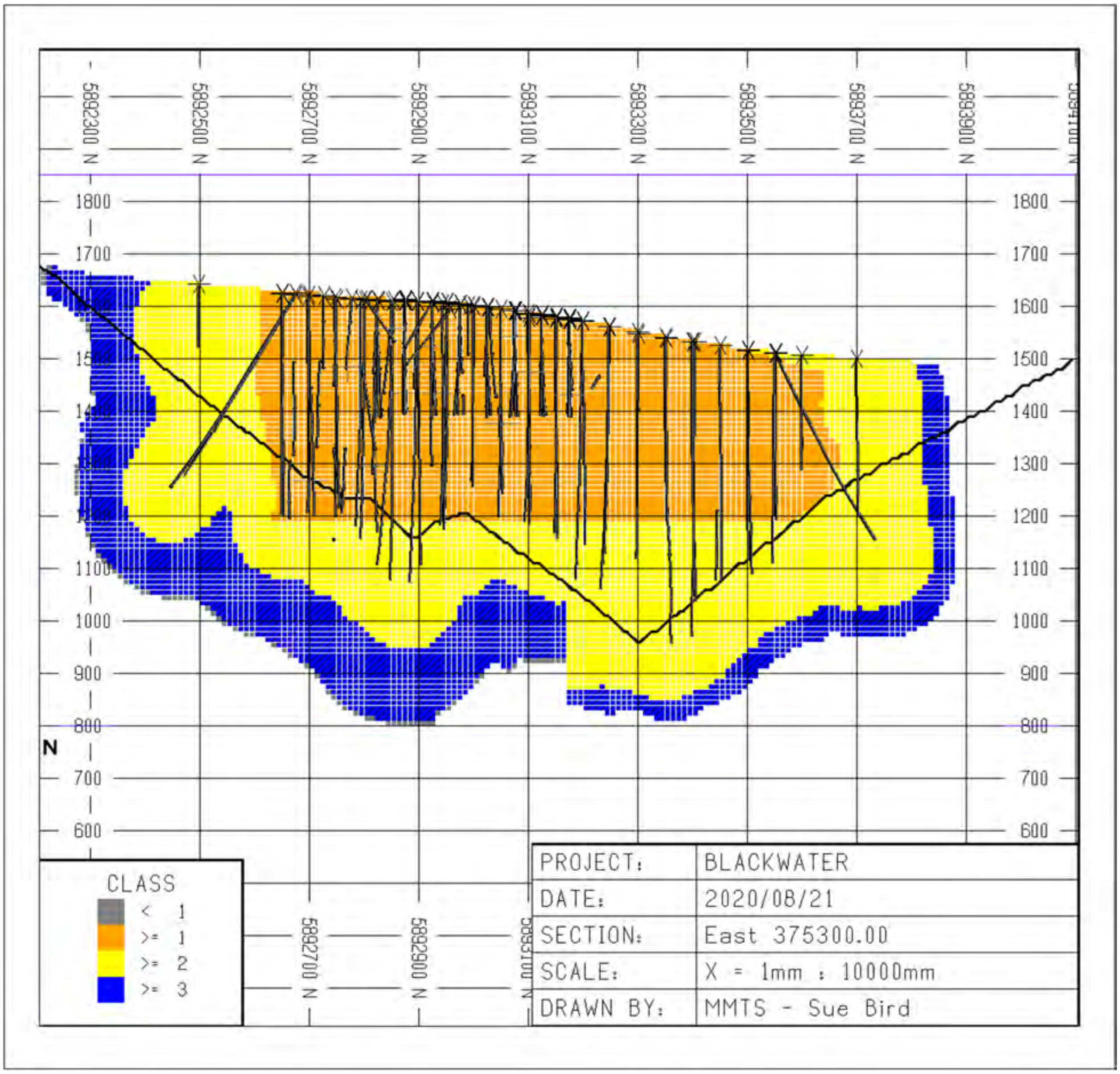
Table 14-11: Summary of Initial Classification Parameters

Class	Domain		
	1 (distance in m)	2 (distance in m)	3 (distance in m)
Measured	50	30	30
Indicated	260	100	100
Inferred	All other blocks interpolated with Au		

A solid shape encompassing the volume of blocks that were predominately classified as Measured was created with all blocks inside the shape given a final classification of Measured, and blocks outside the shape assigned as Indicated or Inferred based on the distance criteria in Table 14-11.

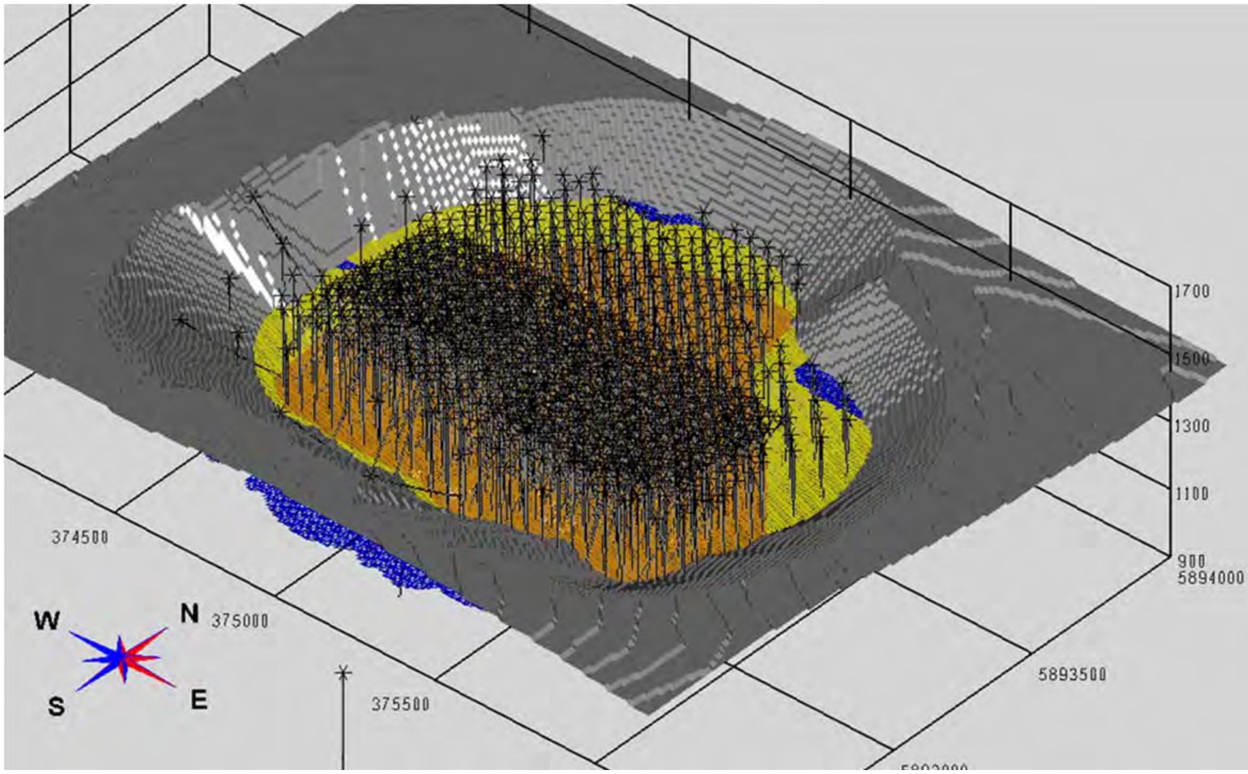
Figure 14-6, which is a north–south section through the centre of the deposit and Figure 14-7, which is a three-dimensional image, illustrate the final block classification and show the drill hole density. The drill hole spacing to a depth of about 1,400 m is 25 m and to a depth of 1,200 m is about 50 m, which supports the central portion of the resource pit being classified as Measured.

Figure 14-6: Illustration of Classification, DH Density and Resource Pit – 375300N



Note: Figure prepared by Moose Mountain, 2020. Class 1: Measured Resource; Class 2: Indicated Resource, Class 3: Inferred Resource.

Figure 14-7: Three Dimensional View of the Confidence Classification at elev. =1350 (mid pit), the Drill Pattern, and the Resource Pit



Note: Figure prepared by Moose Mountain, 2020. Class 1: Measured Resource; Class 2: Indicated Resource, Class 3: Inferred Resource.

14.11 Model Validation

The capping, modelling methods, and search parameters were chosen so that the final interpolated grades closely match the de-clustered composite data (using a nearest-neighbour (NN) model) while showing appropriate smoothing.

In order to perform appropriate validations, a NN model was created in order to compare the de-clustered composites to the modelled grades. To validate the amount of smoothing in the model, the NN model was corrected for block size using an indirect lognormal theoretical correction, based on the global variogram parameters and mean grades for each domain.

14.11.1 Global Grade Validation

Resource validation to ensure there was no global bias compared NN grades to those of the final grade interpolation at zero cut-off.

Table 14-12 summarizes this comparison by domain, illustrating that the difference in gold grades by domain is within 4% overall. For silver, the comparison shows mean modelled grades within 2.6% for all domains.

Table 14-12: Summary of Model Grade Comparison with De-Clustered Composites by Domain

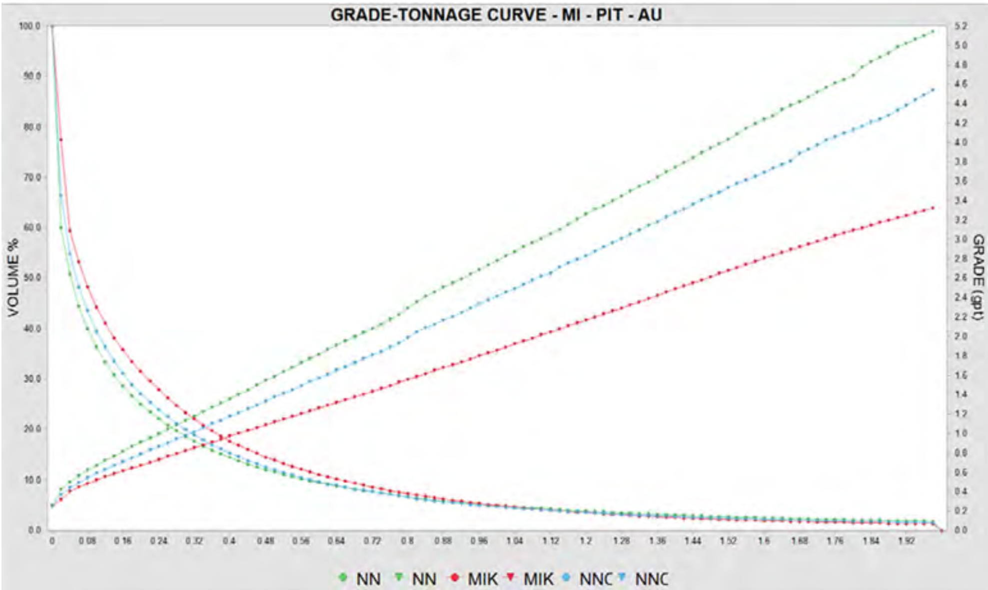
	Parameter	**De-Cluster Au Composites (NN)**				**Au - MIK**			
		1	**2**	**3**	**ALL**	**1**	**2**	**3**	**All**
Au	Num samples	202,692	23,8675	167,359	608,726	189,523	238,667	167,353	595,543
	Num missing	4,139	6,842	19,427	30,408	17,308	6,850	19,433	43,591
	Min (g/t)	0.003	0.003	0.003	0.003	0.001	0.001	0.001	0.001
	Max (g/t)	45.9	76.8	99	99	3.929	13.965	16.771	16.771
	Wtd mean (g/t)	0.1819	0.3158	0.265	0.2573	0.1797	0.2934	0.2581	0.2473
	Weighted CV	2.9479	4.8225	5.2657	4.8294	1.7694	2.0026	2.0593	2.0308
	Difference (1-NN/MIK)				-1.2%		-7.6%	-2.7%	-4.0%
	Parameter	**De-Cluster Ag Composites (NN)**				**Ag - OK**			
		1	**2**	**3**	**ALL**	**1**	**2**	**3**	**All**
Ag	Num samples	178,489	221,930	161,713	340,202	202,696	238,675	167,359	370,055
	Num missing	28,342	23,587	25,073	53,415	4,135	6,842	19,427	23,562
	Min (g/t)	0.1	0.1	0.1	0.1	0.1	0.1	0.1	0.1
	Max (g/t)	615.5	999	999	999	303.2	374.1	444.8	444.8
	Wtd mean (g/t)	3.85	2.51	4.2	4.02	3.63	2.47	4.27	3.92
	Weighted CV	3.62	3.21	4.43	4.06	2.39	2.05	2.41	2.41
	Difference (1-NN/MIK)				-6.1%		-1.6%	1.6%	-2.6%

Note: Num = number, Wtd = weighted, CV = co-efficient of variation.

14.11.2 Grade–Tonnage Curves

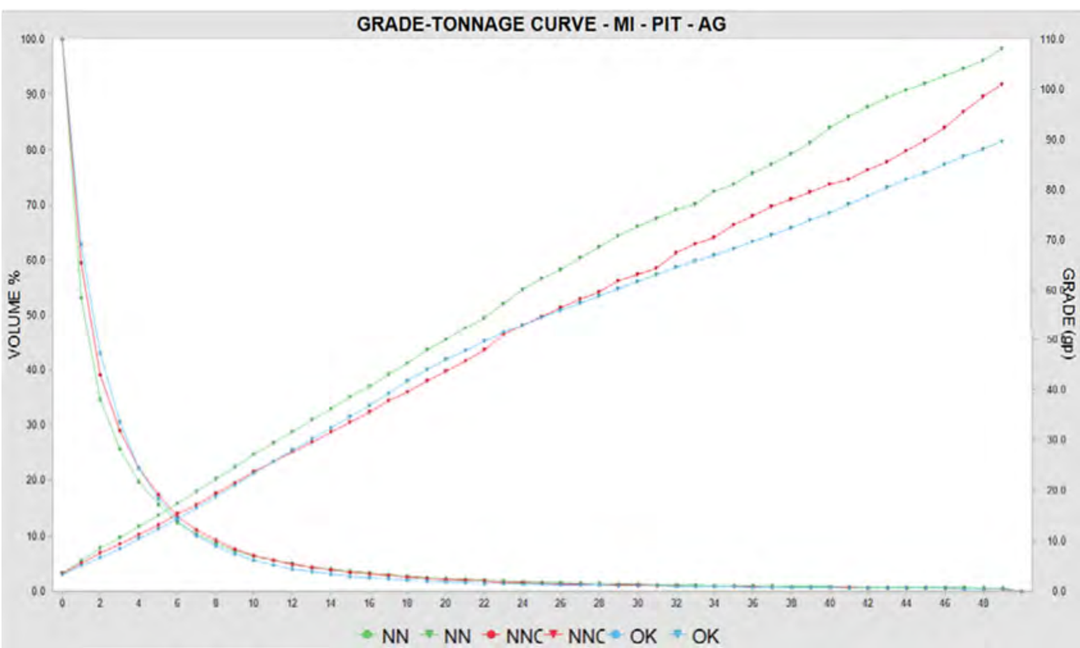
Grade–tonnage curves were created to compare the Au-MIK and Ag-OK interpolated grades with de-clustered composite grades. The de-clustered composites were corrected for the volume–variance effect by applying an indirect lognormal correction to the NN grades. This correction applies a factor to reduce the variance based on the block size (which is similar to the selective mining unit or SMU) in order to ensure that the modelled grades have had appropriate smoothing applied. Figure 14-8 and Figure 14-9 illustrate this comparison for gold and silver respectively, showing increased smoothing (reduced grades and increased tonnage) compared to the uncorrected NN grade curves, but a similar distribution compared to the theoretical NN–indirect lognormal correction grades.

Figure 14-8: Grade–Tonnage Curve Comparison for Au – MI within the Resource Pit



Note: Figure prepared by Moose Mountain, 2020

Figure 14-9: Grade–Tonnage Curve Comparison for Ag – MI within the Resource Pit



Note: Figure prepared by Moose Mountain, 2020

14.11.3 Visual Comparisons

Further validation on local grade estimation was completed through visual comparisons of the modelled grades with the assay and composite grades in section, plan and through three-dimensional checks. Figure 14-10 to Figure 14-17 illustrate the block grades and composite grades in north–south cross-sections throughout the area of the resource pit. The resource pit is illustrated on each section. Figure 14-10 through Figure 14-13 are sections for gold grade comparisons and Figure 14-14 through Figure 14-17 are the same sections comparing the silver grades. Both gold and silver grades show similar grade distributions and values throughout the model to that of the drill hole data. On all sections, the drill hole data shown is ±10 m of the section, also illustrating the close drill hole spacing throughout the deposit.

Figure 14-10: Au Grade – Model Compared to Composite – 37500E

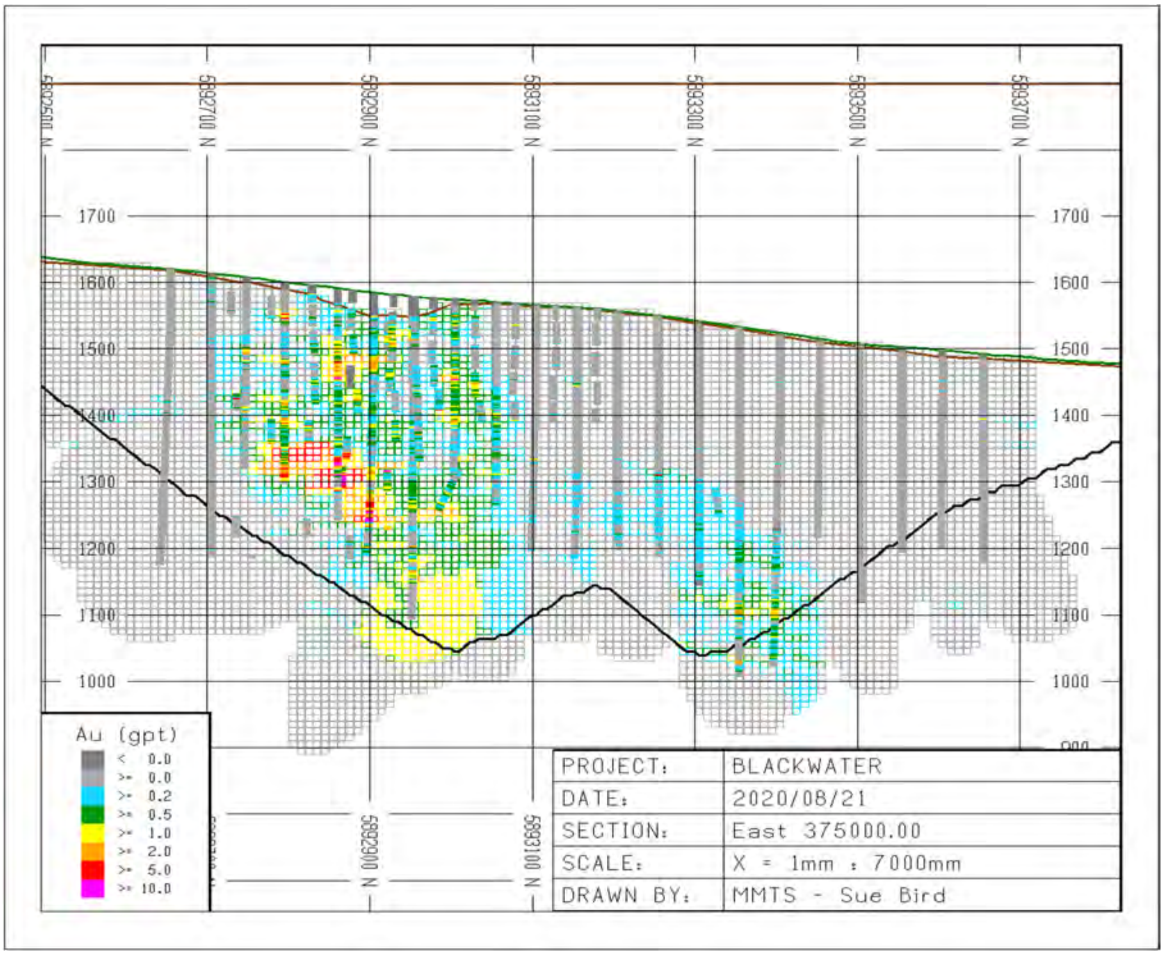


Figure 14-11: Au Grade – Model Compared to Composite – 375300E

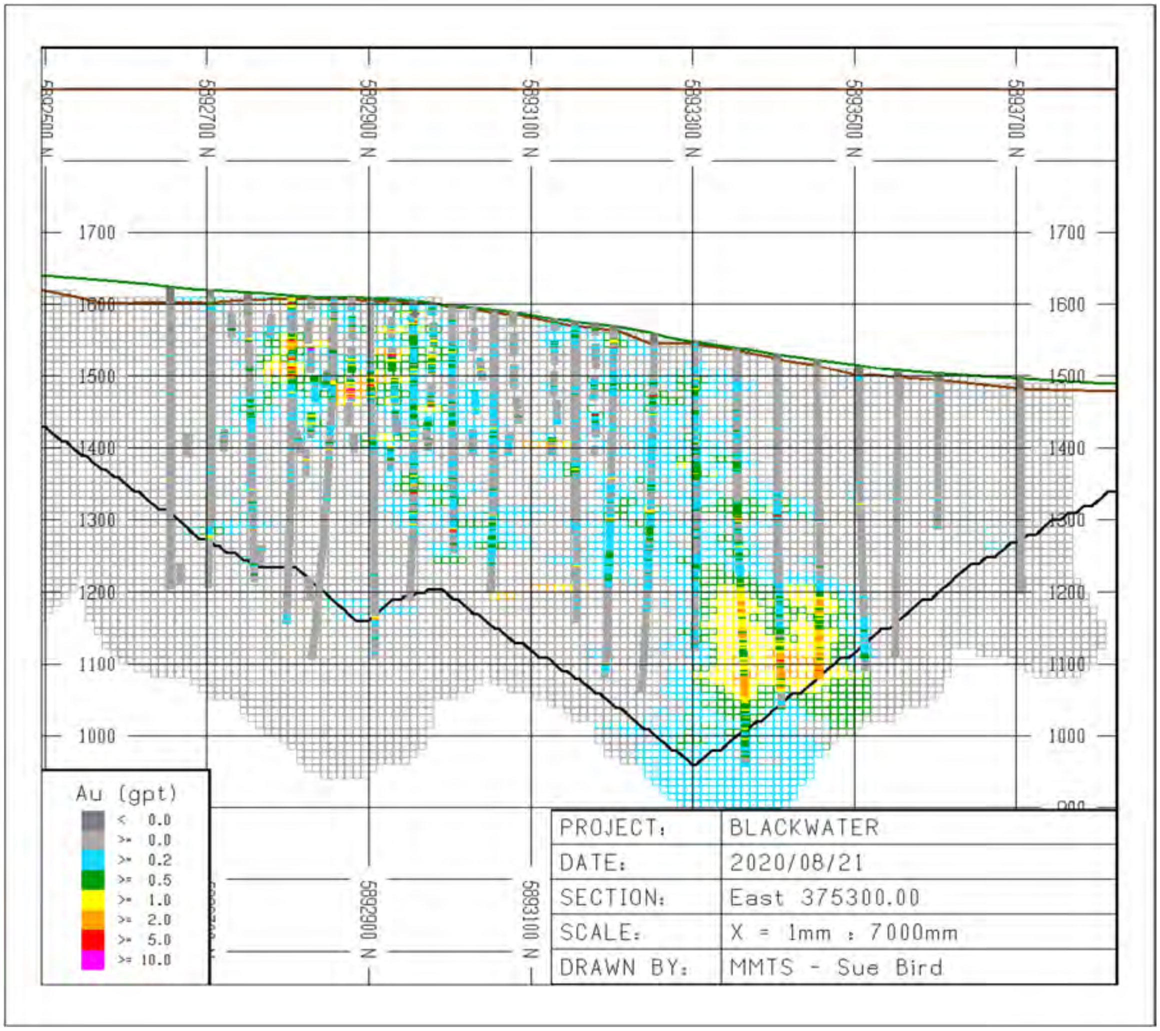


Figure 14-12: Au Grade – Model Compared to Composite – 375500E

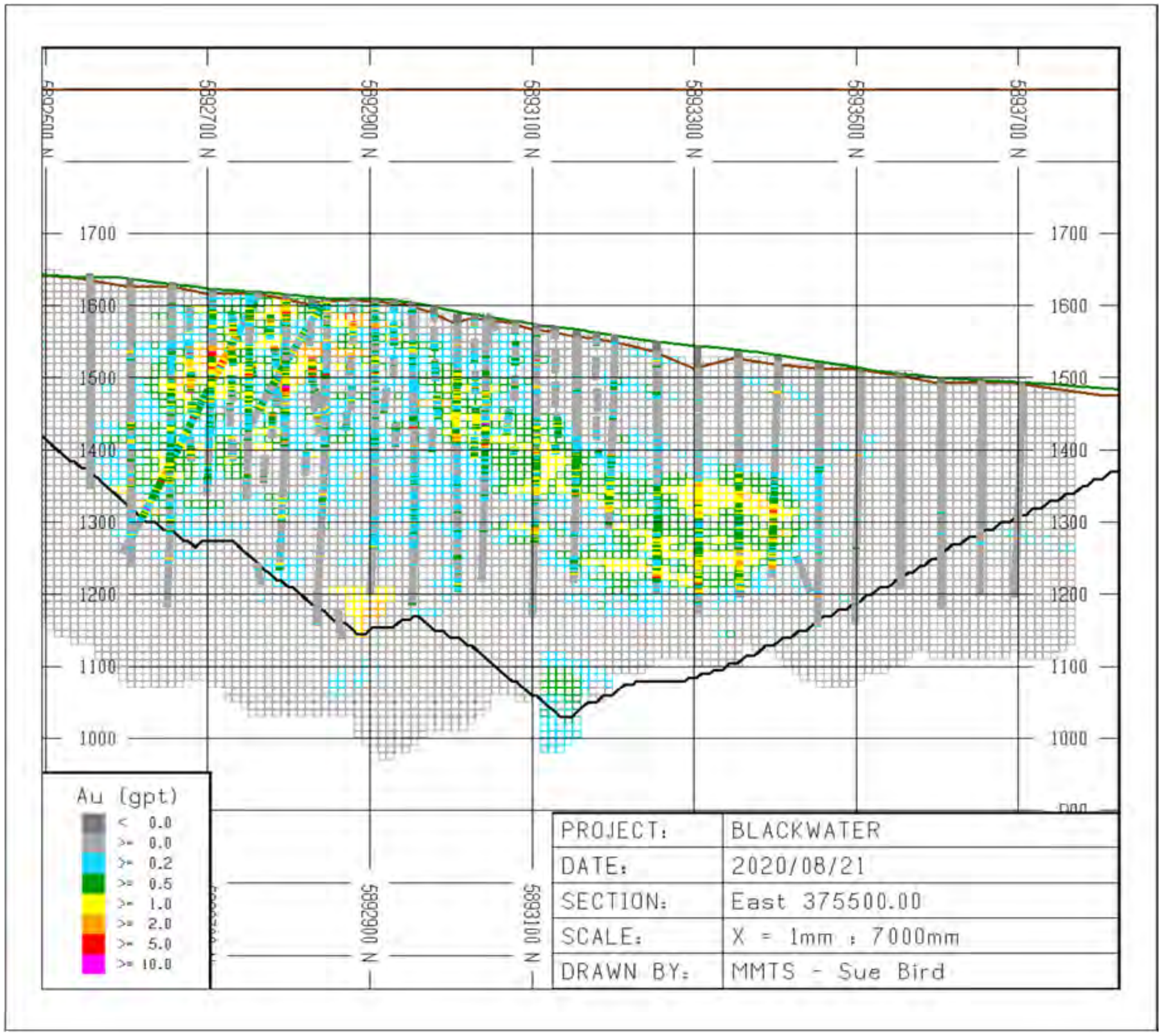


Figure 14-13: Au Grade – Model Compared to Composite – 375800E

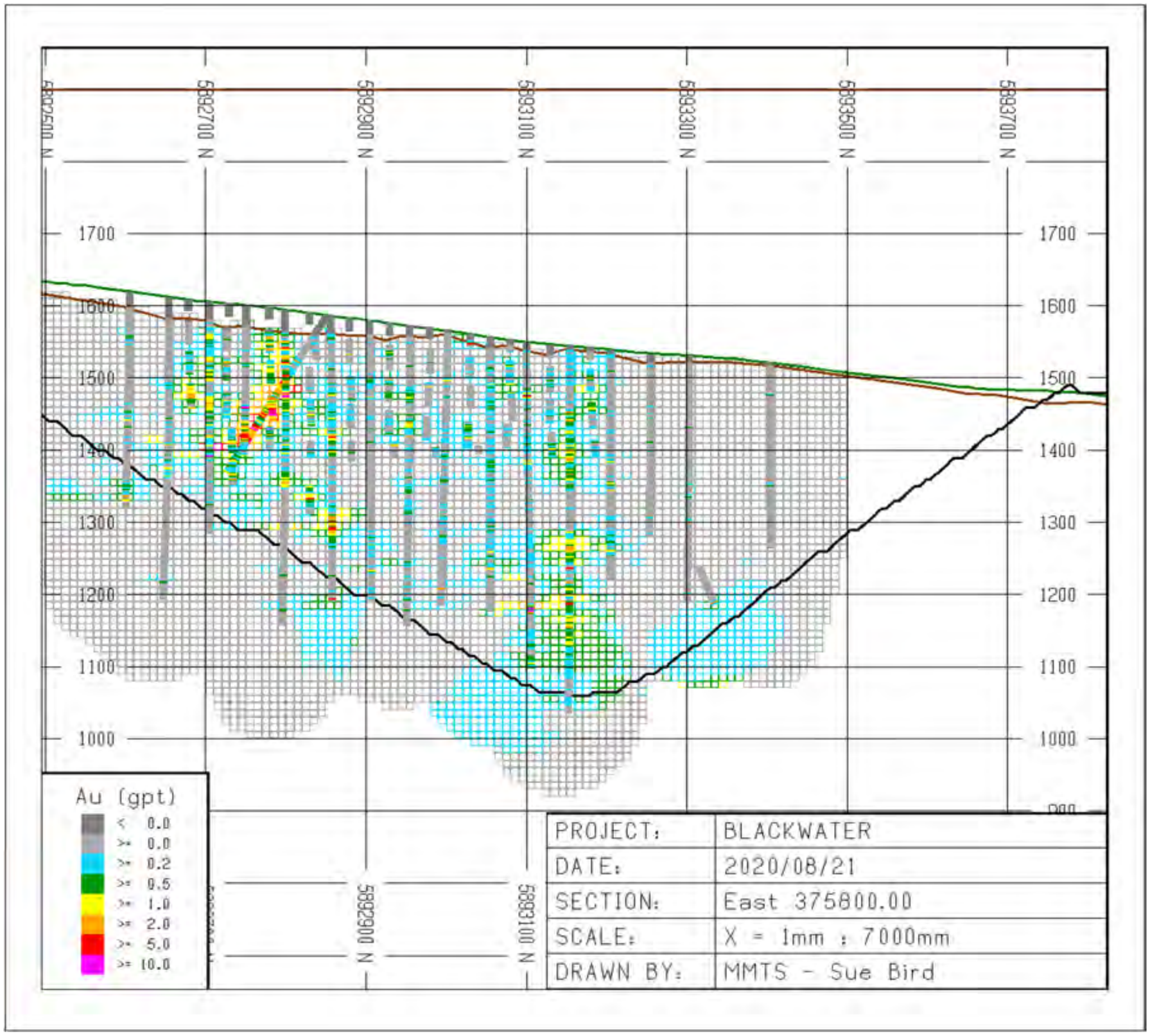


Figure 14-14: Ag Grade – Model Compared to Composite – 375000E

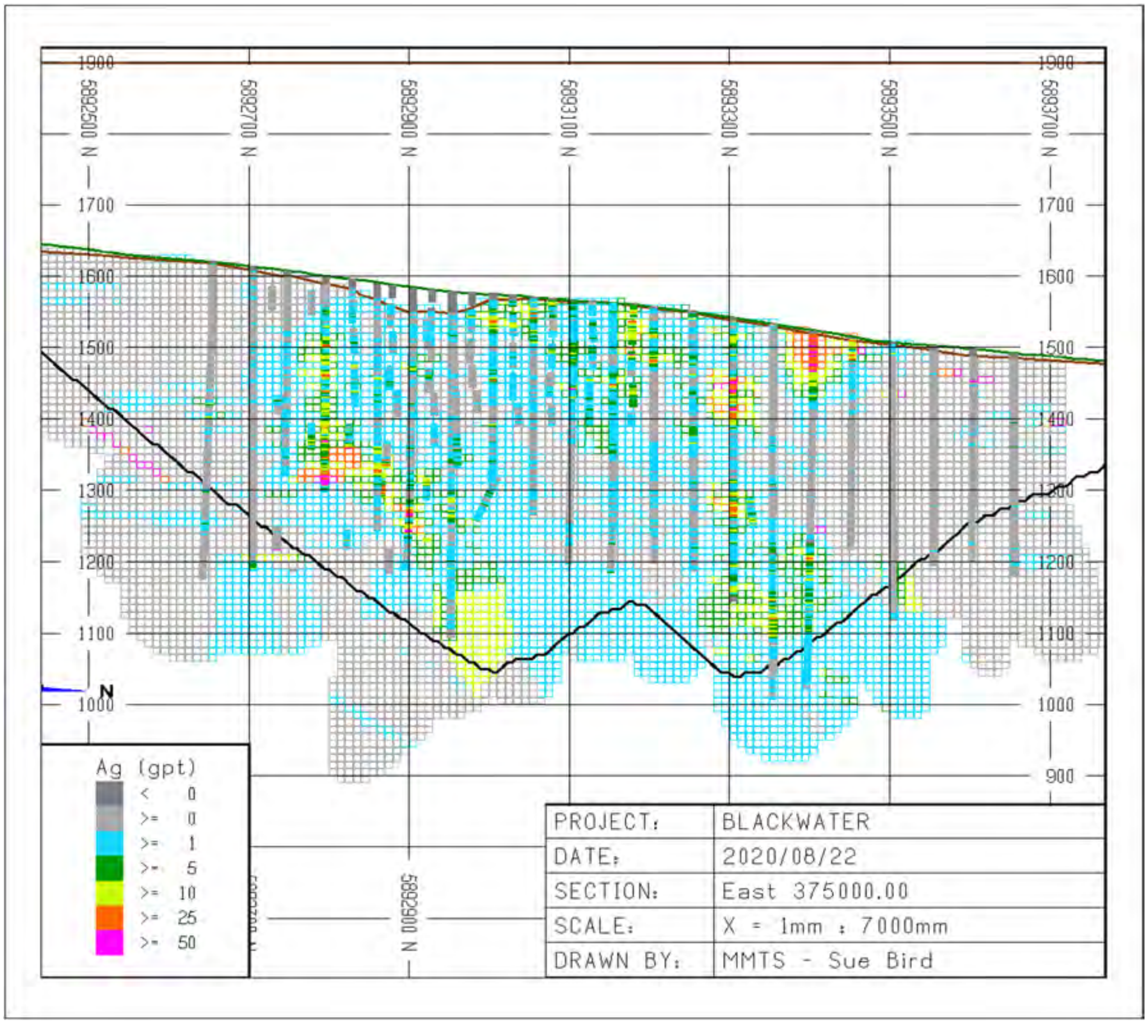


Figure 14-15: Ag Grade – Model Compared to Composite – 375300E

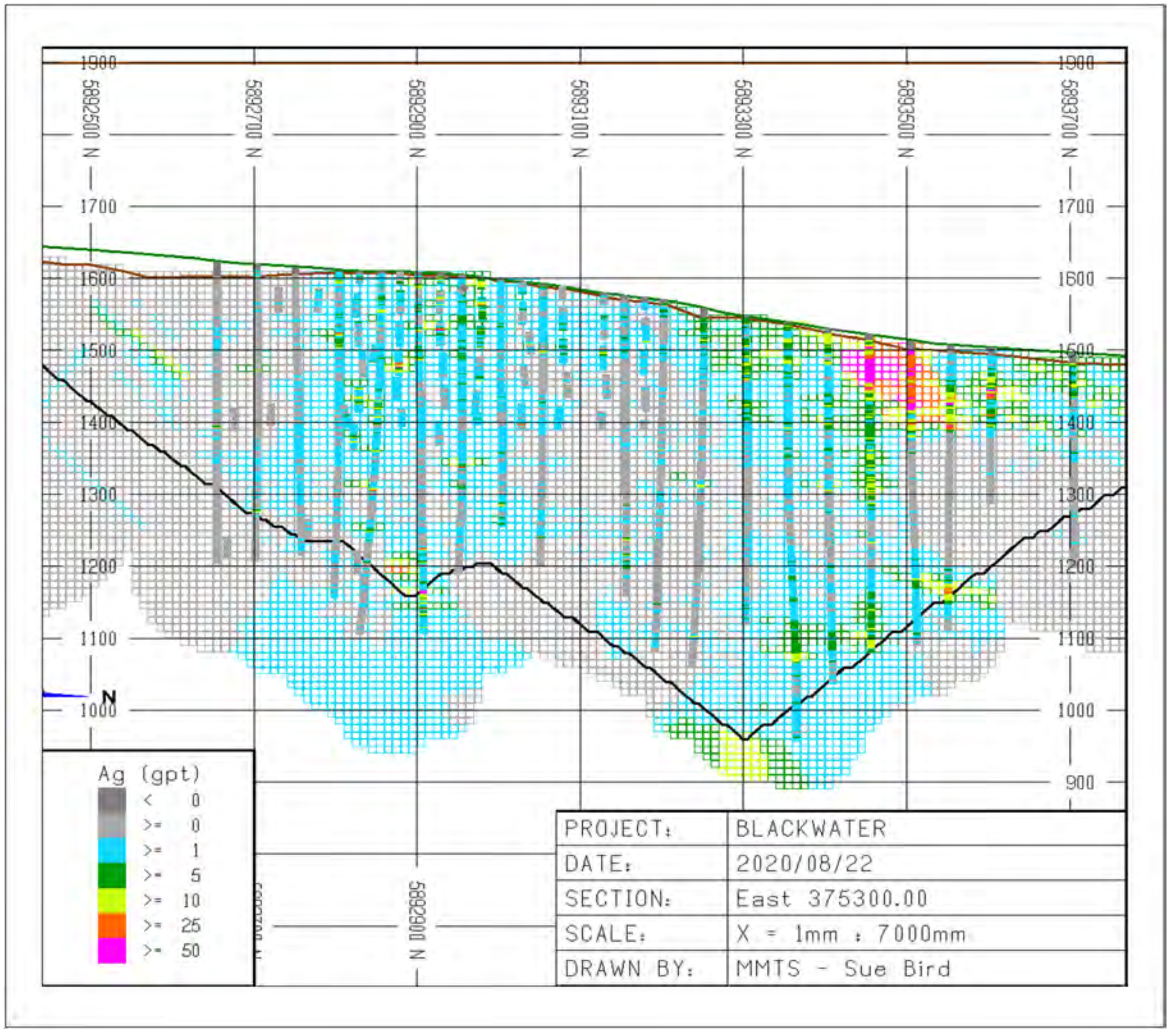


Figure 14-16: Ag Grade – Model Compared to Composite – 375500E

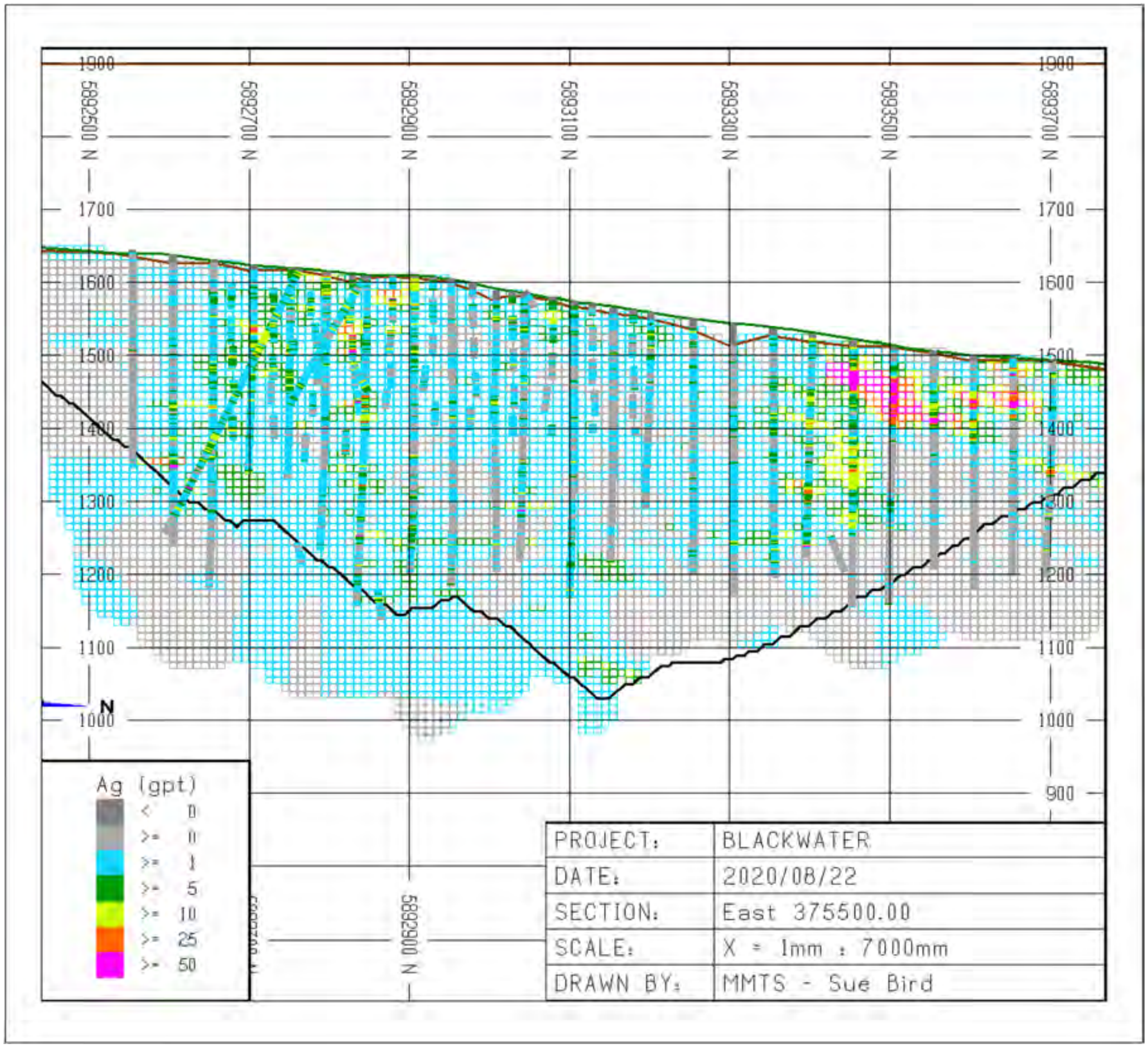
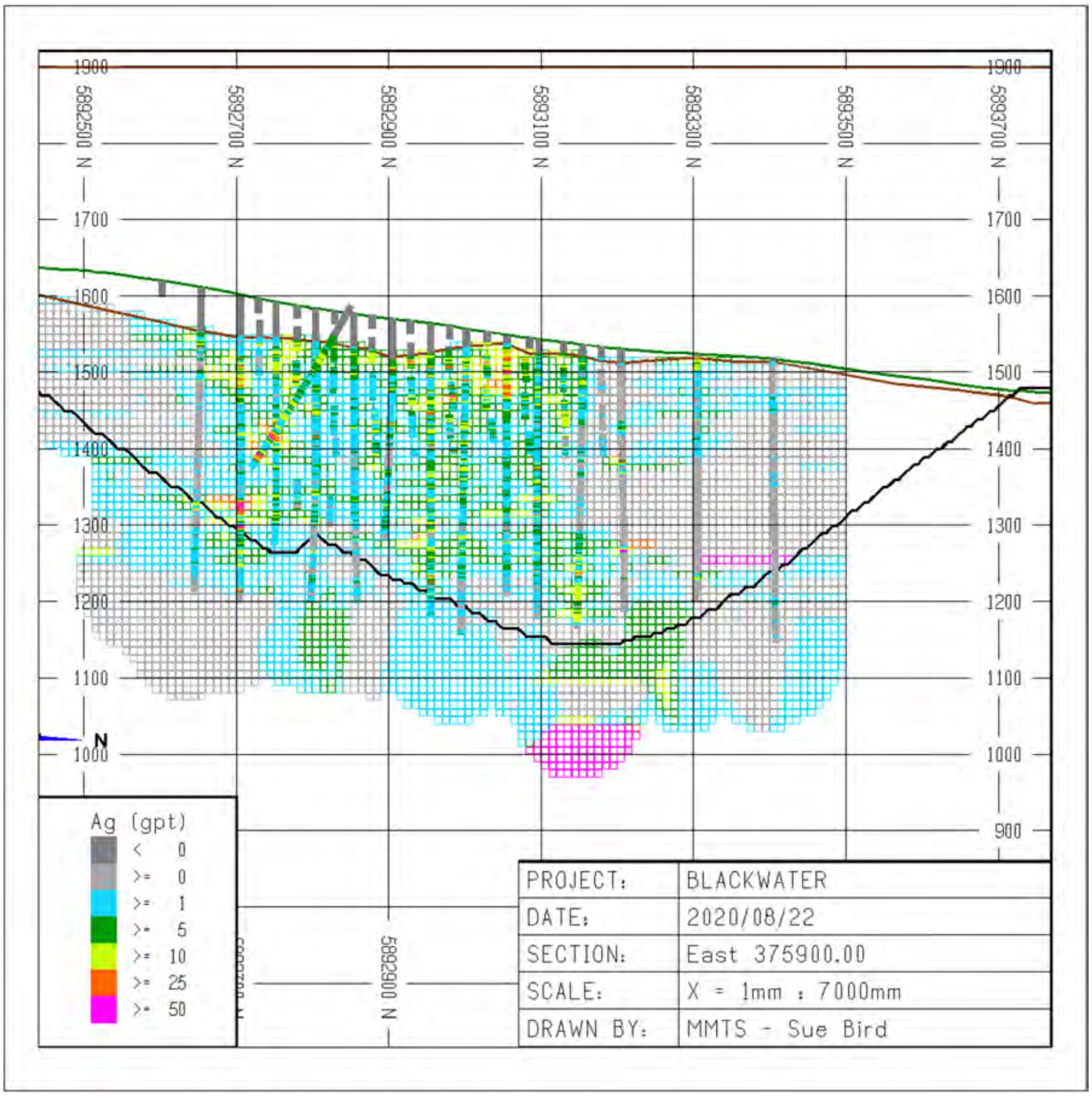


Figure 14-17: Ag Grade – Model Compared to Composite – 375900E



14.12 Reasonable Prospects for Eventual Economic Extraction

An open pit created using Lerchs–Grossmann (LG) pit optimization was done on a series of pits with varying price assumptions. The base case price, cost, smelter terms, foreign exchange and recoveries are summarized in Table 14-13.

Table 14-13: Summary of Base Case Economic Inputs

Parameter	Units	Value
Gold price	US$/oz	1,400.00
Silver price	US$/oz	15.00
Foreign exchange rate	($US:C$)	0.75
Gold payable	%	99.9
Silver payable	%	95.0
Gold offsites	US$/oz	8.50
Silver offsites	US$/oz	0.25
Royalty	%	1.5%
Net smelter gold price	C$/g	58.79
Net smelter silver price	C$/g	0.59
Gold process recovery	%	93
Silver process recovery	%	55

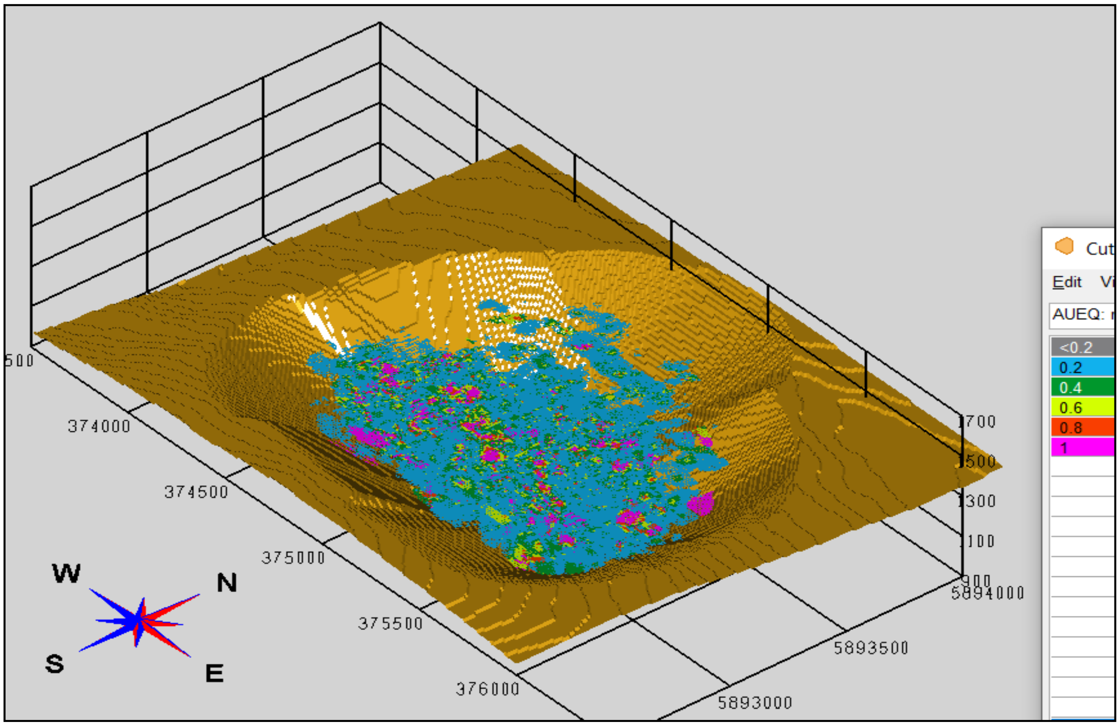
The resulting gold equivalent (AuEq) equation used is:

- $AuEq = \text{A}u + 0.006 * AgGrade$.

The base case cut-off grade of 0.20g/t AuEq is considered appropriate using the assumptions above. The final resource pit has been based on the LG pit using 135% Au price and 109% Ag price in order to ensure that the resource pit will be large enough to encompass any potential reserves and is representative of an eventual economic extraction shape.

The resulting pit shape for “reasonable prospects of eventual economic extraction” is illustrated in Figure 14-18 with the AuEq grade for all blocks above cut-off.

Figure 14-18: Three Dimensional View of the Resource Pit and AuEq Blocks Above 0.2g/t AuEq



Note: Figure prepared by Moose Mountain, 2020

For the LG pit optimizations, the costs given in Table 14-14 were used. Constant pit slopes at 40º were used for the resource pit.

Table 14-14: Costs Used for Lerchs–Grossmann Resource Pit

Cost	Units	Value
Mineralization mining costs	$C/t	C$2.50
Waste mining costs	$C/t	C$2.30
*Bench incremental mining costs *starting at 1,500 m: $0.025/t is added for every 10 m elevation drop	$C/t	C$0.025
Processing costs	$C/t of mineralization	C$12.00
G&A costs	$C/t of mineralization	C$4.50

14.13 Factors That May Affect the Mineral Resource Estimate

Areas of uncertainty that may materially impact the Mineral Resource estimate include:

- Commodity price assumptions;
- Metal recovery assumptions;
- Mining and processing cost assumptions.

There are no other known factors or issues known to the QP that materially affect the estimate other than normal risks faced by mining projects in the province in terms of environmental, permitting, taxation, socio-economic, marketing, and political factors.

14.14 Risk Assessment

A description of potential risk factors is given in Table 14-15 along with either the justification for the approach taken or mitigating factors in place to reduce any risk.

Table 14-15: List of Risks and Mitigations/Justifications

#	Description	Justification/Mitigation
1	Classification criteria	The deposit is extremely well drilled off to 1,200 m elevation.
2	Gold price assumption	Conservative for cut-off grade, reasonable for pit size.
3	Capping	Cumulative probability plots, swath plots and grade–tonnage curves show model validates well with composite data throughout the grade distribution.
4	Grade continuity and MIK Interpolation	Grade control model based on assays of 561 RC holes of 48–72 m length for total of 33,216 m illustrates conservatism in the modelled Au tonnage and grades. The potential drill hole bias was accounted for in this analysis.
5	Processing and mining costs	Same costs are used as for the mine planning pits, and are therefore conservative for a "reasonable prospect of eventual economic extraction" assessment.

15.0 MINERAL RESERVE ESTIMATES

15.1 Introduction

The Mineral Reserves for Blackwater are a subset of the Measured and Indicated Mineral Resources, described in Section 14, as supported by the 2024 Expansion Study open pit (LOM) plan, described in Section 16.

Proven and Probable Mineral Reserves are modified from Measured and Indicated Mineral Resources. Inferred Mineral Resources are set to waste.

15.2 Mineral Reserves Statement

Mineral Reserves are estimated using the CIM 2019 Best Practices Guidelines and are classified using the 2014 CIM Definition Standards.

Mineral Reserves are reported at the point of delivery to the primary crusher, and have an effective date of 10 September, 2021.

The Qualified Person for the estimate is Mr. Marc Schulte, P.Eng., a member of Moose Mountain.

Proven and Probable Mineral Reserves are summarized in Table 15-1. Inferred Mineral Resources are set to waste.

Table 15-1: Mineral Reserves Statement

Confidence Category	Tonnage (Mt)	Gold Grade (g/t Au)	Contained Gold Metal (Moz Au)	Silver Grade (g/t Ag)	Contained Silver Metal (Moz Ag)	AuEq Grade (g/t)
Proven	325.1	0.74	7.8	5.8	60.4	0.78
Probable	9.2	0.80	0.2	5.8	1.7	0.83
Total Proven and Probable	***334.3***	***0.75***	***8.0***	***5.8***	***62.2***	***0.78***

Notes to accompany Mineral Reserves table:

1. Mineral Reserves are reported at the point of delivery to the primary crusher, inclusive of mining loss and dilution, using the 2014 CIM Definition Standards, and have an effective date of 10 September, 2021.
2. Mineral Reserves are supported by the 2024 Expansion Study life of mine plan.
3. The Qualified Person for the estimate is Mr. Marc Schulte, P.Eng., a member of Moose Mountain Technical Services.
4. Mineral Reserves are reported at a net smelter return (NSR) cut-off of C$13.00/t. The NSR cut-off covers processing costs of C$9.00/t, administrative (G&A) costs of C$2.50/t and stockpile rehandle costs of C$1.50/t. The NSR cut-off assumes US$1,400/oz Au and US$15/oz Ag at a currency exchange rate of 0.75 US$ per C$; 99.9% payable gold; 95.0% payable silver; US$8.50/oz Au and US$0.25/oz Ag offsite costs (refining, transport, and insurance); a 1.5% NSR royalty; and uses a 93% metallurgical recovery for gold and 55% recovery for silver.
5. Gold equivalent (AuEq) values are calculated using the same parameters as NSR listed above, resulting in the following equation: AuEq = Au g/t + (Ag g/t x 0.006).
6. Numbers have been rounded as required by reporting guidelines

The open pit is based on the results of Pseudoflow sensitivity analysis, and then designed into detailed pit phases for production scheduling purposes. The Mineral Reserves by pit phase are shown in Table 15-2.

Table 15-2 Mineral Reserves within Designed Pit Phases

Pit Phase	Pit Name	Mill Feed (Mt)	Gold Grade (g/t)	Silver Grade (g/t)	Waste (Mt)	Strip Ratio (t/t)
Construction borrow pit	P650	0.2	0.19	25.1	11.7	-
Starter phase	P651i	32.7	1.00	6.3	30.5	0.9
East pushback 1	P652i	13.4	0.81	6.8	19.4	1.4
East pushback 2	P653i	32.1	0.95	6.3	49.8	1.6
West pushback	P654i	32.1	0.84	4.3	56.3	1.8
North pushback 1	P655i	43.3	0.64	3.8	44.6	1.0
North pushback 2	P656i	27.3	0.47	13.2	55.9	2.0
Southeast pushback	P657i	66.3	0.64	5.3	148.1	2.2
Ultimate pushback southwest	P658i	86.9	0.75	4.7	256.7	3.0
Total Open Pit	*P658*	*334.3*	*0.75*	*5.8*	*672.9*	*2.0*

Note: An NSR cut-off of $13.00/t is applied. Mined tonnes and grade include operational modifying factors such as loss and dilution. Mineral Reserves in this table are not additive to the Mineral Reserves in Table 15-1. Footnotes to Table 15-1 also apply to this table.

15.3 Factors that May Affect the Mineral Reserves

Mineral Reserves are based on the engineering and economic analysis described in Sections 16 to 22 of this Report. Changes in the following factors and assumptions may affect the Mineral Reserve estimate:

- Metal prices and foreign exchange rates;
- Interpretations of mineralization geometry and continuity of mineralization zones;
- Geotechnical and hydrogeological assumptions;
- Changes to pit designs from those currently envisaged;
- Ability of the mining operation to meet the annual production rate;
- Changes to operating and capital cost assumptions;
- Mining and process plant recoveries;

- Ability to meet and maintain permitting and environmental license conditions and the ability to maintain the social license to operate.

15.4 QP Comments on Section 15

The Mineral Reserve estimates are based on the most current knowledge, permit status, and engineering constraints. The QP is of the opinion that the Mineral Reserves were estimated using industry-accepted practices.

16.0 MINING METHODS

The Mineral Reserves stated in Section 15 are supported by the open pit mine plan summarised in this section.

Open pit mine designs, mine production schedules and mine capital and operating costs were developed for the Blackwater deposit at a feasibility study level of engineering.

16.1 Key Mine Design Criteria

The following mine planning design inputs were used:

- Topography is based on a LiDAR survey of the region;
- Whole block resource block model on 10 m spacing in all three dimensions, with diluted gold and silver grades, SGs, and resource classifications;
- Inferred mineral resources are treated as waste rock with no economic value;
- Gold metallurgical process recovery of 93%, silver metallurgical process recovery of 55%;
- Open pits, stockpiles and haul roads are planned to fall within existing permitted disturbance areas.

16.2 Net Smelter Price, Net Smelter Return and Cut-off Grade

NSR was defined as the dollar value in a block in $/t, available to the local operation (i.e. inside the property gates). The NSR value accounts for in situ grades, process recoveries and the net smelter price (NSP). The NSPs were based on the market price for gold and silver and deducting all off-site costs (Table 16-1).

Table 16-1: Net Smelter Price Inputs

Description	Units	Values
Gold price	US$/oz.	$1,400
Silver price	US$/oz.	$15.00
US:Canadian exchange rate	US$/$CDN	0.75
Payable gold	%	99.9
Payable silver	%	95.0
Gold offsite costs	US$/oz	8.50
Silver offsite costs	US$/oz	0.25
Royalty	%	1.5

Using a gold market price of US$1,400/oz resulted in an NSP value of C$1,829/oz or C$58.79/g. Using a silver market price of US$15/oz resulted in an NSP value of C$18/oz or C$0.59/g.

The NSR C$/t of each block in the block model provided net revenue to the Blackwater Gold Mine economics, to cover the mining, processing, and any other attributed costs from the operation. The NSR in each block of the model was calculated using the following formula:

- NSR = NSP Gold ($/g) * Gold Grade (g/t) * Gold Process Recovery (%) + NSP Silver ($/g) * Silver Grade (g/t) * Silver Process Recovery (%).

With the NSR value calculated for each mineralized block in the 3D block model, the pit benches, sub-benches, or individual blocks were examined for their contribution to positive cash flow.

The cut-off was based on the calculated NSR. Low-grade resource blocks, internal in the pit design, must be mined to expose higher-grade blocks below them; therefore, they can still contribute to a positive cash flow if they have an in situ grade value that is greater than the incremental operating cost. Since the cost of mining from the pit is covered if the block needs to be mined as waste, then, if the NSR value is greater than the process and administrative costs, as well as additional costs to rehandle from a stockpile, the block can contribute positively on an incremental basis to the cash flow.

An economic mine planning NSR cut-off of C$13.00/t was selected. This cut-off covered the incremental production costs of processing, administration, and stockpile rehandle.

16.3 Mining Loss and Dilution

Whole block diluted gold and silver grades and tonnages were used for mine planning. Block sizes are on 10 m spacing in all three dimensions. It was estimated that the effects of dilution and loss introduced via mining operations were covered within the whole block measurements for tonnage and grade.

It was estimated that, in the range of the cut-off, the block model grades had a 17% reduction in gold grade compared to the de-clustered composite data. This comparison is described in Section 14, was used to validate the model, and included a correction to the grade–tonnage curve for volume–variance effects.

Edge dilution introduced from ore to waste contacts within the whole block model was estimated to be 9.5%. It is anticipated that the dilution introduced by using whole block tonnages and grades is sufficient to cover the estimated effects of mining operation dilution and loss.

16.4 Geotechnical Considerations

Pit designs were configured on 10 m bench heights, with berms placed every two benches, or double benching. Bench face angles, inter-ramp angles and bench widths were unique to each prescribed geotechnical domain.

Geotechnical domains were based on geotechnical conditions described by Knight Piésold (Knight Piésold, 2013a). Geotechnical conditions were estimated from geomechanical and hydrogeologic data collected during past site investigation programs, characterization of geology, rock mass structure, and rock mass quality, and analysis of kinematic and rock mass stability.

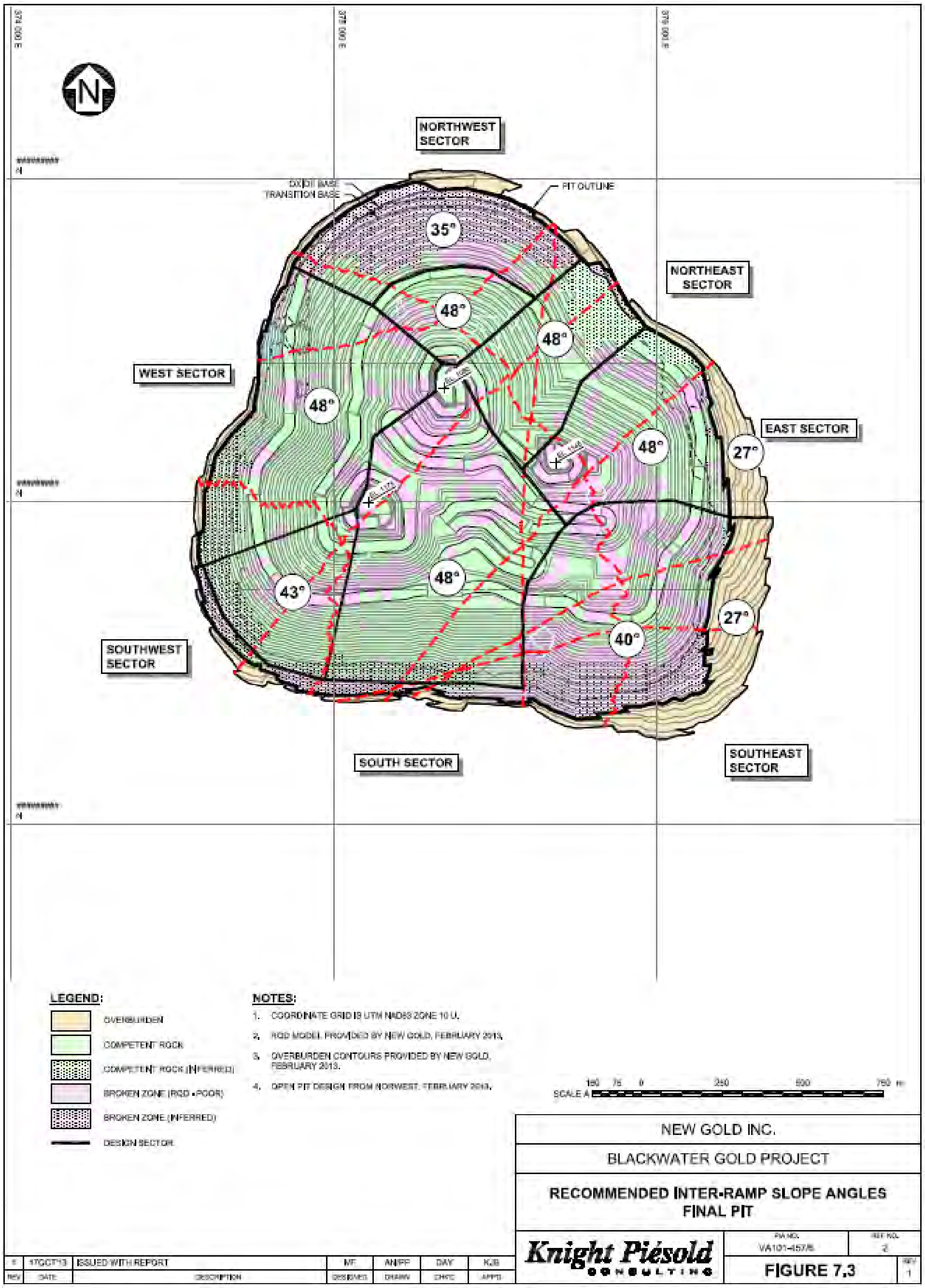
Three geotechnical domains were defined for the purposes of the slope stability analyses:

- Surficial material: glacial till deposits typically ranging from 5 to 20 m in thickness throughout the deposit area, which increases up to 110 m depth along the eastern side of the deposit;
- Broken zone: delineated within the rock mass RQD block model when the values are generally <40%;
- Competent rock: defined as all rock mass regions where the RQD is >40%.

Slope stability analyses were conducted for each pit design sector and indicate that the main geotechnical controls on pit slope design are adverse faults and fractured rock masses. The achievable slope geometry is largely controlled by the location and extent of the broken zone.

Geotechnical domains are illustrated in Figure 16-1, and pit slopes and configurations for the domains are shown in Table 16-2.

Figure 16-1: Open Pit Geotechnical Domains



Note: Figure prepared by Knight Piésold, 2013.

Table 16-2: Pit Slope and Configuration Inputs

Domain	Bench Height (m)	Bench Face Angle (º)	Inter-Ramp Angle (º)	Bench Width (m)
NW Lower	20	70	48	10.7
NW Upper	20	60	35	17.0
NE	20	70	48	10.7
E	20	70	48	10.7
SE	20	60	40	12.3
S	20	70	48	10.7
SW	20	70	43	14.2
W	20	70	48	10.7
OVB	20	25	20	12.1

Note: BFA = bench face angles; IRA = inter-ramp angles.

Additionally, a maximum inter-ramp height of 150 m in broken zones, and 200 m in competent rock, were maintained by including geotechnical berms.

In-pit haul roads and geotechnical berms were added to the pit slopes and flattened the inter-ramp angles out to a shallower overall slope in all domains. Geotechnical berms were placed so that a maximum inter-ramp height of 150 m in broken zones, and 200 m in competent rock, was maintained wherever in-pit ramps are not present.

The implementation of slope design requires effective slope depressurization, good-controlled blasting and excavation practices, and regular inspection and systematic slope monitoring.

An updated geotechnical interpretation was completed by Knight Piésold (Knight Piésold, 2022a) that includes refinement of the pit design sectors and recommended inter-ramp slope angles. Results of this updated interpretation have not been incorporated into the 2024 pit designs and mine engineering. Checks were completed using this updated interpretation, and potential changes to the pit designs would negligibly affect (<1%) the current Mineral Reserve estimate and not materially affect the strip ratio associated with the Mineral Reserve estimate (<5%). It is recommended that future mine engineering and planning incorporate this updated interpretation.

16.5 Hydrogeological Considerations

Water inflows to the Blackwater open pit will include both groundwater and surface water runoff. The contributions from groundwater will progressively increase as the pit extends below the groundwater table. The contributions from surface water will be direct precipitation into the pit and runoff from the limited contributing catchments around the pit excavation. The inflows from direct precipitation will increase with increasing pit area in conjunction with groundwater inflows as the pit increases in depth.

A combination of in-pit and perimeter pumping wells will be used for slope depressurization and pit dewatering. Pumping wells will typically be installed to a nominal depth of approximately 350 m below ground level and up to a maximum depth of 450 m below ground level depending on drilling conditions encountered.

The in-pit groundwater wells will target water removal from storage in the higher permeability zone and groundwater inflow to the higher permeability zone from the surrounding lower permeability bedrock. Perimeter dewatering wells will be established as needed to lower and extend the cone of depression (when natural drainage is not sufficient) to provide the required depressurization identified from the open pit wall stability analyses.

There will be approximately 12 dewatering wells spaced at 150–200 m intervals to achieve an adequate cone of depression to lower the groundwater level.

The maximum groundwater dewatering rates are estimated to be approximately 65 L/s as the open pit is expanded over the LOM.

In-pit water from surface runoff will be directed via in-pit ditching and grading towards actively progressing in-pit sumps. Skid-mounted diesel dewatering pumps will transfer this water via in-pit piping to a junction header near the pit rim. The flows will be directed to the plant site location.

16.6 Pit Optimization

The economic pit limits are determined using the Pseudoflow algorithm. This algorithm uses the ore grades and SG for each block of the 3D block model and evaluates the costs and revenues of the blocks within potential pit shells. The routine uses input economic and engineering parameters and expands downwards and outwards until the last increment is at break-even economics.

Additional cases were included in the analysis to evaluate the sensitivities of resources to strip ratio/topography and high-grade/low-grade areas of the deposit. The various cases or pit shells were generated by varying the input NSP values and comparing the resultant waste and mill feed tonnages and metal grades for each pit shell.

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

For each pit shell, an undiscounted cash flow (UCF) was generated based on the shell contents and the economic parameters listed in Table 16-1 and Table 16-3.

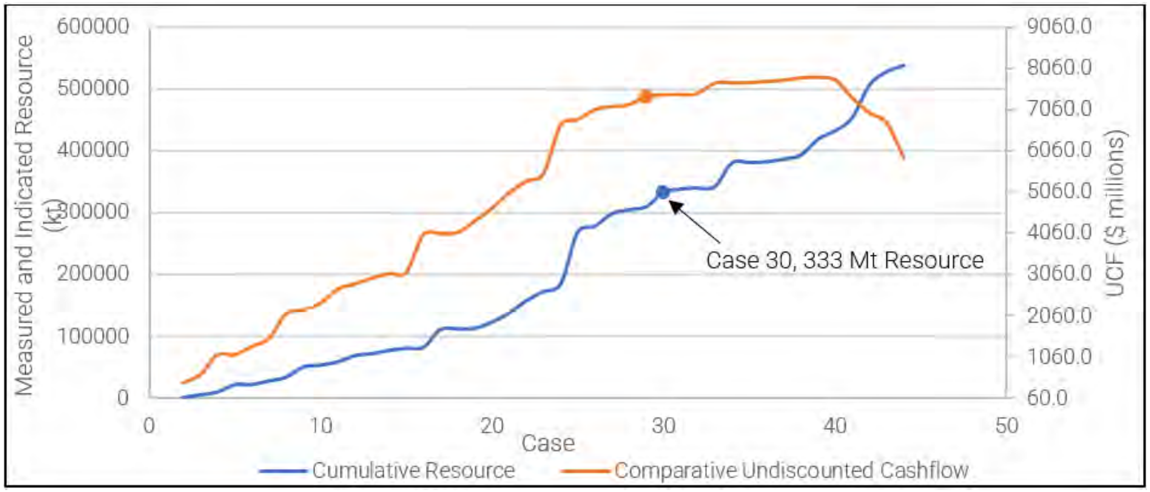
Table 16-3: Inputs into Pseudoflow Pit Optimization

Item	Parameter
Pit rim ore mining cost	$2.50/t, pit rim of 1,500 m
Pit rim waste mining cost	$2.00/t, pit rim of 1,500 m
Incremental bench haulage cost	$0.025 per every 10 m bench below pit rim
Processing cost	$10.00/t
G&A cost	$3.00/t
Gold process recovery	93%
Silver process recovery	55%

The UCFs for each case were compared to reinforce the selected point at which increased pit expansions do not materially increase the value. Note that the economics are only applied for comparative purposes to assist in the selection of an optimum pit shell for further mine planning; they do not reflect the actual financial results of the mine plan.

The pit shell generated from Case 30, whose comparative contents and UCF value are illustrated in Figure 16-2, was selected as the ultimate pit limit and was used for further mine planning as a target for detailed open pit designs with berms and ramps.

Figure 16-2: Pseudoflow Pit Shell Resource Contents by Case

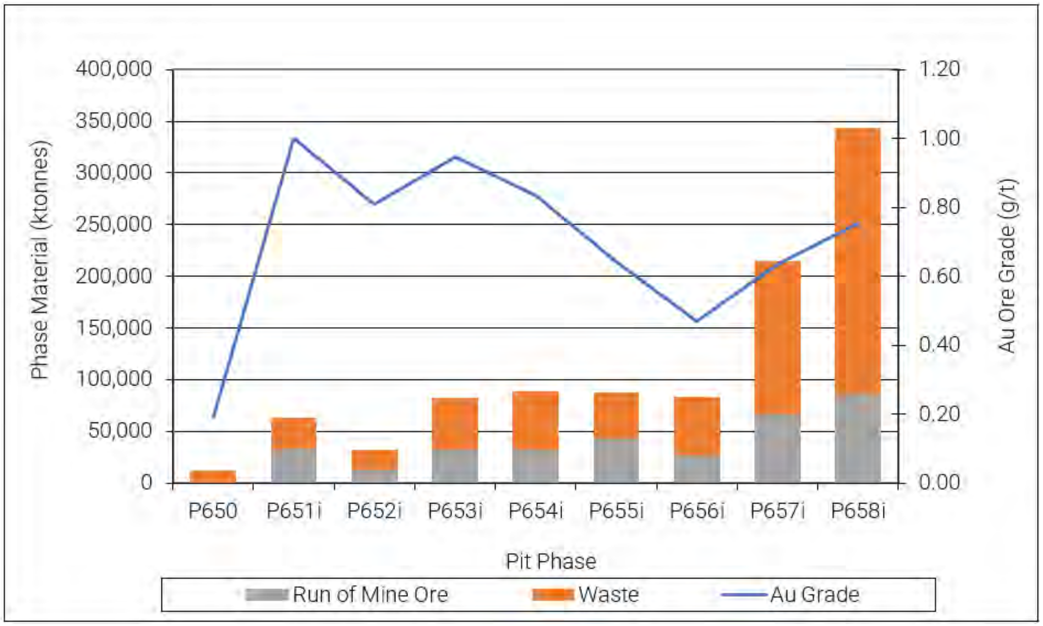


Note: Figure prepared by Moose Mountain, 2021.

16.7 Pit Designs

Contents of the designed open pits were provided in Table 15-2. The contents for each designed pit phase are shown in Figure 16-3.

Figure 16-3: Designed Phase Pit Contents



Note: Figure prepared by Moose Mountain, 2021.

16.7.1 In-Pit Haul Roads

Two-way in-pit haul roads of 32 m widths were designed to support the use of 230 t payload haul trucks. Haul road grades were limited to a maximum of 10%.

Access ramps were not designed for the last two benches of the pit bottom, on the assumption that the bottom ramp segments will be removed using some form of retreat mining. The bottom two ramped benches of the pit use one-way haul roads of 23 m width and 12% grade since bench volumes and traffic flow are reduced.

16.7.2 Pit Phasing

Ultimate pit limits are generally split into phases or pushbacks to target higher economic margin material earlier in the mine life as well as reducing the waste stripping hurdles as the pit develops. Minimum pushback distances of 75 m are honoured, with most of the bench pushbacks well over 100 m.

The Blackwater pit is split into nine phases:

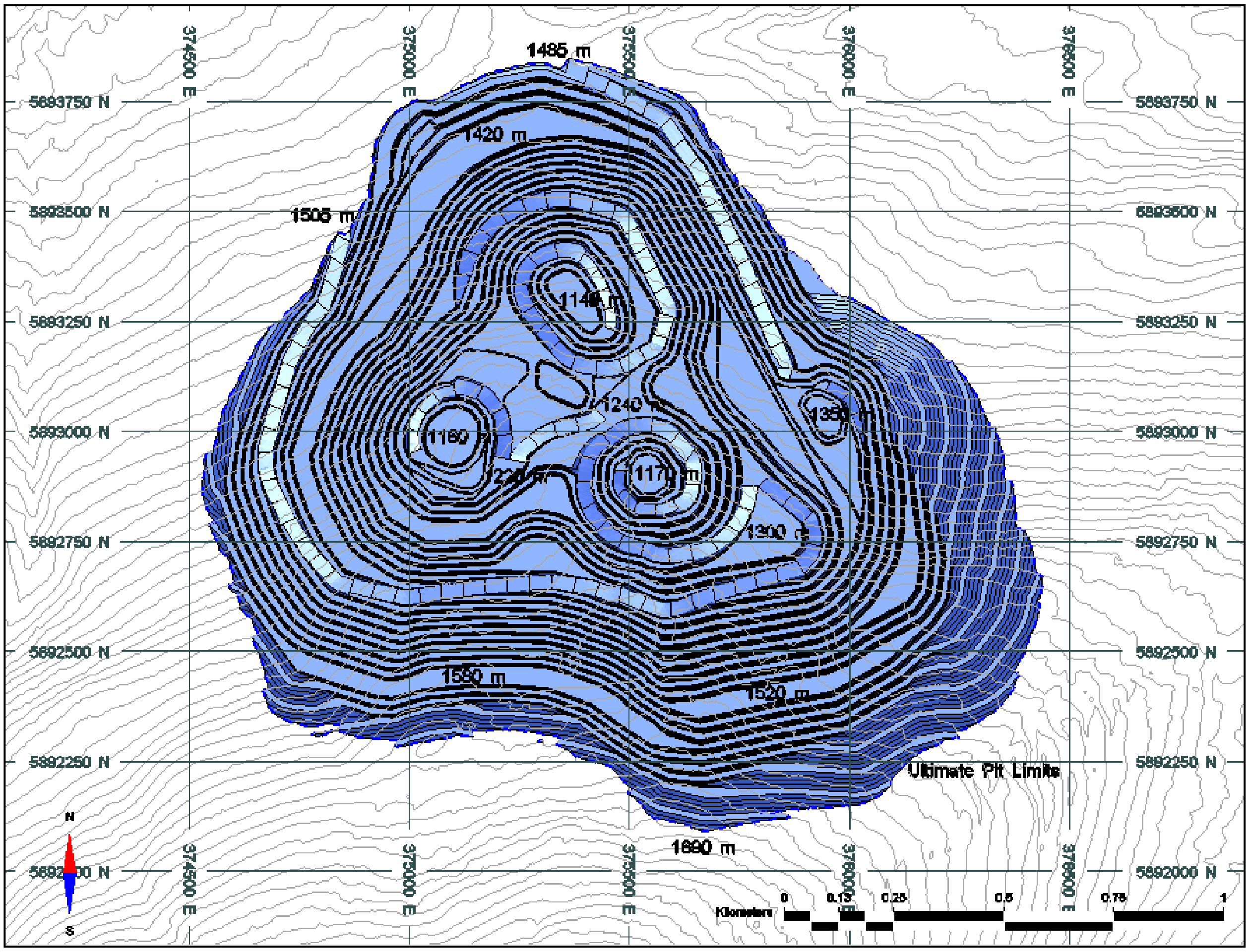
- The first phase acts as a borrow pit targeting NAG waste rock and glacial till overburden, which will be used during the early year construction periods;.

- The second phase targets higher-grade, lower-strip-ratio areas of the pit defined by Case 8 of the optimization runs described in Section 16.7 and providing mill feed over the initial years of the Blackwater Gold Mine;
- Phases then proceed from the highest economic margin to lowest.

16.7.3 Pit Designs

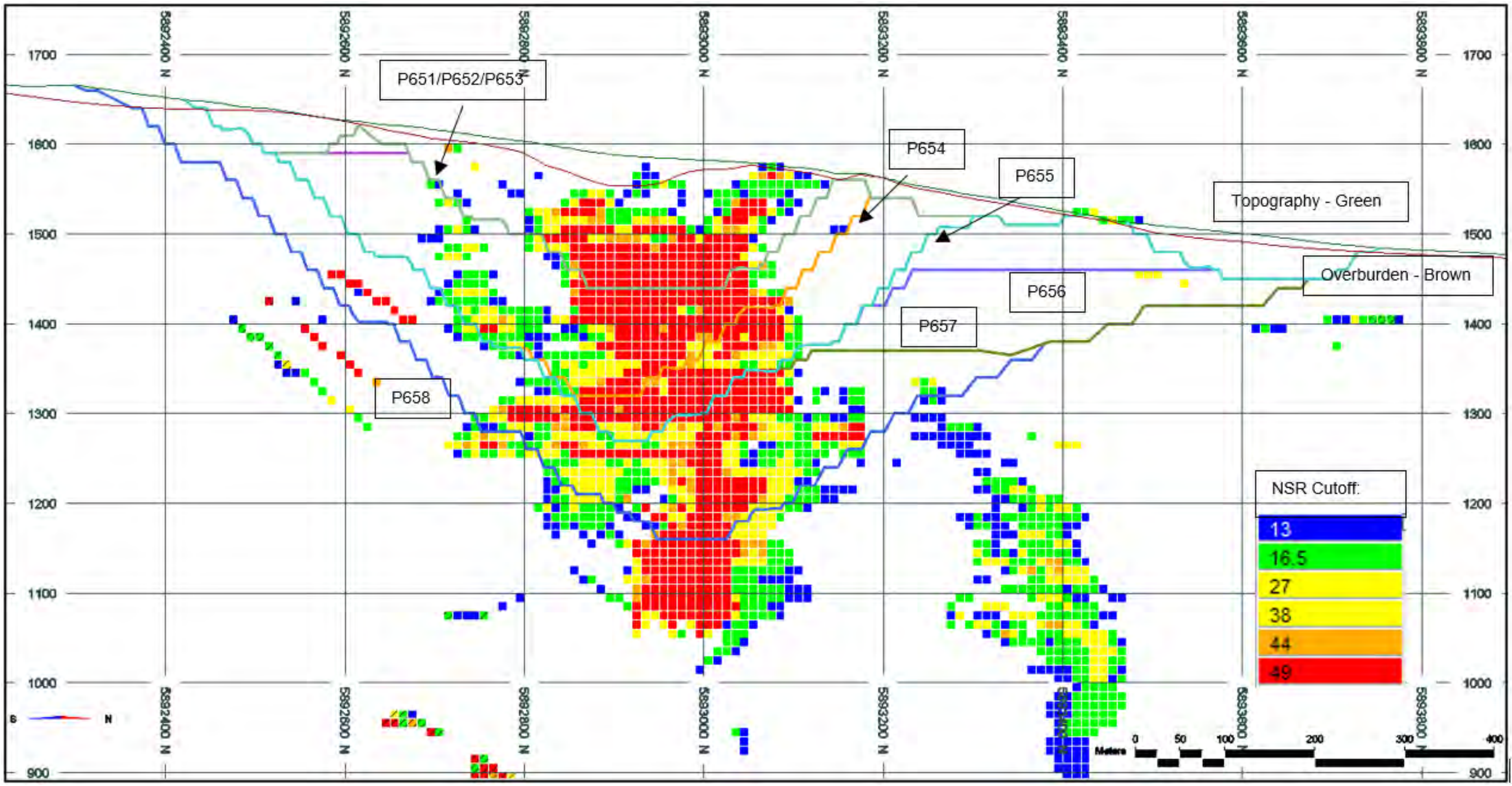
The ultimate Blackwater pit design is shown in Figure 16-4. Sections through the deposit showing the whole block gold and silver grades are illustrated in Figure 16-5 to Figure 16-8. Table 16-4 summarizes the pit phases. Where metre (m) elevations are listed in Table 16-4, they refer to the mine grid z-dimension elevation, which is metres above sea level (masl).

Figure 16-4: Final Pit Outline



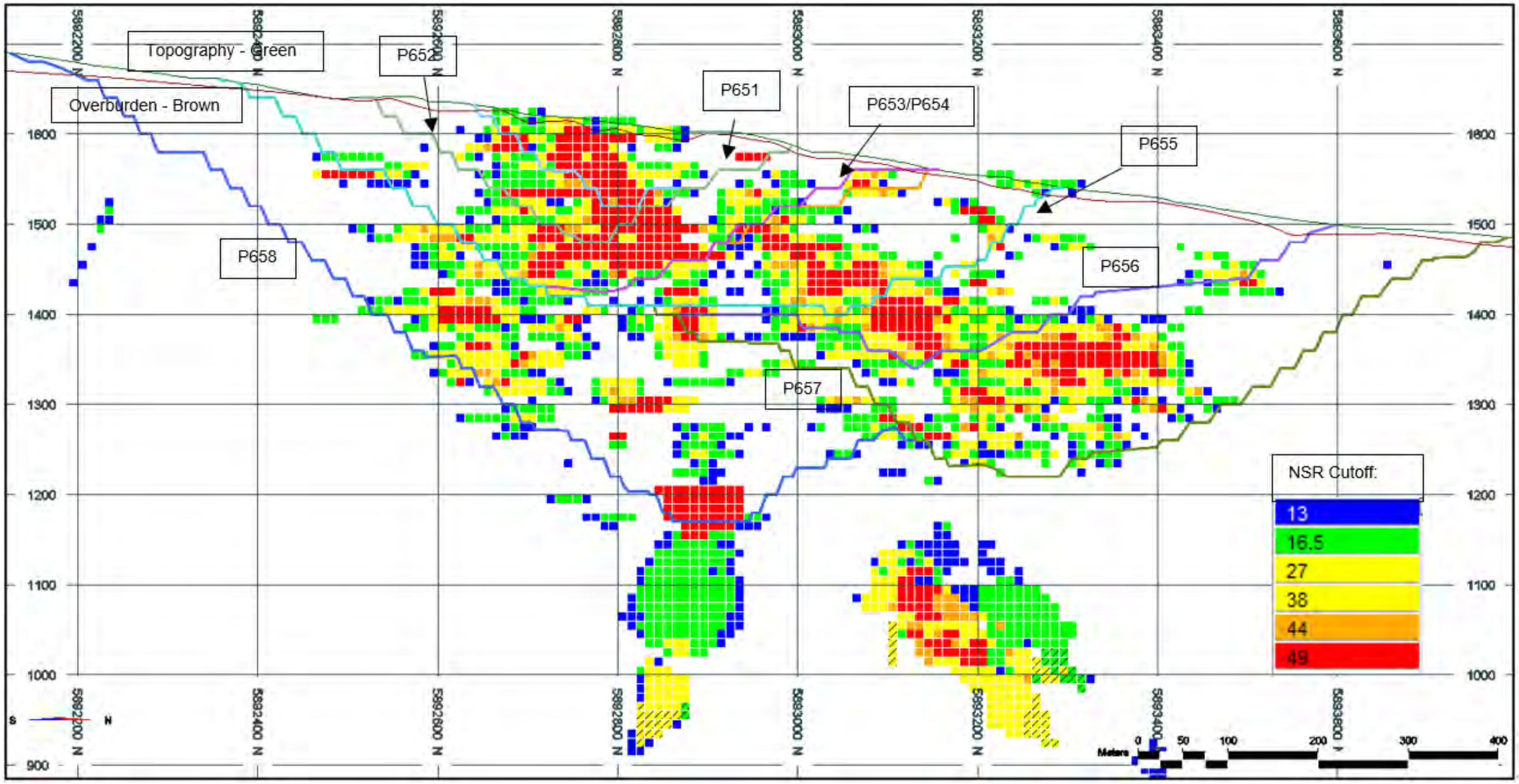
Note: Figure prepared by Moose Mountain, 2021.

Figure 16-5: North–South Section, 375,050E Looking West



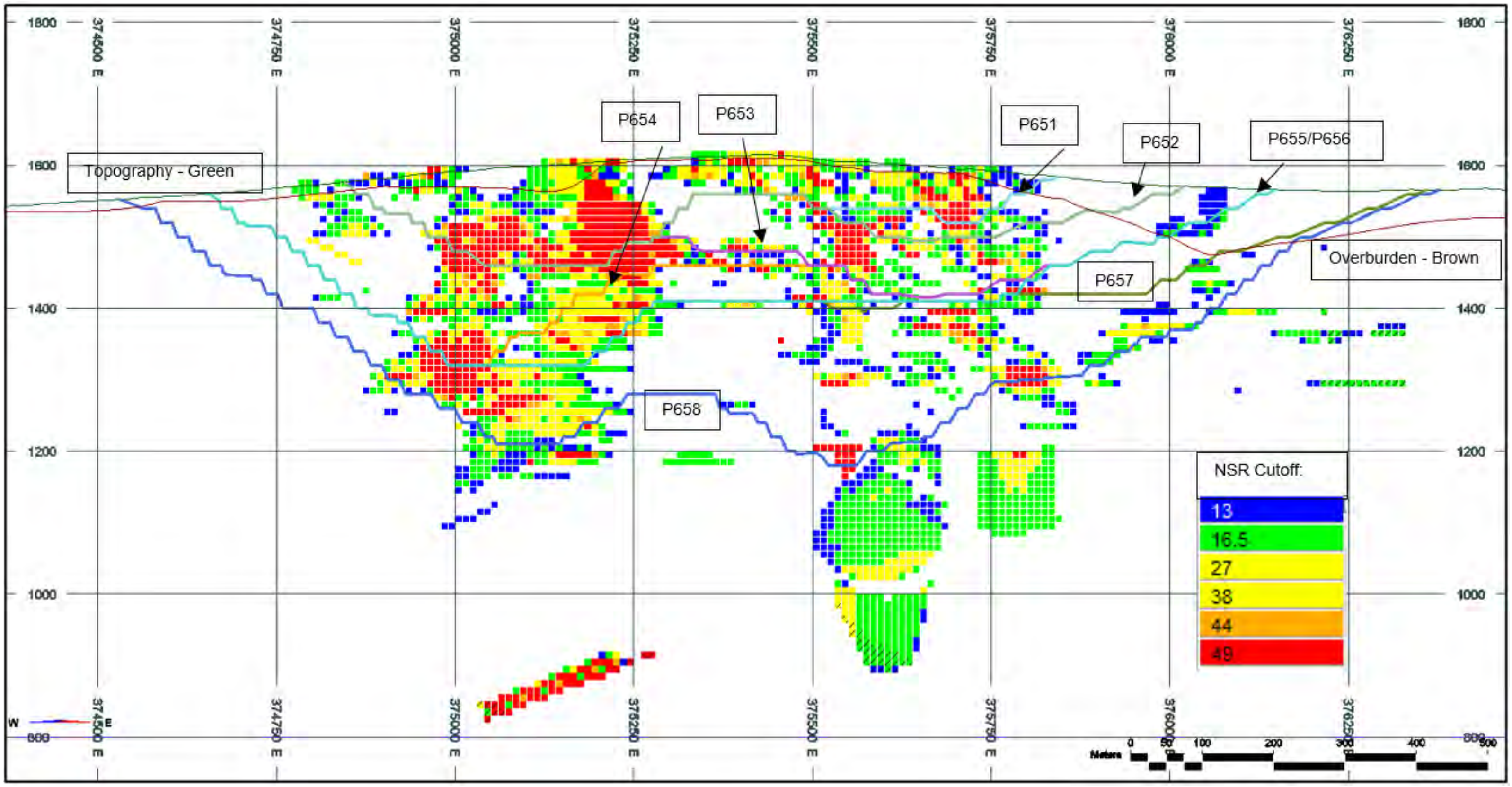
Note: Figure prepared by Moose Mountain, 2021.

Figure 16-6: North–South Section, 375,550E Looking West



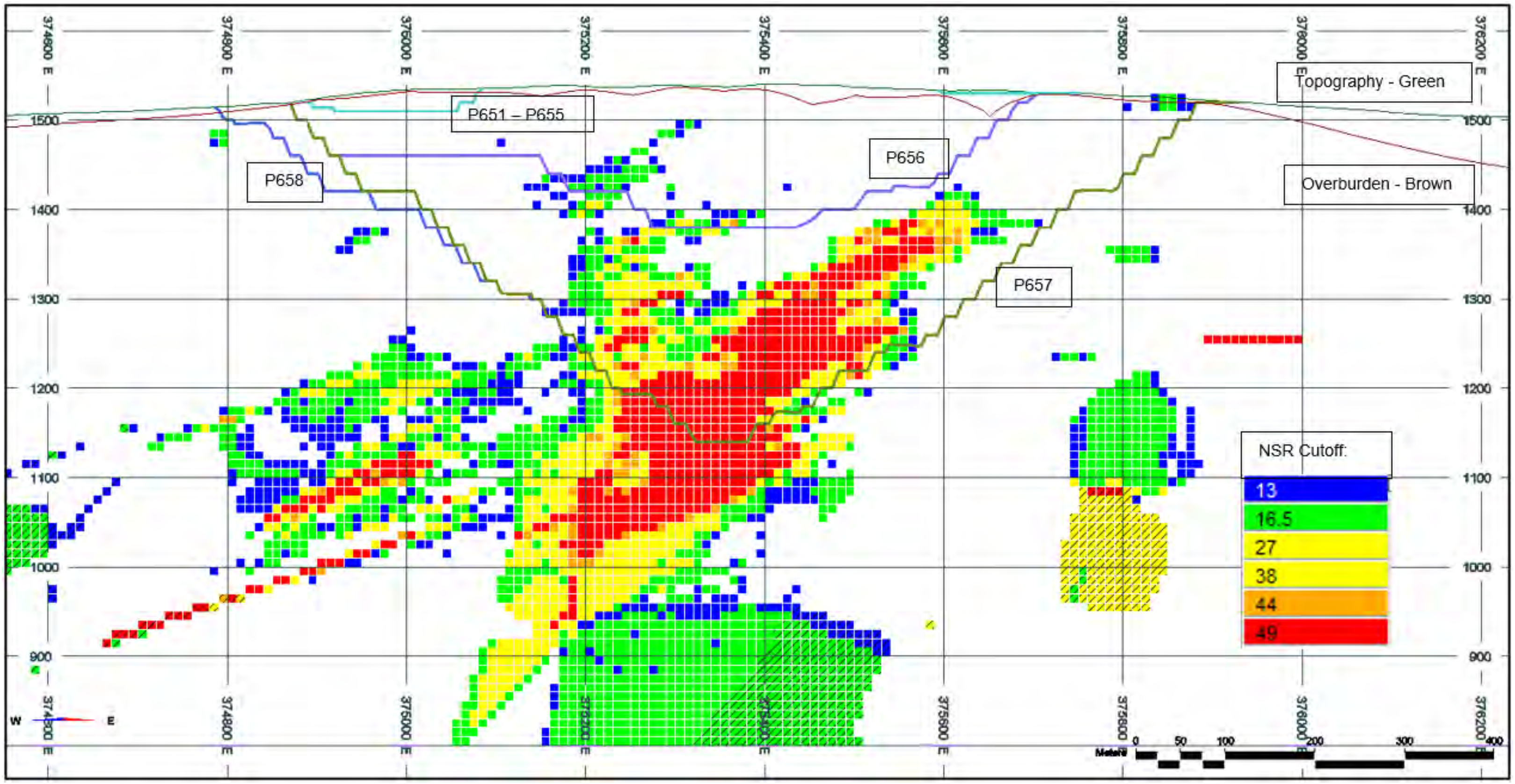
Note: Figure prepared by Moose Mountain, 2021.

Figure 16-7: East–West Section, 5,892,850N Looking North



Note: Figure prepared by Moose Mountain, 2021.

Figure 16-8: East–West Section, 5,893,350N Looking North



Note: Figure prepared by Moose Mountain, 2021.

Table 16-4: Pit Phases

Phase	Note
Construction Borrow Pit, P650	This phase targets near surface NAG waste rock and glacial till overburden for mine construction purposes, contained with three small pit areas within the ultimate pit limits. A small amount of PAG waste rock and mineralization is also contained within this initial phase. All three pit bottoms of this phase will be accessed by ex-pit haul roads on the hillside.
Starter Pit, P651	This phase targets the high-grade, low-strip-ratio southern portion of the deposit. This phase contains about three years' worth of mill feed within two separately-accessed pit bottoms. The western portion mines from the pit crest at the 1630 m elevation, down to the pit bottom at the 1,440 m elevation. The ramp runs counter clockwise down from the pit exit at the 1,555 m elevation in the west. The eastern portion of the pit is accessed from a secondary ramp running counter clockwise from the 1,560 m elevation down to the 1,520 m elevation. Upper benches of the pit will be accessed via ex-pit haul roads on the hillside.
East Pushback 1, P652	This phase pushes out the eastern portion of the previous phase. This phase contains about two years' worth of mill feed and mines from the pit crest at the 1,650 m elevation, down to the pit bottom at the 1,460 m elevation. The main ramp runs clockwise from the pit exit at the 1,560 m elevation in the north of the pit. Upper benches of the pit will be accessed via ex-pit haul roads on the hillside.
East Pushback 2, P653	This phase pushes out the eastern portion of the previous pit phase, with enough room left over for a final push back to the ultimate pit limits in future phases. The phase contains about four years' worth of mill feed and mines from the pit crest at the 1,665 m elevation, down to the pit bottom at the 1,390 m elevation. The main ramp runs clockwise from the pit exit at the 1,530 m elevation in the northeast of the pit. A sub-phase in the north part of this phase mines from the 1,530 m elevation down to the 1,450 m elevation via a counter clockwise ramp from the pit exit in the north. Upper benches of the pit will be accessed via ex-pit haul roads on the hillside.
West Pushback, P654	This phase pushes out the pit to the west, with enough room left over for a final push back to the ultimate pit limits in future phases. The phase mines from the pit crest at the 1,620 m elevation, down to the pit bottom on the 1,320 m elevation. The main ramp runs counter clockwise from the pit exit at the 1,540 m elevation in the west of the pit. Upper benches of the pit will be accessed via ex-pit haul roads on the hillside.
North Pushback 1, P655	This phase pushes out the pit to the north. The phase mines from the pit crest at the 1,570 m elevation, down to the pit bottoms on the 1,270 m. The main ramp runs clockwise from the pit exit at the 1,520 m elevation in the west of the pit, bridging across the phase on the 1,410 m bench, then proceeding counter clockwise to switchbacks on the 1,360 and 1,320 m benches. Upper benches of the pit will be accessed via ex-pit haul roads on the hillside.
North Pushback 2, P656	This phase further pushes out the pit to the north. The phase mines from the pit crest at the 1,540 m elevation, down to the pit bottoms on the 1,310 m. The main ramp runs clockwise from the pit exit at the 1,490 m elevation in the north of the pit. Upper benches of the pit will be accessed via ex-pit haul roads on the hillside.
Southeast Pushback, P657	This phase pushes out the pit to the ultimate limits in the north, east and southeast. The phase mines from the pit crest at the 1,630 m elevation, down to the pit bottom on the 1,140 m elevation. The main ramp runs clockwise from the pit exit at the 1,485 m elevation in the

Phase	Note
	north of the pit. Upper benches of the pit will be accessed via ex-pit haul roads on the hillside. An extra wide geotechnical berm is left behind on the 1,420 m bench. Future design iterations should implement shallower pit slopes in the broken zones of north wall between the 1,280 and 1,420 m elevations, with a narrower geotechnical berm.
Southwest Pushback, P658	This phase pushes out the pit to the ultimate pit limits in the south and west. The phase mines from the pit crest at the 1,690 m elevation, down to the two pit bottoms on the 1,170 m and 1,160 m elevations, bridging between the two bottoms on the 1,240 m bench. The main ramp runs counter clockwise from the pit exit at the 1,505 m elevation in the west of the pit, with switchbacks on the 1,300 m and 1,220 m benches. Upper benches of the pit will be accessed via ex-pit haul roads on the hillside. Additional geotechnical berms are left behind on the 1,580 m and 1,520 m benches in the south and on the 1,400 m bench in the west.

Note: PAG = potentially acid generating; NAG = non-acid generating.

16.8 Ex-Pit Haul Roads

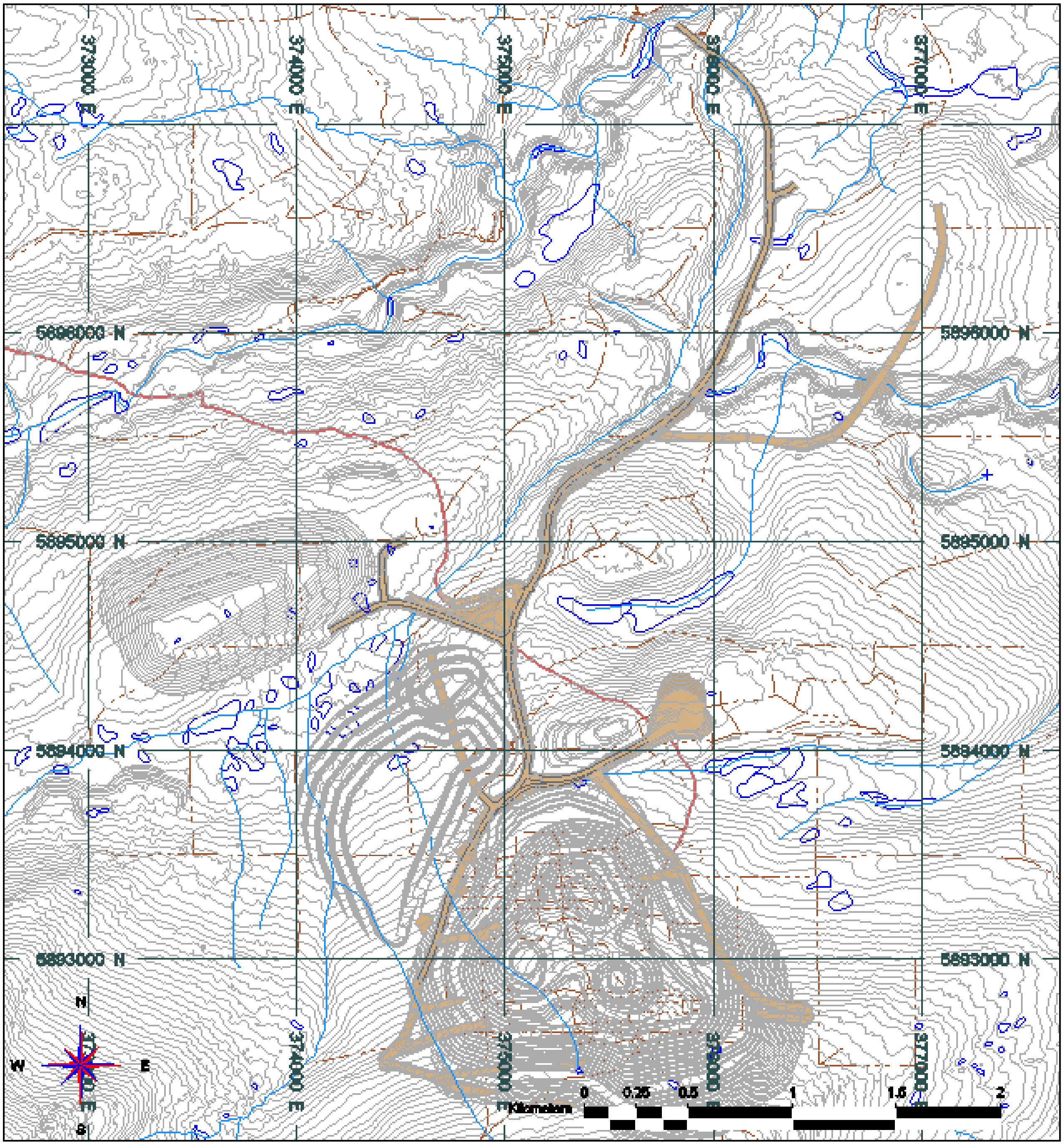
Mine haul roads external to the open pits are designed to haul ore and waste materials from the open pits to the scheduled destinations. The mine haul roads were designed with the following key inputs:

- 37 m wide ex-pit haul roads that incorporated a dual-lane running width and berms on both edges of the haul road;
- Sized to handle 230 t payload rigid-frame haul trucks;
- 10% maximum grade.

The designed ex-pit haul roads are shown as highlighted surfaces in Figure 16-9.

The existing exploration road network was incorporated into the access development for the pit. Haul road routes will be initially pioneered by dozers as single-lane, balanced cut/fill accesses. The roads will be expanded into full-width mine haul roads by means of end-dumping with suitable NAG waste rock from the pit.

Figure 16-9: Ex-Pit Mine Haul Road Location Plan



Note: Figure prepared by Moose Mountain, 2021. Roads shown in brown.

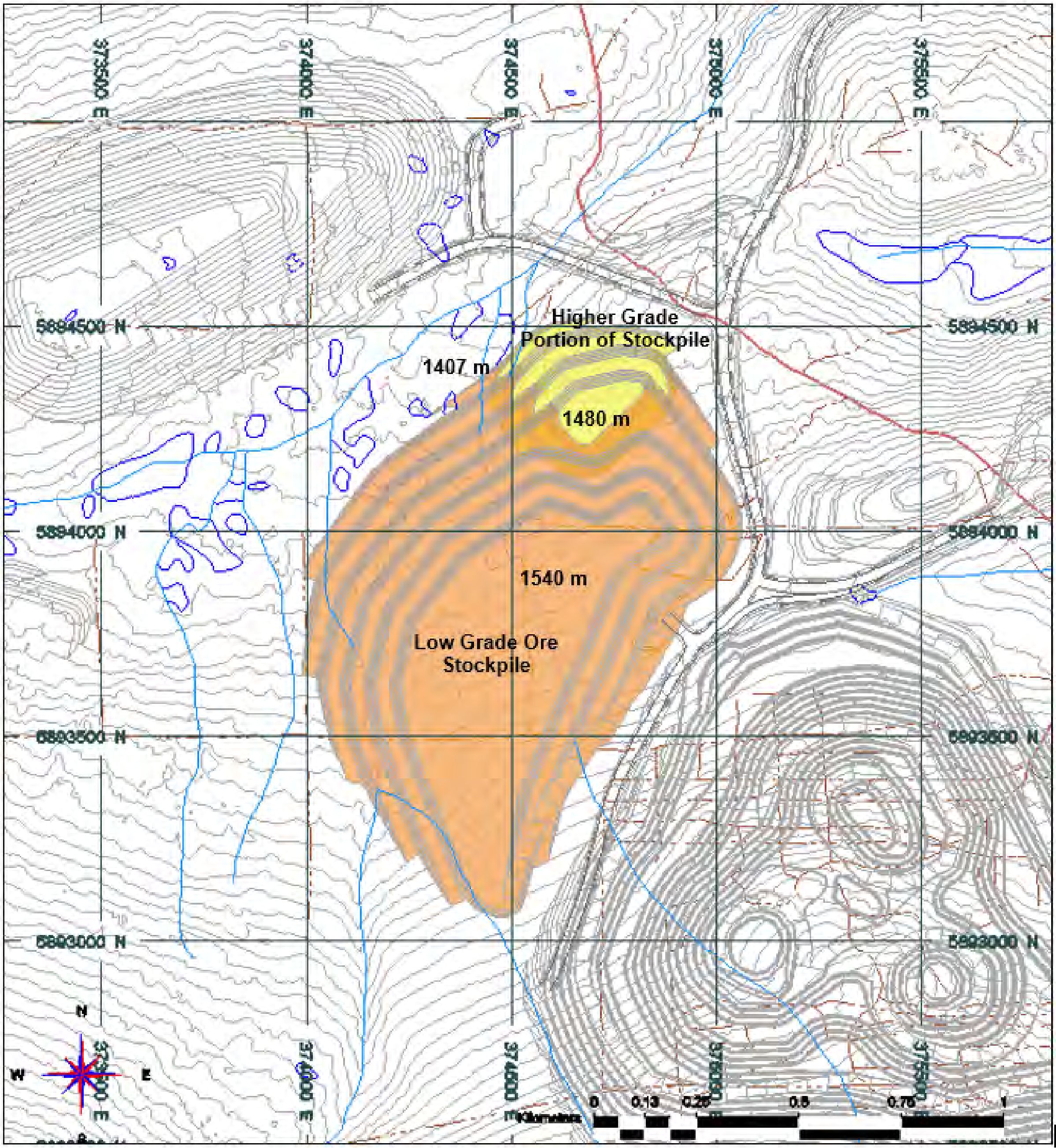
16.9 Ore Stockpile Facilities

When ore is mined from the pit, it will either be delivered to the crusher, the run-of-mine (ROM) stockpile located next to the crusher, or the low-grade ore stockpile.

The crusher and ROM stockpiles are located <1 km northeast of the pit limits.

Throughout the life of operations, ore at between C$13.00/t and C$27/t NSR will be stockpiled in a low-grade ore stockpile located just outside the pit limits to the northwest, shown in Figure 16-10.

Figure 16-10: Ore Stockpile Facilities Location Plan



Note: Figure prepared by Moose Mountain, 2021.

Cut-off grade optimization on the mine production schedule also sends ore above C$27/t NSR, mined in the first two years of the mine life, to a discrete area within the northeast corner of the stockpile footprint.

The stockpiled ore is planned to be re-handled back to the crusher during the mine life. The higher-grade material will be re-handled in advance of the lower-grade material.

The ore stockpile is designed within the existing EA certified project description boundary. The stockpile will be built on the hillside, on six 20 m lifts dumped out at angle of repose (1.3H:1V). Each facility is planned at a 3H:1V overall slope from the 1,407 masl elevation to the 1,540 masl elevation.

Low-grade ore is classified as potentially-acid generating (PAG) with a relatively short lag time, and the stockpile is expected to generate acidic drainage with elevated metals until the ore is processed. The low-grade ore will be placed on a low-permeability base with a drainage collection system. The drainage will be collected, neutralized with lime at the process plant, and discharged to the TSF.

16.10 Waste Storage Facilities

Waste storage facility designs were completed that will have the capacity to store all non-ore materials from the Blackwater open pit.

Overburden and non-acid generating (NAG) waste rock will be used for construction of the downstream and some upstream portions of the TSF dams. PAG waste rock will be used for construction of sections of the upstream portion of the TSF dams or stored subaqueously within the TSF itself. Additional details related to the TSF are included in Section 18 of this Report.

16.10.1 Waste Classification

A block model was developed based on the acid-base accounting (ABA) test results and exploration geological metal dataset to classify waste rock and ore blocks. The same criteria for waste rock were used for ore blocks.

The likelihood of acid rock drainage (ARD) of rock or other material was based on the neutralization potential ratio (NPR, where NPR = neutralization potential (NP)/acid potential (AP)), where PAG material was defined as an NPR ≤2 and NAG material defined by an NPR >2.

Waste materials were grouped as either PAG or NAG as defined by the calculated NPR. NAG waste rock was further classified as to its metal leaching (ML) potential based on zinc content, as that metal (and others, including cadmium) shows elevated values in some of the mine waste. Overall, the classification criteria were defined as follows:

- Waste rock:
    - PAG1: NPR ≤ 1.0 (PAG);

    - PAG2: 1.0 < NPR ≤ 2.0 (PAG);
    - NAG3: NPR > 2.0 and Zn ≥ 1,000 ppm (NAG–metal leaching);
    - NAG4: NPR > 2.0 and 600 ≤ Zn < 1,000 ppm (NAG);
    - NAG5: NPR > 2.0 and Zn < 600 ppm (NAG);
- Overburden (NAG);
- Ore and low-grade ore (PAG);
- Tailings (PAG).

Waste rock with zinc values > 1,000 ppm (NAG3) will be segregated as this material has a higher ML potential. NAG4 and NAG5 waste rock has a low ML potential due to the relatively low zinc content. NAG4 and NAG5 waste rock can be used in construction or stored in out-of-pit storage piles, while NAG3 waste rock will be subaqueously stored in the TSF to limit the potential for neutral metal leaching.

All PAG waste rock and tailings will be subaqueously stored in the TMF.

Ore and low-grade ore will be temporarily stockpiled before being milled and ultimately disposed in the TSF as tailings.

In closure, a cover composed of NAG material will be placed on the final tailings beach to minimize oxidation and interaction with meteoric water.

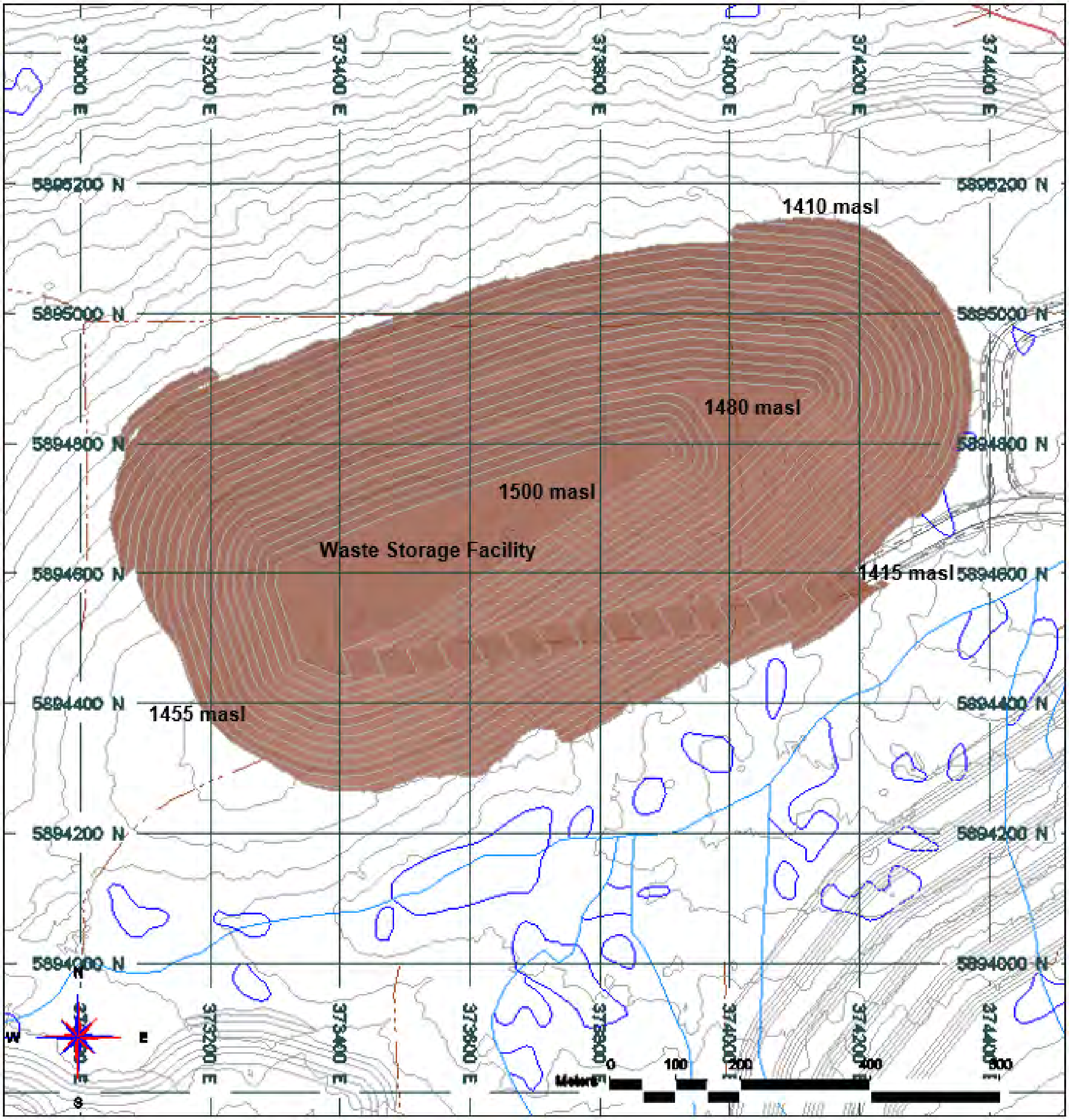
16.10.2 Waste Handling

Most of the waste materials from the open pit will be used for construction of TSF or placed in the TSF itself.

A storage pile is planned for surplus NAG waste materials from the open pit.

Overburden and NAG waste not used in the construction of the TSF will be placed in the WRSF. This facility is shown in Figure 16-11. It is designed within the existing Environmental Assessment-certified project description boundary.

Figure 16-11: Waste Rock Storage Facility Location Plan



Note: Figure prepared by Moose Mountain, 2021.

The WRSF will be located 1.5 km northwest of the pit limits and is planned on four 20 m lifts dumped out at angle of repose and will store a mixture of NAG waste rock and overburden. It is planned at a 4H:1V overall slope from the 1,370 to the 1,470 masl elevations. As some of the material will be rehandled to the tailings facility for dam construction at the end of the LOM, the final elevation of this stockpile is planned to sit at 1,470 masl elevation.

16.11 Production Schedule

16.11.1 Production Scheduling

Production requirements by period, mine operating considerations, product prices, recoveries, destination capacities, equipment performance, haul cycle times and operating costs are used to determine the optimal production schedule from the pit phase Mineral Reserves.

The production schedule is based on the following parameters:

- The Mineral Reserve estimate quantities are split by phase and bench;
- Annual periods are scheduled out over the life of mine:
  - Monthly period detail is scheduled out for the pre-production period and Year 1;
  - Quarterly period detail is scheduled out for Years 2 and 3;
- An annual mill feed rate of 6 Mt is targeted for the first year of operation, increasing to 15 Mt for the next five years, and 25 Mt thereafter:
  - Mill commissioning and ramp up is planned during the pre-production period with no capture of economic benefits via the ramp up milling;
  - Year 2 throughput of 9 Mt is targeted, with expansion planned for the latter half of the year;
- Within a given phase, each bench is fully mined before progressing to the next bench;
- Pit phases are mined in sequence, where the second pit phase does not mine below the first pit phase;
- Pit phase vertical progression is limited to no more than 80 m in each year:
  - Average annual vertical phase progression is 50 m;
- Ore tonnes released in excess of the mill capacity are stockpiled;
- Low-grade ore is stockpiled and re-handled to the crusher before the end of mine life.

The overall production schedule is included in Table 16-5.

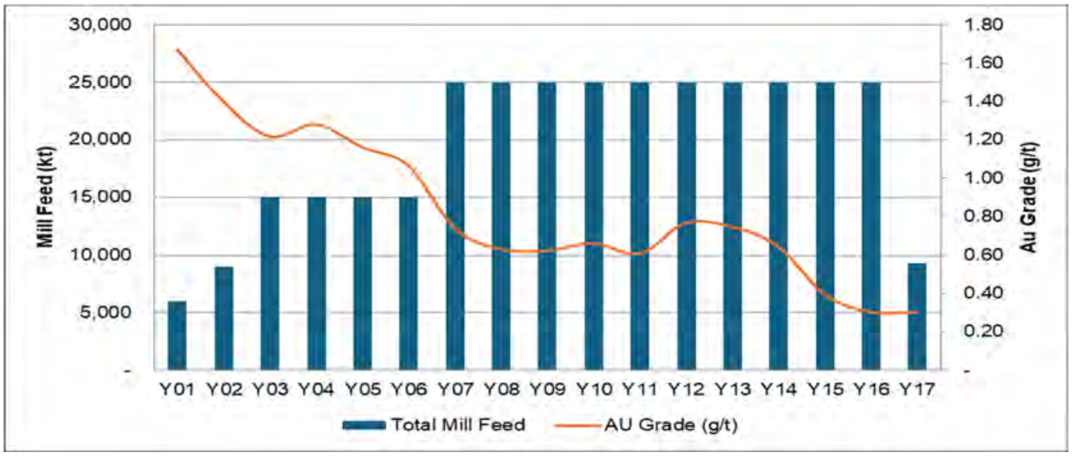
Table 16-5: Production Schedule

	Unit	LOM	Y1	Y2	Y3	Y4	Y5	Y6	Y7	Y8	Y9	Y10	Y11	Y12	Y13	Y14	Y15	Y16	Y17
Ore mill feed	*kt*	*334,279*	*6,000*	*9,000*	*15,000*	*15,000*	*15,000*	*15,000*	*25,000*	*25,000*	*25,000*	*25,000*	*25,000*	*25,000*	*25,000*	*25,000*	*25,000*	*25,000*	*9,278*
Au	*g/t*	*0.75*	*1.67*	*1.40*	*1.22*	*1.28*	*1.16*	*1.06*	*0.74*	*0.63*	*0.62*	*0.66*	*0.61*	*0.77*	*0.74*	*0.64*	*0.40*	*0.30*	*0.30*
Ag	*g/t*	*5.78*	*9.10*	*6.58*	*7.84*	*6.29*	*9.29*	*8.08*	*6.33*	*4.96*	*2.89*	*3.82*	*4.65*	*5.04*	*5.46*	*4.87*	*6.61*	*6.75*	*6.75*
Ore mined from pit	kt	333,934	18,231	14,561	27,853	30,323	30,370	30,946	29,895	25,000	23,010	17,000	16,000	24,000	25,000	19,000	2,745	—	—
Au	g/t	0.75	0.83	0.99	0.80	0.80	0.71	0.68	0.65	0.63	0.63	0.79	0.78	0.78	0.74	0.75	1.14	—	—
Ag	g/t	5.78	6.67	5.64	6.56	5.13	10.65	7.51	6.69	4.96	2.59	2.50	3.46	4.97	5.46	4.28	5.45	—	—
Ore mined directly to mill	kt	251,707	5,951	9,000	15,000	15,000	15,000	15,000	25,000	25,000	23,010	17,000	16,000	24,000	25,000	19,000	2,745	—	—
Au	g/t	0.89	1.68	1.40	1.22	1.28	1.16	1.06	0.74	0.63	0.63	0.79	0.78	0.78	0.74	0.75	1.14	—	—
Ag	g/t	5.47	9.10	6.58	7.84	6.29	9.29	8.08	6.33	4.96	2.59	2.50	3.46	4.97	5.46	4.28	5.45	—	—
Ore mined to stockpile	kt	82,227	12,280	5,561	12,852	15,323	15,370	15,946	4,895	—	—	—	—	—	—	—	—	—	—
Au	g/t	0.32	0.42	0.33	0.31	0.32	0.28	0.31	0.22	—	—	—	—	—	—	—	—	—	—
Ag	g/t	6.74	5.48	4.11	5.07	3.99	11.99	6.96	8.56	—	—	—	—	—	—	—	—	—	—
Stockpile retrieval to mill	kt	82,572	49	—	—	—	—	—	—	—	1,990	8,000	9,000	1,000	-	6,000	22,255	25,000	9,278
Au	g/t	0.32	1.00	—	—	—	—	—	—	—	0.59	0.40	0.30	0.30	-	0.30	0.30	0.30	0.30
Ag	g/t	6.73	9.42	—	—	—	—	—	—	—	6.32	6.61	6.75	6.75	-	6.75	6.75	6.75	6.75
Stockpile balance	kt		12,576	18,137	30,989	46,312	61,682	77,628	82,523	82,523	80,533	72,533	63,533	62,533	62,533	56,533	34,278	9,278	—
Au	g/t		0.42	0.39	0.36	0.35	0.33	0.33	0.32	0.32	0.31	0.30	0.30	0.30	0.30	0.30	0.30	0.30	—
Ag	g/t		5.45	5.04	5.06	4.70	6.52	6.61	6.73	6.73	6.74	6.75	6.75	6.75	6.75	6.75	6.75	6.75	—
Waste mined	kt	*670,531*	*23,513*	*38,420*	*49,744*	*65,116*	*64,276*	*60,200*	*61,187*	*53,000*	*54,990*	*56,155*	*48,517*	*44,740*	*31,177*	*18,934*	*564*	—	—
NAG rock waste	kt	117,276	3,649	8,831	11,407	8,907	8,106	5,812	10,930	11,586	15,734	16,271	12,615	3,078	278	74	-	—	—
PAG rock waste	kt	467,698	12,029	15,833	25,134	47,298	47,274	36,796	46,457	38,424	34,867	35,731	35,871	41,662	30,899	18,860	564	—	—
Overburden waste	kt	85,466	7,835	13,752	13,203	8,911	8,893	17,591	3,792	2,989	4,384	4,084	32	—	—	—	—	—	—
Wasted Inferred	kt	91	—	4	—	—	3	—	8	1	6	70	—	—	—	—	—	—	—
Au	g/t	0.32	—	0.23	—	—	0.11	—	0.11	0.11	0.24	0.36	—	—	—	—	—	—	—
Ag	g/t	4.48	—	1.70	—	—	21.50	—	21.50	21.50	2.40	1.96	—	—	—	—	—	—	—
Waste rehandle	kt	9,936	—	—	—	—	—	—	—	—	—	—	—	4,696	4,000	200	200	742	98
Waste destination summary																			
OVB to construction	kt	1,200	—	500	700	—	—	—	—	—	—	—	—	—	—	—	—	—	—
NAG to construction	kt	2,820	400	420	200	200	200	200	200	200	200	200	200	200	—	—	—	—	—
OVB to tailing dam	kt	54,422	3,095	3,476	11,457	7,254	6,111	7,521	3,685	2,885	4,069	3,599	32	—	—	200	200	742	98
PAG to tailing dam	kt	29,803	—	—	—	—	738	2,468	6,758	2,335	5,497	5,447	3,035	3,525	—	—	—	—	—
NAG to tailing dam	kt	113,383	1,709	8,183	11,207	8,084	5,856	5,482	10,730	10,022	15,352	15,526	9,312	7,574	4,278	75	—	—	—
PAG to TSF	kt	437,986	12,029	15,837	25,134	47,298	46,539	34,328	39,707	36,090	29,375	30,354	32,836	38,137	30,899	18,860	564	—	—
OVB to WRSF	kt	31,084	4,740	9,776	1,046	1,657	2,782	10,070	107	104	315	485	—	—	—	—	—	—	—
NAG to WRSF	kt	9,769	1,540	227	-	623	2,050	130	-	1,364	182	545	3,103	—	—	—	—	—	—
Waste:ore ratio		2.0	1.3	2.6	1.8	2.1	2.1	1.9	2.0	2.1	2.4	3.3	3.0	1.9	1.2	1.0	0.2	—	—
Cumulative waste:ore ratio			1.4	1.9	1.9	2.0	2.0	2.0	2.0	2.0	2.1	2.1	2.2	2.2	2.1	2.0	2.0	2.0	2.0
Total material mined	kt	1,004,465	41,744	52,981	77,596	95,438	94,646	91,146	91,082	78,000	78,000	73,156	64,517	68,740	56,177	37,934	3,309	—	—
Cumulative material mined	kt		44,488	97,469	175,065	270,504	365,150	456,295	547,378	625,378	703,378	776,533	841,050	909,790	965,967	1,003,901	1,007,210	1,007,210	1,007,210
Total material moved	kt	1,096,973	41,793	52,981	77,596	95,438	94,646	91,146	91,082	78,000	79,990	81,155	73,517	74,436	60,177	44,134	25,764	25,742	9,376

Note: Y = year, OVB = overburden, NAG = non-acid generating, PAG = potentially acid generating.

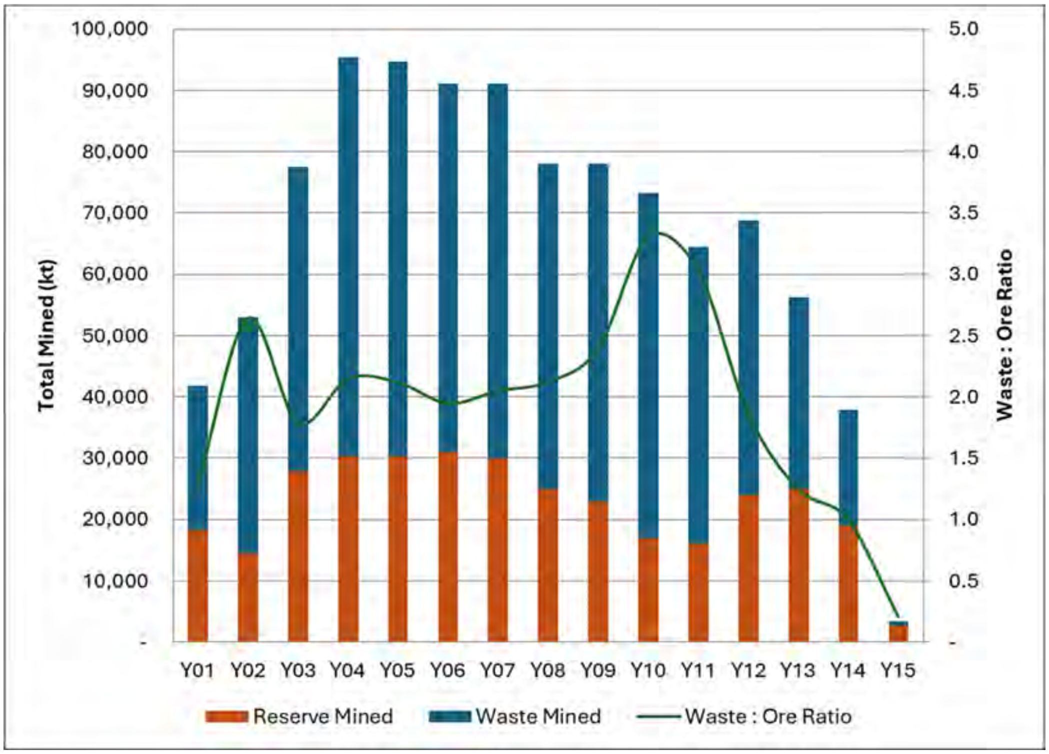
The mill feed is illustrated in Figure 16-12 and shows the production tonnage and grade forecast; Figure 16-13 provides an illustration of the projected material mined and strip ratio.

Figure 16-12: Planned Mill Feed Tonnes and Grade



Note: Figure prepared by Moose Mountain, 2021.

Figure 16-13: Planned Material Mined and Waste:Ore Ratio



Note: Figure prepared by Moose Mountain, 2021.

16.11.2 Mine Sequence

The pit operations will run for 15 years, including three months of pre-production. Following pit operations, stockpile re-handling operations will continue for two additional years.

LOM activities are summarized in Table 16-6. End-of-period layouts for Year 1, Year 3, Year 6, Year 11, Year 15, and Year 17 are illustrated in Figure 16-14 to Figure 16-19. Each end-of period layout plan represents a point in time where there are material changes to the mine operational strategy.

Table 16-6: Annual Mine Operations Activities

Year	Activity
Year 1	Clearing and grubbing of pit and stockpile areas (annually throughout mine life). Starter pit mined down to 1,540 bench. Stockpiling of excess high-grade ore to the ore stockpile. Delivery of low-grade ore to the ore stockpile (continues until Year 7). Haul road expansion from the pits to the tailings dam (continues to Year 3). Delivery of construction fill to TSF C dam (continues until Year 14). Stockpiling of PAG rock in the TCF C footprint (continues to Year 5). Delivery of excess mined overburden and NAG rock to the WRSF (continues until Year 11). Pit electrification and purchase of electric driven drills and shovels.
Year 2 to Year 3	Starter pit mined to the pit bottom on the 1,440 bench. East pushback 1 and 2 pits mined to the 1,490 bench. West pushback pit mined to the 1,550 bench. Mill feed increased to 15 Mt/a during Year 2. Delivery of construction fill to TSF D dam (continues until Year 14).
Year 4 to Year 6	East pushback 1 pit mined to the pit bottom on the 1,460 bench (Year 4). East pushback 2 pit mined to the pit bottom on the 1,390 bench (Year 5). West pushback pit mined to the 1,360 bench. North pushback 1 and 2 pits mined to the 1,410 bench. Southeast pushback pit mined to the 1,470 bench. Southwest pushback pit mined to the 1,630 bench. Stockpiling of PAG rock in the TCF D footprint (continues to Year 15).
Year 7	West pushback pit mined to the pit bottom on the 1,320 bench. North pushback 1 pit mined to the 1,330 bench. North pushback 2 pit mined to the 1,370 bench. Southeast pushback pit mined to the 1,420 bench. Southwest pushback pit mined to the 1,610 bench. Mill feed increased to 25 Mt/a.
Year 8 to Year 10	North pushback 1 pit mined to the pit bottom on the 1,270 bench (Year 9). North pushback 2 pit mined to the pit bottom on the 1,310 bench (Year 9). Southeast pushback pit mined to the 1,270 bench. Southwest pushback pit mined to the 1,540 bench. Re-handle of all stockpiled high-grade ore (5 Mt) to the crusher. Re-handle of stockpiled low-grade ore (5 Mt) to the crusher.
Year 11 to Year 15	Southeast pushback pit mined to the pit bottom on the 1,140 bench (Year 11). Southwest pushback pit mined to the pit bottom on the 1,160 bench. Re-handle of stockpiled low-grade ore (38 Mt) to the crusher. Re-handle of WRSF material (9 Mt) to the TSF C and D dams.

Year	Activity
Year 16 to Year 17	Re-handle to crusher of remaining stockpiled low-grade ore (34 Mt, stockpiled depleted). Re-handle of WRSF material (1 Mt) to the TSF D dam. Initiate work on closure plan for pits and WSF.

Figure 16-14: EOP Mine Operations, Year 1

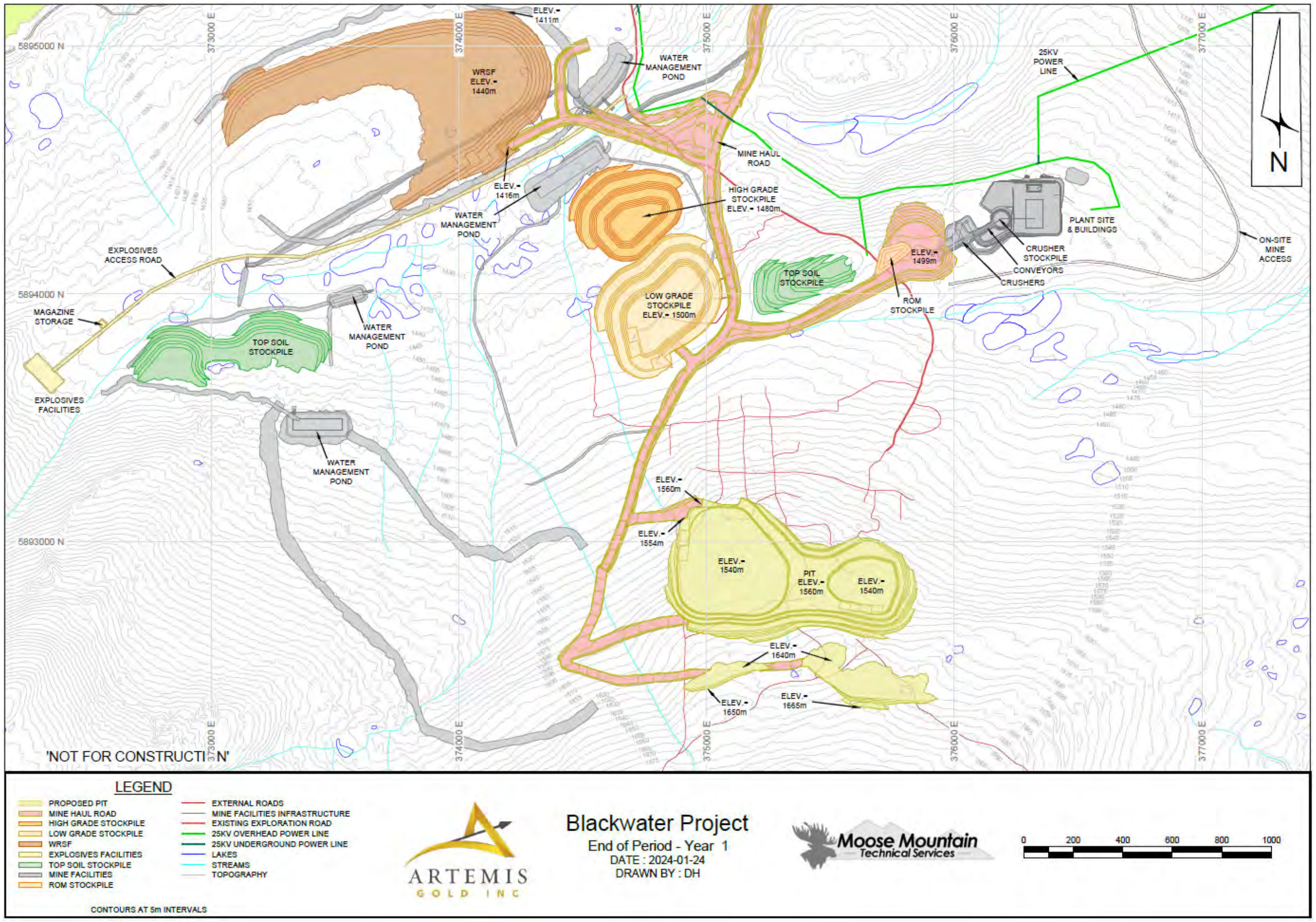


Figure 16-15: EOP Mine Operations, Year 3

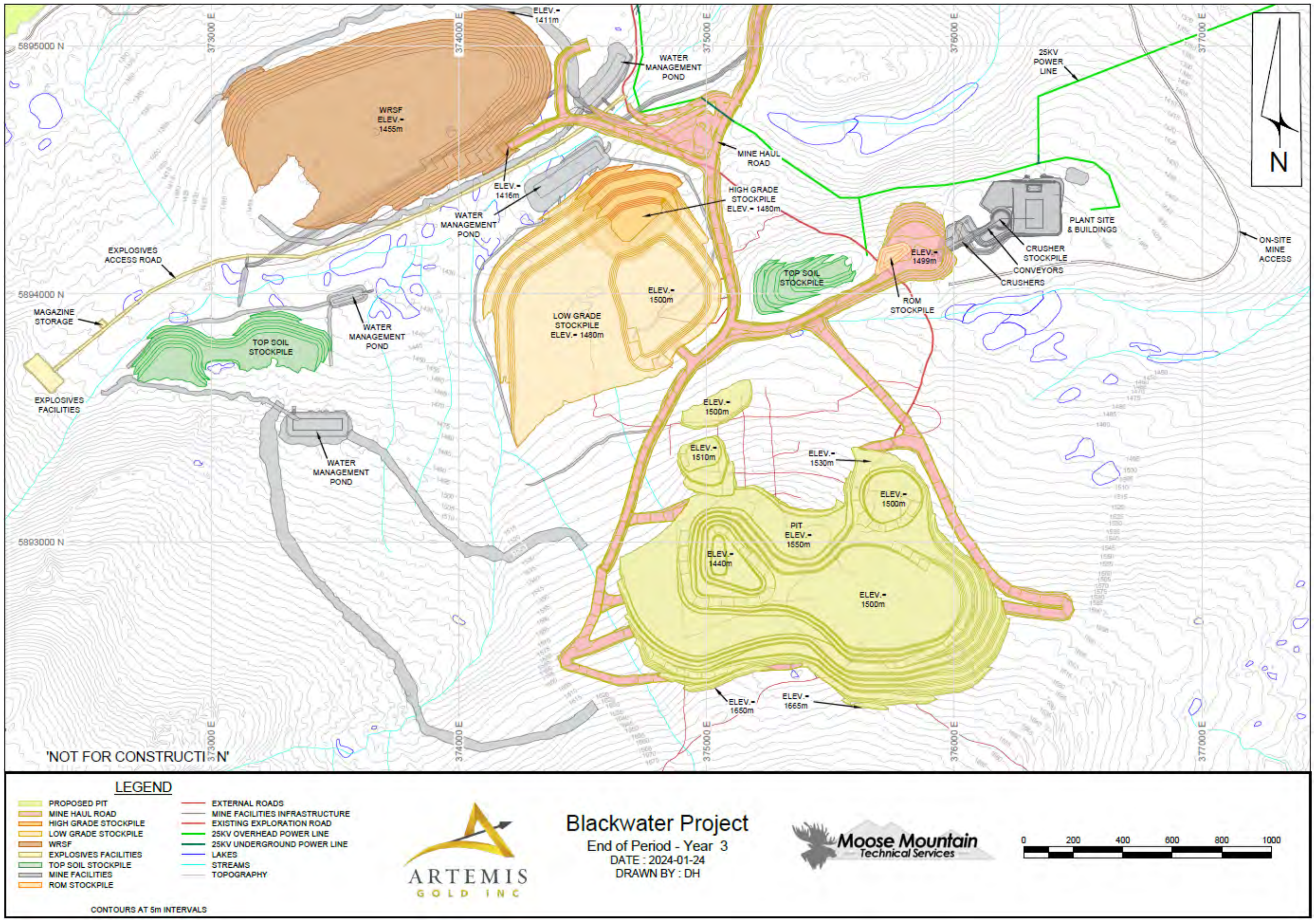


Figure 16-16: EOP Mine Operations, Year 6

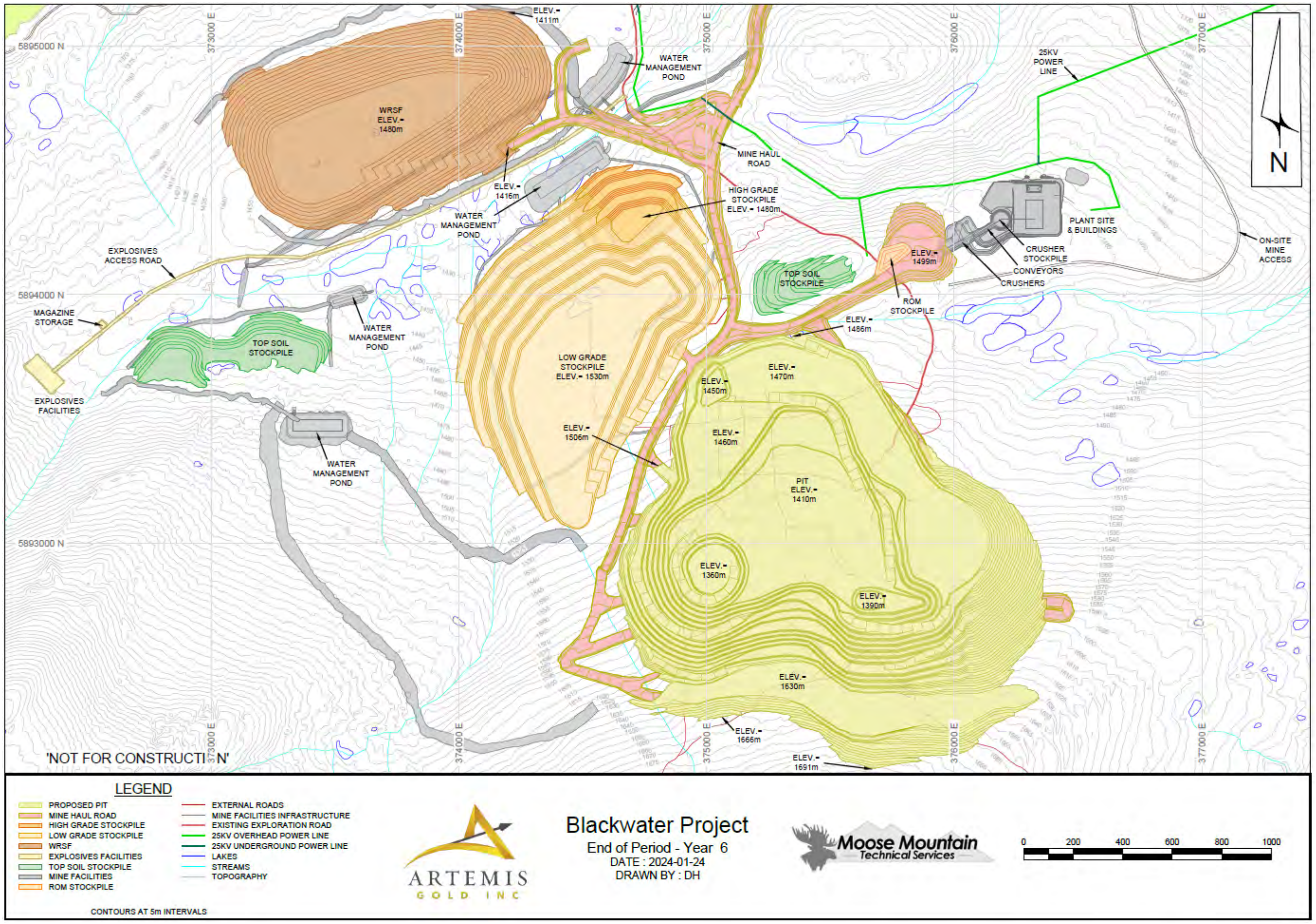


Figure 16-17: EOP Mine Operations, Year 11

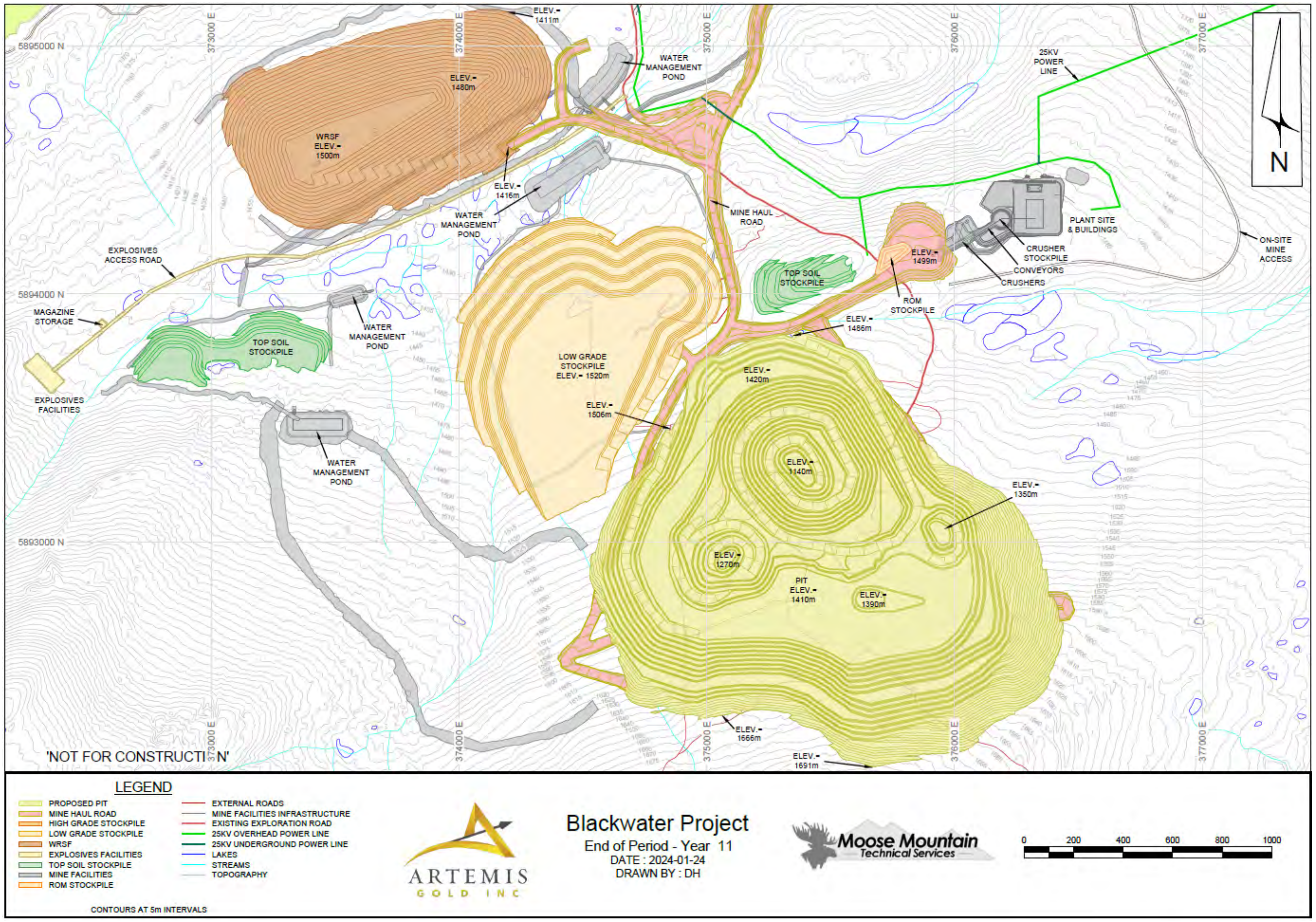


Figure 16-18: EOP Mine Operations, Year 15

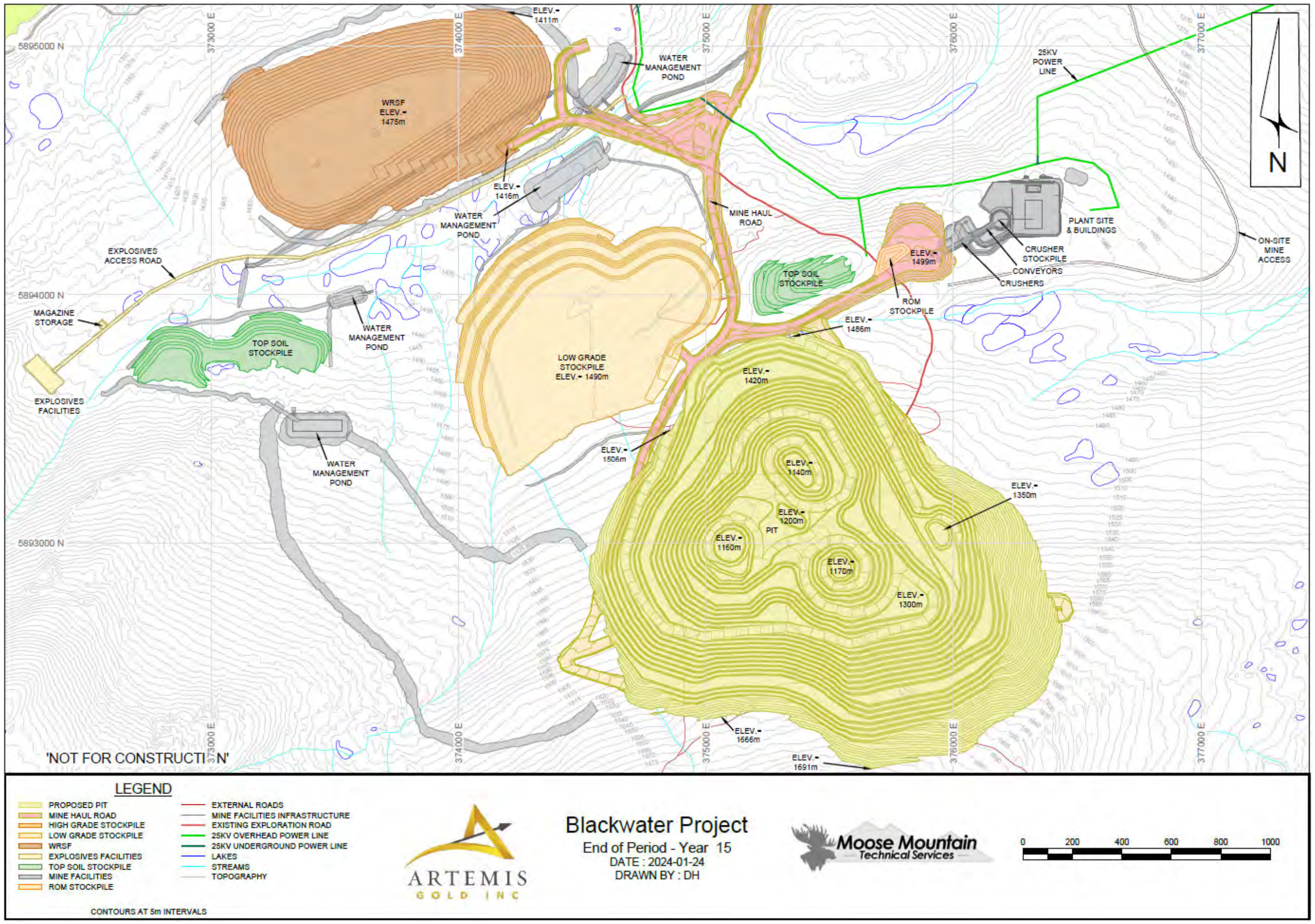
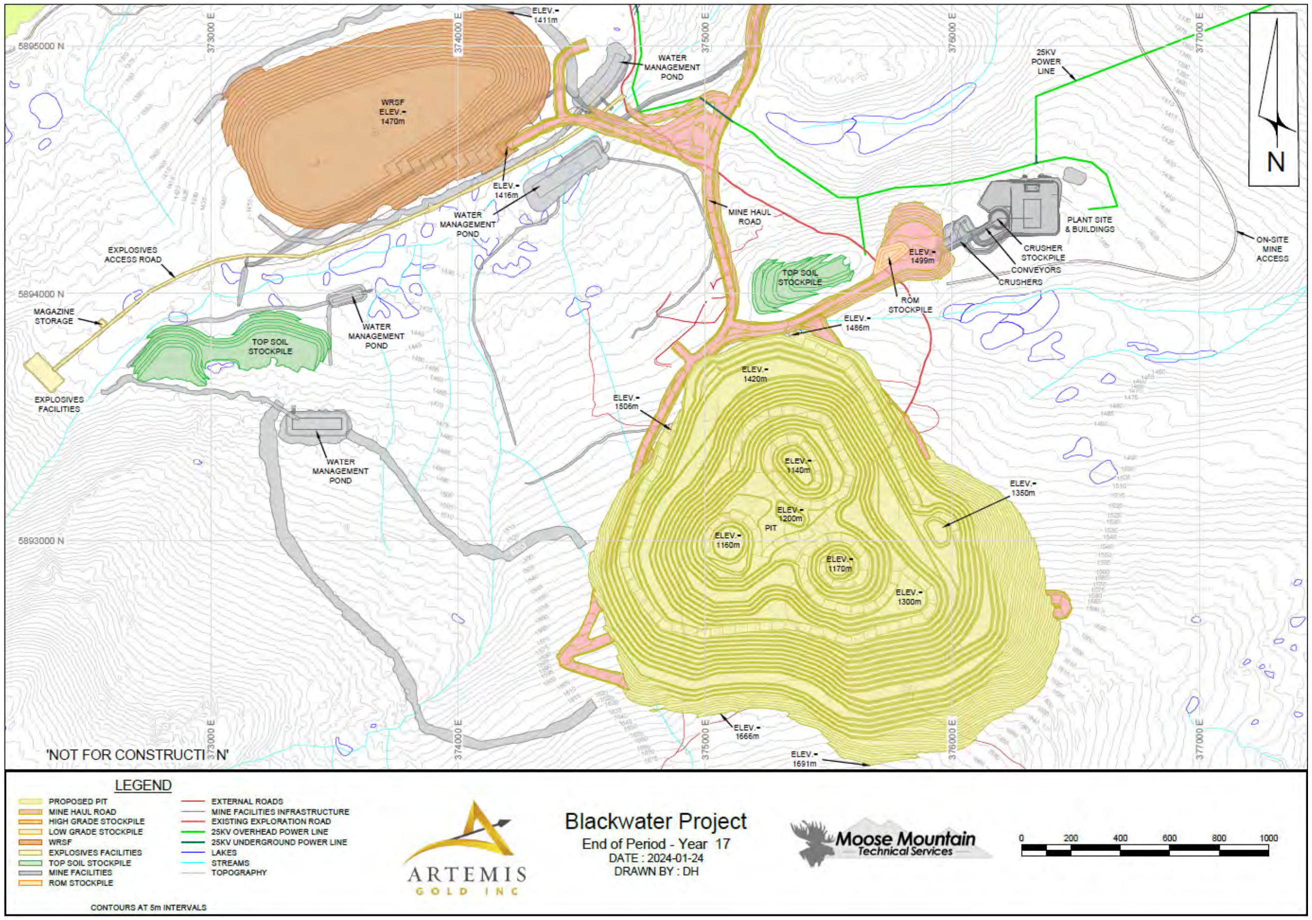


Figure 16-19: EOP Mine Operations, Year 17



16.12 Mine Operations

Mining operations are planned to be typical of similar open pit operations in interior British Columbia.

Grade control drilling will be carried out to better delineate the resource in upcoming benches. An ore control system is planned to provide field control for the loading equipment to selectively mine ore-grade material separately from the waste.

In situ rock will be drilled and blasted on 10 m benches to create suitable fragmentation for efficient loading and hauling of both ore and waste rock. There may be a requirement for frost blasting in the winter months; otherwise, no drilling or blasting is planned for the overburden materials. Various drill and blast patterns and powder factors are planned for various in pit materials, as well as for wet and dry in situ conditions. Target powder factors average 0.25 kg/t in ore and 0.22 kg/t in waste. Cushion blasting will be used for any blast patterns adjacent to an interim or final pit wall to prevent overbreak of the wall and to maintain its overall stability and integrity. This will also reduce the surface area of the ultimate walls and limit acid production and metal leaching.

Drilling activities are planned to fall under a contract service arrangement for the first three years of operations, with contractor supplied diesel run drills, then planned to convert to an Owner-operated fleet of electric powered drills for the remainder of the LOM.

The blasting activities are planned to fall under a contract service agreement with the explosive supplier. Blasting in both wet and dry conditions is proposed to be done using a blended emulsion product, with the proportion of emulsion varying with in hole water conditions. On average an estimated 25% of blast holes are expected to be wet. The explosives storage facility will be located to the northwest of the pit.

Loading in ore zones will be completed with hydraulic excavators on 10 m benches, and possibly 5 m split benches if required for ore control. Loading in waste zones will be completed with hydraulic front shovels and wheel loaders on 10 m benches.

Ore and waste rock will be hauled out of the pit and to scheduled destinations with off-highway rigid-frame haul trucks.

Mine pit services will include:

- Haul road maintenance;
- Pit floor and ramp maintenance;
- Stockpile maintenance, including spreading of TSF destined materials;
- Ditching;
- Dewatering;
- Fuel and lube services;

- Snow removal;
- Lighting;
- Cable handling;
- Transporting personnel and operating supplies;
- Mine rescue.

Direct mining operations and mine fleet maintenance are planned as an Owner's fleet, with the exception of the contract drill and blast services. Equipment ownership and labour will be undercharged to mine operations.

Mining operations are based on 365 operating days per year with two 12-hour shifts per day. An allowance of 10 days of no production was built into the mine schedule to allow for adverse weather conditions.

The number of hourly mine operations personnel, including hourly maintenance personnel, will peak at 330 persons. Due to the shift rotation, only one-quarter of full personnel complement will be on shift at a given time. Salaried personnel of approximately 50 persons will be required for mine operations, including the mine and maintenance supervision, mine engineering, and geology departments.

16.13 Mine Equipment

Grade control drilling will be carried out with 140 mm (5.5") diesel RC drills, with sampling and assaying on 3 m intervals. Production drilling will be carried out with 250 mm (10”) electric rotary drills and 200 mm (8”) diesel rotary drills.

Reliable mining equipment commonly found in the open pit mining industry has been selected and properly sized for the loading and hauling fleet. Hydraulic excavators (22 m^3 bucket) are proposed for ore loading, based on their ability to minimize losses and dilution for the ore control operations. Hydraulic front shovels (34 m^3 bucket) are proposed for waste loading based on their efficient pass match to the haulers and productivity on 10 m benches. A front-end wheel loader (12 m^3 bucket) is also proposed to load the crusher when required, and back up the main loading fleet.

Initially all equipment is planned to be diesel driven. The pit is planned to be electrified in Year 1, and additional production drills and hydraulic front shovels purchased in this period, and beyond, are proposed as electric drive. The diesel driven equipment will continue to operate after the pit is electrified. There is also potential to retrofit existing diesel drive units.

Rigid-frame haulers (240 t payload) are proposed to be flexible enough to meet the targeted production levels and to maintain productivity of the loading units. Three smaller rigid frame haulers (140 t payload) and two articulated haulers (40 t payload) are proposed to supplement the fleet and provide additional flexibility for construction of the pits, haul roads, and tailings dam, as well as mining of pit bottom benches.

Graders (5.5 m and 4.9 m blade) will be used to maintain the haul routes for the haul trucks and other equipment within the pits and on all routes to the various waste storage locations and the crusher. Haul trucks that are outfitted with a water tank (115 kL) are also included for haul road maintenance.

Track dozers (450 kW) are included to handle waste rock and overburden to the various construction and waste storage locations. Track dozers (325 kW) are included to support in pit mining activities.

A wheel loader (7.0 m^3 bucket) and hydraulic excavators (4.5 m^3 and 3.0 m^3 bucket) are included as pit support, ore control support, floor cleanup, loading tools for the articulated haulers, ditching tools, and back-up loaders for the main fleet.

Custom fuel/lube trucks are included for mobile fuel/lube support.

A cable reeler is included once the pit is electrified to handle electric cable movements for the drills and shovels.

Various small mobile equipment pieces are proposed to handle all other pit service and mobile equipment maintenance functions.

Pits will be dewatered with conventional dewatering equipment (skid-mounted diesel pumps).

Mine fleet maintenance activities will generally be performed in the maintenance shop facilities that will be located within 1 km of the northern pit limits.

Primary annual mining equipment requirements for the LOM plan are summarized in Table 16-7. Annual support equipment requirements for the LOM plan are listed in Table 16-8.

Table 16-7: Primary Mining Fleet Requirements on an Annual Basis

Purpose	Type	Y1	Y2	Y3	Y4–Y5	Y6–Y8	Y9–Y12	Y13	Y14	Y15	Y16	Y 17
Drilling	Electric rotary tracked drill, 250 mm (10”) holes	—	—	—	3	3	3	2	2	1	—	—
	Diesel rotary tracked drill, 200 mm (8”) holes	—	—	—	5	5	4	4	3	1	—	—
	Diesel RC tracked drill, 140 mm (5.5”) holes	1	1	2	2	2	2	2	2	2	—	—
Loading	Electric hydraulic front shovel, 34 m^3 bucket	1	1	2	3	3	3	2	2	2	2	1
	Diesel hydraulic excavator, 22 m^3 bucket	2	2	3	3	3	3	2	2	1	—	—
	Wheel loader, 12 m^3 bucket	1	1	1	1	1	1	1	1	1	1	1
Hauling	Rigid frame haul truck, 240 t payload	12	17	23	28	33	33	33	26	8	8	3
	Rigid frame haul truck, 140 t payload	3	3	3	3	3	3	3	3	3	3	3
	Articulated haul truck, 40 t payload	2	2	2	2	2	2	2	2	2	2	2

Note: Y = year.

Table 16-8: Support Equipment Requirements on an Annual Basis

Unit	Function	Maximum Units
Motor grader (5.5 m blade)	Haul road maintenance	4
Motor grader (4.9 m blade)	Haul road/site road maintenance	1
Water/gravel truck	Haul road maintenance	3
Track dozer (450 kW)	Dump maintenance, construction support	4
Track dozer (325 kW)	Pit maintenance, shovel support, snow clearing, site prep, construction	4
Wheel loader (7 m^3)	Pit maintenance, shovel support, snow clearing, site prep, construction	2
Hydraulic excavator (4.5 m^3)	Ore cleaning, prep for ore loading, pit support	2
Hydraulic excavator (3.0 m^3)	Ditching, construction activities	2
Fuel and lube truck	Fuel/lube support of excavators, wheel loader, drills, and support equipment	3
Shuttle bus	Employee transportation	4
Pickup trucks	Staff transportation	15
Light plants (6 kW)	Pit lighting	9
Water pumps (150 m^3/h)	Pit sump dewatering	6
On-highway dump truck	Utility material movement	2
Flatbed picker truck	Material transport, pump crew support	1
Maintenance trucks	Mobile maintenance crew and tool transport	4
Mobile crane (30 t capacity)	Mobile maintenance material handling	1
Float trailer (55 t capacity)	Equipment transport	1
Forklift (5 t capacity)	Shop material and tire handling	1
Cable reeler (WL, 10 t capacity)	Shovel and drill support	1
Scissor lift	Maintenance support	1
Mobile manlift	Mobile maintenance support	1

17.0 RECOVERY METHODS

17.1 Introduction

The process plant design is based on a metallurgical flowsheet designed for recovery with minimum operating costs using the testwork results discussed in Section 13. The flowsheet is based upon unit operations that are well proven in industry.

The process plant consists of a crushing circuit, milling, gravity, pre-oxidation, leach and CIL circuit, cyanide detox, and an elution plant. A second plant gold room will be included to house additional electrowinning cells as well as new gold pouring facilities. The existing plant reagents, oxygen, air, and water services will be re-used and or expanded as required. Un-thickened tailings flow by gravity to the TSF. A dedicated decant return line is used to make-up process water as required.

Plant design comprises:

- Phase 1 (existing): 6 Mt/a;
- Phase 2: expansion by 9 Mtpa to 15 Mt/a;
- Phase 3: expansion by 9.5 Mt/a to 24.5 Mt/a.

The optimization of Phase 1, will add an additional 0.5 Mt/a bringing the overall plant capacity to 25 Mt/a (Phase1 + Phase 2 + Phase 3).

The design basis assumptions for each phase are summarized in Table 17-1.

Table 17-1: Design Basis

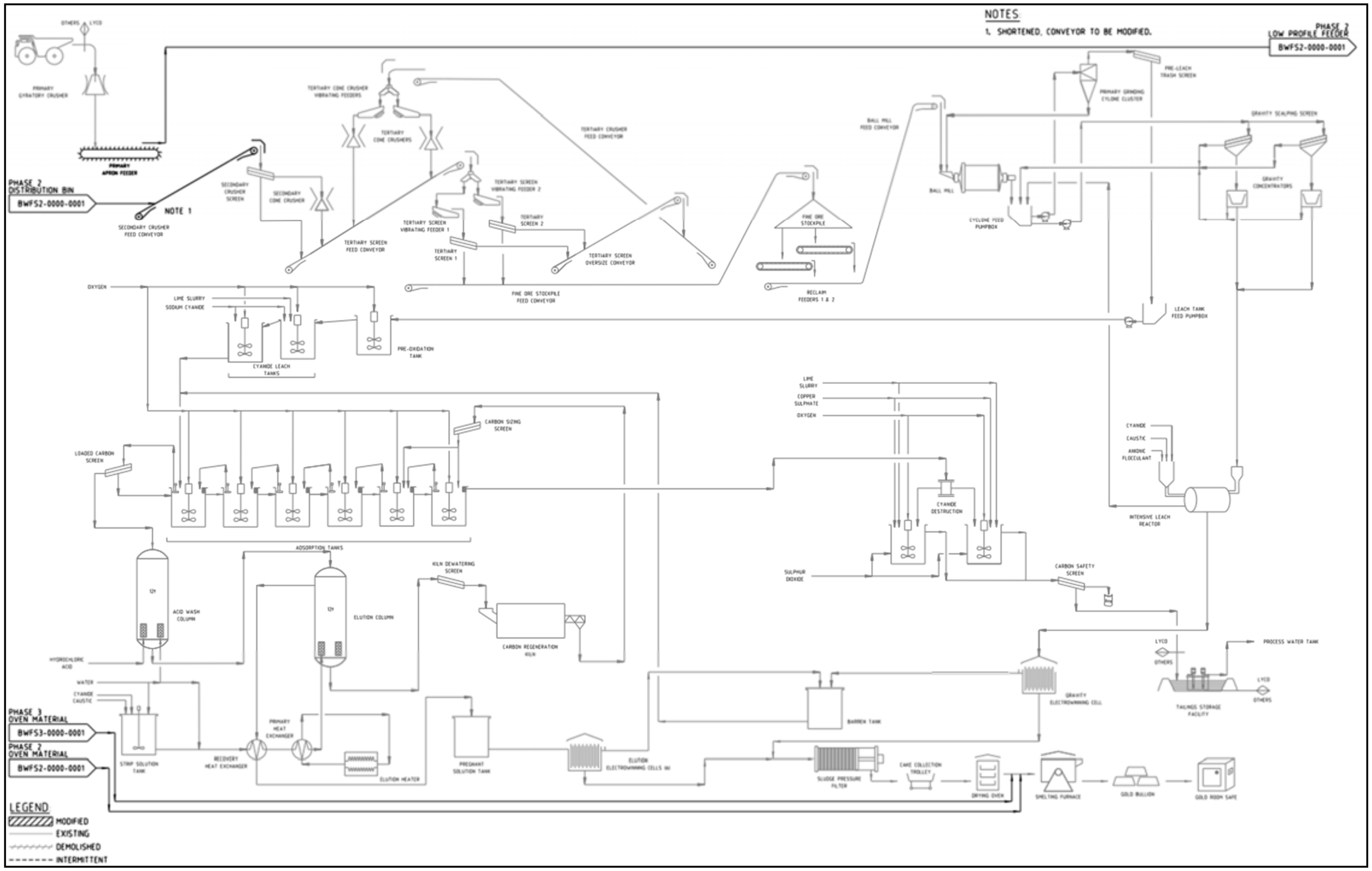
Item	Phase 1	Phase 2	Phase 3	Phase 1 Optimisation
Nominal throughput	Existing stream of 6.0 Mt/a of mill feed	Dedicated 9.0 Mt/a processing stream to achieve an overall plant throughput of 15 Mt/a.	Dedicated 9.5 Mt/a processing stream to achieve an overall plant throughput of 25 Mt/a with the optimised Phase 1 and Phase 2 circuit combined	Optimization of Phase 1 circuit to treat 6.5 Mt/a of ore feed in conjunction with Phase 3 expansion
Crushing annual utilization (dry plant)	6,132 h Primary gyratory crusher with secondary and tertiary cone crushing and a crushed ore stockpile	6,570 h Primary gyratory crusher, used for both Phase 1 and Phase 2 processing streams, distribution bin with secondary cone crushing and a crushed ore stockpile	6,570 h Primary gyratory crusher with secondary cone crushing and a crushed ore stockpile	6,132 h Primary gyratory crusher with secondary and tertiary cone crushing and a crushed ore stockpile.
Milling annual utilization (wet plant)	8,059 h Single stage ball mill grinding and classification circuit, gravity concentration, hybrid CIP/CIL with pre-aeration, cyanide destruction, and gold recovery operations.	8,059 h Single stage ball mill grinding and classification circuit, gravity concentration, hybrid CIP/CIL with pre-aeration, cyanide destruction, and gold recovery operations.	8,059 h Single stage ball mill grinding and classification circuit, gravity concentration, hybrid CIP/CIL with pre-aeration, cyanide destruction, and gold recovery operations.	8,059 h Single stage ball mill grinding and classification circuit, gravity concentration, hybrid CIP/CIL with pre-aeration, cyanide destruction, and gold recovery operations.

Note: Optimization of Phase 1 plant is undertaken in conjunction with Phase 3.

17.2 Process Flow Sheet

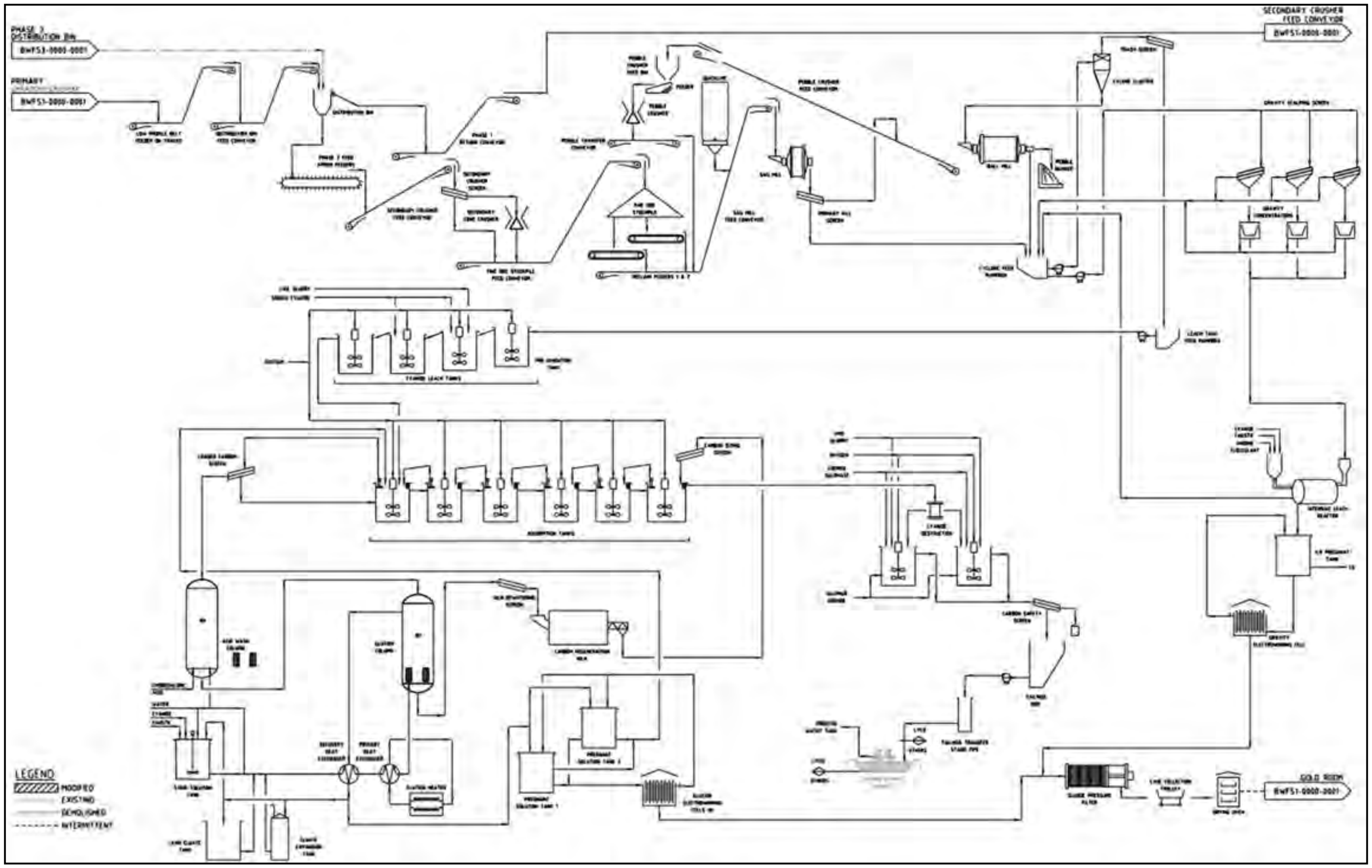
Process flowsheets for Phases 1, 2, and 3 are provided in Figure 17-1, Figure 17-2, and Figure 17-3, respectively.

Figure 17-1: Phase 1 Flowsheet



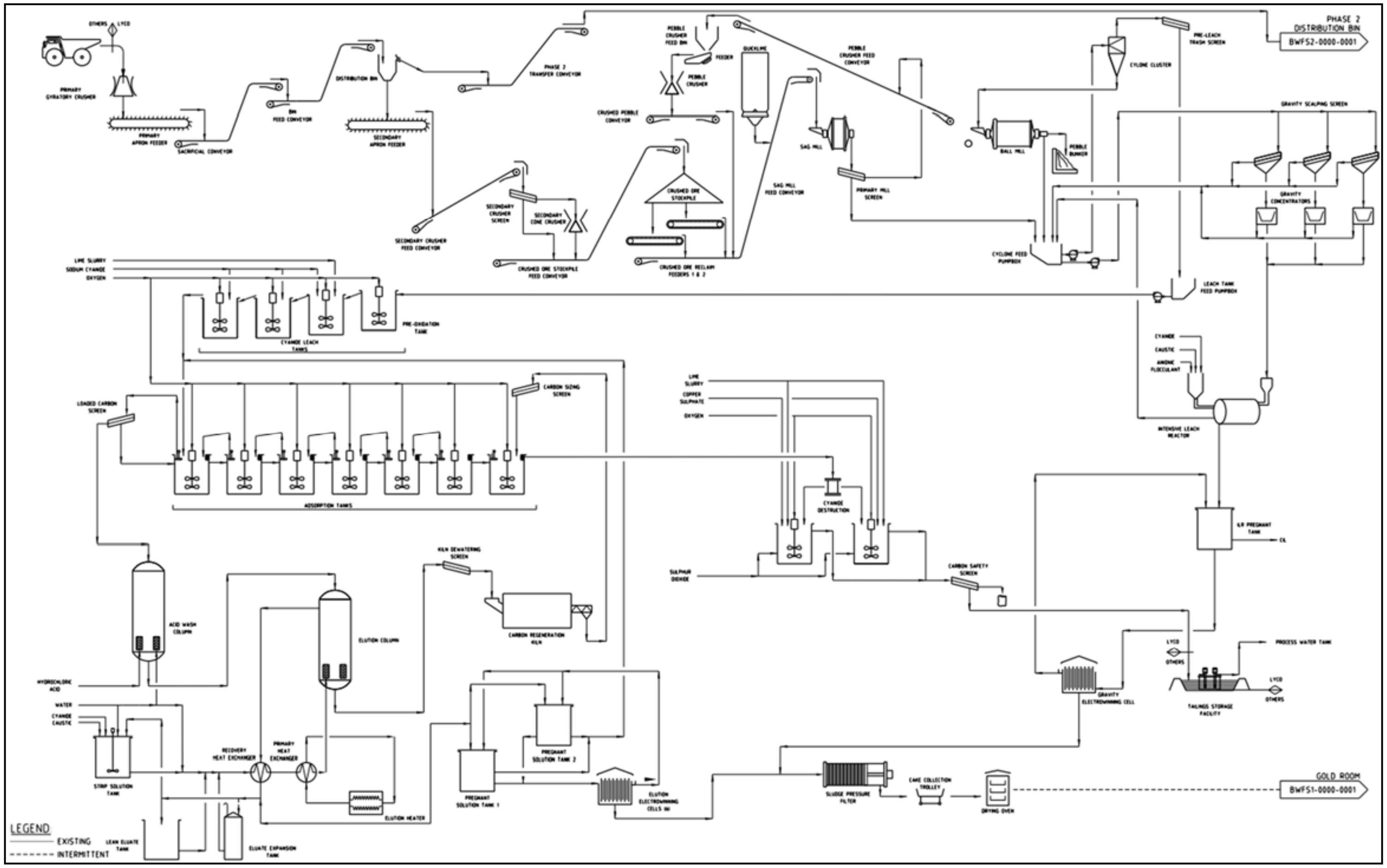
Note: Figure prepared by Lycopodium, 2024.

Figure 17-2: Phase 2 Flowsheet



Note: Figure prepared by Lycopodium, 2024.

Figure 17-3: Phase 3 Flowsheet



Note: Figure prepared by Lycopodium, 2024.

17.3 Plant Design

The process flow is summarized in Table 17-2. Plant design criteria are provided in Table 17-3.

Table 17-2: Process Flow

Item	Phase 1	Phase 2	Phase 3
Primary crushing	Primary gyratory crushing of ROM material	Primary gyratory crushing of ROM material, shared with the Phase 1 circuit, to a distribution bin splitting material back to the Phase 1 circuit with the remainder to the Phase 2 circuit.	Primary gyratory crushing of ROM material with a portion of the material transferred to the Phase 1 circuit for increased throughput via a distribution bin.
Secondary crushing	Secondary and tertiary cone crushing circuit with classification screens, to produce a fine product for storage on a fine ore stockpile ahead of the milling plant	Secondary cone crushing circuit with classification screen, to produce a crushed product for storage on a stockpile ahead of the milling plant.	
Grinding	Single-stage ball mill operating in closed circuit with classification cyclones.	SABC circuit with the SAG mill operating in closed circuit with a pebble crusher and the ball mill operating in closed circuit with classification cyclones.	
Gravity recovery	Gravity recovery of mill discharge material by semi-batch centrifugal gravity concentrators, followed by intensive cyanidation of the gravity concentrate and electrowinning of the pregnant leach solution in a dedicated electrowinning cell located in the goldroom		
Trash screening	Trash screening of cyclone overflow before leaching.		
Oxygen	Pre-aeration with pure oxygen ahead of the leach and CIL circuit with pure oxygen addition throughout.		
Washing, elution, regeneration	Acid washing of loaded carbon and AARL elution circuit followed by electrowinning and smelting to produce doré. Carbon regeneration by rotary kiln.		
Cyanide destruction	SO_2/oxygen process.		
Tailings	Tailings slurry transfer to the TSF via gravity	Tailings slurry transfer to the tailings disposal facility via pumping	

Note: ROM = run-of-mine; SAG = semi-autogenous grind, SABC = SAG, ball mill, and crusher circuit; CIL = carbon in leach; AARL = Anglo American Research Laboratory; TSF = tailings storage facility.

Table 17-3: Design Criteria

Parameter	Units	Phase 1	Phase 2	Phase 3	Phase 1 Optimisation
Plant throughput (processing stream)	Mt/a	6.0	9.0	9.5	6.5
Plant throughput (overall plant)	Mt/a	6.0	15.0	25.0	6.5
Gold head grade (LOM)	g/t Au	0.75	0.75	0.75	0.75
Silver head grade (LOM)	g/t Ag	5.78	5.78	5.78	5.78
Crushing plant annual utilization	h	6,132	6,570	6,570	6,132
Milling, leach, and refinery annual utilization	h	8,059	8,059	8,059	8,059
Bond crusher work index (CWi) – design (75th percentile)	kWh/t	15.1	15.1	15.1	15.1
Bond ball mill work index (BWi) – design (75th percentile)	kWh/t	19.1	19.1	19.1	19.1
Abrasion index (Ai) – average	g	0.168	0.168	0.168	0.168
SMC Axb (75th percentile)	kWh/t	29.1	29.1	29.1	29.1
Primary crusher lump feed size	F_{100}	800	800	800	800
Crushing plant product size, P_{80}	mm	9	49	48	9
Crushed ore stockpile live capacity	tonnes	5,956	13,401	14,145	5,956
SAG mill	MW		14.0	14.0	
Ball mill	MW	14.0	14.0	14.0	14.0
Cyclone overflow size, P_{80}	µm	150	150	150	150
Gravity gold recovery – design	%	31.5	31.5	31.5	31.5
Leaching tails solid grade (gravity tails leaching)	g(Au)/t	0.08	0.08	0.08	0.08
Pre-aeration time - target	h	2	2	2	2
Leach time - target	h	8	8	8	8
Adsorption time - target	h	16	16	16	16
Leach tails solution grade	g/m^3 Au	0.015	0.015	0.015	0.015
Sodium cyanide addition (NaCN)	kg/t material	0.73	0.73	0.73	0.73
Lime addition (at 90% CaO purity)	kg/t material	1.22	1.22	1.22	1.22
Elution column size	tonnes	12	18	18	12
Number of carbon strip per week	#	10	12	12	12
Leach tails CN_{WAD}	ppm	200	200	200	200
Detoxed tails CN_{WAD}	ppm	< 25	< 25	< 25	< 25

Note: Phase 1 optimization is implemented after Phase 3.

17.3.1 Crushing

Run of mine (ROM) ore will be trucked from the pit to the ROM pad and either directly tipped into the feed bin or reclaimed by front end loader into the bin that directly feeds the primary gyratory crusher. Phases 1 and 2 will share a primary crusher and phase 3 will have a dedicated primary crusher. Extraction of the dust from the discharge points below the crusher will be done with dry dust extraction systems.

During Phase 1, ore drawn from the primary crusher discharge vault will feed a secondary crusher screen, of which the middlings and oversize feed the secondary cone crusher in open circuit. The secondary crusher product, secondary screen undersize and tertiary crusher product will combine to feed the tertiary screen twin bin, with two tertiary crusher screens and dedicated feeders for each. Each tertiary crusher screen oversize and middlings will feed a dedicated tertiary crusher while the screen undersize will be conveyed to an eight hour live stockpile which feeds a dedicated ball mill circuit.

In Phase 2, ore drawn from the primary crusher discharge vault will be conveyed to a distribution bin, which will return a fraction to the Phase 1 secondary crusher screen feed conveyor while the remainder will be conveyed to the Phase 2 secondary crusher screen. This secondary crusher screen oversize and middlings will feed a secondary cone crusher in open circuit. Secondary crusher product and secondary screen undersize will combine to feed a second 12 hour live stockpile which will feed a dedicated semi-autogenous grind (SAG), ball and crusher (SABC) circuit.

During Phase 3, ore drawn from the Phase 3 primary crusher discharge vault will be conveyed to a distribution bin, with a portion transferred via the Phase 2 distribution bin to the Phase 1 circuit with the remainder reporting to the secondary crusher screen. This secondary crusher screen oversize and middlings will feed a secondary cone crusher in open circuit. Secondary crusher product and secondary screen undersize will combine to feed a third 12h live stockpile which will feeds dedicated SABC circuit.

Dry dust extraction systems will be placed at key transfer points, weightometers will be strategically placed for control and accounting purposes, and tramp metal magnet systems will be located before secondary cone crushing to protect downstream equipment.

17.3.2 Crushed Ore Reclaiming and Conveying

Crushed ore will be withdrawn from a reclaim tunnel beneath the covered stockpile, by two variable speed reclaim belt feeders (Phase 1) and by two variable speed apron feeders (Phases 2 and 3) (both running with option for one to be on standby). Each feeder is sized for 100% of plant capacity. The feeders will discharge onto the mill feed conveyor, which will convey the crushed ore to the primary mill feed box. The mill feed conveyor will be fitted with a weightometer, used for controlling the speed of the belt feeder and hence the feed rate to the grinding circuit. Extraction of the dust from the feeder discharge points will be achieved using dry dust extraction system. Crushed ore reclaim tunnel ventilation fans will displace the air in the crushed ore reclaim tunnel.

For Phases 2 and 3, quick lime will be added directly to the mill feed conveyor via a lime silo using a rotary valve. Lime addition will be adjusted by a pH meter in the leach circuit and by the mill feed rate as measured by the mill feed conveyor weightometer.

17.3.3 Grinding and Classification Circuit

During Phase 1 operations, crushed ore will be fed to a single-stage ball mill. Mill discharge will pass through a trommel screen where the oversize (scats) will be directed to a bunker. Screen underflow will report to the mill sump before being pumped to hydro-cyclones for classification. Cyclone overflow, at the targeted grind size P_{80} of 150 µm, will gravitate to the trash screen for grit and woodchip removal. Cyclone underflow will gravitate back to the ball mill feed for further grinding.

In Phases 2 and 3, crushed ore will be fed to a SAG mill in closed loop with a pebble crusher. SAG mill discharge will pass through a vibrating pebble dewatering screen where the oversize (pebbles) will be directed to a pebble crushing circuit which will recycle back to the SAG mill feed. Pebble dewatering screen undersize will gravitate to the mill discharge sump before being pumped to hydro-cyclones in closed loop with a ball mill. Cyclone overflow, at the targeted grind size P_{80} of 150 µm, will gravitate to the trash screen for grit and woodchip removal. Cyclone underflow will report to a ball mill for further grinding. Ball mill discharge will pass through a trommel screen with oversize (scats) reporting to a bunker and the undersize combining with SAG mill discharge in the mill discharge sump.

All grinding media for the SAG and ball mills will be loaded via kibbles into the media chutes discharging into the respective mill’s feedbox.

17.3.4 Gravity Recovery Circuit

The gravity circuit will comprise two (Phase 1) or three (Phase 2 and 3) centrifugal concentrators complete with two (Phase 1) or three (Phase 2 and 3) gravity scalping screens. Feed to the circuit will be pumped with a dedicated pumping system from the mill discharge hopper to distributor splitting the feed to gravity scalping screen. Gravity scalping screen oversize will gravitate back to the mill discharge hopper. Each scalping screen undersize will gravitate to a dedicated gravity concentrator. Gravity concentrator tails will gravitate to the mill discharge hopper. During the batch cycles the scalping screen undersize will periodically bypass the concentrators and be directed to the mill discharge hopper.

The operation of the gravity concentrator will be semi-batch and the gravity concentrate will be collected in an intensive leach reactor (ILR) feed storage hopper and subsequently leached by the ILR circuit.

17.3.5 Intensive Leach Reactor

Concentrate from the gravity concentrator will gravitate to the ILR to recover the contained gold by cyanide leaching. Each phase circuit will have its own dedicated ILR.

The concentrate from the gravity concentrators will gravitate to the ILR gravity concentrate storage hopper and will then transferred be to the ILR drum after the predetermined weight is reached at the storage hopper, at which point batch processing will be initiated.

Process water will be added to the ILR drum. ILR leach solution (~1.2%w/v NaCN and ~2.0%w/v NaOH) will be stored in the ILR solution storage hopper and transferred to the ILR drum. Caustic soda solution will be added to maintain the pH above 10.5. The ILR circulating pump will split the solution between the ILR solution hopper and the ILR drum. Oxygen will be added to the circulation line through a sparger during leaching, for a predetermined period of time. Flocculant will be added to the ILR pump hopper and circulate through the ILR solution storage hopper and ILR drum. Flocculant will allow the fine flocculated solids to settle in the ILR reactor drum.

After completion of leaching, the ILR drum will be stopped, and the solution will be transferred to the ILR solution storage hopper to settle the solids. Sodium hydroxide will then be added to the solution storage hopper and the decanted clarified solution will be drained to the ILR pump hopper and pumped to the ILR electrowinning feed tank in the goldroom as a pregnant solution for gold sludge recovery using a dedicated electrowinning cell. The sludge will be combined with the sludge from the carbon electrowinning cells and be smelted, or may be separately smelted for metallurgical accounting purposes.

Once the pregnant solution transfer is complete, the ILR drum and ILR solution storage hopper will be washed, and the slurry will be collected in the ILR pump hopper. The slurry will then be pumped to the mill discharge hopper.

17.3.6 Trash Screening

Cyclone overflow will gravitate via a sampling system to a trash screen, to remove foreign material prior to leaching. Trash will report to the trash bins, which will be periodically emptied. Screen undersize will gravitate to a leach tank feed pump box from where the slurry will be pumped to the leach circuit. Each phase will have a dedicated trash screening and pumping system.

An automatic slurry sampler, installed on the feed to the trash screens will collect a representative sample of the stream for plant control and metallurgical accounting purposes.

17.3.7 Pre-Aeration, Leach, and CIL Circuit

The leach circuit will consist of one pre-aeration tank (per Phase), two (Phase 1) or three (Phase 2 and 3) leach tanks, and six (Phase 1 and 2) or seven (Phase 3) carbon-in-leach (CIL) tanks. Slurry can bypass the pre-aeration tank or any of the leach and CIL tanks when necessary. Oxygen will be added to all tanks via the agitator shaft to maintain adequate dissolved oxygen levels for leaching. Cyanide solution can be added to all leach tanks and the first CIL tank in each phase circuit as required, but generally will only be added to the first two tanks with a control loop with the cyanide analyzer.

Fresh carbon or regenerated carbon from the respective phase's carbon regeneration circuit will be returned to the last tank of the CIL circuit with the alternative option of addition to the second last CIL tank and will be advanced counter-currently to the slurry flow by recessed impeller carbon advance pumps in each CIL tank. The intertank screen in each CIL tank will retain the carbon whilst allowing the slurry to flow by gravity to the downstream tanks. This counter-current process will be repeated until the gold-loaded carbon reaches the first CIL tank. Recessed impeller pumps will be used to transfer slurry from the first tank to the respective phase's loaded carbon screen mounted above the acid wash column in the elution circuit. Undersize from the loaded carbon screen will gravitate to the leach feed distribution box. The recessed impeller pump in CIL tank 2 will be used to transfer slurry from the second tank to the loaded carbon screen when the first tank is offline.

Slurry from the last CIL tank will gravitate to the respective phase's cyanide destruction tank distributor. For phase 1 the main pH control will be done by a milk of lime slurry to the pre-aeration tank. For all phases there will be an option to add milk of lime to the leach tanks as required to maintain the pH above 10.5 during when quick lime addition to mill feed conveyor is not sufficient (Phase 2 and 3) or milk of lime addition to the pre-aeration tank (Phase 1). pH measurement and control will be included in the pre-aeration tank to ensure a sufficiently high pH is maintained prior to cyanide addition.

17.3.8 Carbon Acid Wash, Elution and Regeneration Circuit

Each Phase will have a dedicated acid wash, elution, and regeneration circuit. Phase 1 will use a 12 t carbon stripping system and phases 2 and 3 an 18 t carbon stripping system.

Carbon Acid Wash

Before carbon stripping (elution), loaded carbon will be treated with a 3% hydrochloric acid solution to remove calcium, magnesium, and other salt deposits that would otherwise render the elution less efficient or be 'baked on' in the subsequent elution and carbon regeneration steps and ultimately foul the carbon.

Loaded carbon from the loaded carbon recovery screen will gravitate to the acid wash column.

Entrained water will be drained from the column which will then be refilled with a 3% hydrochloric acid solution, from the bottom up. Once the column is filled with the carbon, it will be left to soak in the acid for 30 minutes, after which the spent acid will be drained for reuse or discarded to the tailings hopper and the carbon rinsed and pH corrected with caustic.

The acid washed carbon will then be hydraulically transferred to the elution column for carbon stripping.

Carbon Stripping (Elution)

Carbon stripping (elution) will use the Anglo American Research Laboratory (AARL) process (Phase 1) or split AARL process (Phase 2 and 3).

The elution sequence will commence with the injection of a set volume of strip solution into the bottom of the elution column. The strip solution will be made up of 3.0% w/w NaOH and 2.0% w/w NaCN. Strip solution will be re-circulated through the strip solution heater to bring the solution up to 130°C before entering the elution column. Eluate leaving the elution column will pass through the recovery heat exchanger to either the strip solution tank, lean eluate tank (Phase 2 and 3 only) or one of two pregnant solution tanks depending on the sequence step. For Phases 2 and 3, lean eluate tank solution will be used in the elution step and refilled in the rinse and cooling steps in preparation for the next elution.

Upon completion of the cool down sequence, the carbon will be hydraulically transferred to the carbon regeneration kiln feed hopper via a dewatering screen. A bypass option will be provided to bypass the carbon regeneration if required. Anti-scalant dosing and sulphamic acid cleaning systems will be provided.

Electrowinning and Goldroom

Gold is recovered from the pregnant eluate by electrowinning and smelted to produce doré bars.

A new gold room is built during Phase 2 execution with all Phase 1 gold room equipment relocated to the new building. During Phase 3 the gold room is expanded to accommodate the additional equipment. Each Phase has dedicated electrowinning circuits and sludge filters for each dedicated elution and gravity gold circuits. The drying ovens, furnace and other facilities are shared for all Phases.

Pregnant eluate from each elution circuit is transferred to one of two pregnant solution tanks to allow an elution to occur to one tank while the other is recirculating through the electrowinning cells or pumping out barren solution to the leach circuit. During an electrowinning cycle pregnant eluate passes through four of six electrowinning cells per phase with stainless steel mesh cathodes. Gold is deposited on the cathodes and the solution returns by gravity to the selected pregnant solution tank for recirculation until it is deemed barren, or the sequence time has been complete.

ILR pregnant solution from each ILR electrowinning tank passes through one additional dedicated electrowinning cell per phase to process ILR pregnant solution. Barren ILR solution is pumped back to the leach feed distribution box in the leach circuit.

The gold-rich sludge is washed off the steel cathodes in the electrowinning cells using high pressure water sprays and gravitates to the filter feed hopper (one per phase). The sludge is then pumped into a sludge pressure filter (one per phase), dried, mixed with fluxes, and smelted in an electric induction furnace to produce gold doré.

The electrowinning and smelting process takes place within a secure and supervised goldroom equipped with access control, and security systems.

Carbon Reactivation

Carbon will be reactivated in an electric rotary kiln, with one kiln per phase. Dewatered barren carbon from the stripping circuit will be held in a kiln feed hopper. A screw feeder will meter the carbon into the reactivation kiln, where it will be heated to 650–750°C in an atmosphere of superheated steam to restore the carbon activity. Carbon discharged from the kiln will be quenched in water and pumped to a carbon sizing screen to remove undersized carbon fragments. The undersized fine carbon will be pumped to the carbon safety screen and the oversize will be pumped to either of the last two CIL tanks.

As carbon will be lost by attrition, new carbon will be added to the circuit via the carbon quench vessel. The new carbon will then be transferred via the carbon sizing screen into the circuit the same way as reactivated carbon.

17.3.9 Cyanide Destruction

The CIL tails will gravitate to the cyanide destruction distribution box and into both two cyanide destruction tanks operating in parallel. Total residence time in series will be approximately 90 minutes to reduce maximum weakly acid dissociable cyanide (CN_{WAD}) design levels from 200 ppm (100 ppm operating) to <25 ppm.

Cyanide destruction will be undertaken using the SO_2/oxygen method. The reagents required will be oxygen, milk of lime, copper sulphate, and SO_2 gas. The cyanide destruction tanks will be equipped with oxygen addition points and an agitator to ensure that the oxygen and the reagents are thoroughly mixed with the tailings slurry. After the cyanide destruction process, the slurry will gravitate to the vibrating carbon safety screen to recover any carbon leaking from worn screens or overflowing tanks. Screen underflow will gravitate to the TSF. Screen oversize (recovered carbon) will be collected in a fine carbon collection bin for potential return to the circuit.

17.4 Equipment

The major equipment requirements are summarized in Table 17-4.

Table 17-4: Equipment List

Item	Note
Primary crushing	Gyratory crushers
	Primary crusher discharge bins
	Apron feeders, with a low profile belt feeder for Phase 2
	Distribution bins
	Distribution bin feed conveyors and secondary crusher feed conveyors
	Double deck secondary crusher screens
	Secondary crushers
	Tertiary crusher screen feed conveyor (Phase 1 stream only)
	Tertiary crusher screen feed twin bin (Phase 1 only)
	Two tertiary crusher screen vibrating feeders (Phase 1 only)
	Two double deck tertiary crusher screens (Phase 1 only)
	Tertiary crusher feed conveyor (Phase 1 only)
	Twin tertiary crusher feed twin bin (Phase 1 only)
	Two tertiary crusher vibrating feeders (Phase 1 only)
	Two tertiary crushers (Phase 1 only)
	Crushed ore stockpile feed conveyors
	Crushed ore stockpile (8 h Phase 1, 12 h Phase 2 and 3)
	Dust extraction at strategic equipment discharge points
Crushed ore reclaim circuit	Two crushed ore reclaim belt feeders (Phase 1)
	Two crushed ore reclaim apron feeders (Phase 2 and 3)
	Mill feed conveyor
	Lime system (Phase 2 and 3)
Grinding and classification circuit	SAG mill (Phase 2 and 3)
	Primary mill vibrating pebble screen (Phase 2 and 3)
	Pebble crusher feed conveyor (Phase 2 and 3)
	Pebble crusher feed bin (Phase 2 and 3)
	Pebble crusher (Phase 2 and 3)
	Crushed pebble conveyor (Phase 2 and 3)
	Ball mill with trommel screen
	Classification cyclones
	Mill discharge hopper and pumps
	Milling feed and discharge area sump pumps
Gravity recovery circuit	Two (Phase 1) or three (Phase 2 and 3) gravity scalping screens

Item	Note
	Two (Phase 1) or three (Phase 2 and 3) gravity concentrators
Intensive leach reactor circuit	ILR feed storage tank
	ILR drum
	ILR pump hopper
	ILR solution storage hopper
	ILR circulating pump
	ILR area sump pump with imbedded gold trap
Trash screening circuit	Trash screen
	Leach feed automatic sampler
	Leach feed hopper and pumps
Pre-aeration, leach, and CIL circuit	Pre-aeration tank with agitator
	Two (Phase 1) or three (Phase 2 and 3) leach tanks with agitators
	Six (Phase 1 and 2) or seven (Phase 3) CIL tanks with agitators, interstage screens and recessed impeller carbon transfer pumps
	Gantry crane
	Cyanide analyzer and HCN gas monitor
	CIL area sump pumps
Acid wash circuit	Loaded carbon screen
	Acid wash column
Stripping circuit	Elution column
	Strip solution heater with heat input and heat recovery heat exchangers
	Strip solution tank
	Lean eluate tank (Phases 2 and 3)
	Elution area sump pump
Electrowinning circuit and goldroom	Two pregnant solution tanks (per Phase).
	ILR electrowinning feed tank (one per Phase)
	Six electrowinning cells with dedicated rectifiers (per Phase)
	Filter feed hopper and sludge pressure filter (one set per Phase)
	Drying oven (one after Phase 1, two after Phase 2, two after Phase 3)
	Flux mixer
	Electric induction smelting furnace with bullion moulds
	Bullion vault and safe
	Fume extraction and ventilation systems
	Goldroom crane
	Goldroom security system

Item	Note
	HCN & ammonia gas monitors
	Goldroom sump pump with embedded gold trap
Carbon reactivation circuit	Carbon dewatering screen (one per phase)
	Regeneration kiln including feed hopper and screw feeder (one per phase)
	Carbon quench tank and pump (one per phase)
	Carbon sizing screen (one per phase)
	Fine carbon tank and pump (one per phase)
	Regenerated carbon transfer tank and pump (one per phase)
Cyanide destruction circuit	Two agitated cyanide destruction tanks (per phase)
	Vibrating carbon safety screen (per phase)
	HCN gas monitor and CN_{WAD} analyser (per phase)
	Cyanide destruction area sump pump (per phase

17.5 Energy, Water, and Process Materials Requirements

17.5.1 Energy

Power is discussed in Section 18.11.

17.5.2 Water

Raw water will be supplied from the water management pond and depressurisation wells into a raw water storage tank.

Tailings return water and mill cooling water return will meet most of the process water requirements. Raw water and contact water will provide any additional make-up water requirements.

Potable water for plant use will be supplied from a potable water treatment plant that will be installed at the accommodations camp.

17.5.3 Consumables

The following reagent systems are required for the process: quick lime, sodium cyanide, sodium hydroxide, hydrochloric acid, copper sulphate, sulphur dioxide, flocculant, activated carbon, anti-scalant, and smelting fluxes. High pressure air and oxygen supplies will be required. Grinding media are needed for the mills.

18.0 PROJECT INFRASTRUCTURE

18.1 Introduction

The overall Blackwater Gold Mine facilities and major infrastructure cover the mine site area, TSF, WRSF, camp site, main access road, and site wide water management systems.

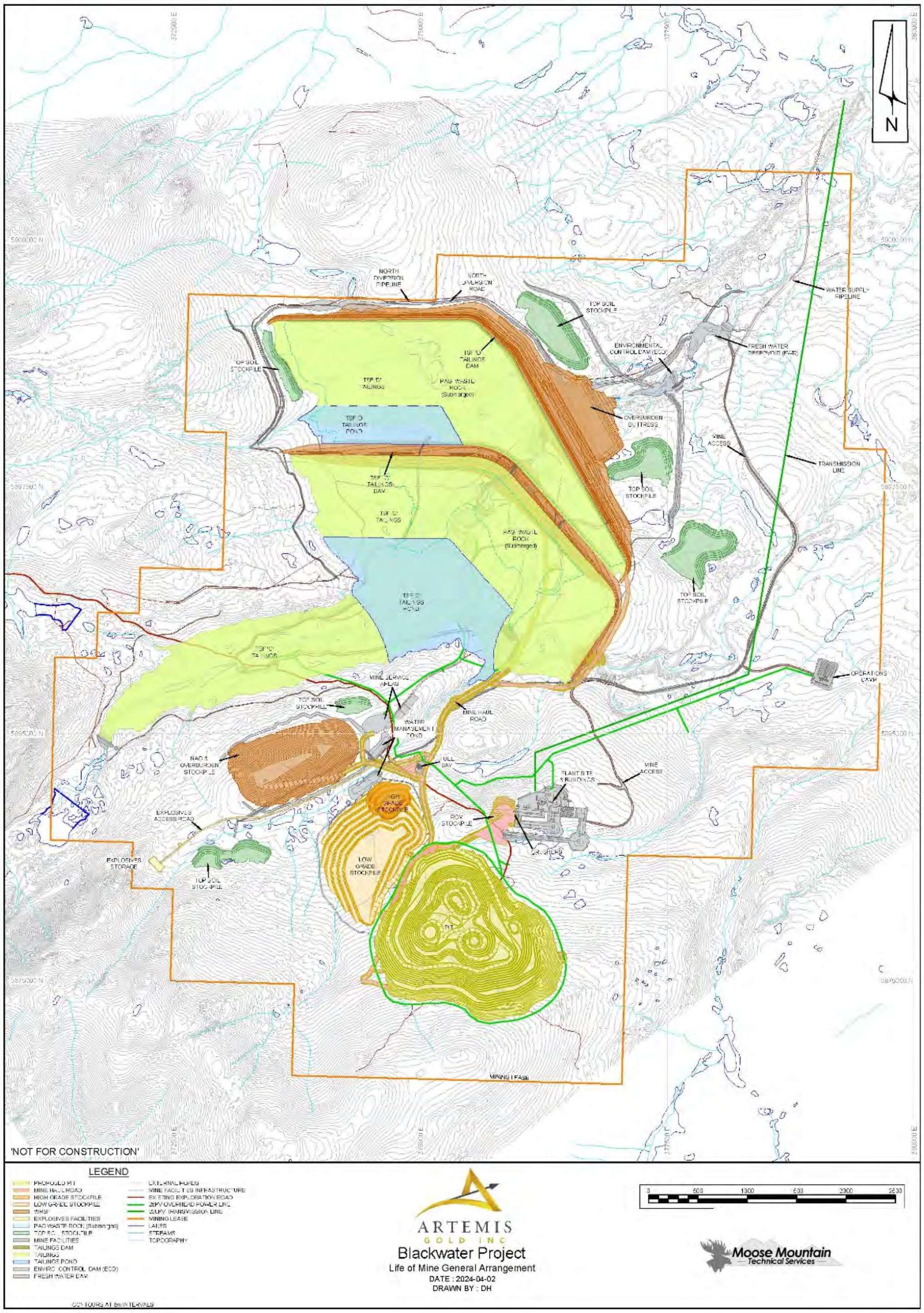
The following Phase 1 Infrastructure is assumed to be in place:

- Power supply;
- Roads (access and haul roads);
- Process plant (Phase 1);
- Tailings storage facility (stage 1) and associated water management structures;
- Mine services (heavy mine equipment (HME) workshop, wash bay, tire service area, explosive storage, bulk fuel, and lube facilities);
- Pit development (pit clearing, bench access, low-grade stockpiles, WRSFs);
- Accommodations (operations and construction, 537 beds);
- Ancillary building (mining, process, laboratory, warehousing, and security);
- Water treatment (metals and lime neutralization);
- Communications (fibre optic cable).

The 2024 Expansion Study assumes that the existing facilities will be built out as necessary to meet the Phase 2 and Phase 3 expansion requirements.

An overall layout plan is presented in Figure 18-1.

Figure 18-1: Blackwater Gold Mine Infrastructure Location Map



18.2 Built Infrastructure

A description of the key non-process infrastructure required is provided in Table 18-1.

Table 18-1: Infrastructure

Item	Note
Laboratory building	Sized to accommodate the proposed mine expansion and as such no expansion of the laboratory building will be required.
Mine fleet maintenance facilities	A staged approach is taken to construction and use. The existing facility will be expanded in Years 2 and 3 of operations with the addition of four fabric covered bay structures (each 22 x 21 m) will be built within the existing mine services area. Two of the bays will have overhead crane availability, with a 15 t capacity, and clearance in the bays for 230 t payload class rigid frame haul trucks. The expanded facility will include: • 5 x HME service bays; • 1 x support equipment service bay; • 1 x washdown bay; • 1 x tire service area.
Mine offices	Offices for engineering and geology will be located adjacent the ROM pad area. No expansion of this facility will be required for Phase 2 or Phase 3.
Mine dry	Will consist of a modular change room complex will be located at the operations camp. The mine dry will be expanded in Year 3 by retro-fitting existing facilities on site to meet the increasing personnel numbers. The final capacity will be 385 persons.
Fuel	The existing bulk fuel facility will have two 200,000 L single-walled vertical storage tanks, each with receiving/dispensing capabilities, and one diesel exhaust fluid system. The fuel storage is located within a high-density polyethylene (HDPE)-lined containment area, and the fuel farm allows for filling of two Cat 793 rock trucks simultaneously. The fuel farm will be expanded to accommodate the expanding fleet required for the mine operations. An additional 200,000 L of vertical storge will be added to accommodate the expanding mine fleet. This facility will be located within the existing HDPE-lined containment. Expansion of the diesel receiving/dispensing facility, and diesel exhaust fluid storage and dispensing module will also be undertaken. The system will provide a minimum storage capacity of 4.5 days' worth of diesel fuel.
Potable water	Will be supplied to the camp, process plant and mine services area via water well TW13-02 which is approximately 240 m to the north of the operations camp. Water for domestic use is stored in two 57 m^3 reservoirs located in each area. Water treatment plants located at the operations camp and process plant will treat the water to potable standards. Potable water for the mine services area will be trucked and stored for use. Separate fire protection and potable water distribution systems will convey water from the reservoirs to the camp buildings.
Sewage and sanitation	Sanitary sewage will be collected from the individual camp buildings and stored in four holding tanks providing approximately three days storage capacity. Initially, sewage will be transported via pump truck to a treatment plant located to the south of the process facility prior to this facility being relocated in Year 3 to the operations camp. Treated effluent will be pumped to a

Item	Note
	disposal field located away from the camp pad. In tear 5 treatment will be expanded with the addition of a 300-person treatment facility at the operations camp to meet peak personnel requirements.
Security	Several security systems will be in place to ensure the security of personnel, materials, and product. Wherever possible, security systems will be automated or computerized to reduce requirements for security staff. Primary access to the site will be regulated by swipe cards, which will be issued to all employees. This system will control access to highly-sensitive areas on site, such as the gold room and process control server room. Visual security will be provided by a network of closed-circuit television cameras installed throughout the site. Camera imagery will be fed to the central security control room and the mill process plant control room. Security systems will be expanded to accommodate Phase 2 and Phase 3.

Building requirements for the process area include:

- Mill building;
- Gravity circuit building;
- Gold room;
- Leach hut;
- Process and potable water pump houses;
- Reagent and grinding media storage sheds.

18.3 Road and Logistics

On site roads will provide access to the plant site, mine services area, accommodation, and explosives storage facility. These roads will support two-way, light vehicle traffic and will be wider as required for the passage of mine trucks.

A new 15 km-long road will be built as part of the fresh water supply system along the pipeline corridor connecting the pumping station on Tatelkuz Lake to the mine site in Year 7. This road will make partial use of existing trails.

18.4 Borrow Sources

Geotechnical and hydrogeological investigations in support of facility location and design are summarized in Section 10.7.

Naturally-occurring materials that are expected to be suitable for construction of the TSF embankment seal zone (Zone S) and shell zone (Zone C) and for general fill (Zone G) purposes for other Project components will be sourced from glacial till borrow areas. The TSF embankments will primarily be constructed from overburden and non ML NAG waste rock from the open pit.

However, when production of these construction materials directly from the open pit, or stockpiled from the open pit, is insufficient to meet construction demands, the materials will be sourced from glacial till borrow areas. Glacial till borrow areas have been developed in the TSF basin and have been identified around the TSF basin for future development.

Processed material for embankment drains and filters (Zones F and T) and other armouring (i.e., riprap) or aggregate materials will be produced from esker borrow areas. An esker borrow area has been developed in an esker complex identified near the Main Dam C Stage 1 construction area and an aggregate screening area has been established in the relatively flat area adjacent to the esker. A larger esker complex is located further downstream of the TSF area along the planned mine access road within the permitted mine boundary. This additional source represents a contingency borrow area for TSF construction and may be used as a construction material borrow area for other Project components, and possibly as a concrete aggregate source.

18.5 Stockpiles

Geotechnical and hydrogeological investigations in support of facility location and design are summarized in Section 10.7.

The stockpiling strategy and ore stockpile facilities are discussed in Section 16.9.

18.6 Waste Rock Storage Facilities

Geotechnical and hydrogeological investigations in support of facility location and design are provided in Section 10.7.

The waste rock storage facility design and capacities are discussed in Section 16.10.

18.7 Tailings Storage Facility

18.7.1 Site Selection

A tailings alternatives assessment for the Project was completed in 2015 (ERM, 2015) in response to requests from the BC Environmental Assessment Office and to meet requirements within the Canadian Environmental Assessment Act. The study included evaluation of the best available technology and best available practices for tailings and waste rock management for the Project, including a comprehensive assessment of the TSF design alternatives and management strategies for tailings and PAG/ML waste rock.

The assessment considered and compared the best available technologies and practices for tailings management, considering the safety, technical, water balance, and lifecycle costs for all Project phases, as well as the implications for environmental, health, social, and economic values. The assessment also considered how the options interact and affect values under Section 5 of the Canadian Environmental Assessment Act. The assessment demonstrated that thickened slurry tailings with PAG/ML NAG waste rock stored underwater in the Davidson Creek valley is the best available technology and preferred alternative.

The selected alternative from the tailings alternatives assessment, including the best available practices, formed the design basis that underwent a coordinated provincial and federal environmental assessment that was initiated in 2012 and ended successfully in 2019. The design of the TSF presented in this Report generally follows the tailings technology and overall concept presented in the tailings alternatives assessment. Differences are primarily related to the timing of mining and milling rates, as well as the corresponding adjustments to staging of the TSF and associated water management refinements.

18.7.2 TSF Hazard Classification

The Canadian Dam Association Dam Safety Guidelines (Canadian Dam Association, 2013; Canadian Dam Association, 2019) and the Part 10 Guidance Document for the Health, Safety and Reclamation Code for Mines in British Columbia (BC Ministry of Energy, Mines and Petroleum Resources, 2016) were used to determine the dam hazard classification and suggested minimum target levels for some design criteria, such as the inflow design flood and earthquake design ground motion for the TSF. A dam classification of ‘very high’ was selected for the TSF embankments.

Target design flood and earthquake criteria were selected for the TSF while considering the long-term design life of the facility, the minimum target levels in the guidelines listed above, and emerging international best practices for tailings management (Global Tailings Review, 2020; Montana Code Annotated, 2019). The following target levels were adopted for the design basis of the TSF:

- Inflow design flood: the probable maximum flood;
- Earthquake design ground motion: the 1/10,000-year return period seismic event or maximum credible earthquake, whichever is greater.

The selected inflow design flood and earthquake design ground motion exceed minimum recommendations from the Provincial regulations (BC Ministry of Energy, Mines and Low Carbon Innovation, 2022) and the associated guidelines (Canadian Dam Association, 2013; Canadian Dam Association, 2019; BC Ministry of Energy, Mines and Petroleum Resources, 2016) for ‘very high’ consequence structures during the construction, operations, and active-care closure phases of mine life, and are consistent with the requirements for a facility in closure.

18.7.3 Tailings Characteristics

Laboratory testing programs were conducted in 2013 (Knight Piésold, 2013b) and 2022 (Knight Piésold, 2022b) to determine the geotechnical and physical characteristics of the tailings. The tailings comprise non-plastic, sandy silt to silty sand with some clay. The particle size distribution of the tailings ranged from 44–52% sand, 35–46% silt, and 10–13% clay. Specific gravity of the tailings solids ranged from 2.75–2.79.

The testwork shows that an initial settled dry density of approximately 1.3 t/m³ is achievable for the tailings in both drained and undrained settling and a final average dry density of approximately 1.5–1.6 t/m³ should be achieved at the stress levels proposed in the TSF.

The tailings should fully consolidate within approximately 10 years following the end of tailings deposition. Surface desiccation of the exposed tailings surfaces (beaches) could be expected to further enhance consolidation and densification of the deposit at closure if required for reclamation purposes.

18.7.4 Design

Geotechnical and hydrogeological investigations in support of the TSF design are summarized in Section 10.7.

The TSF was designed to permanently store tailings, PAG waste rock, and potentially ML NAG waste rock generated during mine operation. The facility was designed to hold 470 Mm³ of tailings and waste rock material and up to 12 Mm³ of pond storage under normal operating conditions. Additional freeboard allowances are included in the design to manage seasonal inflows and provide protection from severe natural flooding.

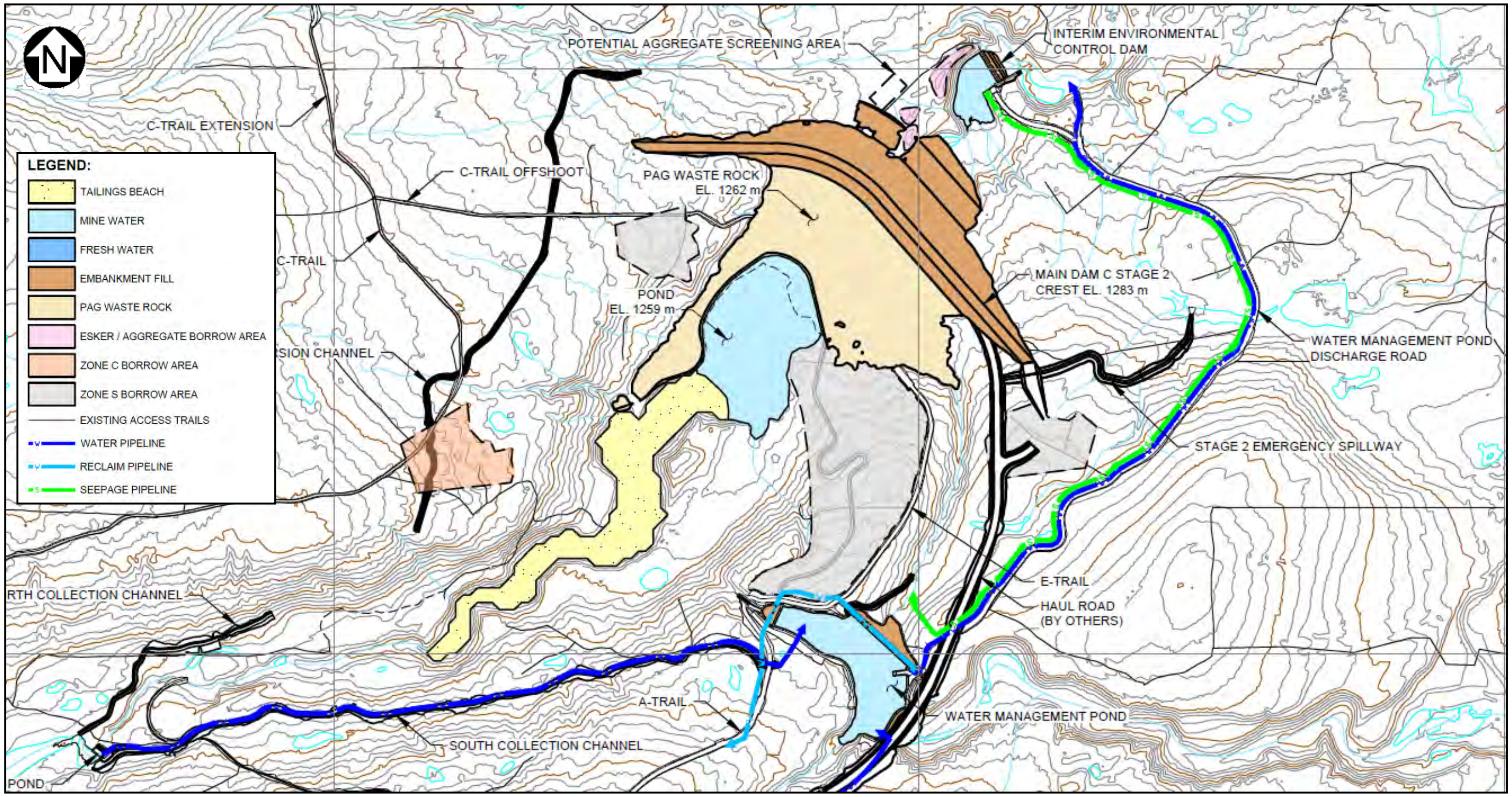
The TSF will comprise two adjacent valley-fill style impoundment facilities, referred to as TSF C and TSF D.

TSF C will be formed by the construction of three water retaining earth–rockfill embankments: Main Dam C, the Saddle Dam, and the West Dam in the upper reaches of the Davidson Creek drainage area. TSF C has been designed to receive PAG/ML NAG waste rock until Year 5, disposed of directly upstream of Main Dam C, and tailings for 15 years from Year 1 to the end of Year 15. The ultimate TSF C arrangement was sized to contain 231 Mm³ of tailings and 32 Mm³ of waste rock.

The construction of TSF C is in progress with construction activities for Stage 1 of Main Dam C underway.

Tailings will initially be discharged into TSF C from points on the west side of the facility and flow east towards the PAG/ML NAG waste rock disposal area. The supernatant pond will form at the interface of the tailings and waste rock disposal area. The Stage 1 and Stage 2 arrangements of TSF C will each include an emergency spillway to pass the inflow design flood in the highly unlikely event that the event occurs during initial construction or the first year of operations. Subsequent stages of the TSF have been designed to impound the inflow design flood. The general arrangement of the Stage 2 TSF at the end of Year 1, depicting the general footprint and extent of the TSF during the early operations period, is shown on Figure 18-2.

Figure 18-2: Year 1, TSF General Arrangement



Note: Figure prepared by Knight Piésold, 2024.

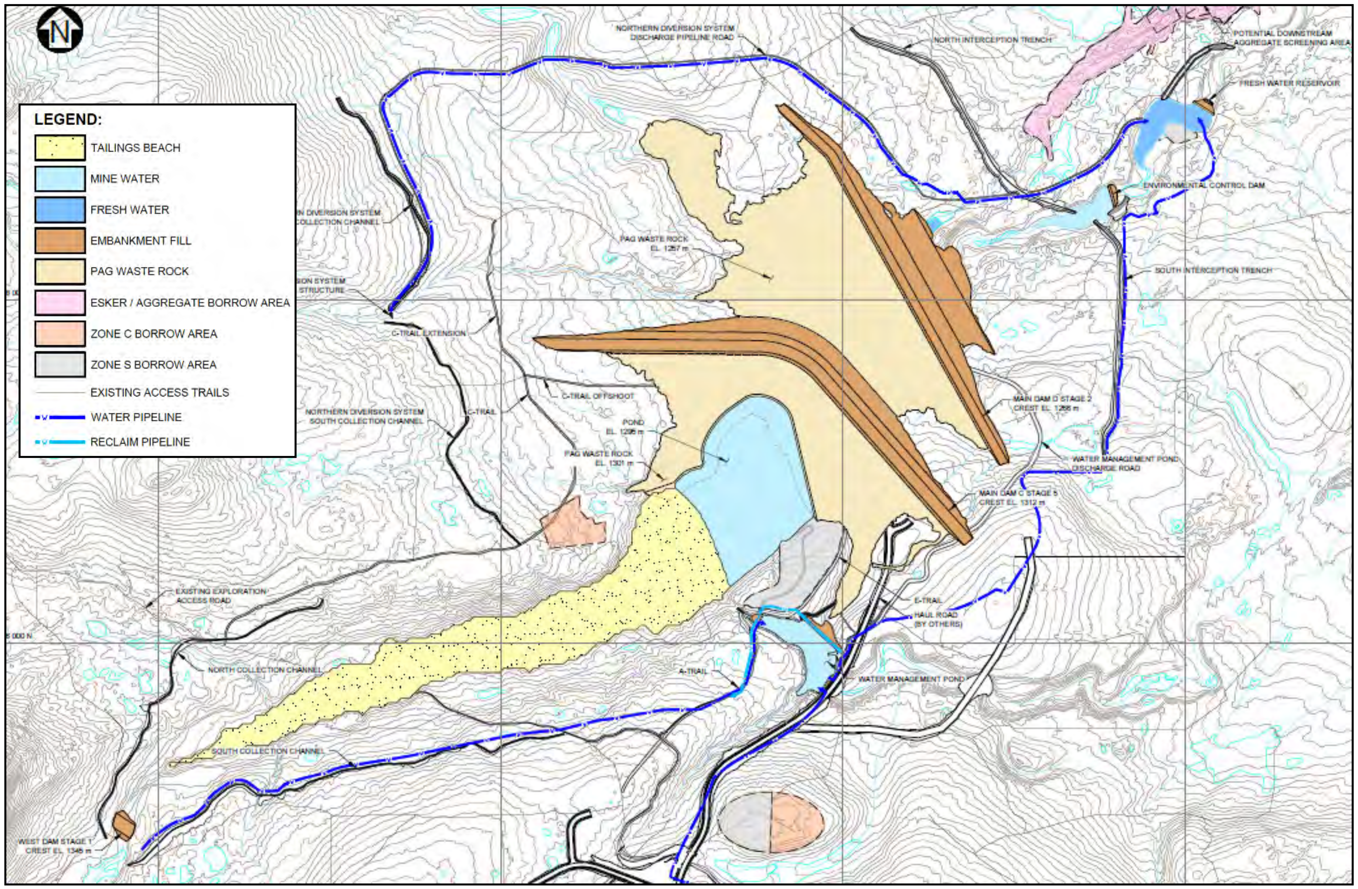
The West Dam will initially be constructed in Year 3 to establish the western limit of TSF C and ultimate upstream tailings discharge location. The West Dam will be raised to its ultimate crest elevation in Year 8 or as required to support on-going tailings deposition.

The timing of the Saddle Dam raises and its crest elevation will match those of Main Dam C starting in approximately Year 7. The Saddle Dam will join with Main Dam C above an elevation of 1,350 masl to form one continuous embankment.

Tailings will be discharged into TSF C from both the west and east sides of the facility starting in approximately Year 7, during the Phase 3 expansion. The discharge from the east will be from the face of Main Dam C and the Saddle Dam and will cover and submerge the waste rock disposal area. TSF D will be formed by construction of one water retaining earth–rockfill embankment: Main Dam D, adjacent to and downstream of TSF C within the Davidson Creek drainage area. TSF D construction will start in approximately Year 3, during the Phase 2 expansion, to provide additional storage capacity for PAG/ML NAG waste rock and tailings.

The general arrangement of the TSF at the end of Year 4 following construction of Main Dam D, representing the general footprint and extent of the TSF during Phase 2 of mine operations, is shown on Figure 18-3.

Figure 18-3: Year 4, TSF General Arrangement

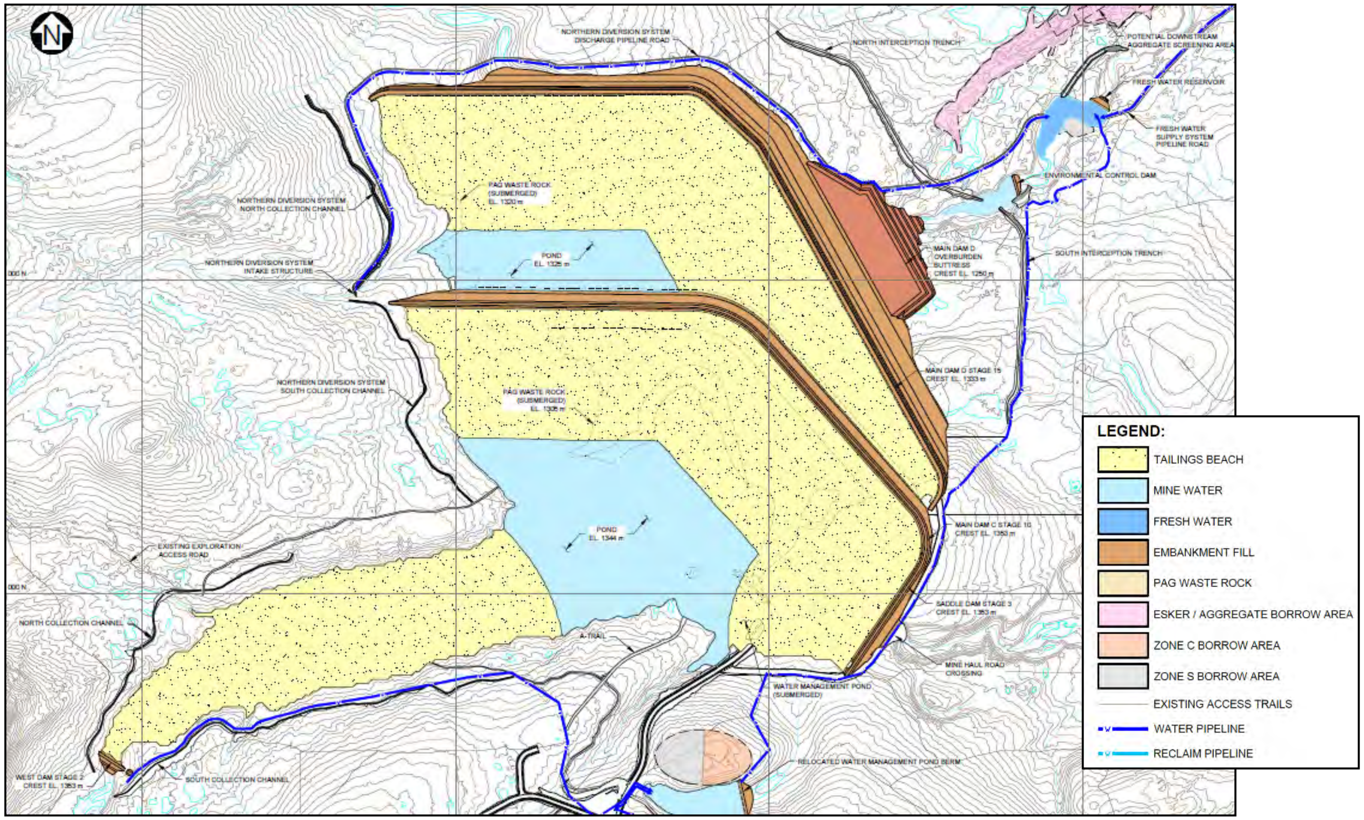


Note: Figure prepared by Knight Piésold, 2024.

TSF D is designed to receive waste rock for 12 years starting in Year 4 to the end of Year 15, then tailings for up to two years starting in Year 16 when TSF C reaches design capacity, the timing of which depends on the rate of mining and tailings consolidation processes. TSF D was sized to contain 181 Mm^3 of waste rock and 26 Mm^3 of tailings. Process water recovered following discharge of tailings to TSF D will be pumped to the supernatant pond in TSF C for reuse in ore processing. Buttresses constructed from shell zone and surplus overburden materials from the open pit development will be constructed with the raises of Main Dam D to provide additional stability and limit material directed to stockpile to avoid unnecessary re-handling. The surplus overburden buttress will also serve as a material source for reclamation activities in closure.

The general arrangement of the TSF at the end of mine life is shown on Figure 18-4.

Figure 18-4: End of Mine, TSF General Arrangement



Note: Figure prepared by Knight Piésold, 2024.

The TSF C and TSF D embankment raising schedules are summarized in Table 18-2.

Table 18-2: TSF C and TSF D Embankment Raising Schedules

Mine Year	TSF C					TSF D		
	Stage (1, 2)	Crest Elevation (masl)			Total Capacity (Mm^3)	Stage (1)	Main Dam D Crest Elev. (masl)	Total Capacity (Mm^3)
		Main Dam C	Saddle Dam (3)	West Dam				
-1	1	1,273	—	—	22	—	—	—
1	2	1,283	—	—	35	—	—	—
2	3	1,296	—	1,345	57	—	—	—
3	4	1,306	—	—	80	1	1,256	17
4	5	1,312	—	—	98	2	1,266	32
5	6	1,321	—	—	131	3	1,275	49
6	—	—	—	—	—	4	1,283	66
7	7	1,331	1,331	—	177	5	1,291	86
8	—	—	—	1,353	—	6	1,297	101
9	8	1,340	1,340	-	228	7	1,302	114
10	—	—	—	—	—	8	1,307	128
11	—	—	—	—	—	9	1,312	143
12	9	1,349	1,349	-	287	10	1,318	162
13	—	—	—	—	—	11	1,323	178
14	—	—	—	—	—	12	1,325	185
15	10	1,353	1,353	-	314	13	1,327	192
16	—	—	—	—	—	14	1,331	206
17	—	—	—	—	—	15	1,333	213

Notes:

1. Stage 1 to 6 of Main Dam C and all stages of the West Dam and Main Dam D will be constructed with centreline raises.
2. Stage 7 to 10 of Main Dam C, and all stages of the Saddle Dam, will be constructed with downstream raises. The downstream raises of Main Dam C will toe onto the TSF D PAG/ML NAG waste rock dump surface.
3. The Saddle Dam will join with Main Dam C above an elevation of 1,350 masl to form one continuous embankment.

Seepage through the TSF embankments will primarily be controlled by the low permeability upstream seal zones and the cut off trenches tied into the low permeability subgrade of Main Dam C and Main Dam D and facing high-density polyethylene geomembrane on the TSF C embankments above 1,312 masl. The embankment drainage collection systems will provide additional control and will collect and direct seepage to the seepage control dams (the Interim

Environmental Control Dam, initially, and the Environmental Control Dam, during and after construction of Main Dam D) for collection and pump back to the TSF supernatant pond.

The major embankment construction material types and their placement specifications are summarized in Table 18-3.

Table 18-3 TSF Embankments Construction Materials

Material	Description	Locations		Placing and Compaction Requirements [3]
		Source	Placement [1]	
Zone S	Well-graded, low-permeability, silty sand with some gravel with a fines content of 20 to 60% (passing the #200 sieve)	Primarily open pit/borrows as required	Seal Zone [2]	Placed, moisture conditioned, and spread in max. 300 mm lifts and compacted by combination of smooth drum vibratory rollers and pad foot compactors to a minimum of 95% of standard proctor maximum dry density.
Zone C	Random fill comprising pervious overburden and specific non-ML NAG waste rock	Primarily open pit/borrows as required	Shell Zone	Dumped and spread in max. 1,000 mm lifts and uniformly compacted with smooth drum vibratory rollers and/or selective truck routing across the main fill to produce a uniformly compacted lift. The edges of the fill will be compacted using smooth drum vibratory rollers to produce a uniformly compacted lift.
Zone F	Clean, non-reactive, fine to coarse sand and gravel	Processed from borrows	Filter Zone	Placed and spread in max. 600 mm lifts loose and compacted using smooth drum rollers with a minimum of 4 passes.
Zone T	Non-reactive fluvial or colluvial material or selected waste rock	Processed from borrows	Transition Zone	

Note.

1. Material is placed at Main Dam C, Main Dam D, the Saddle Dam, and the West Dam, unless otherwise noted.
2. Material placed at Main Dam C and Main Dam D only.
3. Compaction method specifications may be adjusted during construction to other methods that are sufficient to meet the design intent of the embankment construction materials.

18.7.5 Monitoring

Geotechnical monitoring instruments will be installed along sections of the TSF and seepage control dam embankments to monitor the response of the embankment fill and foundation as the facilities are constructed and operated. The instruments will typically be connected to an automated data acquisition system that will provide real-time access to the data. Flow monitoring devices, water level meters, and groundwater wells may be measured manually in non-critical areas. The collected data will be analyzed and evaluated against quantifiable performance objectives and remedial actions will be taken if thresholds defined in the trigger action response plan are exceeded.

18.8 Water Management

18.8.1 General

Geotechnical and hydrogeological investigations in support of the water management designs are summarized in Section 10.7.

The design of the water management structures has taken into consideration the following requirements:

- Temporary and secure storage of fresh water within the mine site area in engineered water storage facilities;
- Limit accumulation of surplus water within the TSF to the maximum practicable extent;
- Control, collection, and diversion of non-contact surface water flows not needed for mine operations;
- Control and collect contact surface water prior to use/release;
- Controlled release of surface water flows to Davidson Creek downstream of the mine to reduce the potential environmental impacts of the project to the extent reasonably practicable;
- The inclusion of monitoring features to confirm performance goals are achieved and design criteria are met;
- Staged development of the facilities over the life of the Project as the disturbed mine site areas change.

Water management plans were developed by identifying the size and position of the planned mine site facilities and establishing estimated catchment area boundaries based on the mine site development concept. Drainage from the majority of the mine area will flow by gravity into the TSF, following natural topographical drainages, which simplifies water management, spill control, and mine closure. Mine-affected runoff will be captured and recycled for use as process water. Surplus water not required to support mine operations will be sampled and analyzed, compared to applicable water quality criteria, and if compliant, will be used to augment flow in lower Davidson Creek.

The water management systems required during construction and operations are summarized in Table 18-4 and are shown for different stages of the mine development in Figure 18-2, Figure 18-3, and Figure 18-4.

Table 18-4: Water Management Systems

Item	Note
Central diversion system	Diverts runoff around the TSF or directs runoff to a water transfer pond and subsequently the Water Management Pond and later to the Relocated Water Management Pond. Major facilities will include collection and diversion channels, a water transfer pond, and a pipeline and pump station. The system will have an initial location during Phase 1 and be relocated during Phase 2 in Years 2 and 3.
Northern diversion system	Will be constructed in Year 4, up-gradient of TSF D to allow for diversion of upstream flows around the TSF or for release of flows to the TSF. Infrastructure will include collection channels, a water transfer pond and intake structure, and a gravity flow pipeline.
Fresh water reservoir	Collects runoff from contributing areas and water delivered via gravity pipelines from collection points. The reservoir will provide flows to lower Davidson Creek to meet instream flow needs downstream of the mine and will provide water for mine operations when required.
Fresh water supply system	May be established if additional water is needed to maintain flows in Davidson Creek. Will convey water from Tatelkuz Lake to the Fresh Water Reservoir, as needed for supplementing the available water for release into lower Davidson Creek or providing make up water to the mine when required. Infrastructure will include an intake and pump station at the lake, a booster pump station, and a water pipeline.
Water management pond	Manages runoff from contributing areas and water pumped from collection points. Will provide fresh make-up water to the process plant to support ore processing. Water not needed to support mine operations will be discharged to the Fresh Water Reservoir, and from there will be used to augment flow in lower Davidson Creek. Will be relocated as part of Phase 3, prior to the initial location being affected by the rising TSF C tailings filling level.
Reclaim water system	Routes supernatant water from the TSF to the plant site to support ore processing.
Stockpile water management structures	Diverts runoff from undisturbed areas and contains seepage and surface water runoff from the footprints of the low-grade stockpile and the waste stockpiles. Infrastructure will include collection and diversion channels, collection ponds, and pump and pipeline systems.
Lake 16 diversion berm and Lake 15–16 connector channel	Divert flows from Lake 16 in the headwaters of Davidson Creek towards Lake 15 in the headwaters of Creek 705 to reduce the amount of flow through the mine site.
Water treatment plants	Includes a metals water treatment plant and lime neutralization circuit near the process plant.

18.8.2 Monitoring

Geotechnical monitoring instruments will be installed along sections of the water management structure embankments to monitor the response of the embankment fill and foundation as the facilities are constructed and operated.

Water discharged from the fresh water reservoir will be monitored using ultrasonic flow meters installed at each of the design outlets. These instruments will typically be connected to an automated data acquisition system that will provide real-time access to the data. Flow monitoring devices, water level meters, and groundwater wells may be measured manually in non-critical areas. The collected data will be analyzed and evaluated against quantifiable performance objectives and remedial actions will be taken if thresholds defined in the trigger action response plan are exceeded.

18.8.3 Water Balance

A LOM water balance model was prepared in 2021 (Knight Piésold, 2021d). The model represented proposed Project development and sequencing in the 2021 Feasibility Study and simulated water management flows, surface water, and groundwater flows during all phases of mine development. Conclusions from that water balance model iteration are inferred to be relatively unchanged for the 2024 Expansion Study throughput rates. Mine operations during Phase 1 should be manageable with a low likelihood of needing external make-up water sources. The ramp up of ore throughput to 15 Mt/a at Phase 2 and 25 Mt/a at Phase 3 will progressively increase the frequency and quantity of make-up water required to support ore processing while maintaining downstream flows within prescribed limits. It is recommended that a water balance model be updated considering the ramp up to 15 Mt/a to further evaluate site water availability and how sources of available water will be managed to meet the ore processing and environmental flow demands, if/when the decision is made to proceed with this expansion plan.

18.9 Water Treatment

Collected contact water from the pit and WRSF will be treated through a metals water treatment plant, as required depending on where the water will be directed. The initial installed capacity of 55 L/sec will be increased to 110 L/sec in Year 2 followed by an additional increase to 155 L/sec in Year 6.

18.10 Workforce

Mine operations will be staffed with a combination of locally- and regionally-based personnel. All personnel will be transported to site using buses from Williams Lake, Quesnel, Prince George and Vanderhoof. Regionally-based personnel will be flown to Prince George, from where they will be bussed to site.

18.11 Camps and Accommodation

The camp accommodations will be located approximately 2.8 km to the northeast of the main mine processing area (approximately 5 km via the access road).

At start-up, the camp will accommodate 312 personnel. Prior to Phase 3 construction, the operations camp pad will be extended, and the construction camp relocated to the same area to allow for pit expansion to the north. Additional accommodations required to meet peak occupancy for Phase 2 and Phase 3 construction will be provided through short-term dormitory rental. This will include two 45-man dormitories for Phase 2 and five 45-man dormitories for Phase 3 to meet peak occupancy requirements.

Electricity will be supplied via the 24 kV powerline. A 0.8 MW backup generator complete with a diesel fuel tank will also be located on the camp pad.

Propane will be used to power furnaces and the kitchen for cooking. A 40,000 L propane tank to be located on the camp pad will provide propane storage.

18.12 Power and Electrical

The Blackwater Gold Mine will require up to 113 MW (70 MW for Phases 1/2 and 43 MW for Phase 3) of power once the full mill throughput is realized in Year 7. A 135 km, 230 kV overland transmission line is currently under construction to connect to the BC Hydro grid at the Glenannan substation located near the Endako mining operations, 65 km west of Vanderhoof, BC.

The incoming transmission line will terminate at the site main substation adjacent to the main process facilities. The substation will have incoming circuit breakers, motorized isolating disconnect switches, power transformers, switchgear, and protective equipment for the stepdown of power from the transmission voltage level of 230 kV to the site distribution/utilization level of 25 kV.

The primary power supply around the mine site will be a single 25 kV feed pole line running from the primary distribution centre at the main substation. This distribution system will be expanded to accommodate electrification of the open pit in Year 1. Portable substations will transform the power to 4.16 kV for the mine shovels and drills.

A 15 km long 25 kV feed line and portable substations (25 kV/4.16 kV stepdown) will be required for the fresh water supply system in Year 8. All other mine power will be supplied using pole-mounted transformers to step the voltage down from 25 kV to 600 V.

A total of 12,500 kVA of stand-by power (7,500 kVA in Phases 1/2 and 5,000 kVA in Phase 3) will be available from standby generators sized to provide power to the process and ancillary electrical equipment in the event of a utility power failure. The temporary power generation equipment will be installed as the source of backup power supply for the permanent camp.

18.13 Communications

Fiber optic cable will be brought to the mine site along the incoming 230 kV powerline from the BC Hydro Glenannan substation for telecommunications. This fiber aerial cable will land at the mine substation. During the construction phase prior to the fiber optic line reaching site and being installed, a satellite system will provide internet service to the Blackwater Gold Mine.

19.0 MARKET STUDIES AND CONTRACTS

19.1 Market Studies

No formal marketing studies were completed.

There are many markets in the world where gold and silver are bought and sold, and it is not difficult to obtain a market price at any time. The gold and silver market is highly liquid with many well-informed potential buyers and sellers active at any given time.

The economic analysis in Section 22 uses the payability forecasts summarized in Table 19-1.

Table 19-1: Payability Assumptions

Item	Units	Value
Gold payable	%	99.9
Silver payable	%	95.0
Selling costs	C$/AuEq oz	3.0

19.2 Commodity Price Projections

Mineral Resources were estimated using a gold price of US$2,000/oz Au and a silver price of $16/oz Ag.

Mineral Reserves were estimated using a gold price of US$1,400/oz Au and a silver price of $15/oz Ag.

Long-term commodity pricing for the economic analysis in this Report uses metal prices and exchange rates consistent with current consensus estimates provided by a consortium of banks in Q1 2024. The long-term guidance gold price is US$1,800/oz Au. The cashflow analysis uses a reverting price curve for gold. The average January 2024 spot price is approximately US$2,000/oz Au and this reverts to long-term pricing from Year 5 onwards (Table 19-2).

Table 19-2: Commodity Price and Exchange Rate Forecasts Used in Economic Analysis

	Unit	Year 1	Year 2	Year 3	Year 4	Year 5 to Year 17
Gold price	US$/oz	2,000	1,950	1,900	1,850	1,800
Silver price	US$/oz	23	23	23	23	23
C$/US$ foreign exchange rate	C$:US$	0.74	0.74	0.74	0.74	0.74

The long-term guidance silver price used in the economic analysis is US$23/oz. The Canadian:US dollar exchange rate forecast for the estimate is 0.74.

Artemis Gold has a modest hedging program in place to secure the returns on capital invested in the early years of operations and de-risk the servicing loan facility during the pay-back period. To date, Artemis Gold has entered into forward sales agreements to deliver 190,000 ounces of gold between March 2025 and December 2027 at a weighted average sales price of C$2,851 per ounce. During Q4 2023, Artemis Gold also executed zero cost collars associated with 30,000 ounces of gold with settlement dates from December 2024 to February 2025. The collars have a weighted average put price of C$2,600 per ounce and a weighted average call price of C$3,353 per ounce.

19.3 Streaming Agreement

Artemis Gold has an existing gold stream agreement with Wheaton, whereby Wheaton will purchase 8.0% of the refined gold produced from the Blackwater Gold Mine (see discussion in Section 4.7).

Once 464,000 ounces of refined gold have been delivered to Wheaton, the gold stream will reduce to 4.0%. Wheaton will make payments for the gold purchased equal to 35% of the US dollar gold price quoted by the London Bullion Market Association two days prior to delivery.

19.4 Contracts

Artemis Gold 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 doré elsewhere in the world.

Other than for product sales, the largest contracts will cover items such as mine equipment, drill-and-blast operations, bulk commodities, and technical services. Artemis Gold is currently negotiating contract terms. Contracts will be re-negotiated and renewed as needed during the LOM.

19.5 QP Comments on Item 19 "Market Studies and Contracts"

The QP notes the following.

- The doré to be produced by the mine is readily marketable;
- Metal prices are set corporately for Mineral Resource and Mineral Reserve estimation;
- The metal prices used in the cashflow analysis are based on consensus estimates provided by a consortium of banks. The cashflow analysis uses a reverting price curve for gold.

The QP has reviewed commodity pricing assumptions, marketing assumptions, and the current major contract areas, and considers the information acceptable for use in estimating Mineral Resources and Mineral Reserves, and in the economic analysis that supports the Mineral Reserves.

20.0 ENVIRONMENTAL STUDIES, PERMITTING, AND SOCIAL OR COMMUNITY IMPACT

20.1 Introduction

The Blackwater Gold Mine is supported by a suite of environmental, social, economic, and cultural heritage baseline studies and potential effects were fully assessed. The Blackwater Gold Mine was granted an Environmental Assessment Certificate (EAC) #M19-01 on 21 June, 2019 (BC Environmental Assessment Office, 2019b) under the 2002 Environmental Assessment Act and an Environmental Assessment Decision Statement on 15 April, 2019 under the Canadian Environmental Assessment Act, 2012 (CEA Agency, 2019). The Blackwater Gold Mine is supported by the Lhoosk'uz Dené Nation and the Ulkatcho First Nation, who submitted letters of support following completion of the Environmental Assessment process.

To manage potential effects, an Environmental Management System is supported by a comprehensive set of management plans.

20.2 Baseline and Supporting Studies

Comprehensive biophysical studies were completed to support the Application/EIS and permit applications, and Artemis Gold continues to collect data in conformance with conditions in provincial and federal authorizations. Studies completed are summarized in Table 20-1.

Table 20-1: Baseline and Supporting Studies

Biophysical Studies	Years
Air quality	2012, 2013, 2022–current
Archaeology	2011–2013, 2022-current
Fish and aquatic resources	2011–2013, 2017, 2021–current
Geochemistry	2013, 2020
Groundwater quality	2011–current
Groundwater quantity	2012–current
Meteorology	2011–current
Noise	2011–2013
Sediment quality	2011–2013, 2017, 2021–current
Soils, terrain, and surficial geology	2013, 2017, 2021–current
Surface water quality	2011–current
Surface water quantity	2011–current
Vegetation (incl invasive plants)	2011–2013, 2017, 2021–current

Biophysical Studies	Years
Wetlands	2011–2013, 2017, 2021, 2022
Wildlife	2011–2013, 2016, 2018, 2022–current

20.3 Environmental Considerations

20.3.1 Environmental Monitoring Plans

Artemis Gold has an Environmental Management System to organize and guide activities to ensure environmentally and socially responsible development of the mine. The individual management plans and supporting documentation that form the basis of the Environmental Management System are required by the *Mines Act*, *Environmental Management Act*, or the Environmental Assessment Certificate #M19-01 Conditions and were developed by qualified professionals and subject matter experts. The management plans were developed and revised to incorporate comments/reviews from Indigenous groups provided through review of monitoring results and/or previous iterations of the plan.

20.3.2 Water Management

Water management objectives were defined within the context of meeting instream flow needs in Davidson Creek and water quality guidelines or science-based environmental benchmarks in Davidson Creek, Creek 661 and Chedakuz Creek.

A Mine Site Water and Discharge Monitoring and Management Plan was developed. This will provide an early detection system in regard to both process and receiving surface and groundwater quality trends so that impacts can be reported, investigated, and mitigated, and future recurrences can be avoided. In addition, ongoing monitoring will be used to evaluate predictions, calibrate models, and update models and mitigations throughout the life of mine. The plan incorporates monitoring plans for all discharges of mine contact water to the receiving environment (surface water or groundwater). Monitoring of discharge will include measurement of discharge volumes, flow rates, and water quality at a frequency that meets federal (Metal and Diamond Mining Effluent Regulations) and provincial permit limits.

Application of the water management system is anticipated to mitigate water flow and quality issues. With water management and mitigation (e.g., water treatment and erosion and sediment control measures) in place, water quality is not predicted to exceed BC or Canadian Council of Ministers of the Environment water quality guidelines for the protection of aquatic life in the receiving environment or the science-based environmental benchmarks approved for dissolved aluminum in Davidson Creek.

Adaptive management is intended to address the circumstances that will require implementation of alternate or additional mitigation measures to address effects of the Blackwater Gold Mine if the monitoring shows that those effects are approaching the triggers (based on the instream flow needs, water quality guidelines and science-based environmental benchmarks, identified in the Aquatic Effects Monitoring Program Plan and Mine Site Water and Discharge Monitoring and Management Plan.

20.3.3 Waste Management

To mitigate the effects of water quality and quantity on downstream aquatic receiving environments, the Blackwater Gold Mine has comprehensive water management structures designed to capture and convey mine contact water (water that is in contact with disturbed lands and build facilities), to water treatment facilities or to the TSF which will convey effluent for treatment prior to discharge to the environment. A site-wide Mine Site Water and Discharge Monitoring and Management Plan is in effect in conformance to the Code and *Environmental Management Act* Effluent Discharge permits #110650 and #110652.

An air quality monitoring program will be undertaken during construction and operations. Key air quality mitigation measures include the maintenance of emissions equipment to design standards, implementation of fugitive dust management measures, and application of trigger-action-response plans to manage periodic near exceedances of permitted levels. Dust generation and discharge from the TSF, overburden and waste rock stockpiles will be monitored.

Monitoring of wildlife for the mine site including the waste stockpiles, and management of wildlife use of the TSF ponds will be guided by the Wildlife Mitigation and Monitoring Plan.

20.3.4 Offset Plans

Offsetting plans were prepared to mitigate potential impacts to fish and fish habitat, wetlands, and southern mountain caribou.

The Blackwater Gold Mine received an Authorization under Paragraph 35(2)(b) of the *Fisheries Act* dated 30 June, 2023 and approval of the amendments to Schedule 2 of the Metal and Diamond Mining Effluent Regulations (under Section 36 of the *Fisheries Act*) dated 9 June, 2023. The associated offset plans require restoration and creation of fish habitat in, and adjacent to, local and regional watercourses.

Artemis Gold has developed a Wetland Monitoring and Offsetting Plan in accordance with an Environmental Assessment Certificate M#19-01 Condition 24 to offset the impacts on wetland function. The Mathews Creek wetland complex has been selected as the primary offsetting site. The wetland complex is located off the Blackwater Gold Mine area on lands partially owned by Artemis Gold (fee-simple) with the balance being Crown land. Additional wetland offsetting is planned to be achieved at the Dykam Ranch wetland complex and Capoose Road Rehabilitation project associated with caribou offsetting work.

The local population of woodland caribou, Southern Mountain population (*Rangifer tarandus caribou*) is listed as threatened under Schedule 1 of the *Species at Risk Act,* as special concern by Committee on the Status of Endangered Wildlife in Canada, and blue-listed by the province (BC Conservation Data Centre, 2020). The Mine overlaps the eastern boundary of the Tweedsmuir local population unit (BC Ministry of Forests, Lands, Natural Resource Operations and Rural Development, 2020). Artemis Gold has developed a Caribou Mitigation and Monitoring Plan to avoid, reduce and offset adverse effects on caribou and its critical habitat as defined in the Recovery Strategy for the Woodland Caribou, Southern Mountain population in Canada (Environment Canada, 2014). A Caribou Offsetting Plan has been developed in accordance with EAC and DS conditions, and consists of a habitat securement in the Capoose mineral tenure area and funding for caribou habitat restoration initiatives.

20.4 Closure Considerations

Closure and reclamation of the Blackwater Gold Mine will conform to the requirements of the Health, Safety, and Reclamation Code for Mines in BC (BC Ministry of Energy, Mines and Low Carbon Innovation, 2022) and the Reclamation and Closure Plan approved through *Mines Act* Permit M-246. End land use and land capability objectives were integrated into planning and design. The Reclamation and Closure Plan will be reviewed with each five-year mine plan update and will inform the mine reclamation liability cost estimate and reclamation security bond amount.

Closure costs are discussed in Section 22.

20.5 Permitting

The Blackwater Gold Mine holds a provincial Environmental Assessment Certificate #M19-01 (EAC) under the 2002 *Environmental Assessment Act* (BC Environmental Assessment Office, 2019b) and a federal Decision Statement under the *Canadian Environmental Assessment Act,* 2012 (CEA Agency, 2019). In 2020 the EAC Certificate was transferred from New Gold to Artemis Ltd., a wholly-owned subsidiary of Artemis Gold, in accordance with provincial and federal requirements.

Participants in the Application/EIS process included representatives from federal government agencies (Canadian Environmental Assessment Agency, Fisheries and Oceans Canada, Environment and Climate Change Canada and Health Canada), BC government agencies (Ministry of Energy, Mines and Low Carbon Innovation, Ministry of Environment and Climate Change Strategy, Ministry of Forests, Lands, Natural Resource Operations and Rural Development and Northern Health), Indigenous Groups (Lhoosk'uz Dené Nation, Ulkatcho First Nation, Nadleh Whut'en First Nation, Stellat'en First Nation, Saik'uz First Nation and Nazko First Nation) and local governments. The public also had opportunities to provide review and input on the Application/EIS.

The Blackwater Gold Mine is supported by the Lhoosk'uz Dené Nation and the Ulkatcho First Nation, who submitted letters of support following completion of the Environmental Assessment process.

The Environmental Assessment Certificate contains 43 binding conditions, which identify requirements for environmental and social management plans, consultation requirements related to management plans, and requirements for an Environmental Monitoring Committee, Community Liaison Committee, Independent Environmental and Aboriginal Monitor(s).

The federal Decision Statement includes 102 binding conditions, which identify requirement for plans to offset impacts, consultation requirements for offset plans and follow-up programs, and specific mitigation measures. Artemis Gold is addressing these conditions in accordance with the timelines specified by the Environmental Assessment Certificate and Decision Statement.

Artemis Gold holds the necessary permits to construct and operate Phase 1 of the Blackwater Gold Mine, including those required for off-site infrastructure.

Key federal and provincial permits are listed in Table 20-2 and Table 20-3, respectively.

Table 20-2: Key Federal Authorizations

Authorizing Legislation	Authorization	Issue Date	Purpose
Fisheries Act	Fisheries Act Section 35(2) Authorization (21-HPAC-01447)	30 June, 2023	Avoid, mitigate, and offset impacts to fish habitat as a result of activities that are likely to result in the harmful alteration, disruption, or destruction of fish habitat.
Metal and Diamond Mine Effluent Regulations (MDMER)	Schedule 2 Amendment	8 June, 2023	List the upper reaches of Davidson Creek (development of the tailings storage facility) and Creek 661 (development of the upper overburden and ore stockpiles) in Schedule 2 of the MDMER.

Table 20-3: Key Provincial Authorizations

Authorizing Legislation	Authorization	Issue Date	Purpose
Mines Act	Permit M-246 Early Works	22 June, 2021	Approves Blackwater early works program within Permitted Mine Area, encompassing 1,018.9 ha.
Mines Act	Permit M-246 Major Works	8 March, 2023	Approves Blackwater Major Works within Permitted Mine Area, encompassing 4,296.2 ha.
Mines Act	Permit G-11-164	21 August, 2021	Authorizes excavation and removal of sand and gravel a gravel pit near 135 km Kluskus Forestry Service Road.
Environmental Management Act	Permit 110602	24 June, 2021	Authorizes discharge of treated stormwater effluent to ground from early-stage construction activities.

Authorizing Legislation	Authorization	Issue Date	Purpose
Environmental Management Act	Permit 110650	3 May, 2023	Authorizes discharge to air.
Environmental Management Act	Permit 110652	3 May, 2023	Authorizes discharge of effluent.
Forest Act	Licence of Occupation Transmission Line 7410296	April 25, 2023	Transmission line.
Forest Act	Licence of Occupation Transmission Line 7409823	April 25, 2023	Transmission line.
Water Sustainability Act	Water Licence - Freshwater supply system (100350124)	June 30, 2023	Authorizes water supply from Tatelkuz Lake to the mine site.

Mines Act Permit M-246 was issued in March 2023 which authorizes construction, operation, and closure of the Mine. Permit M-246 is required to be reviewed at minimum every five years. Major amendments to the approved five-year mine plan may necessitate amendments as required.

Permit M246 conditions include numerous health, safety, geotechnical and environmental reporting requirements. *Environmental Management Act* (EMA) Permits #110650 and #110652, which authorize discharges to the environment from approved works, and specify environmental monitoring and reporting requirements, were issued in May 2023. Artemis Gold.

The Blackwater Gold Mine is currently authorized for a milling capacity of 55,000 t/day (or 20 Mt/a). Prior to implementing the Phase 3 mill capacity increase, amendments to several permits may be required, including the *Mines Act* Permit M-246 and *Environmental Management Act* Permits #110650 and #110652.

20.6 First Nations

The Blackwater Gold Mine is located within the traditional territories of Ulkatcho First Nation, Lhoosk'uz Dené Nation. The power transmission line and other infrastructure pass through the traditional territories of Nadleh Whut'en First Nation, Saik'uz First Nation, and Stellat'en First Nation (collectively, the Nechako First Nations).

It is Artemis Gold's policy to have agreements in place with the primary First Nations identified in the Community Effects Monitoring and Management Plan for the Blackwater Gold Mine.

Since acquiring the Project in August 2020, Artemis Gold has engaged with Indigenous Groups to provide updates on the Blackwater Gold Mine's permitting requirements, review of draft regulatory applications, and construction progress. Artemis Gold's engagement with Indigenous Groups is facilitated in part through:

- Focused meetings on specific technical areas;
- Blackwater Gold Mine update teleconferences and meetings with groups individually or collectively;
- Meetings with the Environmental Monitoring Committee established by Environmental Assessment Certificate Condition 19; and
- Meetings with the Community Liaison Committee established by Environmental Assessment Certificate Condition 37.

20.7 Considerations of Social and Community Impacts

The region has diverse non-traditional land uses driven by commercial and non-commercial recreation, forestry, agriculture and mining and mineral exploration. The mine site has a relatively simple non-traditional land use setting with only a few overlapping uses including three guide-outfitter certificates, three provincially registered traplines, and one range tenure.

21.0 CAPITAL AND OPERATING COSTS

21.1 Introduction

Capital and operating cost estimates are at a minimum at a pre-feasibility level of accuracy.

For cost estimation purposes, Phase 1 capital costs are assumed to be sunk costs.

21.2 Capital Cost Estimates

21.2.1 Basis of Estimate

The estimate was based on inputs developed by Artemis Gold, Knight Piésold, Moose Mountain, Lycopodium, JAT MetConsult, and other third parties, using budget quotes and historical data, and is an amalgamation of engineering, material take offs and in-house benchmarks.

Data supporting the estimate were obtained during Q4 2023 and Q1 2024.

The capital cost estimate excluded government taxes and duties. These costs were included where applicable in the financial model.

The estimate assumes that the process plant will be executed using an engineering and design, procurement, and construction management (EPCM) approach.

The construction of the mine, TSF, site-wide bulk earthworks, roads, borefields, power station, camps, and non-process infrastructure buildings (outside of the plant area) will be either managed by the Owner's team or by specialist consultants and/or contractors engaged directly by the Owner. This execution approach was used as the basis for the preliminary implementation schedule and capital cost estimate.

There is no allowance for escalation beyond the base date of the estimate.

21.2.2 Mine Capital Costs

Mine capital costs are derived from purchase contracts entered into by Artemis Gold, vendor quotations, and operational data collected by other Canadian open pit mining operations.

The initial mine equipment fleet is purchased via various lease arrangements, with agreed-upon commercial terms from the equipment suppliers. Down payments and lease payments are reported as sustaining capital, as the fleet is paid off and ownership transfers to Artemis Gold.

The costs of expanding the mine equipment fleet over the first six years of operations are treated as growth capital. Required mining equipment replacement purchases, for units retired after their useful life, are treated as sustaining capital.

Development costs throughout mine operations were capitalized, including:

- Clearing and grubbing;

- Topsoil removal and stockpiling;
- Haul road construction;
- Crush rock production for construction and operations;
- Piping for pit dewatering.

The following items were also capitalized:

- Pit electrification;
- Site GPS and wireless data networks
- Fleet machine onboard GPS and wireless data receiving.
- Fleet management, dispatch, and health systems
- Mine survey gear and supplies;
- Geology, grade control and mine planning software licenses;
- Maintenance tooling and supplies;
- Mine rescue gear;
- Mine communications systems (handheld and machine onboard);
- Pit and stockpile geotechnical instrumentation;
- Drill and blast contractor demobilization.

The mining cost estimate is included as Table 21-1.

Table 21-1: Mining Capital Cost Estimate

Mining	Area	C$ M
Growth capital costs	Site development	79
	Mine equipment	202
	Mine infrastructure	—
	Indirects	4
	Subtotal	*285*
Sustaining capital costs	Site development	31
	Mine equipment	659
	Mine infrastructure	11
	Indirects	23
	Subtotal	*724*
Total		***1,009***

Note: Growth capital includes deferred initial capital costs. Numbers have been rounded.

Contingency was included in the mining capital cost estimate where required.

Portions of the estimated mine operating costs, particularly mine operating costs related to tailings dam expansion as well as maintenance of mine fleet major components, were allocated to capital costs, and treated as growth and sustaining capital respectively.

21.2.3 Process Plant Capital Costs

The capital cost estimate for the process plant is based on the inputs summarized in Table 21-2.

Table 21-2: Process Plant Capital Cost Inputs

Item	Note
Quantities	The estimate is an amalgamation of engineering, material take offs and in-house benchmarks. The level of accuracy and detail in the estimate varied based on the engineering progress of the given scope and discipline.
Design growth allocation	Design growth allowances were made to cover undefined items of work within the defined scope. The design allowance was intended to make allowances for the final quantities between the current known definition and the final design.
Supply costs	Pricing cost of major and long lead mechanical and electrical equipment was inclusive of all costs necessary to purchase the goods ex-works, excluding delivery to site. Bulk equipment supply costs and material costs were mostly derived from supplier quotes. Budgetary quotations were requested from select vendors for the supply of major and long lead items. The mechanical packages were sole-sourced to keep commonality of spares with existing Phase 1 plant.

Item	Note
Contractor	Installation rates assumed contractors based at site. Budgetary quotations were requested from various contractors for packages with developed bills of quantity.
Labour hours	Installation hours for all equipment and commodities and systems were assembled by the estimating team, supported by construction data, and contractor budgetary bids where available.
Construction indirect costs	Indirect costs include: • Deliverables (deliverables related to the scope of work and contractor overhead costs); • Mobilisation and demobilisation (personnel and equipment); • Recurring costs (management, supervision, operating costs); • Included execution strategy and inputs provided by Artemis Gold. Construction indirect costs were developed based on an EPCM execution strategy, and inputs from Artemis Gold.
Freight costs	Freight costs were estimated as follows: • Unit rate per container based on the number of container loads per Phase; • Percentage of supply cost; Freight costs were validated against provided budget quote from logistics contractors.
Spares parts	A value for spares was estimated by Lycopodium and included in the estimate. Estimates for wear consumption rates for major equipment were based on vendor data, and where applicable, were factored to account for differences in process variables.
Vendor representatives	Major equipment vendors were identified, and rates applied to calculate the vendor representatives' costs.
Heavy lift cranes	A rental provision for a 330 t crawler crane and a 550 t all-terrain crane was included in the estimate.
Temporary facilities	Estimated based on construction duration and site requirements. Construction services, power, water, personal protective equipment, communications, computers, IT services, servers and telephones are included as Owner's costs.
Material management	Estimated based on construction duration and site requirements.
Construction mobile equipment	Included for warehousing and plant operations and was based on construction duration and site requirements.
Site safety	Estimated based on construction duration and site requirements and includes consumables, safety tools, winter clothing, personal protective equipment, and safety awards.
Fuel	Diesel fuel consumption based on information supplied by contractors. A fuel rate if $1.35/L is assumed.

Estimation methods incorporated, or were based on, the following:

- A 3D model to support engineering quantity estimates for earthworks, concrete, steelwork, mechanical and electrical for the crushing plant, processing plant, conveying systems, and infrastructure;
- Unit rates, which reflect current market conditions, for the supply of bulk materials, benchmarked against projects that were either under construction or completed.

All bulk earthworks were costed on an Owner-managed basis, using local resources and Owner's equipment, with applicable rates and distributables included to support the works.

A summary of the process plant capital costs is provided in Table 21-3.

Table 21-3: Process Plant Capital Cost Estimate

Process	Area	Phase 2 C$ M	Phase 3 C$ M	Total C$ M
Growth capital cost estimate	General, site-wide, construction indirects	55	63	***118***
	Water management structures	12	12	***24***
	Crushing	36	56	***92***
	Process plant buildings	36	32	***68***
	Grinding	114	122	***235***
	Leaching and adsorption	49	50	***99***
	Elution and goldroom	28	25	***52***
	Tails handling	8	7	***14***
	Reagent	19	26	***44***
	Process plant utilities	25	29	***54***
	Process plant infrastructure	10	17	***27***
	Process plant indirects	71	81	***152***
Total		***462***	***519***	***981***

Notes: Growth capital cost estimate includes deferred initial capital costs. Numbers have been rounded.

The estimate assumes that the process plant will be executed using an EPCM approach. EPCM services costs for Phase 2 were factored at 12.5% of total installed direct cost excluding mining, water management, TSF and infrastructure costs.

Artemis Gold made a capital cost estimate provision for a front-end engineering and design (FEED) study and early work phase prior to the Phase 2 EPCM, which is not included in the capital cost estimate.

Phase 3 EPCM services costs were factored at 13% of total installed direct costs, excluding mining, water management, TSF and infrastructure costs.

Each line item in the process plant capital estimate had a contingency value allocated based on the apparent risk level. The assessment of the risk level was based on the present engineering progress, information provided by vendors and Artemis Gold, and database information.

Contingency was individually applied to quantities, unit rates and costs and compiled as a weighted average to form the overall recommended contingency.

Contingency assumptions are provided in Table 21-4.

Table 21-4: Process Plant Contingency

Item	Unit	Phase 2	Phase 3
Process plant	%	9.33	11.52
Indirects	%	13.28	13.67
EPCM		Included in Owner's cost	Included in Owner's cost

21.2.4 Tailings Storage Facility and Water Management Capital Costs

The capital cost estimate for the TSF and water management structures/systems was developed based on the current understanding and level of design detail available.

The estimated material quantities comprise material volumes, areas, and neat lengths and quantities from design drawings. The unit rates were developed by Knight Piésold from a combination of data collected during pre-production construction, vendor quotations, manufacturer information, and industry standards and rates.

Material for construction is assumed to be sourced from the pit with loading and hauling included in the mining costs.

The estimated growth and sustaining capital costs are summarized in Table 21-5. These estimated costs exclude earthworks costs captured in the mining cost estimate.

Table 21-5: Tailings Storage Facility and Water Management Capital Cost Estimate

Description	Growth Capital (C$ M)	Sustaining Capital (C$ M)
Water management structures	45	64
Tailings storage facility	32	183
Subtotal direct costs	*77*	*247*
Annual engineering activities and construction QA/QC, field reviews, and design office support	11	34
Contingency	9	0
Subtotal indirect costs	*20*	*34*
Total	***97***	***281***

Note: Growth capital cost estimate includes deferred initial capital costs. Numbers have been rounded.

A contingency was added to growth capital costs where required.

21.2.5 Infrastructure Capital Costs

Infrastructure capital costs were derived from vendor quotations for buildings, heating, ventilation, and air conditioning, tankage, and recent construction activities. Quantities were estimated for the foundations and civil work within the mine services area and overall pricing was applied to those quantities. Where detailed quantities were not developed, a factored approach was taken in line with the level of definition for this study.

Onsite infrastructure costs total C$17 M over the LOM.

21.2.6 Owner's Capital Costs

Owner's capital costs were derived from Phase 1 cost data, and are summarized in Table 21-6.

Table 21-6: Owner's Capital Cost Estimate

Owners	Area	C$ M
Growth capital cost estimate	Personnel/labour	17
	Camp services	26
	Site services	7
	Health & safety	0
	General expenses	16
	Environment	32
	Infrastructure/equipment	21
	Subtotal	*118*
Sustaining capital cost estimate	Personnel/labour	9
	Camp services	13
	Site services	1
	Health & safety	0
	Environment	58
	Infrastructure/equipment	2
	PST	34
	Subtotal	*117*
Total		***235***

Notes: Growth capital cost estimate includes deferred initial capital costs. Numbers have been rounded. Numbers expressed as a zero are due to rounding.

Key elements within the estimate were: environmental management system programs; camp services; administrative salaries and wages; principal repayments; insurance and establishment fees; transport, equipment hire, freight and logistics; fuel and consumables; vehicles and mobile equipment; repairs, maintenance, and tooling; safety, training, and development; winter preparation; IT and communications; consultants and contractors; and general expenses.

21.2.7 Capital Cost Summary

The overall capital cost estimate is provided in Table 21-7.

Table 21-7: Capital Cost Estimate Summary Table

Major Area	Discipline Area	C$ M
Growth capital cost estimate	Mining	285
	Process	981
	TSF and water management	97
	Infrastructure	17
	Owners	118
	Subtotal	*1,497*
Sustaining capital cost estimate	Mining	724
	Process	—
	TSF and water management	281
	Infrastructure	—
	Owners	117
	Subtotal	*1,122*
Growth + sustaining capital cost estimates	***Total***	***2,619***
Closure costs (net of salvage value)		250

Notes: The growth capital cost estimate includes deferred initial capital costs. Numbers have been rounded.

21.3 Operating Cost Estimates

21.3.1 Basis of Estimate

Data supporting the estimate were obtained during Q4 2023 and Q1 2024.

The operating costs have been derived using first principal estimates based on typical operating data/standard industry practice and inputs compiled from a variety of sources, including the following:

- Labour pays rates and manning as advised by Artemis Gold;
- Grid power costs as based on BC Hydro's industrial tariff, which includes an energy price at a rate of 0.051 C$/kWh and energy demand cost of 8.78 C$/kVA billed at the highest average in a 30 minute interval per billing period;
- Diesel fuel cost of $1.35/L;
- Consumable prices from supplier budget quotations and prices confirmed by Artemis Gold.

21.3.2 Mine Operating Costs

The mine operating cost estimates are built up from first principles and consist of the following components:

- Equipment operating cost: the activities of grade control, drilling, blasting, loading, hauling, mining support, and equipment maintenance. Equipment operating cost estimates are based on the total required annual operating hours (calculated from the equipment productivities) and the cost per SMU hour to operate the equipment. The largest component of the estimated mine operating costs is for the hauling function, and a significant portion of the planned hauls for Blackwater are downhill loaded, especially early in the mine life;
- Salary and hourly personnel: mine department salary staff and general mining labour; fleet operator and maintenance labour costs are included as part of the equipment operating costs, so distributed into the direct mining functions (drill, load, haul, etc.). During peak production, the mine is expected to employ 50 salaried personnel and 330 hourly personnel;
- General mine expense costs: miscellaneous tools and equipment necessary to support mine operation, such as surveying, mine planning software, geotechnical instrumentation for pit and dumps, office costs and overheads for mine operations, mine maintenance and technical services departments.

The total mining cost is approximately C$3.0 billion to mine 1.0 Bt of material, and move 1.1 Bt of material, including ore and waste rehandle. The total unit cost is approximately C$2.97/t mined (Table 21-1).

Table 21-8: Mine Operating Costs (operations period)

Area	LOM (Years 1–17)		Phase 1 (Years 1–2)		Phase 2 (Years 3–6)		Phase 3 (Years 7–15)		Stockpile (Years 16–17)
	C$M	C$/t	C$M	C$/t	C$M	C$/t	C$M	C$/t	C$M
Total mine operating cost	2,979	2.97	239	2.52	911	2.54	1,783	3.24	46
Less: capitalized tailings dam expansion	-83	-0.00	-42	-0.02	-41	-0.03	—	—	—
Less: capitalized mine fleet major component maintenance	-407	-0.41	-5	-0.05	-134	-0.37	-259	-0.47	-9
Add: PST for mining operations	11	0.01	1	0.01	2	0.01	7	0.01	1
Adjusted Mine Operating Costs	***2,500***	***2.57***	***193***	***2.46***	***738***	***2.15***	***1,531***	***2.78***	***38***

Note: Unit costs expressed as C$/t mined. Tailings expansion mine operating costs are allocated to growth capital costs, mining fleet major component maintenance costs are allocated to sustaining capital costs. Numbers have been rounded.

The portion of mine operating costs related to construction of TSF C and D dams in Years 1, 2, and 6 (estimated as C$83 M covering the mining of 32 Mt), and the portion of mine operating costs related to equipment major component rebuilds/replacement over the entire LOM (estimated as C$407 M), are treated as growth capital in the financial model. With the re-allocation, the total unit cost is approximately C$2.57/t mined (refer to Table 21-1).

The estimated impact of provincial sales tax (PST) in BC was added to the mining cost estimate.

21.3.3 Process Operating Costs

Process costs assume the following:

- Phase 1: 6–6.5 Mt/a feeding the existing Phase 1 circuit;
- Phase 2: expansion by 9 Mt/a to 15 Mt/a;
- Phase 3: expansion by 9.5 Mt/a to 25 Mt/a;
- Grind size: P_{80} grind size of 150 µm;
- Operating: 24 hour per day operation, 365 days per year.

Processing operating costs have been developed for a LOM blend. It is expected that the plant will operate on a range of mineralised material blends. The LOM processing costs are a weighted average of the various mineralised material type processing costs based on the LOM blend.

The estimate basis is summarized in Table 21-9. The process cost estimates by phase are included in Table 21-10.

Table 21-9: Process Cost Estimate Inputs

Item	Note
Reagent and consumables	Individual reagent consumption rates were estimated based on the metallurgical testwork results, industry practice and peer-reviewed literature. Each reagent cost was obtained through the updated quote from vendors or suppliers. Other consumables (e.g. liners for the crusher, screen deck panels, ball mill liners and grinding media) were estimated using the following inputs: • Metallurgical testwork results (abrasion and bond work indices); • Comminution study of Orway Mineral Consultants; • Supplier/vendor quote.
Plant maintenance	Annual maintenance costs were estimated based on a total installed mechanical cost by area using factors applied to the equipment cost. The cost estimate of plant mobile equipment maintenance was based on the industrial practice referred to the calculation of similar size of processing plant.

Item	Note
	The rental fees, fuel, and maintenance costs for the mobile equipment and special personal protective equipment are all included in the process maintenance centre.
Laboratory	Laboratory cost was based on a quote from SGS, which was prepared according to the estimation of sample counts, assay elements and turnaround requirements etc.
Labour	Staffing was estimated by benchmarking against similar size of processing plants. The labour costs integrated for plant operation include management/administration, metallurgy, operations, and maintenance.
Service and utilities	No direct water supply costs were included at Artemis Gold's request.
Process plant pre-production and working capital costs	Pre-production cost related to the processing labour, the construction and commissioning of the Phase 2 and 3 circuit is not included as the Phase 1 operation will be operational. First fill reagents and other consumables required for the Phase 2 and Phase 3 process plants are expected to be paid for in the commissioning and ramp up period, from the existing Phase 1 operating cost budget.

Table 21-10: Process Cost Estimate

Area	Phase 1		Phase 2		Phase 3	
	C$M/a	C$/t	C$M/a	C$/t	C$M/a	C$/t
Reagents and consumables	38.8	6.47	62.7	6.97	65.8	6.92
Plant maintenance	2.8	0.46	3.9	0.43	4.2	0.45
Laboratory	2.3	1.60	0.35	0.04	0.57	0.06
Power	9.6	1.99	14.1	1.57	14.6	1.53
Labour	12.0	0.39	4.2	0.47	4.2	0.45
Total	***65.5***	***10.92***	***85.3***	***9.48***	***89.4***	***9.41***

Note: Unit costs expressed as C$/t milled. Numbers have been rounded.

For the first year of each operating phase a reduction in liner and mechanical consumables costs were applied. The net result is a reduction of the average Phase process operating cost over the period of operation as indicated in Table 21-11.

Table 21-11: Process Cost Estimate (period average)

Process Cost Centre	Units	Phase 1 (Year 1–2)	Phase 2 (Year 3–6)	Phase 3 (Year 7–15)	Stockpile Phase (Year 16–17)	LOM (Year 1–17)
Reagents & consumables	$/t milled	6.55	6.77	6.81	6.81	6.79
Plant maintenance	$/t milled	0.30	0.37	0.41	0.44	0.40
Laboratory	$/t milled	0.32	0.18	0.13	0.13	0.15
Power	$/t milled	1.58	1.58	1.55	1.55	1.56
Labour	$/t milled	1.68	1.08	0.82	0.82	0.91
PST	$/t milled	0.08	0.08	0.08	0.08	0.08
Total	***$/t milled***	***10.51***	***10.06***	***9.80***	***9.83***	***9.88***

Note: Numbers have been rounded.

21.3.4 Infrastructure Operating Costs

Infrastructure operating costs including fuel, power and maintenance are included in general and administrative costs.

21.3.5 General and Administrative Operating Costs

G&A costs included senior management, administration, insurance, environmental, camp and travel, health and safety, emergency response and other costs required to support the operation. Expenses covered under site service costs are, among others, power costs, water treatment costs and powerline/road maintenance costs.

The G&A operating costs are summarized in Table 21-12.

Table 21-12: G&A Operating Cost Estimate

Area	Phase 1 (Years 1–2)		Phase 2 (Years 3–6)		Phase 3 (Years 7–15)		Stockpile Phase (Years 16–17)		LOM (Years 1–17)	
	C$M	C$/t	C$M	C$/t	C$M	C$/t	C$M	C$/t	C$M	C$/t
Personnel	39.3	2.62	89.3	1.49	229.1	1.02	25.4	0.74	383.1	1.15
Camp services	32.4	2.16	99.8	1.66	275.9	1.23	34.7	1.01	442.8	1.32
Site services	7.8	0.52	16.7	0.28	37.8	0.17	5.1	0.15	67.5	0.20
Total	***79.6***	***5.30***	***205.8***	***3.43***	***542.8***	***2.41***	***65.2***	***1.90***	***893.4***	***2.67***

Note: Unit costs expressed as C$/t milled. Numbers have been rounded.

21.3.6 Operating Cost Summary

Operating costs are summarized in Table 21-13.

Table 21-13: LOM Operating Cost Summary

Area	Units	Phase 1 (Years 1–2)	Phase 2 (Years 3–6)	Phase 3 (Years 7–15)	Stockpile Phase (Years 16–17)	LOM (Years 1–17)
Mining	$/t mined	2.46	2.15	2.78	n/a	2.57
Process	$/t milled	10.51	10.06	9.80	9.83	9.88
G&A	$/t milled	5.30	3.43	2.41	1.90	2.67

Note: Mining costs excludes the cost of mine fleet major component maintenance which are reported as sustaining capital and include low-grade ore stockpile rehandle. Numbers have been rounded.

22.0 ECONOMIC ANALYSIS

22.1 Cautionary Statement

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

- Mineral Resource and Mineral Reserve estimates ;
- Assumed commodity prices and exchange rates;
- Mine production plans;
- Projected recovery rates;
- Sustaining and operating cost estimates;
- Assumptions as to closure costs and closure requirements;
- Assumptions as to environmental, permitting, and social risks.

Additional risks to the forward-looking information include:

- Changes to costs of production from what is assumed;
- Changes to construction execution strategy from what is assumed;
- Unrecognized environmental risks;
- Unanticipated reclamation expenses;
- Unexpected variations in quantity of mineralized material, grade, or recovery rates;
- Geotechnical and hydrogeological considerations during mining being different from what was assumed;
- Failure of plant, equipment, or processes to operate as anticipated;
- Accidents, labour disputes and other risks of the mining industry.

22.2 Methodology Used

All dollar amounts are expressed in Q1 2024 Canadian dollars, unless otherwise noted.

The expansion study assumes that construction of the 6 Mt/a Phase 1 process plant is completed. The purpose of the expansion study is to optimize the timing of mine expansion through the advancement of Phase 2 to Year 3 of operations at an increased production capacity of 15 Mt/a, and Phase 3 to Year 7 of operations at an increased production capacity of 25 Mt/a.

Phase 1 capital costs of C$730–C$750 M are considered sunk costs for the purposes of the economic analysis in this Report. The net present value (NPV) is reported net of the scheduled repayment of the C$385 million loan facility associated with Phase 1, with all gold and silver stream participations included.

The NPV at a 5% discount rate is discounted to the commencement of Phase 2 construction.

22.3 Financial Model Parameters

The economic analysis is based on the metallurgical recovery predictions in Section 13.3, the Mineral Reserve estimates in Section 15, the mine plan discussed in Section 16, infrastructure requirements set out in Section 18, payability assumptions in Section 19.1, the commodity price forecasts in Section 19.2, closure cost estimates in Section 20.4, and the capital and operating costs outlined in Section 21.

The economic analysis is based on 100% equity financing and is reported on a 100% ownership basis. The economic analysis assumes constant prices with no inflationary adjustments, and uses a reverting gold price curve as discussed in Section 19.2.

The cashflow includes provision for two private NSR royalties at 1.0% and 1.5% over portions of the Mineral Reserve, which were applied to the economic cash flow model based upon the mine plan.

Estimated payments to Indigenous groups are included in the cash flow model.

Closure costs are estimated at approximately C$250 M, which includes a salvage value of C$37 M. Closure costs are assumed to be applied in Year 18. Bonding of the reclamation and closure costs has been applied throughout the model, and are based on progressive disturbance.

The cashflow assumes the repayment of the loan facility associated with Phase 1 as follows:

- The loan facility of C$385 million will be repaid in quarterly instalments over a six-year term commencing 31 May, 2025, with a repayment holiday during Years 4 and 5;
- An annual interest rate of Canadian Dollar Offered Rate (assumed at 4.0%) plus a margin of 4.75% up to 31 May, 2025 with the margin reducing to 4.25% thereafter;
- Commitment fees of 1.75% associated with the unused C$40 M cost overrun facility;
- Phase 2 and Phase 3 growth capital is assumed to be funded through operating cashflow.

22.4 Taxes

Key provincial and federal tax considerations in the economic analysis include:

- BC mining tax: 2% provincial minimum tax payable on net current proceeds which is creditable against the 13% effective mining tax rate which is calculated based on operating profit less applicable capital cost deductions. The mining tax is deductible in computing provincial and federal income tax;
- BC provincial income tax: 12.0%, payable after applicable deductions are used;
- Canadian federal income tax: 15.0%, payable after applicable deductions are used.

22.5 Economic Analysis

The after-tax NPV is C$3.25 B, using a 5% discount rate. The Blackwater Gold Mine is cashflow positive in Phase 1, and so there is no internal rate of return or payback period that is relevant to this Report.

A cashflow summary table is provided in Table 22-1.

Table 22-1: Summary Cashflow Analysis

Item	Unit	First 5 Years	First 10 Years	LOM (17 years)
Average throughput capacity	Mt/a	12	18	20
Gold grade	g/t	1.29	0.91	0.75
Silver grade	g/t	7.75	5.92	5.78
Gold equivalent grade	AuEq g/t [1]	1.36	0.96	0.79
Gold recoveries	%	93	93	93
Average annual gold production	Au oz	463,000	478,000	438,000
Average annual silver production	Ag oz	1,944,000	2,165,000	2,376,000
Average annual gold equivalent production	AuEq oz [2]	488,000	506,000	469,000
Strip ratio	Waste:ore	1.99	2.13	2.01
Growth capital [3,4]	C$ M	1,174	1,497	1,497
Sustaining capital [4]	C$ M	499	874	1,122
Operating costs	C$/t milled	26.86	23.00	20.03
Cash costs [5]	US$/oz	456	577	645
All-in sustaining costs [6]	US$/oz	615	712	781
Average annual free cash flow [7]	C$ M	552	489	413
After-tax NPV5% [8]	C$ B	3.25		

Notes: Numbers have been rounded.

1. Gold equivalent grades were determined using a gold price of US$1,800/oz, a silver price of US$23/oz, a gold metallurgical recovery of 93%, a silver metallurgical recovery of 65%, and mining smelter terms for the following equation: AuEq = Au g/t + (Ag g/t x 0.0085).

2. Gold equivalent ounces were determined using a gold-to-silver ratio of 78:1 (US$1,800:US$23).
3. Includes deferred initial capital costs.
4. Excludes closure costs and salvage value.
5. Cash costs include selling costs, royalty payments, operating costs, less silver by-product credits and adjustments to stockpile inventory, divided by payable gold ounces.
6. All-in sustaining costs include cash costs as defined above, sustaining capital and closure costs, divided by payable gold ounces.
7. Free cash flow = operating cash flow less sustaining capex, closure costs and taxes.
8. After-tax NPV represents the net present value of after-tax cash flows, discounted at a rate of 5%. The after-tax cash flows take into account the repayment of the Project loan facility of $385 million, as well as the effect of the gold stream and silver stream arrangements.

The full cashflow on an annualized basis is included as Table 22-2 (Years 1 to 10) and Table 22-3 (Years 11 to 17).

Table 22-2: Annualized Cashflow (Years 1 to 10)

Year	Units	*Totals*	1	2	3	4	5	6	7	8	9	10
Stage	Phase		1	1	2	2	2	2	3	3	3	3
Mill throughput	Mt	***334***	6	9	15	15	15	15	25	25	25	25
Recovered gold	('000) oz	***7,453***	300	377	546	574	521	478	553	472	467	495
Recovered silver	('000) oz	***40,399***	1,141	1,238	2,457	1,972	2,911	2,533	3,305	2,593	1,509	1,994
Net revenue	C$ (M)	***17,798***	775	954	1,347	1,350	1,211	1,111	1,314	1,119	1,075	1,150
Operating cost	C$ (M)	***6,697***	194	236	373	408	400	366	526	505	515	502
Growth capital	C$ (M)	***1,497***	323	298	132	83	338	286	29	—	8	—
Sustaining capital	C$ (M)	***1,122***	62	78	107	138	115	97	64	78	61	75
Closure costs	C$ (M)	—	—	—	—	—	—	—	—	—	—	—
Debt service; Phase 1	C$ (M)	***477***	82	139	112	80	5	58	—	—	—	—
Working capital and bonding	C$ (M)	***135***	(19)	28	5	7	7	8	8	8	10	10
Pre-tax cash flow	C$ (M)	***7,870***	133	175	618	635	345	296	687	529	481	563
Taxes	C$ (M)	***2,786***	12	76	224	268	188	158	211	155	144	184
Post-tax cash flow	C$ (M)	***5,083***	121	98	394	367	157	138	475	374	338	379
Free cash flow [1]	C$ (M)	***7,192***	507	564	643	536	507	490	512	382	355	389

Note: Free cash flow is net revenue less operating costs, sustaining capital costs, closure capital costs (net of salvage value) and taxes. Closure costs are discussed in Section 22.3. Numbers have been rounded.

Table 22-3: Annualized Cashflow (Years 11 to 17)

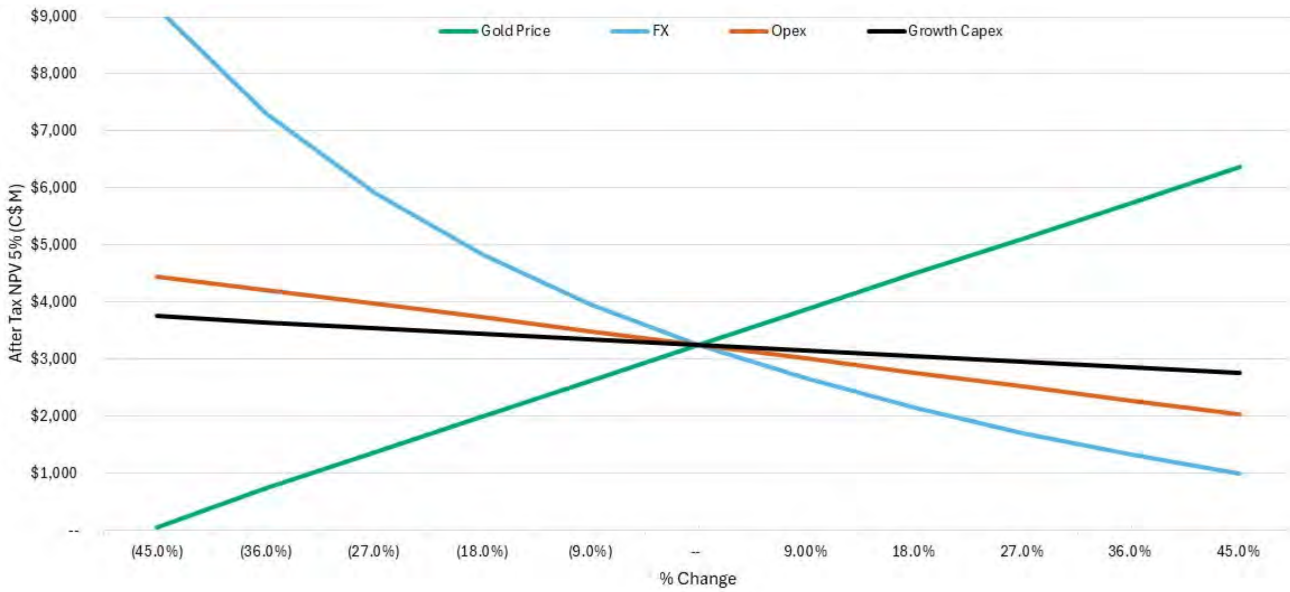
Year	Units	11	12	13	14	15	16	17
Stage	Phase	3	3	3	3	3	3	3
Mill throughput	Mt	25	25	25	25	25	25	9
Recovered gold	('000) oz	455	572	557	482	295	226	84
Recovered silver	('000) oz	2,428	2,633	2,850	2,547	3,452	3,527	1,309
Net revenue	C$ (M)	1,064	1,328	1,341	1,154	728	571	208
Operating cost	C$ (M)	484	500	478	437	334	319	120
Growth capital	C$ (M)	—	—	—	—	—	—	—
Sustaining capital	C$ (M)	63	64	47	27	20	21	7
Closure costs	C$ (M)	—	—	—	—	—	—	—
Debt service; Phase 1	C$ (M)	—	—	—	—	—	—	—
Working capital and bonding	C$ (M)	9	9	9	9	10	9	9
Pre-tax cash flow	C$ (M)	508	755	807	681	364	221	72
Taxes	C$ (M)	167	264	282	236	123	75	18
Post-tax cash flow	C$ (M)	341	491	525	445	241	147	53
Free cash flow [1]	C$ (M)	350	500	534	454	251	156	62

Note: Free cash flow is net revenue less operating costs, sustaining capital costs, closure capital costs (net of salvage value) and taxes. Closure costs are discussed in Section 22.3. Numbers have been rounded.

22.6 Sensitivity Analysis

A sensitivity analysis was performed examining capital costs, operating costs, foreign exchange rate, gold grade and gold price as shown in Figure 22-1 and Table 22-4.

Figure 22-1: After-Tax NPV 5% Sensitivity Analysis



Note: Figure prepared by Artemis Gold, 2024. FX = foreign exchange rate, opex = operating cost; capex = capital cost.

Table 22-4: Base Case After-Tax NPV 5% Sensitivity to Gold Price and Foreign Exchange (C$B)

C$:US$	Long-term Gold Price (US$/oz)															
	1,100	**1,200**	**1,300**	**1,400**	**1,500**	**1,600**	**1,700**	**1,800**	**1,900**	**2,000**	**2,100**	**2,200**	**2,300**	**2,400**	**2,500**	**2,600**
0.71	1.6	1.9	2.2	2.4	2.7	2.9	3.2	3.4	3.7	3.9	4.2	4.5	4.7	5.0	5.2	5.5
0.72	1.6	1.9	2.1	2.4	2.6	2.9	3.1	3.4	3.6	3.9	4.1	4.4	4.6	4.9	5.1	5.4
0.73	1.6	1.8	2.1	2.3	2.6	2.8	3.1	3.3	3.6	3.8	4.0	4.3	4.5	4.8	5.0	5.3
0.74	1.5	1.8	2.0	2.3	2.5	2.8	3.0	**3.2**	3.5	3.7	4.0	4.2	4.5	4.7	4.9	5.2
0.75	1.5	1.7	2.0	2.2	2.5	2.7	2.9	3.2	3.4	3.7	3.9	4.1	4.4	4.6	4.9	5.1
0.76	1.4	1.7	1.9	2.2	2.4	2.7	2.9	3.1	3.4	3.6	3.8	4.1	4.3	4.5	4.8	5.0
0.77	1.4	1.6	1.9	2.1	2.4	2.6	2.8	3.1	3.3	3.5	3.8	4.0	4.2	4.5	4.7	4.9

Note: Report base case is highlighted. Numbers have been rounded.

The NPV is most sensitive to fluctuations in gold price (gold grade) and foreign exchange rate assumptions, and less sensitive to variations in capital and operating costs. The gold grade is not presented in the sensitivity graph because the impacts of changes in the gold grade mirror the impact of changes in the gold price.

23.0 ADJACENT PROPERTIES

This section is not relevant to this Report.

24.0 OTHER RELEVANT DATA AND INFORMATION

This section is not relevant to this Report.

25.0 INTERPRETATION AND CONCLUSIONS

25.1 Introduction

The QPs note the following interpretations and conclusions in their respective areas of expertise, based on the reviews and interpretations of data available for this Report.

25.2 Mineral Tenure, Surface Rights, Water Rights, Royalties and Agreements

The mineral tenure held is valid, and the granted mining licence is sufficient to support Mineral Resource and Mineral Reserve estimation.

Surface and water rights are granted, and sufficient to support mining operations.

The mineral claims are subject to a number of NSR royalties, ranging from 1–3%. Artemis Gold may purchase a portion of selected NSRs. A portion, but not all, of the mining lease is subject to NSR royalties that range from 1–3%.

The purchase agreement between Artemis Gold and New Gold included a gold stream agreement. New Gold maintained a security interest over the Blackwater Gold Mine in connection with the gold stream agreement. On 13 December, 2021, New Gold announced the sale of the gold stream agreement to Wheaton.

25.3 Geology and Mineralization

The Blackwater deposit is considered an example of a volcanic-hosted, epithermal-style gold–silver deposit. The QP considers that exploration programs that use an epithermal deposit model are appropriate to the Project area.

The geological understanding of the settings, lithologies, and structural and alteration controls on mineralization in the different zones is sufficient to support estimation of Mineral Resources and Mineral Reserves. The geological knowledge of the area is also considered sufficiently acceptable to reliably inform mine planning.

The mineralization style and setting are well understood and can support declaration of Mineral Resources and Mineral Reserves.

The Project area offers excellent exploration potential as the deposit is open at depth, particularly in the northwest of the deposit where an increasing trend in gold grade is noted.

25.4 Exploration, Drilling and Analytical Data Collection in Support of Mineral Resource Estimation

The exploration programs completed to date are appropriate for the deposit style.

Sampling methods are acceptable for Mineral Resource and Mineral Reserve estimation.

Sample preparation, analysis and security are generally performed in accordance with exploration best practices and industry standards.

The quantity and quality of the lithological, geotechnical, collar and down-hole survey data collected during the exploration and delineation drilling programs are sufficient to support Mineral Resource and Mineral Reserve estimation. The collected sample data adequately reflect deposit dimensions, true widths of mineralization, and the deposit style. Sampling is representative of the gold and silver grades in the deposits, reflecting areas of higher and lower grades.

The QA/QC programs adequately address issues of precision, accuracy, and contamination. Drilling programs typically included blanks, duplicates, and CRM samples. QA/QC submission rates meet industry-accepted standards.

The data verification programs concluded that the data collected adequately support the geological interpretations and constitute a database of sufficient quality to support the use of the data in Mineral Resource and Mineral Reserve estimation.

25.5 Metallurgical Testwork

Metallurgical testwork and associated analytical procedures were appropriate to the mineralization type, appropriate to establish the optimal processing routes, and were performed using samples that are typical of the mineralization style.

Samples selected for testing were representative of the various types and styles of mineralization to be mined and processed within the first five years of operation. Samples were selected from a range of depths within the deposits. Sufficient samples were taken so that tests were performed on sufficient sample mass. Additional variability testwork is recommended for mineralization in the later periods of the mine plan.

Recovery factors estimated are based on appropriate metallurgical testwork, and are appropriate to the mineralization types and the selected process route. A recovery of 93% for gold and 65% for silver is recommended for mine planning purposes.

There are no deleterious elements known that would affect process activities or metallurgical recoveries.

25.6 Mineral Resource Estimates

Mineral Resources are reported using the 2014 CIM Definition Standards, and assume open pit mining methods.

Factors that may affect the Mineral Resource estimates include: metal price assumptions; changes to the assumptions used to generate the gold equivalent cut-off grade; changes in local interpretations of mineralization geometry and continuity of mineralized zones; changes to geological and mineralization shapes, and geological and grade continuity assumptions; changes to density and domain assignments; changes to geotechnical, mining and metallurgical recovery assumptions; changes to the input and design parameter assumptions that pertain to the conceptual pit constraining the estimates; and assumptions as to the continued ability to access the site, retain mineral and surface rights titles, maintain environmental and other regulatory permits, and maintain the social license to operate.

25.7 Mineral Reserve Estimates

Proven and Probable Mineral Reserves were modified from Measured and Indicated Mineral Resources. Inferred Mineral Resources were set to waste.

The Mineral Reserves are supported by the 2024 Expansion Study Mine Plan and classified in accordance with the 2014 CIM Definition Standards.

Blackwater Proven and Probable Mineral Reserves total 334.3 Mt grading 0.75 g/t Au and 5.8 g/t Ag (0.78 g/t AuEq).

Changes in the following factors and assumptions may affect the Mineral Reserve estimate: metal prices and foreign exchange rates; interpretations of mineralization geometry and continuity of mineralization zones; geotechnical and hydrogeological assumptions; changes to pit designs from those currently envisaged; ability of the mining operation to meet the annual production rate; changes to operating and capital cost assumptions; mining and process plant recoveries; and the ability to meet and maintain permitting and environmental license conditions and the ability to maintain the social license to operate.

25.8 Mine Plan

The mining operations will use conventional open pit mining methods and equipment.

Mining will be based on a phased approach with stockpiling to bring high-grade forward and provide operational flexibility.

A reasonable open pit mine plan, mine production schedule and mine capital and operating cost estimates were developed.

An annual mill feed rate of 6 Mt is targeted for the first year of operation, increasing to 15 Mt for the next five years, and 25 Mt thereafter. Low-grade ore is stockpiled and re-handled to the crusher before the end of mine life.

Open pit mining operations are anticipated to run for 15 years, excluding pre-production mining. Following mining operations, stockpiled low-grade material is expected to be processed for an additional two years, resulting in a total mine life of 17 years.

Pit layouts and planned mine operations are typical of other open pit gold operations in Canada, and the unit operations within the developed mine operating plan are proven to be effective for these other operations.

The mine plan supports the cash flow model and financials developed for the 2024 Expansion Study.

25.9 Recovery Plan

The process methods are conventional to the industry. The comminution and recovery processes are widely used in the industry with no significant elements of technological innovation.

The process plant flowsheet design was based on testwork results and industry standard practices.

The plant design envisages an initial throughput rate for Phase 1 of 6 Mt/a, an upgrade of 9 Mt/a in Phase to for an overall production rate of 15 Mt/a, a de-bottlenecking of Phase 1 to allow 6.5 Mt/a production, and a Phase 3 upgrade of 9.5 Mt for an overall production rate in Phase 3 of 25 Mt/a.

The proposed process facilities are appropriate to the mineralization style.

The plant will produce variations in recovery due to the day-to-day changes in ore type or combinations of ore type being processed. These variations are expected to trend to the forecast recovery value for monthly or longer reporting periods.

25.10 Infrastructure

The overall facilities and major infrastructure cover the mine site area, TSF, WRSF, camp site, main access road, and site wide water management systems.

At start-up, the camp will accommodate 312 personnel. Additional accommodations required to meet peak occupancy for Phase 2 and Phase 3 construction will be provided through short-term dormitory rental.

The TSF was designed to permanently store tailings, PAG waste rock, and potentially metals leaching NAG waste rock generated during mine operation. Two valley-fill style impoundments are planned, TSF C (Stage 1 construction is currently underway) and TSF D (initial construction will be required in approximately Year 3 to support Phase 2 expansion).The TSF embankments will be engineered, water retaining, zoned earth-rockfill structures constructed primarily from NAG waste rock and overburden materials generated during open pit development and from external borrows located in the vicinity of the TSF. Site water management structures are designed to collect and divert non contact surface water not needed for mine operations and to collect and control mine affected contact water. The water management systems will be adjusted progressively during mine operations as the TSF footprint expands.

There is sufficient area within the mining lease for the TSF and water management facilities envisaged in the LOM plan, and sufficient material available to progressively construct the TSF dams.

Power will be sourced from the BC Hydro grid. The Blackwater Gold Mine will require up to 113 MW (70 MW for Phases 1/2 and 43 MW for Phase 3) of power once the full mill throughput is realized in Year 7. Standby generators will provide power to the process plant and ancillary electrical equipment in the event of a utility power failure.

25.11 Environmental, Permitting and Social Considerations

The Blackwater Gold Mine is supported by a suite of environmental, social, economic, and cultural heritage baseline studies and potential effects were fully assessed.

The Blackwater Gold Mine was granted an Environmental Assessment Certificate #M19-01 and an Environmental Assessment Decision Statement in 2019. Assessment of components to address updates in the design were considered in recent permits.

To manage potential effects of the Blackwater Gold Mine, an Environmental Management System is supported by a comprehensive set of management plans.

Reclamation of the Blackwater Gold Mine area will conform to the requirements of the Health, Safety, and Reclamation Code for Mines in BC. The Reclamation and Closure Plan is approved through *Mines Act* Permit M-246 and describes how end land use and land capability objectives will be achieved.

All major provincial and federal permits, licenses, and authorizations for construction and operation were issued. The Blackwater Gold Mine is currently authorized for a milling capacity of 55,000 t/day (or 20 Mt/a). Prior to implementing the Phase 3 mill capacity increase, amendments to several permits will be required including *Mines Act* Permit M-246 and *Environmental Management Act* Permits #110650 and #110652.

The Environmental Assessment Certificate contains 43 binding conditions, which identify requirements for environmental and social management plans, consultation requirements related to management plans, and requirements for an Environmental Monitoring Committee, Community Liaison Committee, Independent Environmental and Aboriginal Monitor(s). The federal Decision Statement includes 102 binding conditions, which identify requirement for plans to offset impacts, consultation requirements for offset plans and follow-up programs, and specific mitigation measures. Artemis Gold is addressing these conditions in accordance with the timelines specified by the Environmental Assessment Certificate and Decision Statement.

The Blackwater Gold Mine is located within the traditional territories of Ulkatcho First Nation, Lhoosk’uz Dené Nation. The power transmission line and other infrastructure pass through the traditional territories of Nadleh Whut’en First Nation, Saik’uz First Nation, and Stellat’en First Nation (collectively, the Nechako First Nations).

Since acquiring the Project in August 2020, Artemis Gold has engaged with Indigenous Groups to provide updates on the Blackwater Gold Mine's permitting requirements, review of draft regulatory applications, and construction progress.

25.12 Markets and Contracts

No formal marketing studies were completed. The planned doré product is readily marketable.

Artemis Gold has an existing gold stream agreement with Wheaton, whereby Wheaton will purchase 8.0% of the refined gold produced from the Blackwater Gold Mine. Once 371,908 ounces of refined gold have been delivered to Wheaton, the gold stream will reduce to 4.0%. Wheaton will make payments for the gold purchased equal to 35% of the US dollar gold price quoted by the London Bullion Market Association two days prior to delivery.

Mineral Resources were estimated using a gold price of US$2,000/oz Au and a silver price of $16/oz Ag.

Mineral Reserves were estimated using a gold price of US$1,400/oz Au and a silver price of $15/oz Ag.

The cashflow analysis uses a reverting price curve for gold. The average January 2024 spot price is approximately US$2,000/oz Au and this reverts to long-term pricing of US$1,800/oz Au from Year 5 onwards. The long-term guidance silver price used in the economic analysis is US$23/oz. The Canadian:US dollar exchange rate forecast for the estimate is 0.74.

Artemis Gold 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 doré elsewhere in the world.

Other than for product sales, the largest contracts will cover items such as mine equipment, drill-and-blast operations, bulk commodities, and technical services. Artemis Gold is currently negotiating contract terms. Contracts will be re-negotiated and renewed as needed during the LOM.

25.13 Capital Cost Estimates

Capital cost estimates are at a minimum pre-feasibility study level. Phase 1 costs are assumed to be sunk costs.

Data supporting the estimate were obtained during Q4 2023 and Q1 2024.

The estimate was based on inputs developed by Artemis Gold, Knight Piésold, Moose Mountain, Lycopodium, and other third parties, using budget quotes and historical data, and is an amalgamation of engineering, material take offs and in-house benchmarks.

The estimate assumes that the Blackwater Gold Mine will be implemented using an EPCM approach.

Capital costs include mining, infrastructure (TSF, water management, mining infrastructure), process, and Owner costs.

The LOM total growth and sustaining capital cost estimate for Phases 2 and 3 is C$2,619 M.

25.14 Operating Cost Estimates

Operating cost estimates are at a minimum pre-feasibility study level.

Operating cost estimates were derived using first principal estimates based on typical operating data/standard industry practices, and inputs compiled from a variety of sources. Data supporting the estimate were obtained during Q4 2023 and Q1 2024.

Operating costs include mining, process, and general and administrative costs.

LOM plan operating cost estimates total C$20.03/t milled.

25.15 Economic Analysis

The after-tax NPV is C$3.25 B, using a 5% discount rate. The Blackwater Gold Mine is cashflow positive in Phase 1, and so there is no internal rate of return or payback period that is relevant to this Report.

The NPV is most sensitive to fluctuations in gold price (gold grade) and foreign exchange rate assumptions, and less sensitive to variations in capital and operating costs. Impacts of changes in the gold grade mirror the impact of changes in the gold price.

25.16 Risks and Opportunities

25.16.1 Risks

The major risks include:

- Changes to metal prices and exchange rate assumptions;
- Capital cost growth;
- Increases in operating costs;
- Changes to productivity assumptions;
- Changes to mining grade and dilution control assumptions;
- Presence of high-grade silver in the mill feed;
- Geotechnical and hydrogeological uncertainty;

- Climate uncertainty and associated water management needs;
- Changes as to assumptions on integration of mining operations and the TSF construction;

25.16.2 Opportunities

Project opportunities include:

- Mineral Resources exclusive of Mineral Reserves: a portion of the estimated Measured and Indicated Mineral Resources were not converted to Mineral Reserves. This material represents upside potential for extending the mine life once studies have been completed that support conversion to Mineral Reserves;
- Gold price: Mineral Reserves are based on a US$1,400/oz gold price. Using a higher gold price for pit design and a higher cut off represents upside potential for extending the mine life;
- Inferred Mineral Resources: there is upside potential for the estimates if mineralization that is currently classified as Inferred can be upgraded to higher-confidence Mineral Resource categories;
- Mineralization remains open at depth: particularly in the northwest of the deposit where an increasing trend in gold grade is noted. Additional drilling and evaluation may support estimation of Mineral Resources in this area;
- Exploration potential: the Blackwater land package remains largely under-explored, and warrants additional exploration efforts;
- Evaluation of alternative methods for transportation of waste material: the ex-pit haul route for waste material is expected to be relatively fixed for the LOM, opening up the possibility to perform hauling of waste material using alternative methods which may have the potential to significantly reduce operating costs and lower greenhouse gas emissions;
- Electrification of the hauling fleet: deployment of battery electric vehicles has the potential to significantly reduce operating costs and lower greenhouse gas emissions;
- Automation of hauling operations: the potential to automate hauling operations presents an opportunity to optimize production efficiencies and reduce operating costs;
- Process engineering initiatives: evaluation of alternative processing methodologies which may result in lower capital and operating costs for Phase 3.

25.17 Conclusions

Under the assumptions described in this Report, the proposed LOM plan is achievable, and the economic analysis supports declaration of Mineral Reserves.

26.0 RECOMMENDATIONS

26.1 Introduction

A single phase work program is proposed. All elements of the program can be conducted concurrently. The estimated budget to complete the program is approximately C$2.7 M.

26.2 Mining

The following recommendations are made with regard to improving future mine planning and operational performance during the mine operating phase:

- Complete additional core drilling for geotechnical purposes, including refining the boundaries between the broken zone and competent zone in the southeast and east pit design sectors, refining the pit slope design in the northwest pit design sector. Complete vertical sonic drill holes on the southeast side of the pit to further characterize the overburden materials for potentially steeper overburden slope angles and further assess consistency and suitability of overburden materials for use in dam construction. Incorporate updated open pit geotechnical interpretations into an updated set of open pit designs;
- Detail out opportunities to mine on larger benches in waste zones, using a larger equipment fleet;
- Run trade-off studies to investigate the merits of implementing technologies such as fleet automation, trolley assist, crushing and conveying systems, or other electrification initiatives for transporting waste rock from the open pit to the TSFs;
- Investigate other technologies to optimize material movement from the open pit to various mine infrastructures.

These field programs and desktop studies are estimated to cost approximately C$1.5 M.

26.3 Metallurgy and Process

Accelerating the development rate will mean that ore will be treated earlier in the production cycle and its behaviour could have a greater influence on cash flows.

It is recommended that ore samples from areas in the latter half of the production cycle be tested using the flow sheet to be used (gravity, CIL leach of the gravity tailings) and some samples from early in the mining cycle also be tested for comparison purposes. Some 20 samples should be tested with appropriate duplicates or repeats. Higher silver grade samples should be part of this testwork, because such grades may potentially impact the carbon loadings, leach rates, elution circuit size and electrowinning circuit size. The proposed work is estimated to require a budget of approximately C$140,000.

A second recommendation is to complete flotation, ultrafine grinding, and leach testwork in support of the Phase 3 circuit to potentially reduce capital costs. The proposed work is estimated to require a budget of approximately C$1 M.

26.4 Geotechnical and Hydrological

An acceleration to currently planned geotechnical site investigation work and updates to the LOM water balance model will be required to support implementation of the 2024 Expansion Study plan.

Recommendations for if/when the decision is made to proceed with the 2024 Expansion Study plan are as follows:

- The existing site investigation plan (Knight Piésold, 2024a) should be updated for the earlier construction of Main Dam D. Site investigation work will need to be completed in Year 1 and Year 2 to support detailed design work and permit condition fulfilment over that time to facilitate the proposed start of Main Dam D construction in Year 3. Site investigation costs in Year 1 and Year 2 and engineering capital costs in Year 2 for the detailed design of Main Dam D are carried in the Phase 2 growth capital costs;
- The LOM water balance model should be updated considering the ramp up to 15 Mt/a to further evaluate site water availability and how sources of available water will be managed to meet the processing and environmental flow demands. The update to the LOM water balance model was not carried in the capital cost estimate and is estimated to cost approximately C$100,000.

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