Blackwater Gold Project

NI 43-101 Technical Report on Updated Feasibility Study

British Columbia, Canada

Effective Date: September 10, 2021

Prepared for: Artemis Gold Inc.
595 Burrard Street, Suite 3083, Vancouver, British Columbia V7X 1L3

Prepared by: Ausenco Engineering Canada Inc.
855 Homer Street, Vancouver, British Columbia V6B 2W2

List of Qualified Persons:

Robin Kalanchey, P. Eng., Ausenco Engineering Canada • Sue Bird, P.Eng., Moose Mountain Technical Services • George Dermer, P. Eng., Moose Mountain Technical Services • Marc Schulte, P.Eng., Moose Mountain Technical Services • Daniel Fontaine, P.Eng., Knight Piésold Ltd., Vancouver • James Garner, P. Eng., Allnorth • Rolf Schmitt, P. Geo., ERM • John Dockrey, P. Geo., LORAX • John Alan Thomas, P.Eng., JAT Met Consult Ltd.

CERTIFICATE OF QUALIFIED PERSON
Robin Kalanchey, P. Eng.

I, Robin Kalanchey, P.Eng., certify that I am employed as Vice President, Transportation and Logistics with Ausenco Engineering Canada Inc. with an address of 855 Homer Street, Vancouver, BC, V6B 2W2. This certificate applies to the technical report titled, “Blackwater Gold Project, NI 43-101 Technical Report on Updated Feasibility Study,” that has an effective date of September 10, 2021 (the “Technical Report”).

I graduated from University of British Columbia, in Vancouver, BC, in 1996 with a Bachelor of Applied Science degree in Metals and Materials Engineering. I am a Professional Engineer in good standing registered with the Association of Professional Engineers and Geoscientists of Alberta, (member #61986), and with the Engineers and Geoscientists British Columbia, (registration #53123).

I have practiced my profession for 25 years since graduation. I have been directly involved in the process development, design, construction, operation and evaluation of mining properties, mineral processing plants and metallurgical operations for the winning of base and precious metals, including gold and silver, in numerous countries and jurisdictions, including British Columbia, Canada.

I have read the definition of “Qualified Person” set out in the National Instrument 43-101 Standards of Disclosure for Mineral Projects (“NI 43-101” or the “Instrument”) and certify that by virtue of my education, affiliation to a professional association and past relevant work experience, I fulfill the requirements to be a “Qualified Person” for the purposes of NI 43-101.

I visited the Blackwater Project site on July 6, 2021 for a visit duration of 1 day. I am responsible for sections 1.1, 1.2,1.3, 1.5.1, 1.5.3, 1.18, 1.19, 1.19.1, 1.26, 1.27, 1.29, 1.30, 1.31, 2.1, 2.2, 2.3, 2.4, 2.4.1, 2.5, 2.6, 2.7, 2.8, 3.1, 5.1, 5.3, 5.4, 5.7, 6, 17, 18.1, 18.7, 18.8, 18.9.1, 18.9.2, 18.9.3, 18.9.4, 18.9.5, 18.11, 18.12, 18.14, 21.1.1, 21.1.2, 21.1.3, 21.1.4, 21.1.5, 21.1.6, 21.1.7, 21.1.8, 21.1.9, 21.1.10, 21.1.12, 21.1.14, 21.1.17, 21.1.18, 21.2.1, 21.2.3, 21.2.4, 24.1.1, 24.1.2, 25.10, 25.11, 25.19, 25.20, 25.22, 25.23, 27 of the Technical Report.

I am independent of Artemis Gold Inc. as independence is defined in Section 1.5 of NI 43-101. I have had no previous involvement with Blackwater Project.

I have read NI 43-101 and the sections of the Technical Report for which I am responsible have been prepared in compliance with that Instrument. As of the effective date of the Technical Report, to the best of my knowledge, information and belief, the sections of the Technical Report for which I am responsible contain all scientific and technical information that is required to be disclosed to make those sections of the Technical Report not misleading.

Dated: October 25, 2021

“Signed and sealed”

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Robin Kalanchey, P. Eng.

CERTIFICATE OF QUALIFIED PERSON

Sue Bird, P.Eng.

I, Sue Bird, P.Eng. am employed as a Geological Engineer 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 Project, NI 43-101 Technical Report on Updated Feasibility Study," that has an effective date of September 10, 2021 (the "Technical Report").

I graduated with a Geologic Engineering degree (B.Sc.) from Queen's University in 1989 and an M.Sc. in Mining from Queen's University in 1993. I am a member of the self-regulating Association of Professional Engineers and Geoscientists of British Columbia (License #25007). 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, New Mexico, Spanish Mountain, BC, Marban, QB, Garrison, ON as well as numerous due diligence gold projects in the southern USA done confidentially for various clients.

I have read the definition of "Qualified Person" set out in the National Instrument 43-101 Standards of Disclosure for Mineral Projects ("NI 43-101" or the "Instrument") and certify that by virtue of my education, affiliation to a professional association and past relevant work experience, I fulfill the requirements to be a "Qualified Person" for the purposes of NI 43-101.

I visited the property on July 15, 2020, for a visit duration for 1 day. I am responsible for Sections 1.4, 1.6 through 1.14, 2.4.2, 3.2, 4.1 through 4.6, 4.9, 7 through 12, 14, 23, 25.1, 25.2, 25.3, 25.4, 25.5, 25.7, 26.1 and 26.2 of the Technical Report.

I am independent of Artemis Gold Inc. as independence is described by Section 1.5 of NI 43–101. I previously co-authored the report titled "Blackwater Gold Project British Columbia, NI 43-101 Technical Report, on Pre-Feasibility Study" that has an effective date of August 26, 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: 25 October 2021

"Signed and sealed"

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Sue Bird, P. Eng.

CERTIFICATE OF QUALIFIED PERSON

George Dermer, P. Eng.

I, George Dermer, P.Eng., certify that I am employed as a Mining Engineer 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 Project, NI 43-101 Technical Report on Updated Feasibility Study," that has an effective date of September 10, 2021 (the "Technical Report").

I graduated from the University of British Columbia in 2005 with a Bachelor of Applied Science in Mining Engineering. I am a member of the self-regulating Association of Professional Engineers and Geoscientists of the Province of British Columbia (License #33924). I have practiced my profession for 16 years since graduation. I have been directly involved in Mineral Resource Estimates and Mineral Reserve estimates and mine planning for precious metals, base metals, and coal mining projects, for both project engineering studies and mine operations. I have also worked on mining financial models for the purpose of operational budgets and mergers and acquisitions.

I have read the definition of "Qualified Person" set out in the National Instrument 43-101 Standards of Disclosure for Mineral Projects ("NI 43-101" or the "Instrument") and certify that by virtue of my education, affiliation to a professional association and past relevant work experience, I fulfill the requirements to be a "Qualified Person" for the purposes of NI 43-101.

I have not visited the Backwater Gold Project.

I am responsible for 1.28, 1.29, 1.30, 3.3, 19, 22, 24.1, 25.20, 25.21 of the Technical Report.

I am independent of Artemis Gold Inc as independence is defined in Section 1.5 of NI 43-101. I have had no previous involvement with the Blackwater Gold Project.

I have read NI 43-101 and the sections of the Technical Report for which I am responsible have been prepared in compliance with that Instrument. As of the effective date of the Technical Report, to the best of my knowledge, information and belief, the sections of the Technical Report for which I am responsible contain all scientific and technical information that is required to be disclosed to make those sections of the Technical Report not misleading.

Dated: October 25, 2021

"Signed and sealed"

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George Dermer, P. Eng.

CERTIFICATE OF QUALIFIED PERSON

Marc Schulte, P.Eng.

I, Marc Schulte, P.Eng., am employed as a Mining Engineer 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 Project, NI 43-101 Technical Report on Updated Feasibility Study," that has an effective date of September 10, 2021 (the "Technical Report").

I am a member of the self-regulating Association of Professional Engineers, Geologist and Geophysicists of Alberta (License #71051). 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 a total of 18 years since my graduation from university. I have worked on Mineral Reserve estimates and mine planning for precious metals, base metals and coal mining projects, for both project engineering studies and mine operations.

I have read the definition of "Qualified Person" set out in the National Instrument 43-101 Standards of Disclosure for Mineral Projects ("NI 43-101" or the "Instrument") and certify that by virtue of my education, affiliation to a professional association and past relevant work experience, I fulfill the requirements to be a "Qualified Person" for the purpose of NI 43-101.

I visited the Blackwater Gold Project on July 14, 2020, for a visit duration of one day. I am responsible for Sections 1.15, 1.17, 1.26, 1.27, 1.29, 1.30, 1.31, 2.4.4, 15, 16, 21.1.11, 21.2.2, 24.1.1, 24.1.2, 25.8, 25.9, 25.19, 25.20, 26.1 and 26.4 of the Technical Report.

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

I have previously co-authored the following report on the Blackwater Gold Project:

- Bird, S., Fontaine, D., Meintjes, T., Schulte, M., and Thomas, J., 2020: Blackwater Gold Project British Columbia NI43-101 Technical Report on Pre-Feasibility Study, effective date August 26, 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: October 25, 2021

"Signed and sealed"

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Marc Schulte, P. Eng.

CERTIFICATE OF QUALIFIED PERSON

Daniel Fontaine, P. Eng.

I, Daniel Fontaine, P.Eng., certify that I am employed as a Specialist Engineer and Associate with Knight Piésold Ltd. (KP), with an office address of 1400-750 West Pender Street, Vancouver, British Columbia, V6C 2T8. This certificate applies to the technical report titled, "Blackwater Gold Project, NI 43-101 Technical Report on Updated Feasibility Study," that has an effective date of September 10, 2021 (the "Technical Report").

I graduated from McGill University in Montreal, Quebec, in 2006 with a bachelor's degree in Civil Engineering. I am a registered Professional Engineer of Engineers and Geoscientists of British Columbia (License #36208). I have practiced my profession for 15 years. 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.

I have read the definition of "Qualified Person" set out in the National Instrument 43-101 Standards of Disclosure for Mineral Projects ("NI 43-101" or the "Instrument") and certify that by virtue of my education, affiliation to a professional association and past relevant work experience, I fulfill the requirements to be a "Qualified Person" for the purposes of NI43-101.

I visited the Blackwater Gold Project site most recently on November 3-8, 2020, as well as on several other occasions previously between 2011 and 2019. I am responsible for Sub-sections 1.5.2, 1.5.4, 1.5.5, 1.21, 1.22, 2.4.3, 5.2, 5.5, 5.6, 18.3, 18.4, 18.5, 18.6, 21.1.13, 25.13, 25.14, 26.1 and 26.5 of the Technical Report.

I am independent of Artemis Gold Inc. as independence is defined in Section 1.5 of NI 43-101. I have been involved with the Blackwater Gold Project continuously since 2011, performing and overseeing work related to climate and hydrological baseline studies, geotechnical and hydrogeological site investigations, site characterization, tailings and water management studies, and the design of the project components related to the sections of the Technical Report that I am responsible for preparing. I was a co-author of the report, titled "Blackwater Gold Project British Columbia, NI 43-101 Technical Report on Pre-Feasibility Study", prepared for Artemis Gold Inc. that has an effective date of August 26, 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: October 25, 2021

"Signed and sealed"

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Daniel Fontaine, P. Eng.

CERTIFICATE OF QUALIFIED PERSON
James Garner, P. Eng.

I, James Garner, P. Eng., certify that I am employed as a Civil Engineering Group Lead with Allnorth Consultants Limited, with an office address of 100, 275 Lansdowne St, Kamloops, BC, V2C 1E9. This certificate applies to the technical report titled "Blackwater Gold Project, NI 43-101 Technical Report on Updated Feasibility Study," that has an effective date of September 10, 2021 (the "Technical Report").

I graduated from the University of British Columbia in 2015 with a degree in Applied Science in Civil Engineering. I am a member of Engineers and Geoscientists British Columbia (License #49570). I have practiced my profession for six years since graduation. I have been directly involved in the civil engineering design and inspection of infrastructure for mining projects during this time period.

I have read the definition of "Qualified Person" set out in the National Instrument 43-101 Standards of Disclosure for Mineral Projects ("NI 43-101" or the "Instrument") and certify that by virtue of my education, affiliation to a professional association and past relevant work experience, I fulfill the requirements to be a "Qualified Person" for purposes of NI 43-101.

I have not visited the Blackwater Gold project site. I am responsible for 1.19, 1.26, 1.27, 18.2, 18.9.6, 18.10, 18.13, 21.1.15, 24.1.1, 24.1.2 of the Technical Report.

I am independent of Artemis Gold Inc. as independence is defined in Section 1.5 of NI 43-101. I have had no previous involvement with the Blackwater Gold Project.

I have read NI 43-101 and the sections of the Technical Report for which I am responsible have been prepared in compliance with that Instrument. As of the effective date of the Technical Report, to the best of my knowledge, information and belief, the sections of the Technical Report for which I am responsible contain all scientific and technical information that is required to be disclosed to make those sections of the Technical Report not misleading.

Dated: October 25, 2021

"Signed and sealed"

____

James Garner, P. Eng.

CERTIFICATE OF QUALIFIED PERSON

Rolf Schmitt, P Geo.

I, Rolf Schmitt, P Geo. certify I am employed as a Technical Director with ERM Consultants Canada Ltd., with an office address of 1111 West Hastings St. Vancouver, BC. This certificate applies to the Technical Report titled "Blackwater Gold Project, NI-43-101 on Updated Feasibility Study," that has an effective date of September 10, 2021 (the "Technical Report").

I graduated from the University of British Columbia with a Bachelor of Geology (Hons) in 1977, University of British Columbia with a Master of Science (Regional Resource Planning) in 1985, and University of Ottawa with a Master of Science Geology (1993), with specialization in exploration geochemistry. I am a Professional Geologist, registered with Engineers and Geoscientists of British Columbia (member #121446). I have practiced my profession continuously since 1977 and have been involved in: mineral exploration for porphyry copper-gold and VMS deposits in British Columbia, exploration geochemical surveys across Canada, regional land use policy and planning and mining regulations in British Columbia, in environmental assessment and permitting of mines in British Columbia, and ESG due diligence of base and precious metal projects throughout North and South America.

I have read the definition of "Qualified Person" set out in the National Instrument 43-101 Standards of Disclosure for Mineral Projects ("NI 43-101" or the "Instrument") and certify that by virtue of my education, affiliation to a professional association and past relevant work experience, I fulfill the requirements to be a "Qualified Person" for the purpose of NI 43-101.

I have not visited the Blackwater Gold Project. I am responsible for Sections 1.23, 1.24, 1.25, 4.7, 4.8, 20, 25.15, 25.16, 25.17 and 25.18 of the Technical Report.

I am independent of Artemis Gold Inc. as independence is defined in Section 1.5 of NI 43-101. I have been involved with the Blackwater Gold Project since 2016 and have worked on the permitting and reclamation and closure planning and provided senior technical review (Environmental Assessment Certificate #M19-01, 2019), and worked on the Special Use Permit (Mine Access Road), senior technical review and writing of sections for the joint *Mines Act / Environmental Management Act* permits Application.

I have read NI 43-101 and the sections of the Technical Report for which I am responsible have been prepared in compliance with that Instrument. As of the effective date of the Technical Report, to the best of my knowledge, information and belief, the sections of the Technical Report for which I am responsible contain all scientific and technical information that is required to be disclosed to make those sections of the Technical Report not misleading.

Dated: October 25, 2021

"Signed and sealed"

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Rolf Schmitt, P. Geo.

CERTIFICATE OF QUALIFIED PERSON

John Alan Thomas, P. Eng.

I, John Alan Thomas, P. Eng., certify that I am employed as president of JAT Metconsult Ltd. with an office address of 5766 Goldenrod Crescent, Delta, BC V4L 2G6, Canada. This certificate applies to the technical report titled, "Blackwater Gold Project NI 43-101 Technical Report Updated Feasibility Study," that has an effective date of September 10, 2021, (the "Technical Report").

I graduated from Manchester University with a B.Sc. M.Sc. and Ph.D. in in chemical engineering, awarded in 1969, 1972 and 1973, respectively. I am a professional Engineer of The Association of Professional Engineers of the Province of British Columbia, License number 34665. I have practiced my profession for forty-eight years. I have been directly involved in the development, engineering and operation of ore treatment processes for a wide variety of metals, with the main involvement in gold ore processing. I have been involved with gold projects for Bolivar Gold Corp. as vice president operations, (2003 – 2005), Infinito Gold, as vice president operations, (2005 – 2012), Edgewater Exploration, as chief operating officer, (2012 – 2013, Atlantic Gold, as vice president projects, (2015 – 2017) and Almaden Minerals, as vice president project development, (2018 – 2020). I have prepared certain sections of the technical report Minera IRL NI 43-101 Ollachea Gold Project Technical Report (Preliminary Economic Assessment, 9 August 2021).

I have read the definition of "Qualified Person" set out in the National Instrument 43-101 Standards of Disclosure for Mineral Projects ("NI 43-101" or the "Instrument") and certify that by virtue of my education, affiliation to a professional association and past relevant work experience, I fulfill the requirements to be a "Qualified Person" for the purposes of NI 43-101.

I visited the Blackwater site on May 7th. 2020 for a duration of one day. I am responsible for Sections 1.3, 1.16, 1.17, 1.19, 1.26, 1.31, 2.4.5, 13, 18.2.1, 21.1.16, 21.2, 21.2.5, 24.1.1, 24.1.2, 25.6, 25.19, 25.20, 26.1 and 26.3 of the Technical Report.

I am independent of Artemis Gold Inc.as independence is defined in Section 1.5 of NI 43-101. I have previously prepared certain sections of the technical report titled "Blackwater Gold Project British Columbia, NI 43-101 Technical Report on Pre-Feasibility Study" that has an effective date of August 26, 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: October 25, 2021

"Signed and sealed"

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John Alan Thomas, P. Eng.

CERTIFICATE OF QUALIFIED PERSON

John Dockrey, P. Geo.

I, John Dockrey, P. Geo., certify that I am employed as a Senior Geochemist with Lorax Environmental Services Ltd., with an office address of 2289 Burrard St., Vancouver, BC, V6J 3H9. This certificate applies to the technical report titled "Blackwater Gold Project, NI43-101 Technical Report on Updated Feasibility Study," that has an effective date of September 10, 2021 (the "Technical Report").

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 am a *registered member with* Engineers and Geoscientists BC (Permit to Practice #1001840). I have practiced my profession for *11 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.

I have read the definition of "Qualified Person" set out in the National Instrument 43-101- *Standards of Disclosure for Mineral Projects* ("NI 43-101" or the "Instrument") and certify that by virtue of my education, affiliation to a professional association and past relevant work experience, I fulfill the requirements to be a "Qualified Person" for the purposes of NI 43-101.

I have not visited the Blackwater Project.

I am responsible for the Waste Characterization sections *1.20 and 25.12* of the Technical Report.

I am independent of *Artemis Gold Inc.* as independence is defined in Section 1.5 of NI 43-101. I have been involved with the *Blackwater Project in support of geochemical assessments since 2016*.

I have read NI 43-101 and the sections of the Technical Report for which I am responsible have been prepared in compliance with that Instrument. As of the effective date of the Technical Report, to the best of my knowledge, information and belief, the sections of the Technical Report for which I am responsible contain all scientific and technical information that is required to be disclosed to make those sections of the Technical Report not misleading.

Dated: October 25, 2021

"Signed and sealed"

____

John Dockrey, P. Geo.

Important Notice

This report was prepared as National Instrument 43-101 Technical Report for Artemis *(*Artemis*)* by Ausenco Engineering Canada Inc. (Ausenco), Moose Mountain Technical Services, Knight Piésold Ltd., Allnorth Consultants Limited, LORAX Environmental Services Ltd, ERM Consultants Canada Ltd, and JAT Met Consult Ltd., collectively the Report Authors. The quality of information, conclusions, and estimates contained herein is consistent with the level of effort involved in the Report Authors' services, based on i) information available at the time of preparation, ii) data supplied by outside sources, and iii) the assumptions, conditions, and qualifications set forth in this report. This report is intended for use by Artemis subject to terms and conditions of its contracts with each of the Report Authors. Except for the purposed legislated under Canadian provincial and territorial securities law, any other uses of this report by any third party is at that party's sole risk.

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1 SUMMARY

1.1 Introduction

Ausenco Engineering Canada Inc. (Ausenco), Moose Mountain Technical Services (MMTS), Knight Piésold Ltd. (KP), Allnorth (Allnorth), LORAX Environmental Services Limited (LORAX), ERM Consultants Canada Ltd. (ERM), and a JAT Met Consult Ltd. (JAT Metco) have prepared a technical report (the Report) for Artemis Gold Inc. (Artemis) on a Feasibility Study (2021 FS) evaluation of the Blackwater Gold Project (the Project), located in British Columbia, Canada. BW Gold Ltd. (BW Gold) is the holding entity for the mineral claims. BW Gold is a wholly owned subsidiary of Artemis. For the purposes of this Report, Artemis is used interchangeably for the subsidiary and parent companies.

The six contiguous claim blocks held by Artemis, specifically the Blackwater, Capoose, Auro, Key, Parlane and RJK claim blocks, are referred to as the Property for the purposes of this Report.

The Project refers to exploration and development activity related to the Blackwater deposit which is contained within the Blackwater claim block.

1.2 Key Findings

The key findings of the 2021 FS are:

- At the base case cut-off grade of a 0.20 g/t gold equivalent (AuEq), 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 million AuEq ounces.
- Of the total Measured and Indicated Mineral Resources, 75% are in the Measured category.
- Proven and Probable Mineral Reserves total 334.3 Mt at 0.75 g/t Au and 5.8 g/t Ag (0.78 g/t AuEq).
- Ore processing commences with a nominal milling rate of 16,500 tpd (6.0 Mtpa, Phase 1). The ore processing facilities will be expanded to achieve 33,000 tpd (12 Mtpa, Phase 2) starting in Year 5 with a final expansion to achieve 55,000 tpd (20 Mtpa, Phase 3) starting in Year 10 of operation. Phase 4 will commence in year 17, when mining ends and treatment of stockpiled ore commences at a throughput of 55,000 tpd (20 Mtpa, Phase 4).
  - The Phase 1 crushing circuit utilizes a gyratory primary crusher with the balance of the equipment specified to allow for minimum modifications required to expand to Phase 2 capacity.
  - Single stage 14 MW ball milling, gravity concentration, leaching, and carbon-in-leach (CIL) will be used for recovering gold and silver
  - The average gold feed grade will be 1.62 g/t Au over the first five years.
  - The initial capital cost estimate is C$645 million, including an overall 12.2% contingency applied. Expansion capital is C$347 million for Phase 2 and C$374 million for Phase 3. No additional expansion capital is required for Phase 4.

- The life of mine (LOM) operating costs are estimated at C$17.96/t of ore milled. Total LOM all-in sustaining cash costs are estimated at C$850/oz Au recovered.
- For the base financial case:
    - After-tax net present value (NPV) at a 5% discount rate is estimated at C$2,151 million.
    - After-tax internal rate of return is 32%.
    - After-tax initial capital payback is estimated at 2.3 years.

1.3 Terms of Reference

The Report supports disclosures in Artemis' press release entitled "Artemis Announces Feasibility Study for Blackwater Project" dated 13 September 2021.

All currencies are expressed in Canadian dollars (C$) unless otherwise stated. Years expressed in this summary are for illustrative purposes only, as the decision to implement production is at the discretion of Artemis and permits to support operation still have to be obtained. Mineral Resources and Mineral Reserves are estimated using the 2019 edition of the Canadian Institute of Mining, Metallurgy and Exploration (CIM) estimation of Mineral Resources and Mineral Reserves Best Practice Guidelines (2019 CIM Best Practice Guidelines) and are reported using the 2014 CIM Definition Standards for Mineral Resources and Mineral Reserves (2014 CIM Definition Standards).

For the purposes of the Report, two terms are used for the mine production; LOM which refers to the life of mine including the pre-production period; the operational period refers to the mine life excluding the pre-production duration.

1.4 Project Description and Location

1.4.1 Location

The Blackwater Project is located in central British Columbia (BC), approximately 112 km, southwest of Vanderhoof and 446 km northeast of Vancouver (Figure 1-1). The Project site is readily accessible by forest service and mine roads. Driving time from Vanderhoof to the property is about 2.5 hours. Helicopter access is available from bases in Vanderhoof, Quesnel, or Prince George.

Figure 1-1: **Blackwater Project Location Map**

Note: prepared by Artemis, 2020

1.4.2 Mineral Tenure

Artemis holds a 100% recorded interest in 329 mineral claims coverage an area of 148,902 ha distributed among the Blackwater, Capoose, Auro, Key, Parlane and RJK claim blocks. The Blackwater claim block comprises 76 mineral cell claims totalling 30,791ha. All claims are 100% held in the name of BW Gold and expire in 2022. There are no other parties with beneficial interests in these mineral rights. None of the Blackwater cell claims are known to overlap any legacy or Crown granted mineral claims, or no-staking reserves.

1.4.3 Surface Rights

A review of surface rights in the vicinity of the Property was undertaken in September 2020. The majority of the Blackwater mineral claims are located on Crown lands. The review identified an overlapping private parcel, land reserves/notations, a transfer of administration/control area, grazing tenures, forest recreation sites, forest tenures, trap lines, guide outfitters, and an ungulate winter range. Sixteen (16) of the Capoose claims have minor portions overlapping onto Entiako Provincial Park.

A review of surface rights in the vicinity of proposed electrical transmission lines, water pipeline, and access roads (Linear Infrastructure) was undertaken in December 2013 and in September 2020. This review identified private parcels; a Land Act license, rights of way, reserves/notations and a transfer of administration/control area; grazing tenures; forest tenures;

forest recreation sites; traplines; guide outfitter areas; a wildlife management area; an agriculture land reserve; and third-party mineral tenures overlapping or in close proximity to the proposed Linear Infrastructure route.

1.4.4 Royalties and Encumbrances

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

- A 1.5% NSR royalty is payable on mineral claim 515809 (Dave Claim). The claim covers a portion of the Blackwater deposit.
- A 1% NSR royalty is payable on mineral claim 515810 (Jarrit Claim). The claim covers a portion of the Blackwater deposit.
- The current agreement would allow Artemis. to purchase two-thirds of three Blackwater Claims (637203, 637205, and 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.

Artemis's 100% interest in the property, assets and rights related to the Blackwater Project and six contiguous claim blocks (Blackwater, Capoose, Auro, Key, Parlane and RJK) is subject to the following consideration:

- A secured gold stream participation in favor of New Gold, whereby New Gold will purchase 8.0% of the refined gold produced from the Project. Once 279,908 ounces of refined gold have been delivered to New Gold, the gold stream will reduce to 4.0%.

New Gold will make payments for the gold purchased equal to 35% of the US dollar gold price quoted by the London Bullion Market Association (LBMA) two days prior to delivery. In the event that commercial production at Blackwater is not achieved by the 7th, 8th, or 9th anniversary of closing, being August 21, 2020 (i.e., 7th anniversary is August 21, 2027, 8th anniversary is August 21, 2028, and 9th anniversary is August 21, 2029), New Gold will be entitled to receive additional cash payments of C$28 million on each of those dates.

New Gold maintains a security interest over the Project in connection with the gold stream agreement.

1.5 Accessibility, Climate, Local Resources, Infrastructure and Physiography

1.5.1 Accessibility

The Blackwater site will be accessed via the Kluskus Forest Service Road (FSR). Artemis will likely become the primary operator and user of the FSR by the time the Project is constructed, considering that reduced logging operations are anticipated in the area at that time, and will be responsible for primary maintenance. Artemis will upgrade part of the FSR to meet future year-round operational Project needs.

A new 16 km long mine access road will replace the existing exploration access road to the site. Some sections of the planned water supply pipeline, fibre-optic cable and power transmission line will parallel this road. The road will be used for heavy traffic during mine operation and has been designed for year-round, all-weather access.

1.5.2 Climate

The climate is sub-continental, characterized by brief warm summers and long cold winters resulting from the influence of cold arctic air. The 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 to 440 mm.

The weather is not expected to present any unusual difficulties for year-round mining operations.

1.5.3 Local Resources

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.

There is no grid-connected power in the direct vicinity of the Project. The main BC Hydro 500 kV transmission lines supplying western BC are approximately 100 km to the north. Several interconnection points from the 500 kV lines to existing 230 kV substations and transmission lines are possible in an area between Fraser Lake and Vanderhoof. Power for the current Blackwater exploration camp is provided by generators. The deposit is located on the north slope of Mt. Davidson, and the proposed Project infrastructure including the mill facilities, waste stockpiles and tailings storage will be sited predominantly in the Davidson Creek watershed. Precipitation run-off and groundwater from pit dewatering will be the primary water sources for mineral processing. A groundwater well field will supply potable water for the camp.

1.5.4 Physiography

The elevation of the Project ranges 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. Bedrock outcrops are limited and most of the area is covered with thick glacial deposits of 2 m or more, except at high elevations near the summit of Mt. Davidson and several localized areas lower in elevation.

The Nazko Upland sub-region is the primary biogeoclimatic region. 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 Engelmann spruce are dominant above 1,500 m. The pine beetle epidemic infested almost all the lodgepole pine forests within this sub-region. The Nazko Upland sub-region also contains an extensive network of lakes, rivers, and wetland complexes. Atmospheric heating of these water bodies can result in convective activity and sporadic summer showers.

1.5.5 Regional Tectonics and Seismicity

The Project site is situated in central BC – an area with historically low levels of seismic activity. While neighboring regions experience higher seismicity, the Project site is too distant from those areas to raise any significant seismic hazards. First, while the Queen Charlotte-Fairweather fault system and the Alaskan panhandle experience higher seismicity, those levels drop off rapidly when moving away from the coast and northbound. Similarly, while the Cascadia and Explorer subduction zones in southwestern BC have potential for large magnitude earthquakes, those zones are also too distant to pose seismicity concerns.

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 completed the Project acquisition on 21 August 2020. Artemis 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 drillholes totalling 324,839 m were drilled in the block model area between 2009 and January 2013 by Richfield and New Gold. Drilling completed between 1981 and the end of 2006 consists of 81 holes totaling 7,633 m. This legacy drilling is not used in resource estimation.

The exploration drilling carried out since 2009 was predominantly HQ (63.5 mm core diameter) drill core except where a reduction to NQ diameter (47.6 mm) was required to attain target depths. Drilling for metallurgical used PQ diameter (85 mm) core. Some of the condemnation drilling was undertaken using reverse circulation (RC) methods.

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 metre. 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 drillhole 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 drillholes, 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.

BW Gold drilled 561 Reverse Circulation (RC) holes for a total of 33,216 m during a Pre-Production Grade Control Program during the winter 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 drillhole database is supported by over 43,000 QA/QC check assays.

Eco Tech Stewart Group Laboratories (Eco Tech) in Kamloops and ALS Mineral Laboratories (ALS) in Vancouver, Vanderhoof, Terrace, Reno, and Elko were used for sample preparation. Eco Tech in Kamloops and ALS in North Vancouver were used as the primary assay laboratories. Both primary laboratories were accredited and are independent of New Gold and Artemis.

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 the laboratories but were generally similar.

Metallurgical samples were selected from the designated metallurgical holes and samples from numerous resource holes across the deposit. The samples were collected and dispatched 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.

Specify gravity measurements were made in 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.

1.12 Data Verification

Data verification programs were completed by Sue Bird, Principal of MMTS. The QP reviewed the sample database for interval errors and missing sample intervals.

A site visit was undertaken by the QP on 14 July 2020 to review the site location, core storage, core, geology and protocols. The QP concluded that the QA/QC with respect to the results received for the drill programs between 2009 and 2012 were acceptable. The protocols were reviewed and were well documented. The drillhole database was adequate to support the geological interpretations and Mineral Resource estimate in this Report.

1.13 Mineral Resource Estimates

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 drillholes. The drillhole database is supported by analysis of over 43,000 quality assurance/quality control (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 drillhole 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 40 degrees.

1.14 Mineral Resource Statement

The Qualified Person for the resource estimate is Sue Bird, P. Eng. of MMTS. 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 1-1are 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: May 5, 2020 (base case is highlighted)

Classification			In-situ Grades			In-situ Contained Metal		
	Cut-off (g/t AuEq)	Tonnage (kt)	AuEq (g/t)	Au (g/t)	Ag (g/t)	AuEq (koz)	Au (koz)	Ag (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
	0.70	3,328	1.20	0.97	36.9	128	104	3,951

Notes:

1. The Mineral Resource estimate was prepared by Sue Bird, P.Eng., the Qualified Person for the estimate and employee of MMTS. The estimate has an effective date of May 5, 2020.
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.

1.15 Mineral Reserve Estimates

Proven and Probable Mineral Reserves are modified from the Measured and Indicated Mineral Resources and are summarized in Table 1-2. Inferred Mineral Resources are set to waste. Mineral Reserves are supported by the 2021 FS mine plan.

Table 1-2: Proven and Probable Mineral Reserves

	Run of Mine (Mt)	Gold Grade (Au, g/t)	Contained Metal (Au, Moz.)	Silver Grade (Ag, g/t)	Contained Metal (Ag, Moz.)	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 Reserve	**334.3**	**0.75**	**8.0**	**5.8**	**62.2**	**0.78**

Notes:
1. The Mineral Reserve estimates were prepared by Marc Schulte, P.Eng. (who is also the independent Qualified Person for these Mineral Reserve estimates), reported using the 2014 CIM Definition Standards, and have an effective date of September 10, 2021.
2. Mineral Reserves are based on the 2021 Feasibility Study life of mine plan.
3. Mineral Reserves are mined tonnes and grade; the reference point is the mill feed at the primary crusher and includes consideration for operational modifying factors such as loss and dilution.
4. Mineral Reserves are reported at an 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.
5. 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.
6. The AuEq values were calculated using the same parameters as NSR listed above, resulting in the following equation: AuEq = Au g/t + (Ag g/t x 0.006).
7. Numbers have been rounded as required by reporting guidelines.

Open pits are based on the results of Pseudoflow sensitivity analysis, and then designed into detailed pit phases to develop reserves for production scheduling.

Factors that may affect the Mineral Reserves estimates include metal prices and foreign exchange rate, changes in interpretations of mineralization geometry and continuity of mineralization zones, geotechnical and hydrogeological assumptions, ability of the mining operation to meet the annual production rate, process plant and mining recoveries, the ability to meet and maintain permitting and environmental license conditions, and the ability to maintain the social license to operate.

1.16 Metallurgy and Processing

The process flowsheet was designed using historical testwork as a basis to broadly define the process (whole ore leaching) and more recent testwork carried out in 2019 and 2020 to design the process in detail as it was an integrated program and included gravity concentration in all recovery testwork, something that was not used in previous testwork.

The more recent metallurgical program, completed in 2019, was completed with the primary objective of confirming and optimizing the flowsheet and design criteria using a combination of new testwork, results from the historical and previous testwork programs, and trade-off studies completed since the 2013 FS. Drill core from site was sent to Base Metallurgical Laboratories Ltd. (BaseMet) in Kamloops, BC for testwork that included core splitting, sample preparation, interval assaying, mineralogy, gravity concentration, cyanide leach and cyanide destruction. Some additional work was carried out in 2020 to provide certain design parameters.

The test program included three larger composites for optimization testwork and 48 samples covering the deposit to establish the variability of the ore to the chosen flow sheet.

The mineralogy indicated that the sulphur content is mainly associated with pyrite, pyrrhotite and sphalerite.

The comminution testwork included semi-autogenous grind (SAG) mill comminution (SMC) on the new drill core, Bond rod mill work index (RWi), Bond ball mill work index (BWi) and abrasion index (Ai) tests. The results indicate the material is hard with results ranging from 11.8 to 24.6 kWh/t and the 75th percentile of the samples tested was 21.1 kWh/t for the variability samples. The high degree of variability in ore hardness suggested difficulties in designing and operating a SAG mill/ball mill combination and, as a result, a three-stage crushing and ball milling was selected for the 2021 FS.

Tests showed that gravity concentration was effective in recovering gold and should be incorporated in the flow sheet. All the more recent testwork included gravity recovery of gold prior to leaching, and this resulted in significantly higher overall recoveries compared with historical testwork.

The results obtained from three composites and 48 variability samples show that an overall gold recovery of 93% and a silver recovery of approximately 65% can be obtained with gravity gold recovery of 34.2%. A grind size of 80% passing 150 µm was shown to be adequate.

Sodium cyanide and lime consumptions are expected to be 0.6 kg/t and 1 kg/t respectively.

Tests on carbon loading of the gold / silver leach solution indicated a combined (Au+ Ag) of 6,000 g/t.

Cyanide destruction testwork showed air / SO_2 to be effective, with a 4:1 SO_2: weakly acid dissociable (WAD) cyanide ratio. Hydrogen peroxide was also shown to be effective on clear solution.

1.17 Mining Methods

Mining is based on conventional open pit methods suited for the Project location and local site requirements. Open pit operations will commence 15-18 months prior to mill start-up and are anticipated to run for 17 years. Following mining operations, stockpiled low-grade material will be processed for an additional five years, resulting in a total life-of-mine (LOM) of 22 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 pre-production phase will target suitable overburden and waste rock for construction whilst exposing near-surface, high-grade material. The first phase will target higher-grade, lower-strip-ratio ore, providing mill feed over the initial years of the Project. The remaining phases will expand the pit to the north targeting progressively deeper ore.

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

Mill feed targets are 6.0 Mtpa over the first five years of operation, increasing to 12 Mtpa for the next five years of operation, and finally to 20 Mtpa until the end of the planned mine life.

During the pre-stripping phase of mine operations, all ore mined in the pit will be stockpiled. Throughout the life of operations, all ore grading between C$13/t and C$16.50/t NSR will be stockpiled. Cut-off grade optimization on the mine production schedule will also send ore above C$16.50/t NSR to an ore stockpile in certain planned periods. The stockpiled Mineral Reserves are planned to be re-handled back to the crusher once the pit is exhausted.

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.

Initially, mining will be undertaken using 400 t class hydraulic shovels and 190 t payload class haul trucks. As production requirements increase, the load and haul fleet will be expanded with 600 t class hydraulic shovels and 230 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 mine equipment fleet is planned to be purchased via various lease arrangements.

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 to the 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.

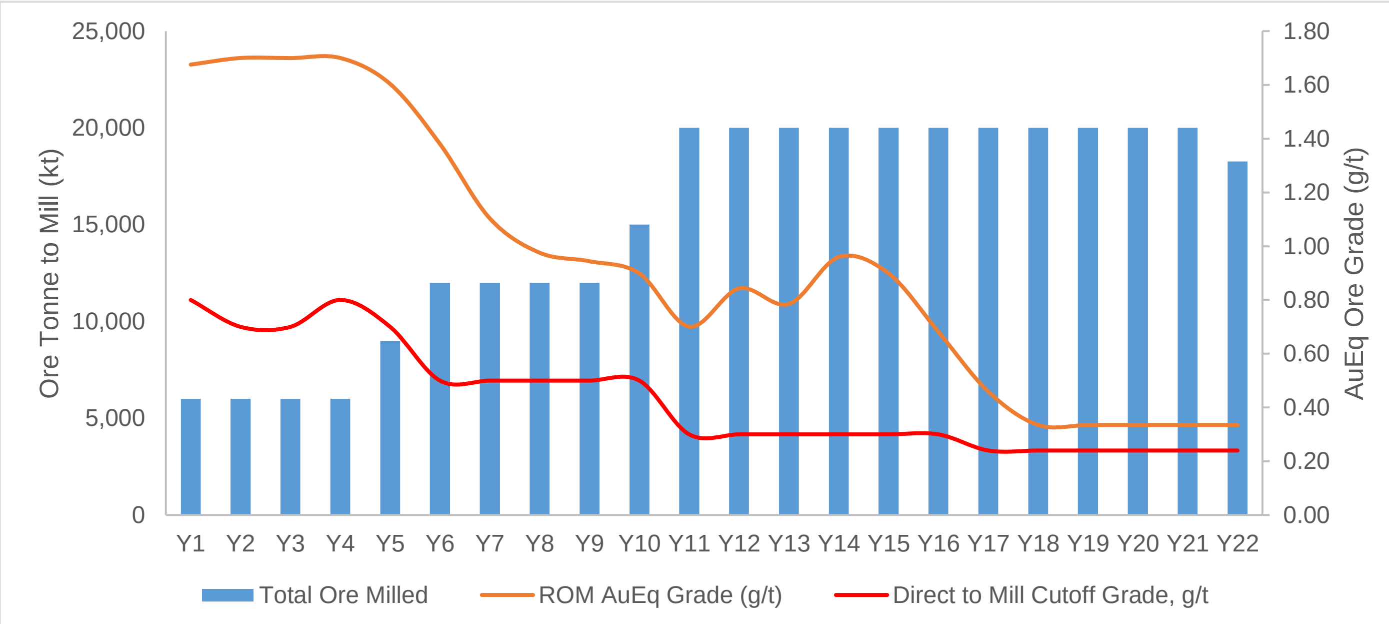
Maintenance on mine equipment will be performed in the field with major repairs to mobile equipment conducted in the workshops that will be located west of the plant facilities.

Annual mine operating costs per tonne mined will range from C$1.96–C$3.64/t with a LOM average of C$2.60/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 Blackwater are downhill loaded hauls. Mobile equipment maintenance operations will also be managed by the Owner and are included in the mine planning and costs.

After mining is completed, the mining equipment will be removed, and the pits will be allowed to fill with water-producing ponds. Contouring and re-vegetation of the fill areas will be completed.

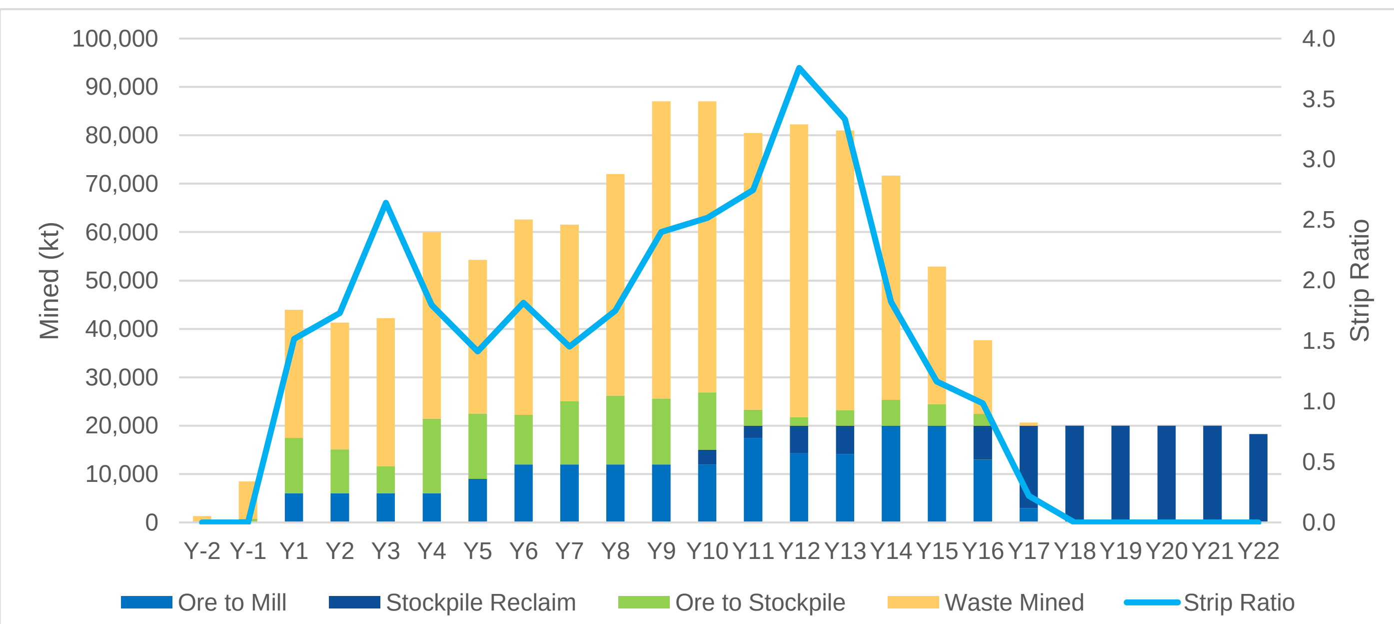
Figure 1-2 and Figure 1-3 summarize the proposed ore and waste schedule for the 2021 FS mine plan.

Figure 1-2: Planned Mill Feed Tonnes and Grade



Note: prepared by MMTS, 2021

Figure 1-3: Planned Material Mined and Strip Ratio



Note: prepared by MMTS, 2021

1.18 Recovery Methods

Gold-silver mineralisation in the Blackwater deposit is associated with sulphides, occurring in veins and disseminations; free gold is also present to varying extents. Gold and silver values from the types of mineralisation present in the Blackwater deposit are largely recoverable by a combination of gravity processes and conventional cyanidation. The preferred process flowsheet selected for recovery of the gold and silver values was derived from the testwork results and tailored to support a robust production profile over the life of mine. The unit operations that were included in the selected process are well proven at the commercial scale, and typical in the industry.

The process plant will be constructed in four distinct phases, as outlined below:

- Phase 1 (6Mtpa) operating for years 1 to 5
- Phase 2 (12Mtpa) operating for years 5 to 10
- Phase 3 (20Mtpa) operating for years 10 to 17
- Phase 4 (20Mtpa) operating for years 17 to 22

The Phase 1 process plant will treat 6Mtpa or 745t/h based on an availability of 8,059 hrs per annum or 92%. The crushing section design is set at 70% availability and the gold room availability is set at 52 weeks per year including seven operating days and five smelting days per week. The plant will operate with two shifts per day, 365 days per year, and will produce doré bars.

Run-of-Mine (ROM) material will be hauled to the primary gyratory crusher where a front-end wheel loader (FEL) will supplement the direct-tip feed from the ROM stockpile to maintain a continuous crushing operation. The crushing circuit will include secondary and tertiary cone crushing and screening to produce a crushing circuit product size with a P80 of 8mm. Crushed ore will be stored on a covered conical stockpile. The crushed material will be processed through a dual pinion ball mill in closed circuit with cyclones producing a final product with a P80 of 150 µm. The installed ball mill power will be 14 MW and the mill dimensions will be 7.3m x 12.5m (internal diameter x effective grinding length) with a circulating load of 400%.

A portion of the ball mill discharge will feed two parallel gravity concentrator trains. The intensive cyanidation circuit will receive gravity gold concentrate on a batchwise basis for treatment in an intensive leach reactor.

The cyclone overflow will pass over a trash screen and will be pumped to the leach-adsorption circuit consisting of one pre-aeration tank, three leach tanks and seven carbon-in-leach (CIL) adsorption tanks. The combined leach and adsorption circuit residence time will be 24 h at 45% w/w solids. Gold and silver leached in the leach-CIL circuit will be recovered onto activated carbon and eluted in an Anglo-American Research Laboratory (AARL)-style elution circuit and then precipitated by electrowinning in the gold room. The gold–silver precipitate will be dried in a drying oven and then mixed with fluxes and smelted in a furnace to pour doré bars. Carbon will be re-activated in a carbon regeneration kiln before being returned to the CIL circuit.

Slurry exiting CIL Tank 7 will gravitate to two cyanide detoxification tanks which are designed based on the conventional O_2/SO_2 process. The detoxified slurry stream will gravitate to the carbon safety screen and on to the tailings pump box, from where it flows by gravity through a single pipeline to the TSF. A lime neutralization system for run-off will be installed for the low-grade ore (LGO) stockpile and will be neutralized in the processing plant through lime addition prior to discharge to the TSF.

The installed power for the Phase 1 process plant is estimated to be 32.5 MW, and power consumption will be 32.3kWh/t of material treated for the processing plant. Raw water will be supplied from the water management pond and depressurisation wells to a raw-water storage tank. Potable water will be supplied from the potable water treatment plant at the camp. Gland water will be supplied from the raw-water tank. Process water will primarily consist of TSF reclaim water, supplemented by contact water and raw water. Reagents will include quicklime, sodium cyanide, sodium hydroxide, copper sulphate, hydrochloric acid, elemental sulphur and oxygen.

The Phase 2 expansion will treat an additional 6.0 Mtpa (12 Mtpa total) through minor upgrades to the Phase 1 crushing circuit, and addition of new milling, leaching, adsorption, elution, and detox capacity.

The Phase 3 expansion will include a new process line consisting of crushing, grinding, leaching, adsorption and detox circuits with a capacity of 8.0 Mtpa. The Phase 1 and 2 acid wash, elution, electrowinning and gold room facilities will be used, and combined throughput will increase from 12 Mtpa to 20 Mtpa.

Phase 4 will be a continuation of processing per the completed Phase 3 expansion with ore being rehandled from stockpiles for processing.

1.19 Onsite Infrastructure

The Project can be accessed from Highway 37, west of Vanderhoof, via the Kluskus and Kluskus-Ootsa FSRs. A new 15.6km portion of access road will be built to connect the FSR with the mine plant site for transportation of equipment and materials.

Presently, the mine plant site and existing camp is accessed from the Kluskus-Ootsa FSR by an exploration road. This road will be partially decommissioned following completion of the new mine access road. The remaining portions of the exploration road within in the mine site boundary will be used for local construction access and mine operations. The sections of the exploration road located within the TSF will be inundated in approximately Year 6.

Approximately 8 km of on-site roads will be constructed to provide access to the truck shop, mine plant site, accommodations and explosives store. These roads will be approximately 10 m wide, gravel roads, to allow two-way traffic.

A 15km road will connect the planned pumping station at Tatelkuz Lake to the access road.

The process plant will consist of the crushing, screening, stockpile, grinding, gravity separation, intensive leaching reactor, leaching, CIL, carbon elution and regeneration, reagents, cyanide detoxification, and the gold room. The grinding mills, reagent storage and gold room will be in enclosed buildings. The gold room will be constructed as a pre-engineered building, complete with a heavy-duty building enclosure, closed-circuit televisions (CCTVs), motion sensors and alarms to prevent unauthorized entry.

A fibre-optic backbone will be included throughout the plant to provide an ethernet-type system for voice, data, and control systems bandwidth requirements.

The truck shop, mine dry, and administration offices will be located in a shared building of modular construction. In the initial construction of the truck shop facility, it will include two bays to service the mining fleet. There will be overhead crane availability, with a 15 t capacity, and clearing for 230 t haul trucks. A further expansion of the truck shop facilities is planned in year 1. This building will also include meeting rooms, wash facilities, toilets, closed offices and an open work area equipped with workstations.

The plant offices will be adjacent to the main plant buildings and will house all plant operating and maintenance offices. The central control room will be in this complex, with CCTV coverage of all parts of the plant. The plant offices will also include a change room and toilets. The construction will be modular, similar to the administration offices.

The laboratory modular construction will be modified to allow solid floors where necessary for heavy equipment such as crushers or fire assay furnaces. It will include toilets and a change room. Some area will be available for sample storage, but the main storage will be in an unheated adjacent building.

1.19.1 Power Supply

Power will be supplied by connection to the BC Hydro grid via a 135 km long 230kV transmission line. The line will follow existing resource roads and other previously-disturbed areas as much as practicable. Emergency back-up diesel generators will be located at the process plant, plant offices and control room and at the cellular tower.

1.20 Waste Characterization

A testing program has been completed to geochemically characterize the geologic materials that will be produced by the Project. This program characterized waste rock, ore, tailings and overburden, analyzing these materials using methods that included acid base accounting, metal assays, and humidity cell tests. These results have been used to define the acid generating potential and metal leaching potential of mine waste. Overall, the results show that the Project will produce waste rock and tailings that are classified as potentially acid generating (PAG) and/or metal leaching (ML). A classification scheme has been developed to rate the ARD and ML potential of mine waste and is used to guide waste management.

1.21 Tailings Storage Facility

The TSF was designed to permanently store tailings, PAG waste rock, and potentially ML NAG waste rock that will be generated during operations. The facility was designed to hold 469 Mm^3 of tailings and waste rock material, and up to 12 Mm^3 of pond storage under normal operating conditions. The TSF will comprise two adjacent sites, TSF C and TSF D.

TSF C will be constructed first to provide storage capacity for start-up of the process plant. It was designed to contain tailings for approximately 21 years of mine operations and PAG/ML waste rock generated during the first six years of mining. TSF C will comprise a valley-fill style impoundment formed by construction of three embankments (Main Dam C, the West Dam, and the Saddle Dam) in the upper reaches of the Davidson Creek drainage basin. Main Dam C will be initially constructed during the preproduction phase to form TSF C and will be raised annually through Year 6 using centreline construction methods. The embankment will be an engineered, water retaining, zoned earth-rockfill dam with a compacted low-permeability seal zone and appropriate filter/transition zones flanked by a pit-run shell zone. Thereafter, the dam will be raised periodically in stages approximately 8 m high using downstream construction methods comprising zoned earth-rockfill complete with high density polyethylene (HDPE) geomembrane facing. The West Dam and Saddle Dam will be constructed in Years 6 and 12, respectively, and raised periodically in stages along with the later stages of Main Dam C. Construction of the embankments for TSF C requires placement of approximately 54 Mm^3 of fill material to reach a final elevation of 1,353 masl.

TSF D will be formed adjacent to and downstream of TSF C beginning in Year 5, during the Phase 2 expansion, to provide additional storage capacity to contain PAG/ML waste rock generated between Year 6 and the end of mining and up to two years of tailings beginning in approximately Year 21 when TSF C reaches design capacity. TSF D will be formed by construction of one embankment (Main Dam D). The embankment will be an engineered, water retaining, zoned earth-rockfill dam with a compacted low-permeability seal zone and appropriate filter/transition zones flanked by a pit-run shell zone. Main Dam D will be raised annually using centreline construction methods. Construction of Main Dam D requires placement of approximately 48 Mm^3 of fill material to reach a final elevation of 1,331 masl.

The 2021 FS is supported by the detailed design of the Stage 1 TSF and associated facilities required for the start of mine operations. Main Dam C Stage 1 was designed as a water-retaining dam with a crest elevation of 1,273 masl. The Stage 1 TSF will provide sufficient capacity to impound tailings and PAG/ML waste rock generated during the first year of operations and a supernatant pond up to 2 Mm^3, with additional capacity to manage seasonal water volume fluctuations and the environmental design flood. An emergency spillway will be constructed along the right abutment to pass the inflow design flood. Main Dam C will require placement of approximately 3.25 Mm^3 of fill material for Stage 1 that will be sourced from local external borrow sources and pre-stripping of the open pit during Year -1 of mine development.

1.22 Water Management Structures

The principal design objectives for the water management structures are to manage surface water during mine operations and active closure. Surface water is to be managed in a manner that allows for the beneficial use of the water to support mine operations and to divert flow not needed to the downstream receiving environment. Drainage from the majority of the mine area will flow by gravity into the TSF, following natural topographical drainages mapped for the Project; which simplifies water management, spill control, and mine closure. 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 2021 FS is supported by detailed design of the water management facilities required at the start of mine operations. Specific water management structures and systems planned for the mine operations period include:

- Fresh Water Reservoir (FWR) to store water and provide flows to lower Davidson Creek to meet instream flow needs downstream of the mine and to provide water for mine operations when required.
- A lined Water Management Pond (WMP) located downslope of the open pit and stockpiles area and within the ultimate footprint of TSF C to manage runoff from contributing areas and water pumped from collection points. The WMP will provide fresh make-up water to support ore processing. Water not needed to support mine operations will be used to augment flow in lower Davidson Creek. The WMP will be relocated in approximately Year 12.
- A discharge system to route freshwater from the WMP to the FWR, and a raw water supply system to route freshwater from the WMP to the plant site.
- Reclaim water system to route supernatant water from the TSF to the plant site.
- Central Diversion System (CDS) to divert fresh water around the TSF or to a water transfer pond, from where it can be pumped to the WMP.
- Northern Diversion System (NDS) to divert freshwater around TSF D to the FWR or allow it to bypass diversion and flow into TSF D, depending on the needs of the mine, beginning during the Phase 2 expansion.
- Fresh water from Tatelkuz Lake supplied by the Fresh Water Supply System (FWSS) to the FWR beginning during the Phase 2 expansion.
- Stockpile water management structures to divert and contain seepage and surface water runoff from the LGO stockpile and the waste stockpiles.

- Water treatment plants (WTPs), including a metals WTP, membrane WTP, and lime neutralization circuit at the mill.

The facilities listed above will be used to achieve the following requirements in the mine water management plan:

- 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.

1.23 Closure and Reclamation Plan

The Closure and Reclamation Plan will take advantage of progressive reclamation opportunities through the life of the mine. In particular, mining operations will cease in the Open Pit in the later Operations phase. Closure of major mine infrastructure is anticipated to take one to three years after cessation of ore processing from stockpiled ore. Reclamation of the Project area will conform to the requirements of the Health, Safety, and Reclamation Code for Mines in BC (BC EMLI 2021). As much as possible, disturbed areas will be reclaimed to native ecosystems and waterways restored to pre-disturbance flow patterns. In the extended Closure and Post-closure phases, activities will focus on monitoring vegetation and geotechnical stability of reclaimed areas, and water treatment, as required.

1.24 Environmental, Social, Economic, and Cultural Heritage Considerations

The Project is supported by a suite of environmental, social, economic, and cultural heritage baseline studies. The potential Project effects to environmental, social, economic, and cultural heritage components have been fully assessed. The Project was granted an Environmental Assessment Certificate (EAC) #M19-01 on June 21, 2019 (EAO 2019c) under the provincial *Environmental Assessment Act* (2002) and an Environmental Assessment Decision Statement (DS) on April 15, 2019 under the federal *Canadian Environmental Assessment Act, 2012* (CEA Agency 2019b). Assessment of components to address updates in the Project design have been considered in recent permits (Table 20-1) and will be addressed in permit applications currently in progress (Table 20-2 and Table 20-3). To manage potential effects of the Project, an Environmental Management System supported by a comprehensive set of management plans is being developed for the permitting phase of the Project (Section 20.1).

1.25 Permitting

A complete set of provincial and federal permits, licenses, and authorizations to approve the construction and operation of the Project have been identified and are in progress. Key federal approvals include: 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: permit approving mine plan and reclamation program (*Mines Act*), effluent discharge permit and air discharge permit (*Environmental Management Act*), licence of occupation to occupy crown land for transmission line (*Land Act*), and water licence for use of offsite water (*Water Sustainability Act*).

1.26 Capital Cost Estimate

The capital cost estimates have been summarized at the levels indicated by Table 1-3 and stated in Canadian dollars with a base date of Q2-2021 and an accuracy range of +15%/-10%. No provisions for forward escalation have been included. This estimate collectively presents the entire costs for the project including all Third-Party estimates, Owner's scope and Ausenco's scope. The estimate is summarized in Table 1-3.

Table 1-3: Estimate Summary Level 1 Major Facility (C$ M)

WBS Lvl 1	WBS Description	Initial Capital	Expansion / Growth Capital	Sustaining Capital	Deferred Capital	Total LOM Capital
1000	Mining	64.5	62.8	430.1	4.0	561.4
2000	Site development / tailings storage facility / waste rock facility	73.1	175.9	273.2	0.2	522.5
3000	Ore crushing and reclaim	55.6	45.2	0	0	100.8
4000	Process plant	141.7	254.6	0	0	396.3
5000	On-site infrastructure	30.5	31.5	17.0	26.1	105.2
6000	Off-site infrastructure	100.7	0	22.8	12.7	136.2
	Subtotal Direct Costs	466.2	570.0	743.2	43.0	1,822.40
7000	Indirects	16.5	26.2	2.6	0.1	45.4
8000	Engineering and project management	60.9	58.1	22.7	3.5	145.1
9000	Provisions	101.7	66.3	62.2	5.1	235.3
	Subtotal Indirect Costs	179.0	150.6	87.5	8.6	425.8
	Project Total (C$ M)	645.2	720.6	830.7	51.6	2,248.2

1.27 Operating Cost Estimate

For operating cost estimating purposes, the Project was divided into three areas; mining, processing, and general and administrative (G&A). The costs for each department include labour, operating and maintenance supplies, freight, and utilities as appropriate. The expected accuracy range of the operating cost estimate is +15%/-10%.

Operating costs of operation are summarized in Table 1-4 below.

Table 1-4: Summary of Operating Costs

	Units	Years 1 -5	Years 6 - 10	Years 11 - 17	Years 18-22	LOM
Mining	C$/t Mined	2.26	2.36	2.79	n/a	2.60
	C$/t Milled	15.59	13.77	7.51	0.99	7.57
Process	C$/t Milled	9.47	8.73	8.06	8.06	8.32
G&A	C$/t Milled	4.12	2.60	1.89	1.31	2.07
Total	**C$/t Milled**	**29.18**	**25.09**	**17.45**	**10.36**	**17.96**

1.28 Economic Analysis

1.28.1 Cautionary Statement

The results of the economic analysis discussed in this section represent forward-looking information as defined under Canadian securities law. Actual results may differ materially from those expressed or implied by forward-looking information.

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;
- 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.

1.28.2 Cashflow Basis

The economic analysis was carried out using a discounted cash flow model with base case metal price assumptions of:

- Gold – US$1,600/oz;
- Silver – US$21.33/oz;
- Exchange rate – 0.79 (US$/C$).

The economic analysis is presented as a Base Case, which assumes no leverage, and a Leveraged Case, which assumes debt financing. Financing of the Project is not a measure of the economic viability and technical feasibility of the Project, but a measure of the ability of Artemis to secure debt financing for the Project.

1.28.3 Base-Case

For the 22-year mine life and 334 Mt mill feed, the following after-tax Base Case financial parameters were calculated:

- C$2,151 million NPV at 5.0% discount rate;
- 32% IRR;
- 2.3 year initial capital payback.

1.28.4 Leveraged Case

For a leveraged case assuming C$360 M (plus up to C$25 M in capitalized interest) in project debt financing at an annual interest rate of 5.5%, an upfront financing fee of 3%, and a seven-year term post commencement of commercial production with a balloon payment of 30% of the principal at maturity, the following after-tax Leveraged Case financial parameters were calculated::

- C$2,158 million NPV at 5.0% discount rate;
- 43% IRR;
- 2.4 year initial capital payback.

The Leveraged Case is based on the Base Case and the following additional assumptions:

- C$360 M (plus up to C$25 M in capitalized interest) in in project debt financing;
- Annual interest rate of Canadian Dollar Offered Rate (assumed at 0.5% in the 2021 FS) plus a margin of 4.25% up to the date of completion, with the margin reducing to 3.75% once the Project is effectively in commercial production;

- Customary upfront and standby financing fees;
- Six-year term post commencement of commercial production with Principal and capitalized interest repayable in quarterly instalments over six years, commencing one year following achievement of commercial production, with a repayment holiday during years 4 and 5 of production when Artemis expects to undertake its Phase 2 expansion;
- Expansion capital is assumed to be funded through operating cashflow.

1.28.5 Sensitivity Analysis

NPV sensitivity analysis was performed on the Project base case using gold price, gold grade, exchange rate, operating costs and initial capital costs. The impacts of changes in the gold grade mirror the impact of changes in the gold price. The Project is more sensitive to changes in the gold price (grade) and the US$:C$ exchange rate than to changes in capital or operating costs.

1.29 Risks and Opportunities

The major risks to the Project are identified as:

- Changes to metal prices and exchange rate assumptions;
- Capital cost growth;
- Increases in operating costs;
- Productivity assumptions;
- Mining grade and dilution control;
- Presence of high-grade silver in the mill feed;
- Geotechnical and hydrogeological uncertainty;
- Climate uncertainty and associated water management needs;
- Integration of mining operations and the TSF construction;
- Permitting delays;
- Lack of social license affecting permit grant.

Project opportunities include:

- Delineation of additional mineralization that could support higher-confidence resource categories through additional drilling;
- Use of a trolley-assist system later in the mine life;

- Assessment of methods to reduce waste mining costs;
- Value engineering initiatives.

1.30 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.

Ausenco considers that the scientific and technical information available on the Project can support a mine construction decision. However, the decision to proceed with a mining operation on the Project is at the discretion of Artemis.

1.31 Recommendations

It is recommended Artemis complete additional field and laboratory-based programs, across the development and operations phase of the project, at an estimated value of C$3 M.

2 INTRODUCTION

2.1 Introduction

Ausenco Engineering Canada (Ausenco), Moose Mountain Technical Services (MMTS), Knight Piésold Ltd. (KP), Allnorth Consultants Ltd. (Allnorth), Lorax Environmental Services Ltd. (Lorax), ERM Consultants Canada Ltd., and JAT Met Consult Ltd. (JAT Metco) have prepared a technical report (the Report) for Artemis Gold Inc (Artemis) on a feasibility study (FS) evaluation (2021 FS) of the Blackwater Gold Project (the Project), located 112 km southwest of Vanderhoof in British Columbia, Canada.

2.2 Terms of Reference

The Report supports disclosures in in Artemis' press release entitled Artemis Announces Feasibility Study for Blackwater Project, dated September 13, 2021.

All measurement units used in this Report are metric, and currency is expressed in Canadian dollars (C$) unless stated otherwise. Years used in the mine plan are for illustrative purposes only, as the decision to implement production is at the discretion of Artemis and permits to support operation still have to be obtained. Mineral Resources and Mineral Reserves are estimated in accordance with using the 2019 edition of the Canadian Institute of Mining, Metallurgy and Exploration (CIM) Estimation of Mineral Resources and Mineral Reserves Best Practice Guidelines (2019 CIM Best Practice Guidelines) and are reported using the 2014 CIM Definition Standards for Mineral Resources and Mineral Reserves (2014 CIM Definition Standards).

BW Gold Ltd. (BW Gold) is the holding entity for the mineral claims, and party to the purchase agreement with New Gold Inc. (New Gold). BW Gold is a wholly-owned subsidiary of Artemis. For the purposes of this Report, Artemis is used interchangeably for the subsidiary and parent companies.

The six contiguous claim blocks held by Artemis, specifically the Blackwater, Capoose, Auro, Key, Parlane and RJK claim blocks, are referred to as the Property for the purposes of this Report. The Project refers to exploration and development activity related to the Blackwater deposit which is contained within the Blackwater claim block.

2.3 Qualified Persons

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

- Robin Kalanchey, P. Eng., Ausenco Engineering Canada
- Sue Bird, P.Eng., Moose Mountain Technical Services;
- George Dermer, P. Eng., Moose Mountain Technical Services
- Daniel Fontaine, P.Eng., Knight Piésold Ltd.;

- James Garner, P. Eng., Allnorth
- Marc Schulte, P.Eng., Moose Mountain Technical Services;
- Rolf Schmitt P. Geo., ERM
- John Dockrey, P. Geo., LORAX
- John Alan Thomas, P.Eng., JAT Met Consult Ltd.

2.4 Site Visits and Scope of Personal Inspection

Site visits were performed as follows:

2.4.1 Robin Kalanchey Site Visit

Robin Kalanchey visited the site on 6 July 2021. During his visit, Robin viewed the proposed location of the processing plant, the open pit, Waste Rock Storage Facilities (WRSFs), haul and access roads and Tailings Storage Facility (TSF). Additionally, he reviewed selected drill core samples, and the on-site and off-site infrastructure options

2.4.2 Sue Bird Site Visit

Sue Bird visited the site on 14 July 2020. Sue reviewed the drillhole 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 drillholes were also inspected and reviewed.

2.4.3 Daniel Fontaine Site Visit

Daniel Fontaine visited the site most recently on November 3–8, 2020 as well as on several other occasions previously between 2011 and 2019. During his visits, Daniel viewed the proposed locations of the processing plant, open pit, low-grade ore and waste stockpiles, select haul roads, TSF and associated dam sites, and select locations associated with water management components of the proposed Project.

2.4.4 Marc Schulte Site Visit

Marc 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.

2.4.5 John Thomas Site Visit

John Thomas visited the site on 07 May 2020. During his visit, John 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.

2.5 Effective Dates

The Report has the following effective dates:

- Effective date of the Mineral Resource estimate: 5 May 2020;
- Effective date of the Mineral Reserve estimate: 10 September 2021;
- Effective date of the economic analysis: 10 September 2021.

The overall Report effective date is taken to be the date of the economic analysis and is 10 September 2021.

2.6 Information Sources and References

Information sources used in compiling this Report are included in Section 27.

2.7 Previous Technical Reports

Artemis has filed the following technical report:

- 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:

- 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 March 2, 2011, re-addressed June 6, 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 September 19, 2011.
- 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 March 7, 2012.
- 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.
- 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.

2.8 Abbreviations

Table 2-1: Name Abbreviation

Abbreviation	Description
%	percent
AARL	Anglo-American Research Laboratory
AAS	atomic absorption spectrometry
ABA	Acid Base Counting
AEMP	Aquatic Effects Monitoring Program
Ag	silver
Ai	Abrasion Index
AQDMP	Air Quality and Dust Management Plan
ARD	Acid Rock Drainage
Artemis	Artemis Gold Inc.
Au	gold
AuEq	gold equivalent
BAP	Best Available Practices
BAT	Best Available Technology
BC	British Columbia
BWi	Bond Ball Mill Work Index
C$	Canada Dollars
CCME	Canadian Council of Ministers
CDS	Central Diversion System
CEMMP	Community Effects Monitoring and Management
CFN	Cheslatta Carrier Nation
CIL	Carbon-in-Leach
CIM	Canada Institute of Mining
CIP	Carbon-in-Pulp
CLC	Community Liaison Committee
CMMP	Caribou Mitigation and Monitoring Plan
CRD	Cariboo Regional District
CPP	Cumulative Probability Plots
CSS	Closed Side Settings
CWD	Coarse Woody Debris
CV	Coefficients of Variation
DAF	Dissolved Air Flotation
DCF	Discounted Cash Flow
DS	Decision Statement
DSCR	Dam Site Characterization Report
EAC	Environmental Assessment Certificate (EAC) #M19-01
EC	Environment Canada
ECCC	Environment and Climate Change Canada
ECD	Environmental Control Dam
EDGM	Earthquake Design Ground Motion
EMLI	Ministry of Energy, Mining, and Low Carbon Innovation
ELUP	End Land Use Plan

EMA	Environmental Management Act
EMS	Environmental Management System
ENV	Ministry of Environment and Climate Strategy
EOA	Environmental Assessment Office
EOP	End-of Period
ERM	ERM Consultants Canada Ltd.
FA	Fire Assay
FEL	Front-End Wheel Loader
FSR	Forest Service Road
FWR	Freshwater Reservoir
FWSS	Fresh Water Supply System
GPS	Global Positioning System
HDPE	High Density Polyethylene
IDF	Inflow Design Flood
IECD	Interim Environmental Control Dam
IFN	Instream Flow Needs
kLCM	Thousand Loose Cubic Metres
KP	Knight Piésold Ltd
LDN	Lhoosk'uz Dené Nation
LG	Lerchs-Grossmann
LGO	Low-Grade Ore
LOM	Life of Mine
LPS	Low-Permeability Subgrade
LRMP	Land & Resource Management Plan
MA	Mines Act
MBBR	Moving Bed Biofilmreactor
MDE	Maximum Design Earthquake
MDMER	Metal and Diamond Mining Effluent Regulations
MIK	Multiple Indicator Kriging
ML	Metal leaching
MSDP	Mine Site Water and Discharge Monitoring and Management Plan
MSTCP	Mine Site Traffic Control Plan
NAG	Non-Acid Generating
ND	Nominal Diameter
NDS	Northern Diversion System
NFN	Nazko First Nation
NP	Neutralization Potential
NPR	Neutralization Potential Ratio
NTBB	Nee Tahi Buhn Band
NWFN	Nadleh Whut'en First Nation
OK	Ordinary Kriging
OMS	Operations Maintenance and Surveillance
PA	Participation Agreement
PAG	Potentially Acid Generating
PMF	Probable Maximum Flood
POC	Parameters of Concern
POPC	Parameters of Potential Concern

PPDT	Pore Pressure Dissipation Tests
PPE	Personal Protective Equipment
PST	Provincial Sales Tax
QP	Qualified Professional
RDBN	Regional District of Bulkley-Nechako
RC	Reverse Circulation
RCP	Reclamation and Closure Plan
RMR	Rock Mass Rating
RMZ	Resource Management Zone
ROM	Run of Mine
RQD	Rock Quality Designation
RTK	Real Time Kinematic
RWi	Rod Mill Work Index
SAG	Semi-Autogenous Grind
SCP	Sediment Control Pond
SEPSCP	Surface Erosion Prevention and Sediment Control Plan
SFN	Saik'uz First Nation
SMC	Semi-Mill Comminution
SOP	Standard Operating Procedures
SPT	Standard Penetration Tests
StFN	Stellat'en First Nation
TSF	Tailings Storage Facility
UFN	Ulkatcho First Nation
UCF	Undiscounted Cash Flow
VMP	Vegetation Monitoring Program
WAD-CN	Weak Acid Dissociable Cyanide
WBM	Water Balance Model
WHC	Waste Hazard Class
WMMP	Wildlife Mitigation and Monitoring Plan
WMP	Water Management Pond
WMOP	Wetland Monitoring and Offsetting Plan
WSF	Waste Storage Facility
WTA	Waste Transfer Areas
WTP	Water Treatment Plant
YFN	Yekooche First Nation

Table 2-2: Unit Abbreviations

Abbreviation	Description
US$	United States dollar
C$	Canadian dollar
°C	degree Celsius
°F	degree Fahrenheit
%	percent
µ	micro
µm	micrometre

Abbreviation	Description
C$	Canadian dollar
cm	centimetre
ft	feet
ft^2	square feet
g	gram
g/t	grams per tonne
ha	hectare
hr	hour
HP	horsepower
km	Kilometre (Canada) Kilometer (US)
koz	thousand ounces
kV	kilovolt
kW	kilowatt
kWh	kilowatt-hour
kWh/t	Kilowatt per tonne
kN/m^3	kilonewton per cubic metre
MW	megawatt
kPa	kilopascal
kN	kilonewton
masl	metres above sea level
mamsl	metres above mean sea level
L/s	litre per second
M	million
m	metre
m/a	metres per annum
m^2	square metre
m^3	cubic metre
mm	millimetres
Mtpa	million tonnes per annum
t	metric tonne
st	short ton
Mt	million tonnes
oz	ounce
Moz	million ounces
ppb	parts per billion
ppm	parts per million
t	Metric tonne
t/hr	tonnes per hour
tpd	tonnes per day
t/a	tonnes per annum
w/w/ w/s	gravimetric moisture content (weight of water/weight of soil)
wt	weight

3 RELIANCE ON OTHER EXPERTS

3.1 Introduction

The authors of this Report state that they are qualified persons (QP) for those areas identified in the "Certificate of Qualified Person" for each QP, as included in this Report. The QPs have relied and believe there is a reasonable basis for this reliance, upon the following other expert reports, which provided information regarding mineral rights, surface rights, and taxation in sections of this Report as noted below.

3.2 Mineral Tenure

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 experts and experts retained by Artemis for this information through the following documents:

- Blake, Cassels and Graydon, LLP, 2020: ownership confirmation letter addressed to Artemis dated 25 August 2020.

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. The QPs have assumed that the information in such letter is accurate and understand that the information in such letter may not be relied upon by any other party without the consent of Blake, Cassels & Graydon LLP.

3.3 Taxation

The QP's are not experts with respect to Canadian Federal or Provincial income taxes, and have fully relied upon information provided by Artemis Gold Inc., and disclaim responsibility for the application of this information in Section 22, and in support of the Mineral Reserve estimate in Section 15.

The information related to taxation matters was provided by Artemis Gold Inc. in the form of e-mails and Excel spreadsheet reviews during the preparation of this Technical Report in September 2021. These matters were not independently verified by the QPs but appear to be reasonable representations that are suitable for inclusion in this report.

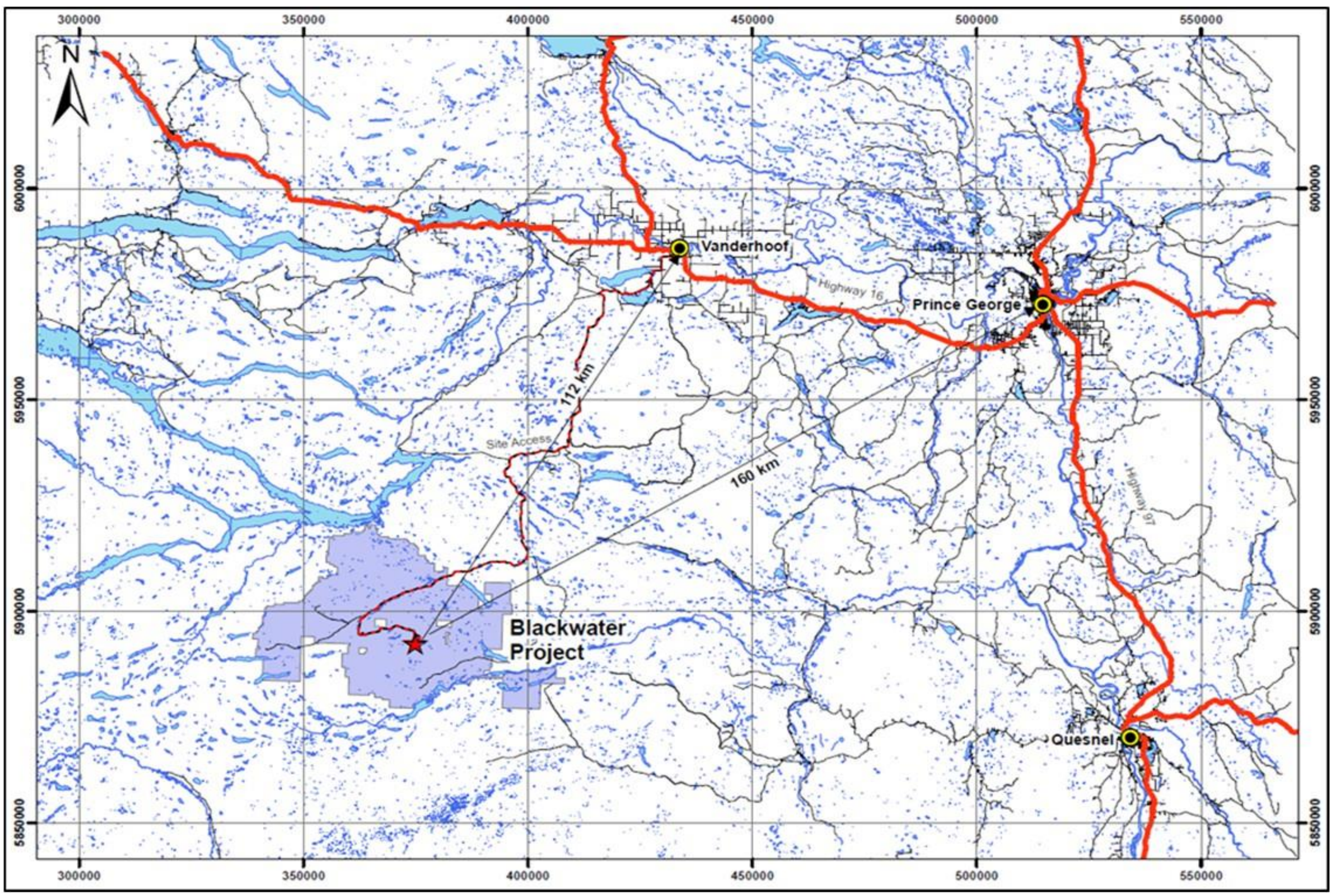
4 PROPERTY DESCRIPTION AND LOCATION

4.1 Introduction

The Property and Project lie in central British Columbia, approximately 112 km southwest of Vanderhoof and 446 km northeast of Vancouver, as shown in Figure 4 1. The Project is within NTS map sheet 93F/02 and is centred at 5893000 N and 375400 E (UTM NAD83).

The Property comprises six contiguous claim blocks held by Artemis (Blackwater, Capoose, Auro, Key, Parlane and RJK) as shown in Figure 4-2.

Figure 4-1: Location Plan Blackwater Project



Note: prepared by MMTS, 2020.

4.2 Property Ownership History

The area was initially explored by Granges Inc. 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.

All mineral claims are cell claims.

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 a further mineral claim to form the Auro claim block. An additional infill claim was recorded and added to the Auro claim block in 2013. New Gold acquired the Key claim block in 2013 from Troymet Exploration Corporation (Troymet).

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

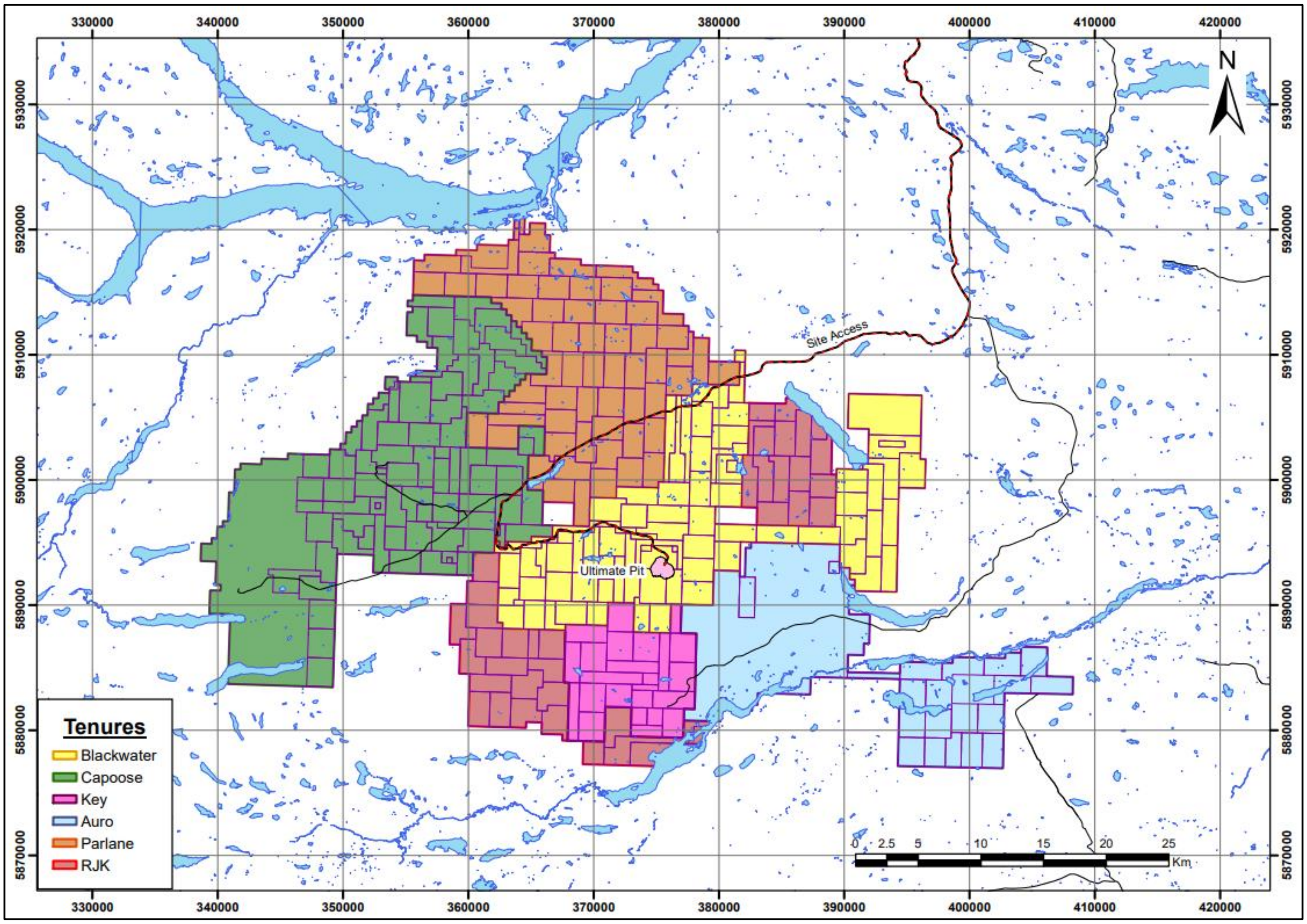
In June 2017, New Gold acquired the BW East, BW West and BW South claims from RJK Explorations Ltd. (RJK) as well as the Big Bear property from Parlane Resource Corp. (Parlane).

On August 21, 2020, Artemis acquired all of New Gold's mineral tenure, assets and rights related to the Blackwater Project and now holds a 100% interest in the Blackwater Project.

4.3 Mineral Tenure

Artemis holds a 100% recorded interest in 329 mineral claims covering an area of 148,902 ha (Figure 4-2).

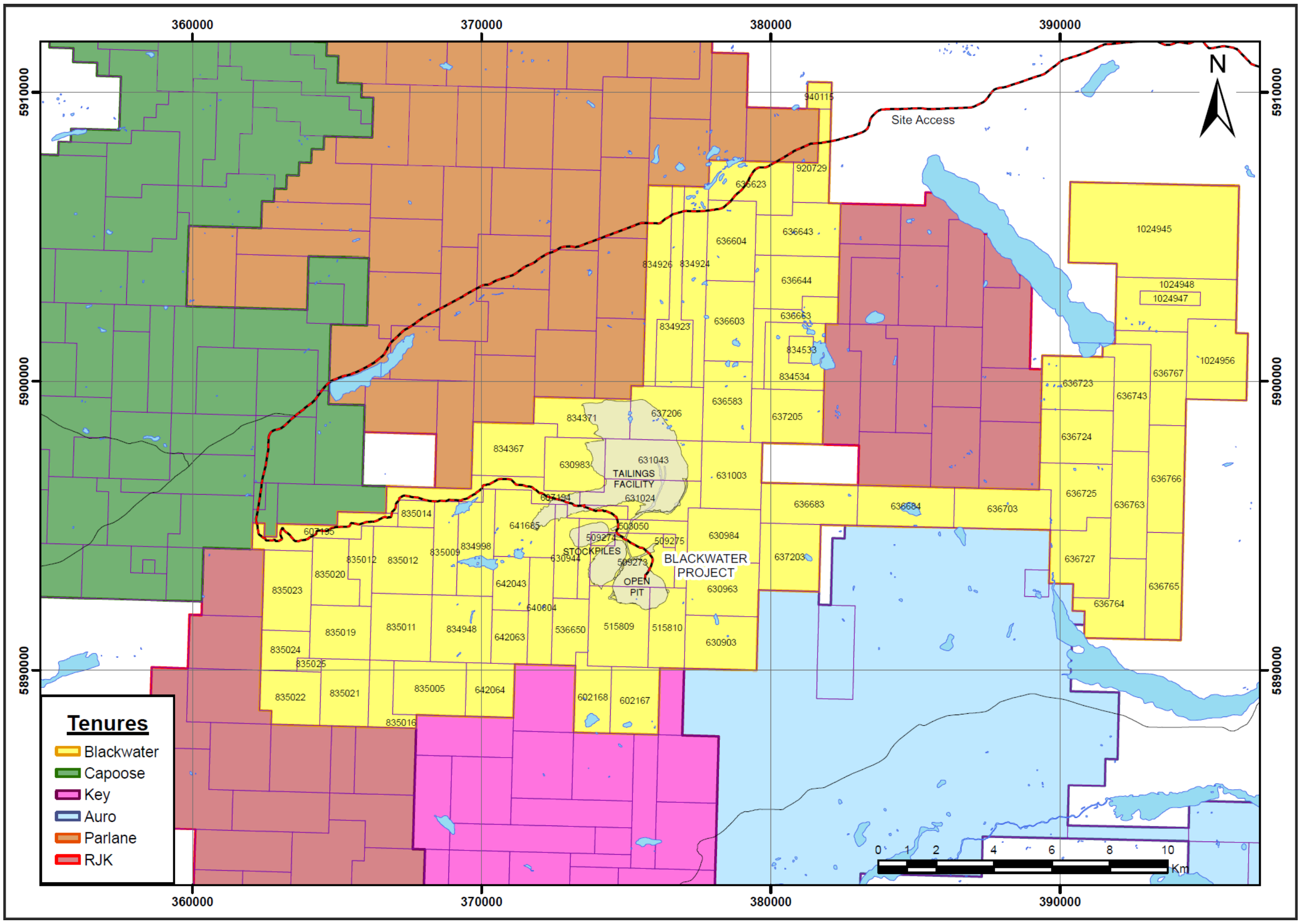
Figure 4-2: Mineral Claim Blocks of the Property



Note: prepared by MMTS, 2021.

Mineral claims covering the Project area are shown in Figure 4-3.

Figure 4-3: Mineral Claim Blocks of the Project



Note: prepared by MMTS, 2021. Infrastructure locations shown on the figure for the Blackwater Project are proposed locations; none of this infrastructure has been constructed.

4.3.1 Blackwater Claim Block

The Blackwater claim block comprises 76 mineral cell claims totaling 30,791ha (Table 4-1). All Blackwater claims are 100% held in the name of Artemis. Artemis holds both the recorded and beneficial interest in these claims.

None of the Blackwater cell claims are known to overlap any legacy or Crown granted mineral claims, or no-staking reserves. The Blackwater deposit spans the Davidson (509273), Dave (515809) and Jarrit (515810) claims.

4.3.2 Capoose Claim Block

The Capoose claim block is situated west of the Blackwater claim block and consists of 106 mineral claims totaling 42,191 hectares. All Capoose claims are 100% held in the name of Artemis. Capoose mineral claims are summarized in Table 4-1.

All claims in the Capoose claim block, excepting claim 238045, are cell claims. Claim 238045 is a four post claim. Capoose claims 706597 and 645063 partially overlap portions of legacy claims held by a third party.

None of the Capoose claims are known to overlap with areas of any Crown granted mineral claims. Three Capoose claims partially overlap no-staking reserves, and 16 claims partially overlap the Entiako Provincial Park.

4.3.3 Auro Claim Block

The Auro claim block lies southeast of the Blackwater claim block and contains 21 mineral claims totaling 22,591 hectares. All Auro claims are 100% held in the name of Artemis. The Auro mineral claims are summarized in Table 4-1.

No Auro claims are known to overlap any legacy or Crown granted mineral claims. One claim (831124) partially overlaps a no-staking reserve.

4.3.4 Key Claims

The Key claim block comprises 24 mineral claims located immediately south of the Blackwater and west of the Auro claim blocks. The claim block totals 8,854 hectares. All Key claims are 100% held in the name of Artemis. Key mineral claims are summarized in Table 4-1.

All claims in the Key claim block are cell claims with no predecessors.

No Key claims are known to overlap any legacy or Crown granted mineral claims. No Key claims overlap any no-staking reserves or parks.

4.3.5 Parlane Claims

The Parlane claim block comprises 62 mineral claims immediately north of the Blackwater and east of the Capoose claim blocks. The claim block totals 27,470 hectares. All Parlane claims are 100% held in the name of Artemis. Parlane mineral claims are summarized in Table 4-1.

No Parlane claims are known to overlap any legacy or Crown Granted mineral claims. No Parlane claims overlap any mineral reserves or parks.

4.3.6 RJK Claims

The RJK claim block comprises 40 mineral claims south and to the north-east of the Blackwater claim blocks. The claim block totals 17,006 hectares. All RJK claims are 100% held in the name of Artemis. RJK mineral claims are summarized in Table 4-1.

No RJK mineral claims are known to overlap any legacy or Crown granted mineral claims. No RJK mineral claims overlap any no-staking reserves or parks.

Table 4-1: Claims Table Listing

Title Number	Claim Name	Title Type	Title Sub Type	Map Number	Issue Date	Good To Date	Area (ha)	Block
646683	PRINCESS	Mineral	Claim	093F	2009/OCT/03	2022/AUG/29	407.2	Auro
745822	NG1	Mineral	Claim	093F	2010/APR/12	2022/AUG/29	485.5	Auro
745842	NG2	Mineral	Claim	093F	2010/APR/12	2022/AUG/29	485.5	Auro
745862	NG3	Mineral	Claim	093F	2010/APR/12	2022/AUG/29	465.9	Auro
745882	NG4	Mineral	Claim	093F	2010/APR/12	2022/AUG/29	465.9	Auro
745902	NG5	Mineral	Claim	093F	2010/APR/12	2022/AUG/29	465.8	Auro
745922	NG6	Mineral	Claim	093F	2010/APR/12	2022/AUG/29	485.7	Auro
745942	NG7	Mineral	Claim	093F	2010/APR/12	2022/AUG/29	485.2	Auro
745962	NG8	Mineral	Claim	093F	2010/APR/12	2022/AUG/29	485.7	Auro
745982	NG9	Mineral	Claim	093F	2010/APR/12	2022/AUG/29	485.4	Auro
746002	NG10	Mineral	Claim	093F	2010/APR/12	2022/AUG/29	485.7	Auro
746022	NG11	Mineral	Claim	093F	2010/APR/12	2022/AUG/29	485.7	Auro
746042	NG12	Mineral	Claim	093F	2010/APR/12	2022/AUG/29	466.5	Auro
746062	NG13	Mineral	Claim	093F	2010/APR/12	2022/AUG/29	486.0	Auro
746082	NG14	Mineral	Claim	093F	2010/APR/12	2022/AUG/29	466.5	Auro
746102	NG15	Mineral	Claim	093F	2010/APR/12	2022/AUG/29	466.0	Auro
746182	NG15	Mineral	Claim	093F	2010/APR/12	2022/AUG/29	388.8	Auro
746202	NG16	Mineral	Claim	093F	2010/APR/12	2022/AUG/29	330.4	Auro
831124	AURO PROPERTY	Mineral	Claim	093F	2010/AUG/05	2022/AUG/29	14,026.0	Auro
982702	BW BRIDGE	Mineral	Claim	093F	2012/APR/26	2022/AUG/29	194.1	Auro
1018105		Mineral	Claim	093F	2013/MAR/27	2022/AUG/29	77.5	Auro
503050	WHITEWATER	Mineral	Claim	093F	2005/JAN/13	2022/AUG/29	348.8	Blackwater
509273	GOT	Mineral	Claim	093F	2005/MAR/19	2022/AUG/29	484.4	Blackwater
509274	got2	Mineral	Claim	093F	2005/MAR/19	2022/AUG/29	38.8	Blackwater
509275	got3	Mineral	Claim	093F	2005/MAR/19	2022/AUG/29	19.4	Blackwater
515809		Mineral	Claim	093F	2005/JUL/01	2022/AUG/29	581.6	Blackwater
515810		Mineral	Claim	093F	2005/JUL/01	2022/AUG/29	349.0	Blackwater
536650	NIGHT FLIGHT	Mineral	Claim	093F	2006/JUL/06	2022/AUG/29	271.4	Blackwater
602167	BWD	Mineral	Claim	093F	2009/APR/05	2022/AUG/29	387.9	Blackwater
602168	BWD2	Mineral	Claim	093F	2009/APR/05	2022/AUG/29	310.3	Blackwater
607194	BLACKWATER 2	Mineral	Claim	093F	2009/JUL/08	2022/AUG/29	464.9	Blackwater
607195	BLACKWATER 1	Mineral	Claim	093F	2009/JUL/08	2022/AUG/29	348.7	Blackwater
630903	BW1	Mineral	Claim	093F	2009/SEP/09	2022/AUG/29	465.3	Blackwater
630944	BW2	Mineral	Claim	093F	2009/SEP/09	2022/AUG/29	251.9	Blackwater
630963	BW3	Mineral	Claim	093F	2009/SEP/09	2022/AUG/29	465.2	Blackwater
630983	BW4	Mineral	Claim	093F	2009/SEP/09	2022/AUG/29	387.3	Blackwater
630984	BW5	Mineral	Claim	093F	2009/SEP/09	2022/AUG/29	465.0	Blackwater
631003	BW6	Mineral	Claim	093F	2009/SEP/09	2022/AUG/29	484.1	Blackwater
631024	BW7	Mineral	Claim	093F	2009/SEP/09	2022/AUG/29	445.5	Blackwater
631043	BW8	Mineral	Claim	093F	2009/SEP/09	2022/AUG/29	464.7	Blackwater
636583	KASSY 1	Mineral	Claim	093F	2009/SEP/18	2022/AUG/29	464.5	Blackwater
636603	KASSY 2	Mineral	Claim	093F	2009/SEP/18	2022/AUG/29	464.3	Blackwater
636604	KASSY 3	Mineral	Claim	093F	2009/SEP/18	2022/AUG/29	464.0	Blackwater
636623	KASSY 4	Mineral	Claim	093F	2009/SEP/18	2022/AUG/29	463.8	Blackwater
636643	KASSY 5	Mineral	Claim	093F	2009/SEP/18	2022/AUG/29	483.3	Blackwater
636644	KASSY 6	Mineral	Claim	093F	2009/SEP/18	2022/AUG/29	483.5	Blackwater
636663	KASSY 7	Mineral	Claim	093F	2009/SEP/18	2022/AUG/29	290.2	Blackwater
636683	RIGHT STUFF 1	Mineral	Claim	093F	2009/SEP/18	2022/AUG/29	464.9	Blackwater
636684	RIGHT STUFF 2	Mineral	Claim	093F	2009/SEP/18	2022/AUG/29	464.9	Blackwater
636703	RIGHT STUFF 3	Mineral	Claim	093F	2009/SEP/18	2022/AUG/29	464.9	Blackwater
636723	RIGHT STUFF 4	Mineral	Claim	093F	2009/SEP/18	2022/AUG/29	464.4	Blackwater
636724	RIGHT STUFF	Mineral	Claim	093F	2009/SEP/18	2022/AUG/29	464.6	Blackwater
636725	RIGHT STUFF 6	Mineral	Claim	093F	2009/SEP/18	2022/AUG/29	484.2	Blackwater
636727	RIGHT STUFF 7	Mineral	Claim	093F	2009/SEP/18	2022/AUG/29	484.4	Blackwater
636743	RIGHT STUFF 8	Mineral	Claim	093F	2009/SEP/18	2022/AUG/29	483.8	Blackwater
636763	RIGHT STUFF 9	Mineral	Claim	093F	2009/SEP/18	2022/AUG/29	464.8	Blackwater

Title Number	Claim Name	Title Type	Title Sub Type	Map Number	Issue Date	Good To Date	Area (ha)	Block
636764	RIGHT STUFF 10	Mineral	Claim	093F	2009/SEP/18	2022/AUG/29	484.6	Blackwater
636765	RIGHT STUFF 11	Mineral	Claim	093F	2009/SEP/18	2022/AUG/29	465.1	Blackwater
636766	RIGHT STUFF12	Mineral	Claim	093F	2009/SEP/18	2022/AUG/29	464.8	Blackwater
636767	RIGHT STUFF13	Mineral	Claim	093F	2009/SEP/18	2022/AUG/29	464.4	Blackwater
637203	OZZY	Mineral	Claim	093F	2009/SEP/19	2022/AUG/29	484.4	Blackwater
637205	BABY JANE	Mineral	Claim	093F	2009/SEP/19	2022/AUG/29	464.6	Blackwater
637206	DAVID DALE	Mineral	Claim	093F	2009/SEP/19	2022/AUG/29	464.6	Blackwater
640804	PUREANDY	Mineral	Claim	093F	2009/SEP/25	2022/AUG/29	310.2	Blackwater
641685	RICHFIELD-ADJACENTCC	Mineral	Claim	093F	2009/SEP/26	2022/AUG/29	445.6	Blackwater
642043	BW	Mineral	Claim	093F	2009/SEP/27	2022/AUG/29	232.6	Blackwater
642063	BW 2	Mineral	Claim	093F	2009/SEP/27	2022/AUG/29	232.7	Blackwater
642064	BW3	Mineral	Claim	093F	2009/SEP/27	2022/AUG/29	310.3	Blackwater
834367	RICH 1	Mineral	Claim	093F	2010/SEP/27	2022/AUG/29	484.1	Blackwater
834371	DAVIDSON	Mineral	Claim	093F	2010/SEP/27	2022/AUG/29	425.9	Blackwater
834533	DAVIDSON 1	Mineral	Claim	093F	2010/SEP/29	2022/AUG/29	77.4	Blackwater
834534	DAVIDSON 2	Mineral	Claim	093F	2010/SEP/29	2022/AUG/29	406.3	Blackwater
834923	DAVIDSON 3	Mineral	Claim	093F	2010/OCT/02	2022/AUG/29	483.6	Blackwater
834924	DAVIDSON 4	Mineral	Claim	093F	2010/OCT/02	2022/AUG/29	483.5	Blackwater
834926	DAVIDSON 5	Mineral	Claim	093F	2010/OCT/02	2022/AUG/29	483.5	Blackwater
834948		Mineral	Claim	093F	2010/OCT/03	2022/AUG/29	484.7	Blackwater
834998	RICH 2	Mineral	Claim	093F	2010/OCT/04	2022/AUG/29	426.3	Blackwater
835005		Mineral	Claim	093F	2010/OCT/04	2022/AUG/29	465.5	Blackwater
835009		Mineral	Claim	093F	2010/OCT/04	2022/AUG/29	271.3	Blackwater
835011		Mineral	Claim	093F	2010/OCT/04	2022/AUG/29	484.7	Blackwater
835012		Mineral	Claim	093F	2010/OCT/04	2022/AUG/29	484.5	Blackwater
835013		Mineral	Claim	093F	2010/OCT/04	2022/AUG/29	174.4	Blackwater
835014	DAVE	Mineral	Claim	093F	2010/OCT/04	2022/AUG/29	116.2	Blackwater
835016		Mineral	Claim	093F	2010/OCT/04	2022/AUG/29	232.8	Blackwater
835019		Mineral	Claim	093F	2010/OCT/04	2022/AUG/29	387.8	Blackwater
835020		Mineral	Claim	093F	2010/OCT/04	2022/AUG/29	329.5	Blackwater
835021	BW WEST	Mineral	Claim	093F	2010/OCT/04	2022/AUG/29	387.9	Blackwater
835022	BW WEST 2	Mineral	Claim	093F	2010/OCT/04	2022/AUG/29	368.6	Blackwater
835023		Mineral	Claim	093F	2010/OCT/04	2022/AUG/29	465.2	Blackwater
835024	Dave2	Mineral	Claim	093F	2010/OCT/04	2023/FEB/04	213.3	Blackwater
835025	BW WEST2	Mineral	Claim	093F	2010/OCT/04	2022/AUG/29	38.8	Blackwater
920729	JONBLK	Mineral	Claim	093F	2011/OCT/21	2022/AUG/29	251.2	Blackwater
940115	NOREADD	Mineral	Claim	093F	2012/JAN/06	2022/AUG/29	77.3	Blackwater
1024945	BW NE	Mineral	Claim	093F	2014/JAN/09	2022/AUG/29	1,817.2	Blackwater
1024947	BW-N 1	Mineral	Claim	093F	2014/JAN/09	2022/AUG/29	96.7	Blackwater
1024948	BW-N 2	Mineral	Claim	093F	2014/JAN/09	2022/AUG/29	599.6	Blackwater
1024956	BW-N 3	Mineral	Claim	093F	2014/JAN/09	2022/AUG/29	561.1	Blackwater
238045	CAP	Mineral	Claim	093F025	1978/SEP/18	2022/AUG/29	100.0	Capoose
512838		Mineral	Claim	093F	2005/MAY/17	2022/AUG/29	811.9	Capoose
534364	JAG-1	Mineral	Claim	093F	2006/MAY/24	2022/AUG/29	482.8	Capoose
534365	JAG-2	Mineral	Claim	093F	2006/MAY/24	2022/AUG/29	482.9	Capoose
534366	JAG-3	Mineral	Claim	093F	2006/MAY/24	2022/AUG/29	482.6	Capoose
534367	JAG-4	Mineral	Claim	093F	2006/MAY/24	2022/AUG/29	289.7	Capoose
552493	NE CAPOOSE	Mineral	Claim	093F	2007/FEB/22	2022/AUG/29	483.1	Capoose
552494	NE CAPOOSE 2	Mineral	Claim	093F	2007/FEB/22	2022/AUG/29	483.0	Capoose
552495	E CAPOOSE	Mineral	Claim	093F	2007/FEB/22	2022/AUG/29	483.3	Capoose
552497	NE CAPOOSE3	Mineral	Claim	093F	2007/FEB/22	2022/AUG/29	483.0	Capoose
553489	PAW	Mineral	Claim	093F	2007/MAR/03	2022/AUG/29	19.4	Capoose
555053	CAP	Mineral	Claim	093F	2007/MAR/26	2022/AUG/29	251.3	Capoose
557495	JAG-5	Mineral	Claim	093F	2007/APR/23	2022/AUG/29	482.7	Capoose
557496	JAG-6	Mineral	Claim	093F	2007/APR/23	2022/AUG/29	482.5	Capoose
564372	CAPOOSE S	Mineral	Claim	093F	2007/AUG/09	2022/AUG/29	464.2	Capoose
564373	CAPOOSE SW	Mineral	Claim	093F	2007/AUG/09	2022/AUG/29	464.2	Capoose

Title Number	Claim Name	Title Type	Title Sub Type	Map Number	Issue Date	Good To Date	Area (ha)	Block
564375	CAPOOSE SW2	Mineral	Claim	093F	2007/AUG/09	2022/AUG/29	483.5	Capoose
564376	CAPOOSE E2	Mineral	Claim	093F	2007/AUG/09	2022/AUG/29	483.5	Capoose
564377	CAPOOSE E3	Mineral	Claim	093F	2007/AUG/09	2022/AUG/29	483.2	Capoose
580086	CAPOOSE NORTH	Mineral	Claim	093F	2008/APR/01	2022/AUG/29	77.3	Capoose
598000	BUCK	Mineral	Claim	093F	2009/JAN/26	2022/AUG/29	38.7	Capoose
601527	FAWN	Mineral	Claim	093F	2009/MAR/23	2022/AUG/29	19.4	Capoose
606724	FAWN	Mineral	Claim	093F	2009/JUN/27	2022/AUG/29	174.3	Capoose
606728	MALAPUT E-W	Mineral	Claim	093F	2009/JUN/27	2022/AUG/29	96.9	Capoose
617183	BUCK 2	Mineral	Claim	093F	2009/AUG/10	2022/AUG/29	96.9	Capoose
625583	M-1	Mineral	Claim	093F	2009/AUG/29	2022/AUG/29	484.1	Capoose
625603	M-2	Mineral	Claim	093F	2009/AUG/29	2022/AUG/29	484.2	Capoose
625623	M-3	Mineral	Claim	093F	2009/AUG/29	2022/AUG/29	484.0	Capoose
625624	M-4	Mineral	Claim	093F	2009/AUG/29	2022/AUG/29	464.5	Capoose
625625		Mineral	Claim	093F	2009/AUG/29	2022/AUG/29	483.7	Capoose
641983	FAWN	Mineral	Claim	093F	2009/SEP/27	2022/AUG/29	19.4	Capoose
641984	FAWN 2	Mineral	Claim	093F	2009/SEP/27	2022/AUG/29	154.9	Capoose
642544	FAWNIE DOME	Mineral	Claim	093F	2009/SEP/28	2022/AUG/29	116.1	Capoose
642564	FD 2	Mineral	Claim	093F	2009/SEP/28	2022/AUG/29	464.4	Capoose
642565	FD 3	Mineral	Claim	093F	2009/SEP/28	2022/AUG/29	348.4	Capoose
642583	FD 4	Mineral	Claim	093F	2009/SEP/28	2022/AUG/29	309.6	Capoose
642603	TOP LAKE	Mineral	Claim	093F	2009/SEP/28	2022/AUG/29	174.1	Capoose
643103	BUCK 1	Mineral	Claim	093F	2009/SEP/29	2022/AUG/29	484.1	Capoose
643104	BUCK 2	Mineral	Claim	093F	2009/SEP/29	2022/AUG/29	445.5	Capoose
643106	BUCK 3	Mineral	Claim	093F	2009/SEP/29	2022/AUG/29	406.6	Capoose
643107	BUCK 4	Mineral	Claim	093F	2009/SEP/29	2022/AUG/29	483.9	Capoose
643108	BUCK 5	Mineral	Claim	093F	2009/SEP/29	2022/AUG/29	483.9	Capoose
643109	BUCK 6	Mineral	Claim	093F	2009/SEP/29	2022/AUG/29	483.7	Capoose
643110	BUCK 7	Mineral	Claim	093F	2009/SEP/29	2022/AUG/29	483.7	Capoose
643123	BUCK 8	Mineral	Claim	093F	2009/SEP/29	2022/AUG/29	484.1	Capoose
643323	TOP	Mineral	Claim	093F	2009/SEP/29	2022/AUG/29	309.4	Capoose
644244	CAPOOSE M6	Mineral	Claim	093F	2009/SEP/30	2022/AUG/29	484.3	Capoose
644283	CAPOOSE M7	Mineral	Claim	093F	2009/SEP/30	2022/AUG/29	484.3	Capoose
644285	CAPOOSE M8	Mineral	Claim	093F	2009/SEP/30	2022/AUG/29	465.0	Capoose
644323	CAPOOSE M9	Mineral	Claim	093F	2009/SEP/30	2022/AUG/29	464.7	Capoose
644363	CAPOOSE M10	Mineral	Claim	093F	2009/SEP/30	2022/AUG/29	310.0	Capoose
645063	CAPOOSE M11	Mineral	Claim	093F	2009/SEP/30	2022/AUG/29	465.0	Capoose
645064	CAPOOSE M12	Mineral	Claim	093F	2009/SEP/30	2022/AUG/29	465.1	Capoose
645065	CAPOOSE M13	Mineral	Claim	093F	2009/SEP/30	2022/AUG/29	426.4	Capoose
645066	CAPOOSE M14	Mineral	Claim	093F	2009/SEP/30	2022/AUG/29	232.6	Capoose
649243	JAG-8	Mineral	Claim	093F	2009/OCT/08	2022/AUG/29	483.1	Capoose
704807	PAWING	Mineral	Claim	093F	2010/JAN/26	2022/AUG/29	213.3	Capoose
704817	PAWS	Mineral	Claim	093F	2010/JAN/26	2022/AUG/29	213.3	Capoose
704825	FAWN WEST	Mineral	Claim	093F	2010/JAN/26	2022/AUG/29	387.2	Capoose
704826	FW 2	Mineral	Claim	093F	2010/JAN/26	2022/AUG/29	464.6	Capoose
704827	FW 3	Mineral	Claim	093F	2010/JAN/26	2022/AUG/29	406.6	Capoose
704828	FW 4	Mineral	Claim	093F	2010/JAN/26	2022/AUG/29	387.4	Capoose
704829	FW 5	Mineral	Claim	093F	2010/JAN/26	2022/AUG/29	387.0	Capoose
704830	FW 6	Mineral	Claim	093F	2010/JAN/26	2022/AUG/29	193.6	Capoose
704854	FW 7	Mineral	Claim	093F	2010/JAN/27	2022/AUG/29	464.6	Capoose
704855	FW 8	Mineral	Claim	093F	2010/JAN/27	2022/AUG/29	309.6	Capoose
704863	FW 9	Mineral	Claim	093F	2010/JAN/27	2022/AUG/29	445.1	Capoose
706011	FW 10	Mineral	Claim	093F	2010/FEB/11	2022/AUG/29	193.7	Capoose
706593	CPN1	Mineral	Claim	093F	2010/FEB/19	2022/AUG/29	482.9	Capoose
706594	CPN2	Mineral	Claim	093F	2010/FEB/19	2022/AUG/29	482.6	Capoose
706595	CPN3	Mineral	Claim	093F	2010/FEB/19	2022/AUG/29	444.0	Capoose
706596	CPN4	Mineral	Claim	093F	2010/FEB/19	2022/AUG/29	328.1	Capoose
706597	CPW1	Mineral	Claim	093F	2010/FEB/19	2022/AUG/29	484.4	Capoose
706598	CPW2	Mineral	Claim	093F	2010/FEB/19	2022/AUG/29	484.3	Capoose

Title Number	Claim Name	Title Type	Title Sub Type	Map Number	Issue Date	Good To Date	Area (ha)	Block
706599	CPW3	Mineral	Claim	093F	2010/FEB/19	2022/AUG/29	484.3	Capoose
706600	CPW4	Mineral	Claim	093F	2010/FEB/19	2022/AUG/29	484.3	Capoose
706602	CPW5	Mineral	Claim	093F	2010/FEB/19	2022/AUG/29	484.0	Capoose
706603	CPW6	Mineral	Claim	093F	2010/FEB/19	2022/AUG/29	483.8	Capoose
706605	CPW7	Mineral	Claim	093F	2010/FEB/19	2022/AUG/29	483.9	Capoose
706606	CPW8	Mineral	Claim	093F	2010/FEB/19	2022/AUG/29	484.0	Capoose
706607	CPW9	Mineral	Claim	093F	2010/FEB/19	2022/AUG/29	484.2	Capoose
706608	CPW9	Mineral	Claim	093F	2010/FEB/19	2022/AUG/29	484.4	Capoose
706609	CPW10	Mineral	Claim	093F	2010/FEB/19	2022/AUG/29	484.5	Capoose
706610	CPW11	Mineral	Claim	093F	2010/FEB/19	2022/AUG/29	484.7	Capoose
706612	CPW12	Mineral	Claim	093F	2010/FEB/19	2022/AUG/29	484.5	Capoose
706613	CPW13	Mineral	Claim	093F	2010/FEB/19	2022/AUG/29	484.5	Capoose
706614	CPW14	Mineral	Claim	093F	2010/FEB/19	2022/AUG/29	484.7	Capoose
706615	CPW15	Mineral	Claim	093F	2010/FEB/19	2022/AUG/29	484.8	Capoose
706616	CPW16	Mineral	Claim	093F	2010/FEB/19	2022/AUG/29	485.0	Capoose
706617	CPW17	Mineral	Claim	093F	2010/FEB/19	2022/AUG/29	485.2	Capoose
706618	CPW18	Mineral	Claim	093F	2010/FEB/19	2022/AUG/29	485.4	Capoose
706619	CPW19	Mineral	Claim	093F	2010/FEB/19	2022/AUG/29	485.4	Capoose
706620	CPW20	Mineral	Claim	093F	2010/FEB/19	2022/AUG/29	485.2	Capoose
706621	CPW21	Mineral	Claim	093F	2010/FEB/19	2022/AUG/29	485.0	Capoose
706622	CPW22	Mineral	Claim	093F	2010/FEB/19	2022/AUG/29	484.9	Capoose
706623	CPW23	Mineral	Claim	093F	2010/FEB/19	2022/AUG/29	485.2	Capoose
706625	CPW24	Mineral	Claim	093F	2010/FEB/19	2022/AUG/29	485.0	Capoose
706626	CPW25	Mineral	Claim	093F	2010/FEB/19	2022/AUG/29	485.2	Capoose
706627	CPW26	Mineral	Claim	093F	2010/FEB/19	2022/AUG/29	485.4	Capoose
706628	CPW27	Mineral	Claim	093F	2010/FEB/19	2022/AUG/29	485.5	Capoose
706629	CPNW1	Mineral	Claim	093F	2010/FEB/19	2022/AUG/29	290.1	Capoose
706630	CPNW2	Mineral	Claim	093F	2010/FEB/19	2022/AUG/29	154.6	Capoose
706638	PAWS 2	Mineral	Claim	093F	2010/FEB/19	2022/AUG/29	407.3	Capoose
843656	JOHNNY NORTH	Mineral	Claim	093F	2011/JAN/20	2022/AUG/29	213.2	Capoose
843657	JOHNNY NW	Mineral	Claim	093F	2011/JAN/20	2022/AUG/29	407.0	Capoose
843658	JOHNNY W	Mineral	Claim	093F	2011/JAN/20	2022/AUG/29	310.0	Capoose
564994	KEY 1	Mineral	Claim	093F	2007/AUG/24	2022/AUG/29	485.2	Key
564995	KEY 2	Mineral	Claim	093F	2007/AUG/24	2022/AUG/29	485.2	Key
564996	KEY 3	Mineral	Claim	093F	2007/AUG/24	2022/AUG/29	485.2	Key
564997	KEY 4	Mineral	Claim	093F	2007/AUG/24	2022/AUG/29	466.0	Key
564998	KEY 5	Mineral	Claim	093F	2007/AUG/24	2022/AUG/29	388.3	Key
564999	KEY 6	Mineral	Claim	093F	2007/AUG/24	2022/AUG/29	388.2	Key
565000	KEY 7	Mineral	Claim	093F	2007/AUG/24	2022/AUG/29	116.5	Key
565001	KEY 8	Mineral	Claim	093F	2007/AUG/24	2022/AUG/29	97.1	Key
589167	LOCK 1	Mineral	Claim	093F	2007/AUG/24	2022/AUG/29	485.6	Key
589177	LOCK 2	Mineral	Claim	093F	2007/AUG/24	2022/AUG/29	485.8	Key
589183	LOCK 3	Mineral	Claim	093F	2007/AUG/24	2022/AUG/29	484.9	Key
589231	LOCK 4	Mineral	Claim	093F	2007/AUG/24	2022/AUG/29	485.2	Key
589232	LOCK 5	Mineral	Claim	093F	2007/AUG/24	2022/AUG/29	485.4	Key
589234	LOCK 6	Mineral	Claim	093F	2007/AUG/24	2022/AUG/29	388.4	Key
589236	LOCK 7	Mineral	Claim	093F	2007/AUG/24	2022/AUG/29	466.2	Key
589238	LOCK 8	Mineral	Claim	093F	2007/AUG/24	2022/AUG/29	233.1	Key
589241	LOCK 9	Mineral	Claim	093F	2007/AUG/24	2022/AUG/29	407.6	Key
589242	LOCK 10	Mineral	Claim	093F	2007/AUG/24	2022/AUG/29	465.7	Key
589243	LOCK 11	Mineral	Claim	093F	2007/AUG/24	2022/AUG/29	194.0	Key
589244	LOCK 12	Mineral	Claim	093F	2007/AUG/24	2022/AUG/29	388.6	Key
642003	YELLOW and BLACK	Mineral	Claim	093F	2007/AUG/24	2022/AUG/29	233.2	Key
642004	YELLOW and BLACK	Mineral	Claim	093F	2007/AUG/24	2022/AUG/29	464.7	Parlane
642023	BLACK	Mineral	Claim	093F	2007/AUG/24	2022/AUG/29	483.9	Parlane
642024	YELLOW	Mineral	Claim	093F	2007/AUG/24	2022/AUG/29	483.8	Parlane

Title Number	Claim Name	Title Type	Title Sub Type	Map Number	Issue Date	Good To Date	Area (ha)	Block
694043		Mineral	Claim	093F	2007/AUG/24	2022/AUG/29	464.6	Parlane
694044		Mineral	Claim	093F	2007/AUG/24	2022/AUG/29	445.0	Parlane
694045		Mineral	Claim	093F	2007/AUG/24	2022/AUG/29	483.8	Parlane
694046		Mineral	Claim	093F	2007/AUG/24	2022/AUG/29	464.6	Parlane
694063		Mineral	Claim	093F	2007/AUG/24	2022/AUG/29	445.0	Parlane
694064		Mineral	Claim	093F	2007/AUG/24	2022/AUG/29	483.8	Parlane
694065		Mineral	Claim	093F	2007/AUG/24	2022/AUG/29	483.7	Parlane
694066		Mineral	Claim	093F	2007/AUG/24	2022/AUG/29	464.1	Parlane
694083		Mineral	Claim	093F	2007/AUG/24	2022/AUG/29	483.6	Parlane
694084		Mineral	Claim	093F	2007/AUG/24	2022/AUG/29	464.2	Parlane
694085		Mineral	Claim	093F	2010/JAN/04	2022/AUG/29	483.6	Parlane
694086		Mineral	Claim	093F	2010/JAN/04	2022/AUG/29	464.2	Parlane
694087		Mineral	Claim	093F	2010/JAN/04	2022/AUG/29	463.9	Parlane
694088		Mineral	Claim	093F	2010/JAN/04	2022/AUG/29	464.0	Parlane
694089		Mineral	Claim	093F	2010/JAN/04	2022/AUG/29	464.1	Parlane
694090		Mineral	Claim	093F	2010/JAN/04	2022/AUG/29	463.9	Parlane
694103		Mineral	Claim	093F	2010/JAN/04	2022/AUG/29	483.2	Parlane
694123		Mineral	Claim	093F	2010/JAN/04	2022/AUG/29	464.1	Parlane
694143		Mineral	Claim	093F	2010/JAN/04	2022/AUG/29	444.6	Parlane
694144		Mineral	Claim	093F	2010/JAN/04	2022/AUG/29	464.2	Parlane
694145		Mineral	Claim	093F	2010/JAN/04	2022/AUG/29	463.9	Parlane
694146		Mineral	Claim	093F	2010/JAN/04	2022/AUG/29	425.4	Parlane
694147		Mineral	Claim	093F	2010/JAN/04	2022/AUG/29	463.8	Parlane
694148		Mineral	Claim	093F	2010/JAN/04	2022/AUG/29	483.0	Parlane
694163		Mineral	Claim	093F	2010/JAN/04	2022/AUG/29	348.0	Parlane
694183		Mineral	Claim	093F	2010/JAN/04	2022/AUG/29	463.6	Parlane
694184		Mineral	Claim	093F	2010/JAN/04	2022/AUG/29	463.6	Parlane
694185		Mineral	Claim	093F	2010/JAN/04	2022/AUG/29	463.6	Parlane
694186		Mineral	Claim	093F	2010/JAN/04	2022/AUG/29	463.6	Parlane
694187		Mineral	Claim	093F	2010/JAN/04	2022/AUG/29	463.6	Parlane
694287		Mineral	Claim	093F	2010/JAN/04	2022/AUG/29	463.6	Parlane
713362	KL1	Mineral	Claim	093F	2010/MAR/04	2022/AUG/29	482.7	Parlane
713382	KL2	Mineral	Claim	093F	2010/MAR/04	2022/AUG/29	482.7	Parlane
713402	KL3	Mineral	Claim	093F	2010/MAR/04	2022/AUG/29	482.7	Parlane
713422	KL4	Mineral	Claim	093F	2010/MAR/04	2022/AUG/29	482.7	Parlane
713442	KL6	Mineral	Claim	093F	2010/MAR/04	2022/AUG/29	444.1	Parlane
713462	KL7	Mineral	Claim	093F	2010/MAR/04	2022/AUG/29	463.6	Parlane
713482	KL8	Mineral	Claim	093F	2010/MAR/04	2022/AUG/29	463.4	Parlane
713502	KL9	Mineral	Claim	093F	2010/MAR/04	2022/AUG/29	463.6	Parlane
713522	KL10	Mineral	Claim	093F	2010/MAR/04	2022/AUG/29	347.6	Parlane
713542	KL11	Mineral	Claim	093F	2010/MAR/04	2022/AUG/29	463.2	Parlane
713562	KL12	Mineral	Claim	093F	2010/MAR/04	2022/AUG/29	463.2	Parlane
713582	KL13	Mineral	Claim	093F	2010/MAR/04	2022/AUG/29	463.2	Parlane
713602	KL14	Mineral	Claim	093F	2010/MAR/04	2022/AUG/29	463.2	Parlane
713622	KL15	Mineral	Claim	093F	2010/MAR/04	2022/AUG/29	463.2	Parlane
713642	KL16	Mineral	Claim	093F	2010/MAR/04	2022/AUG/29	463.2	Parlane
713662	KL17	Mineral	Claim	093F	2010/MAR/04	2022/AUG/29	463.0	Parlane
713682	KL18	Mineral	Claim	093F	2010/MAR/04	2022/AUG/29	463.0	Parlane
713702	KL19	Mineral	Claim	093F	2010/MAR/04	2022/AUG/29	463.0	Parlane
713722	KL20	Mineral	Claim	093F	2010/MAR/04	2022/AUG/29	463.0	Parlane
713742	KL21	Mineral	Claim	093F	2010/MAR/04	2022/AUG/29	463.0	Parlane
713782	KL22	Mineral	Claim	093F	2010/MAR/04	2022/AUG/29	463.0	Parlane
713802	KL22	Mineral	Claim	093F	2010/MAR/04	2022/AUG/29	463.0	Parlane
713822	KL23	Mineral	Claim	093F	2010/MAR/04	2022/AUG/29	462.9	Parlane
713842	KL24	Mineral	Claim	093F	2010/MAR/04	2022/AUG/29	462.9	Parlane
713862	KL25	Mineral	Claim	093F	2010/MAR/04	2022/AUG/29	482.1	Parlane
713882	KL26	Mineral	Claim	093F	2010/MAR/04	2022/AUG/29	482.2	Parlane
713902	KL27	Mineral	Claim	093F	2010/MAR/04	2022/AUG/29	482.1	Parlane

Title Number	Claim Name	Title Type	Title Sub Type	Map Number	Issue Date	Good To Date	Area (ha)	Block
713922	KL28	Mineral	Claim	093F	2010/MAR/04	2022/AUG/29	462.7	Parlane
1046035	THE CUB	Mineral	Claim	093F	2016/AUG/18	2022/AUG/29	38.5	Parlane
1046802	THE CUB 2	Mineral	Claim	093F	2016/SEP/19	2022/AUG/29	38.6	Parlane
1046869	THE CUB 3	Mineral	Claim	093F	2016/SEP/22	2022/AUG/29	57.8	Parlane
694164		Mineral	Claim	093F	2010/JAN/04	2023/FEB/06	464.7	Parlane
694188		Mineral	Claim	093F	2010/JAN/04	2023/FEB/06	483.8	Parlane
694203		Mineral	Claim	093F	2010/JAN/04	2023/FEB/06	483.8	Parlane
694204		Mineral	Claim	093F	2010/JAN/04	2023/FEB/06	484.1	Parlane
694205		Mineral	Claim	093F	2010/JAN/04	2023/FEB/06	483.8	Parlane
694206		Mineral	Claim	093F	2010/JAN/04	2023/FEB/06	464.7	Parlane
694207		Mineral	Claim	093F	2010/JAN/04	2023/FEB/04	387.1	RJK
694208		Mineral	Claim	093F	2010/JAN/04	2023/FEB/04	464.4	RJK
694209		Mineral	Claim	093F	2010/JAN/04	2023/FEB/04	483.6	RJK
694210		Mineral	Claim	093F	2010/JAN/04	2023/FEB/04	464.1	RJK
694223		Mineral	Claim	093F	2010/JAN/04	2023/FEB/04	464.1	RJK
694224		Mineral	Claim	093F	2010/JAN/04	2023/FEB/04	290.0	RJK
694225		Mineral	Claim	093F	2010/JAN/04	2023/FEB/04	193.4	RJK
694243		Mineral	Claim	093F	2010/JAN/04	2023/FEB/04	485.8	RJK
694245		Mineral	Claim	093F	2010/JAN/04	2023/FEB/04	485.6	RJK
694263		Mineral	Claim	093F	2010/JAN/04	2023/FEB/04	485.7	RJK
694264		Mineral	Claim	093F	2010/JAN/04	2023/FEB/04	485.5	RJK
694265		Mineral	Claim	093F	2010/JAN/04	2023/FEB/04	466.3	RJK
694283		Mineral	Claim	093F	2010/JAN/04	2023/FEB/04	466.3	RJK
694284		Mineral	Claim	093F	2010/JAN/04	2023/FEB/04	466.1	RJK
694285		Mineral	Claim	093F	2010/JAN/04	2023/FEB/04	427.1	RJK
694286		Mineral	Claim	093F	2010/JAN/04	2023/FEB/04	465.9	RJK
694288		Mineral	Claim	093F	2010/JAN/04	2023/FEB/04	485.2	RJK
694289		Mineral	Claim	093F	2010/JAN/04	2023/FEB/04	485.2	RJK
694290		Mineral	Claim	093F	2010/JAN/04	2023/FEB/04	194.1	RJK
694291		Mineral	Claim	093F	2010/JAN/04	2023/FEB/04	485.2	RJK
694292		Mineral	Claim	093F	2010/JAN/04	2023/FEB/04	465.9	RJK
694293		Mineral	Claim	093F	2010/JAN/04	2023/FEB/04	484.6	RJK
694294		Mineral	Claim	093F	2010/JAN/04	2023/FEB/04	484.9	RJK
694295		Mineral	Claim	093F	2010/JAN/04	2023/FEB/04	465.4	RJK
694296		Mineral	Claim	093F	2010/JAN/04	2023/FEB/04	465.6	RJK
835434	JONECHAKO1	Mineral	Claim	093F	2010/OCT/08	2023/FEB/04	386.8	RJK
835436	JONECHAKO2	Mineral	Claim	093F	2010/OCT/08	2023/FEB/04	193.4	RJK
835527	JONECHAKO3	Mineral	Claim	093F	2010/OCT/08	2023/FEB/04	290.0	RJK
1015566	RJ1	Mineral	Claim	093F	2010/OCT/08	2022/AUG/29	174.9	RJK
1015573	RJK4	Mineral	Claim	093F	2010/OCT/08	2022/AUG/29	311.0	RJK
1015575	RJK5	Mineral	Claim	093F	2010/OCT/08	2022/AUG/29	563.5	RJK
1015577	RJK6	Mineral	Claim	093F	2010/OCT/08	2022/AUG/29	602.6	RJK
1015578	RJK8	Mineral	Claim	093F	2010/OCT/08	2022/AUG/29	466.6	RJK
1015579	RJK9	Mineral	Claim	093F	2012/DEC/31	2022/AUG/29	155.5	RJK

4.4 Surface Rights

Artemis, as the holder of mineral claims (or of a mining lease, once it is obtained), 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 claim blocks. Therefore, any mine infrastructure does not require additional land tenures if it is located on a mining lease and the surface is owned by the Crown. In the case of the Project, the Crown owns the surface in the proposed mine site area.

If Artemis requires exclusive possession of certain areas, then additional rights under the Land Act, such as a surface lease or acquisition of the fee simple may be required.

Once Artemis holds a mining lease that is on unreserved land owned by the government (if it is not lawfully occupied for a purpose other than for mining, and is not protected heritage property), then Artemis could apply for certification by the Minister of Mines and Energy that the surface rights over that area are required by Artemis for the purposes of a mining activity. If that certification is made, then Artemis will have the right to obtain a surface tenure under the Land Act on the terms and conditions set by the minister responsible for the Land Act.

In addition, the Project will require the construction and operation of certain infrastructure, parts of which are to be located outside of the Project area. The key components of this infrastructure are a mine access road, a water pipeline, and an electric transmission line (collectively, the Linear Infrastructure). As currently planned, the proposed routes and locations of this infrastructure appear to be located almost entirely on provincial Crown land. As such, the Linear Infrastructure will require that Artemis obtains additional surface rights from the Crown. In most cases, a surface tenure under the Land Act will be required. There are various types of tenures that can be obtained, including temporary permits, licenses, leases, rights of way, and fee simple interests. The type of surface tenure desired for each component will be determined on a case by case basis.

A review of surface rights in the vicinity of the Linear Infrastructure was undertaken in September 2020. The review used searches of the MTO system, the Integrated Land and Resources Registry (ILRR) system, the BC Government access tool for online retrieval (GATOR), and the Land Title Office, all of which are maintained by the Government of British Columbia. This review identified private parcels; a Land Act licence, rights of way, reserves/notations and a transfer of administration/control area; grazing tenures; forest tenures; a forest recreation sites; traplines; guide outfitter areas; a wildlife management area; an agriculture land reserve; a Land Act reserve, Range Act interests, and third-party mineral tenures overlapping or in close proximity to the proposed electrical transmission line route. The review also identified grazing tenures, forest tenures, traplines, and guide outfitter areas overlapping all elements of the Linear Infrastructure; a forest recreation site overlapping the proposed water pipeline route. The review found no Indian Reserves, federal parks, Ecological Reserves, Protected Areas, Wildlife Habitat Areas, placer mineral tenures, coal tenures, geothermal resource tenures, petroleum and natural gas tenures overlapping the reserves or the Linear Infrastructure.

4.5 Artemis / New Gold Purchase Agreement

The purchase agreement included the following considerations:

- a C$140,000,000 initial payment that was paid to New Gold on the date of acquisition;
- 7,407,407 common shares of Artemis that were issued to New Gold on the date of acquisition;
- a C$50,000,000 cash payment to New Gold that was settled on 23 August 2021; and
- a LOM gold stream with the following attributes:
  - New Gold will receive a percentage of gold production from the Blackwater Project as follows: 8% until 279,908 refined gold ounces are delivered to and purchased by New Gold, then 4% thereafter for the LOM.
  - New Gold will pay a purchase price equal to 35% of the US$ spot price for the gold ounces received. The 65% discount given will be recorded as an increase to the cost of the asset when incurred as variable consideration for the Acquisition.

- The Gold Stream includes a delayed construction/production penalty clause whereby, in the event the Blackwater mineral processing facility has not achieved an average of at least 80% of nameplate capacity for a period of 60 days prior to each of the 7th, 8th, and 9th anniversary dates of closing (being August 21, 2020, i.e., 7th anniversary being August 21, 2027, 8th anniversary being August 21, 2028, and 9th anniversary being August 21, 2029), the Company will be required to make penalty payments to New Gold in the amount of C$28,000,000 per annual deadline missed, up to a maximum of C$84,000,000

New Gold maintains a security interest over the Project in connection with the gold stream agreement.

4.6 Royalties and Encumbrances

Artemis' 100% interest in the Property is subject to a number of net smelter return (NSR) agreements. The majority of these NSRs do not affect the proposed mining operations area. The only NSR royalties that affect the proposed open pit operations are the Dave and Jarrit Options discussed under the Blackwater claims block.

4.6.1 Blackwater Claims Block Agreements and Encumbrances

The Blackwater claim block is subject to four NSR agreements. No other material encumbrances that are recorded against the Blackwater claims that are still active have been identified.

4.6.1.1 Dave Option

A 1.5% NSR royalty is payable on mineral claim 515809 (Dave Claim). The claim covers a portion of the Blackwater deposit.

4.6.1.2 Jarrit Option

A 1% NSR royalty is payable on mineral claim 515810 (Jarrit Claim). The claim covers a portion of the Blackwater deposit.

4.6.1.3 JR Option

The current agreement would allow Artemis to purchase two-thirds of an NSR royalty on three Blackwater claims (637203, 637205, and 637206) for C$1,000,000 at any time, such that a 1% NSR royalty would remain. This royalty does not affect the Blackwater deposit or economics of the 2021 FS.

4.6.1.4 PS Claim

This was purchased by Artemis, along with the Dave2 claim, on 3 February 2021.

4.6.2 Capoose Claims Block Agreements and Encumbrances

4.6.2.1 JAG Option

In December 2011, Silver Quest exercised an option to earn a 100% interest in the JAG Option claims, 534364, 534365, 534366, 534367, 557495, 557496, 552497 and 649243, from a third- party individual. New Gold inherited the ownership and terms of the agreement with its acquisition of Silver Quest. The optionor holds a 2% NSR royalty, of which Artemis may purchase half for C$1,000,000. Artemis has an obligation to pay an advance royalty to the optionor of $30,000 per annum, to be credited against the NSR royalty.

4.6.2.2 Buck Option

New Gold, through Silver Quest, acquired a 100% interest in the Buck Option claims, 643103,

643104, 643106, 643107, 643108, 643109, 643110, and 643123, obtained in December 2011 from Paget Minerals Corporation. The optionor retains a 1.5% NSR royalty. Artemis may purchase two-thirds of the Buck NSR royalty for C$2,000,000, such that a 0.5% NSR royalty would remain.

4.6.2.3 Capoose Property Option

New Gold, through Silver Quest, acquired a 100% interest in the following claims, 641983, 641984, 704825, 704826, 704827, 704828, 704829, 704830, 704854, 704855, 704863, 706011, 642544, 642564, 642565, 642583, 553489, 704807, 704817, 706638, 642243, 642269, 643303, 705004, 705005, 706633, 706634, 706635, 706636 and 706637 obtained prior to December 19, 2011 from an individual. The optionor holds a 1.5% NSR royalty. Artemis may purchase two- thirds of the NSR royalty for C$2,000,000 such that a 0.5% NSR royalty would remain.

4.6.2.4 Capoose Option Joint Venture

New Gold, through Silver Quest, acquired a 100% interest in claim no. 512838, 60% of which was obtained from Bearclaw Capital Corp. (Bearclaw) in 2009, and the remaining 40% of which was obtained from Bearclaw by November 2010. Through an addendum to the original agreement, Silver Quest's mineral claim nos. 552493, 552494, 552495, 564372, 564373, 564375, 564376 and 564377 became part of the subject property upon Silver Quest's exercise of the 60% option in October 2009. The optionor retained a 2.25% NSR royalty. There is a buy-down right contained in the agreement which may entitle Artemis to purchase four-ninths of the NSR royalty for C$1,500,000, such that a 1.25% NSR royalty would remain.

4.6.3 Auro Claims Block Agreements and Encumbrances

In March 2012, New Gold acquired a 100% interest in the Auro claims block (claims 646683, 745822, 745842, 745862, 745882, 745902, 745922, 745942, 745962, 745982, 746002, 746022, 746042, 746062, 746082, 746102, 746182, 746202, and 831124) from Gold Reach. The vendor retained a 2% NSR royalty with no buy-down provision.

4.6.4 Key Claims Block Agreements and Encumbrances

4.6.4.1 Key Agreement

In December 2013, New Gold purchased a 100% interest in the Key claims block (claims 564994, 564995, 564996, 564997, 564998, 564999, 565000, 565001, 589167, 589177, 589183, 589231, 589232, 589234, 589236, 589238, 589241, 589242, 589243, 589244, 642003, 642004, 642023, and 642024) from Troymet Exploration Corporation. Troymet retained a 2% NSR royalty. Artemis may purchase half of the Key NSR royalty for C$2,000,000, such that a 1% NSR royalty would remain.

In October 2010, Troymet acquired a 100% interest in four claims (642003, 642004, 642023, and 642024) comprising part of the Key claims block from a third-party individual. The third- party individual retained a 2% NSR royalty, three-quarters of which (1.5% NSR) can be purchased at any time for C$750,000. New Gold confirmed this royalty with its December 2013 acquisition of Troymet's Key claims block.

In August 2007, Troymet acquired a 100% interest in 20 claims (564994, 564995, 564996, 564997, 564998, 564999, 565000, 565001, 589167, 589177, 589183, 589231, 589232, 589234, 589236, 589238, 589241, 589242, 589243, and 589244) comprising part of the Key claims block. These claims are subject to a 3% NSR royalty in favour of an individual. Two-thirds of the royalty (2% NSR) may be purchased for C$1,000,000 in cash or stock at any time. New Gold confirmed this royalty with its December 2013 acquisition of Troymet's Key claims block.

4.6.5 Parlane Claim Block Agreements and Encumbrances

New Gold acquired through an Option Agreement dated November 25, 2010 a 100% interest in 31 claims (694043, 694044, 694045, 694046, 694063, 694064, 694065, 694066, 694083, 694084, 694085, 694086, 694087, 694088, 694089, 694090, 694103, 694123, 694143, 694144, 694145, 694146, 694147, 694148, 694163, 694183, 694184, 694185, 694186, 694187 and 694287) comprising part of the Parlane Property. These claims are subject to a 2.0% NSR royalty in favour of an individual. One half of the royalty may be purchased for C$1,000,000 resulting in a 1.0% NSR royalty remaining.

New Gold acquired, through an Option Agreement dated October 9, 2010 assigned by an Assignment and Novation Agreement dated January 3, 2012, 28 claims (713362, 713382, 713402,713422, 713442, 713462, 713482, 713502, 713522, 713542, 713562, 713582, 713602, 713622, 713642, 713662, 713682, 713702, 713722, 713742, 713782, 713802, 713822, 713842, 713862, 713882, 713902 and 713922) comprising part of the Parlane Property. These claims are subject to a 2.0% NSR royalty in favour of an individual The NSR royalty may be bought down by a payment of C$1,000,000 to a 1.0% NSR royalty and these claims are also subject to an additional 1.0% NSR royalty in favour of Greencastle Resources Ltd.

4.6.6 RJK Claim Block Agreements and Encumbrances

New Gold acquired through an Option Agreement dated December 1, 2010 a 100% interest in 18 claims (694243, 694245, 694263, 694264, 694265, 694283, 694284, 694285, 694286, 694288, 694289, 694290, 694291, 694292, 694293, 694294, 694295 and 694296) comprising part of the RJK Property. These claims are subject to a 2% NSR royalty in favour of an individual. One half of the royalty may be purchased for C$1,000,000 resulting in a 1.0% NSR royalty remaining.

New Gold acquired through an Option Agreement dated December 3, 2010 a 100% interest in 13 claims (694164, 694188, 694203, 694204, 694205, 694206, 694207, 694208, 694209, 694210, 694223, 694224 and 694225) comprising part of the RJK Property. These claims are subject to a 2.0% NSR royalty in favour of an individual. One half of the royalty may be purchased for C$1,000,000 resulting in a 1.0% NSR royalty remaining.

New Gold acquired through an Option Agreement dated August 2011 a 100% interest in 3 claims (835434, 835436 and 835527) comprising part of the RJK Property. These claims are subject to a 3.0% NSR royalty in favour of an individual. One half of the royalty may be purchased for C$2,000,000 resulting in a 1.5% NSR royalty remaining.

4.7 Environment, Social, Economic, and Cultural Heritage Liabilities and Social License

4.7.1 Environmental, Social, Economic, and Cultural Heritage Liabilities

Environmental, social, economic, and cultural heritage baseline studies have been completed for Project study area and nearby areas. The Project liabilities for valued environmental, social, economic, and cultural heritage components have been assessed in the Project's application for an Environmental Assessment Certificate / Environmental Impact Statement (Application/EIS; New Gold 2015) and for the Joint *Mines Act/Environmental Management Act* application to be submitted in late 2021. The Project has been granted an Environmental Assessment Certificate (EAC) #M19-01 on June 21, 2019 (EAO 2019c) under the 2002 *Environmental Assessment Act* and an Environmental Assessment Decision Statement (DS) on April 15, 2019 under the *Canadian Environmental Assessment Act, 2012* (CEA Agency 2019). The decision includes 43 conditions. Assessment of components to address updates in the Project design have been considered in recent permits (Table 20-1) and will be addressed in permit applications currently in progress (Table 20-2 and Table 20-3). To manage potential effects of the Project, an Environmental Management System supported by a comprehensive set of management plans are being developed for permitting phase of the Project (Section 20.6.1).

The current environmental liabilities are typical of a project where detailed engineering designs have been completed and exploration drilling activities are complete. The major areas that would require rehabilitation include access roads, exploration campsite and drill pads.

4.7.2 Social License

4.7.2.1 Indigenous Nations

The Blackwater mine site is located within the traditional territories of Lhoosk'uz Dené Nation (LDN), Ulkatcho First Nation (UFN), Skin Tyee Nation and Tsilhqot'in Nation. The Kluskus and Kluskus-Ootsa FSRs and Project transmission line cross the traditional territories of Nadleh Whut'en First Nation (NWFN), Saik'uz First Nation (SFN), and Stellat'en First Nation (StFN; collectively, the Carrier Sekani First Nations [CSFNs]) as well as the traditional territories of the Nazko First Nation (NFN), Nee Tahi Buhn Band, Cheslatta Carrier Nation and Yekooche First Nation (EAO 2019a).

Throughout the environmental assessment (EA) process, the Environmental Assessment Office (EAO) consulted with LDN, UFN, WFN, SFN, StFN [collectively, the Carrier Sekani First Nations (CSFNs)] and NFN according to the deeper end of the consultation spectrum described in 2004 by the Supreme Court of Canada in Haida Nation v. British Columbia (Minister of Forests). The EAO consulted with Skin Tyee Nation, Tsilhqot'in Nation, Cheslatta Carrier Nation, Nee Tahi Buhn Band, and Yekooche First Nation at the lower end of the Haida consultation spectrum.

The EAO notified the Skin Tyee Nation, Tsilhqot'in Nation, Cheslatta Carrier Nation, Nee Tahi Buhn Band, and Yekooche First Nation of key milestones in the EA process. The EAO also invited these nations to meet to discuss any Aboriginal Interests potentially affected by Blackwater and shared the relevant sections of the EAO's draft Assessment Report (Part C – Indigenous Consultation Report) for their review and comment. EAO but did not receive any requests to meet or comments on Part C of the Assessment Report, with the exception of the Tsilhqot'in National Government, who commented that it had no further concerns.

4.7.2.1.1 Lhoosk'uz Dené Nation and Ulkatcho First Nation

The EAO engaged UFN and LDN in a collaborative manner throughout the EA review that was guided by a 2016 Memorandum of Understanding (EAO 2019a).

The EAO and the Impact Assessment Agency of Canada (formerly the Canadian Environment Assessment Agency) (Agency) collaborated with UFN and LDN on proposed conditions in the provincial environmental assessment certificate (EAC) and the federal Decision Statement (DS) and an assessment of the potential impacts of the Project on UFN and LFN. As part of the EAO and the Agency's collaboration with the UFN and LDN during the Project's EA, UFN and LDN provided a detailed report setting out their perspectives of the impacts from Blackwater to UFN and LDN (Appendix G to EAO's Assessment Report; EAO 2019a).

The Chiefs for UFN and LDN submitted letters of support to the EAO to share their final conclusions regarding the EA, adequacy of consultation and accommodation, and their consent to the issuance of the EAC (EAO 2019a). In response to the UFN/LDN letter, the EAO advised the Province had committed to negotiate an Economic and Community Development Agreement (ECDA) for Blackwater with the LDN and UFN. Under this agreement, the Province would outline how it intends to share mineral tax revenues from Blackwater with LDN and UFN if Blackwater is developed and operated. Negotiations on the ECDA are ongoing and are led by LDN and UFN and provincial representatives. The Province also communicated that it would be taking a collaborative approach with LDN and UFN on mine permitting and over the life of mine.

There has been a trilateral Participation Agreement in place with Artemis, LDN and UFN since April 2019.

4.7.2.1.2 Carrier Sekani First Nations

The EAO engaged the NWFN, SFN and StFN in a collaborative manner throughout the EA review which was guided by the CSFNs Blackwater Collaboration Plan (EAO 2019a). The EAO and Agency collaborated with the CFSNs on proposed conditions in the provincial EAC and the federal DS. As part of the Collaboration Plan, the CSFNs, EAO, Agency collaboratively developed a report that sets out CSFNs' perspectives on the impacts of the Blackwater Project to CSFNs' Aboriginal rights, title and interests (EAO 2019a).

At the conclusion of the EA review, the CSFNs advised the CSFNs and the EAO it had reached consensus on proposed conditions, and that in particular, the CSFNs valued the incorporation of Yinka Dene Water Law. In response to concerns raised by CSFNs with respect to economic accommodation and compensation, the Province met with CSFNs to discuss their concerns and committed to continue to work with the CSFNs on an economic benefits package in relation to Blackwater.

At the time and in response to the CSFNs' concerns, New Gold committed to continuing negotiations with CSFNs with the goal of reaching a mutually-acceptable "participation agreement" that will include accommodative measures and other benefits, including discussions on financial compensation, business and employment opportunities arising from the Project, environmental matters, and implementation and communication protocols. An agreement has not yet been reached by the parties.

4.7.2.2 Stakeholders and the Public

The Project proponent engaged local residents, including the communities of Vanderhoof, Fort St. James, Burns Lake, Fraser Lake, Quesnel, the City of Prince George, and the Regional Districts of Bulkley-Nechako and Cariboo. In addition, there was engagement with other potentially affected or interested stakeholders including tenure holders and private landowners, commercial and non-commercial land users, service providers, interest groups, and non-government organizations. Stakeholders and the public were also provided with opportunities to review comment on documents produced during the EA review.

Pursuant to EAC Condition 37, BW Gold has established a Community Liaison Committee (CLC) to provide information to Artemis on Project effects in members' communities and mitigation measures to address potential social and economic effects. Committee members include the District of Vanderhoof, Village of Fraser Lake, UFN, LDN, NWFN, SFN, STFN, City of Quesnel, Electoral Area I of Cariboo Regional District, Electoral Area F of Regional District Bulkley Nechako, Northern Health, Nechako Environment and Water Stewardship Society and College of New Caledonia. The CLC will be in place throughout construction, operations and the first five years of closure.

4.7.2.3 Moving Forward

Moving forward engagement with Indigenous nations will be guided by agreements with Artemis and conditions in the Project's EAC M#19-01 (EAO 2019b) and federal DS (CEA Agency 2019). Engagement with stakeholders and the public will be guided by the Project's EAC and DS conditions and permits.

4.8 Permits

Details of the permitting requirements for the Project are presented in Section 20.

4.9 Comments on Section 4

In the opinion of the QPs, the information discussed in this section supports the declaration of

Mineral Resources and Mineral Reserves, based on the following:

- Information from legal and Artemis' experts indicated that Artemis holds 100% of the mineral claims
- 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.
- Surface rights in the vicinity of proposed electrical transmission lines, water pipeline and access roads were reviewed in 2013–2014. The review identified a number of overlapping surface rights in the planned mining operations and Linear Infrastructure areas.
- Artemis will need to apply for additional permits as appropriate under local, Provincial, and Federal laws to allow mining operations
- Notwithstanding the information contained above in this Section 4, there is no guarantee that title to any of the mineral claims will not be challenged or impaired. Third parties may have valid claims affecting the Project, including prior unregistered liens, agreements, transfers or claims, including aboriginal land claims. The titles may be affected by, among other things, undetected defects. As a result, there remains a risk that there may be future constraints on Artemis' ability to operate the Project or Artemis may be unable to enforce rights with respect to the Project.
- Based on the information provided by Artemis 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 ACCESSIBILITY, CLIMATE, LOCAL RESOURCES, INFRASTRUCTURE, AND PHYSIOGRAPHY

5.1 Accessibility

The Project is readily accessible by vehicle from Vanderhoof using the Kluskus FSR and the Kluskus-Ootsa FSR. The Project site can be accessed from the Kluskus-Ootsa FSR using an 18 km-long mine road that was built in 1986 by Granges and improved by Richfield. The FSR provides direct access to the Project area and camp location. Driving time from Vanderhoof to the Project is about 2.5 hours.

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

5.2 Climate

Hydrological and meteorological data have been collected at the project site since early 2011 (KP, 2021a; KP, 2021b). The site-specific data were correlated with available long-term regional data to characterize the hydrometeorological conditions prevalent at the site (KP, 2021c), providing a reasonable basis for engineering design and water balance modelling.

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. 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 to 440 mm.

The weather is not expected to present any unusual difficulties for mining operations and operations are planned on a year-round basis.

5.3 Local Resources and Infrastructure

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.

There is no grid-connected power in the direct vicinity of the Project. The main BC Hydro 500 kV transmission lines supplying western BC are approximately 100 km to the north. Several interconnection points from the 500 kV lines to existing 230 kV substations and transmission lines are possible in an area between Fraser Lake and Vanderhoof. Power for the current Blackwater exploration camp is provided by generators.

Personnel to support development and operation of the mine 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 Project infrastructure including the mill, waste and ore stockpiles, and TSF will be sited predominantly in the Davidson Creek watershed.

Infrastructure required for Project development and operation is discussed in more detail in Section 18.

5.4 Physiography

The elevation of the Blackwater Project ranges 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 property at the summit of Mt. Davidson. Bedrock outcrops on the property are limited and most of the area is covered by thick glacial deposits of 2 m or more, except for the upper 150 m of Mt. Davidson and a few localized areas at lower elevations. The claim area is partially overlain by thick lodgement till and ablation till deposits.

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 m, while subalpine fir and Englemann spruce are dominant above 1,500 m. Lodgepole pine is a major species on dry, fire-prone sites at most elevations. The pine beetle epidemic infested almost all the lodgepole pine forests within this subregion.

5.5 Downstream Drainage Network

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.

5.6 Regional Tectonics and Seismicity

The Project site is situated within a region of BC where the level of recorded historical seismic activity has been low. However, higher seismicity is associated with the Queen Charlotte - Fairweather fault system located offshore of the west coast of BC and the Alaskan panhandle. 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 Project site.

An updated seismicity assessment was carried out for the Project area in 2021 (KP, 2021d), including a review of the regional seismicity and a probabilistic seismic hazard analysis, to provide seismic design parameters for the TSF and other facilities, including the stockpiles and water management dams. Design ground motion parameters provided by the seismic hazard analysis include peak ground acceleration, spectral acceleration (defining the uniform hazard spectrum), and earthquake magnitude.

5.7 Comments on Section 5

In the opinion of the QPs:

- 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.
- Surface rights in relation to the proposed operation are discussed in Section 4.4.

6 HISTORY

The Project ownership history was discussed in Section 4.2.

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: Summary of Work Completed in Project Area

Year	Operator	Work
1973	Granges	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	Granges	Soil sample and ground magnetometer surveys to follow up 1973 soil results.
1977	Granges	Pem claim staked covering most of the presently defined mineral deposit. Pulse EM survey on the Pem claim (12.5 km).
1979	Granges	Vector Pulse electromagnetic (EM) survey on the Pem claim (7 km).
1981	Granges	Helicopter EM and magnetometer survey.
1981	Granges	Horizontal Loop EM survey on the Deb #1 claim.
1981	Granges	Reconnaissance mapping of the Mt. Davidson area.
1982	Granges	Soil geochemistry (220 samples) and ground magnetometer survey (20.8 line km) on the Pem claim.
1983	Granges	Hammer seismic survey.
1984	Granges	Hand-trenching (30 trenches for total of 66 m) and VLF survey (4.8 line km) on the Pem claim. Only 1 trench intersected bedrock.
1985	Granges	Winkie drilling (8 holes for total of 507 m) on the Pem claim.
1985	Granges	Construction of access road from km 146.5 on the Kluskus Haulage road, east 18 km to the Pem grid.
1986	Granges	Percussion drilling (34 holes totaling 1,524 m) on the Pem claim.
1987	Granges	Diamond drilling (23 holes totaling 2,617 m) on the Pem claim.
1992	Granges	Line cutting (58.8 km), soil samples (955), stream silt samples (35), geological mapping (6000 ha), geophysical surveys (50km induced polarization (IP), magnetics, very low frequency (VLF)), diamond drilling (5 holes totaling 785 m).
1994	Granges	Line Cutting (48.2 km), rock samples (29), soil samples (1598), silt samples (23), lake sediment samples (4) Dighem airborne geophysical survey (881 line km of EM, magnetics, radiometrics), Dave claim IP survey (20 km), diamond drilling (5 holes totaling 761.68 m).
1997	Kennecott Canada	Line cutting (4 km) and IP survey; Dave claim.
2005	Silver Quest	Diamond drilling (5 holes totaling 939 m).
2006	Silver Quest	Diamond drilling (2 holes totaling 394 m).
2007	Silver Quest	Soil samples (335).
2009	Richfield	Diamond drilling (18 holes totaling 3,621 m).
2010	Richfield	Diamond drilling (57 holes totaling 21,336 m).

Year	Operator	Work
2011	Richfield	Diamond drilling (59 holes totaling 19,727 m). Initial Mineral Resource estimate and two subsequent resource estimate updates.
2011	New Gold	Diamond drilling (125 holes totaling 49,316 m), metallurgical test holes (7 holes totaling 2,282 m).
2012	New Gold	Diamond drilling (716 holes totalling 207,333 m), geotechnical holes (13 totalling 5,003 m), metallurgical test holes (20 totaling 1,816 m), waste rock characterization (14 holes totalling 2,952 m) hydrological monitoring or pilot holes (7 totaling 2,265 m). Three resource estimate updates. Completion of preliminary economic assessment.
2013	New Gold	Resource estimate update. Draft AIR submitted. Feasibility study; (2013 FS) estimation of Mineral Reserves based on 2013 resource estimate. Diamond drilling (13 holes totaling 7,521 m)
2014	New Gold	Final AIR submitted.
2014–2019	New Gold	Permitting activities, acquisition of additional mineral claims, additional metallurgical testwork. Assessment of the approved Application for an EA Certificate was conducted by EAO from January 12, 2016 to May 17, 2019. EA Certificate # M19-01, which included the Certified Project Description and a Table of Conditions, issued on June 21, 2019. CEAA commenced the environmental assessment on December 21, 2012, and the Decision Statement was issued on April 15, 2019.
2020	Artemis	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	Artemis	Completes feasibility study, estimation of Mineral Resources and Mineral Reserves

7 GEOLOGICAL SETTING AND MINERALIZATION

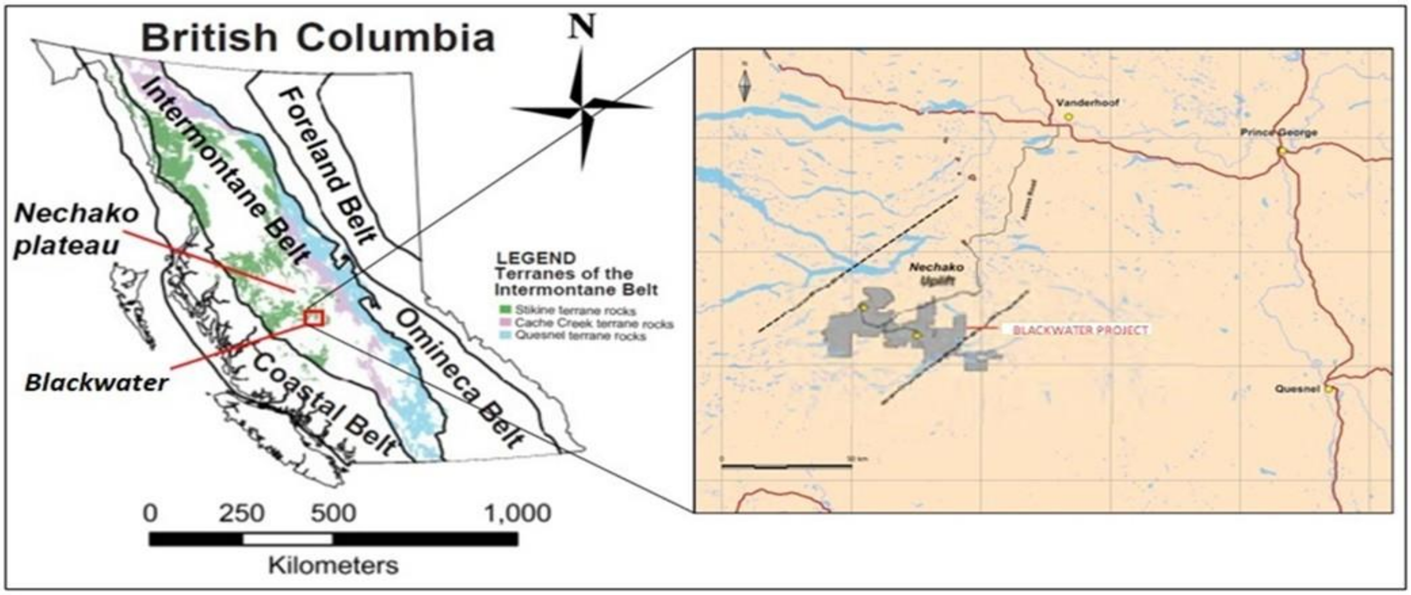
The Property is located on the Nechako Plateau near the geographic centre of British Columbia. The plateau is part of the Intermontane Belt superterrane situated between the Coast Belt to the west and the Omineca Belt to the east (Figure 7-1). Topographic relief for the plateau is moderate with elevations 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 Jurassic to early Tertiary time. These rocks have been 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 and Webster, 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.

7.1 Local and Property Geology

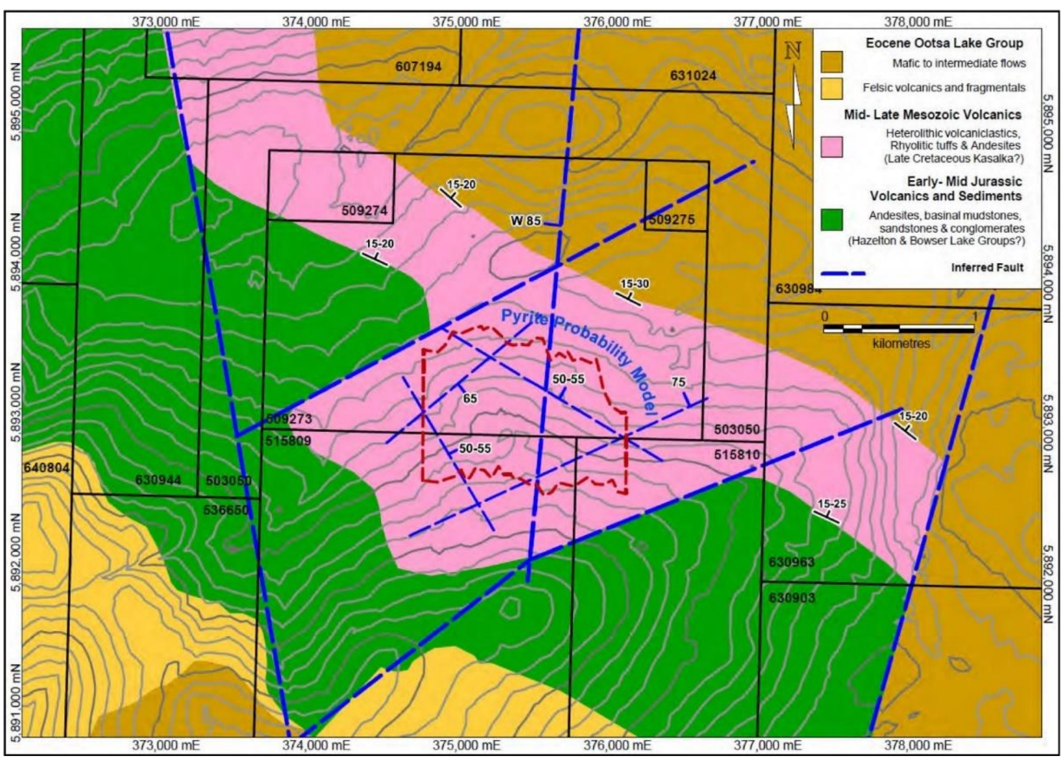
Quaternary glacial overburden, colluvial, and fluvial deposits mask the majority of bedrock within the Project area. Project geology is based on interpretations derived from observations and interpretation of geologic field mapping in conjunction with core and reverse circulation drilling data collected 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 7-1: Blackwater Project Location and Tectono-Stratigraphic Setting



Note: prepared by New Gold, 2014.

Figure 7-2: Top of Bedrock Geology in Vicinity of Blackwater Deposit



Note: 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 north-easterly-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.

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.

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 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.

The lithological codes used in the Blackwater drillhole database have been 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
O	Overburden
AN	Andesite
F	Felsic tuff
FL	Felsic lapilli tuff
V	Volcaniclastic
E	Epiclastic
SE	Argillite/sandstone/conglomerate

7.2 Structural

7.2.1 Overview

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.

7.2.2 Structural Model

A selection of 462 drillholes was re-logged using core photographs to identify major fault intervals used to create the structural models (New Gold, 2014).

A major north–south trending fault dissects the orebody 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 Alteration

7.3.1 Overview

The alteration minerals most commonly identified included muscovite, high and low-temperature illite, ammonium bearing illite, smectite, silica, biotite, and chlorite. Alteration assemblages were defined as follows:

- 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).

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

7.3.2 Alteration Model

The alteration model was initially developed using the methodology outlined in Section 9.4.3 (New Gold, 2014).

An additional 607 drillholes were re-logged from core photos, and a continuous down hole alteration interpretation domain table was constructed for each drillhole using potassic (POT) versus sericitic (SER) altered categories. Any interval that was visibly bleached and altered in excess of approximately 50 vol% was categorized as SER and all others as POT. The ammonium-bearing (NH_4) alteration assemblage could not be used in the photographic re-logs because the minerals associated with that alteration type can only be identified by spectroscopy.

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. Statistical analysis shows that the NH_4 and SER alteration domains closely coincide, and they were therefore combined as a single domain (SER) for resource modelling. A wireframe encompassing the distribution of sericitic alteration was generated from a sectional interpretation of detailed drillhole information. An indicator model of alteration types, simulations of logged silica and sericite intensity, and block estimates of alkali cation percentages, particularly aluminum, provided additional support to the modelled alteration domains.

Statistical comparisons of the modelled sericite and silica between gold and silver, respectively, demonstrate the relationships between mineralization and alteration.

Similar comparisons also demonstrate the relationships between gold and logged sulphides, particularly pyrite and dendritic black sulphide (DBS) mineralization.

7.4 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 at Blackwater 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 the following types:

- Disseminated:
  - As 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 drillholes.

Secondary quartz occurs in several modes:

- Pervasive, amorphous to translucent silicification with associated illite ± sericite.
- Commonly drillholes display intense silicification of felsic units, epiclastics, and more intermediate volcaniclastic rocks with biotite alteration of the matrix (hornfels);
- 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.

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. 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: Drillhole Plan Showing Location of Referenced Cross-Sections

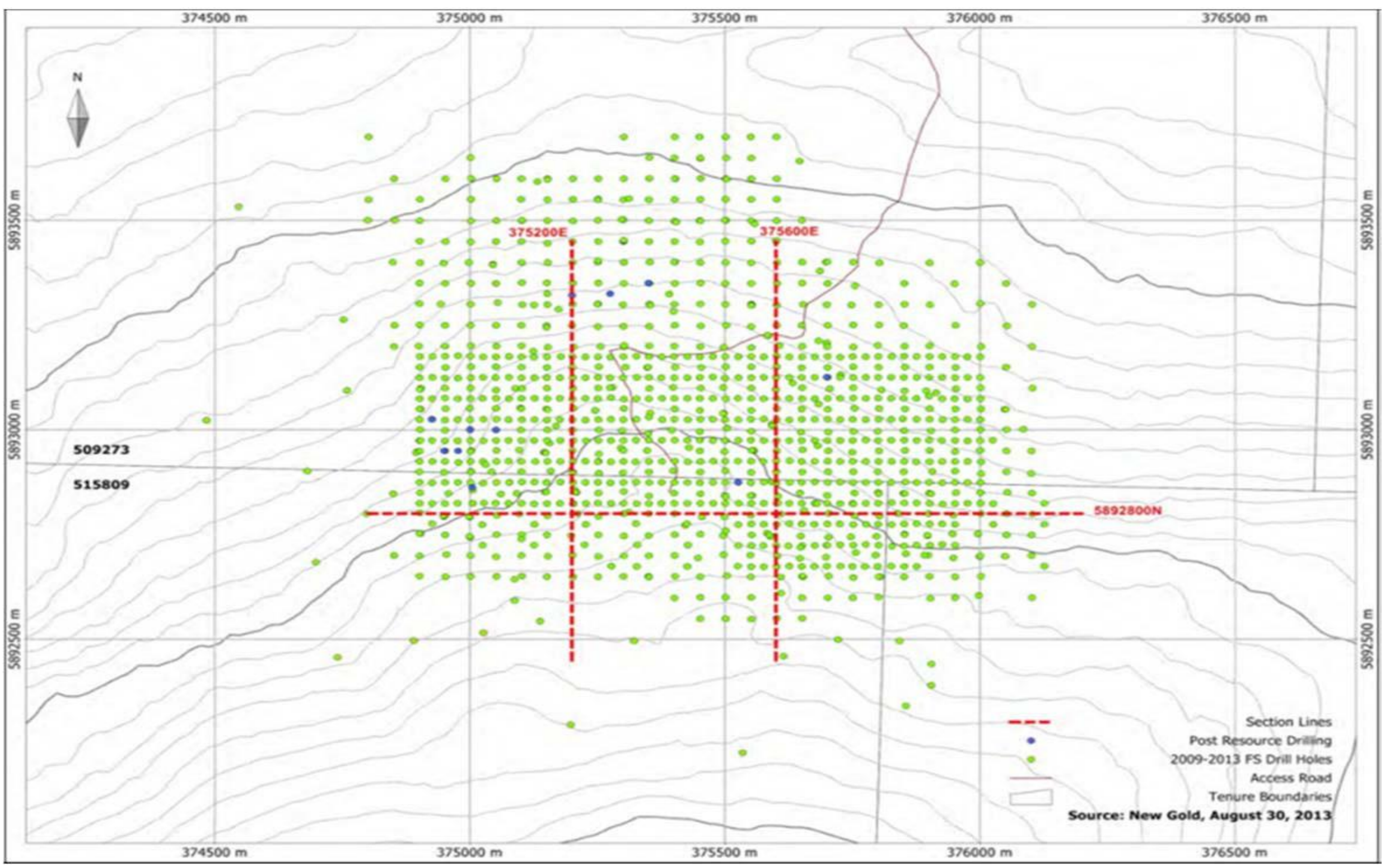
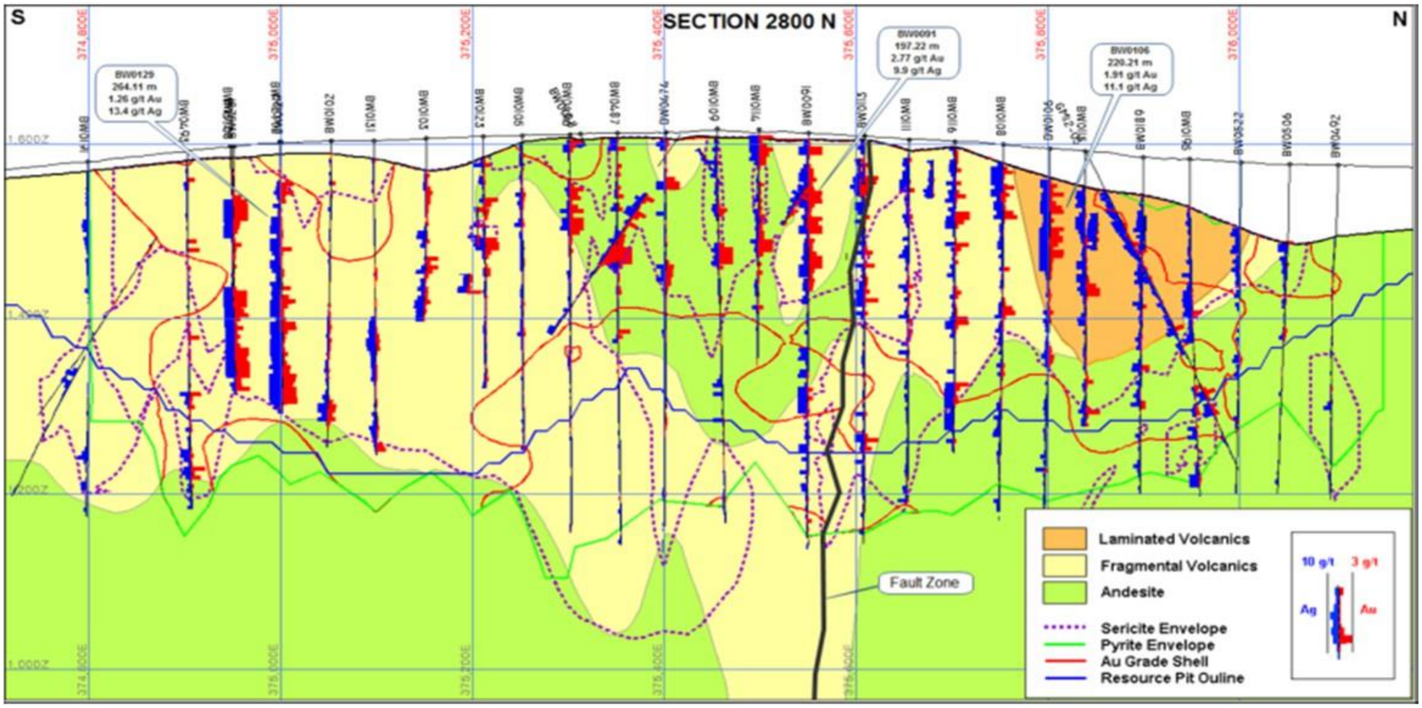
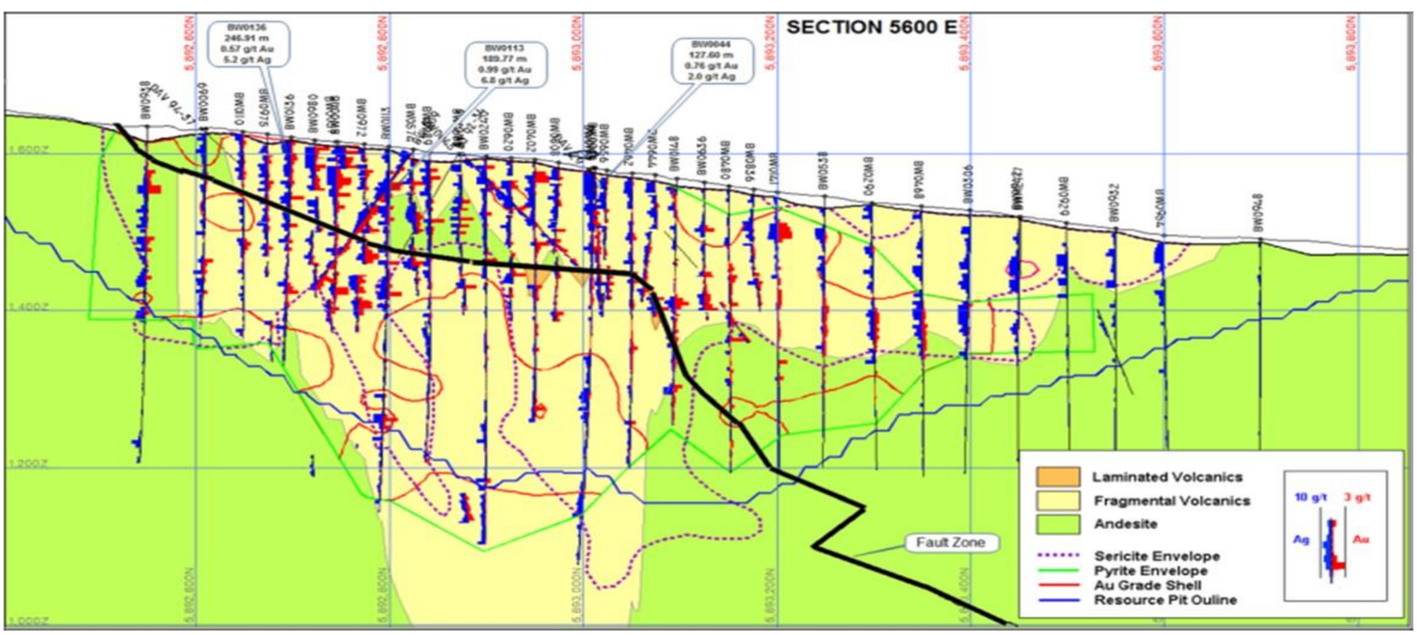


Figure 7-4: Cross-Section 2800 N



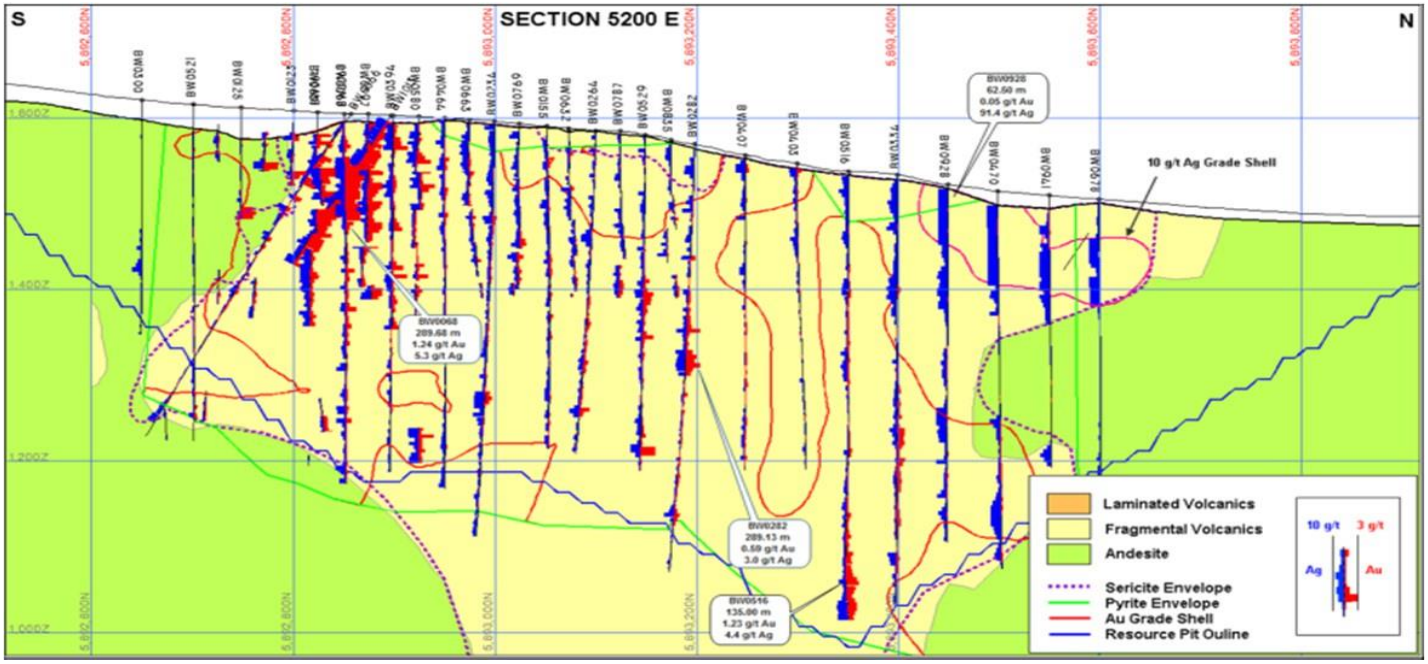
Note: prepared by New Gold, 2014

Figure 7-5: Cross-Section 5600 E



Note: prepared by New Gold, 2114.

Figure 7-6: Cross-Section 5200 E



Note: prepared by New Gold, 2014.

8 DEPOSIT TYPES

8.1 Deposit Model

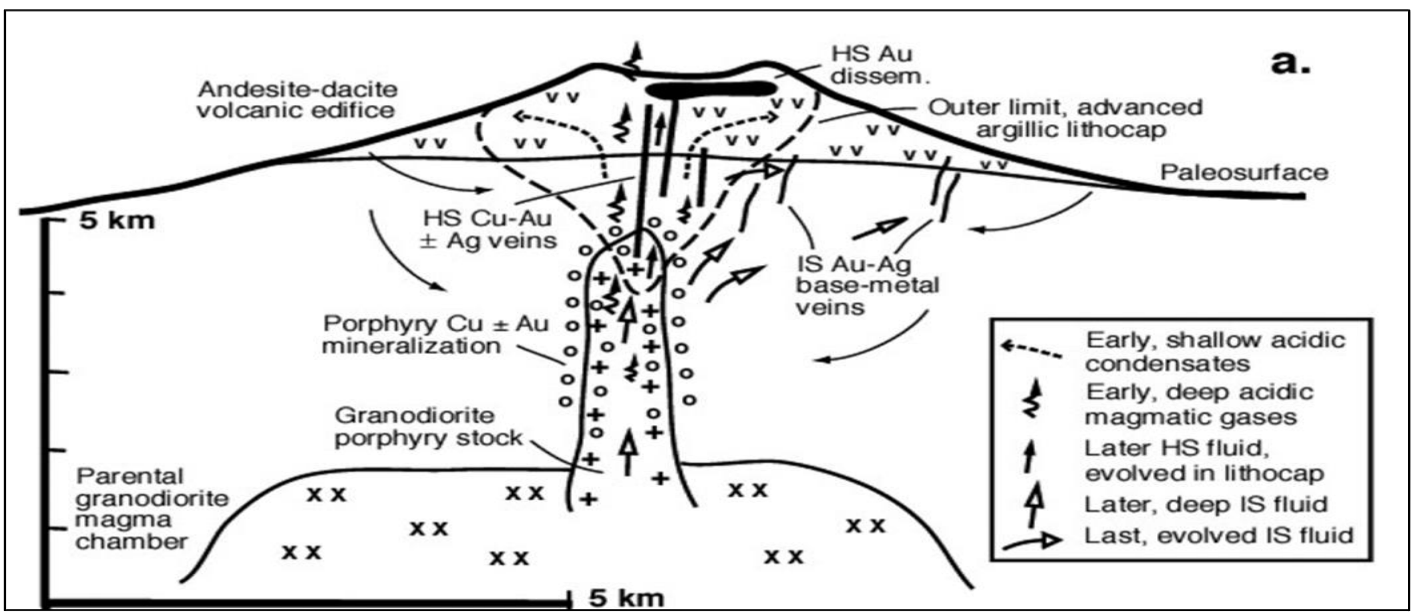
The Blackwater deposit is considered an example of a volcanic-hosted, epithermal-style gold-silver deposit.

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.

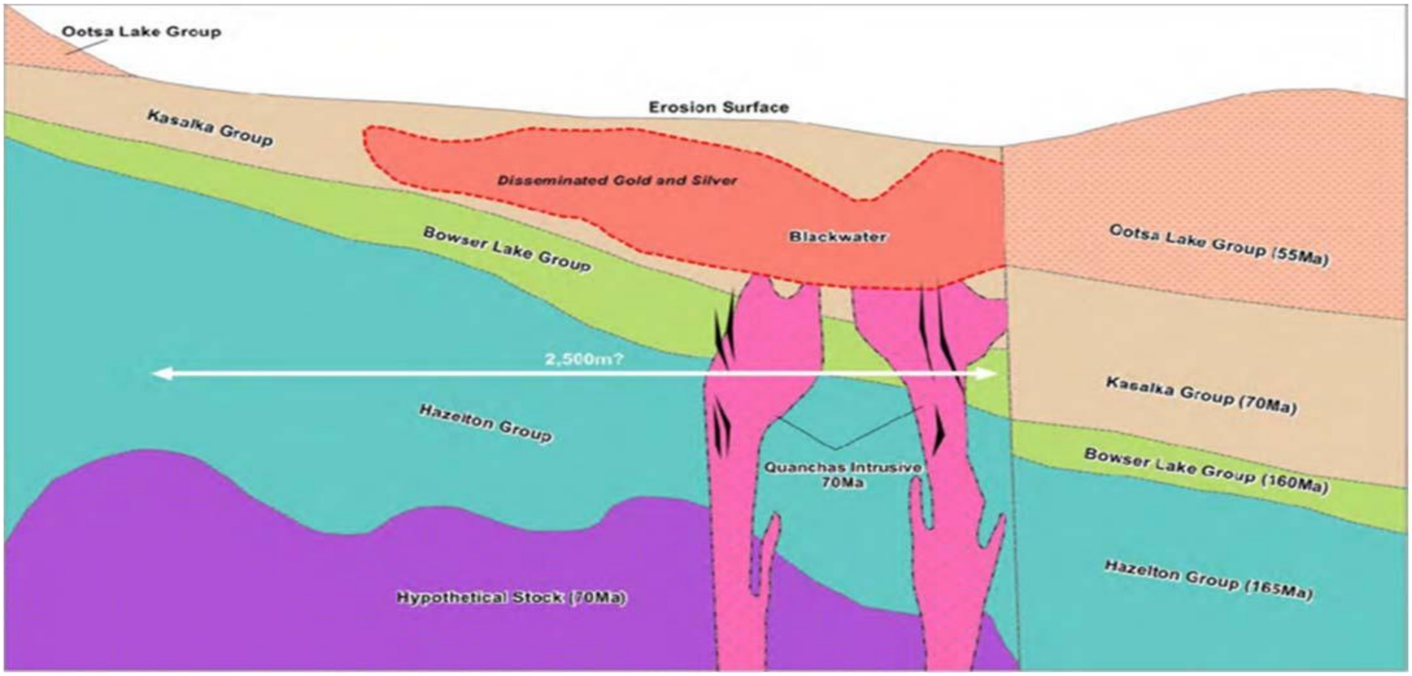
A typical section showing the main features of a calc-alkaline volcanic arc setting and associated epithermal and related mineralization is included as Figure 8-1 (Sillitoe and Hedenquist, 2003). Key features of these deposit styles seen at Blackwater are summarized in Figure 8-1. Figure 8-2 illustrates the hypothesized relationship of the mineralized volcanic rocks to surrounding strata at Blackwater (New Gold, 2014).

Figure 8-1: Schematic Section of Calc-Alkaline Volcanic Arc Setting and Associated Epithermal and Related Mineralization



Note: prepared by New Gold, 2014.

Figure 8-2: Cross-Section of Conceptual Blackwater Model



Note: prepared by New Gold, 2014.

Figure 8-3: Epithermal Gold Deposit Types (after Sillitoe and Hedenquist 2003)

	High sulfidation		Intermediate sulfidation	Low sulfidation	
	Oxidized magma	(Reduced magma) [1]		Subalkaline magma	Alkaline magma
Type example	El Indio, Chile (vein); Yanacocha, Peru (disseminated)	Potosí, Bolivia	Baguio, Philippines (Au-rich); Fresnillo, Mexico (Ag-rich)	Midas, Nevada	Emperor, Fiji
Genetically related volcanic rocks	Mainly andesite to rhyodacite	Rhyodacite	Principally andesite to rhyodacite, but locally rhyolite	Basalt to rhyolite	Alkali basalt to trachyte
Key proximal alteration minerals	Quartz-alunite/APS; quartz-pyrophyllite/ dickite at depth	Quartz-alunite/APS; quartz-dickite at depth	Sericite; adularia generally uncommon	Illite/smectite-adularia	Roscoelite-illite-adularia
Silica gangue	Massive fine-grained silicification and vuggy residual quartz		Vein-filling crustiform and comb quartz	Vein-filling crustiform and colloform chalcedony and quartz; carbonate-replacement texture	Vein-filling crustiform and colloform chalcedony and quartz; quartz deficiency common in early stages
Carbonate gangue	Absent		Common, typically including manganiferous varieties	Present, but typically minor and late	Abundant, but not manganiferous
Other gangue	Barite common, typically late		Barite and manganiferous silicates present locally	Barite uncommon; fluorite present locally	Barite, celestite, and/or fluorite common locally
Sulfide abundance	10-90 vol %		5->20 vol. %	Typically <1-2 vol % (but up to 20 vol % where hosted by basalt)	2-10 vol %
Key sulfide species	Enargite, luzonite, famatinite, covellite	Acanthite, stibnite	Sphalerite, galena, tetrahedrite-tennantite, chalcopyrite	Minor to very minor arsenopyrite ± pyrrhotite; minor sphalerite, galena, tetrahedrite-tennantite, chalcopyrite	
Main metals	Au-Ag, Cu, As-Sb	Ag, Sb, Sn	Ag-Au, Zn, Pb, Cu	Au±Ag	
Minor metals	Zn, Pb, Bi, W, Mo, Sn, Hg	Bi, W	Mo, As, Sb	Zn, Pb, Cu, Mo, As, Sb, Hg	
Te and Se species	Tellurides common; selenides present locally	None known, but few data	Tellurides common locally; selenides uncommon	Selenides common; tellurides present locally	Tellurides abundant; selenides uncommon

Undetermined

Characteristic of Blackwater Deposit

APS, aluminum-phosphate-sulfate minerals

9 EXPLORATION

Artemis 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).

9.1 Geological Mapping

Given the lack of bedrock exposures in the immediate Blackwater deposit area, geologic 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 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 geologic interpretation of the deposit in the subsurface.

9.2 Geochemical Sampling

Soil and stream geochemical surveys were carried out over parts of the Blackwater 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.

Additionally, in 2012 a total of 43 stream silts were collected in key drainage areas around Blackwater. The samples were sent to SGS Laboratories in Vancouver, BC, for analysis. The results indicated anomalous copper and zinc values from streams to the northwest and southeast of the Blackwater deposit. As summarized in Table 6-1, previous operators performed extensive soil geochemistry testing between 1982 and 2007.

9.3 Geophysics

During 2010, Richfield contracted Quantec Geoscience Ltd. (Quantec) f 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 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 lines with 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.

9.4 Other Surveys and Investigations

9.4.1 Topographical Grids and Surveys

Eagle Mapping Ltd. (Eagle Mapping) generated detailed topography in August 2010 from an aerial survey flown on July 7 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 on August 8 and 9, 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 GPS/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.

9.4.2 Petrology, Mineralogy, and Research Studies

Polished section petrographic analysis was conducted on selected drill samples. In 2009 and 2010, Sample suites were selected for the purpose of understanding the nature of the host volcanic and volcaniclastic rocks, and the gold and silver mineralization. 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 drillhole 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.

9.4.3 Alteration Study in Support of Geological Modelling

A two-phase alteration study was completed to develop the alteration model for the deposit. For the first phase some 20 widely-spaced drillholes were re-logged in detail and analyzed by short- wave infrared (SWIR) spectrometer at approximate 10 m spacings down hole. The second phase involved the selection of an additional 135 representative holes, which were collected on approximately 100 m centres 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.

9.5 Exploration Potential

The Blackwater 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 DRILLING

10.1 Introduction

A total of 1,053 core drillholes (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 drillholes 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 has not conducted any core drilling since acquiring the Project.

Table 10-1: 2021 FS Drillhole Summary Table

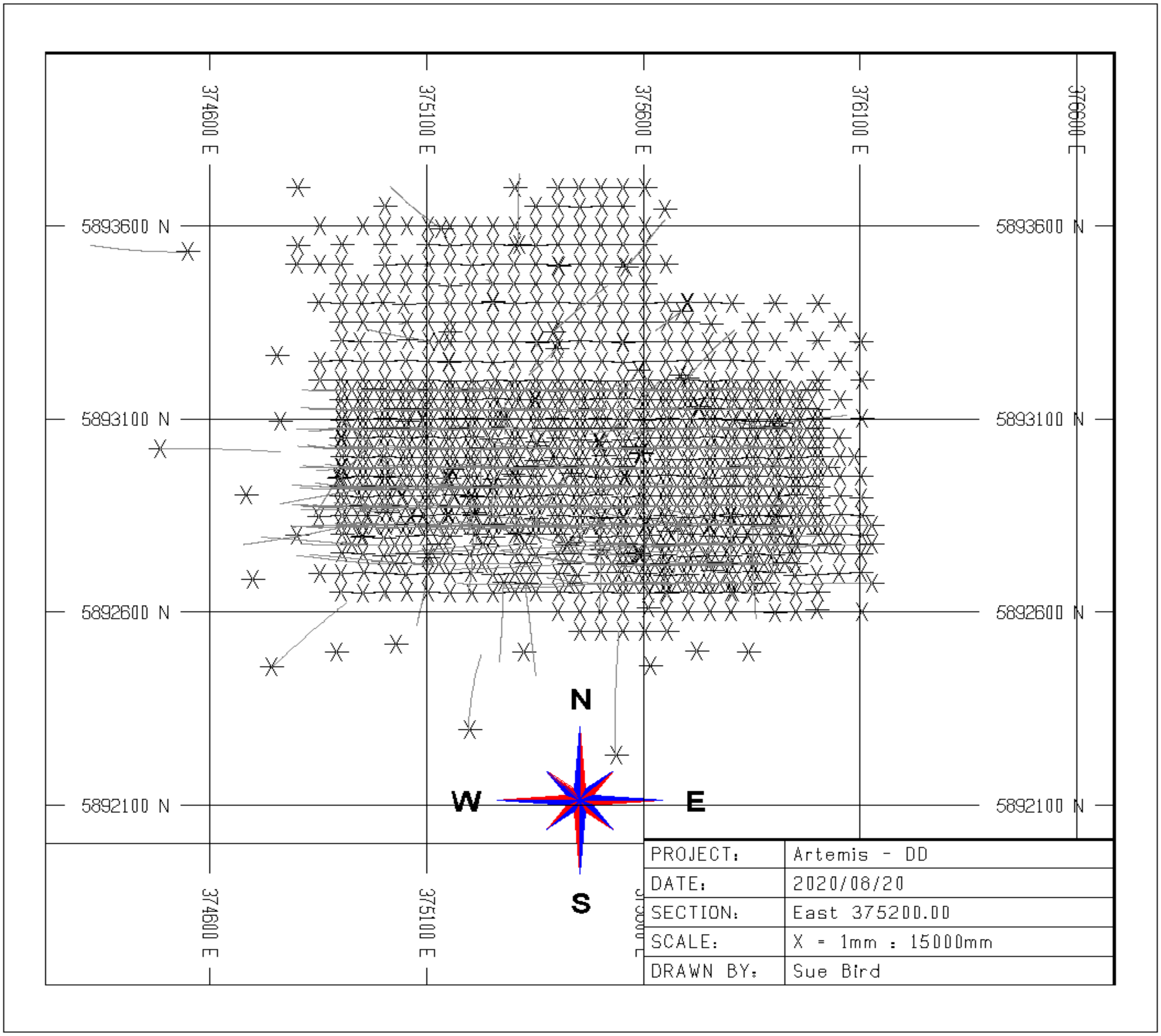
Series	Company	Year	Holes	Total Meters
BW0042 to BW0059	Richfield	2009	18	3,621.23
BW0060 to BW0116	Richfield	2010	57	21,335.92
BW117 to BW0175	Richfield	2011	59	19,727.37
Subtotal	Richfield		*134*	*44,684.52*
BW0176 to BW0298, BW0050R	New Gold	2011	125	49,315.78
BWMET01 to BWMET07	New Gold	2011	7	2,281.91
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 drillholes and grades were included in Section 7 (refer to Figure 7-4 to 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 Drillhole 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. Drill core was transported from drill to camp by four-wheel drive vehicle for core logging.

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 Longear	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

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 metre (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 company 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 drillhole 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 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 Geomechanical and Hydrogeological Drilling

Thirteen specific HQ-size geomechanical drillholes were drilled to collect information for the pit slope stability assessment for the 2013 FS, and the data are used in the 2021 FS. These were numbered GM12-01 to GM12-13. The drillholes were located and inclined to pass through the proposed final pit walls. The total length of these geomechanical holes was 5,003 m.

Twelve specific HQ-size hydrogeological observation wells, termed pilot holes, were drilled to serve as monitoring stations during pumping tests on two 8-inch diameter pumping wells. Several vibrating wire piezometers were installed at each location to measure baseline piezometric elevations in the deposit area and to monitor aquifer response during the pumping tests completed in 2013. These were numbered PH12-2-1, PH13-2-2, PH13-2-3, PH12-3-1, PH12-3-2, PH12-3-3, PH12-4-1, PH12-4-2, PH12-4-3, PH13-1-1, PH13-1-2, and PH13-1-3. The total length of these hydrogeological holes was 4,396 m.

Two 8-inch diameter pumping wells (PW13-1 and PW13-3) were installed using a dual rotary air rig in 2013. Step and constant rate pumping tests were conducted in 2013 to provide information for dewatering and depressurization well design.

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. Information gathered from these drill holes was used in support of the 2021 FS.

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. Information from these drill programs is used in the 2021 FS.

10.10 Condemnation Drilling

Eighteen diamond drillholes (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. Information from these drill programs is used in the 2021 FS.

10.11 Drilling Supporting Mineral Resource Estimation

Some drillholes 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 drillholes are included in the resource database used for estimation purposes as shown in Table 10-3.

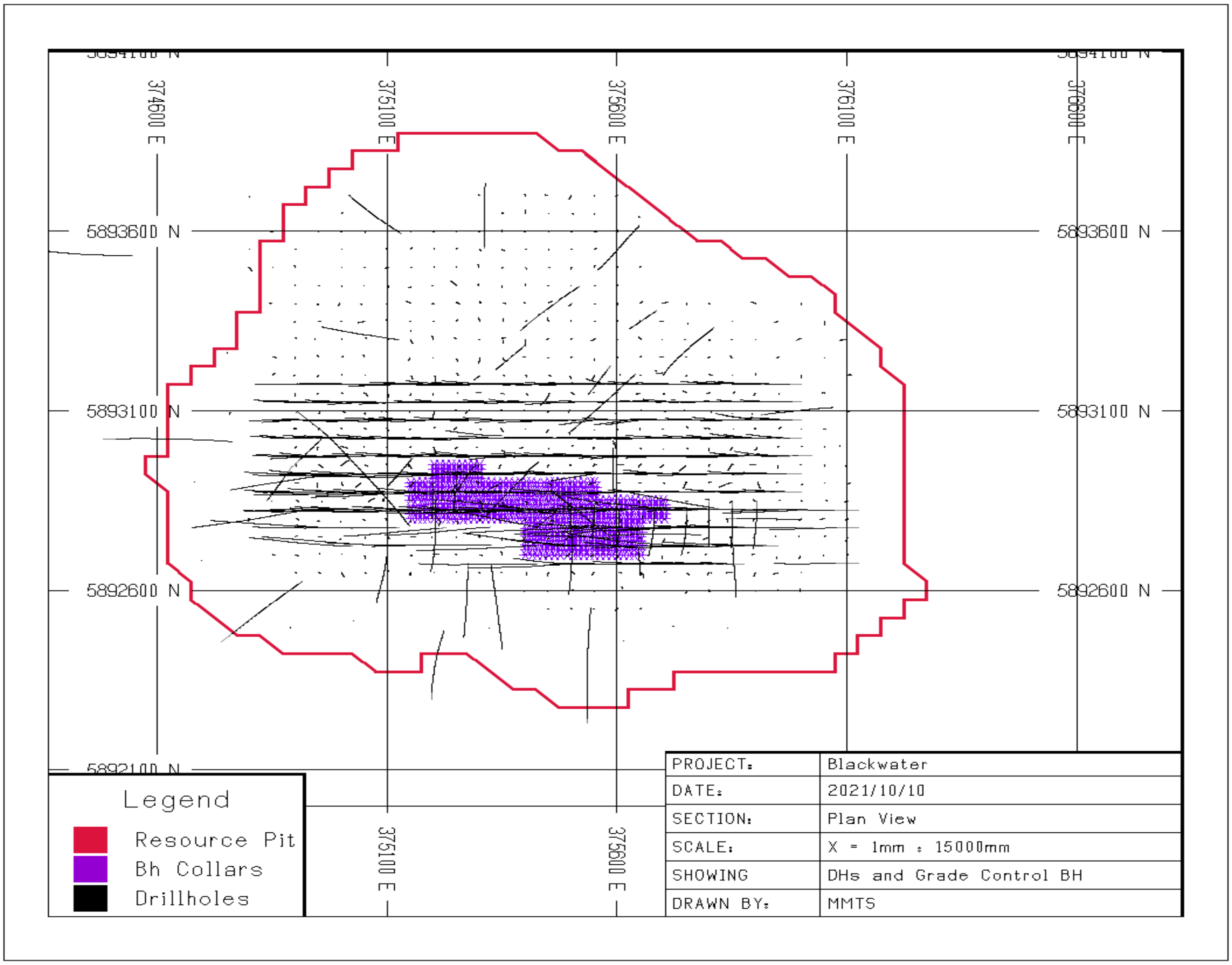
Table 10-3: Resource Database Used In Estimation

Year	Company	Holes	Meters 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 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 drillhole data were not incorporated in the resource estimate, as the drillholes 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.

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.

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

11 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 GeoSim (2011a, b; 2012), AMEC (2012c) and New Gold (2014) and are summarized in this section.

New Gold reviewed the control sample results when received from the laboratory (New Gold, 2012a to 2012j). MMTS reviewed the final QAQC and Grade Control sample results for the current resource estimate.

Quality control (QC) 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 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.

11.1.3 Grade Control Sampling

The pre-production Grade Control Program sampling consisted of reverse circulation drill cuttings being collected at a drill mounted Metzke rotating cone splitter by Artemis 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.

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 Artemis.

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 drillhole database was verified by MMTS, 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

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 %
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
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 (g/t)	Average of 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

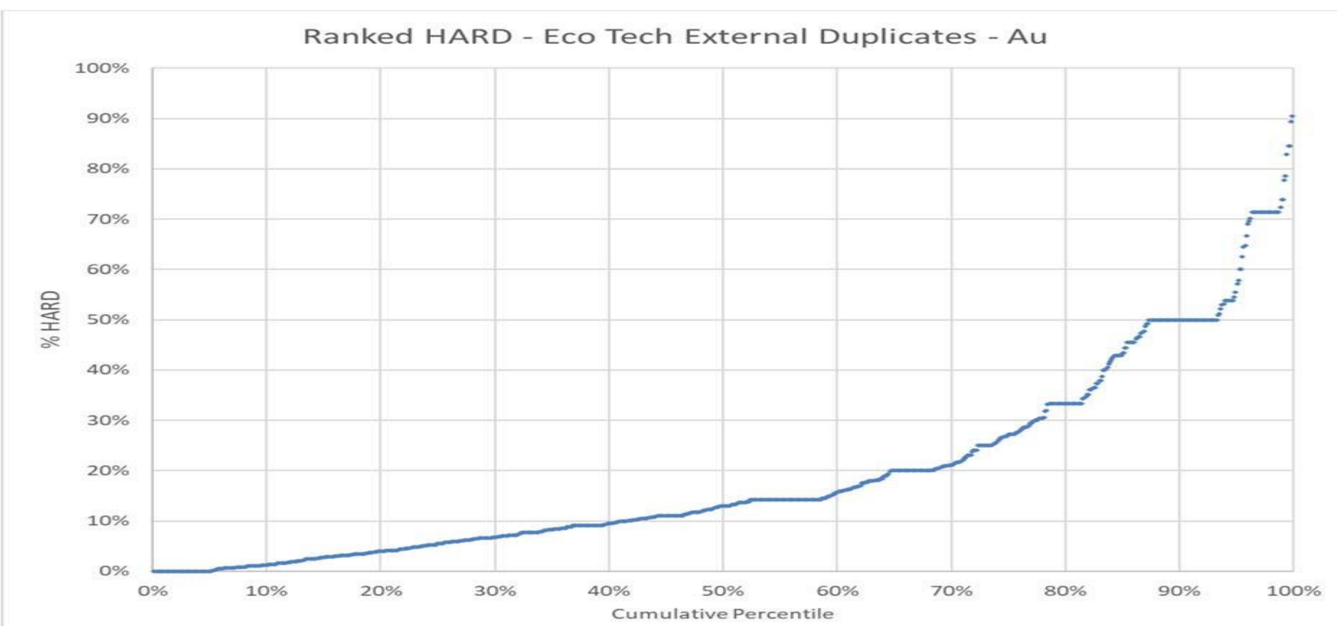
Four types of duplicates were run to assess the precision of the assay analyses; R1 = repeat, D1 = pulp duplicate, D2 = coarse duplicate, and E1 = external check. The insertion rates were 1/10, 1/20, 1/20, and 1/50 respectively. Of most interest are the external checks that are discussed here. The assay 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

		Au Grade	
Lab Comparison	Number of Duplicates	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.

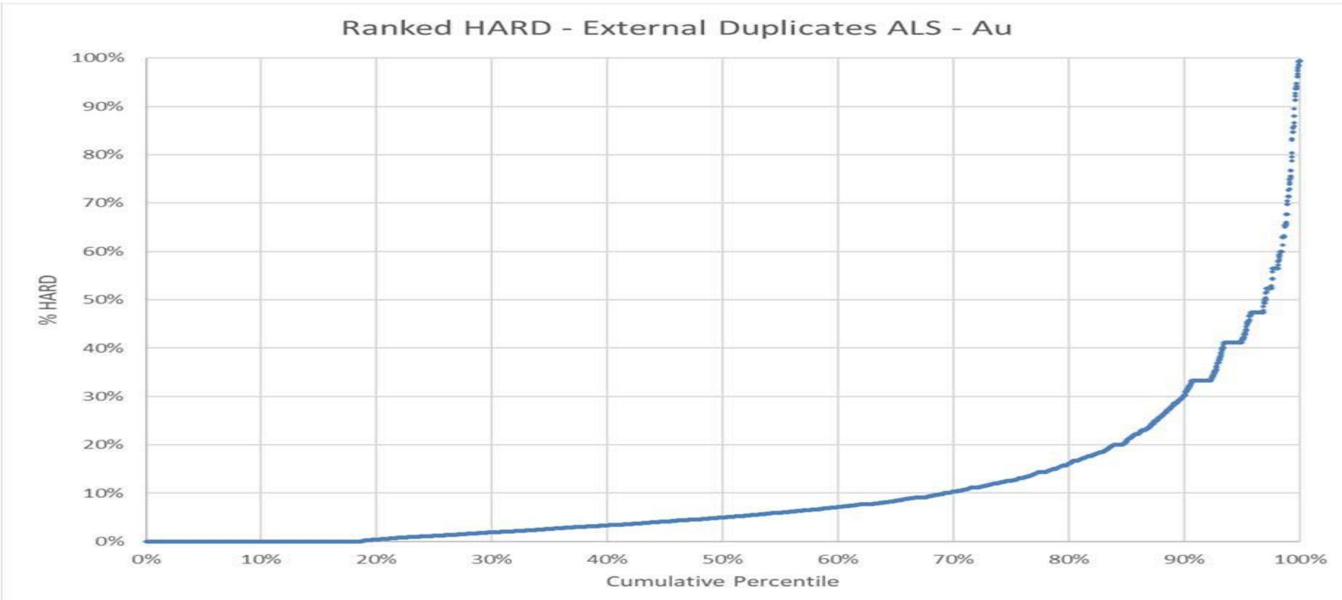
Figure 11-1: Ranked HARD Plot of Eco Tech External Duplicate Pairs – Au



Note: prepared by MMTS, 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: prepared by MMTS, 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.

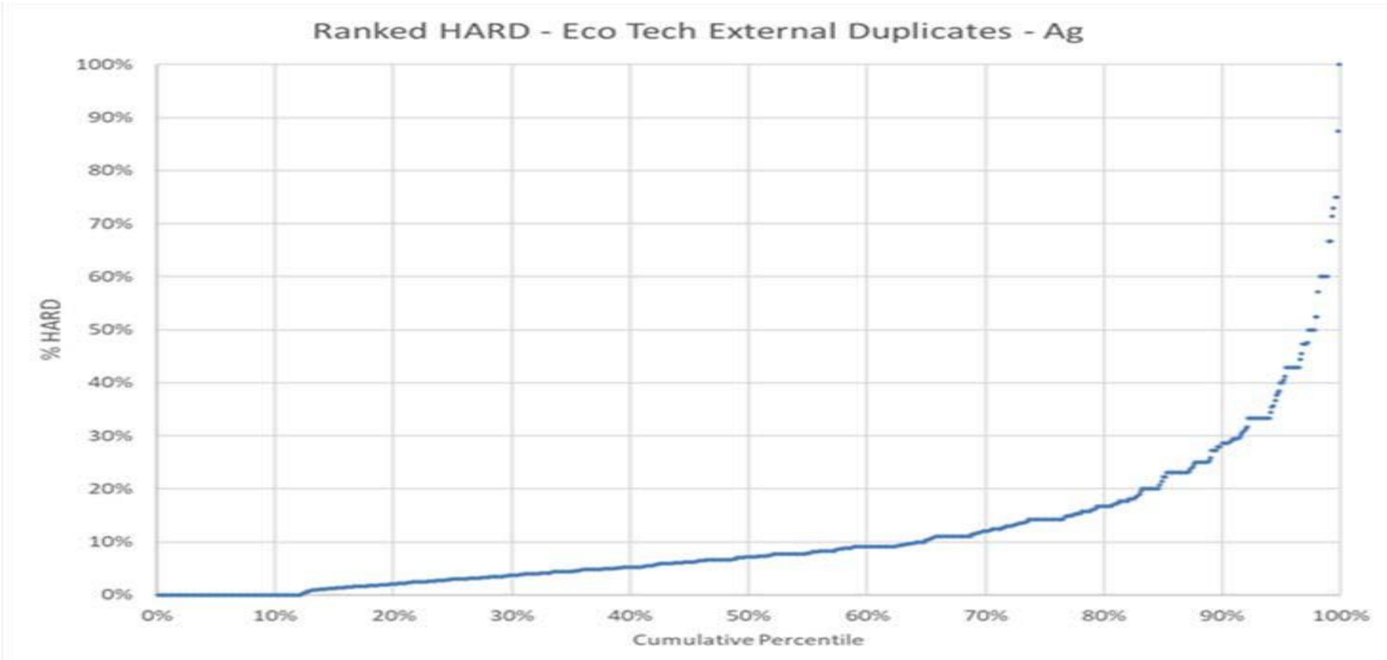
Table 11-5: Summary of External Ag Duplicate Pairs

		Au Grade	
Lab Comparison	Number of Duplicates	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 11-4 shows ranked HARD values for silver assays first done at ALS and gives approximately 63% less than 10% HARD, again considered acceptable.

Figure 11-3: Ranked HARD Plot of Eco Tech External Duplicate Pairs – Ag (Source: MMTS, 2020)



Source: MMTS, 2020

11.6.4 Field Duplicates

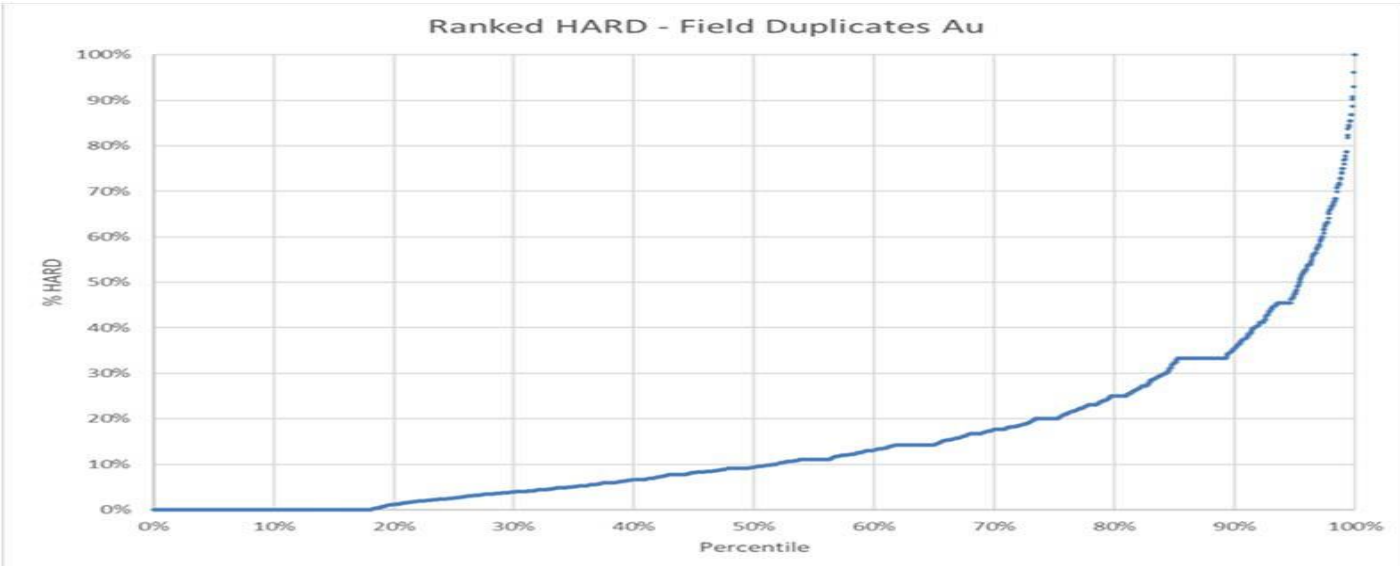
Assay results for 2,482 field duplicate pair results from 2010 to 2011 were analyzed. A summary of statistics is presented in Table 11-6 and shows agreement between means and medians, with the exception being Au mean. When the nine samples with average gold assays above 10.0 g/t were excluded, the re-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

Table 11-6: Summary of Assay Results

Parameter	S1	S2	Difference (%)
Number of Samples	2482		
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 11-4shows 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 11-4: Field Duplicates Ranked HARD plot – Au



Note: MMTS, 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. The Company 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.

11.7 Databases

The current drillhole and assay database for the Project is stored in an Access database administered from the Artemis 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 Corporate Exploration Manager. The database is hosted on the Artemis 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 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 DATA VERIFICATION

12.1 Site Visit

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

- Inspection and verification of the drillhole 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

12.2 Drillhole Database Verification

MMTS reviewed 1% of the assay database (2,137 samples) for accuracy and found no errors. The collar and survey data were validated when imported to MineSight to ensure no errors in the database or upon importing.

12.3 Other Data Verification

Verification of metallurgical, hydrological, environmental baseline and geotechnical data is discussed in the relevant sections of this Report. The data are concluded to be adequate to support the 2021 FS.

12.4 Conclusion

In the opinion of the QP, the data are appropriate to support Mineral Resource and Mineral Reserve estimates.

13 MINERAL PROCESSING AND METALLURGICAL TESTING

13.1 Introduction

The feasibility study published in 2013 presented metallurgical test data from extensive testwork carried out for New Gold Inc. from 2008 to 2013 by several well-known laboratories; these include Inspectorate, G and T Laboratories, SGS, Dawson Metallurgical Laboratories, McClellan Laboratories, Pocock and MetSolve. The results of the work led to the elimination of processing the ore using both heap leaching and flotation and concluded that whole ore leaching of the milled ore was the most appropriate method for recovering gold and silver. All recent work focused on whole ore leaching and incorporated gravity separation as an integral part of the recovery process for gold and silver.

Historical comminution testwork was carried out and the average values obtained are shown in Table 13-1.

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

Ore Type		Axb SAG Mill Breakage Parameter	BMWi Bond Ball Mill Index (kWh/t)	Ai Abrasion Index
Oxide	Average Std. Dev.	55.8 13.0	14.5 2.16	0.111 0.042
Transition	Average Std. Dev.	38.0 8.7	16.0 2.71	0.142 0.099
Sulphide	Average Std. Dev.	31.1 5.3	18.3 1.8	0.206 0.114
All Types	Average Std. Dev.	35.7 11.1	17.4 2.45	0.182 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, and at a head grade of 1 g/t Au, and 5 g/t Ag the extractions for gold and silver 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
Sulfide	87.9	44.8

13.2 2019 Testwork

An extensive program of testwork was carried out in 2019 by BaseMet Laboratories, with some additional work in 2020 The basic leach conditions were initially determined using composites made up of 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.

A further 48 samples were taken from drill holes distributed throughout the deposit. All of these were treated using the proposed flow sheet. In addition to these tests, some comminution testing was carried out and cyanide destruction was also tested, using SO_2 /air. Carbon loading characteristics and slurry rheological parameters were also determined.

13.3 Comminution Testing

Table 13-3 and Table 13-4, the first table shows the results from four twin holes drilled specifically to provide samples, the second table shows the results obtained from composites 1, 2 and 3. The results are similar to these obtained in the previous testwork, but the Bond Ball Mill Work Index is somewhat higher for the composites than for the twinned holes. This highlights the variability in the ore hardness.

Table 13-3: Grinding Tests on Twinned Holes

Sample ID	CSS µm	P_{80} µm	WiBM kWh/tonne	WiRM kWh/tonne	Ai	SMC Axb
BW91:108-122m	212	137	14.4		0.066	27.2
BW624:85.7-108m	212	159	17.7	16.3	0.295	37.1
BW832:45-65m	212	151	11.8	15.0	0.072	36.1
BW832:79.5-95m	212	153	13.8		0.103	52.0

Note- WiBM – Bond ball mill work index, WiRM – Bond rod mill work index, Ai – abrasion index, SMC- breakage parameter

Table 13-4: Grinding Tests on Composites

Sample	CSS µm	P_{80} µm	WiBM kWh/tonne
Comp 1	212 150 106	157 109 78	21.1 20.3 19.3
Comp 2	212 150 106	157 109 78	19.4 18.3 17.4
Comp 3	212 150 106	157 109 78	19.8 19.2 18.9

Note – CSS closed side setting; WiBM = Bond ball mill work index;

The extremely variable results for work index made the sizing of a semi-autogenous grind (SAG) mill/ball mill combination difficult, choosing 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. Taking into account the low abrasion characteristics and more modest tonnage than the 2013 FS, it was decided to opt for three stage crushing and a single ball mill.

13.4 Gravity Concentration

A blend of the composites was used to provide enough sample to assess the effectiveness of gravity concentration and the results presented in Table 13-5 were obtained.

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 P_{80} = 150 µm, 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 P_{80} of 150 µm did not give a significant increase in gold recovery.

13.4.1 Sample Characterization

The analysis of the three composites are shown in Table 13-6.

Table 13-6: Composition of the Three Composites

Sample	Au ppm	Ag ppm	Assays - percent or g/tonne			SO_4 %	S(s) %
			Cu %	Fe %	S(t) %		
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) =

A quantitative evaluation of materials by scanning electron microscope (QEMSCAN) analysis was carried out to determine the minerals present and the sulfur distribution. The results are shown in Table 13-7.

Table 13-7: QEMSCAN Analysis Results

Mineral	Mineral Content - Percent of Total Sulfur		
	Composite 1	Composite 2	Composite 3
Cu sulfides	1.7	0.9	4.9
Galena	0.1	0.7	0.7
Sphalerite	13.8	18.4	9.5
Pyrrhotite	21.4	14.6	15.5
Arsenopyrite	0.1	0.8	1.4

Table 13-8 provides the mineralogical analysis of the three composites tested in 2019.

Table 13-8: Mineralogical Analysis of the Three Composites

Minerals	Mineral Content - percent		
	Composite 1	Composite 2	Composite 3
Copper sulfides	0.1	0.1	0.2
Chrysocolla	<0.1	<0.1	<0.1
Galena	<0.1	0.1	0.1
Sphalerite	0.6	1.1	0.4
Pyrite	1.6	2.3	1.9
Pyrrhotite	0.8	0.7	0.5
Arsenopyrite	<0.1	0.1	0.1
Iron oxides	0.4	0.6	0.4
Quartz	40.5	49.1	47.4
K-Feldspar	26.5	18.5	29.4
Muscovite	12.9	15.8	14.2
Biotite/Phlogopite	13.2	8.3	0.8
Plagioclase Feldspar	0.8	1	1.3
Chlorite	1.4	1.6	2.5
Epidote	0.4	<0.1	<0.1
Kaolinite (clay)	0.4	0.2	0.4
Rutile / anatase	0.1	<0.1	<0.1
Mn-limonite	0.1	0.1	<0.1
Apatite	0.3	0.3	<0.1
Others	0.1	0.2	0.3

Note: Iron oxides include goethite, Limonite and iron minerals. Others include trace amounts of barite and unresolved mineral species.+

Table 13-8 shows that significant levels of copper and zinc are present. Leaching testwork (see section 13.2) showed that three samples from the 48 chosen 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.

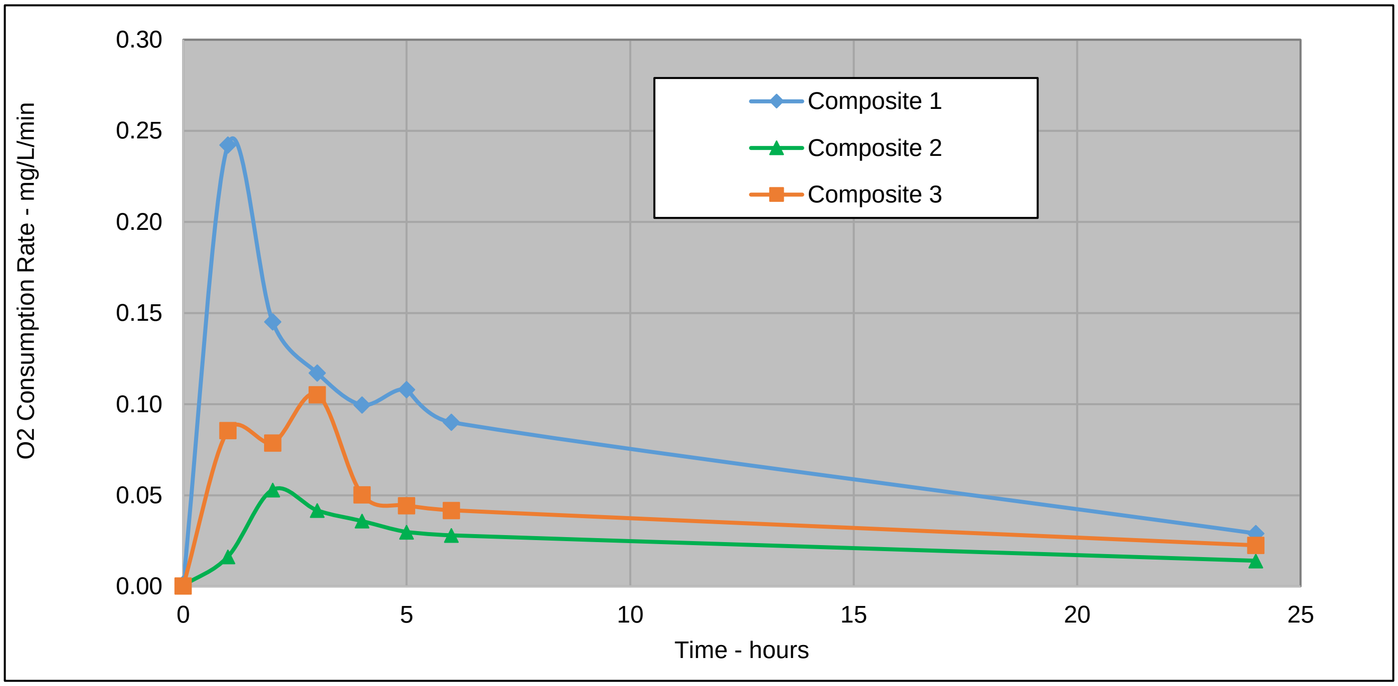
Arsenic was present, 0.1% of arsenopyrite being reported in two of the composites but appeared to have no effect on gold leaching except for one sample which was very high in arsenic (8,950 ppm) and which gave low gold extraction (sample VC25, see section 13.2.6).

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

13.5 Oxygen Uptake Rate

The rate of oxygen uptake was measured for each of the composites (Figure 13-1). 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.

Figure 13-1: Oxygen Uptake Rate for the Three Composites



Note: prepared by Base Met laboratories report BL0452, December 2019

13.6 Leach Testing

The results of the series of tests carried out on the three composites are shown in Table 13-9. A comparison between tests 2-6 and tests 7-11 shows that gravity concentration increases overall recovery. Comparison of tests 8, 12 and 13 shows no difference between initial cyanide concentrations of 500, 1000 and 1500 g/t. Note that up to test 11, an initial cyanide concentration of 1000 mg/L was used. 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.

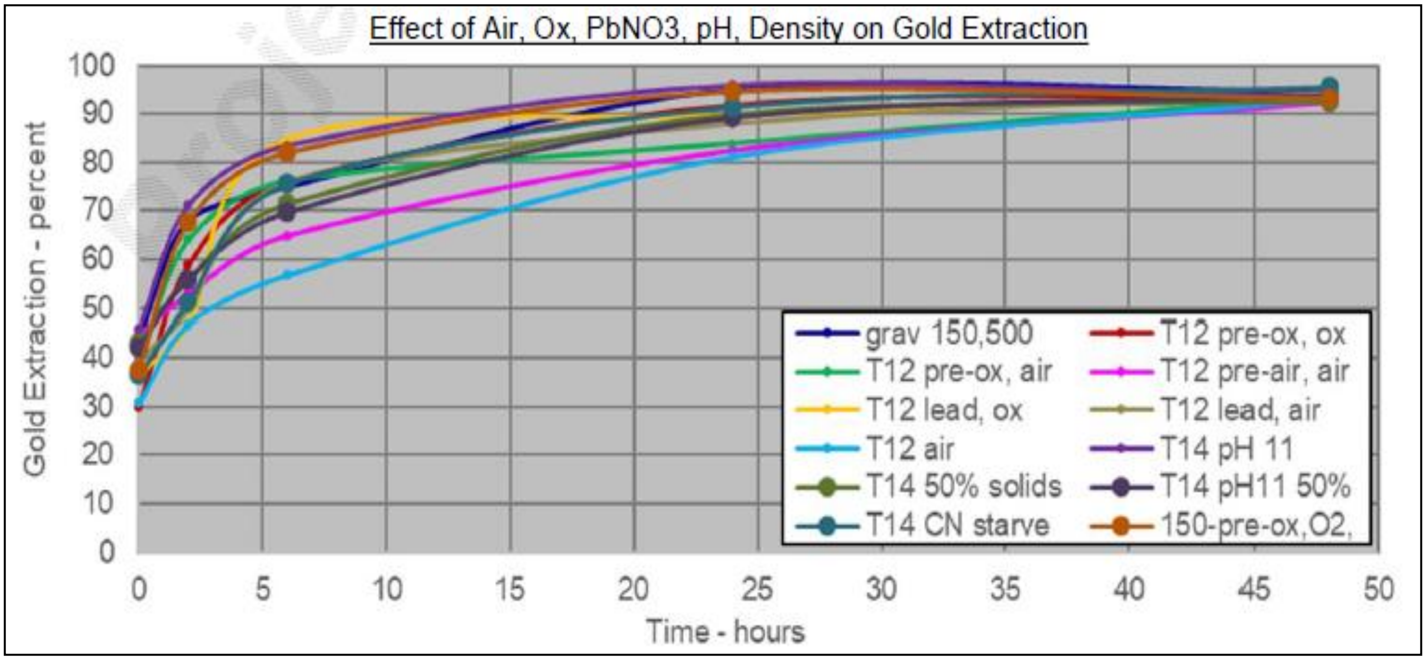
All tests were carried out at 40% solids, apart from test 22 where 50% solids was used. The use of this higher solids percentage appeared to reduce the overall recovery by about 2% but this requires confirmation.

In general, oxygen was sparged into the pulp during leaching and the Figure 13-2 shows that leach kinetics are significantly slower when air is used but the final extractions obtained are very close to those obtained using oxygen. It should be remembered that composite 1, upon which most of the tests were carried out, had the highest oxygen demand of the three composites.

Table 13-9: Results of Leach Conditions Optimization Tests

Sample ID	Test	Conditions Grind, NaCN, Lead Nitrate	Extraction - percent			Reagent Consumption Kg/tonne	
			Au Pan Con	Au 48hrs	Ag 48hrs	NaCN	Lime
Comp 1	1	Rougher 106		90.4	58.4	-	-
	1A	Ro Con Leach	17.0	93.4	45.9	10.1	9.87
	1B	Ro Tl Leach	-	56.9	39.3	1.33	0.61
		Overall	-	89.9	43.2	1.33	-
	2	212	-	91.8	49.1	1.53	1.33
	3	150	-	91.8	56.0	1.31	1.40
	4	106	-	94.9	60.5	1.68	1.40
	5	75	-	95.3	62.0	1.92	1.65
	6	53	-	95.7	61.0	2.15	1.80
	7	grav 212	32.7	90.6	64.3	1.45	1.35
	8	grav 150	41.8	94.6	60.2	1.51	1.35
	9	grav 106	43.4	95.5	67.3	1.65	1.71
	10	grav 75	58.3	96.6	65.3	1.89	1.69
	11	grav 53	58.5	96.5	66.3	2.18	1.69
	12	grav 150, 500	44.1	94.7	62.2	1.33	1.50
	13	grav 150, 1500	43.6	94.7	6.2	1.80	1.40
	14	T12 pre-ox, ox	30.0	94.0	58.0	0.96	1.56
	15	T12 pre-ox, air	39.2	93.3	59.2	0.92	1.65
	16	T12 pre-ox, air	44.9	92.2	56.8	1.10	1.54
	17	T12 lead, ox	36.9	93.4	59.6	1.12	1.24
	18	T12 lead, air	39.6	94.3	61.7	1.03	1.43
	19	T12 air	30.8	93.1	59.0	1.01	1.34
	20	T14 pH 11	46.0	93.7	59.3	0.90	1.95
	21	T14 50% solids	43.3	92.6	56.6	0.86	1.47
	22	T14 pH 11 50%	42.3	93.8	58.4	0.84	1.82
	23	T14 CN starve	36.5	95.6	62.9	0.98	2.09
	46	150-pre-ox, O_2,	37.7	93.2	53.6	1.16	1.42
	76	no grav	-	90.3	58.9	0.82	2.07
Comp 2	24	150, pre-ox, 500	41.5	93.8	59.3	0.67	1.15
	77	No grav	-	92.1	52.1	0.61	1.48
	79	Slurry gen. for detox	64.3	90.5	49.5	0.77	1.03
Comp 3	25	150, pre-ox, 500	41.2	95.4	65.9	0.70	1.32
	78	no grav	-	95.3	67.5		1.58

Figure 13-2: Effect of Various Leach Conditions on Gold Extraction



Note: prepared by Base Met laboratories report BL0452, December 2019

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.

13.7 Leach Variability Testing

To investigate the variability of the extraction over the orebody, 48 samples were chosen, 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. The results obtained are summarized in the Table 13-10.

Table 13-10: Results of Gravity Concentration and Leaching 48 Samples from Throughout the Ore Body

Sample Number VC	Head Grade g/t		Au Ext %	Ag Ext %	Cu ppm	Reagent Consumption		Bond Wi kW-hr/ton
	Gold	Silver				NaCN kg/t	Lime kg/t	
2	3.09	8	95.5	52.3	199	1.0	1.0	15.6
3	4.42	13	96.2	64	429	1.5	0.7	19.8
4	0.43	5	93.2	54.7	176	0.5	1.1	21.5
5	0.46	2	92.4	30.8	206	0.5	1.5	21.6
6	0.62	3	91.9	35.4	63	0.5	0.7	
7	0.79	4	99.2	75.6	2640	5.7	2.9	21.1
8	0.32	2	91.8	68.3	90	0.7	0.9	

Sample Number VC	Head Grade g/t		Au Ext %	Ag Ext %	Cu ppm	Reagent Consumption		Bond Wi kW-hr/ton
	Gold	Silver				NaCN kg/t	Lime kg/t	
9	2.1	22	98.4	61.4	115	0.2	0.9	14
10	6.21	10	94.3	76.1	789	1.4	0.8	16
11	0.61	6	91.4	96.9	63	0.2	1.5	11.8
12	4.71	49	94.2	87.9	131	0.3	0.3	12.1
13	3.66	15	92.9	78.6	135	0.2	0.8	16.8
14								
15	1.79	8	96.9	65.9	133	0.5	0.8	15.1
16	1.47	4	83.0	36.4	40	1.0	1.0	18.8
17	1.63	3	92.1	69.7	182	0.7	1.0	19.7
18	11.95	15	98.3	65.3	70	1.1	3.2	16.1
19	4	9	98.9	78.2	529	0.1	0.1	17.7
20	0.65	13	95.3	87.0	172	1.1	1.7	19.7
21	0.94	3	94.6	69.1	232	0.8	1.0	24.6
22	0.75	10	97.7	97.0	43	0.8	0.6	14.4
23	1.18	5	97.4	48.7	167	0.8	1.3	20.5
24	0.9	2	93.1	87.7	98	1.0	1.4	20.2
25	0.59	8	63,4	66.3	101	1.5	0.5	
26	0.69	5	84.2	93.7	45	0.8	1.0	13.9
27	1.36	9	98.9	66.3	179	1.4	2.2	
28	2.22	3	93.6	69.1	206	0.8	0.8	22.1
29	0.81	13	97.7	90.9	76	0.7	0.7	20.4
30	1.2	7	96.0	56.6	175	0.9	1.0	21.1
31	0.75	5	94.0	83.5	89	1.0	2.0	21.4
32	0.57	2	98.6	78.3	75	1.2	0.9	18.2
33	1.6	40	80,3	66.8	296	1.5	0.8	18.2
34	0.92	4	96.9	65.3	60	1.0	1.1	
35	0.83	3	96.2	91.0	51	0.8	1.1	16.6
36	0.94	13	90.3	57.4	380	1.0	0.6	19.2
37	2.95	37	87.3	89.8	1940	4.5	0.9	19
38	1.74	5	97.2	94.6	363	1.1	1.6	17.7
39	0.3	6	99.2	57.9	292	0.9	0.3	
40	2.81	5	92.5	94.2	418	1.0	0.9	20.3
41	1.09	7	95.7	93.9	99	0.9	1.1	
42	1.41	10	80.8	68.7	1340	2.0	0.4	19.4
43	0.48	12	94.8	52.6	424	1.3	1.0	21.8
44	1.59	6	94.4	60.5	105	0.9	1.1	
45	2.33	8	96.9	85.6	335	0.7	1.2	22.1
46	1.07	3	91.4	70.5	54	0.6	0.7	

Sample Number VC	Head Grade g/t		Au Ext %	Ag Ext %	Cu ppm	Reagent Consumption		Bond Wi
	Gold	Silver				NaCN kg/t	Lime kg/t	kW-hr/ton
47	8.77	9	96.2	58.3	195	0.9	0.9	
48	1.11	25	92.3	88.3	58	0.6	0.4	
49	1.66	13	91.3	85.2	140	0.5	0.9	
50	0.79	7	91.8	74.5	255	1.0	1.7	
Mean	1.83	9.36	92.6	69.8	301	1.0	1.1	18.5

Note: Test 14 was lost

Of the 48 drill composites tested, 6 gave overall extractions (gravity +leach) of less than 90%, all the other samples gave results significantly greater than 90%. The test report has details of the drill holes and intervals of drill core that was used to make up the composites. Of the samples which initially showed low extractions. VC7, VC37 and VC42 all had high copper contents, of 2640, 1940 and 1340 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 as shown in the table, when cyanide concentration was raised. These compare with all the other samples which had an average of 190 ppm copper if these 3 samples were ignored.

Another of the low recoveries (VC-25) occurred in hole BW 0979, which had an abnormally high arsenic content (8950 ppm) and was low in gold grade, as was sample VC26.

It is presumed that in the other 3 cases, still more 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, VC-33 (hole BW 839) and VC46 (hole BW – 91) require further investigation.

13.8 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 are shown in the Table 13-11.

Table 13-11: Results of Cyanide Destruction Testwork

Sample	Detox Test	Target WAD CN	pH	Ret'n Time	Reagents Used		Test Length		Feed/Detox Solution Assays - ppm				
				Mins	SO_2 g/g CN_{MP}	Cu Mg/L	Mins	Number of Displacements	CN_{MP}	Cu	Fe	Ni	Zn
T 46 (Comp 1)	Feed	-	10.3	-	-	-	-	-	367.4	161	9	0.95	55.5
	C1 C2	25	8.4 8.6	60 60	4.0 3.0	15 15	120 180	2 3	4.8 26.3	21.6	<0.2	<0.1	<0.1
	C3 C4	10	8.5 8.6	60 60	4.0 3.5	15 15	240 270	4 5	3.7 7.5	5.4	0.4	<0.1	<0.1
	C5 C6 C7 C8 C9	1	8.5 8.5 8.5 8.5 8.5	60 60 60 60 60	4.0 5.0 4.5 4.5 4.5	15 15 15 0 15	240 180 180 180 480	4 3 3 3 8	3.2 0.3 0.2 14.4 0.2	1.31 0.79	<0.2 0.60	<0.1 <0.1	<0.1 <0.1
T79 (Comp 2)	Feed	-	10.5	-	-	-	-	-	273.0	28.1	3.77	0.86	95.1
	C10 C11 C12 C13	25 10 1 1	8.5 8.5 8.5 8.5	60 60 60 60	3.0 3.5 3.5 3.5	15 15 0 7.5	300 180 60 240	5 3 1 4	11.5 0.4 11.4 0.2	15.6 1.00 5.8 1.09	<0.2 <0.2 <0.2 <0.2	<0.1 <0.1 0.18 <0.1	0.1 0.19 0.2 0.23
T90 (Comp 3)	Feed	-	10.3	-	-	-	-	-	264.0	73.7	12.83	0.485	58.7
	C14 C15 C16	25 10 1	8.5 8.5 8.5	60 60 60	3.0 3.5 4.0	15 15 15	240 240 180	4 4 3	24.8 7.5 0.3	32.9 4.86 1.33	0.10 0.11 <0.1	<0.1 <0.1 <0.1	<0.1 0.10 <0.1

Note: *C1 target WADCN was 25ppm

The results show that very low levels of CNWAD were obtained, using a 60 minute retention time, an initial copper catalyst concentration of 15 mg/l and SO_2:CNWAD 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 and are available.

13.8.1 Carbon Loading

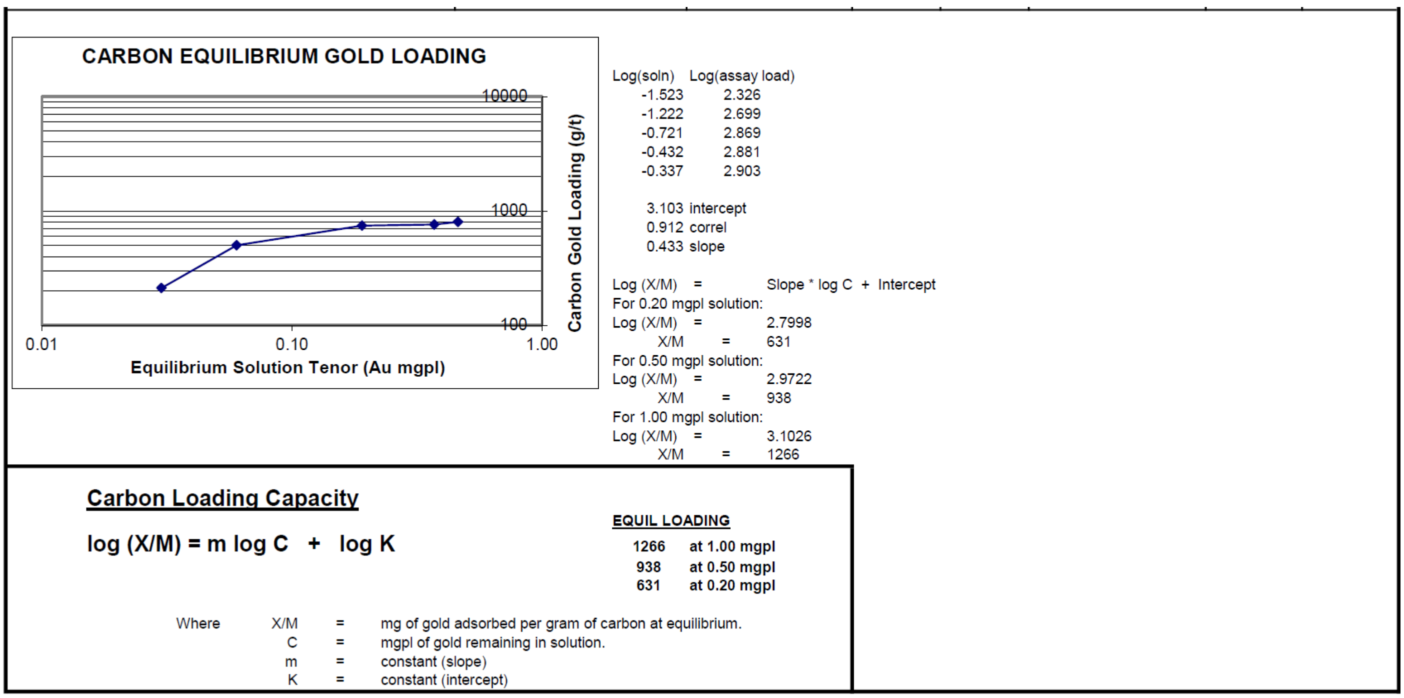
The leach solution produced from composite 3 was used for carbon loading testwork. A sample of carbon was obtained from Quadra Chemicals (type PJ612G-SS240), the likely supplier of carbon for the project. The results are shown in the Table 13-12:

Table 13-12: **Gold and Silver Loading on Carbon**

Composite	Test	Carbon Concentration (g/L)	Carbon Added (g)	Solution mL	Initial Au Mg/L	Initial Ag Mg/L	Final (48 hours) Au Mg/L	Final (48 hours) Ag Mg/L	Loading (48 hours) Au g/t (calc)	Loading (48 hours) Ag g/t (calc)
Year Comp 3	N1	2.50	2.13	850	0.56	3.0	0.03	0.17	212	1148
	N2	1.00	0.85	850	0.56	3.0	0.05	0.7	510	2360
	N3	0.50	0.43	850	0.56	3.0	0.17	1.4	780	3320
	N4	0.25	0.21	850	0.56	3.0	0.34	2.1	880	3840
	N5	0.13	0.11	850	0.56	3.0	0.42	2.4	1120	5040

The analysis of the results for gold are presented in Figure 13-3.

Figure 13-3: **Gold Loading on Carbon**



Note: prepared by Base Met Laboratories report BL 0659 October, 2020

13.8.2 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 sulfides 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 results are shown in Table 13-13:

Table 13-13: Bath Peroxide Cyanide Detoxification Test

Sample	BL769-03 Year Comp 1			BL769-04 Year Comp 3		
	B1	B2	B3	B4	B5	B6
Reactor Size (L)	0.4	0.4	0.4	0.4	0.4	0.4
Elapsed Time (hr)	2	2	2	2	2	2
Feed Volume (L)	0.200	0.200	0.200	0.200	0.200	0.200
Solution Volume (L)	0.200	0.200	0.200	0.200	0.200	0.200
Soln (S.G.)	1.00	1.00	1.00	1.00	1.00	1.00
Slurry (S.G.)	1.00	1.00	1.00	1.00	1.00	1.00
H_2O_2 CN(wad) Ratio	2	4	6	2	4	6
Mass H_2O_2 (g)	0.07	0.15	0.22	0.07	0.14	0.22
H_2O_2 Factor	1.31	1.31	1.31	1.31	1.31	1.31
Cu in Solution (mg/L)	50	50	50	50	50	50
$CuSO_4.5H_2O$ Cu Factor	0.25	0.25	0.25	0.25	0.25	0.25
$CuSO_4.5H_2O$ Added (g)	0.04	0.04	0.04	0.04	0.04	0.04
pH	8.5	8.5	8.5	8.5	8.5	8.5
Lime Addition (g)	2.04	2.04	2.04	2.04	2.04	2.04
Target Final CN_{MP} (ppm)	<1	<1	<1	<1	<1	<1
Initial CN_{MP} (ppm)	244	244	244	237	237	237
Final CN_{MP} (ppm)	2.4	<1	<1	2.6	<1	<1

It can be seen that the WAD cyanide 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 WAD cyanide and 50 mg/l copper as catalyst, removal of WAD cyanide down to <1 mg/l was obtained.

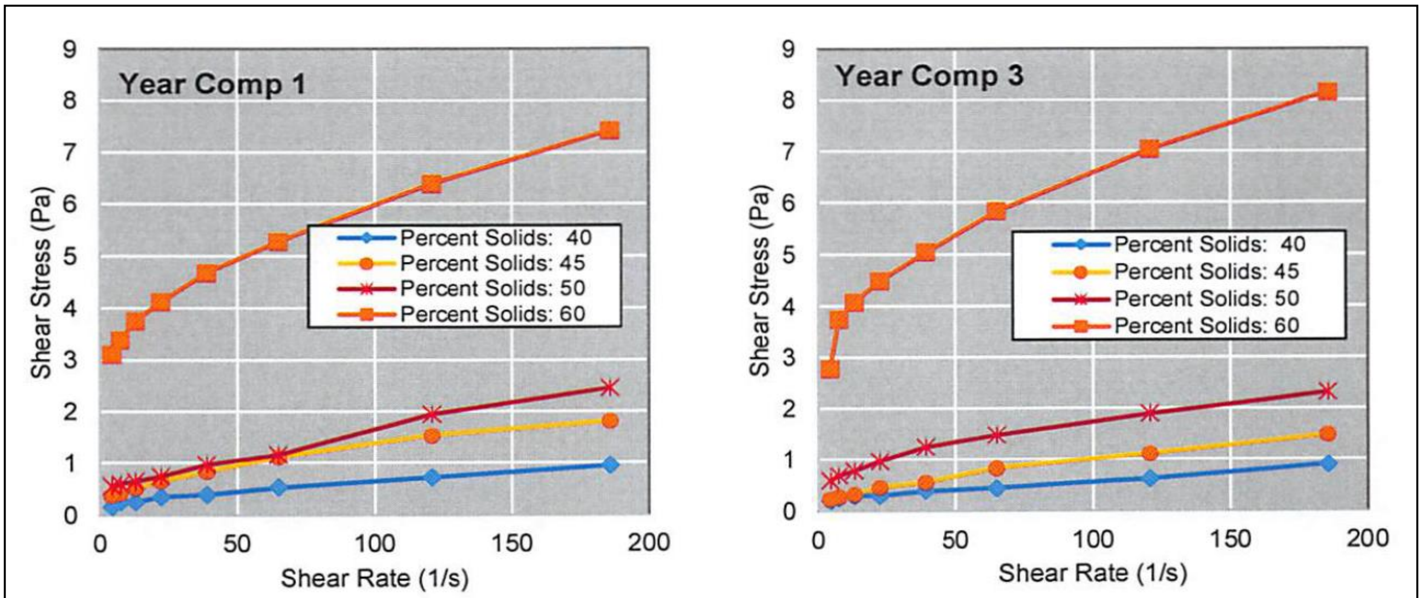
Slurry Viscosity

The sheared viscosity properties of the cyanide tailings from test 3 and 4 were determined using a Brookfield DV2T viscometer. The results are shown in Table 13-14 and Figure 13-4.

Table 13-14: Tailings Viscosity Properties

Sample	Density %	Viscosity (CPS) at Shear Rate (s^{-1})							
		185.5	120.5	64.9	38.9	22.3	13.0	7.4	4.2
Year 1 Comp Test 3 CNTI	40	5	6	8	10	15	18	31	33
	45	10	13	17	21	29	39	56	89
	50	13	16	18	25	33	50	81	133
	60	40	53	81	120	185	289	456	744
Year 3 Test 4 CNTI	40	5	5	6	10	13	21	31	44
	45	8	9	13	14	21	25	38	56
	50	13	16	23	32	44	61	94	144
	60	44	58	90	130	202	314	506	667

Figure 13-4: Viscosity Properties of Cyanide Tailings: Test 3 and 4 Results



Note: prepared by Base Met Laboratories report BL 0769 September, 2020

Gold Recovery

The average extractions for the three composites representing the first 5 years of mining were 94.4% gold and 61.5% silver and the average gold extraction for the 48 variability composites were 93.1% gold and 69.8% silver. Taking into account all of these results, the use of a recovery of 93% gold and 65% 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 0.6kg/t is used.

14 MINERAL RESOURCE ESTIMATES

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 Sue Bird, P.Eng.

14.1 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.

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 (CPPs) and coefficients of variation (CVs) of the gold grades by domain, as discussed in Section 14.4. 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 million 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.

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

			In-situ Grades			In-situ Contained Metal		
Classification	Cut-off (g/t AuEq)	Tonnage (k)	AuEq (g/t)	Au (g/t)	Ag (g/t)	AuEq (koz)	Au (koz)	Ag (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
	0.70	3,328	1.20	0.97	36.9	128	104	3,951

Notes:

1. The Mineral Resource estimate was prepared by Sue Bird, P.Eng., the Qualified Person for the estimate and employee of MMTS. The estimate has an effective date of May 5, 2020.
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.

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.

14.2 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 drillholes within the Blackwater block model and used for interpolation is provided in Table 14-2.

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

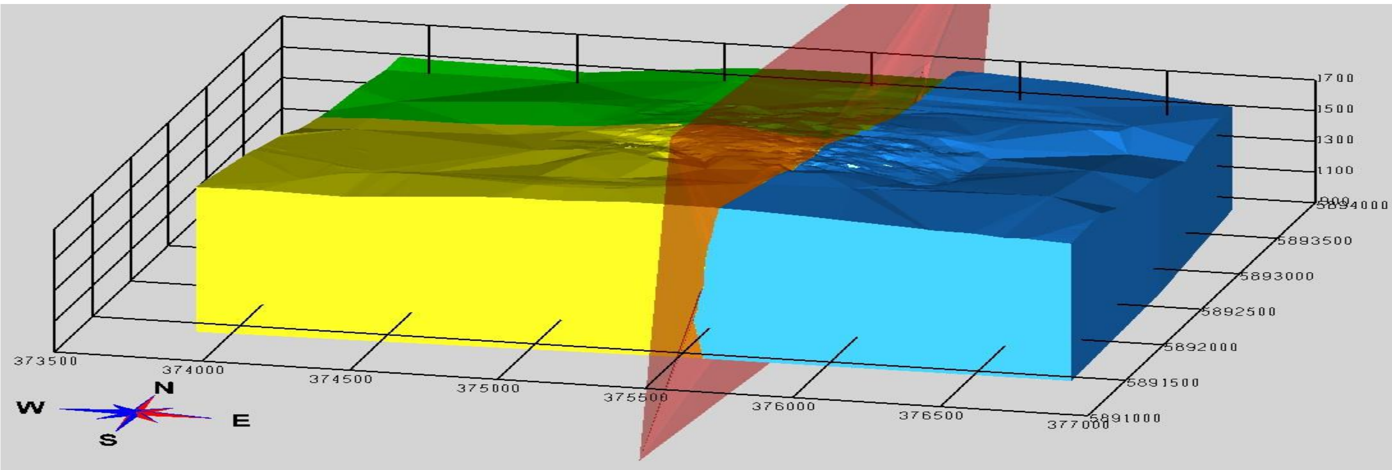
Year	Company	Holes	Metres	Intervals Assayed	Metres Assayed	% Assayed
2009	Richfield	18	3,621	3,450	3,414	94.3
2010	Richfield	56	20,920	20,172	20,048	95.8
2011	New Gold	125	49,316	46,008	46,231	93.7
2011	Richfield	59	19,727	18,484	18,841	95.5
2012	New Gold	743	215,289	200,211	203,416	94.5
2013	New Gold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

An additional 12 drillholes 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.

14.3 Geologic Modelling

The geologic, 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. 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



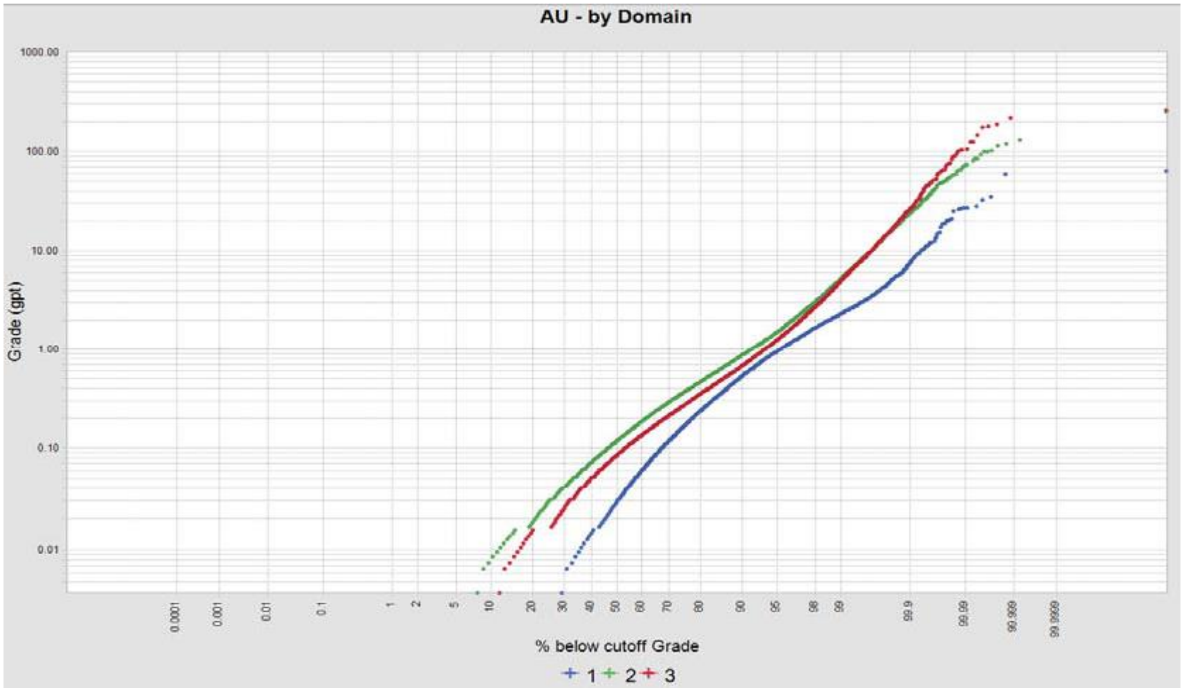
Note: prepared by MMTS, 2020 Green = 1, yellow = 2, blue = 3, red = major N–S fault

14.4 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 14-3illustrate the CPPs by domain for gold and silver respectively.

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.

Figure 14-2: CPP of Au Assay Grades by Domain



Note: prepared by MMTS, 2020

Figure 14-3: CPP of Ag Assay Grades by Domain

Note: prepared by MMTS, 2020

Table 14-3: Summary Statistics of Assays and Composites

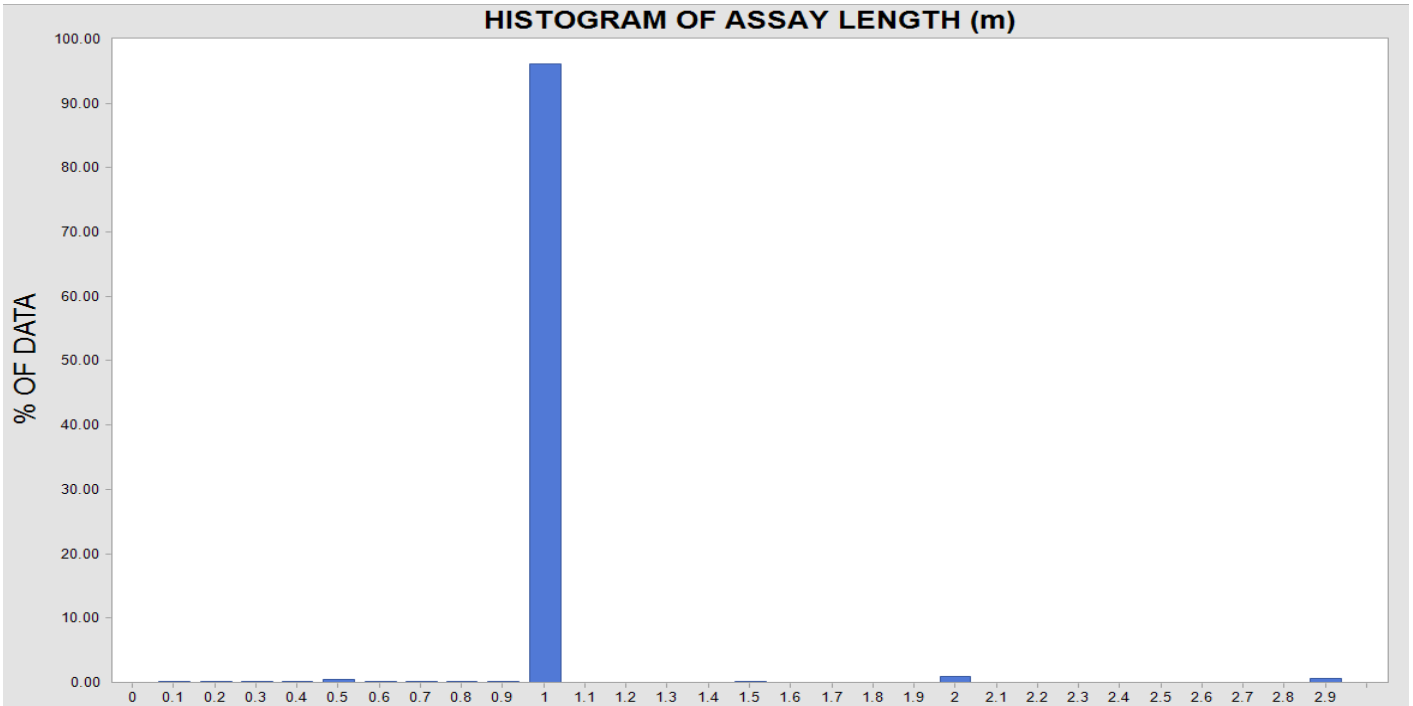
Source	Parameter	Au By Domain			Ag By Domain		
		1	2	3	1	2	3
Assays	Num Samples	65,382	137,890	85,466	65,382	137,890	85,465
	Num Missing Samples	215	49	39	21	49	39
	Min (g/t)	0.003	0.003	0.003	0.10	0.10	0.10
	Max	63.70	252.00	262.00	1045.00	2170.00	1950.00
	Wtd mean (g/t)	0.208	0.446	0.401	3.94	3.04	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	24	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.

14.5 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. 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.

Figure 14-4: Histogram of Assay Lengths



Note: prepared by MMTS, 2020

Composite statistics, for the capped values are summarized in Table 14-14. The C.V. for Au remains rather high, further pointing to MIK as an appropriate interpolation method for gold interpolation. The capping for silver reduced the C.V. 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
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

14.6 Density Assignment

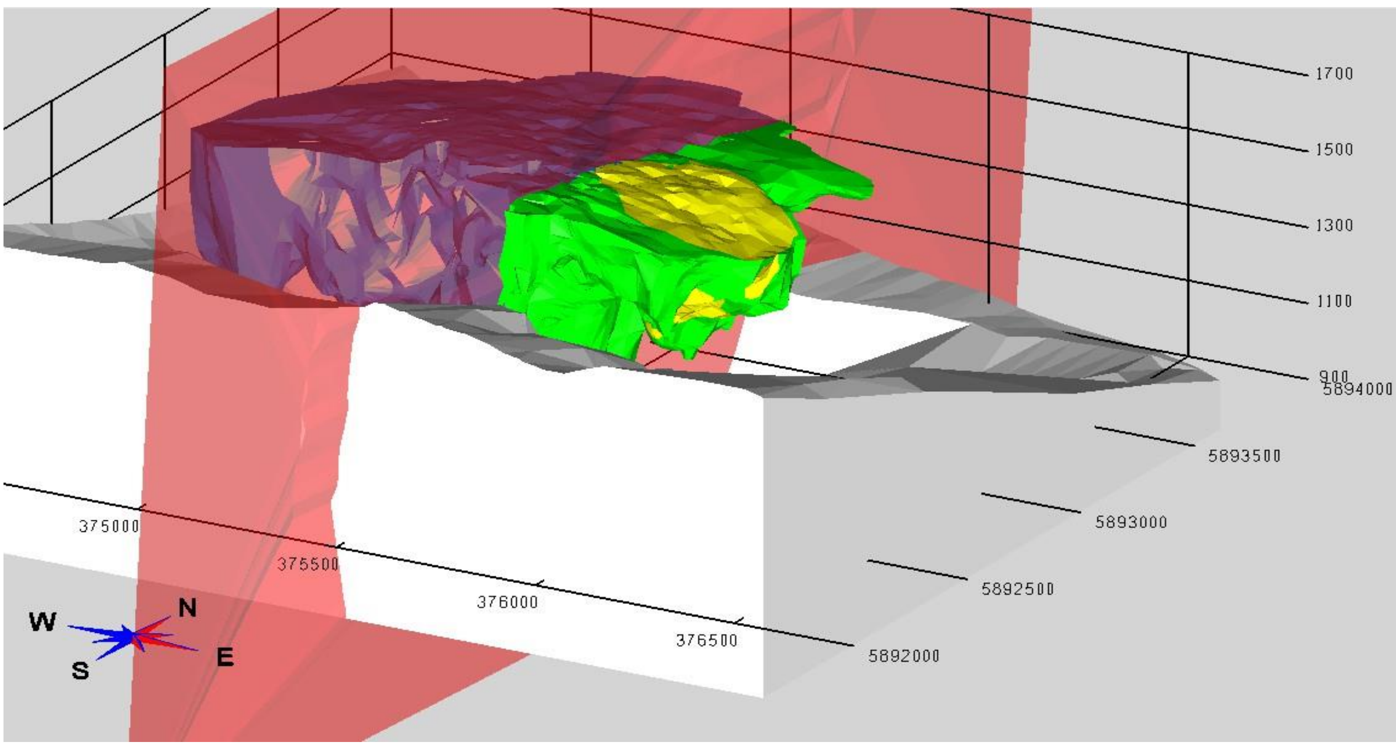
Model blocks were assigned the mean specific gravity value based on lithology and alteration as summarized in Table 14-5

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: prepared by MMTS, 2020. white = sediments, yellow=central sericite, green=east sericite, blue=west sericite, red=fault

14.7 Block Model Interpolations

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

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

14.8 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.

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

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.

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	0.35	0.45	75	50	40	720	300	270
	2	0.146	0	0.35	0.45	75	50	40	720	300	270
	3	0.323	0	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	0	0.3	60	50	40	650	280	220
	6	1.075	0	0	0.3	40	30	30	500	250	220

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")
	7	1.496	0.55	0.25	0.2	40	30	20	400	220	180
	8	2.133	0	0	0.1	20	20	10	250	180	150
	9	3.786	0	0	0	30	10	5			
2	1	0.003	0	0	0.3	50	60	40	200	230	250
	2	0.257	0	0	0.3	50	60	40	200	230	250
	3	0.478	0	0	0.2	50	60	40	200	230	250
	4	0.765	0	0	0.2	30	60	40	160	200	250
	5	1.185	0	0	0.2	20	20	25	120	130	150
	6	1.893	0	0	0.1	30	30	35	120	130	140
	7	3.350	0	0.15	0.05	50	30	50	90	100	110
	8	6.241	0	0	0	45	25	60			
	9	14.244	0.95	0.05	0	30	25	60			
3	1	0.003	0	0	0.3	50	60	40	200	230	250
	2	0.212	0	0	0.3	50	60	40	200	230	250
	3	0.433	0	0	0.2	50	60	40	200	230	250
	4	0.766	0	0	0.2	30	60	40	160	200	250
	5	1.296	0	0	0.2	20	20	25	120	130	150
	6	2.244	0	0	0.1	30	30	35	120	130	140
	7	4.272	0	0.15	0.05	50	30	50	90	100	110
	8	9.143	0	0	0	45	25	60			
	9	25.165	0	0	0	45	25	60			

Table 14-9: Summary of Correlogram Parameters for Ag

Dom	C0	C1	C2	Ranges – Spherical - 1				Ranges – Spherical - 2			Ranges – Spherical - 3		
				C3	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				

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

14.9 Classification of Mineral Resources

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

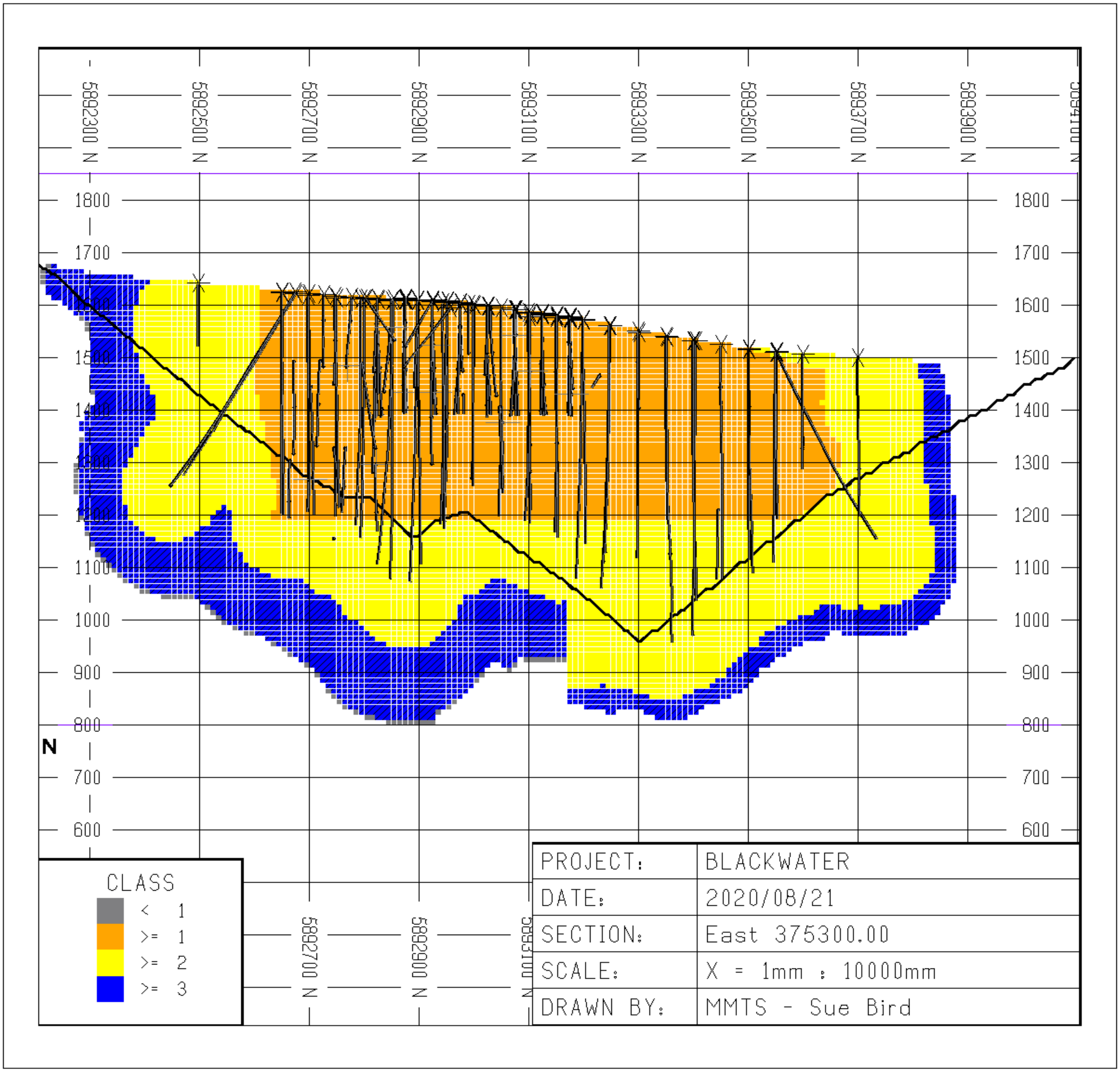
Table 14-11: Summary of Initial Classification Parameters

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 14-1.

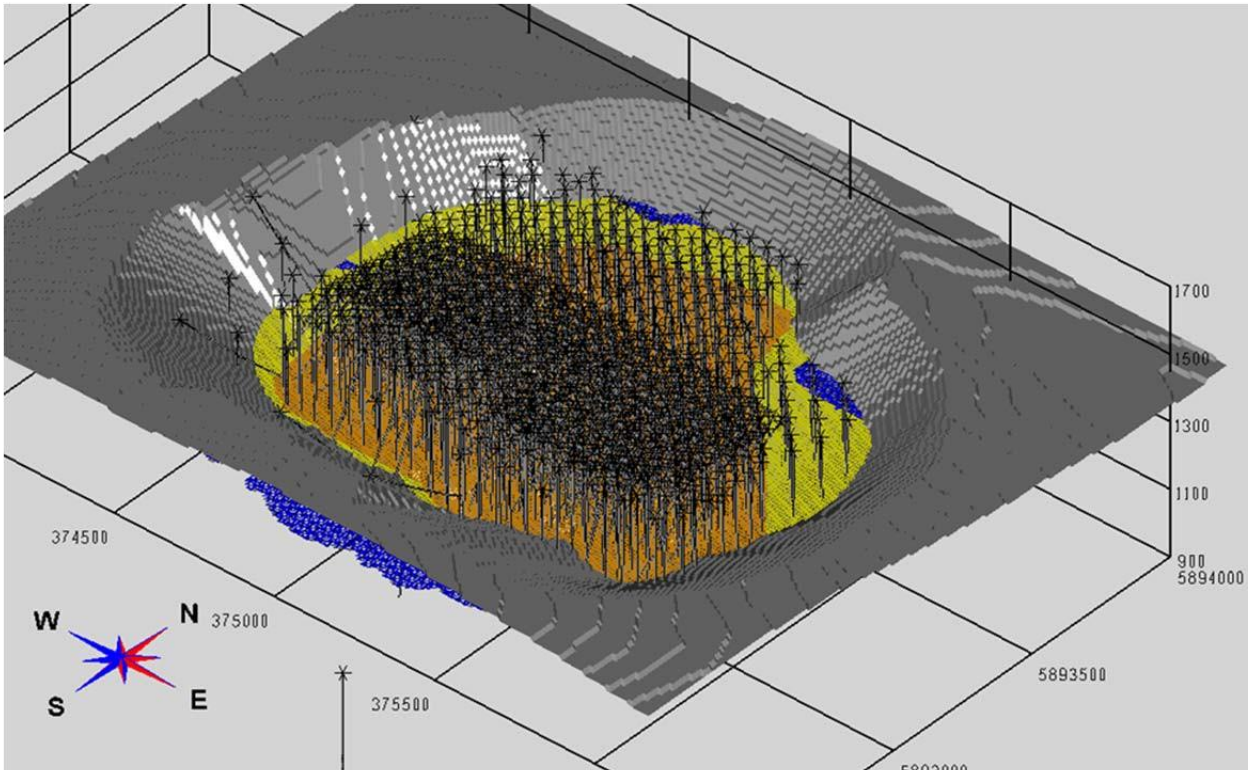
Figure 14-6, which is a north–south section through the centre of the deposit and Figure 14-7, which is a three-dimensional image, illustrate the final block classification and show the drillhole density. The drillhole spacing to a depth of about 1,400 m is 25 m and to a depth of 1,200 m is about 50 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: prepared by MMTS, 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: prepared by MMTS, 2020. Class 1: Measured Resource; Class 2: Indicated Resource, Class 3: Inferred Resource.

14.10 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 or 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.

14.10.1 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.

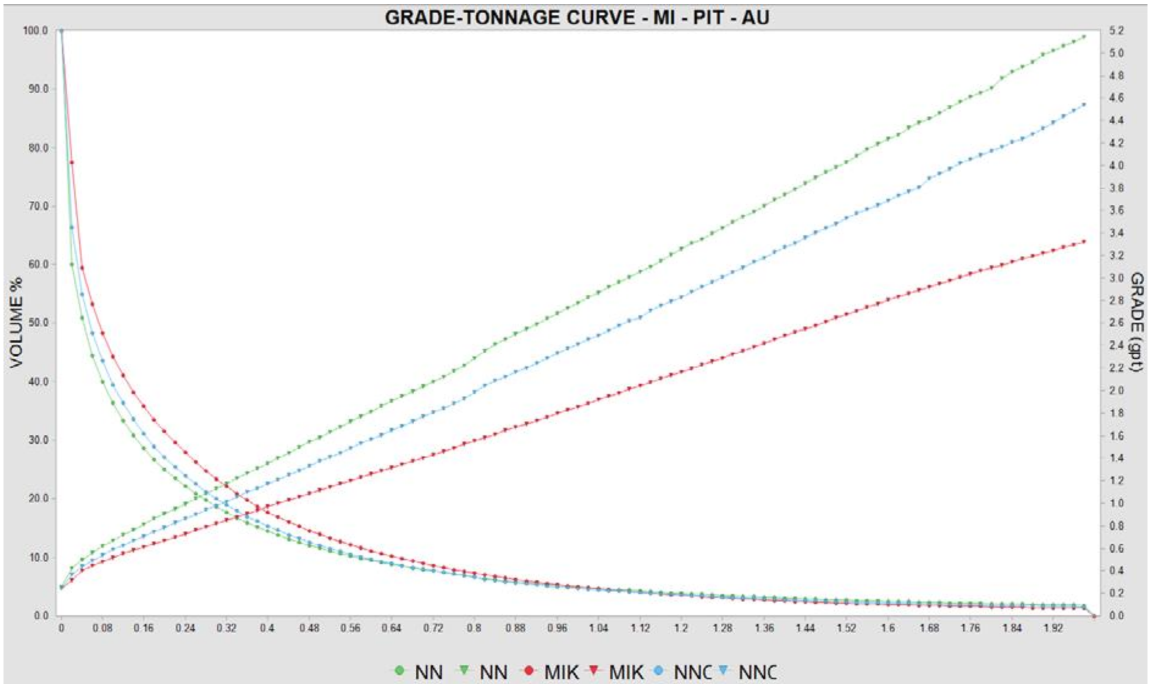
Table 14-12: Summary of Model Grade Comparison with De-Clustered Composites by Domain

Parameter	De-Cluster Au Composites (NN)				Au - MIK			
	1	2	3	ALL	1	2	3	ALL
Num Samples	202,692	23,8675	167,35	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	9	9	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
Num Samples	178,489	221,930	161,71	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.	0.	0.	0.	0.	0.	0.	0.
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.	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%

14.10.2 Grade-Tonnage Curves

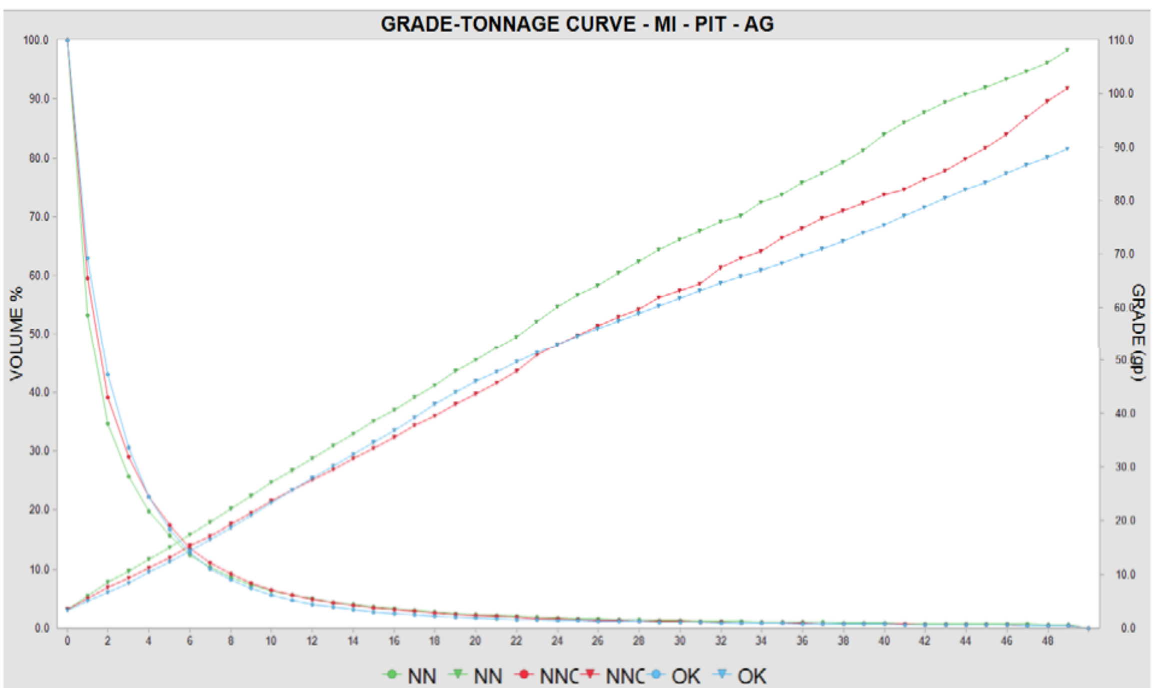
Grade–tonnage curves were created to compare the Au-MIK and Ag-OK interpolated grades with de-clustered composite grades. The de-clustered composites were corrected for the volume–variance effect by applying an indirect lognormal correction (ILC) to the NN grades. This correction applies a factor to 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-ILC grades.

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



Note: prepared by MMTS, 2020

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



Note: prepared by MMTS, 2020

14.10.3 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 14-13are sections for gold grade comparisons and Figure 14-14through 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 drillhole data. On all sections, the drillhole data shown is ±10 m of the section, also illustrating the close drillhole spacing throughout the deposit.

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

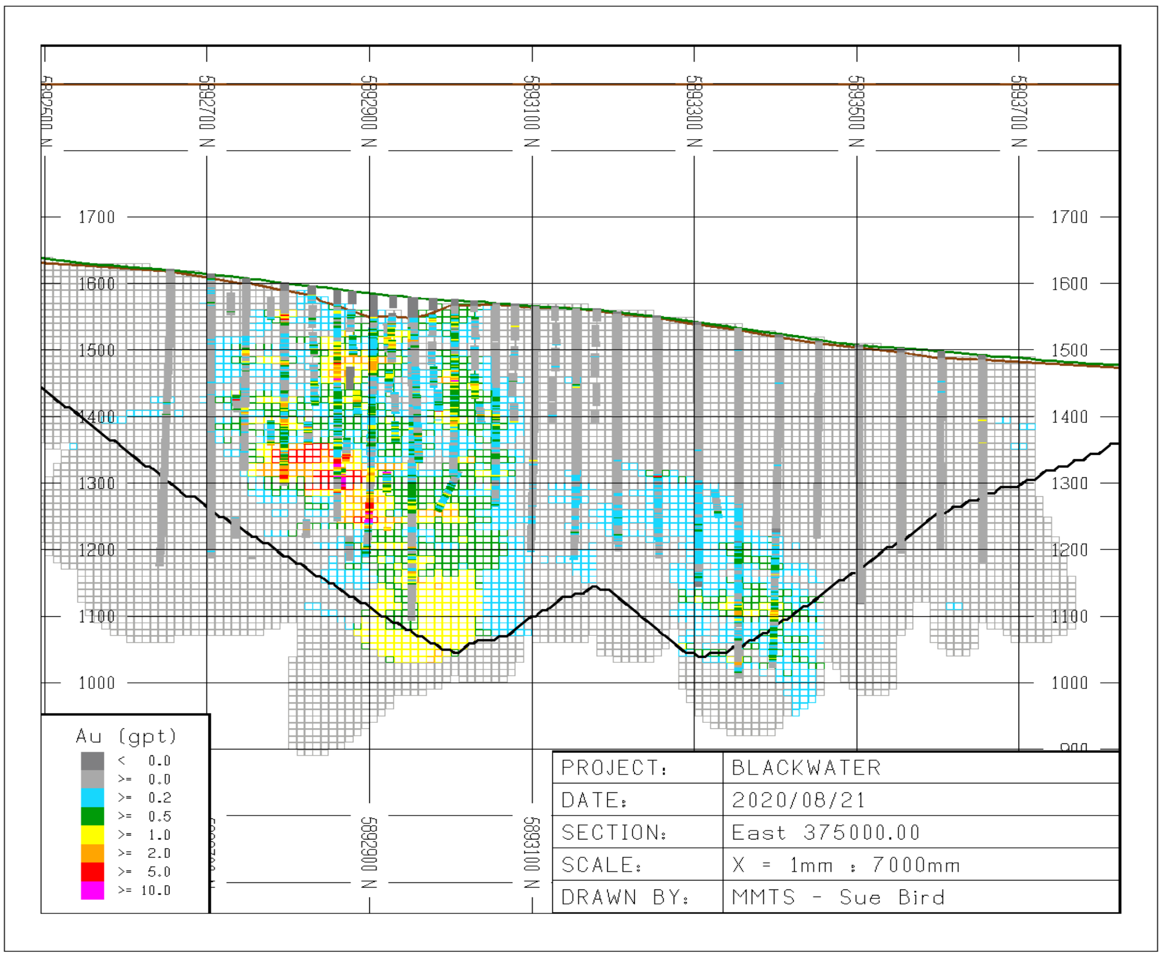


Figure 14-11: Au Grade – Model Compared to Composite – 375300E

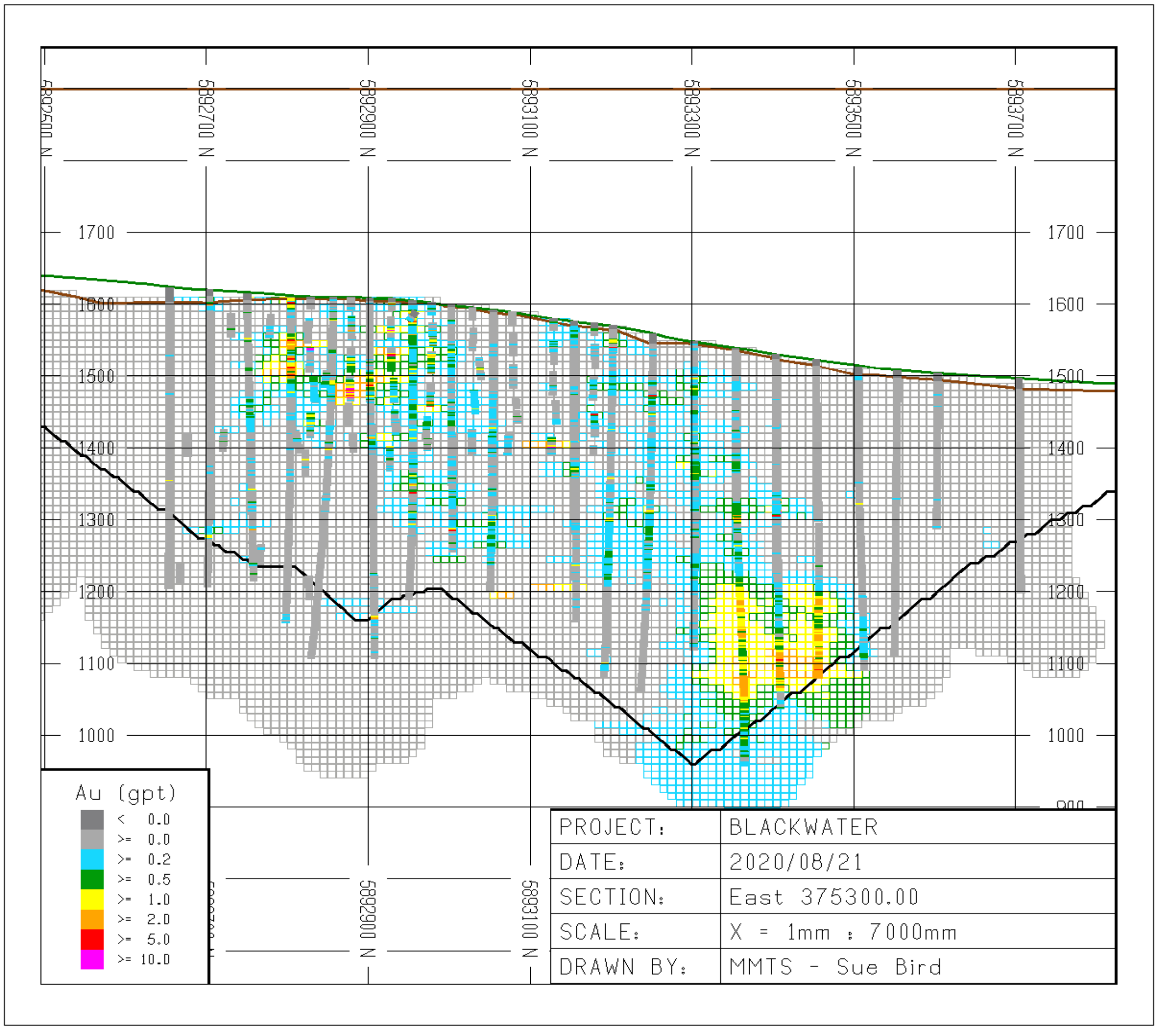


Figure 14-12: Au Grade – Model Compared to Composite – 375500E

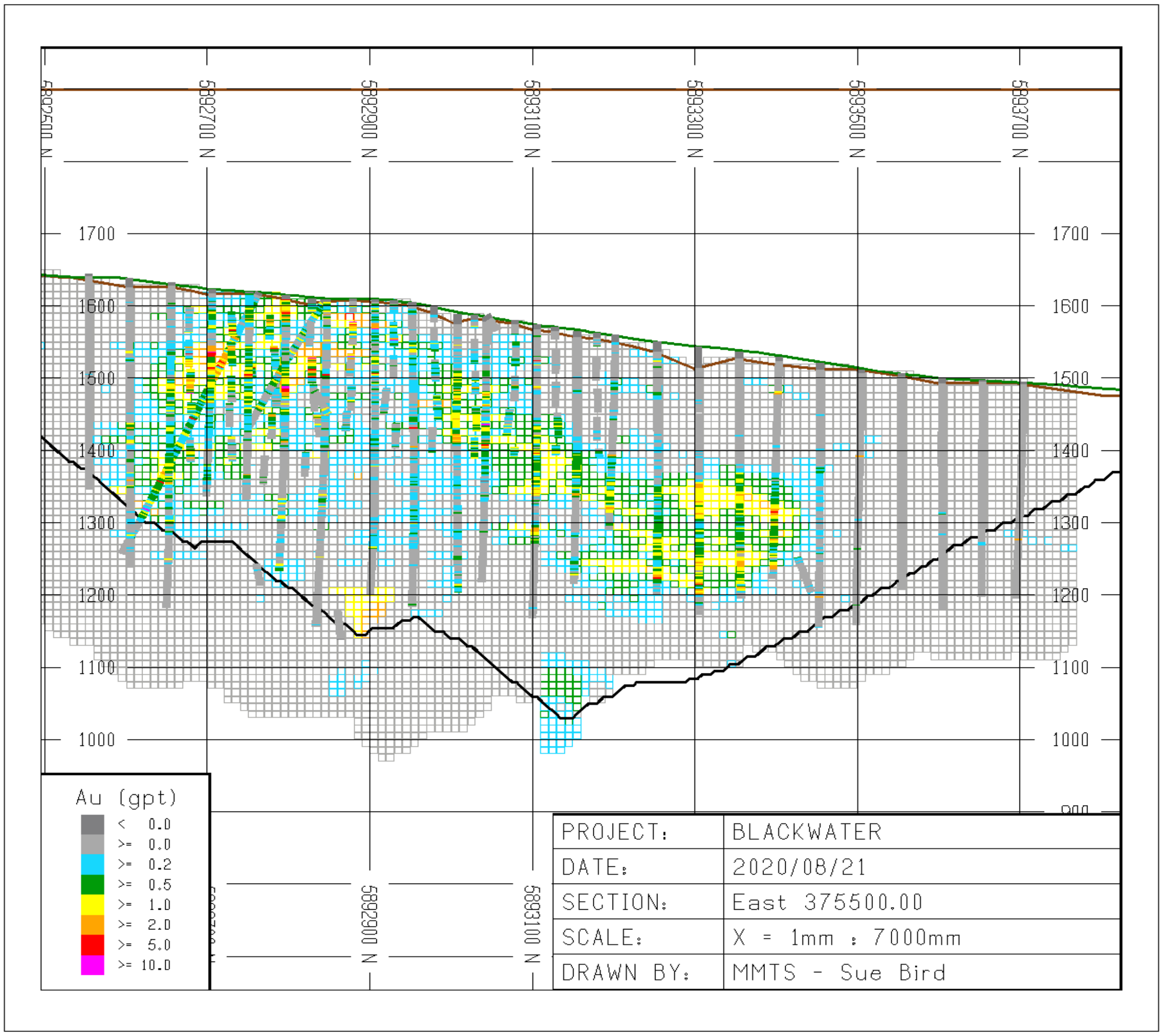


Figure 14-13: Au Grade – Model Compared to Composite – 375800E

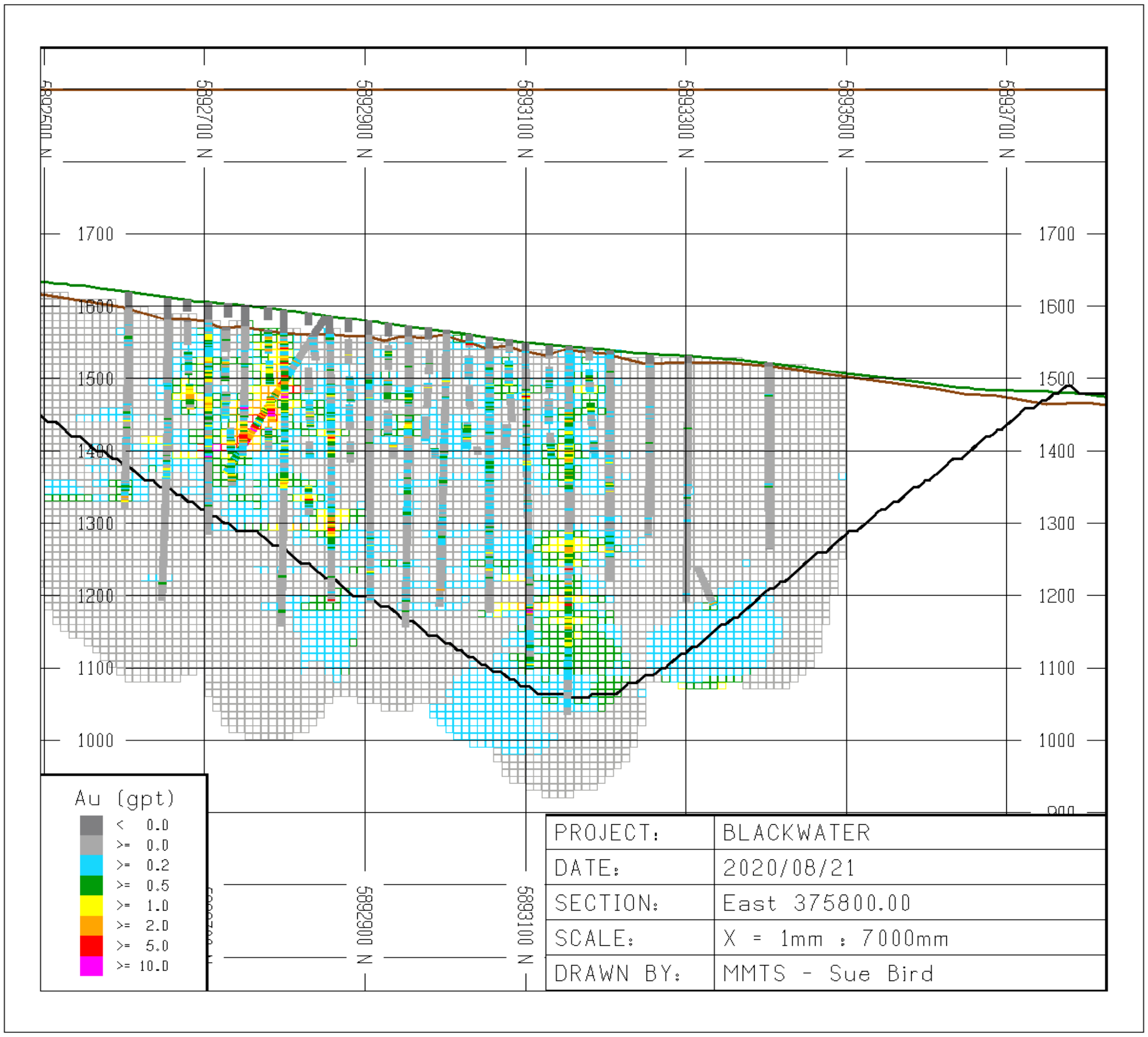


Figure 14-14: Ag Grade – Model Compared to Composite – 375000E

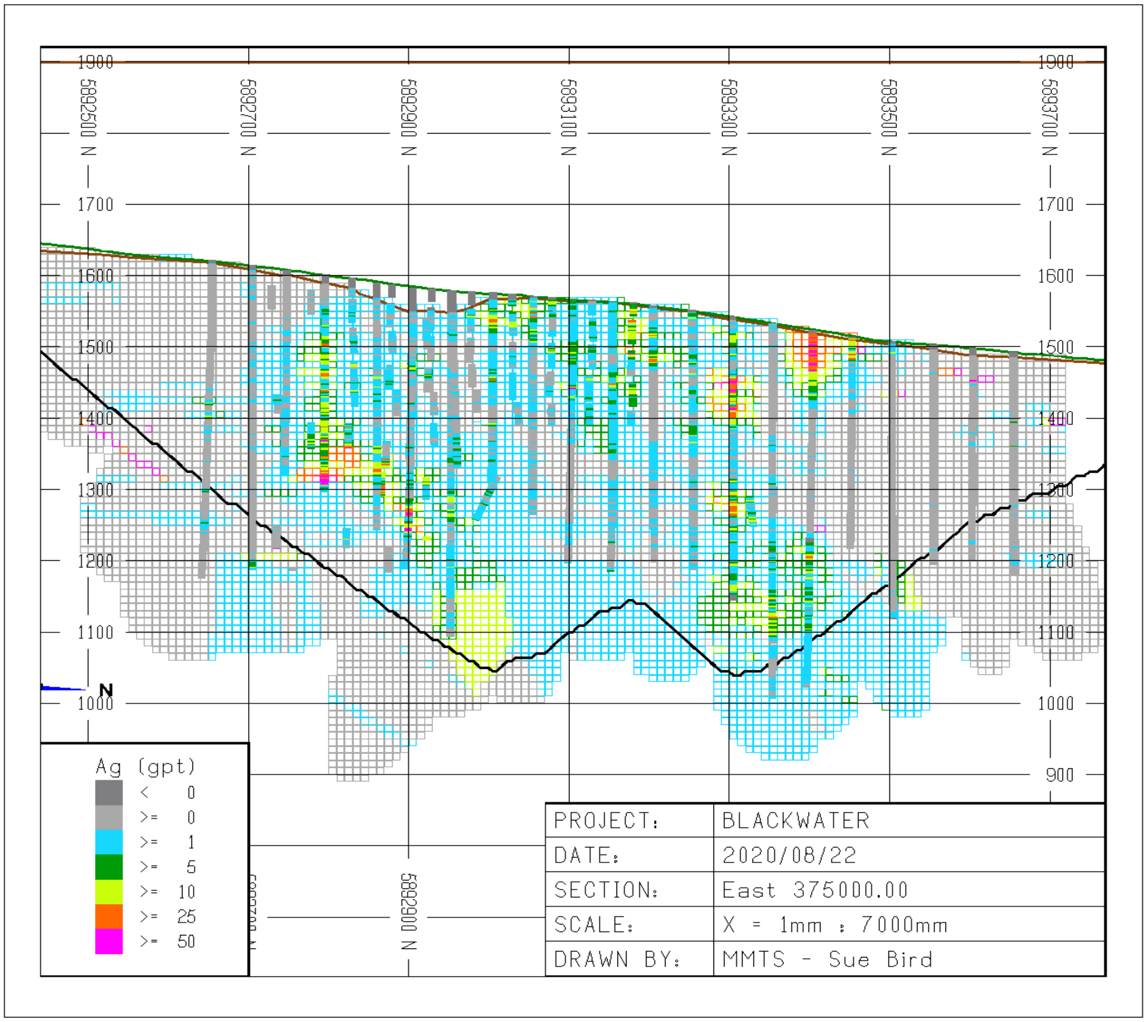


Figure 14-15: Ag Grade – Model Compared to Composite – 375300E

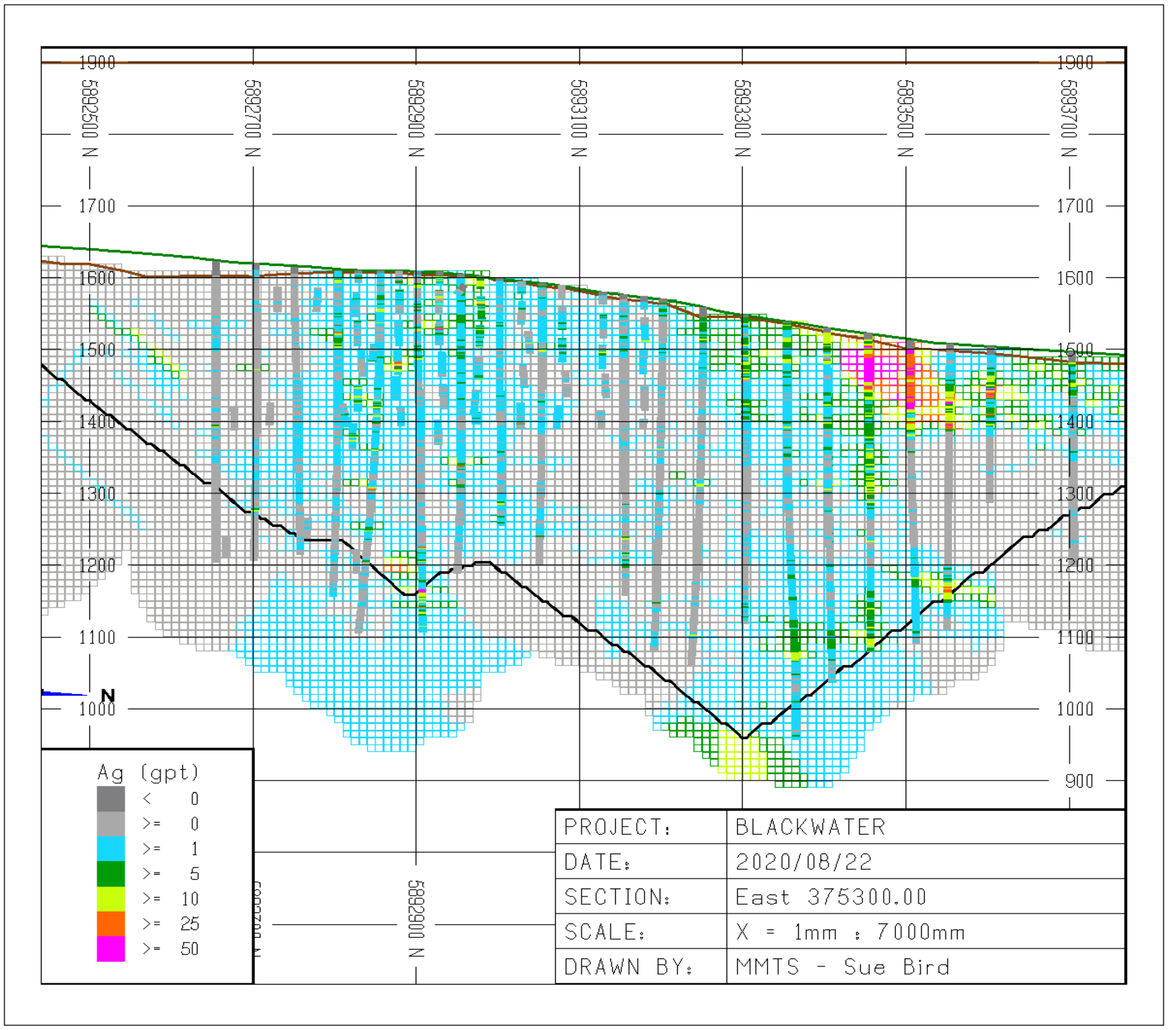


Figure 14-16: Ag Grade – Model Compared to Composite – 375500E

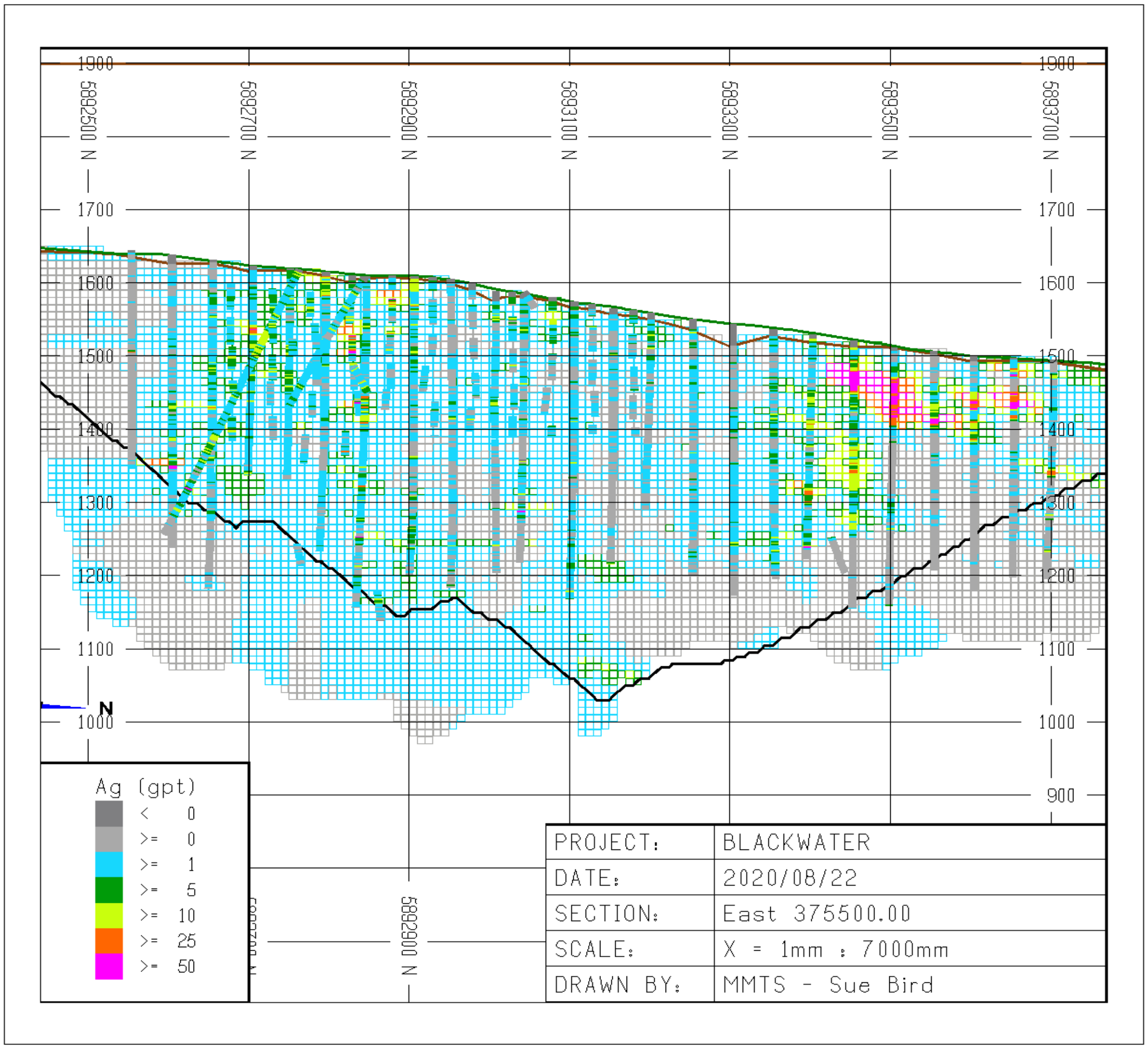
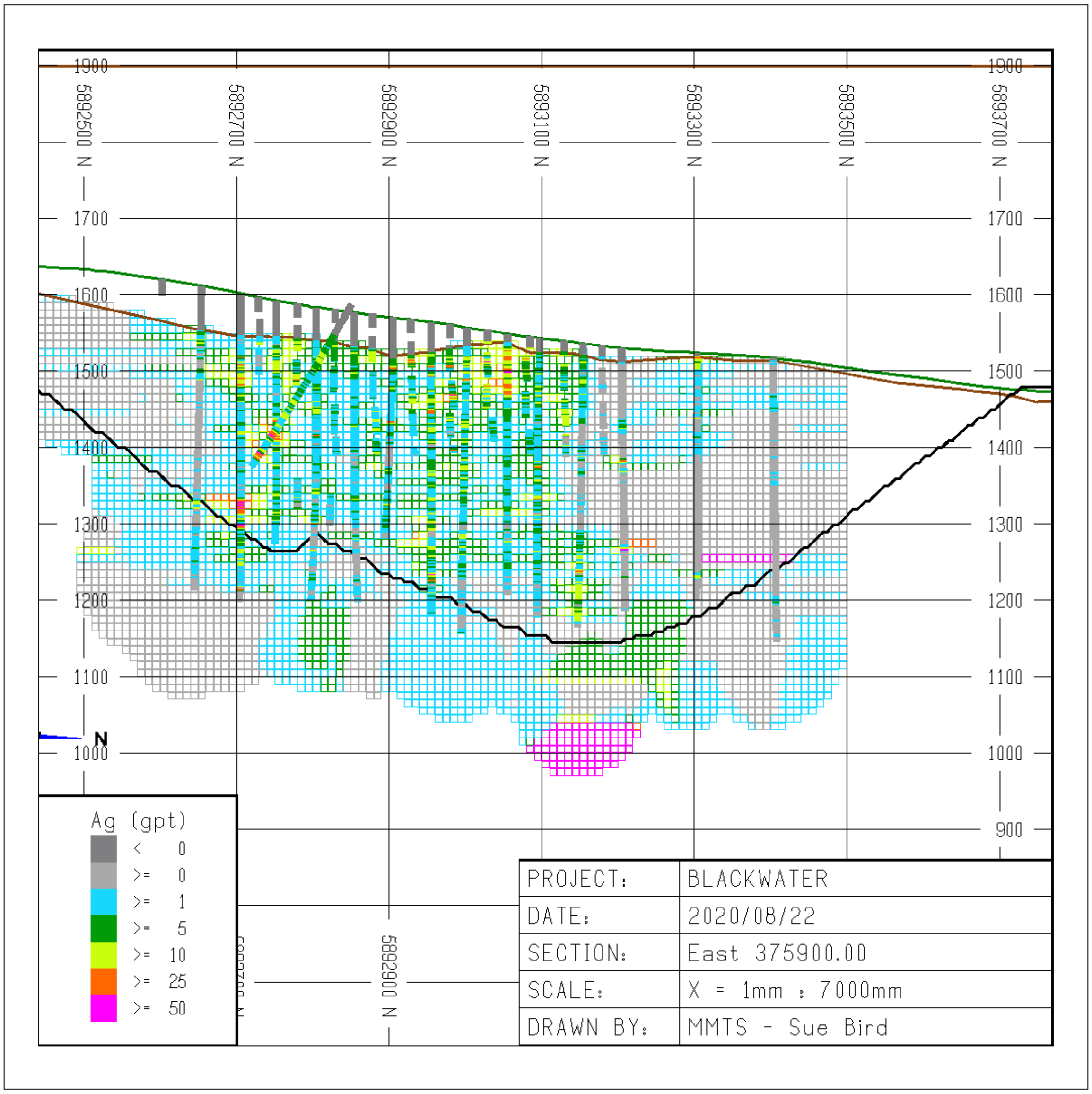


Figure 14-17: Ag Grade – Model Compared to Composite – 375900E



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

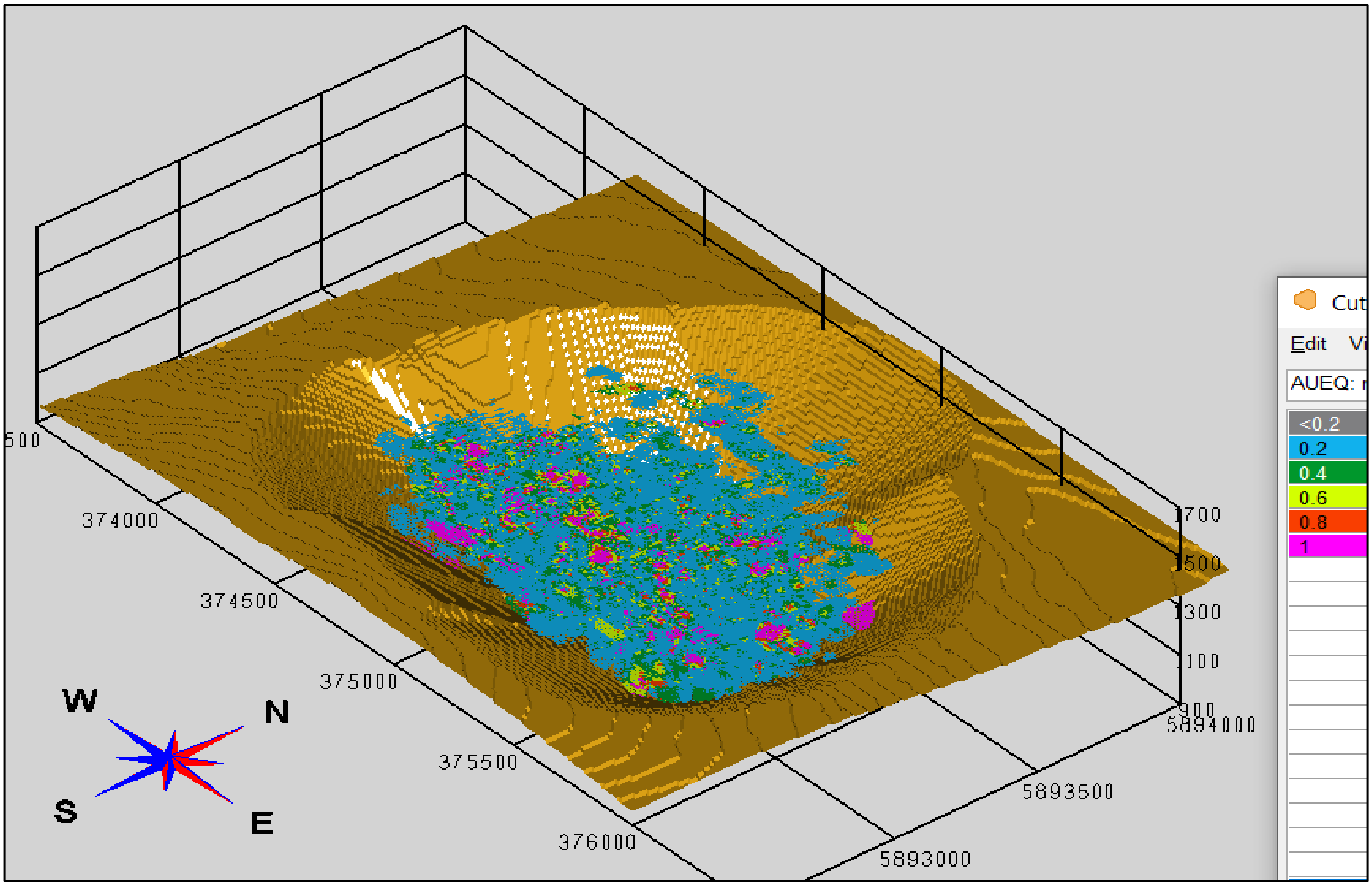
The resulting gold equivalent (AuEq) equation used is:

$$AuEq = \mathrm{Au} + 0.006 * AgGrade$$

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



Source: MMTS, 2020

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

Table 14-14: Costs used for Lerchs-Grossmann Resource Pit

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

14.12 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.

14.13 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 1200m elevation
2	Gold price assumption	Conservative for cut-off grade, reasonable for pit size
3	Capping	CPP, 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 48-72m length for total of 33,216m illustrates conservatism in the modelled Au tonnage and grades. The potential BH 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 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 2021 FS open pit life-of-mine (LOM) plan (LOMP), described in Section 16.

15.2 Mineral Reserves Statement

Proven and Probable Mineral Reserves are modified from Measured and Indicated Mineral Resources and are summarized in Table 15-1. Inferred Mineral Resources are set to waste. Mineral Reserves are estimated using the CIM 2019 Best Practices Guidelines and are classified using the 2014 CIM Definition Standards.

Table 15-1: Mineral Reserve Estimate

Reserve Class	Run of Mine (Mt)	Gold Grade (Au, g/t)	Contained Metal (Au, Moz.)	Silver Grade (Ag, g/t)	Contained Metal (Ag, Moz.)	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 Reserve	334.3	0.75	8.0	5.8	62.2	0.78

Notes:
1. The Mineral Reserve estimates were prepared by Marc Schulte, P.Eng. (who is also the independent Qualified Person for these Mineral Reserve estimates) and have an effective date of September 10, 2021.
2. Mineral Reserves are based on the 2021 Feasibility Study life-of-mine plan.
3. Mineral Reserves are mined tonnes and grade, the reference point is the mill feed at the primary crusher and includes consideration for operational modifying factors such as loss and dilution.
4. Mineral Reserves are reported at an 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.
5. 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.
6. The AuEq values were calculated using the same parameters as NSR listed above, resulting in the following equation: AuEq = Au g/t + (Ag g/t x 0.006).
7. Numbers have been rounded as required by reporting guidelines.

15.3 Mineral Reserves within Pit Phases

Open pits are based on the results of Pseudoflow sensitivity analysis, and then designed into detailed pit phases to develop pit reserves for production scheduling. 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**

Notes:

1. An NSR cut-off of C$13.00/t is applied.
2. Mined tonnes and grade include operational modifying factors.
3. 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.

15.4 Factors that May Affect the Mineral Reserve Estimate

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 rate
- Interpretations of mineralization geometry and continuity of mineralization zones
- Geotechnical and hydrogeological assumptions
- Ability of the mining operation to meet the annual production rate
- Operating cost assumptions
- 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.

15.5 Comments on Section 15

The current 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 have been estimated using industry best practices.

16 MINING METHODS

The Mineral Reserves stated in Item 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 FS level of engineering.

16.1 Key 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.

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

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

Table 16-1: Net Smelter Price Inputs

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

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

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

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

  NSP Silver (C$/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 can be examined for their contribution to positive Project cash flow.

The cut-off grade is based on the calculated NSR. Low grade resource blocks, internal in the pit design, must be mined anyway to expose higher grade blocks below them; therefore, they can still contribute to a positive cash flow if they have an insitu 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 grade of C$13.00/t is chosen. This cut-off grade will cover the incremental production costs of processing, administration, and stockpile rehandle.

16.1.2 Mining Loss and Dilution

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

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

Edge dilution introduced from ore to waste contacts within the whole block model is 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.

16.1.3 Pit Slopes

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

Geotechnical domains are based on geotechnical conditions described by KP (KP, 2013b). Geotechnical conditions are 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:

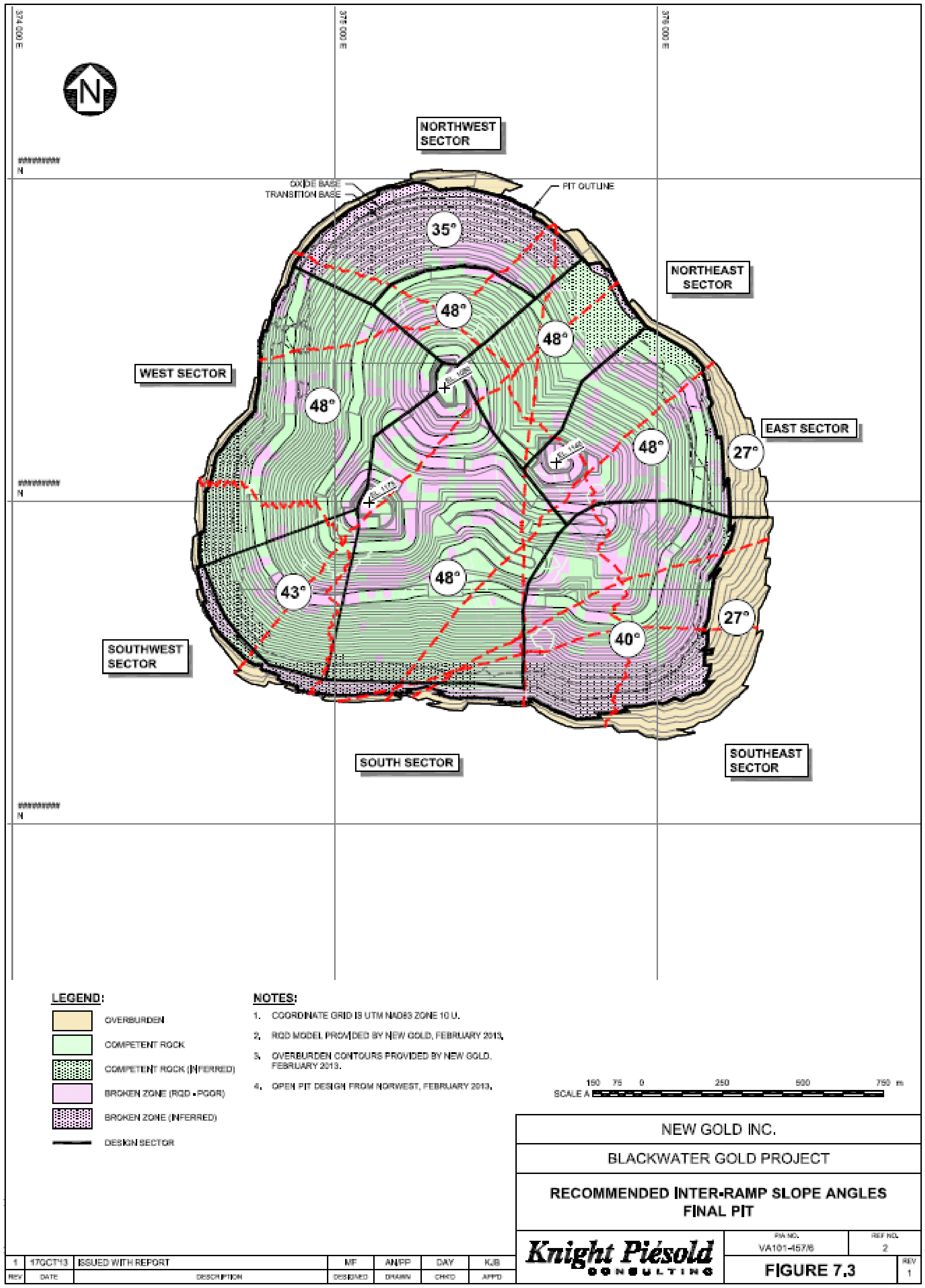
- Surficial material – Glacial till deposits typically ranging from 5 to 20 m in thickness throughout the deposit area, which increases up to 110 m depth along the eastern side of the deposit;

- Broken zone – Delineated within the rock mass RQD block model when the values are generally less than 40%;
- Competent rock – Defined as all rock mass regions where the RQD is greater than 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 configuration for the domains are shown in Table 16-2.

Figure 16-1: Open Pit Geotechnical Domains



Source: KP, 2013c

Table 16-2: Pit Slope and Configuration Inputs

Domain	Bench Height (m)	BFA (degrees)	IRA (degrees)	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

*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, is maintained by including geotechnical berms.

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

The implementation of slope design also requires effective slope depressurization, good-controlled blasting and excavation practices, and regular inspection and systematic slope monitoring. Plans for depressurization are further described in Section 16.8.1.

An updated geotechnical interpretation has been completed by KP (KP, 2021o) that includes refinement of the pit design sectors and recommended inter-ramp slope angles. Timing of this updated interpretation did not allow for effective incorporation into the FS 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.

16.2 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 are 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 are 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 are 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 are evaluated on a relative basis to provide payback on capital and produce a return for the Project. At some point, further expansion does not provide significant added value. A pit limit can then be chosen that has suitable economic return for the deposit.

For each pit shell, an undiscounted cash flow (UCF) is generated based on the shell contents and the economic parameters listed in Table 16-1and Table 16-3. The UCFs for each case are compared to reinforce the selected point at which increased pit expansions do not materially increase the project value. Note that the economics are only applied for comparative purposes to assist in the selection of an optimum pit shell for further mine planning; they do not reflect the actual financial results of the mine plan.

The chosen pit shell is then used as the basis for more detailed design and economic modelling.

Price assumptions for the Pseudoflow runs were provided in Table 16-1 and operating cost assumptions are included as Table 16-3.

Table 16-3: Inputs into Pseudoflow Pit Optimization

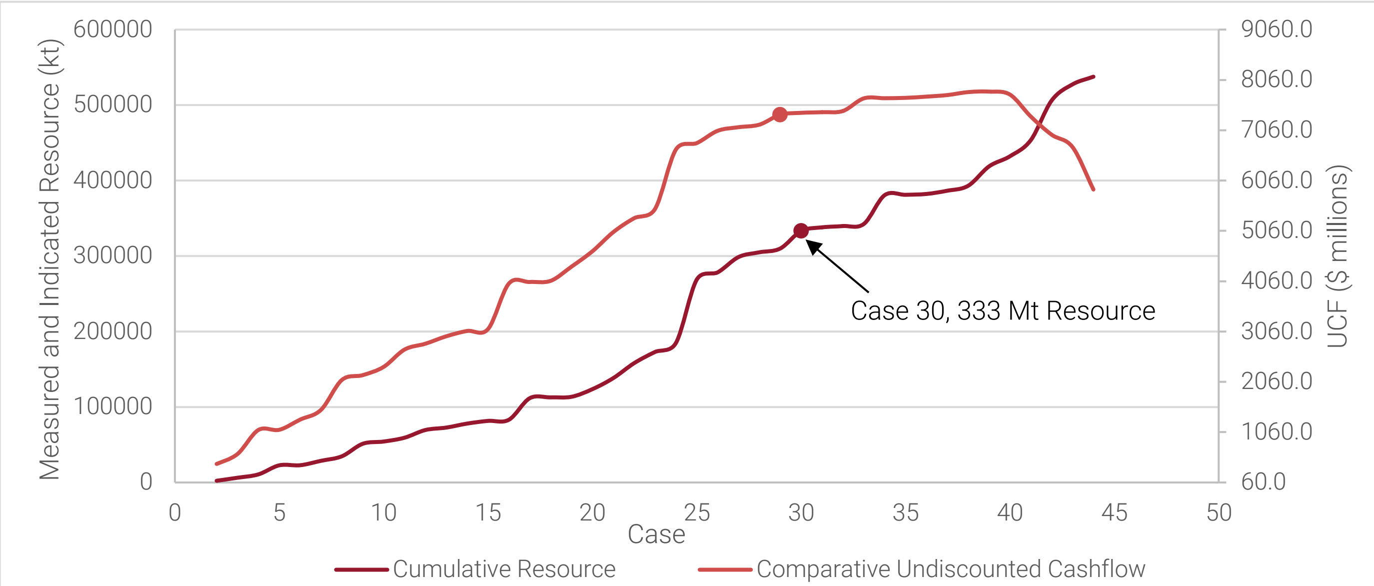
Item	Unit
Pit rim ore mining cost	C$2.50/t, pit rim of 1500 m
Pit rim waste mining cost	C$2.00/t, pit rim of 1500 m
Incremental bench haulage cost	C$0.025 per every 10 m bench below pit rim
Processing cost	C$10.00/t
G&A cost	C$3.00/t
Gold process recovery	93%
Silver process recovery	55%

16.2.1 Ultimate Pit Limits

Figure 16-2 shows the contents of the generated Pseudoflow pit shells. Several inflection points can be seen in the curve of cumulative resources and UCF by pit case. Scoping-level mine plans were produced for pit shells represented at various inflection points and discounted cashflows based on these mine plans indicated Case 30 to be a point at which larger pit shells will not produce significant increases to project value.

The pit shell generated from Case 30 is selected as the ultimate pit limits and is used for further mine planning as a target for detailed open pit designs with berms and ramps.

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

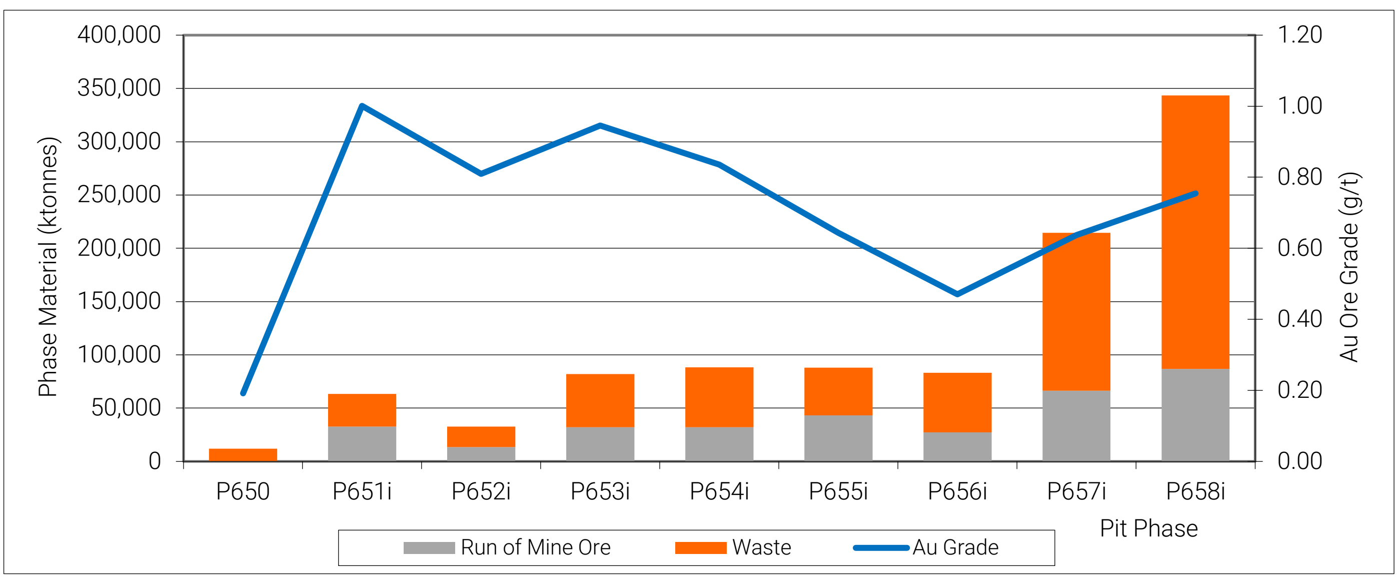


Note: prepared by MMTS, 2021

16.3 Pit Designs

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

Figure 16-3: Designed Phase Pit Contents



Note: prepared by MMTS, 2021

16.3.1 In-Pit Haul Roads

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

16.3.2 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 a vast majority 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 16.2.1 and providing mill feed over the initial years of the Project. Phases then proceed from the highest economic margin to lowest, targeting pit shells represented by inflection points on the curve shown in Figure 16-2.

16.3.3 Pit Design

The phased Blackwater pit designs are discussed in this section and shown in Figure 16-4 to Figure 16-12. Sections through the deposit showing the whole block gold and silver grades are illustrated in Figure 16-13 to Figure 16-16. Where meter (m) elevations are listed below, they refer to the mine grid z-dimension elevation, which is meters above sea level (masl).

- Construction Borrow Pit, P650 – This phase targets near surface NAG waste rock and glacial till overburden for Project 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.

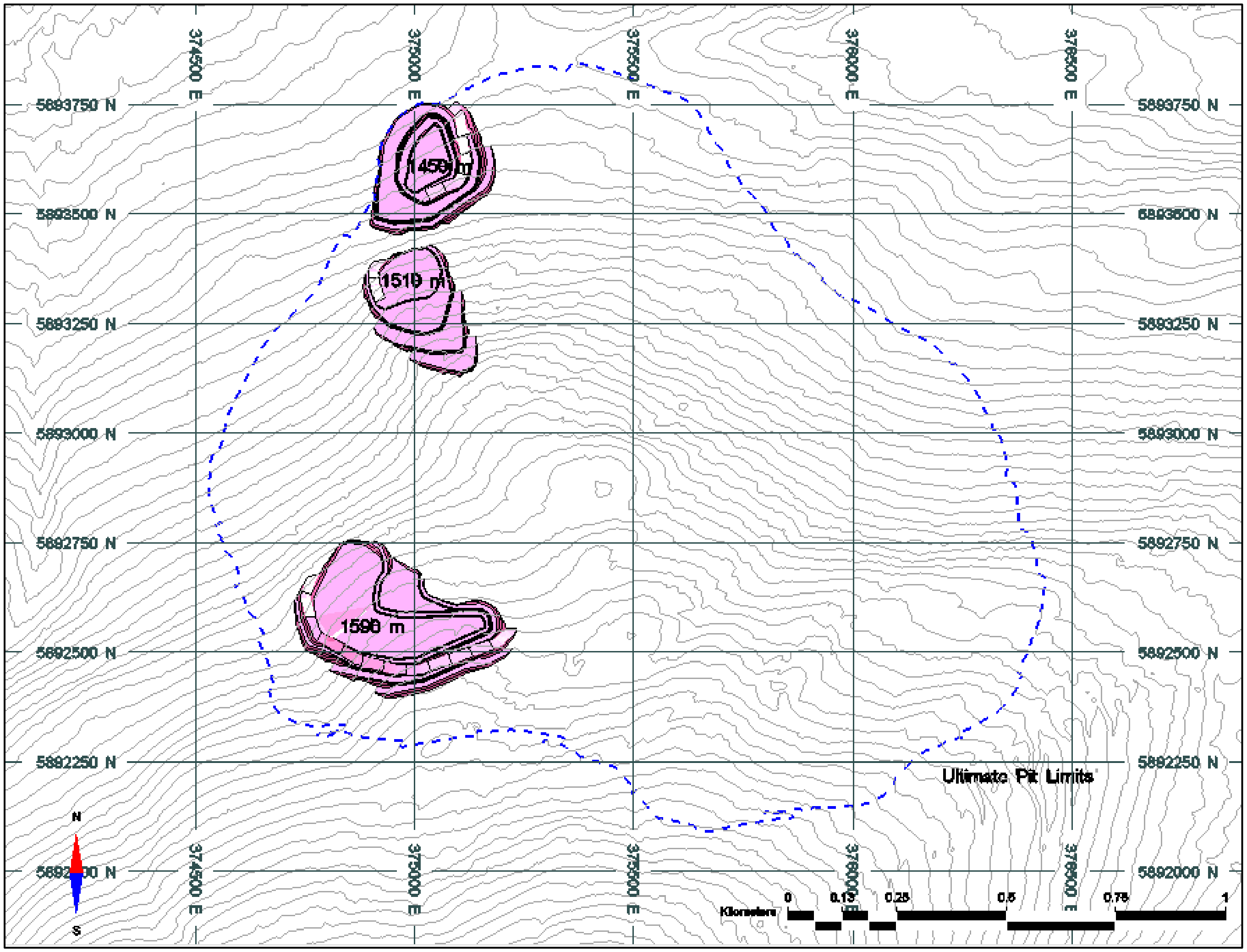
- North Pushback 1, P655 – This phase pushes out the pit to the north. The phase mines from the pit crest at the 1,570 m elevation, down to the pit bottoms on the 1,270 m. The main ramp runs clockwise from the pit exit at the 1,520 m elevation in the west of the pit, bridging across the phase on the 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 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.

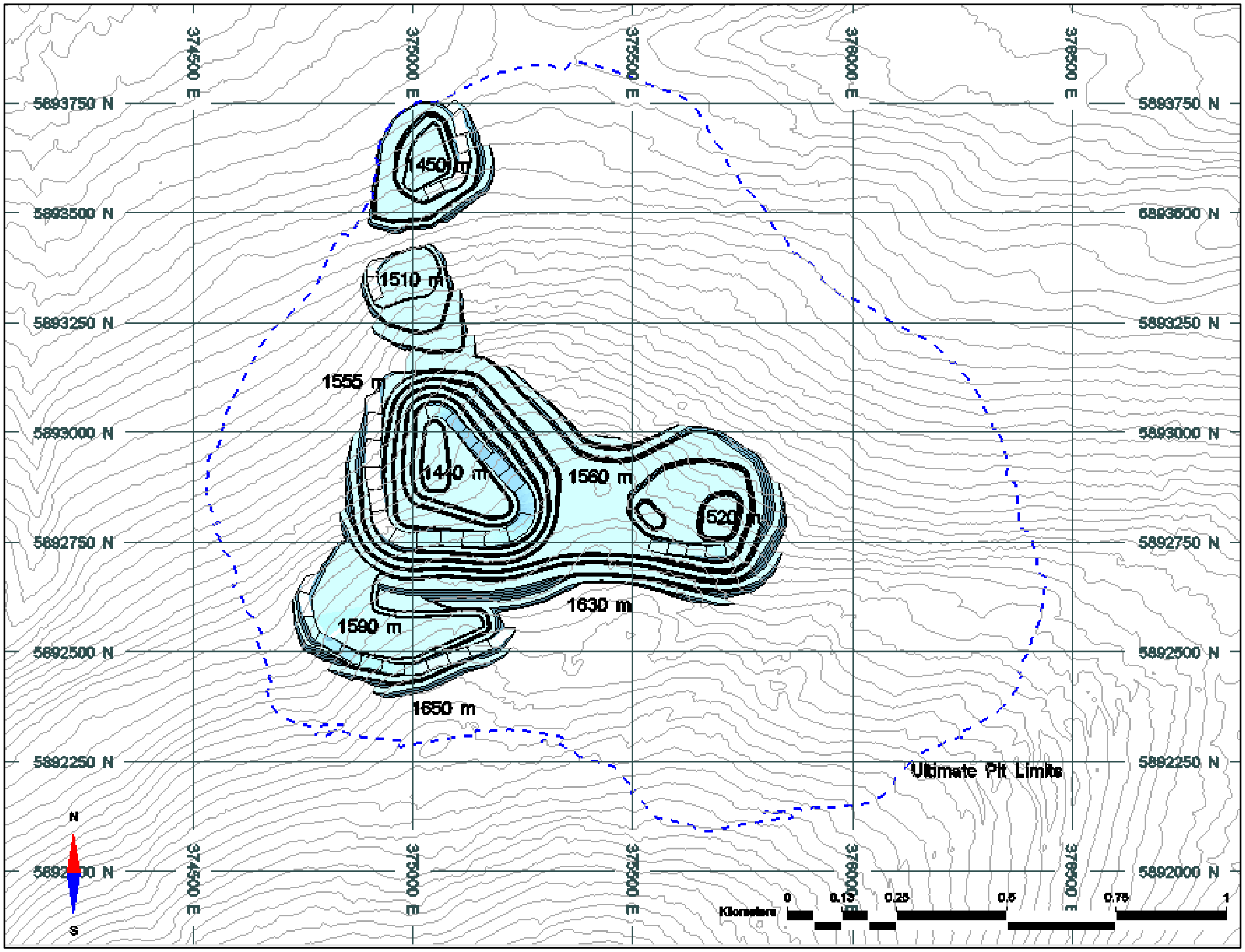
- Southwest Pushback, P658 – This phase pushes out the pit to the ultimate pit limits in the south and west. The phase mines from the pit crest at the 1,690 m elevation, down to the two pit bottoms on the 1,170 m and 1,160 m elevations, bridging between the two bottoms on the 1,240 m bench. The main 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.

Figure 16-4: Construction Borrow Pit, P650



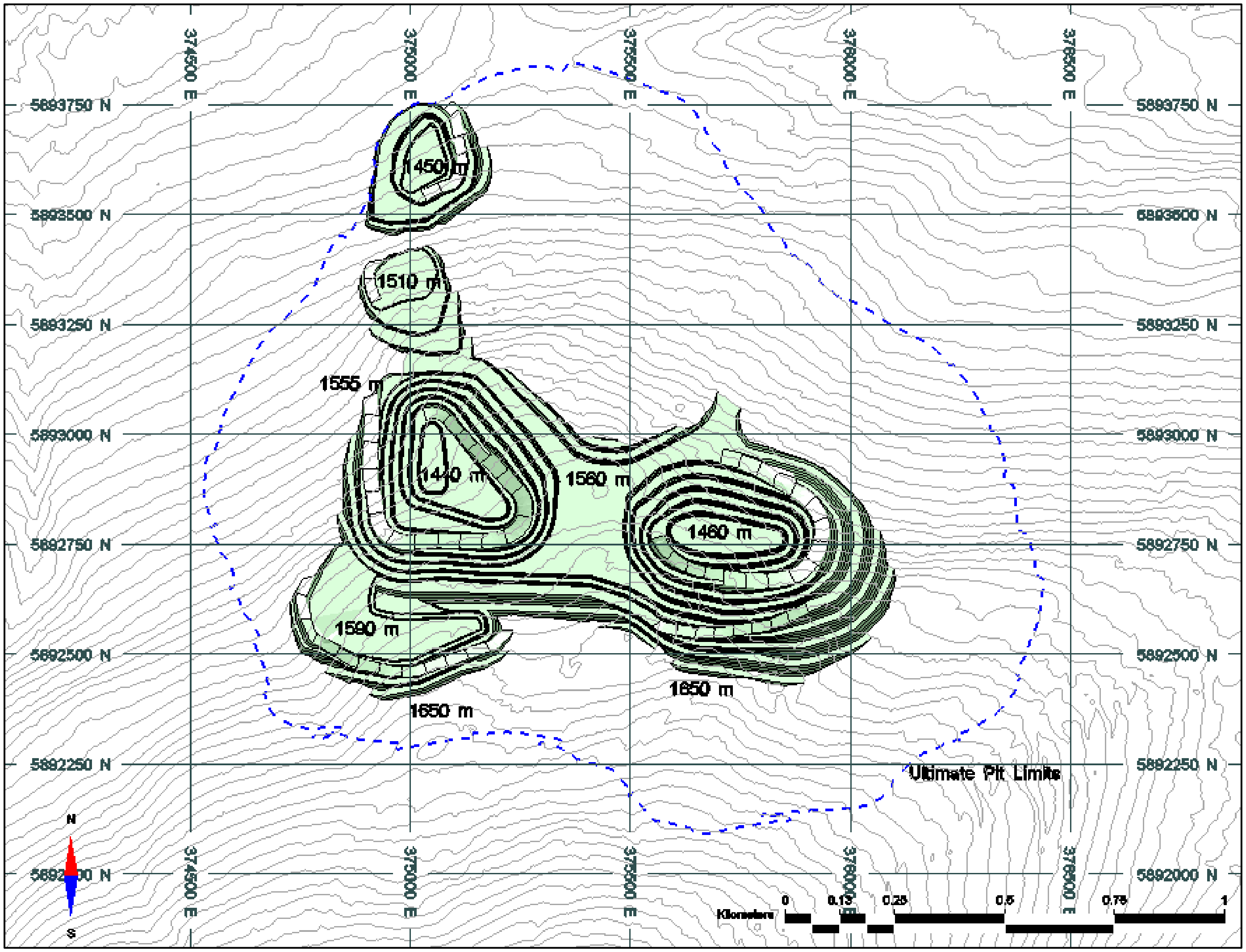
Note: prepared by MMTS, 2021

Figure 16-5: Starter Pit, P651



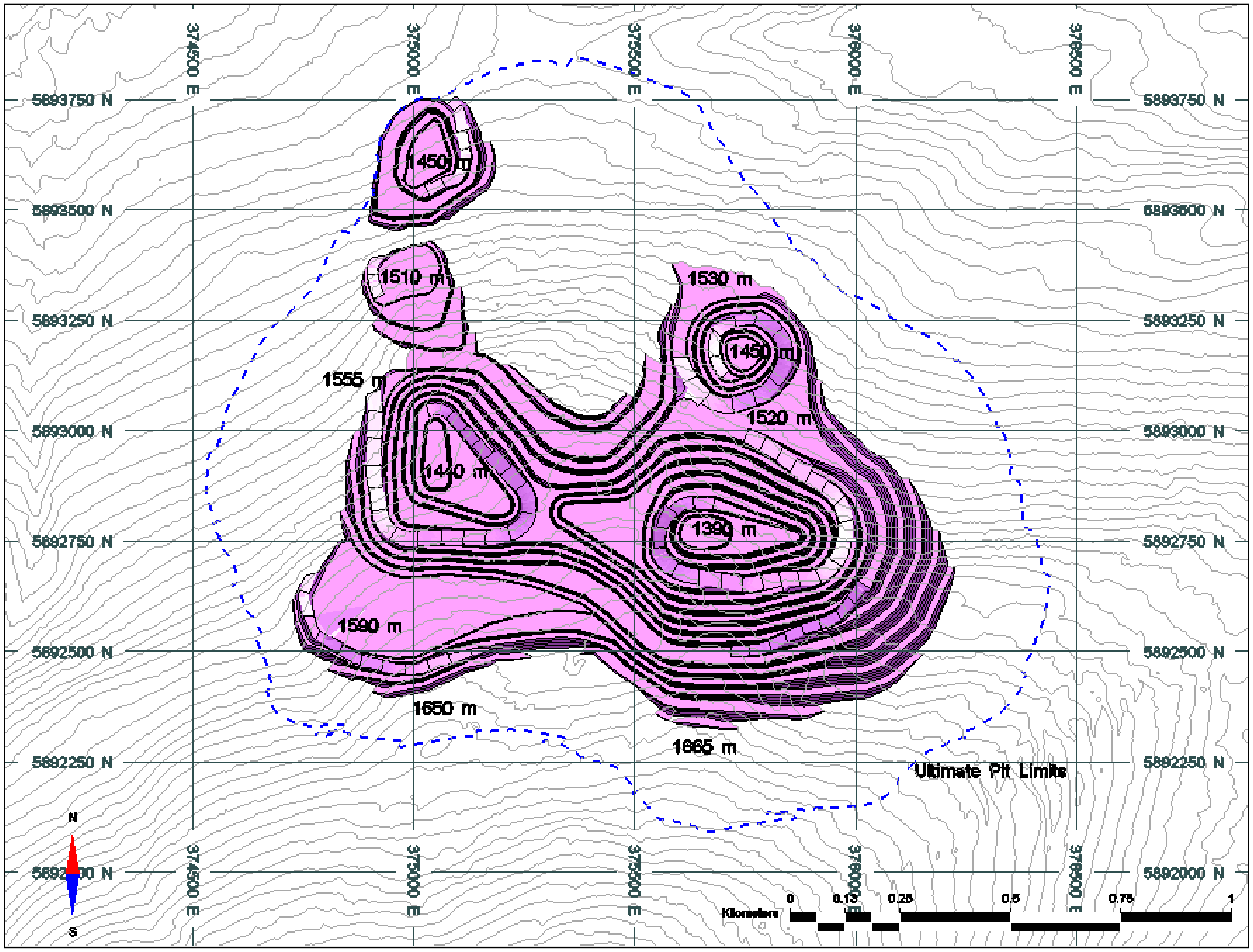
Note: prepared by MMTS, 2021

Figure 16-6: East Pushback 1, P652



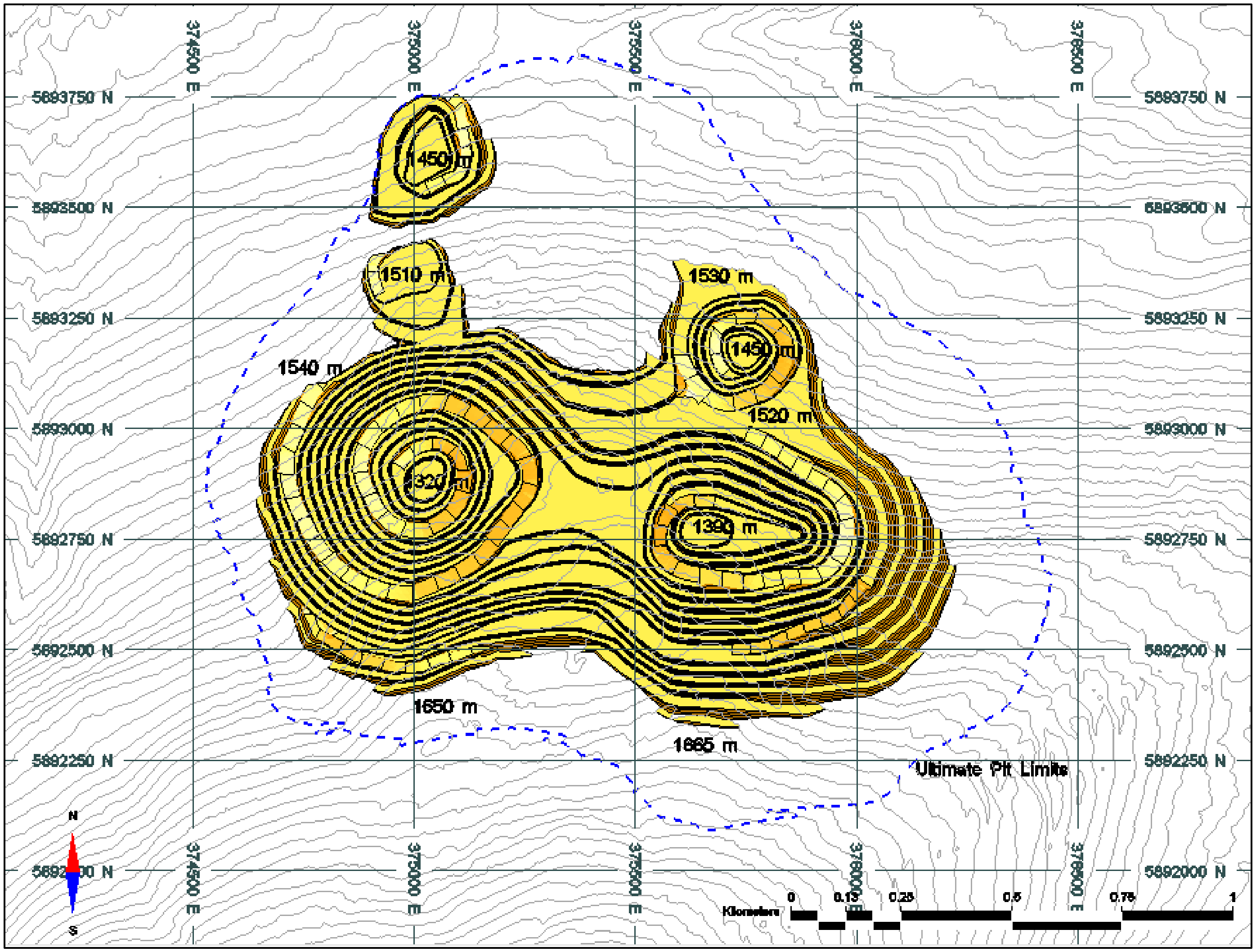
Note: prepared by MMTS, 2021

Figure 16-7: East Pushback 2, P653



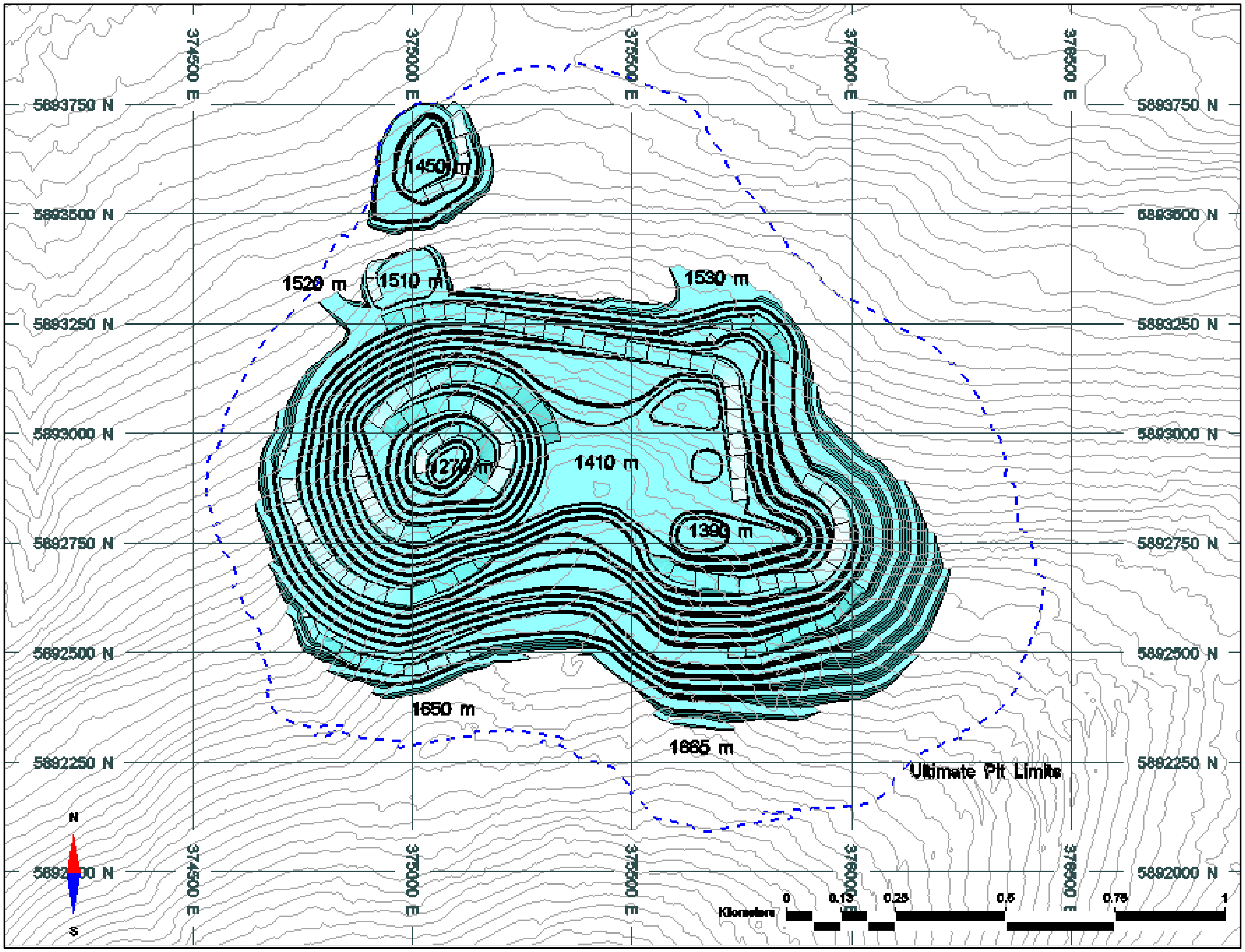
Note: prepared by MMTS, 2021

Figure 16-8: West Pushback, P654



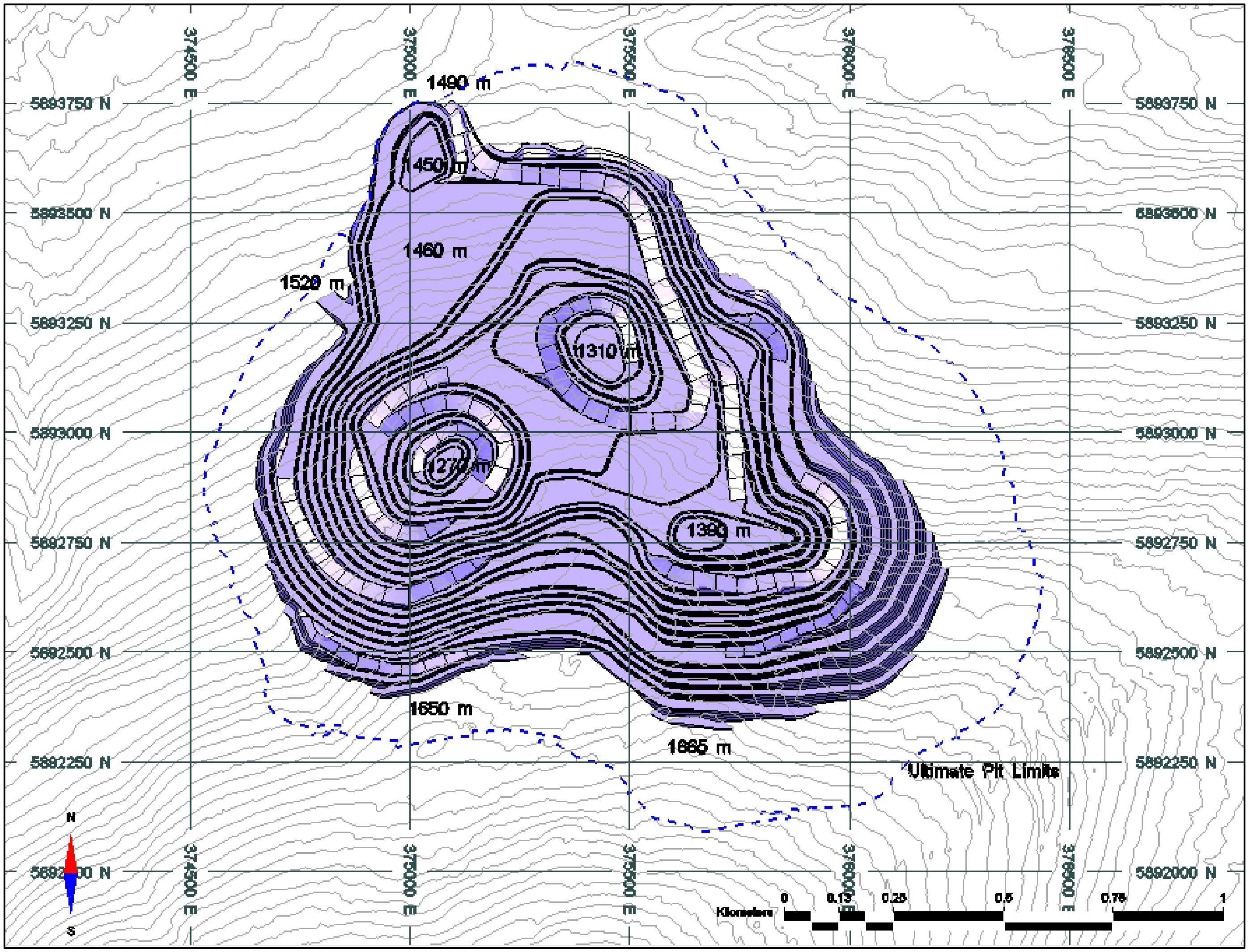
Note: prepared by MMTS, 2021

Figure 16-9: North Pushback, P655



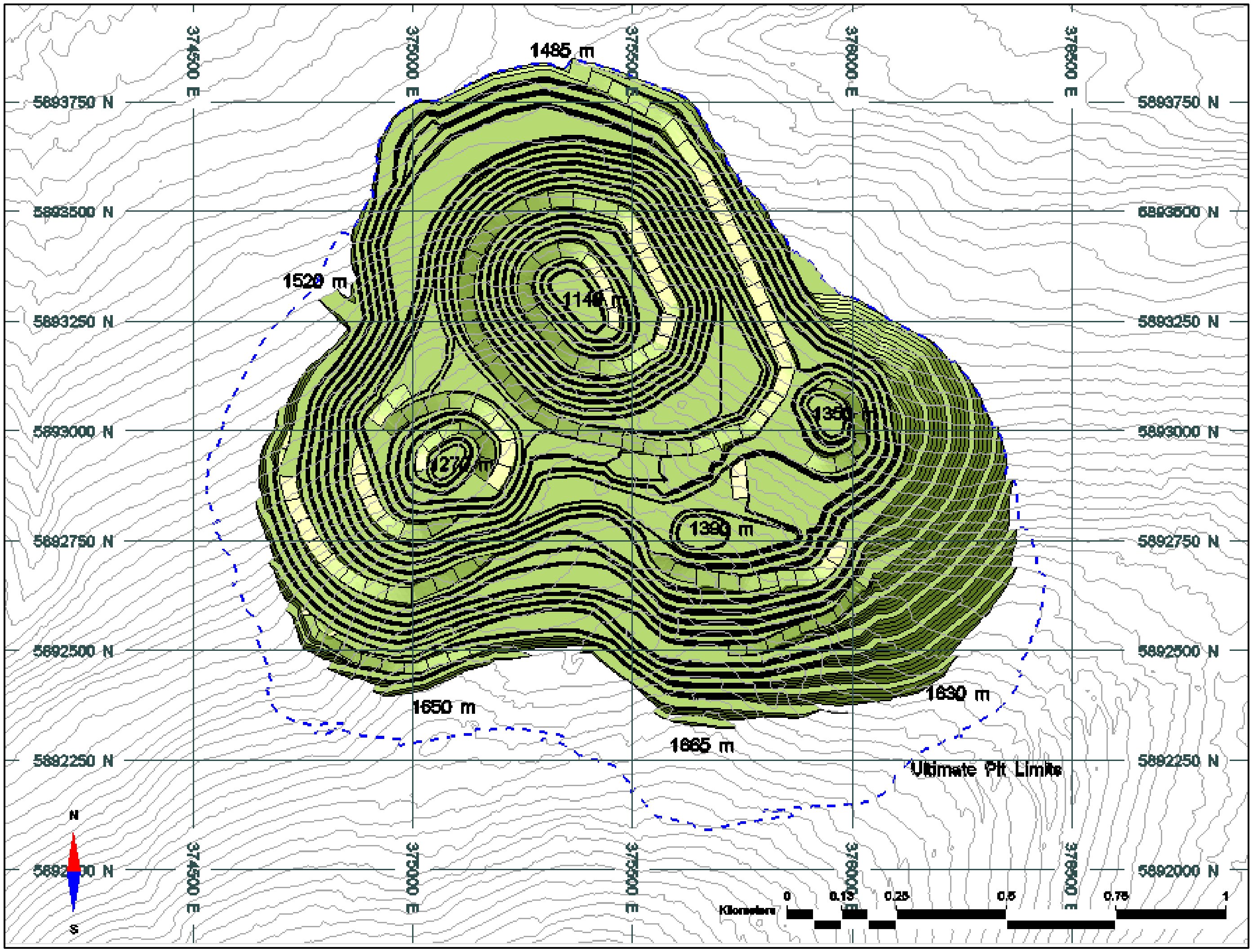
Note: prepared by MMTS, 2021

Figure 16-10: North Pushback 2, P656



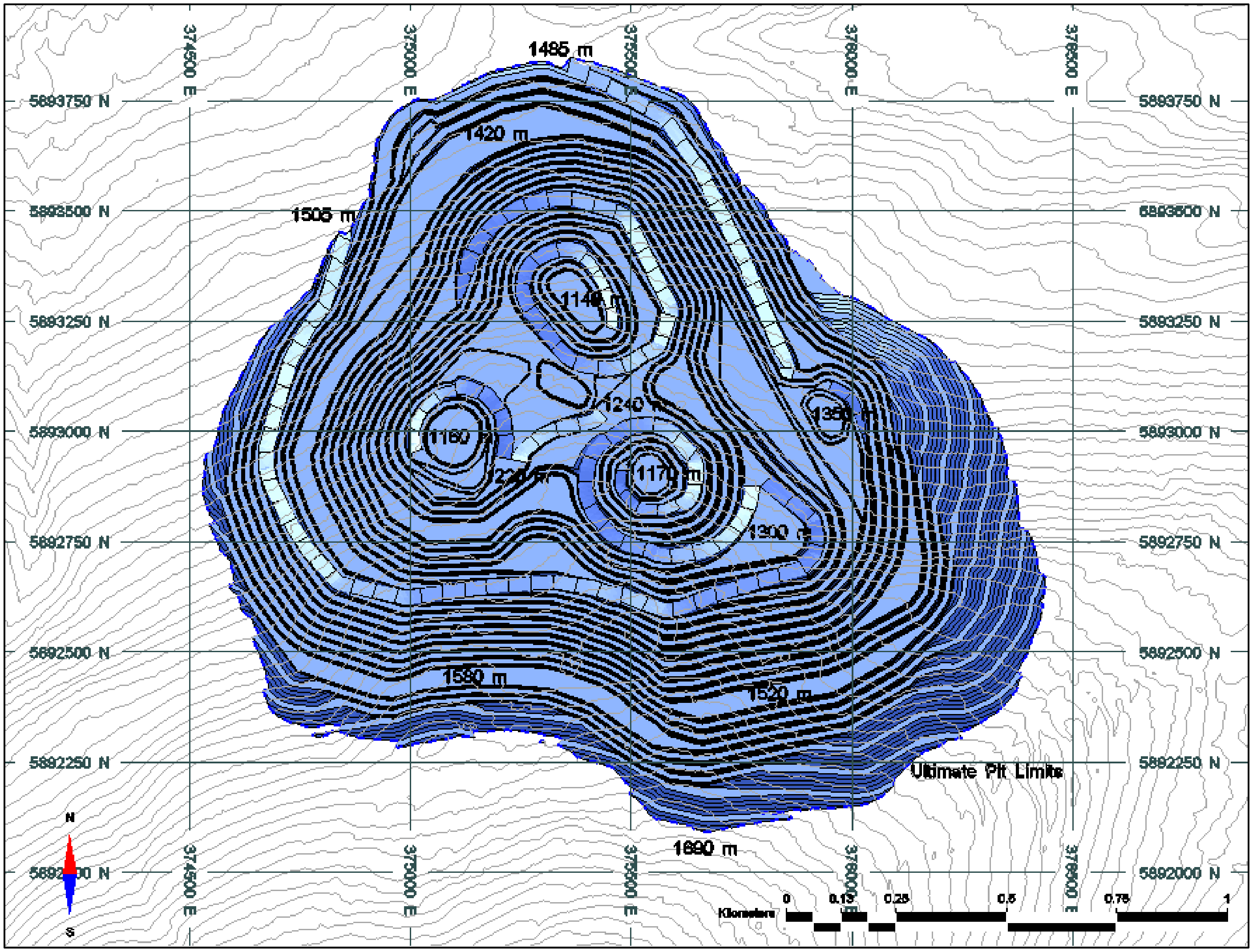
Note: prepared by MMTS, 2021

Figure 16-11: South Pushback, P657



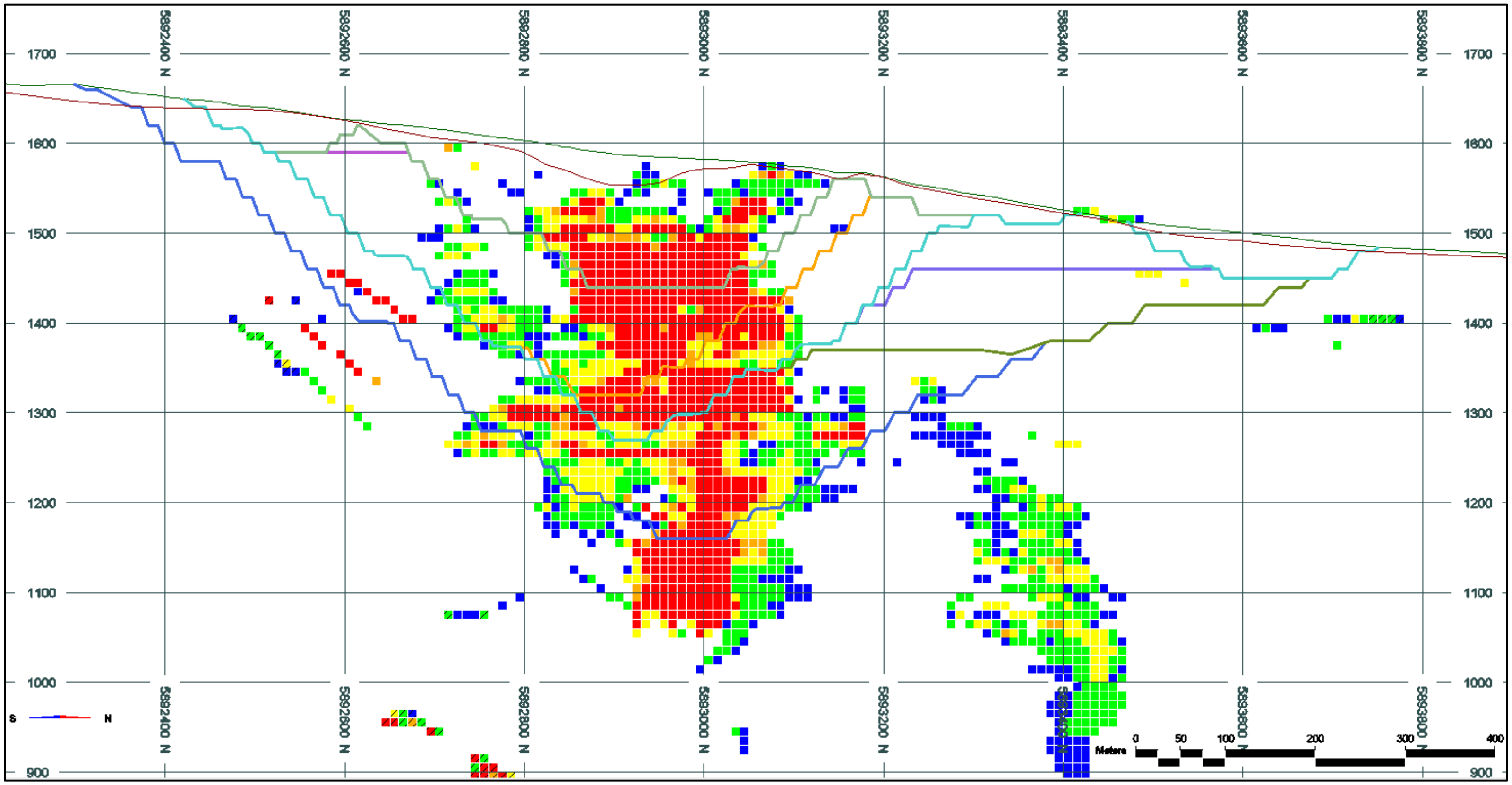
Note: prepared by MMTS, 2021

Figure 16-12: Southwest Pushback, P658



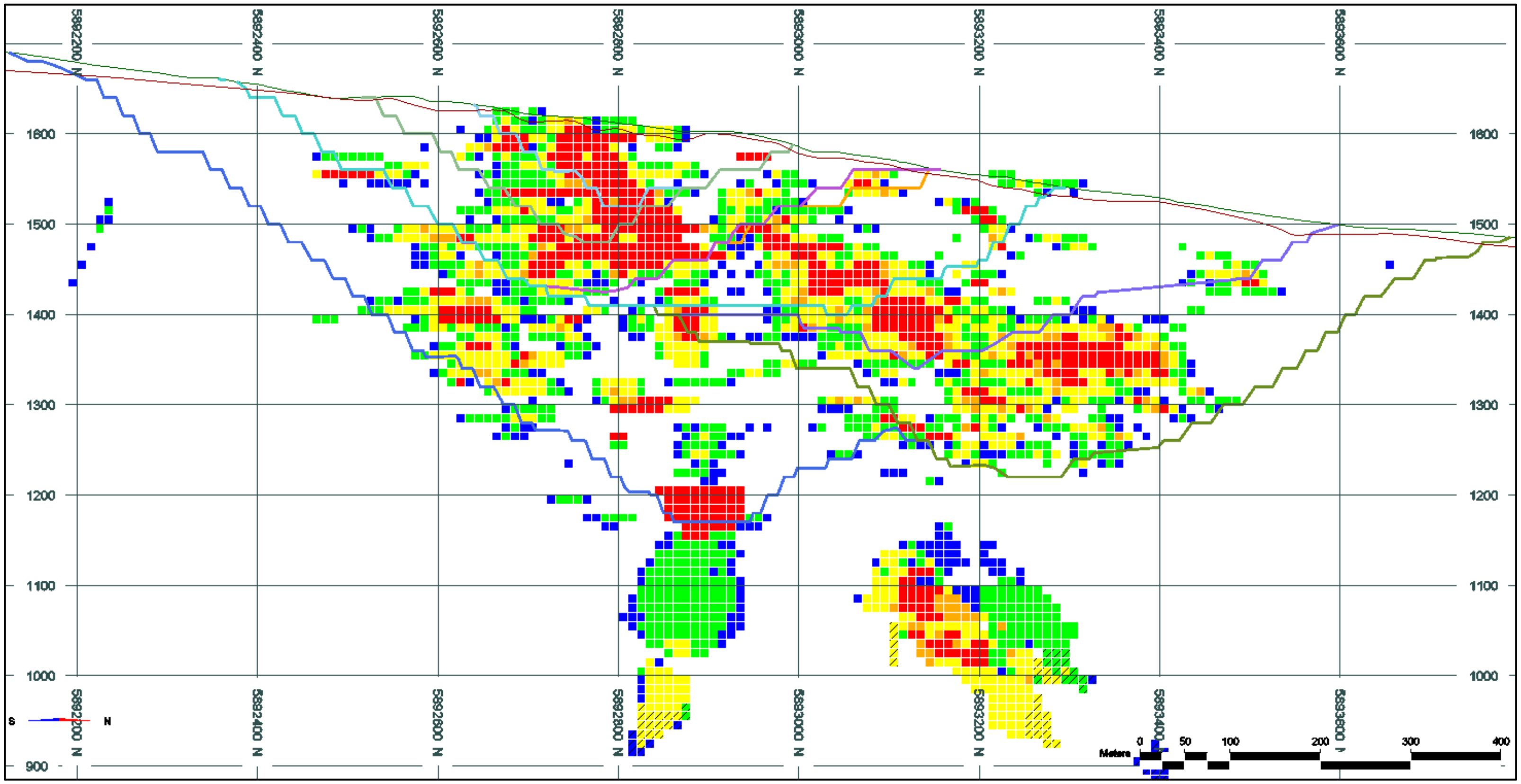
Note: prepared by MMTS, 2021.

Figure 16-13: North South Section, 375,050E Looking West



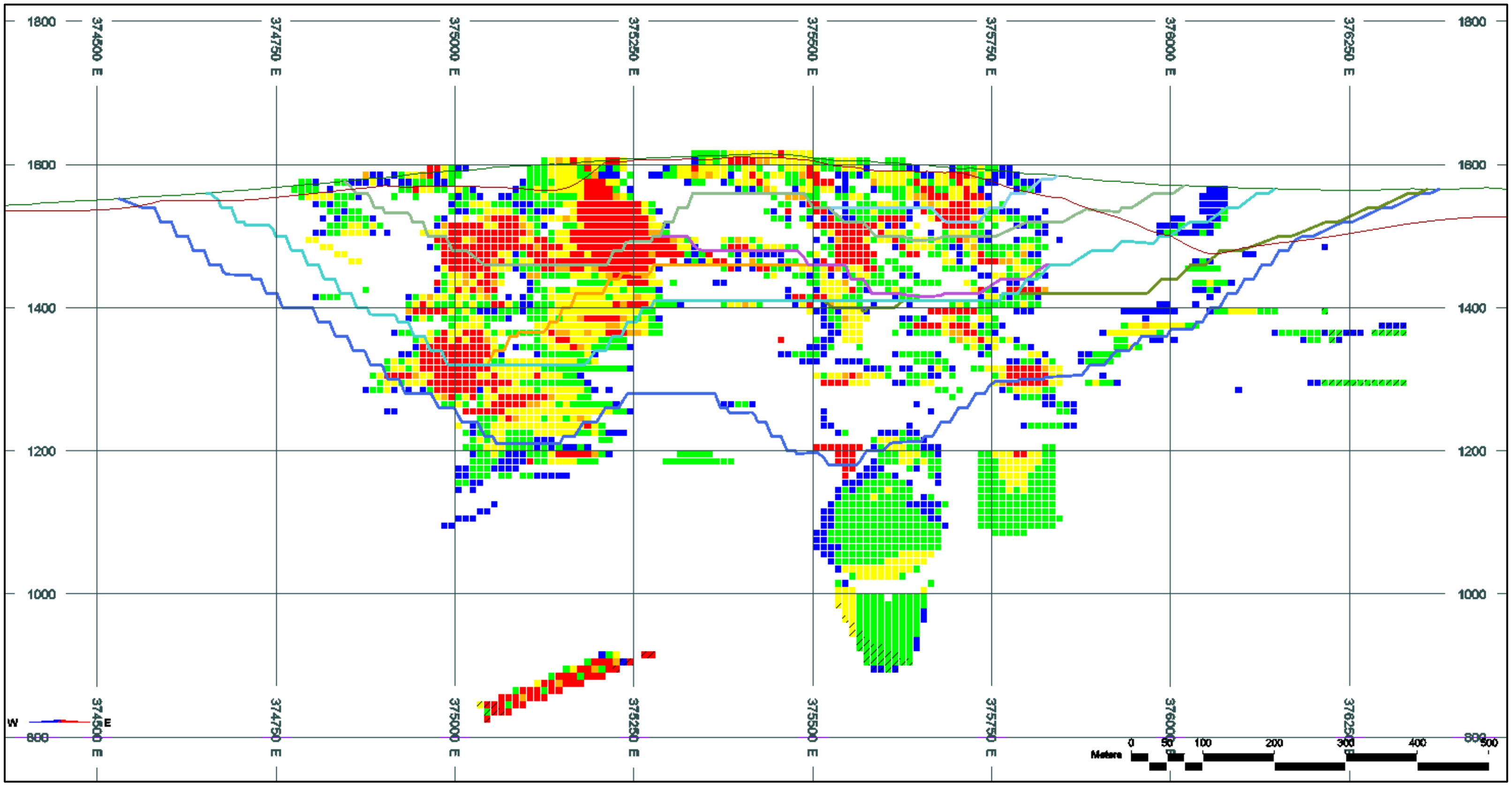
Note: prepared by MMTS, 2021

Figure 16-14: North South Section, 375,550E Looking West



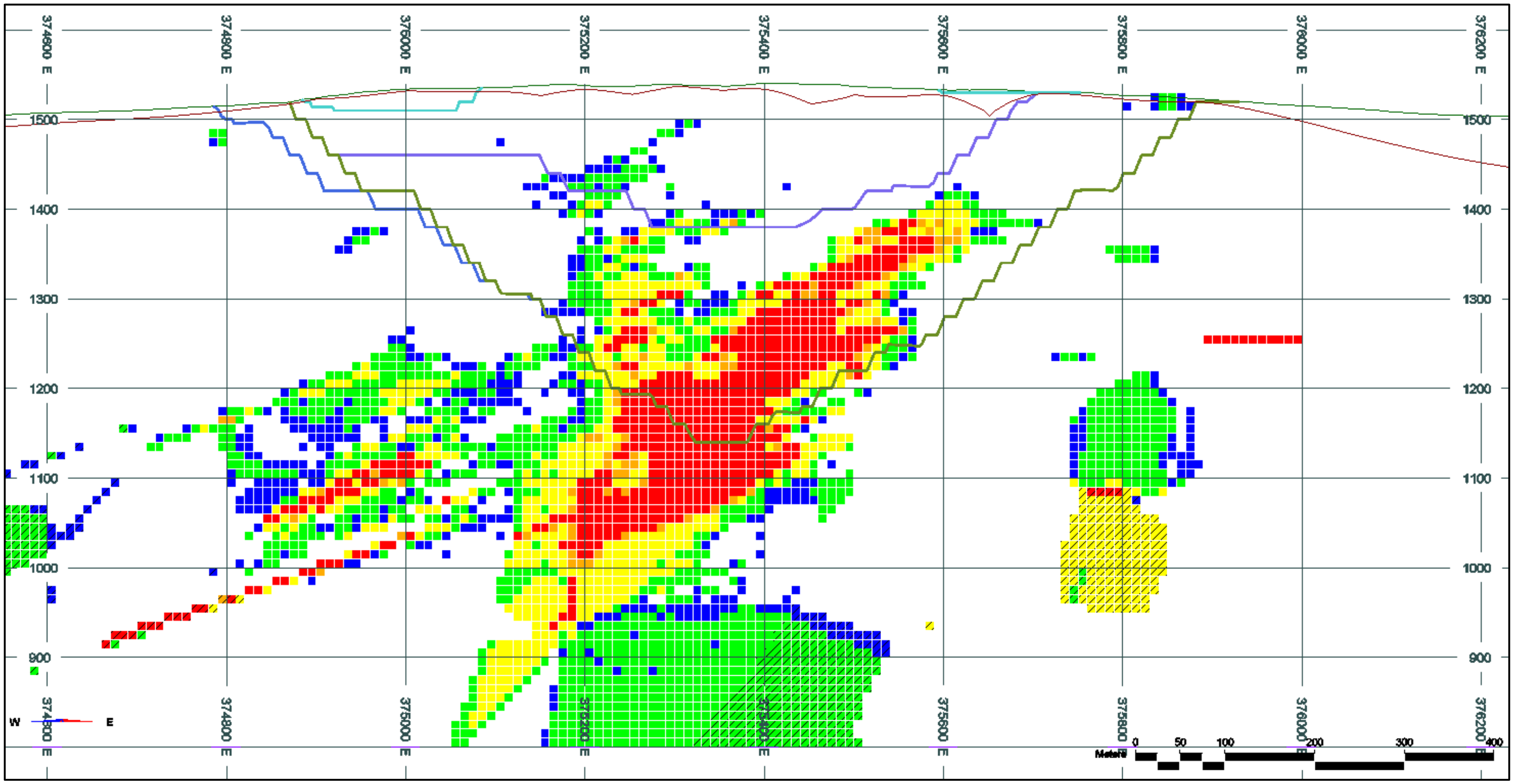
Note: prepared by MMTS, 2021

Figure 16-15: East West Section, 5,892,850N Looking North



Note: prepared by MMTS, 2021

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



Note: prepared by MMTS, 2021

16.4 Ex-Pit Haul Roads

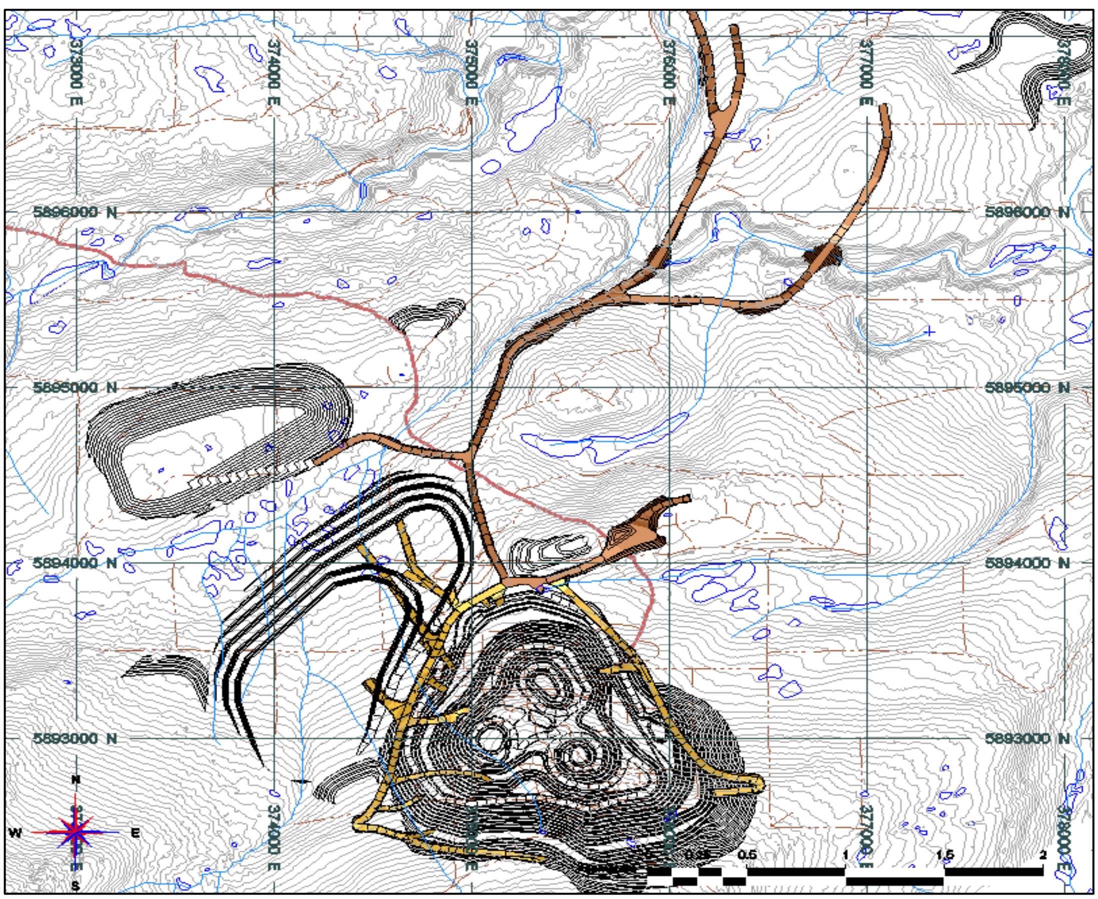
Mine haul roads external to the open pits are designed to haul ore and waste materials from the open pits to the scheduled destinations. The mine haul roads are designed with the following key inputs:

- 37 m wide ex-pit haul roads that incorporate 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 ex-pit haul roads are shown in Figure 16-17.

The existing exploration road network can be 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 16-17: Ex-Pit Mine Haul Roads



Note: prepared by MMTS, 2021

16.5 Ore Storage Facility

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.

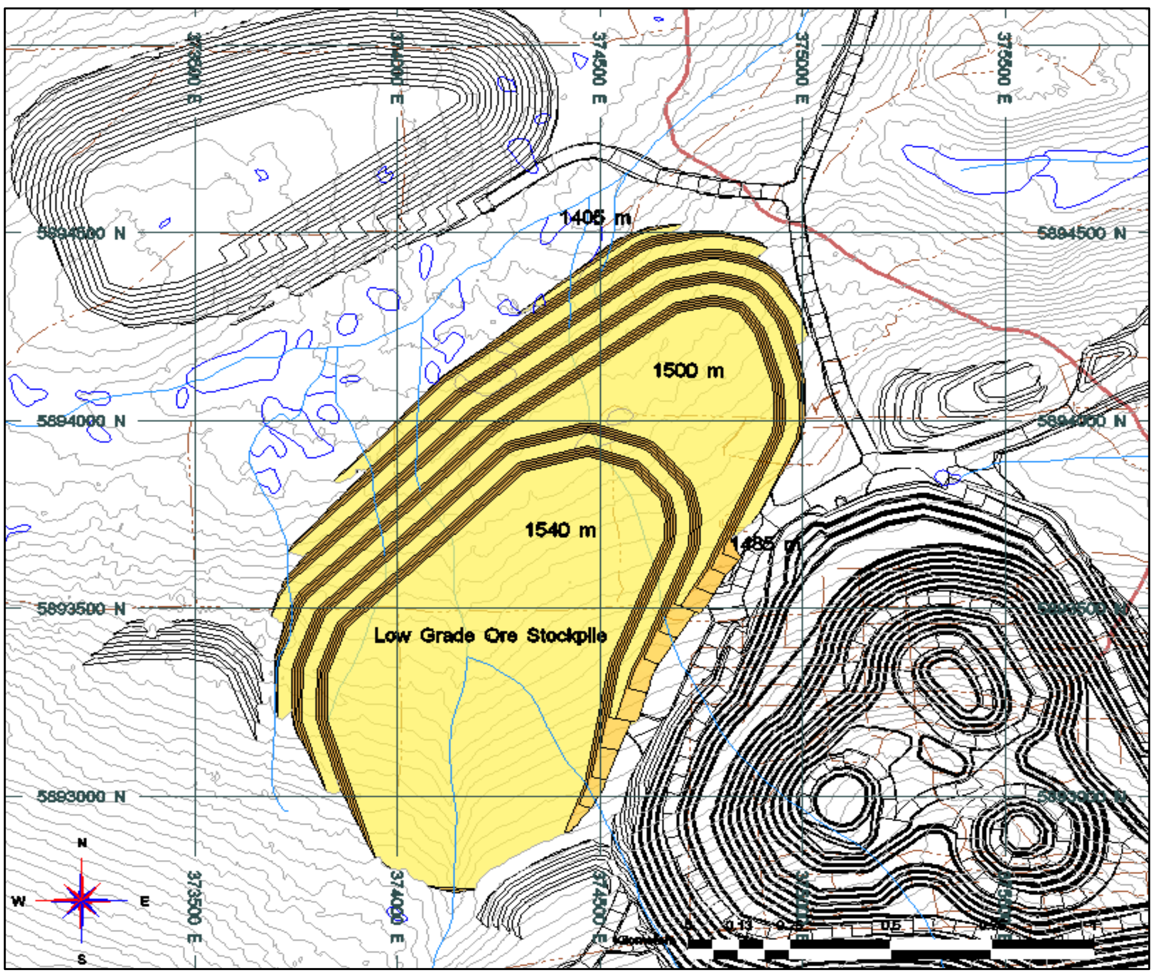
The crusher and ROM stockpiles will be located <1 km northeast of the pit limits.

Throughout the life of operations, ore grading 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 (Figure 16-18).

Cut-off grade optimization on the mine production schedule also sends ore above C$27/t NSR, mined in the first five years of the mine life, to a discrete pile 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 during pit operations and the lower-grade material once the pits are exhausted.

Figure 16-18: Ore Stockpile



Note: prepared by MMTS, 2021

The ore stockpile is designed within the existing EA certified project description boundary. The stockpile will be built on the hillside, on four 20 m lifts dumped out at angle of repose (1.3H:1V). Each facility is planned at a 3H:1V overall slope from the 1,405 masl elevation to the 1,540 masl elevation.

Low-grade ore is classified as 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.

16.6 Waste Rock Storage Facility

Waste stockpile designs have been completed that demonstrate the capacity of storing all non-ore materials from the Blackwater open pit.

Overburden and 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.

16.6.1 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 NRP = neutralization potential (NP)/acid potential (AP)), where PAG material was defined as an NPR ≤2 and NAG material defined by an NPR >2.

Values for NP were determined directly from ABA results or estimated from the calcium values from the larger geological assay dataset; the latter assumed calcium was only present as calcium carbonate and was the sole source of NP. Concentrations of sulphur were converted to AP values for use in the NPR calculation. Separate sulphur and NP block models were prepared and then the estimates combined to determine the NPR for ore and waste rock blocks.

Mine waste was classified according to the geochemical classification scheme developed by AMEC (AMEC, 2014b). As discussed above, 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 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-ML)
    - NAG4 – NPR > 2.0 and 600 ≤ Zn <1,000 ppm (NAG)
    - NAG5 – NPR > 2.0 and Zn < 600 ppm (NAG)
- Overburden (NAG)

- Low-grade ore (PAG)
- Tailings (PAG)

Laboratory kinetic tests (humidity cells) show that most PAG waste rock will have a relatively short lag time to acid production upon exposure to oxidizing conditions. All the PAG1 and some of the PAG2 humidity cell samples developed acidic leachate with little to no lag time. The results of the field bin tests for waste rock were like those for the humidity cell tests, but typically had a longer lag time to acid generation. PAG1 humidity cell samples had the highest dissolved metals loads compared to the other waste rock samples. Metal loads from sub-aqueous waste rock column tests were significantly lower than comparable humidity cell tests, due to oxygen-deficient conditions created by the water cover inhibiting sulphide mineral oxidation. Taken together, the results of waste rock kinetic testing indicate:

- the time to acid generation will be longer under field conditions than in laboratory testing;
- metal loads under neutral pH conditions will be lower than under acidic conditions; and
- underwater storage of waste rock will mitigate acid generation and significantly reduce ML potential.

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.

Most overburden samples (89 of 95) from the open pit, plant site, TSF, and access road area were classified as NAG based on ABA results. Six samples from the overburden-bedrock interface within the open pit footprint were classified as PAG; however, all but one of these samples had low total sulphur values and were classified as PAG only due to low NP. Net acid generating testing conducted on a subset of overburden samples confirmed the NAG character of these materials. Overburden will be monitored for ARD and ML leaching potential during excavation; however, in general, overburden can be used for construction and reclamation purposes.

Low-grade ore is classified as PAG with a relatively short lag time, and the low-grade ore stockpile is expected to generate acidic drainage with elevated metals until the ore is processed.

16.6.2 Waste Handling

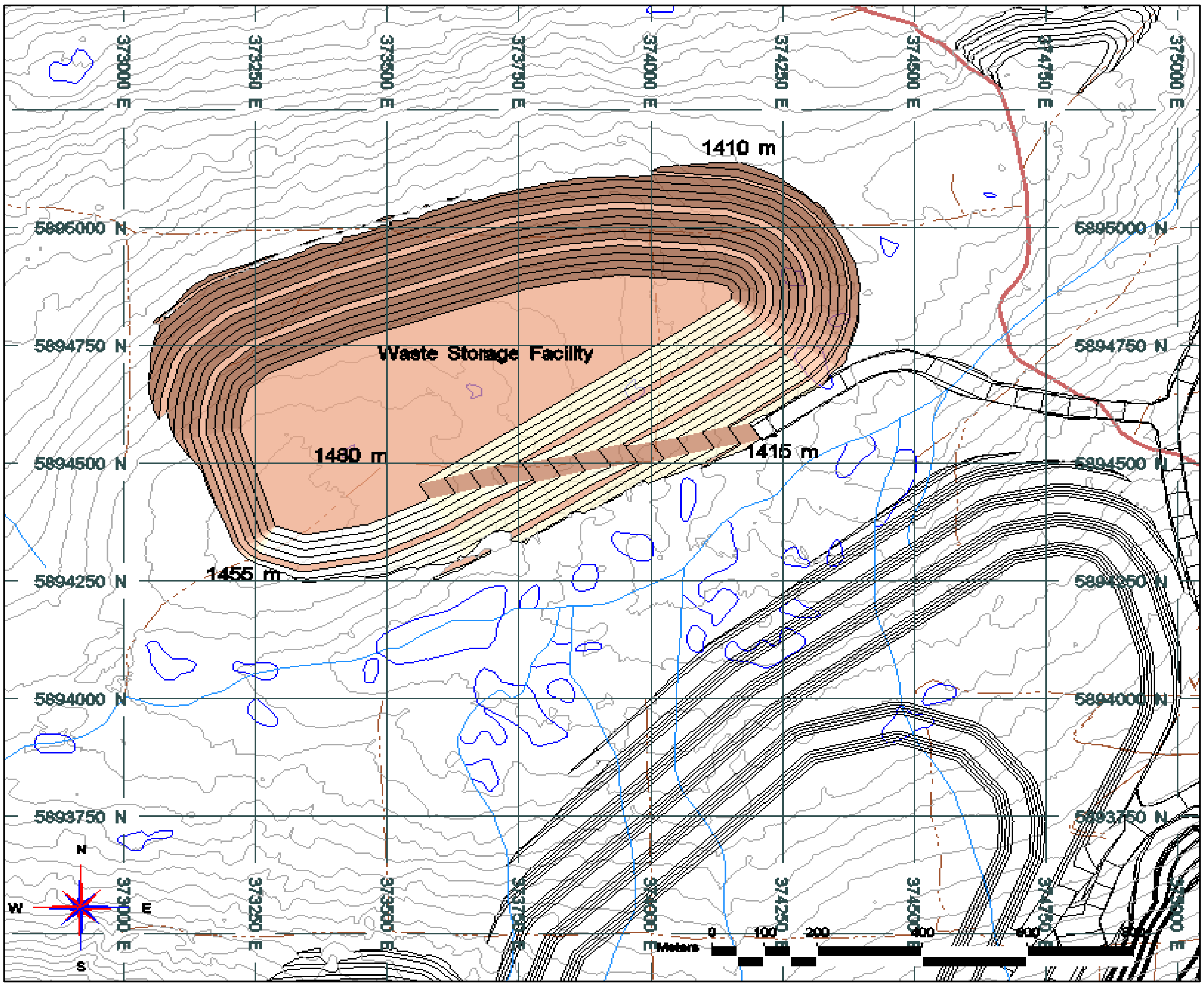
Concepts for handling and management of the various classifications of waste rock to maximize chemical stability were established during previous studies (AMEC, 2014b; KP, 2015) and remain unchanged for the 2021 FS. These were:

- NAG4 and NAG5 waste rock and overburden (no acid rock drainage and minimal metal leaching): Used as a construction material with surplus and unsuitable (geotechnical) materials disposed of in on-land stockpile(s), progressively reclaimed during operations and revegetated at closure.
- NAG3 waste rock (potentially metal leaching): Used to construct internal zones of the TSF embankments (i.e., within the containment provided by the TSF) and surplus co-disposed in the TSF in such a manner that the interstitial space is saturated within 3–5 years;

- PAG1 and PAG2 waste rock (expected metal leaching and acid rock drainage): Disposed within the TSF and used to construct internal zones of the TSF embankments in such a manner that the interstitial space is saturated within one year to limit oxidation and subsequent acid generation.

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 waste storage facility (WSF). This facility is shown in Figure 16-19. It is designed within the existing EA certified project description boundary.

Figure 16-19: Waste Storage Facility



Note: prepared by MMTS, 2021

The WSF 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.

16.7 Production Schedule

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 overall production schedule is included as Table 16-4.

The mill feed is illustrated in Figure 16-20 and shows the production tonnage and grade forecast; Figure 16-20 provides an illustration of the projected material mined and strip ratio.

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 of the Project.
  - Quarterly period detail is scheduled out for Years 2 and 3 of the Project.
- An annual mill feed rate of 6 Mt is targeted for the first five years of operation, increasing to 12 Mt for the next five years, and 20 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 5 throughput of 9 Mt is targeted, with expansion planned for the latter half of the year.
  - Year 10 throughput of 15 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;
- Pre-production mining requirements are as follows
  - 1.9 Mt of overburden and NAG waste rock for haul road construction purposes.
  - 6.0 Mt of overburden and NAG waste rock for tailings dam construction purposes;

- Any excess topsoil, overburden, ore and PAG waste rock that must be moved to access this construction rock is stockpiled (1.2 Mt).
- Ore mined to access this construction rock and for use in commission and mill ramp up (0.7 Mt).

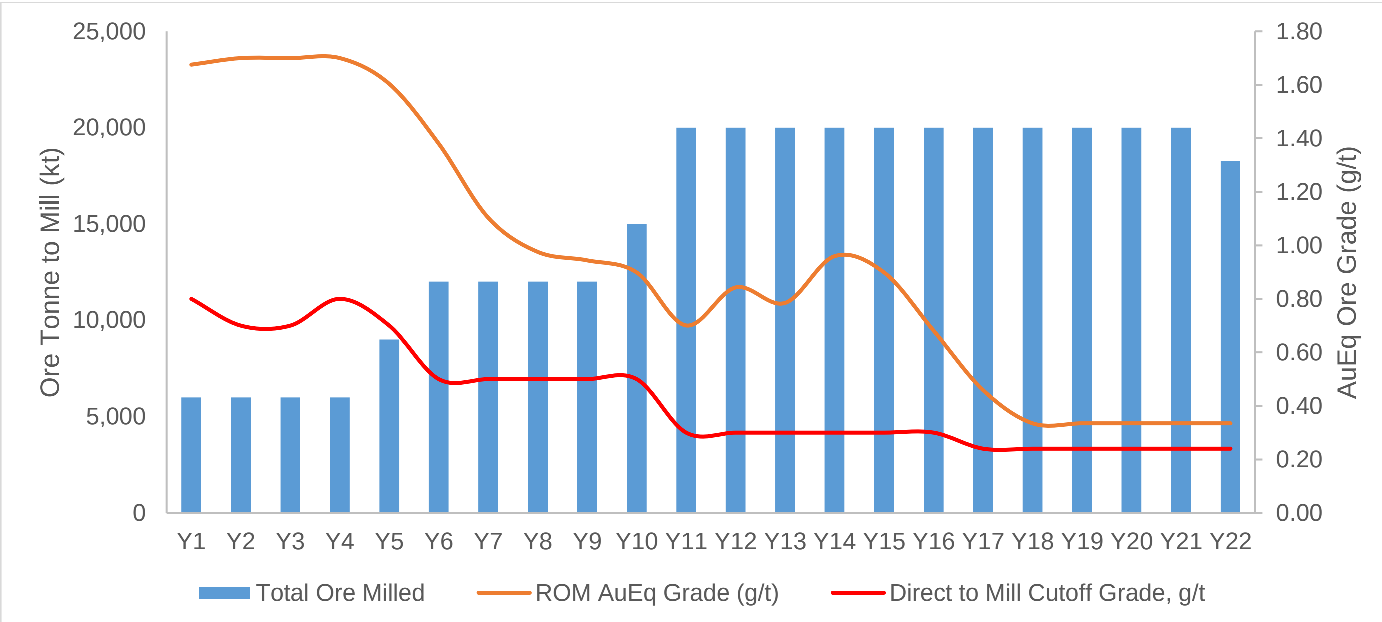
- Ore tonnes released in excess of the mill capacity are stockpiled.
- Low-grade ore is stockpiled and re-handled to the primary crushers at the end of mine life.

Table 16-4: Mine Production Schedule

	Unit	LOM	LOM no Preprod	Y-2	Y-1	Y1	Y2	Y3	Y4	Y5	Y6	Y7	Y8	Y9	Y10	Y11	Y12	Y13	Y14	Y15	Y16	Y17	Y18	Y19	Y20	Y21	Y22	Y23
TOTAL Ore Milled	ktonnes	334,278	334,278	-	-	6,001	6,000	6,000	6,000	9,000	12,000	12,000	12,000	12,000	15,000	20,000	20,000	20,000	20,000	20,000	20,000	20,000	20,000	20,000	20,000	20,000	18,274	-
Au	g/t	0.75	0.75	-	-	1.62	1.66	1.65	1.66	1.56	1.33	1.04	0.94	0.90	0.87	0.68	0.82	0.76	0.93	0.86	0.65	0.42	0.30	0.30	0.30	0.30	0.30	-
Ag	g/t	5.78	5.78	-	-	8.93	7.39	7.55	7.45	6.50	8.70	10.23	5.17	6.87	4.60	3.09	3.57	4.70	5.40	5.90	4.89	5.81	5.89	5.89	5.89	5.89	5.89	-
TOTAL Ore Mined from Pit	ktonnes	334,278	333,554	3	721	17,473	15,119	11,591	21,432	22,476	22,227	25,090	26,194	25,587	23,875	20,817	16,100	17,345	25,382	24,403	15,460	2,982	-	-	-	-	-	-
Au	g/t	0.75	0.75	0.32	0.52	0.83	0.90	1.03	0.76	0.86	0.86	0.64	0.60	0.58	0.64	0.63	0.85	0.72	0.78	0.75	0.74	1.14	-	-	-	-	-	-
Ag	g/t	5.78	5.78	4.73	5.71	6.69	6.13	6.50	6.15	5.01	7.45	11.51	5.33	7.99	4.32	2.55	2.42	3.84	4.94	5.45	4.20	5.37	-	-	-	-	-	-
Ore Mined Directly to Mill	ktonnes	194,885	194,885	-	-	6,001	6,000	6,000	6,000	9,000	12,000	12,000	12,000	12,000	12,000	17,500	14,300	14,100	20,000	20,000	13,000	2,982	-	-	-	-	-	-
Au	g/t	1.04	1.04	-	-	1.62	1.66	1.65	1.66	1.56	1.33	1.04	0.94	0.90	0.95	0.70	0.92	0.83	0.93	0.86	0.83	1.14	-	-	-	-	-	-
Ore Mined to Stockpile	ktonnes	139,393	138,669	3	721	11,472	9,118	5,591	15,432	13,476	10,227	13,090	14,194	13,587	11,874	3,317	1,800	3,244	5,382	4,403	2,460	0	-	-	-	-	-	-
Au	g/t	0.33	0.33	0.32	0.52	0.41	0.40	0.36	0.40	0.40	0.31	0.28	0.32	0.29	0.32	0.26	0.26	0.25	0.25	0.25	0.25	-	-	-	-	-	-	-
Ag	g/t	5.94	5.94	4.73	5.71	5.52	5.30	5.37	5.64	4.02	5.98	12.69	5.46	8.99	4.45	2.11	1.87	2.95	3.24	3.37	3.63	-	-	-	-	-	-	-
Stockpile Retrieval to Mill	ktonnes	139,393	139,393	-	-	-	-	-	-	-	-	-	-	-	3,000	2,500	5,700	5,900	-	-	7,000	17,018	20,000	20,000	20,000	20,000	18,274	-
Au	g/t	0.33	0.33	-	-	-	-	-	-	-	-	-	-	-	0.58	0.58	0.58	0.58	-	-	0.31	0.30	0.30	0.30	0.30	0.30	0.30	-
Ag	g/t	5.94	5.94	-	-	-	-	-	-	-	-	-	-	-	6.27	6.27	6.27	6.27	-	-	5.96	5.89	5.89	5.89	5.89	5.89	5.89	-
Stockpile Balance	ktonnes			3	724	12,196	21,314	26,905	42,337	55,814	66,041	79,131	93,324	106,912	115,786	116,603	112,703	110,047	115,429	119,832	115,292	98,274	78,274	58,274	38,274	18,274	-	-
Au	g/t			0.32	0.52	0.42	0.41	0.40	0.40	0.40	0.39	0.37	0.36	0.35	0.34	0.34	0.32	0.31	0.30	0.30	0.30	0.30	0.30	0.30	0.30	0.30	-	-
Ag	g/t			4.73	5.70	5.53	5.43	5.42	5.50	5.14	5.27	6.50	6.34	6.68	6.46	6.34	6.27	6.18	6.04	5.94	5.89	5.89	5.89	5.89	5.89	5.89	-	-
TOTAL Waste Mined	ktonnes	672,930	663,855	1,296	7,779	26,507	26,164	30,622	38,518	31,769	40,366	36,453	45,806	61,412	60,125	57,183	60,485	57,794	46,319	28,450	15,230	652	-	-	-	-	-	-
NAG Rock Waste (NAG4/NAG5)	ktonnes	117,421	113,898	689	2,833	5,568	3,555	8,655	8,432	3,165	5,042	1,930	2,815	10,189	10,210	18,472	18,140	15,200	2,263	206	57	-	-	-	-	-	-	-
PAG Rock Waste (PAG1, PAG2, NAG3)	ktonnes	468,151	467,159	1	991	12,375	12,563	13,909	22,362	26,997	32,161	31,496	27,780	47,676	40,725	31,674	36,778	42,538	44,057	28,244	15,173	652	-	-	-	-	-	-
Overburden Waste	ktonnes	87,268	82,707	606	3,954	8,564	10,042	8,058	7,724	1,607	3,163	3,027	15,211	3,548	9,185	7,031	5,492	56	-	-	-	-	-	-	-	-	-	-
Wasted Inferred	ktonnes	91	91	-	-	-	4	-	-	-	-	-	-	-	5	6	76	-	-	-	-	-	-	-	-	-	-	-
Au	g/t	0.32	0.32	-	-	-	0.23	-	-	-	-	-	-	-	0.11	0.11	0.35	-	-	-	-	-	-	-	-	-	-	-
Ag	g/t	4.48	4.48	-	-	-	1.70	-	-	-	-	-	-	-	21.50	21.50	2.00	-	-	-	-	-	-	-	-	-	-	-
Waste Rehandle	ktonnes	36,588	36,588	-	-	-	-	-	-	13,964	1,793	3,164	6,791	1,892	-	-	-	-	-	1,655	7,329	-	-	-	-	-	-	-
Waste Destination Summary																												
OVB to Construction	ktonnes	605	-	605	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
NAG to Construction	ktonnes	5,256	3,969	687	600	200	200	200	1,200	200	200	200	200	200	200	200	200	200	200	124	45	-	-	-	-	-	-	-
OVB to Tailing Dam	ktonnes	66,640	62,740	-	3,900	3,170	1,905	5,095	1,655	9,955	4,955	4,340	7,070	5,440	8,000	5,500	4,000	-	-	1,655	-	-	-	-	-	-	-	-
PAG to Tailing Dam	ktonnes	36,475	36,475	-	-	-	-	-	-	-	3,192	6,325	2,663	7,000	6,295	3,875	4,195	2,740	190	-	-	-	-	-	-	-	-	-
NAG to Tailing Dam	ktonnes	107,550	105,450	-	2,100	3,215	2,990	1,060	555	8,580	3,940	3,580	9,405	1,420	10,010	18,272	17,939	15,000	2,062	81	7,341	-	-	-	-	-	-	-
PAG to TSF	ktonnes	431,767	430,775	1	991	12,375	12,566	13,909	22,362	26,997	28,969	25,171	25,117	40,676	34,435	27,805	32,659	39,798	43,867	28,244	15,173	652	-	-	-	-	-	-
OVB to WSF*	ktonnes	20,023	19,967	1	54	5,394	8,137	2,963	6,069	-8,348	-1,792	-1,313	8,141	-1,892	1,185	1,531	1,492	56	-	-1,655	-	-	-	-	-	-	-	-
NAG to WSF*	ktonnes	4,615	4,479	2	133	2,153	365	7,395	6,677	-5,615	902	-1,850	-6,790	8,569	-	-	1	-	1	1	-7,329	-	-	-	-	-	-	-
Strip Ratio (Waste/Resource Mined)		2.0	2.0	-	-	1.5	1.7	2.6	1.8	1.4	1.8	1.5	1.7	2.4	2.5	2.7	3.8	3.3	1.8	1.2	1.0	0.2	-	-	-	-	-	-
Cumulative Strip Ratio				-	12.5	2.0	1.9	2.1	2.0	1.8	1.8	1.8	1.8	1.8	1.9	2.0	2.1	2.2	2.2	2.1	2.0	2.0	2.0	2.0	2.0	2.0	2.0	2.0
Total Material Mined	ktonnes	1,007,208	997,409	1,300	8,499	43,980	41,282	42,213	59,950	54,245	62,593	61,543	72,000	87,000	84,000	78,000	76,585	75,138	71,701	52,853	30,690	3,634	-	-	-	-	-	-
Cumulative Material Mined	ktonnes			1,300	9,799	53,779	95,061	137,274	197,224	251,469	314,062	375,606	447,606	534,606	618,606	696,606	773,191	848,329	920,031	972,884	1,003,574	1,007,208	1,007,208	1,007,208	1,007,208	1,007,208	1,007,208	1,007,208
Total Material Moved	ktonnes	1,183,189	1,173,390	1,300	8,499	43,980	41,282	42,213	59,950	68,209	64,386	64,707	78,791	88,892	87,000	80,500	82,285	81,038	71,701	54,508	45,019	20,652	20,000	20,000	20,000	20,000	18,274	-

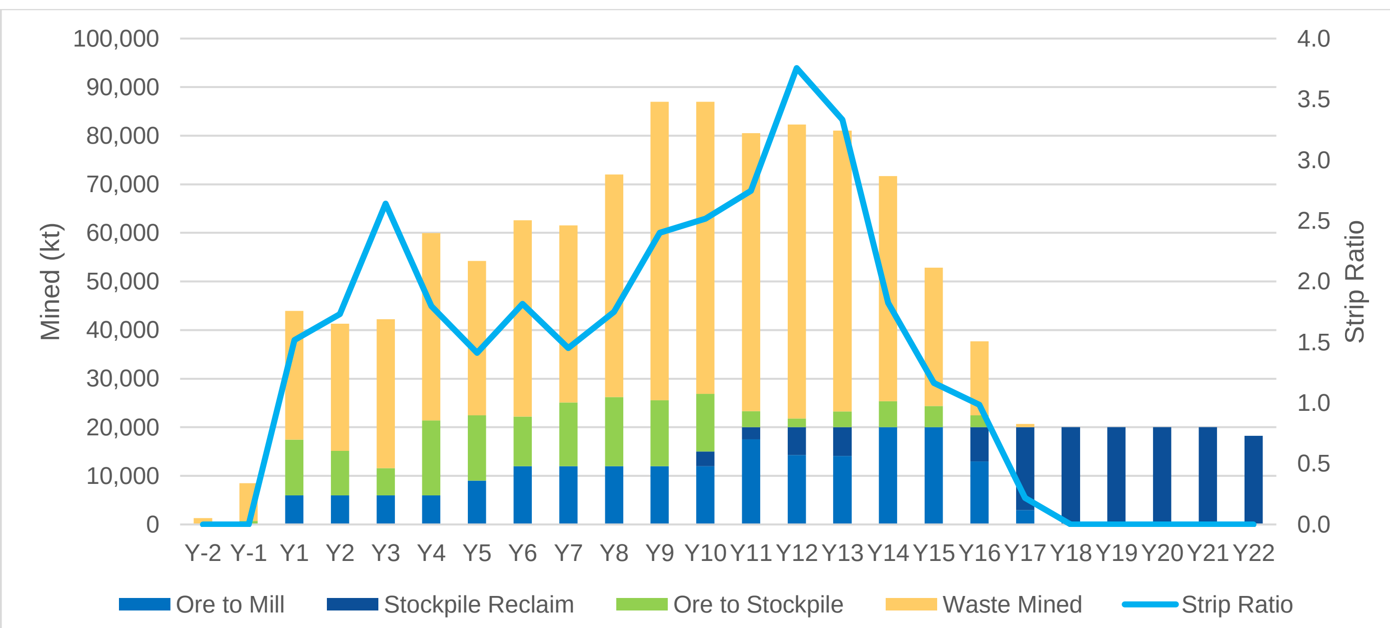
* Waste tonnages are net of rehandle

Figure 16-20: Planned Mill Feed Tonnes and Grade



Note: prepared by MMTS, 2021

Figure 16-21: Planned Material Mined and Strip Ratio



Note: prepared by MMTS, 2021

16.7.1 Mine Sequence

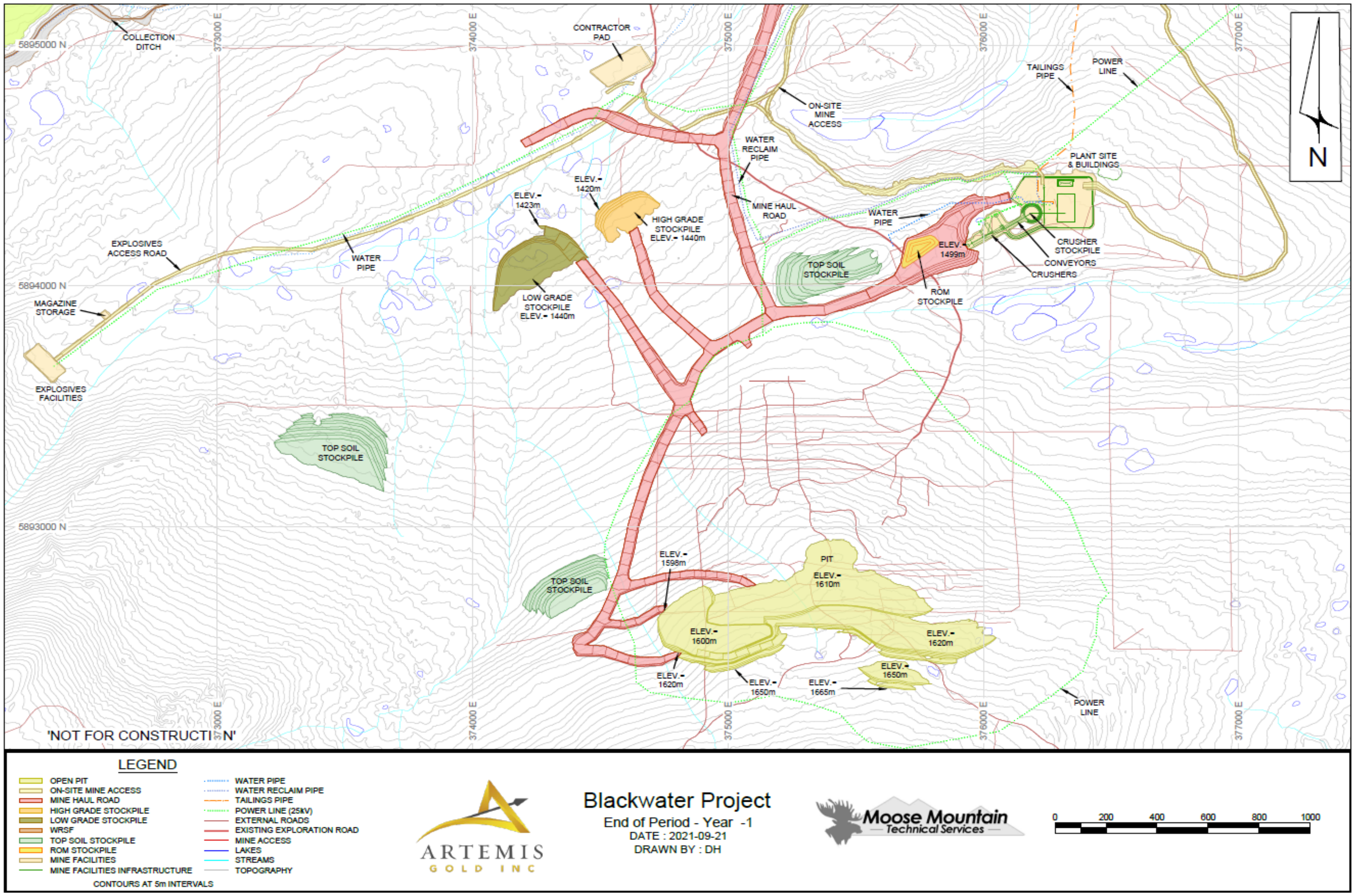
The pit operations will run for 19 years, including two years of pre-production. Following pit operations, stockpile re-handling operations will continue for five additional years. LOM activities are summarized in Table 16-5. End-of-period (EOP) layouts for Year-1, Year 1, Year 2, Year 5, Year 8, Year 13 and Year 17 are illustrated in Figure 16-22 to Figure 16-28.

Table 16-5: Annual Mine Operations

Year	Activity
Y-2 and Y-1	Clearing and grubbing the initial pit phases Clearing and grubbing of ex-pit haul road, and portions of the ore stockpiles and WSF. Haul road construction from the pits to the stockpiles, crusher, and tailings dam. Initial grade control delineation drilling to the 1,580 bench of starter pit Mining of the Construction Borrow pit down to 1,590 bench. Mining of the starter pit down to the 1,600 bench. Delivery of construction rock to the ROM pad. Delivery of construction overburden and rock to TSF C dam. Stockpiling high-grade ore on the ROM pad and ore stockpile for use in mill commissioning. Stockpiling low-grade ore in the ore stockpile for storage until the end of mine life. Delivery of excess mined overburden to the WSF.
Y1	Continued clearing and grubbing of pit and stockpile areas (annually throughout mine life). Construction borrow pit mined down to 1,490 bench; starter pit mined down to 1,540 bench. Continued stockpiling of excess high-grade ore to the ore stockpile (continues until Y5). Continued delivery of low-grade ore to the ore stockpile (continues until Y17). Pit electrification and purchase of electric driven drills and shovels. Delivery of construction fill to TSF C dam (continues until Y16). Delivery of excess mined overburden and NAG rock to the WSF (continues until Y13).
Y2	Construction borrow pit mined to the pit bottom on the 1450 bench. Starter pit mined to the 1,480 bench East Pushback 1 pit mined to the 1,550 bench East Pushback 2 pit mined to the 1,610 bench.
Y3 to Y5	Starter pit mined to the pit bottom on the 1,440 bench (Y3). East Pushback 1 pit mined to the pit bottom on the 1460 bench (Y4). East Pushback 2 pit mined to the 1440 bench; West Pushback pit mined to the 1,470 bench; North Pushback pits 1 and 2 mined to the 1,530 bench. Y5 final period of high-grade ore delivery to the ore stockpile. Y5 start of delivery of construction fill to the TSF D dam (continues until Y16). Rehandle of WSF material (14 Mt) to the TSF D dam. Mill feed increased to 9 Mt in Y5.
Y6 to Y08	Mill feed increased to 12 Mtpa in Y6. East Pushback 2 pit mined to the pit bottom on the 1,390 bench (Y6). West Pushback pit mined to the pit bottom on the 1,320 bench (Y8). North Pushback 1 and 2 pits mined to the 1,390 bench. Southeast pushback pit mined to the 1,490 bench. Re-handle of WSF material (12 Mt) to the TSF D dam.
Y09 to Y13	Mill feed increased to 20 Mtpa in Y11. North Pushback 1 pit mined to the pit bottom on the 1,260 bench (Y11). North Pushback 2 pit mined to the pit bottom on the 1,140 bench (Y14). South Pushback pit mined to the pit bottom on the 1,160 bench (Y18). Re-handle to crusher of remaining stockpiled high-grade ore (Y13, 14, and 15, stockpile depleted). Y17 final period of low-grade ore delivery to the low-grade ore stockpile. Re-handle to crusher of low-grade stockpiled ore (Y15, Y17, and Y18). Delivery of excess mined overburden and NAG rock to the lower NAG and overburden stockpile.

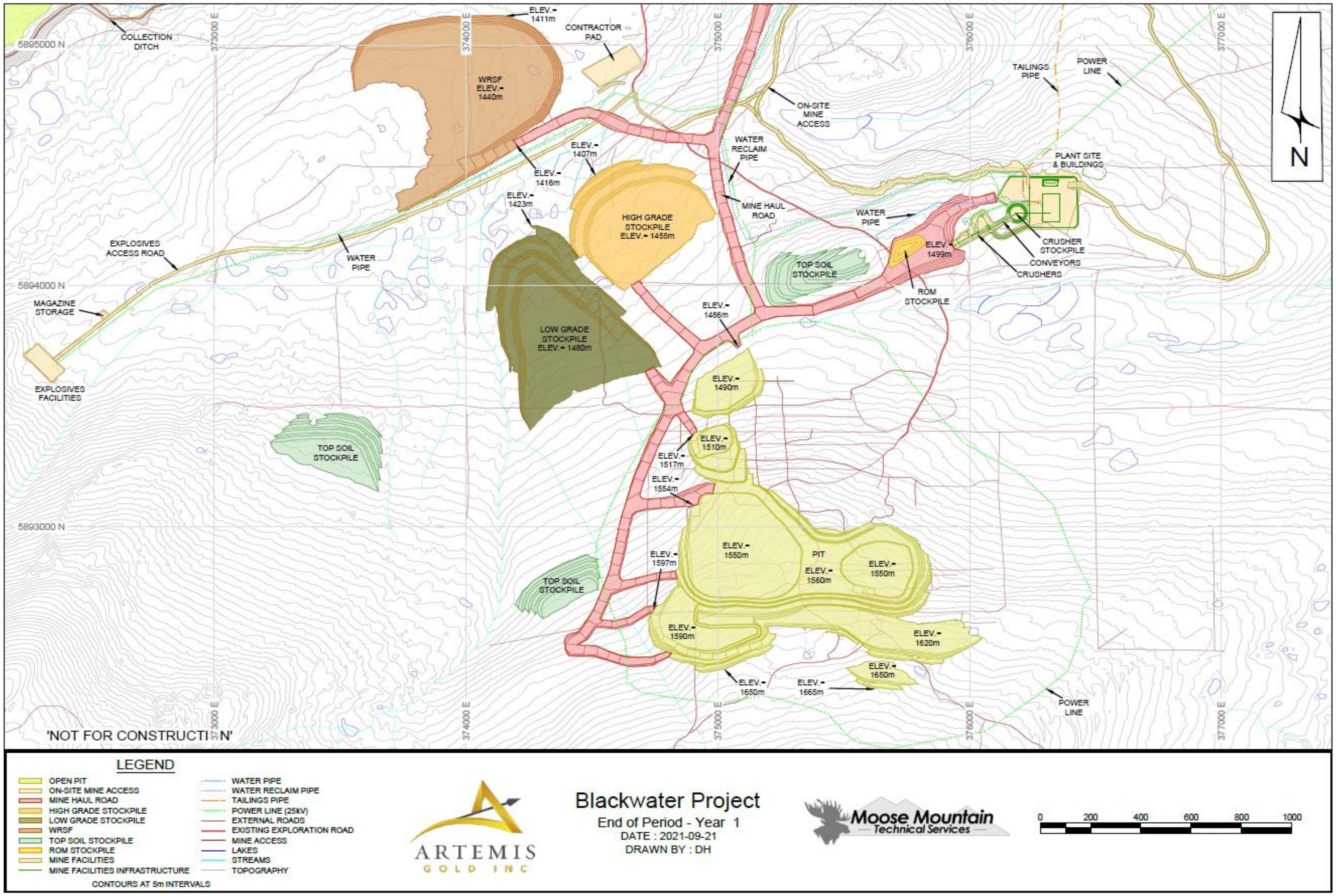
Year	Activity
Y14 to Y17	Southwest pushback pit mined to the pit bottom on the 1,160 bench (Y17). Y16 final period of low-grade ore delivery to the low-grade ore stockpile. Re-handle of WSF material (9 Mt) to the TSF C and D dams. Re-handle of low-grade stockpiled ore (24 Mt) to the crusher.
Y18 to Y22	Re-handle to crusher of remaining stockpiled low-grade ore (98 Mt, stockpiled depleted). Initiate work on closure plan for pits and WSF.

Figure 16-22: EOP Mine Operations, Y-1



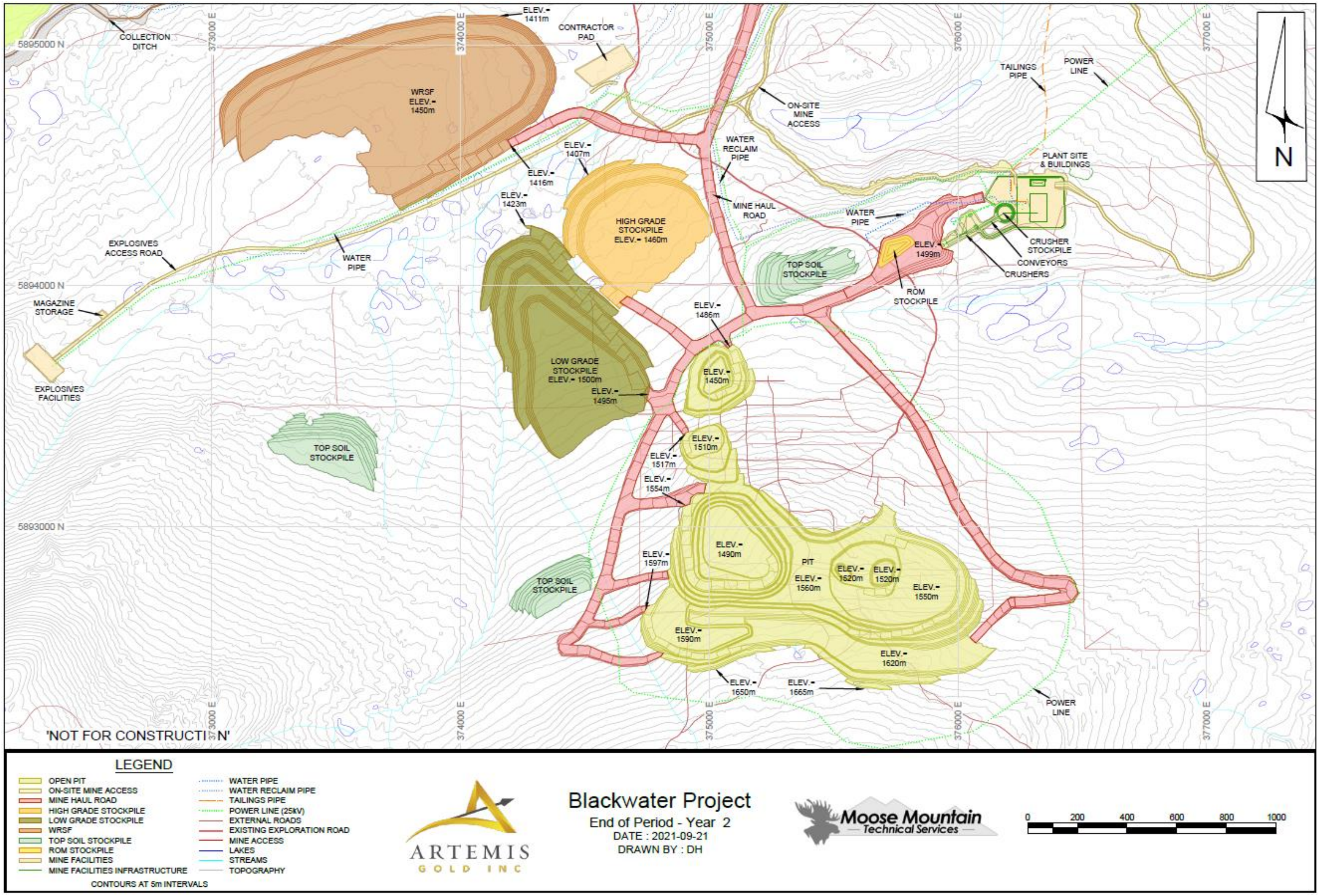
Note: prepared by MMTS, 2021

Figure 16-23: EOP Mine Operations, Y1



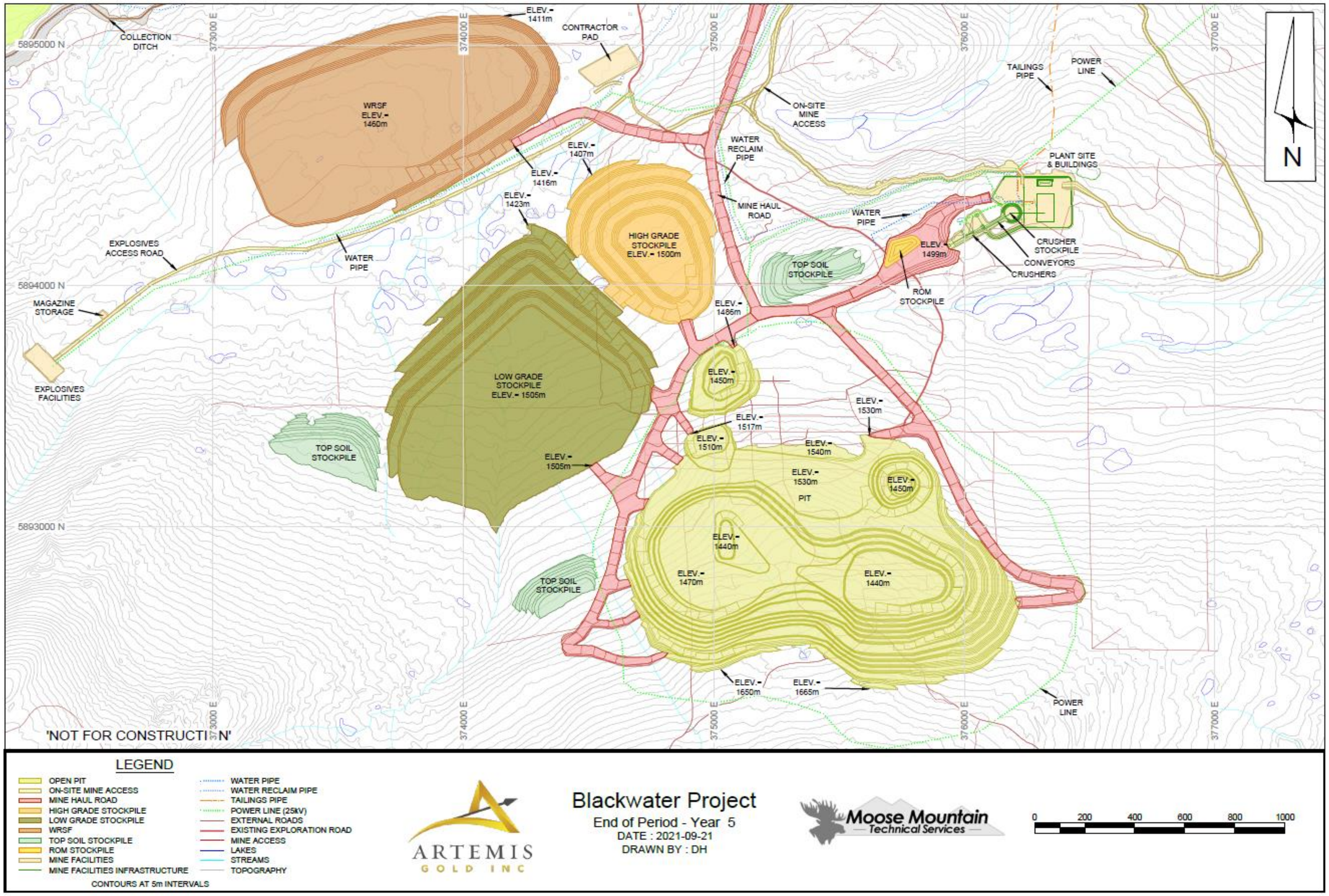
Note: prepared by MMTS, 2021

Figure 16-24: EOP Mine Operations, Y2



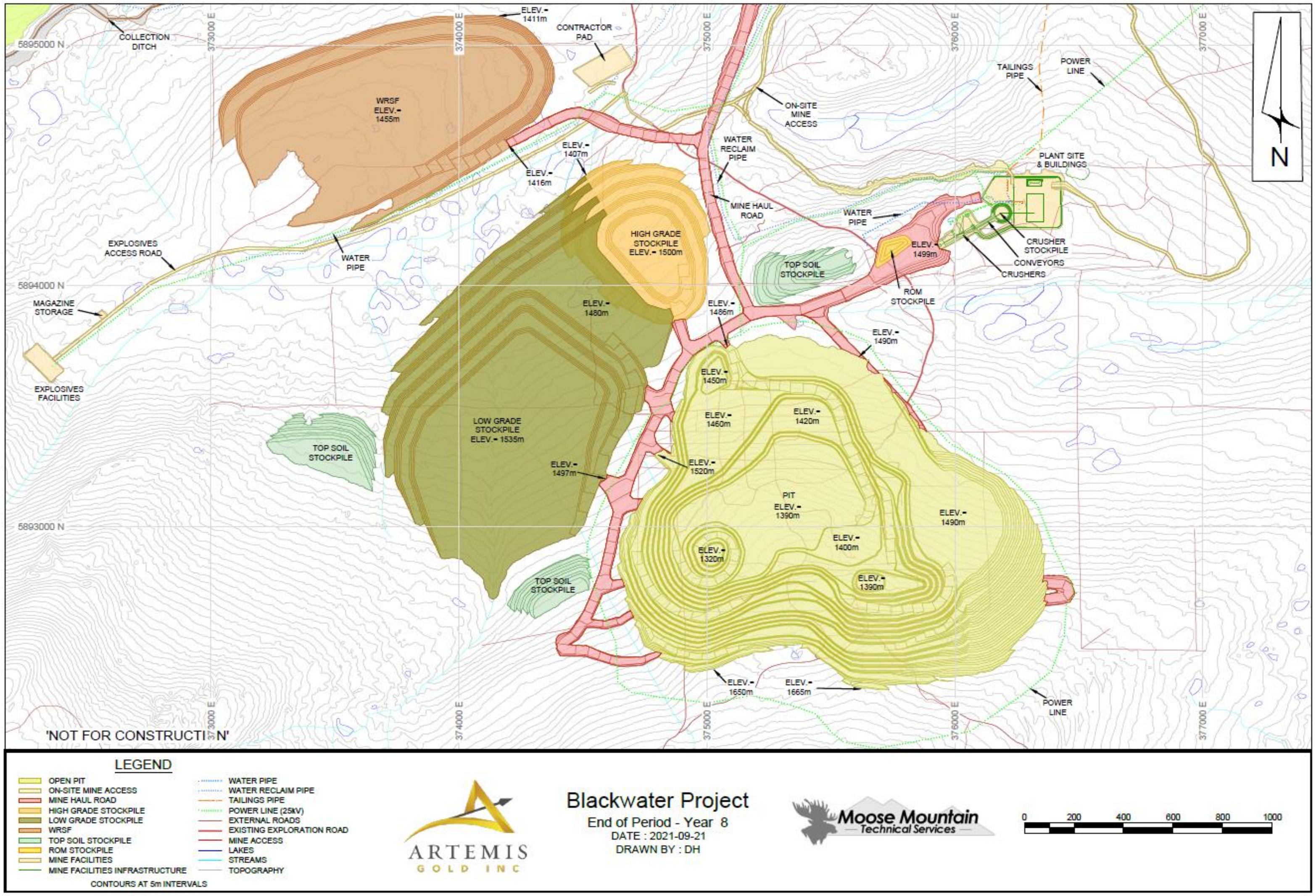
Note: prepared by MMTS, 2021

Figure 16-25: EOP Mine Operations, Y5



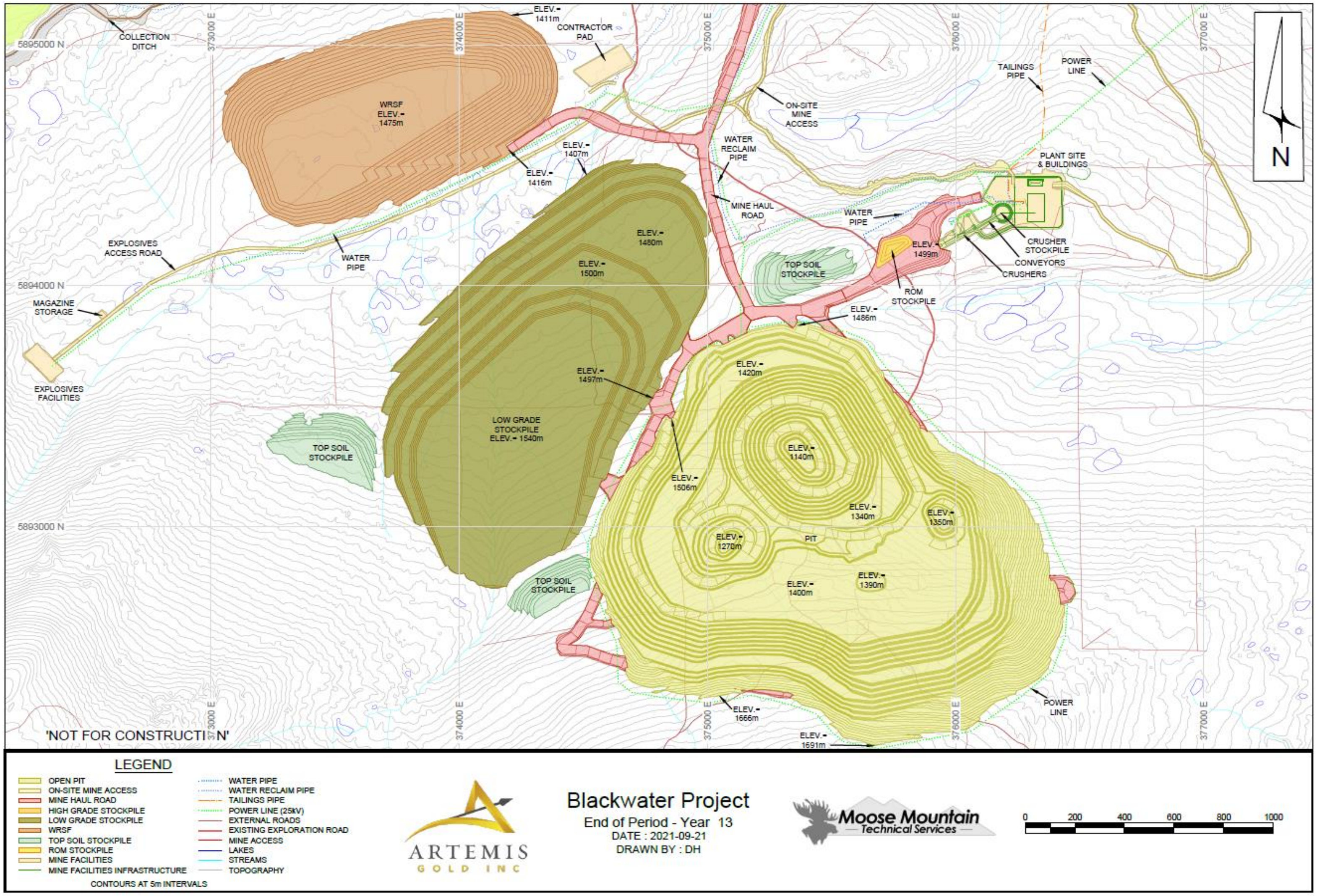
Note: prepared by MMTS, 2021

Figure 16-26: EOP Mine Operations, Y8



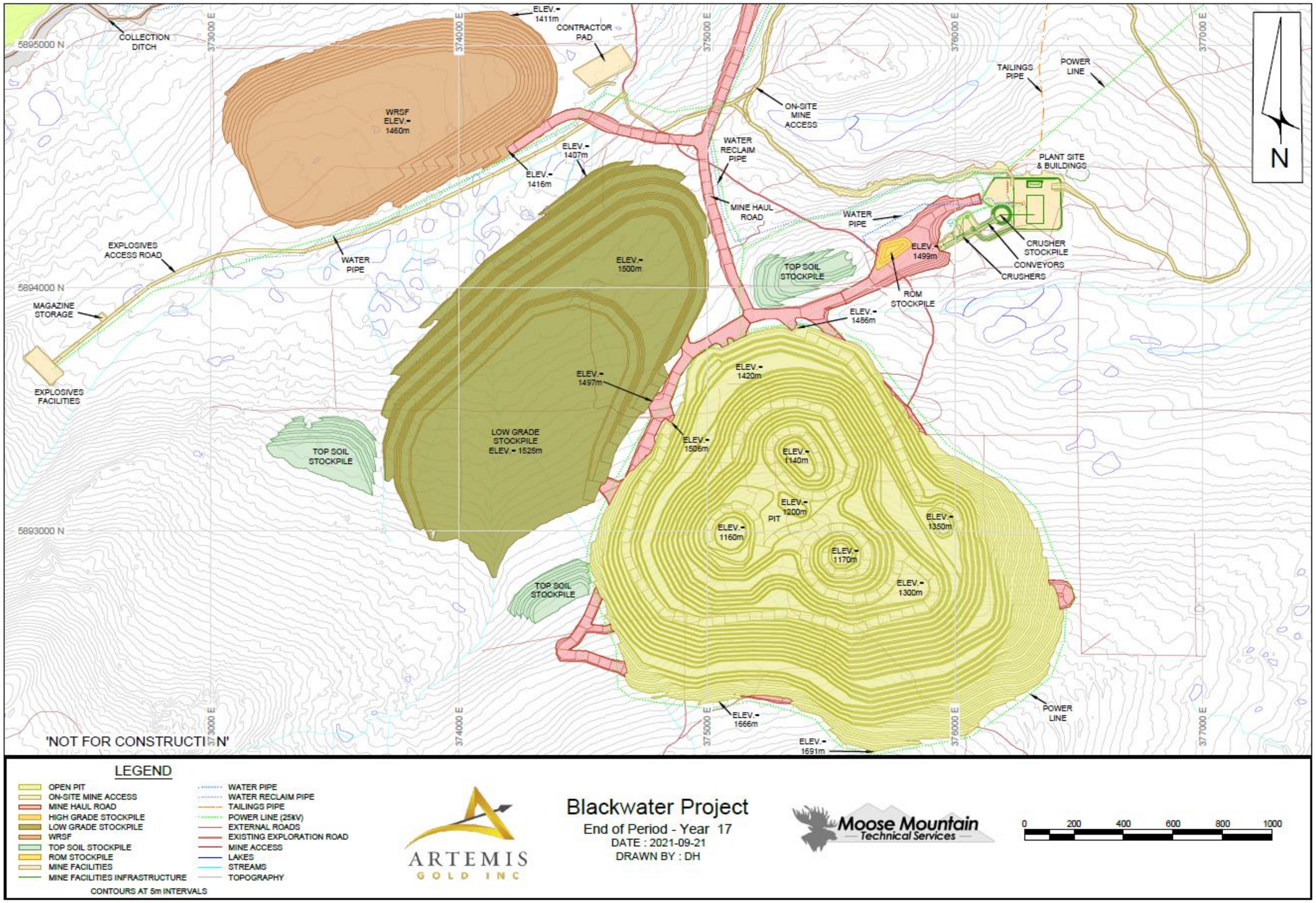
Note: prepared by MMTS, 2021

Figure 16-27: EOP Mine Operations, Y13



Note: prepared by MMTS, 2021

Figure 16-28: EOP Mine Operations, Y17



Note: prepared by MMTS, 2021

16.8 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 insitu conditions. Powder factors average 0.28 kg/t in ore and 0.20 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.

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;
- Ore control;
- 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; 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 348 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 35 persons will be required for mine operations, including the mine and maintenance supervision, mine engineering and geology.

16.8.1 Pit Dewatering

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 to 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 life of mine.

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.

16.9 Mine Equipment

Grade control drilling will be carried out with 144 mm (5.5″) diesel RC drills, with sampling and assaying on 3 m intervals. Production drilling will be carried out with 255 mm (10”) electric rotary drills in waste and 200 mm (8”) diesel rotary drill in ore.

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 (22.0 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 (19.0 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 of the project and additional waste 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 (190 t payload) are proposed to be flexible enough meet the targeted production levels and to maintain productivity of the loading units. Larger rigid frame haulers (230 t payload) are proposed for waste hauling as the stripping ratio increases, the haul distances get longer and additions to the fleet are required in Year 4 of the project and beyond. Four 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.

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. Front-end wheel loaders (12.0 m^3 and 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 next to the process plant, 1.3 km northeast of the pit limits.

Primary mining equipment requirements are summarized in Table 16-6 and Table 16-7.

Table 16-6: Primary Mining Fleet Schedule

	Y-2	Y-1	Y1	Y2	Y3	Y4	Y5	Y6	Y7	Y8	Y9	Y10	Y11	Y12	Y13	Y14	Y15	Y16	Y17	Y18-22
Drilling																				
Electric rotary tracked drill 254 mm (10”) holes	-	-	1	1	1	1	1	1	1	1	2	2	2	2	2	2	2	1	1	-
Diesel rotary tracked drill 203 mm (8”) holes	1	1	3	3	3	4	4	4	4	4	4	4	4	4	4	4	4	3	2	-
Diesel RC tracked drill 144 mm (5.5”) holes	1	1	1	1	1	2	2	2	2	2	2	2	2	2	2	2	2	2	2	-
Loading																				
Electric hydraulic front shovel 34 m^3 bucket	-	-	1	1	1	1	2	2	2	2	3	3	3	3	3	2	2	1	1	1
Diesel hydraulic excavator 22 m^3 bucket	-	1	1	1	1	2	2	2	2	2	2	2	2	2	2	2	2	2	1	-
Wheel loader 19 m^3 bucket	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1
Hauling																				
Rigid frame haul truck 230 t payload	-	-	-	-	-	4	10	10	10	15	21	21	21	21	21	21	14	7	4	4
Rigid frame haul truck 190 t payload	3	5	18	18	18	18	18	18	18	18	18	18	18	18	18	18	18	18	3	-
Articulated haul truck 40 t payload	2	4	4	4	4	4	4	4	4	4	4	4	4	4	4	4	4	4	2	2

Table 16-7: Mining Support Equipment

Unit	Function	Maximum Units
Motor grader (5.5 m blade)	Haul road maintenance	5
Motor grader (4.9 m blade)	Haul road/site road maintenance	1
Water/gravel truck	Haul road maintenance	3
Track dozer (450 kW)	WSF maintenance, construction support	3
Track dozer (325 kW)	Pit maintenance, shovel support, snow clearing, site prep, construction	3
Wheel loader (12 m^3)	Pit maintenance, shovel support, snow clearing, site prep, construction	2
Wheel loader (7 m^3)	Pit maintenance, shovel support, snow clearing, site prep, construction	1
Hydraulic excavator (4.5 m^3)	Ore cleaning, prep for ore loading, pit support	1
Hydraulic excavator (3.0 m^3)	Ditching, construction activities	1
Fuel and lube truck	Fuel/lube support of excavators, wheel loader, drills and support equipment	2
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
Emergency response vehicle	Mine safety and first aid	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 RECOVERY METHODS

17.1 Overall Process Design

Gold and silver values from the types of mineralisation present in the Blackwater deposit are anticipated to be largely recoverable by a combination of gravity processes and conventional cyanidation. The preferred process flowsheet was derived from testwork results and was tailored to support a robust production profile over the life of mine. The unit operations that have been included in the selected process are well proven at the commercial scale, and typical in the industry.

The process plant will be constructed in distinct phases:

- Phase 1 (6Mtpa) operating for Years 1 to 5, year 5 will process at the higher Phase 2 throughput for the latter half of the year
- Phase 2 (12Mtpa) operating for Years 6 to 10, year 10 will process at the higher Phase 3 throughput for the latter half of the year.
- Phase 3 and 4 (20Mtpa) operating for Years 11 to 22

All phases employ a similar flowsheet, with appropriate selection of design criteria to cater for ore variability as follows:

- Hardness increasing with depth over time;
- Contained gold and silver grades peak in Year 7;
- Limited areas of high cyanide soluble copper concentrations in mineralized material.

The Phase 1 process design was developed to a feasibility level of engineering definition while the Phases 2 and 3 process designs were developed to a pre-feasibility level.

17.2 Phase 1 Mill Process Plant Description

The flowsheet developed in the 2020 PFS was retained for Phase 1 with minor adjustments. The crusher circuit equipment selection was refined for a cost-effective expansion to 12 Mtpa, and a carbon-in-leach (CIL) circuit selected for a simpler, lower capital cost alternative to the previously proposed carbon-in-pulp (CIP) circuit. The process design comprises the following circuits:

- Three stage crushing of ROM material;
- Grinding circuit comprising a closed-circuit ball mill with cyclone classification and gravity concentration for a portion of the mill discharge

- Intensive cyanidation of gravity concentrate
- Pre-aeration of the cyclone overflow using oxygen
- Hybrid leach-CIL circuit with 3 stages of leaching and seven stages of CIL adsorption
- Acid washing of loaded carbon and Anglo-American Research Laboratory (AARL) type elution followed by electrowinning and smelting to produce doré
- Carbon regeneration by rotary kiln
- Cyanide destruction of tailings using SO_2/O_2
- Effluent discharged to the TSF

17.2.1 Plant Design Criteria

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

Table 17-1: Phase 1 Key Plant Process Design Criteria

Design Parameter	Units	Design Values
Plant throughput	tpd	16,438
Gold head grade – design	g/t Au	1.58
Silver head grade – design	g/t Ag	10.1
Crushing plant availability	%	70
Mill availability	%	92
Bond Crusher Work Index (CWi)	kWh/t	22.2
Bond Ball Mill Work Index (BWi)	kWh/t	18.7
Bond Abrasion Index (Ai) - average	g	0.134
JK drop weight parameter Axb		31.5
Primary crusher type		Gyratory
Material specific gravity	t/m³	2.70
Ball mill dimensions		7.3m dia x 12.5m EGL
Ball mill installed power	MW	14
Circulating load - nominal	%	400
Classification cyclone overflow density	% w/w	45
Primary Grind size (P_{80})	µm	150
Pre-aeration tanks	#	1
Leach and CIL tanks	#	3 + 7
Leach + CIL residence time	h	24
Elution carbon batch size	t	12
Detox residence time	min	90
Detox CN_{WAD} feed to circuit	mg/L CN_{WAD}	300
Detox CN_{WAD} discharge target - Design	mg/L CN_{WAD}	10

Reagents used in the process and their specific consumption rates are shown in Table 17-2.

Table 17-2: Phase 1 Reagent and Grinding Media Consumption per Tonne Milled

Reagent Description	Units	Consumption
Quick lime (CaO)	kg/t	2.6
NaCN	kg/t	0.60
Oxygen	kg/t	1.58
Activated carbon	g/t	35
NaOH	kg/t	0.15
HCl	kg/t	0.03
SO_2	kg/t	1.5
$CuSO_4$	kg/t	0.08
Ball mill grinding media	kg/t	0.71

17.2.2 Primary Crushing and Stockpiling

ROM material will be hauled to the primary gyratory crusher where a front-end wheel loader will supplement the direct-tip feed from the ROM stockpile to maintain a continuous crushing operation. Mine operations will retrieve any oversize and either use a mobile rock breaker to reduce the lump size or return oversize to the low-grade stockpile.

The crushing plant will produce a fine ore sized to a P_{80} of 8 mm, at a throughput of 978 t/h, and an availability of 70%.

The primary crusher dump pocket will feed the primary gyratory crusher, which will feature a closed size setting of 100 mm, at the feed end of the crushing circuit. The product will be drawn out onto the secondary screen feed conveyor by way of the primary crusher belt feeder. Primary-crushed material will be conveyed to the secondary screen, and this conveyor will be fitted with a tramp magnet to remove steel trash from the primary-crushed material prior to feeding to the secondary/tertiary crushing circuit.

Secondary screen oversize will feed directly into the secondary cone crusher. The secondary and tertiary cone crusher products along with the secondary screen undersize will be conveyed to the tertiary screens. Tertiary screen oversize will feed to two surge bins ahead of two tertiary cone crushers. The secondary and tertiary crushers will have closed size settings of 35 and 15 mm, respectively. The tertiary crushers will be choke fed from the surge bins using feeders. Tertiary screen undersize will report to the tertiary screen undersize conveyor.

The crushed product will be conveyed to a covered crushed ore stockpile (32,100 t total capacity; 8,190 t live capacity). Two reclaim belt feeders will provide two live pockets. The crushed material will feed the mill and will be conveyed from the two fine ore reclaim feeders to the mill feed chute.

The crushed material will be processed through a dual pinion ball mill in closed circuit with cyclones producing a final product with a P_{80} of 150 µm. The mill will be designed for a nominal solids throughput of 16,438 tpd and will be able to process 745 t/h at 92% availability. The installed ball mill power will be 14 MW and the mill dimensions will be 7.3 x 12.5 m (internal diameter x effective grinding length (EGL)) with a circulating load of 400%.

17.2.3 Grinding Circuit

Mill slurry discharge will overflow onto a rubber-lined trommel screen with trommel oversize discharging to a bunker for regular collection and disposal. The trommel undersize will gravitate to the ball mill cyclone pump box where the slurry will be diluted with process water and pumped with a cyclone feed pump to the cyclone cluster. A density meter will monitor and control the amount of process water required to produce a target density to the cyclones.

The cyclone cluster will produce a fine ground overflow product of P_{80} 150 µm, which will be sampled and gravitated to the vibrating trash screen. Cyclone underflow will return to the ball mill directly. Oversize debris will be removed and will fall to a trash bin at ground level. The minus 0.8 mm trash screen underflow will be pumped to the leach-CIL circuit.

17.2.4 Gravity and Intensive Leaching

A portion of the ball mill discharge will feed to two parallel gravity concentrator trains, each sized for 50% duty. The gravity circuit splitter box will provide the feed slurry to two gravity concentrator trains. Each train will consist of a scalping screen and a gravity concentrator.

Oversize from the scalping screen will gravitate to the ball mill cyclone pump box, while the undersize will feed the gravity concentrator. Tailings from the concentrators will be transferred back to the ball mill circuit and the concentrate will gravitate to the intensive cyanidation circuit at ground level.

The intensive cyanidation circuit will receive gravity gold concentrate on a batchwise basis for treatment in an intensive leach reactor. Gold concentrate will be leached in relatively high-strength cyanide solution, in batch fashion, once every 24 hours. The gold-containing pregnant solution will be drained from the reactor and pumped to a dedicated gravity eluate tank in the gold room. The intensive leach tailings will transfer back to the ball mill cyclone pump box in the grinding circuit.

17.2.5 Leaching and Adsorption

Trash screen underflow slurry will be pumped to the leach-adsorption circuit that will consist of one pre-aeration tank, three leach tanks and seven CIL adsorption tanks. The pre-aeration, leach and CIL tanks will be 2,935 m^3 each; the leach and adsorption circuit residence time will be 24 hours at 45% w/w solids.

Barren carbon will enter the adsorption circuit at CIL tank 7. The carbon will advance countercurrent to the main slurry flow during periodic transfers using carbon advance/transfer pumps from a downstream to an upstream tank. Carbon concentrations of 10–15 g/L will be required in all tanks. Carbon will be retained in the upstream tank by an intertank screen. The countercurrent carbon transfer process will be repeated until the carbon becomes loaded and reaches CIL tank 1. Then a recessed impeller pump will transfer slurry and carbon to a loaded carbon recovery screen. The loaded carbon will be washed with water and released to the acid wash column located inside the main plant building, in the desorption area. The slurry will return to CIL tank 1.

Following elution of the loaded carbon and thermal regeneration, the barren carbon will be screened and pumped to CIL tank 7. Fine carbon will be discarded to the tailings pump box.

Tailings slurry from adsorption tank 7 will flow by gravity to cyanide detoxification tank 1 in the cyanide detoxification circuit.

17.2.6 Cyanide Detoxification and Tailings Disposal

Slurry exiting CIL tank 7 will gravitate to two 1,064 m^3 cyanide detoxification tanks that will operate in parallel, which are designed based on the conventional O_2/SO_2 process using elemental sulphur and a $CuSO_4$ solution. The average slurry residence time at 745 t/h of fresh feed will be 1.5 hours.

The detoxified slurry stream will gravitate to the vibrating carbon safety screen to recover any carbon in the event of damage, wear, or other issues with the CIL tank 7 intertank screen. Recovered carbon will collect in a bin that can be manually transferred for re-use or disposal. Tailings discharge from the carbon safety screen will gravitate to the tailings pump box, from where it will flow by gravity through a single pipeline to the TSF. Supernatant water collected from the TSF will be returned by pump to the process water tank.

17.2.7 Lime Neutralization System

A lime neutralization system for run-off will be installed for the low-grade ore stockpile and will be neutralized in the processing plant through lime addition prior to discharge to the TSF.

Stockpile run-off water will be pumped to two agitated neutralization tanks operating in series. Lime will be added to the first neutralization tank in the form of calcium hydroxide slurry until pH 10.0 is reached. Neutralized water will overflow into the second neutralization tank and will be subsequently pumped to the final tailings pump box. Final tailings will flow by gravity to the TSF. Water that does not meet the pH criteria for discharge will be automatically recirculated within the neutralization tanks. The pH of influent and effluent will be measured using pH probes and this information will be used to automate dosing of lime slurry and discharge of effluent to the tailings pump box.

17.2.8 Carbon Acid Wash, Elution and Regeneration Circuit

The acid wash, elution and regeneration circuit is designed to treat 12 batches of carbon per week, with a carbon batch size of 12 t.

17.2.8.1 Carbon Acid Wash

Prior to the gold stripping stage, loaded carbon will be treated with a weak hydrochloric acid solution to remove calcium, magnesium, and other salt deposits that could render the elution less efficient, or become baked on in subsequent steps and ultimately foul the carbon.

Loaded carbon from the loaded carbon recovery screen will flow by gravity to the acid wash column. Entrained water will drain from the column and the column will refill from the bottom up with the hydrochloric acid solution. Once the column is filled with acid, it will be left to soak, after which the spent acid will be rinsed from the carbon and discarded to the tailings pump box.

The acid-washed carbon will be hydraulically transferred to the elution column for gold stripping.

17.2.8.2 Gold Stripping (Elution)

The gold stripping (elution) circuit will use the AARL process.

The elution sequence will commence with the injection of a set volume of water into the bottom of the elution column, along with the simultaneous injection of cyanide and sodium hydroxide solution to achieve a weak NaOH and weak NaCN solution. Once the prescribed volume has been added, the pre-soak period commences. During the pre-soak, the caustic/cyanide solution will be circulated through the column and the elution heater until the working temperature is achieved.

Upon completion of the pre–soak period, additional water will be pumped through the recovery heat exchanger and elution heating system, then through the elution column a rate of 2.0 bed volumes (BV)/h. At this stage, the temperature of the strip solution passing through the column will increase to 120°C and the gold will be stripped off the loaded carbon.

The strip solution will flow up and out of the top of the column, passing through the heat exchanger via the elution discharge strainers and on to the pregnant solution tanks.

After a set volume of solution has passed through the column, a cool down step will be initiated whereby the heating system is bypassed. Upon completion of the cool down sequence, the carbon will be hydraulically transferred to the kiln feed hopper via a stripped carbon dewatering screen.

17.2.8.3 Carbon Reactivation

Carbon will be reactivated in a rotary kiln. Dewatered barren carbon from the stripping circuit will be held in a kiln feed hopper. A screw feeder meters the carbon into the reactivation kiln, where it will be heated to 750°C in an atmosphere of superheated steam to restore the activity of the carbon.

Carbon discharging from the kiln will be quenched in water and pumped over a carbon sizing screen to remove undersized carbon fragments. The screen oversize will be pumped into CIL Tank 7, while the slurry of quench water and fine carbon will report to the tailings pump box via the carbon safety screen for disposal in the TSF.

As carbon will be lost by attrition, new carbon will be added to the circuit using the carbon quench tank. The new carbon will then be transferred along with the regenerated carbon to feed the carbon sizing screen.

17.2.9 Electrowinning and Smelting

Gold will be recovered from the pregnant solution by electrowinning and smelting to produce doré bars. During Phase 1, the gold room will be operated only during the dayshift.

The pregnant solution will be pumped through electrowinning cells with stainless steel mesh cathodes. Gold will be deposited on the cathodes and the resulting barren solution will be pumped to the leach circuit.

The design includes five electrowinning cells; one cell will be dedicated to the intensive cyanidation circuit and the other four cells to the elution circuit.

The electrowinning cell dedicated to the intensive cyanidation circuit will be fed leach solution via a pump from the gravity concentrate eluate tank. Solution will be pumped to the electrowinning cell and then gravitate back into the gravity eluate tank in a closed loop until the target concentration of gold in electrowinning tails is reached. The duration of this cycle will vary with the quantity of gold recovered by gravity but is projected to be less than 24 hours.

The four electrowinning cells dedicated to the elution circuit will operate in a closed loop with one of two pregnant solution tanks. The solution will be pumped through the electrowinning cells and then gravitate back to the pregnant solution tanks. The duration of this cycle will be about nine hours.

The gold-rich sludge will be washed off the steel cathodes in the electrowinning cells using high-pressure spray water and will gravitate to the gold sludge pump box. Sludge will be dewatered in a filter press and then transported manually using trays to the drying oven.

Dried sludge will be removed from the oven the following day and combined with fluxes in a flux mixer before reporting to the smelt furnace. Once all the mixture has been added to the furnace and enough time has elapsed for the material to fully melt, the slag will be poured into a conical slag pot. The liquid metal will then be poured into either 10 or 30 kg moulds on a mould tray. Cooled doré will then be cleaned, weighed, and stamped. The 30 kg bars will be placed in a vault to await shipment to a refinery.

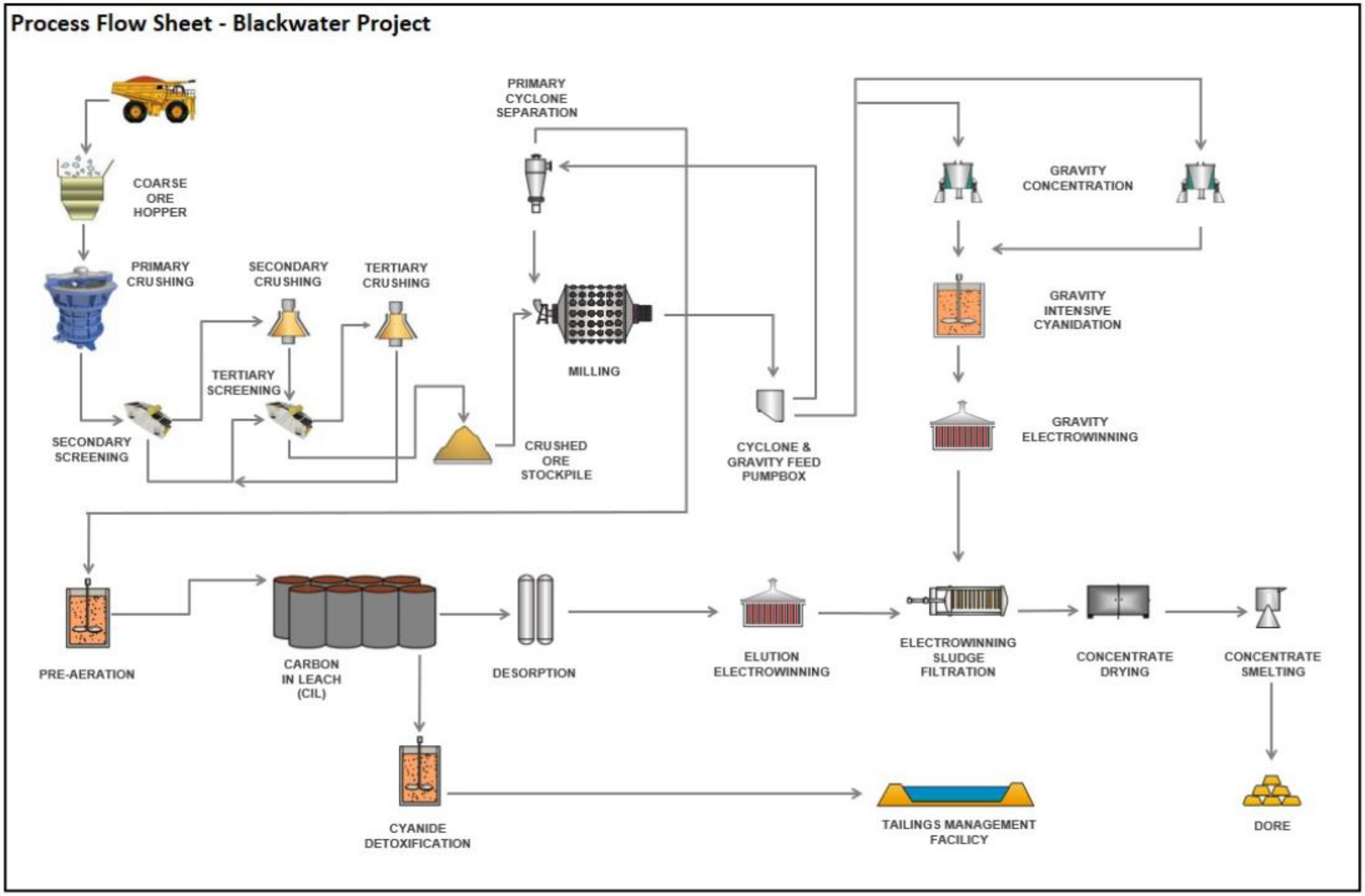
Dust collection will be provided in the gold room for smelting. Extraction fans are planned for the kiln, electrowinning cells, drying oven, and smelting-furnace off gasses.

The electrowinning and smelting process takes place within a secure and supervised gold room.

17.2.10 Flowsheet and Layout Drawings

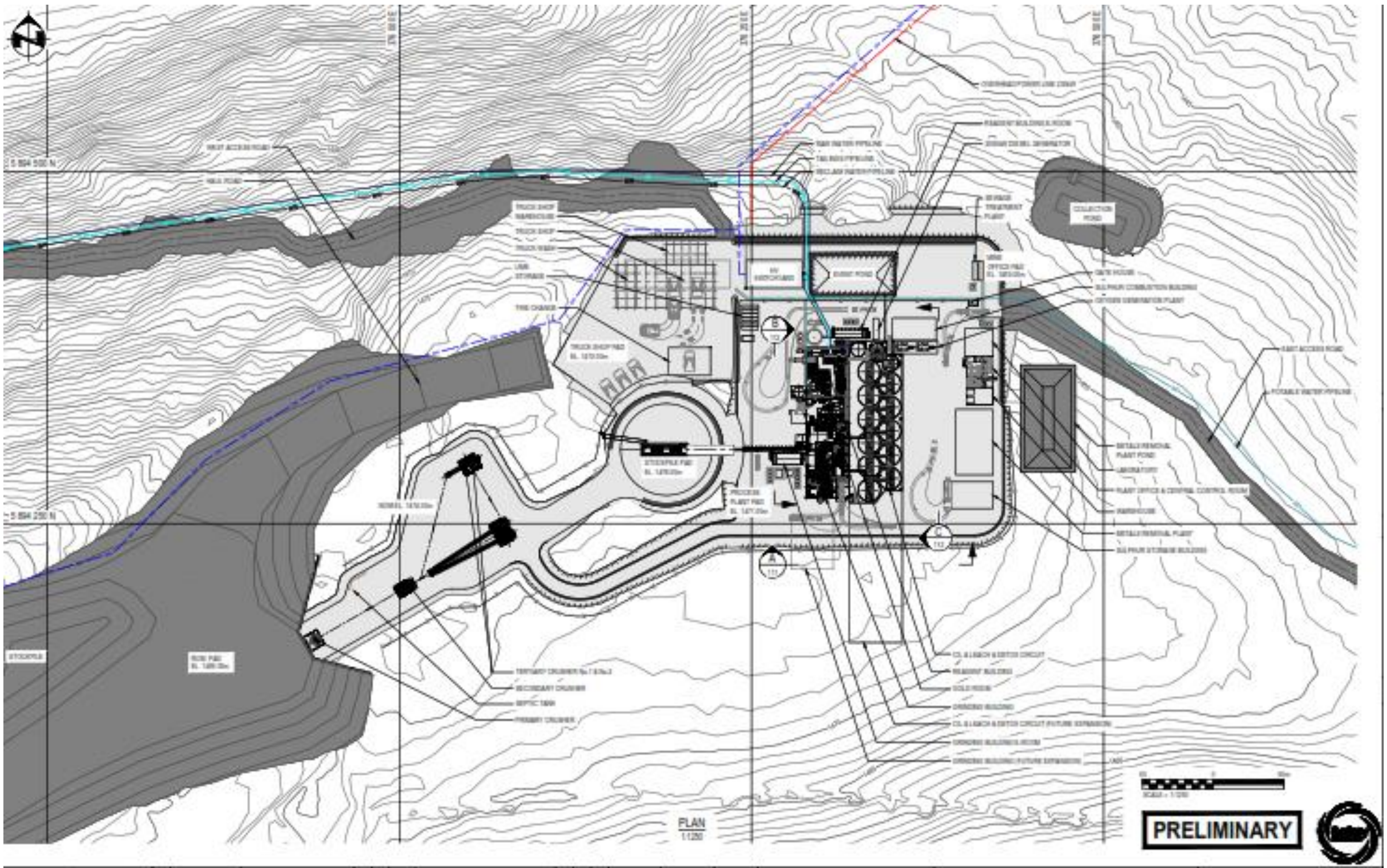
An overall process flow diagram showing the unit operations in the selected process flowsheet is presented in Figure 17-1.

Figure 17-1: Blackwater Flowsheet – Phase 1



Source: Ausenco, 2021

Figure 17-2: Blackwater Plant Layout – Phase 1



Source: Ausenco, 2021.

17.3 Plant Expansion Concept

The plant is designed to accommodate the planned future expansions, and Project development of the project will occur in three Phases. An overview of the additions made in each phase are provided below.

17.3.1 Phase 2 (Years 6-10)

The Phase 2 expansion concept is summarised as follows:

- Additional 6.0 Mtpa treated (12 Mtpa total)
- Additional milling, leaching, adsorption, elution, and detox capacity added
- Uses Phase 1 gold room

Throughput will increase from 6 Mtpa to 12 Mtpa, using the Phase 1 crushing circuit. To account for this, larger motors will be added to the existing conveyors and the belt feeders on the primary and tertiary crushers will be replaced by larger ones. The closed side settings will be increased from 100 to 112 mm on the primary crusher, from 35 to 60 mm on the secondary crusher and from 15 to 25 mm on the tertiary crushers. The crushing circuit P_{80} will increase from 8 to 15 mm. An additional 16MW ball mill (7.9 m internal diameter x 12.6m EGL), cyclone pack, gravity and intensive leaching circuit will be added in series with the Phase 1 ball mill to maintain the target 150 µm product size (P_{80}) for feeding to the leach-CIL circuit.

Phase 2 will include installation of additional leaching, CIL and detox capacity. A new, 14 t elution, desorption, regeneration and electrowinning circuit will be required in Phase 2 to cater for the increase in total contained gold and silver content in the mill feed, which will peak in Year 7. Carbon concentration in the CIL will remain unchanged from Phase 1. The two acid wash, elution, and electrowinning circuits will treat a total batch size of 26 tpd of carbon and treat an average of 10 batches of carbon per week. Use of the same gold room from Phase 1 will be possible through increased equipment utilization (smelting on two shifts). Detox tailings will be directed via two new detox tanks and a new pipeline to the existing TSF.

17.3.2 Phase 3 and 4 (Years 11-22)

The Phase 3 expansion concept is summarised as follows:

- New process line consisting of crushing, grinding, leaching, adsorption and detox circuits with a capacity of 8 Mtpa.
- Utilizes Phase 1 and 2 acid wash, elution, electrowinning and gold room facilities Combined throughput will increase from 12 Mtpa to 20 Mtpa.

The new circuit will consist of three stage crushing using a jaw crusher for the primary duty. The secondary and tertiary crushing equipment will be the same as Phases 1 and 2 to ensure commonality of spares.

Downstream of the Phase 3 crushing circuit, a new grinding circuit (20MW ball mill (8.2 m internal diameter x 12.8 m EGL), cyclone pack, gravity and intensive leaching circuit) will be installed, together with a new cyanide leaching, adsorption and detox circuit. Design criteria for the Phase 3 expansion will largely mirror that of Phases 1 and 2 with respect to the crushing and grinding circuits.

Due to the lower gold and silver feed grades over the life of mine, Phase 3 is only expected to require increased capacity for the leaching, adsorption, and detox circuits, with no additional elution capacity required. The installed Phase 2 capacity for carbon stripping and electrowinning is anticipated to be sufficient for the Phase 3 expansion so no additional expansion of the gold room is required. Detox tailings will be directed via a new pipeline to the TSF.

Phase 4 is a continuation of production throughput as Phase 3, with no expansion required.

17.4 Reagent Handling and Storage

Each set of compatible reagent mixing and storage systems will be located within containment areas to prevent incompatible reagents from mixing. Storage tanks will be equipped with level indicators, instrumentation, and alarms to ensure spills do not occur during normal operation. Appropriate ventilation, fire and safety protection, eyewash stations, and Safety Data Sheet stations will be located throughout the facilities. Sumps and sump pumps will be provided for spillage control. In general reagent makeup is expected to be completed on day shift in Phase 1 and bulk reagents will be stored in 'day' tanks. In Phases 2 and 3, reagents will be made up on both day and night shifts, and bulk reagent storage tanks will provide capacity only for the one shift.

17.4.1 Quick Lime

Quick lime will be delivered to site in bulk by truck and transferred by blower into the lime silo (5.4 day capacity), nine days of bagged lime will be kept in the warehouse. The quicklime will be drawn out by a screw feeder into a detention slaker where water will be added at a controlled rate. Hydrated lime slurry will transfer to the lime storage tank after a screen removes any grit. Two dedicated pumps will transfer lime throughout the plant.

17.4.2 Sodium Cyanide (NaCN)

Sodium cyanide in Phase 1 will be delivered to site as briquettes in bulk boxes. Cyanide safety systems will be in place including area security and monitoring and personal gas detectors. A bag splitter fitted with dust extraction will directly connect to the sodium cyanide mixing tank. In Phase 2 NaCN will be delivered in ISOtainers that will be connected directly to the mixing tank.

After the mixing period is complete, cyanide solution will be transferred to the cyanide storage tank using a transfer pump. Sodium cyanide will be delivered to the leach circuit, intensive cyanidation, and elution circuits with dedicated dosing pumps.

17.4.3 Sodium Hydroxide (NaOH)

Sodium hydroxide (caustic soda) will be delivered in totes as concentrated solution at nominally 50% NaOH by volume. Dosing pumps will be connected to the tote to automatically deliver the reagent to the required locations—elution circuit and eluate tanks—to ensure the dosing requirements are met.

17.4.4 Hydrochloric Acid (HCl)

Hydrochloric acid will be delivered in totes as concentrate solution at nominally 33% HCl by volume. Hydrochloric acid will be delivered to the acid wash circuit using the hydrochloric acid dosing pump connected directly to the tote.

17.4.5 Copper Sulphate Pentahydrate

Copper sulphate will be delivered to site as powder in bulk bags. A bag splitter will be used, which will be directly connected to the copper sulphate mixing tank. Dosing pumps will deliver copper sulphate at 20% by volume to the cyanide detoxification circuit.

17.4.6 Sulphur Dioxide

Elemental sulphur powder will be delivered in bulk bags and off-loaded with hydrated lime powder into two hoppers feeding a sulphur melting tank. Molten sulphur will be pumped from a storage tank to the sulphur furnace to produce SO_2 gas. The gas will be piped to the point of use in the detox circuit for injection into the tanks. Safety systems will include SO_2 area and personnel gas monitors.

17.4.7 Oxygen

Oxygen required for pre-aeration, leaching and cyanide destruction will be generated on site in a vacuum swing adsorption plant. Oxygen gas will be piped to the various points of use in the plant. A back-up liquid oxygen storage facility will be included.

17.4.8 Activated Carbon

Activated carbon will be delivered in solid granular form in bulk bags. When required, the fresh carbon will be introduced to the carbon quench tank, or directly to the final CIL tank.

17.4.9 Flocculant

Liquid flocculant will be delivered to site in drums and stored in the reagent building. Flocculant will be dosed into the intensive cyanidation circuit as required.

17.4.10 Leach Aid

Powdered leach aid (a chemical oxidant used to improve leach recoveries and accelerate the leaching process) will be delivered in bags and added to the intensive cyanidation circuit as required.

17.4.11 Sulfamic Acid

Liquid sulfamic acid will be delivered in drums and utilised to clean heat exchangers in the elution circuit as required.

17.4.12 Gold Room Smelting Fluxes

Borax, silica sand, sodium nitrate, and sodium carbonate are delivered as solid crystals/pellets in bags or plastic containers and stored in the warehouse until required.

17.5 Services and Utilities

Process/Instrument Air

High-pressure air at 750 kPa will be produced by compressors to meet plant requirements. The high-pressure air supply will be dried and used to satisfy both plant air and instrument air demand. Dried air will be distributed via the air receivers located throughout the plant.

17.5.1 Water Supply

17.5.1.1 Raw Water Distribution System

Raw water will be supplied from the water management pond and depressurisation wells into a raw water storage tank. Raw water will be used for all purposes requiring clean water with low dissolved solids and low salt content, primarily as follows:

- Gland water for pumps
- Dust suppression in the crusher area
- Reagent make-up
- Elution circuit make-up
- Fire water for use in the sprinkler and hydrant system
- Cooling water for mill motors and mill lubrication systems (open loop)
- Gravity concentrator fluidisation

The average raw water demand for the Phase 1 process plant is estimated to be 3,723 m^3/day.

17.5.1.2 Potable Water Distribution System

Potable water for plant use will be supplied from a potable water treatment plant that will be installed at the accommodations camp. A dedicated tank and pumps distribute potable water to the plant for use in safety showers and eye wash stations, and to adjacent buildings. This will be expanded in the subsequent phases.

17.5.1.3 Process Water Supply System

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. This will be expanded in the subsequent phases.

17.5.1.4 Gland Water

One dedicated gland water pump will be fed from the raw water tank to supply gland water to all slurry pump applications in the plant. This will be expanded in the subsequent phases.

17.5.2 Projected Energy Requirements

The installed power for the Phase 1 process plant is estimated to be 32.5MW, and power consumption is 32.3 kWh/t of material treated for the processing plant. Section 18 details the power consumption for the life of mine and the surface infrastructure.

18 PROJECT INFRASTRUCTURE

18.1 Project Layout

The overall Project facilities and major infrastructure cover the mine site area, TSF, stockpiles, camp site, main access road, and site wide water management systems. An overall layout plan is presented in Figure 18-1.

18.2 Roads and Logistics

18.2.1 Roads

18.2.1.1 Access Road

The proposed mine access road will be a new road system that will connect the Kluskus-Ootsa FSR with the proposed Blackwater mine site (Figure 18-2). The mine access road will originate at 124.130 km on the Kluskus FSR and will continue approximately 13.8 km south to a termination point near the planned mine plant site. The road will be used by heavy traffic during the construction of the mine and consequently was designed for year-round all-weather access.

The proposed road passes through previously-logged forestry cut blocks; however a significant portion of the route lies within standing mature timber.

The access road will be 5 m wide, single lane, have a design speed of 40 km/hr, and will incorporate five bridges of varying lengths and types. The road design includes ditching to control erosion as well as culverts and cross drains as required. The road is designed to meet the forest industry standards specified in the Ministry of Forests, Lands, Natural Resource Operations and Rural Development Engineering Manual.

The five proposed bridges were designed as permanent single lane bridge with L100 load rating.

The mine access road is planned for construction in the initial stage of Phase 1.

18.2.1.2 Onsite Roads

Onsite roads of approximately 8 km in length will be required to provide access to the plant, truck-shop, accommodation and explosives store. These roads will be approximately 10 m wide to allow two-way, light vehicle traffic and wider where mine trucks will travel. A 15 km access road will also be needed to the pumping station that is planned to be installed on Tatelkuz Lake.

18.2.1.3 Kluskus Forest Service Road

The Kluskus FSR will be the primary means of access to the mine access road. As with most FSRs in Northern BC, heavy traffic will be restricted during the spring breakup, which is typically mid-March to the end of May. During this time load limitations are usually in place for such roads. A 22.3 km section of the Kluskus FSR will require road improvements to ensure future year-round access needs are met.

The improvements required for the Kluskus FSR include realignment of curves for improved safety and efficiency, upgrades of existing roadside ditches, and placing new running surface material. The Kluskus FSR improvements are planned for the initial stage of Phase 1.

18.2.2 Logistics

18.2.2.1 Airstrip

An airstrip will be constructed approximately 13km from the mine site for the transportation of personnel to and from the mine site during the Project operational phase. The airstrip will be sized for King Air 200 class aircraft. Facilities will include an access road, airstrip, and taxiway.

The airstrip was designed with a runway length of 1,000 m, a width of 23 m, and will incorporate a gravel surface for all areas including taxiways and aprons. The airstrip as designed will not have any lighting, instrumentation, navigation aids, nor passenger receiving buildings. Space is available should installation of any of these items be required after initial construction.

The proposed airstrip access road will consist of a series of existing 5 m wide single lane logging roads with pullouts. The road currently has minimal use and will require upgrade to provide year-round access to the airstrip such as brushing, clearing ditches, and application of surfacing material.

Construction of the airstrip and access road is planned for during Phase 1.

18.2.2.2 Bussing

Bus transportation costs were estimated for the construction phase as well as the operational phase, assuming bussing from Prince George and Vanderhoof to the planned mine site. The price to determine the cost per person/per trip was defined using quotes received from local bussing companies.

Figure 18-1: Project Site General Arrangement

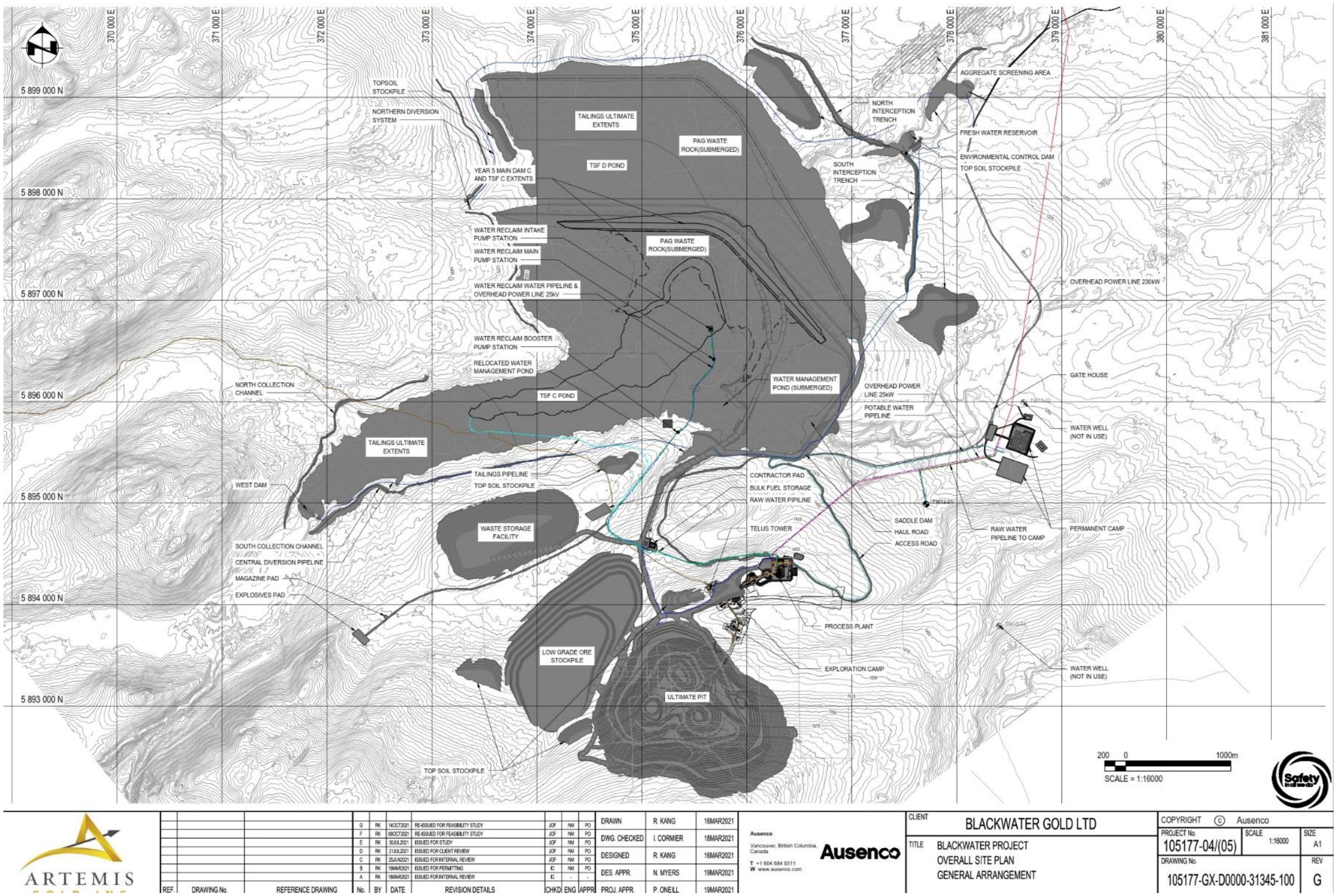
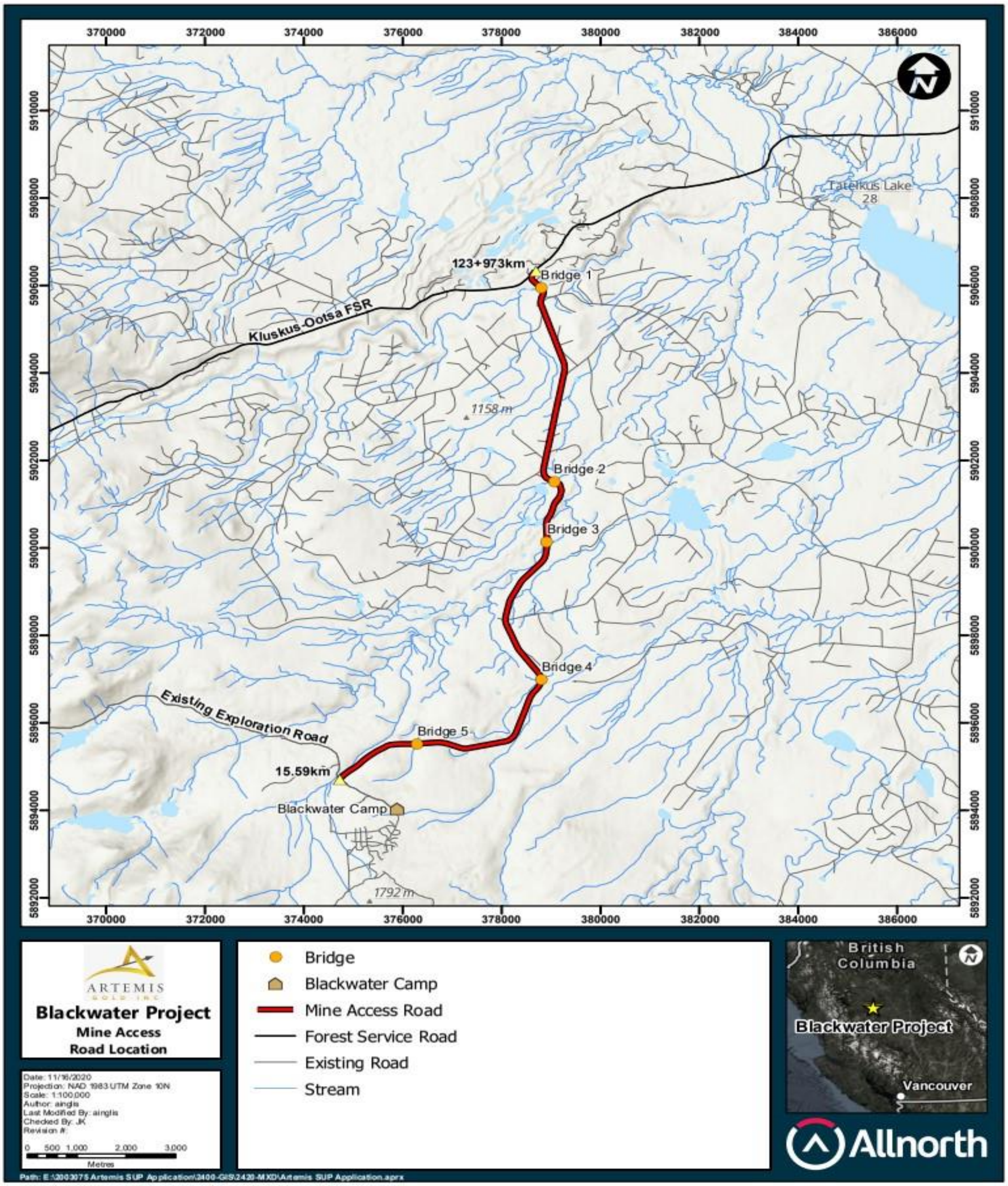


Figure 18-2: Blackwater Mine Access Road



18.3 Geotechnical Investigations

18.3.1 General

KP completed an evaluation of the geotechnical and hydrogeological conditions of the Blackwater area through extensive site investigation programs in 2012 and 2013 (KP, 2013a; KP, 2013b; KP, 2013c; KP, 2013d). These programs supported engineering studies for the tailings and water management systems, plant site, and other mine site infrastructure proposed in the Davidson Creek watershed and for the open pit design on the slopes of Mt. Davidson for the 2013 FS. Drillhole, ground geophysics, and test pit locations were adjusted as the programs progressed and site conditions became better understood. These site investigation programs were supplemented with site investigation work completed in 2019 (KP, 2021e) in the vicinity of the proposed TSF and mine infrastructure areas. Additional site investigation work was completed in 2020-2021 (KP, 2021f) to further investigate the geotechnical and hydrogeological conditions at the planned sites for Main Dam C, the Interim Environmental Control Dam (IECD), the FWR, the proposed plant site location, and in the vicinity of potential overburden and esker borrow source areas.

18.3.2 Site Investigations

The 2012, 2013, and 2019 site investigation programs included:

- Excavating 336 test pits to investigate the near-surface material characteristics and foundation conditions;
- Drilling 36 geotechnical drillholes using overburden drilling excentric (ODEX) drilling techniques with standard penetration tests in the surficial materials and diamond drill coring (HQ3, 61.1 mm core diameter) with packer permeability tests in bedrock;
- Drilling 74 geotechnical drillholes using sonic drilling techniques;
- Drilling 16 geotechnical drillholes in the open pit using geomechanical logging techniques;
- Completing seismic cone penetration tests at five locations to investigate the in-situ condition of glaciolacustrine deposits;
- In-situ packer hydraulic conductivity testing (lugeon single packer) permeability tests during rock mass drilling in ODEX drillholes and geomechanical open pit investigations;
- Airlift hydraulic conductivity testing in overburden;
- Installing 47 standpipe piezometers and 15 vibrating-wire piezometers in select geotechnical drillholes to investigate static groundwater levels and evaluate the rock mass permeability in the TSF area;
- Installing multiple vibrating wire piezometers in 12 observation holes and 12 geomechanical drillholes in the open pit;
- Installing 28 monitoring wells developed for long-term groundwater quality monitoring;
- Installing five closed-bottom polyvinyl chloride pipes for future downhole seismic surveys within the foundation areas of the TSF dams;

- Conducting 34 response tests in the screened completion zone of standpipe piezometers and monitoring wells;
- Conducting downhole seismic surveys at five plant-site drillholes;
- Laboratory testing of surficial materials and rock core samples to determine geotechnical material parameters for the different types of materials encountered;
- Rock strength laboratory testing on selected representative core samples to evaluate strength properties and to verify rock mass classification;
- Completing 35.3 km of seismic refraction surveys to develop profiles of subsurface bedrock and the saturated water table elevations;
- Completing 5.2 km of high-resolution resistivity and induced polarization profiling to develop bedrock and the saturated water table profiles;
- Two pumping wells to assess hydrogeology in the open pit.

The 2020-2021 site investigation program included:

- Excavation of 72 test pits;
- 23 geotechnical sonic drillholes with four additional twin holes for in-situ testing;
- Installation of 12 standpipe piezometers
- Installation of 30 vibrating wire piezometers at 14 locations;
- 12 downhole seismic test locations
- Three seismic cone penetration test probe locations with pore pressure dissipation testing;
- Two pressure meter test locations;
- Off-site laboratory testing of 333 sonic drillhole grab samples and 48 test pit grab samples to support engineering characterization. Laboratory testing included particle size analysis, determination of Atterberg limits, natural moisture content testing, specific gravity tests, and compaction testing.

18.3.3 Site Characterization

The geological, geotechnical, and hydrogeological information collected from the site investigation programs completed between 2012 and 2021 was analyzed and compiled into a stand-alone Dam Site Characterization Report (KP, 2021g). The site characterization combines the site data with information available from other sources and presents the site geological model, site wide geotechnical and hydrogeological conditions, and the interpreted foundation conditions at the TSF and water management structures.

The surficial deposits in the Project area are from the Fraser glaciation, the last period of ice sheet glaciation in BC. The pattern of ice-marginal and subglacial meltwater channels indicates that areas of higher elevation in the vicinity of the mine site became ice-free before valley floors and other low-lying areas. Glacial ice appears to have stagnated in the Davidson Creek valley during late deglaciation producing ice-stagnation landforms such as kettles and kames. The meltwater corridors evolved over the short period during which they were active. Downward-stepping terraces within some meltwater corridors show that active channel floors were progressively lowered by fluvial erosion as the ice melted. The modern drainage system became established as soon as the area was fully deglaciated. Since then, there has been little geomorphic change in the study area.

Massive lodgement glacial tills were deposited at the base of the ice sheet. Ablation tills were also deposited locally as the ice sheet retreated. Sediments deposited during de-glaciation of the area include glaciofluvial and glaciolacustrine sediments. Glaciofluvial sands and gravels are common in valley bottoms and along the valley flanks, occurring as kames, eskers, and terraces. The stratigraphy of the surficial materials and bedrock from surface downward is as follows:

- Holocene deposits;
- Fraser glaciation deposits, including:
  - Glacial till (includes ablation till, lodgement till, undifferentiated till)
  - Glaciofluvial
  - Glaciolacustrine
- Reworked regolith;
- Bedrock
  - Completely weathered bedrock
  - Highly weathered bedrock
  - Intact bedrock

18.3.4 Foundation Conditions for the TSF and Water Management Structures

Topsoil was encountered over the entire Project area with thickness varying from about 0.1 to 2.0 m and typically comprises wet, dark reddish brown silty sand with a high organic cover. Topsoil may be locally thicker in wetland areas and creek bottoms where investigation work was not practical to perform.

Glacial till was the most dominant surficial material encountered and was grouped into three categories: ablation till, lodgement till, and undifferentiated till. Ablation till was encountered at or near surface and generally comprises well graded gravelly silty sand with trace clay and cobbles. Lodgement till was the most common glacial till material forming the foundation of the major structures and comprises well graded, stiff to very stiff, gravelly sandy silt with trace clay. Undifferentiated till is similar in composition to lodgement till; however, it is generally more poorly graded and occurs in thinner layers.

Glaciofluvial materials encountered in the Project area included channelized deposits (eskers, terraces, and buried channels) and non-channelized deposits (kames). The channelized deposits are well-grade coarse-grained sand and gravels with some cobbles and trace silt. They have the lowest proportion of fines of all surficial materials in the area. Meltwater channels form localized terraces along either side of Davidson Creek that can be up to about 10 to 20 m thick in some areas. Buried glaciofluvial deposits were encountered and inferred to represent potential locations of historical meltwater corridors. An inferred subglacial meltwater corridor was identified that is oriented northeast between the proposed locations of Main Dam C and the IECD, which is generally aligned with the orientation of present-day Davidson Creek. Additional work is required as mine development proceeds to further evaluate the extent and continuity of these potential buried channel deposits.

Glaciolacustrine materials were identified and occur in multiple layers with varying thickness. The glaciolacustrine deposits generally comprise uniformly graded, stiff to very stiff, sandy silt with trace to some clay. The structure encountered ranged from massive to laminated to varved with silt and sand layers ranging from 1 to 20 mm thick. Major glaciolacustrine occurrences were summarized based on the elevation and described in the DSCR and relevant design reports. In-situ testing indicates that the major glaciolacustrine units encountered below Main Dam C are expected to behave sand-like and are strongly dilative at current stress levels. The understanding of the lateral continuity and expected behavior of the major glaciolacustrine deposits is limited by the available drilling and in-situ testing information.

Reworked regolith was less prevalent than the glacial deposits within the surficial material and is generally found at depth below the glacial deposits directly above the weathered bedrock. The material is variable in composition but generally comprises abundant altered bedrock clasts within a well graded and very stiff to hard sand silt with trace to some clay matrix.

Completely weathered bedrock was observed in areas across the site and was thickest in the topographically low areas along Davidson Creek. The unit shows evidence of original rock structure; however, it is reduced to soil-like consistency by the drilling process and is generally recovered as stiff to hard sandy silt with some clay and trace gravel. The upper clayey horizon is highly plastic and lightly over-consolidated and may behave in a contractive manner at large strains.

Bedrock below the soil units and completely weathered bedrock generally includes a highly weathered horizon comprising highly oxidized, very weak to weak, mottled, greenish to purplish grey to reddish brown rock. More than half of the material is recovered as continuous core; however, the rest is reduced to finer material and rock fragments by the drilling process and is generally recovered as silty sand with some gravel and clay. It can be difficult to distinguish between the completely weathered bedrock and highly weathered bedrock units due to the disturbance caused by drilling. Bedrock below the highly weathered horizon becomes more intact. Principal rock types defined for the Project area include felsic tuff, felsic lapilli tuff, volcaniclastic, andesite, felsic intrusive, and sedimentary. The bedrock is categorized as a FAIR quality rock with average rock mass rating ranging from 50 to 58 and as a strong rock with average unconfined compressive strength ranging from 60 to 120 MPa.

Additional work is outlined in Section 26.5 order to progressively refine the site geological model, verify conditions assumed for detailed design of the Stage 1 TSF and water management structures, and collected supplemental information supporting the detailed design of subsequent stages of the TSF (KP, 2021g). The recommendations are broken down into a phased five-year investigation plan to be executed prior to and during the initial construction of the mine (Years -2 and -1) and extending over the first several years of operations (Years 1 to 3).

18.3.5 Foundation Conditions for the Low-Grade Ore and Waste Stockpiles

Subsurface materials in the proposed stockpiles areas were assessed using the geological and geotechnical information collected during the field investigation programs in 2012, 2013, and 2019 (KP, 2021g).

The surficial material in the stockpiles area is predominantly glacial till. Kames and kettle topography are expected locally in the lower lying areas to the north and northwest of the open pit, and glaciofluvial materials may be present along historical meltwater corridors flowing towards the north and along a potential northeast trending subglacial meltwater corridor. Topsoil thickness ranged from 0.1 to 1 m where site investigation work was performed but is expected to be thicker locally in lower lying areas where small ponds (kettles) and organic swamps are mapped. The glacial sequence ranges in thickness from 4 to 75 m with the overburden thickness being generally thinner in higher elevation areas to the south near the proposed open pit and north near the WSF and thicker in the lower lying areas. A glaciolacustrine unit was intercepted within the glacial sequence that will require further investigation to understand the lateral continuity and expected behavior during ore stockpiling. Reworked regolith and weathered bedrock were encountered below the glacial sequence overlying the intact bedrock horizon.

Additional site investigation work recommended within the stockpiles area prior to construction is outlined in Section 26.5, which will be performed to verify that the ground conditions are consistent with design assumptions. The recommended work program will include geotechnical drilling and in-situ testing at select locations within the stockpile footprints, and test pits to further investigate the nature and consistency of the surficial materials in the general vicinity of the proposed stockpiles.

18.3.6 Foundation Conditions for the Plant Site

Subsurface materials at the proposed plant site were assessed using the geological and geotechnical information collected from the 2021 drillhole logs, downhole seismic surveys, and laboratory testing data (KP, 2021p). The proposed plant site area is situated on gently to moderately sloped ground covered by overburden. The overburden depth encountered during drilling at five locations in the plant site area encountered a variable overburden thickness ranging from approximately 23 to 46 m. A geotechnical model was developed to support the foundation and earthwork assessment. Major subsurface layers exhibiting distinct engineering characteristics include the following:

- Topsoil
- Glacial till (upper till, middle till, and lower till) with the potential for thin lenses of glaciolacustrine materials at depth
- Reworked regolith
- Weathered bedrock

The site conditions are favourable to apply shallow foundations for the proposed plant site infrastructure. Topsoil and loose surficial materials shall be removed. Shallow spread footings and mat foundations are suitable for the proposed plant site structures. Mat foundations are recommended for the mill and primary crusher where heavy loads and vibrating forces are expected. The upper till unit is considered the primary bearing layer for shallow foundations. Engineered structural fill will be needed for over-excavated foundation areas and for earthwork structures.

18.4 Borrow Sources

18.4.1 General

Local material borrows will be developed to provide fill materials as required for construction of the TSF and water management facilities. The Zone S and Zone C materials are naturally occurring and will be sourced from local glacial till and glaciofluvial borrow areas. The Zone F, T, D, wearing course and riprap bedding materials will require processing and will be sourced from the local esker borrow areas. Material types and placement specifications are further described in Section 18.5.5.

18.4.2 Esker Deposits

An esker complex was identified in close proximity to the Main Dam C Stage 1 construction area. The esker complex is located on the northwest side of Davidson Creek, immediately downstream of Main Dam C. The eskers override the glacial till plain locally with an estimated aggregate volume of greater than 200,000 m^3. The esker complex is free draining and dry and, in most instances, covered with a veneer of organic matter. Groundwater may be encountered at the glacial till contact; however, the static groundwater levels in the area are relatively consistent with the Davidson Creek channel elevation located approximately 20 m below the glacial till contact. The esker deposit is characterized as a relatively clean, well sorted, free-draining sand and gravel. Cobbles and smaller boulders were encountered during test pit excavation. Laboratory testing on selective test pits has shown that the esker deposit is suitable for use in embankment construction and portions will be suitable for use as aggregate. The area adjacent to the esker complex is relatively flat and should be suitable to establish an aggregate screening area during construction.

A larger esker complex is located further downstream along the planned mine access road adjacent to the location of the FWR. The local esker complex described above is an extension of this larger complex further downstream. This resource was characterized during the 2013 FS (KP, 2014; KP, 2013f) and the volume of available aggregate material was estimated to be in excess of 3 Mm^3. This additional aggregate source is being permitted as an external borrow area and represents a contingency borrow area for Stage 1 TSF construction. This source will likely be used as a construction material borrow area for other Project components, such as the FWR, mine access road, and possibly as a concrete aggregate source. The materials were found to be in compliance, with the exception of the testing for alkali–silica reactivity. The addition of supplementary cementing materials such as fly-ash will mitigate against the potential for alkali–silica reactivity expansion by neutralizing the excessive alkalinity of the cement with silicic acid at the early stage of the cement setting.

18.4.3 Zone S and Zone C Borrows

The dominant geologic material type in proximity to the Stage 1 TSF was found to be glacial till. The till deposits are generally well graded and have low permeabilities. The glacial till deposits are sometimes interbedded with glaciofluvial deposits (high and low energy). Borrow areas were identified for naturally occurring materials that are expected to meet the Zone S and Zone C material gradations.

18.5 Tailings Storage Facility

18.5.1 Objectives and Concept

The principal design objectives for the TSF and associated water management facilities are to protect the regional groundwater and surface water during both operations and in the long-term (after closure) and to achieve effective reclamation at mine closure. The design of the TSF has taken into consideration the following requirements:

- Permanent, secure, and total confinement of all solid waste materials within engineered disposal facilities;
- Control, collection, and removal of free-draining liquids from the waste rock and tailings during operations for recycling as process water to the maximum practicable extent;
- Prevention of ARD and minimization of ML from potentially-reactive tailings and waste rock;
- The inclusion of monitoring features for all aspects of the facility to confirm performance goals are achieved and design criteria and assumptions are met;

- Staged development of the facility over the life of the Project.

The TSF was designed to permanently store tailings, PAG waste rock, and potentially ML NAG waste rock that is generated during operation of the mine (KP, 2021i). The facility was designed to hold 469 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 for severe natural flooding.

The TSF comprises two adjacent sites, TSF C and TSF D. TSF C will be constructed first to provide storage capacity for start-up of the process plant. It was designed to contain tailings for approximately 21 years of mine operations and PAG/ML waste rock generated during the first six years of mining. TSF C will comprise a valley-fill style impoundment formed by construction of three embankments (Main Dam C, the West Dam, and the Saddle Dam) in the upper reaches of the Davidson Creek drainage basin. TSF D will be constructed adjacent to and downstream of TSF C beginning in Year 5, during the Phase 2 expansion, to provide additional storage capacity for PAG/ML waste rock and tailings. TSF D will be formed by construction of one embankment (Main Dam D) to contain PAG/ML waste rock generated between Year 6 and the end of mining and up to two years of tailings beginning in approximately Year 21 when TSF C reaches design capacity, the timing of which depends on a variety of factors.

18.5.2 Site Selection

A tailings alternatives assessment (TAA) for the Blackwater Project was completed in 2015 (ERM, 2015) in response to requests from the BC EAO and CEAA. The study included evaluation of the best available technology (BAT) and best available practices (BAP) 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 BAT/BAP 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, 2012. The assessment demonstrated that thickened slurry tailings with PAG/NAG3 waste rock stored underwater in the Davidson Creek valley is the BAT and preferred alternative for the Project. A list of BAPs for the Project was developed and included additional measures to actively manage the water balance during operations that will significantly enhance physical stability while maintaining best practices for geochemical stability.

The selected alternative from the TAA, including the BAP identified for the Project, formed the basis of the Project design 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 TAA. 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.

18.5.3 TSF Hazard Classification

The Canadian Dam Association Dam Safety Guidelines (CDA, 2013; CDA, 2019) and the Part 10 Guidance Document for the Health, Safety and Reclamation Code for Mines in British Columbia (EMPR, 2016) were used to determine the dam hazard classification and suggested minimum target levels for some design criteria, such as the inflow design flood (IDF) and earthquake design ground motion (EDGM) 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 emergency international best practices for tailings management (GTR, 2020; MCA, 2019). The following target levels were adopted for the design basis of the TSF:

- IDF – the probable maximum flood (PMF);
- EDGM – the 1-in-10,000-year return period seismic event or maximum credible earthquake (MCE), whichever is greater.

18.5.4 Tailings Characteristics

Laboratory testing programs were conducted in 2013 (KP, 2013e) and 2021 (KP, 2021i) to determine the geotechnical and physical characteristics of the tailings. The laboratory testwork included determination of index properties of the tailings (composition, particle density, and plasticity) and evaluation of tailings material consolidation, compressibility, and permeability characteristics under a range of confining pressures.

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.

Tailings slurry settling and air drying tests were carried out to measure the initial settled tailings density under a range of settling conditions (undrained settling, drained settling, and air-drying). The testwork shows that an initial settled dry density of approximately 1.3 t/m^3 is achievable for the tailings in both drained and undrained settling.

Laboratory tests were also conducted to determine the consolidation and permeability characteristics of the tailings included slurry consolidometer testing, falling head permeability tests, and seepage induced consolidation testing. The consolidation test results indicate that the tailings should achieve a final average dry density of approximately 1.5-1.6 t/m^3 at the stress levels proposed in the TSF and 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.

18.5.5 Embankment Material Types and Placement Requirements

The earthfill seal zone (Zone S) will be constructed using low-permeability glacial till from nearby external borrows and from pit stripping. The material consists of well-graded silty sand with some gravel with a fines content of 20 to 60% (passing the #200 sieve). This material will generally require no processing except for the occasional removal of oversized particles. The material will be placed in maximum 300 mm lifts loose and compacted by combination of smooth drum vibratory rollers and pad foot compactors to a minimum of 95% of standard proctor maximum dry density or to method specifications established during construction that are sufficient to achieve the design intent of Zone S.

The filter zone (Zone F) will comprise clean, fine to coarse sand. It will be placed adjacent to and downstream of the earthfill seal zone to prevent piping of the Zone S material and to reduce pore pressures within the embankment. This material will be a processed non-reactive sand and gravel material from the esker borrow area. The transition zone (Zone T) will be constructed adjacent to and downstream of the Zone F. The Zone T material will be constructed with processed non-reactive fluvial, colluvial material, or selected waste rock. The transition zone prevents the migration of fines from Zone S and Zone F into the pervious downstream shell zone (Zone C). The Zone F and T materials will be placed and spread in maximum 600 mm lifts loose. Compaction will be carried out using smooth drum vibratory rollers with a minimum of 4 to 6 passes. Method specifications will be established during construction to confirm the equipment and compaction techniques are sufficient to achieve the design intent of Zones F and T.

The shell zones (Zone C) will be constructed with random fill comprising overburden and specific waste rock material types. The Zone C materials will be dumped and spread in maximum 1,000 mm lifts loose. Compaction will be achieved with smooth drum vibratory rollers and/or selective truck routing across the main fill to produce a uniformly compacted lift.

Compaction on the edges of the fill will carried out using smooth drum vibratory rollers with a minimum 4 to 6 passes. Method specifications will be established during construction to confirm the equipment and compaction techniques are sufficient to achieve the design intent of Zone C.

18.5.6 Facility Design

18.5.6.1 TSF C

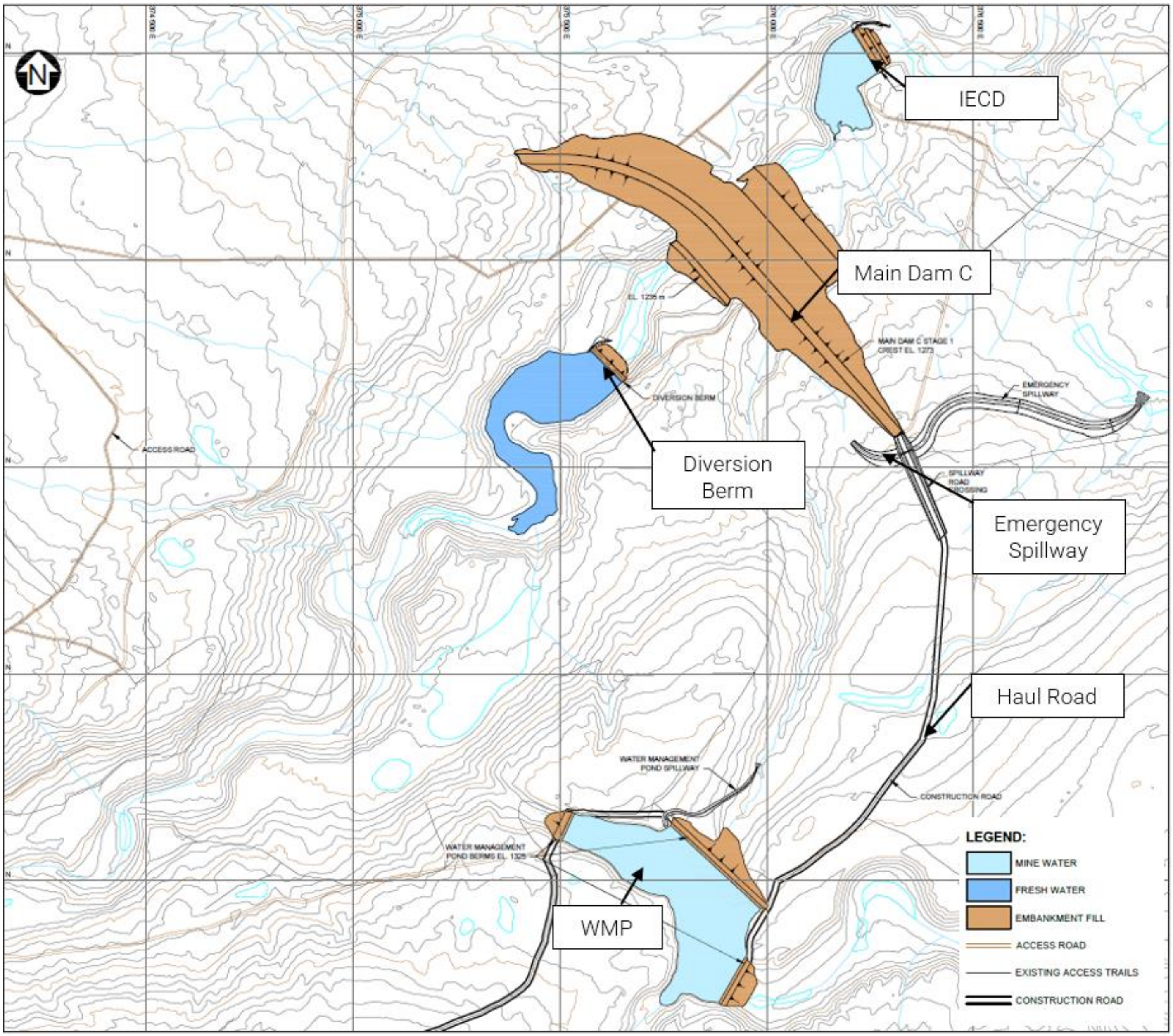
TSF C will be constructed first to provide storage capacity for start-up of the process plant. Specific overall features of the Stage 1 TSF layout are listed below (KP, 2021h):

- Diversion berms, ditches, initial fill placement areas, and pumping systems that will collect and retain or direct water around the TSF during construction;
- A sediment control pond to settle and release construction contact water;
- A zoned water retaining earth-rockfill dam (Main Dam C) that will be approximately 1.2 km in length that will average 25 m in height with a maximum heigh of 60 m at the maximum section in Davidson Creek;
- An embankment drainage collection system comprising perimeter ditches, a chimney drain, longitudinal drains, foundation drains, and embankment outlet drains.
- An IECD and pumpback system that will recover seepage downstream of Main Dam C. The IECD will be located downstream of TSF C and will manage seepage from TSF C and other catchment runoff until construction of TSF D. The IECD is designed with a maximum storage capacity of approximately 58,000m^3 but will be maintained dewatered to the maximum practical extent;
- A HDPE-lined WMP located downslope of the open pit and stockpiles area and within the ultimate footprint of TSF C to manage runoff from contributing areas and water pumped from collection points. The WMP will provide make-up water to support ore processing. Water not needed to support mine operations will be conveyed to the FWR and used to mitigate flow reductions in lower Davidson Creek. The WMP design details are presented in Section 18.6.
- Tailings distribution system and reclaim water systems;
- Designated PAG/ML waste rock disposal areas (located within the TSF);
- Borrow areas, stockpiles, and roads;
- Monitoring instrumentation.

Davidson Creek will be diverted around the initial construction area using a pump system and pipeline installed along the left (north) bank of Davidson Creek. The pipeline will bypass flows and discharge to an erosion protection-lined stilling basin downstream of the Stage 1 TSF initial construction area. A diversion berm will be located approximately 500 m upstream of Main Dam C to facilitate diversion of Davidson Creek. A downstream sediment control pond will be constructed nearer to the initial construction area prior to site preparation and initial embankment construction. The sediment control pond will be decommissioned in Year -1 following construction of the IECD, facilitating continued construction of Main Dam C.

Construction of Main Dam C Stage 1 will commence following completion of the site establishment and water management features described above. The embankment foundations will be cleared and stripped in preparation for fill placement for each stage. Main Dam C Stage 1 was designed as a water retaining dam with a crest elevation of 1,273 masl. The Stage 1 TSF will provide sufficient capacity to impound tailings and PAG/ML waste rock generated during the first year of operations and a supernatant pond up to 2 Mm^3, with additional capacity to manage seasonal water volume fluctuations and the environmental design flood. An emergency spillway will be constructed along the right abutment with an inlet invert elevation of 1,268.3 masl to pass the IDF in the highly unlikely event that the inflow design flood occurs during the first year of operations. Main Dam C will require placement of approximately 3.25 Mm^3 of fill material for Stage 1 that will be sourced from local external borrow sources and pre-stripping of the open pit during Year -1 of mine development. The general arrangement of the Stage 1 TSF at the end of Year -1 is shown on Figure 18-3.

Figure 18-3: TSF Stage 1 General Arrangement

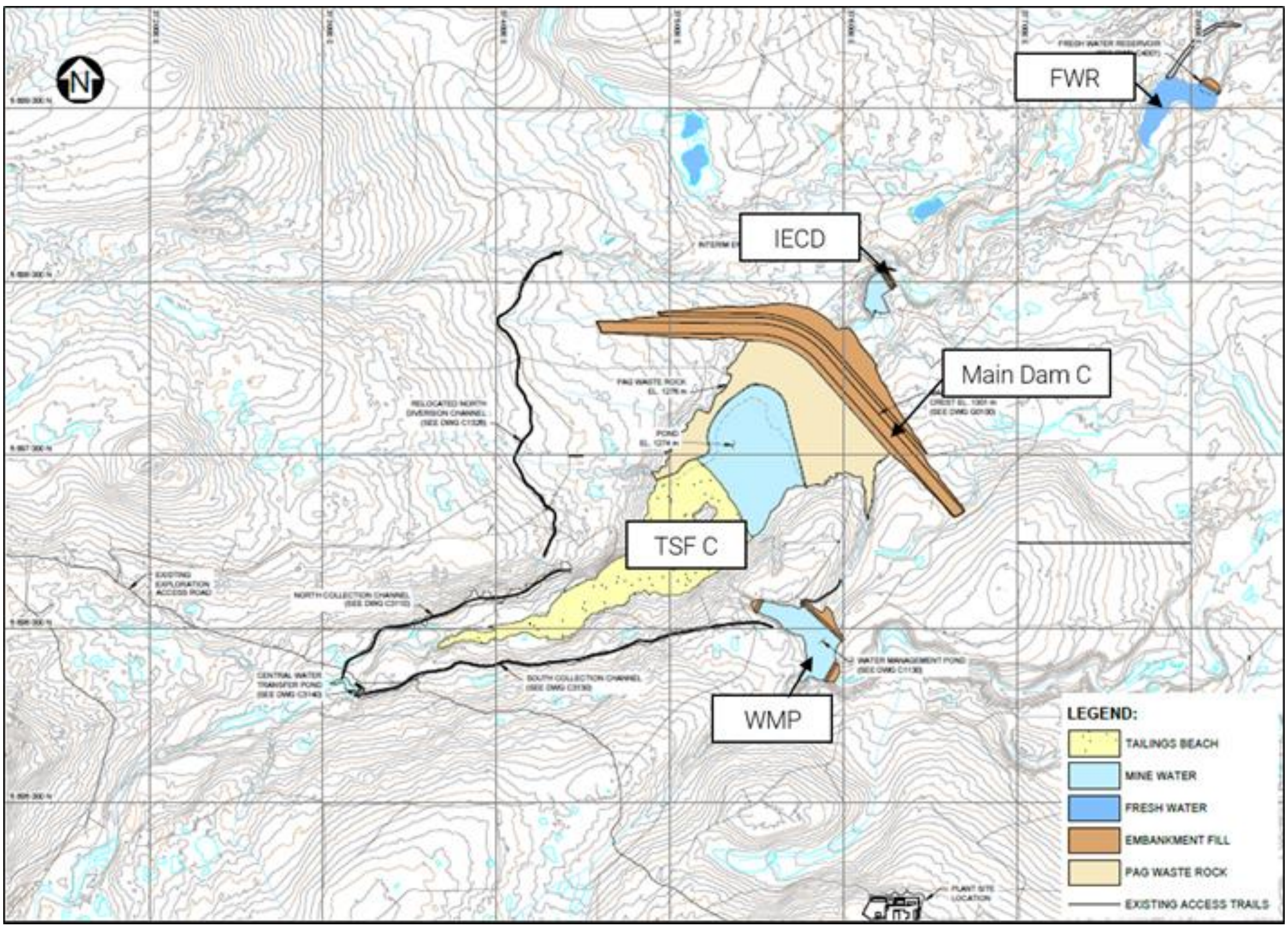


Source: KP, 2021h
Note(s):
1. Main Dam C – Stage 1 General Arrangement from Drawing C1001, included in Appendix D of KP, 2021h.

Main Dam C will be constructed out of NAG open pit materials and locally sourced overburden. The embankment section comprises an impervious earthfill seal zone, designated Zone S, flanked on the downstream side by two filter zones, designated Zones F and T. A downstream shell zone (Zone C) will be constructed using a combination of previous rockfill and sand and gravel from the open pit and less pervious overburden, such as glacial till. Internal drainage systems were designed to control seepage through the embankment (KP, 2021h).

Tailings will initially be discharged into TSF C from one or more points on the west side of the facility with the PAG/ML waste rock disposed of directly upstream of Main Dam C during the first several years of operations. The tailings distribution system corridor will initially follow portions of the existing exploration access road and new purpose-built access until the low point near the current crossing of Davidson Creek. Tailings will be discharged from this location and will flow east towards the waste rock disposal area during the initial six years of ore processing. The supernatant pond will form near the interface of the two disposal areas to allow for efficient saturation of the waste rock interstitial space within one year of placement to meet geochemical objectives. The general arrangement of the TSF at the end of Year 3 is shown on Figure 18-4, representing the general footprint and extent of the TSF during Phase 1 of mine operations.

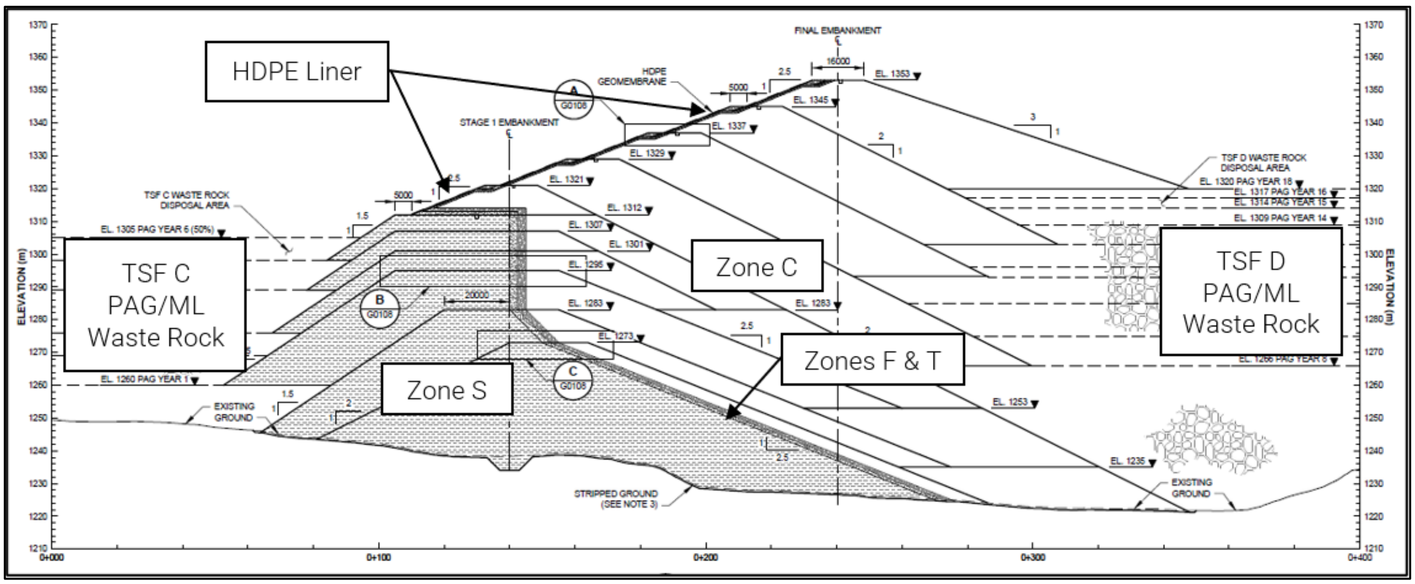
Figure 18-4: TSF General Arrangement – End of Year 3



Source: KP, 2021h
Note(s):
1. Main Dam C – Stage 1 General Arrangement from Drawing C1001, included in Appendix D of KP, 2021h.

Main Dam C will be raised annually through Year 6 to an elevation of 1,321 masl using centreline construction methods. Thereafter, the dam will be raised periodically in stages approximately 8 m high using downstream construction methods. Stages 7 through 11 of Main Dam C will comprise zoned earth-rockfill complete with HDPE geomembrane facing. Construction of Main Dam C will require placement of approximately 52 Mm^3 of fill material to reach a final elevation of 1,353 masl. The typical embankment section for Main Dam C is included on Figure 18-5.

Figure 18-5: Main Dam C – Typical Section



Source: KP, 2021i

Note(s):

1. Tailings Storage Facility – General Arrangement – Year 3 from Drawing G0023, included in Appendix F of KP, 2021.
2. Approximate PAG/ML waste rock disposal area limits shown but tailings materials above excluded for clarity.

The primary drawback of transitioning to downstream raising of Main Dam C is an overall increase in dam construction quantities. However, the transition between design concepts at Main Dam C beginning in Stage 7 has several benefits, including the following:

- Minimizes risk of NAG material shortages (overburden and waste rock) resulting from the concurrent annual construction material needs for raising Main Dam D, which utilizes a similar design concept as the earlier stages of Main Dam C.
- Downstream, periodic raising of Main Dam C at this time limits the frequency of relocation of the tailings distribution infrastructure, which will be extended along the crest of the dam during construction of Stage 7.
- Allows use of PAG/ML waste rock disposed within TSF D to selectively form part of the downstream shell of the dam where the geochemical criteria for saturation can still be achieved.

The Saddle Dam will comprise a zoned earth-rockfill dam with HDPE geomembrane facing and will be constructed in three stages (corresponding with Stages 9 to 11 of Main Dam C) beginning in Year 12 using downstream construction methods to an ultimate elevation of 1,353 masl. The dam will be located east of the WMP, requiring relocation of the WMP and primary mine haul road as waste materials are placed and the elevation of stored materials increases within TSF C. The ultimate dam will be approximately 900 m long with a typical height less than approximately 20 m and a maximum height of 55 m requiring placement of approximately 2 Mm^3 of fill materials. It joins with Main Dam C above an elevation of 1,350 masl to form one continuous embankment.

The West Dam will be constructed in two stages beginning in Year 6 to form the western limit of the TSF. The embankment will initially be constructed to an elevation of 1,345 masl and raised to an ultimate elevation of 1,353 masl in approximately Year 12 or as required to support on-going tailings deposition. The ultimate dam will be approximately 250 m long with a height of up to approximately 20 m. The West Dam will comprise zoned earth-rockfill with HDPE geomembrane facing. Construction of the West Dam requires placement of 150,000 m^3 of fill material to reach the ultimate elevation, the majority of which will be sourced from local excavations within the TSF basin.

The West Dam will be located approximately 750 m downstream of Lake 016852LNRS (Lake 16) in the headwaters of Davidson Creek. A fish offsetting project (design by others) is proposed that will divert flows from Lake 16 towards Lake 01538UEUT (Lake 15) in the headwaters of Creek 705. Drainage from the undiverted catchment areas upstream of the West Dam will accumulate with water from the second phase of the Central Diversion System (CDS) on the west side of the West Dam, forming the revised location of the central water transfer pond. Additional details related to the CDS and other water management facilities are provided in Section 18.6 of this Report.

18.5.6.2 TSF D

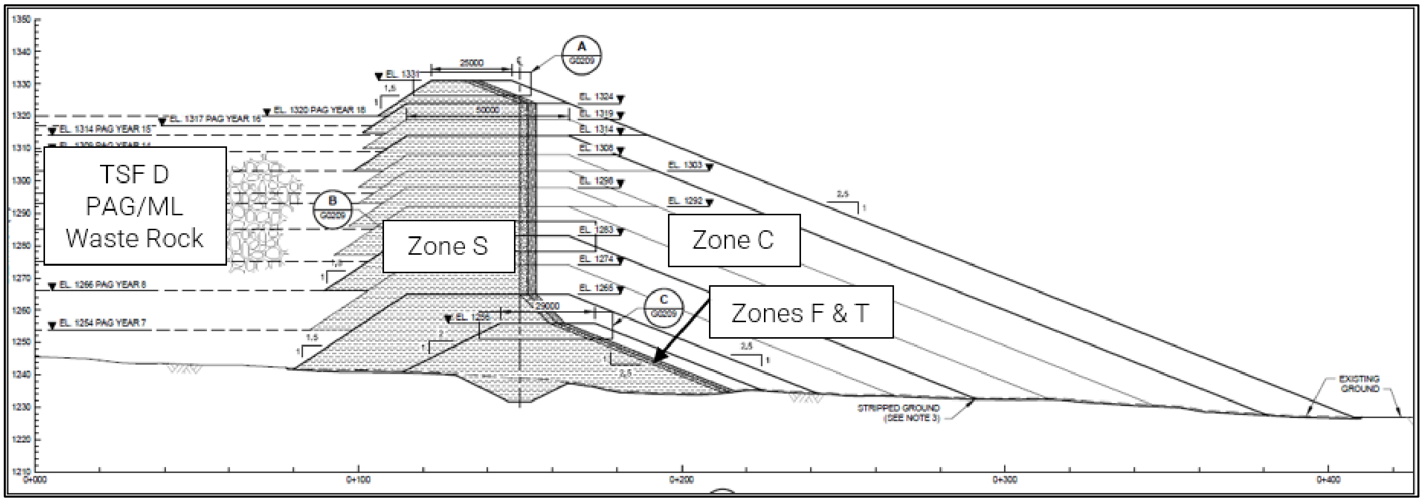
Initial construction of TSF D will begin with construction of Main Dam D Stage 1 in approximately Year 5 as part of the Phase 2 mine expansion. Main Dam D will be constructed just downstream of the IECD. TSF D was sized to contain 180 Mm^3 of waste rock and 25 Mm^3 of tailings. The design includes a nominal pond storage allowance of up to 2 Mm^3 beginning in Year 21 to allow for recycling of process water to TSF C and sufficient additional capacity for seasonal inflows and storage of the IDF throughout at each stage.

Smaller upstream tributaries will be diverted around the initial construction area for Main Dam D prior to initial fill placement and construction of a downstream sediment control pond nearer to the initial construction area. The Stage 1 dam was designed with a crest elevation of 1,256 masl and will be approximately 2.1 km in length averaging 25 m high with a maximum height of approximately 55 m where Davidson Creek is heavily incised. Stage 1 of Main Dam D will comprise a zoned earth-rockfill dam requiring placement of approximately 5 Mm^3 of fill material that will be sourced from local external borrow sources and the open pit waste materials beginning in approximately Year 5 of operations. The dam will be built to elevation 1,265 m by the end of Year 6.

The embankment foundations will be cleared and stripped in preparation for fill placement for each stage. Main Dam D will be raised annually through Year 15 to an elevation of 1,324 masl using centreline construction methods to provide sufficient storage each year for PAG/ML waste rock disposal. Thereafter, the dam will be raised to an ultimate elevation of 1,331 masl prior to tailings deposition into TSF D. The ultimate dam will be approximately 5 km long with an average height of approximately 50 m and a maximum heigh of approximately 130 m at the narrow, heavily incised are where Davidson Creek currently flows. Construction of Main Dam D requires placement of approximately 48 Mm^3 of fill material to reach a final elevation of 1,331 masl.

Main Dam D was designed as a water retaining dam and will be constructed out of NAG open pit materials and locally sourced overburden. The embankment section comprises an impervious earthfill seal zone, designated Zone S, flanked on the downstream side by two filter zones, designated Zones F and T. A downstream shell zone (Zone C) will be constructed using a combination of previous rockfill and sand and gravel from the open pit and less pervious overburden, such as glacial till. Internal drainage systems were designed to control seepage through the embankment (KP, 2021i). The typical embankment section for Main Dam D is included on Figure 18-6.

Figure 18-6: Main Dam D – Typical Section



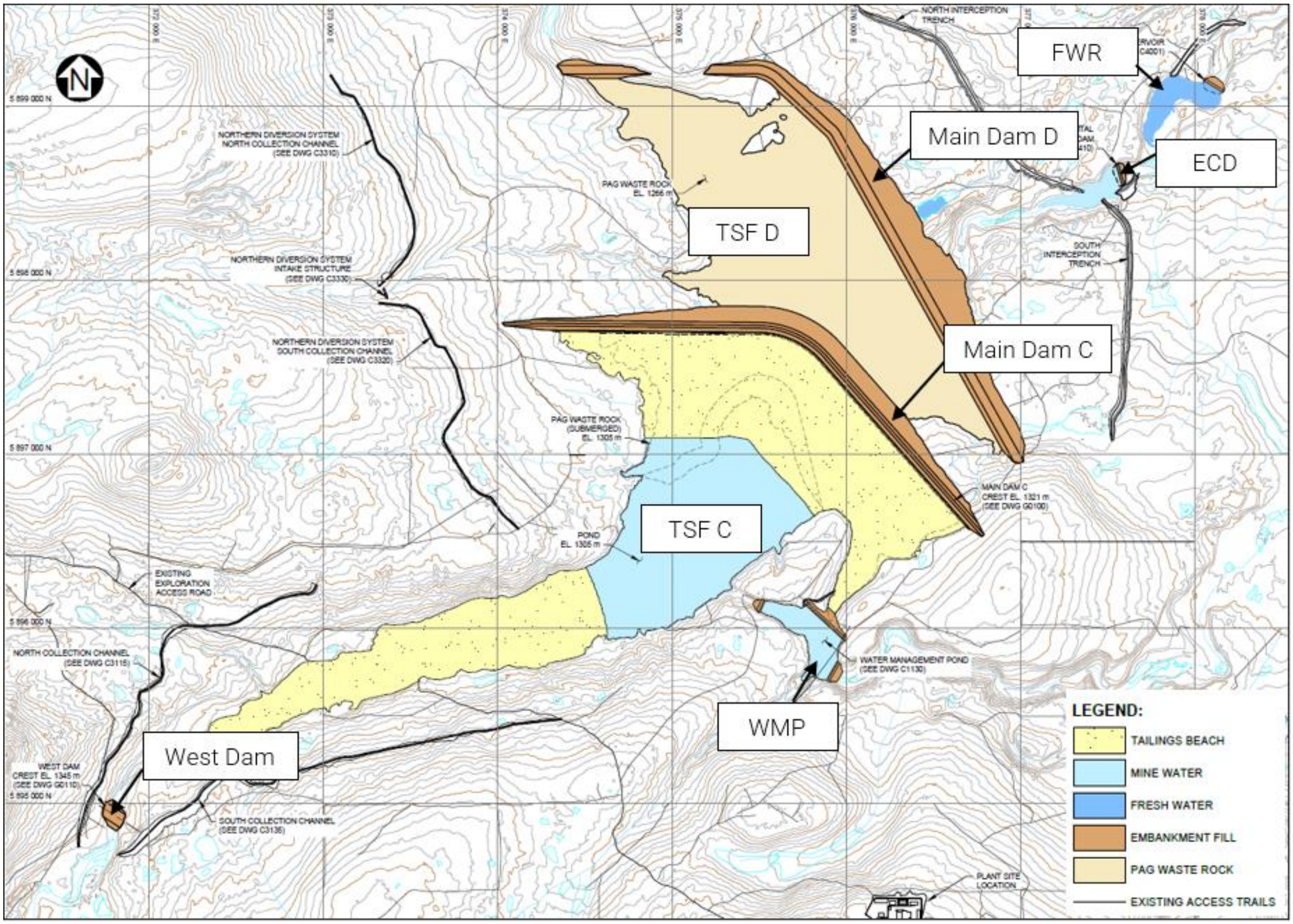
Source: KP, 2021i
Note(s):
1. Typical section based on Main Dam D – Section 3 from Drawing G0205, included in Appendix F of KP, 2021i.
2. Approximate PAG/ML waste rock disposal area limits shown but tailings materials above excluded for clarity.

The Environmental Control Dam (ECD) downstream of TSF D will be constructed in Year 6 approximately 1,000 m downstream of Main Dam D and upstream of the FWR at a topographic low point in Davidson Creek. The ECD replaces the IECD, which will be decommissioned and buried within TSF D during Year 6. Two seepage interception trenches (north and south of Davidson Creek) will be excavated through the surficial sand and gravel terraces downstream of the Main Dam D and will report to the ECD pond. The trenches will be approximately 3.3 km long with a depth ranging from 5 m to 15 m. The ECD will manage seepage and storm water inflows and will utilize a pumpback system to convey the recovered flows to the TSF. The dam will be maintained in a dewatered condition to the maximum extent practical. The ECD has a capacity of approximately 194,000 m^3.

The tailings distribution system will be extended along the crest of Main Dam C during Year 6 to allow for discharge of tailings from the dam crest beginning in Year 7 to cover the submerged PAG/ML waste rock disposal area and manage the location of the supernatant pond. The PAG/ML waste rock will be disposed of directly upstream of Main Dam D and downstream of Main Dam C to form a waste rock storage facility between the two up to an elevation of approximately 1,320 masl.

The general arrangement of the TSF at the end of Year 8 following construction of Main Dam D and tailings discharge from Main Dam C is shown on Figure 18-7, representing the typical operating arrangement of the TSF during Phase 2 of mine operations.

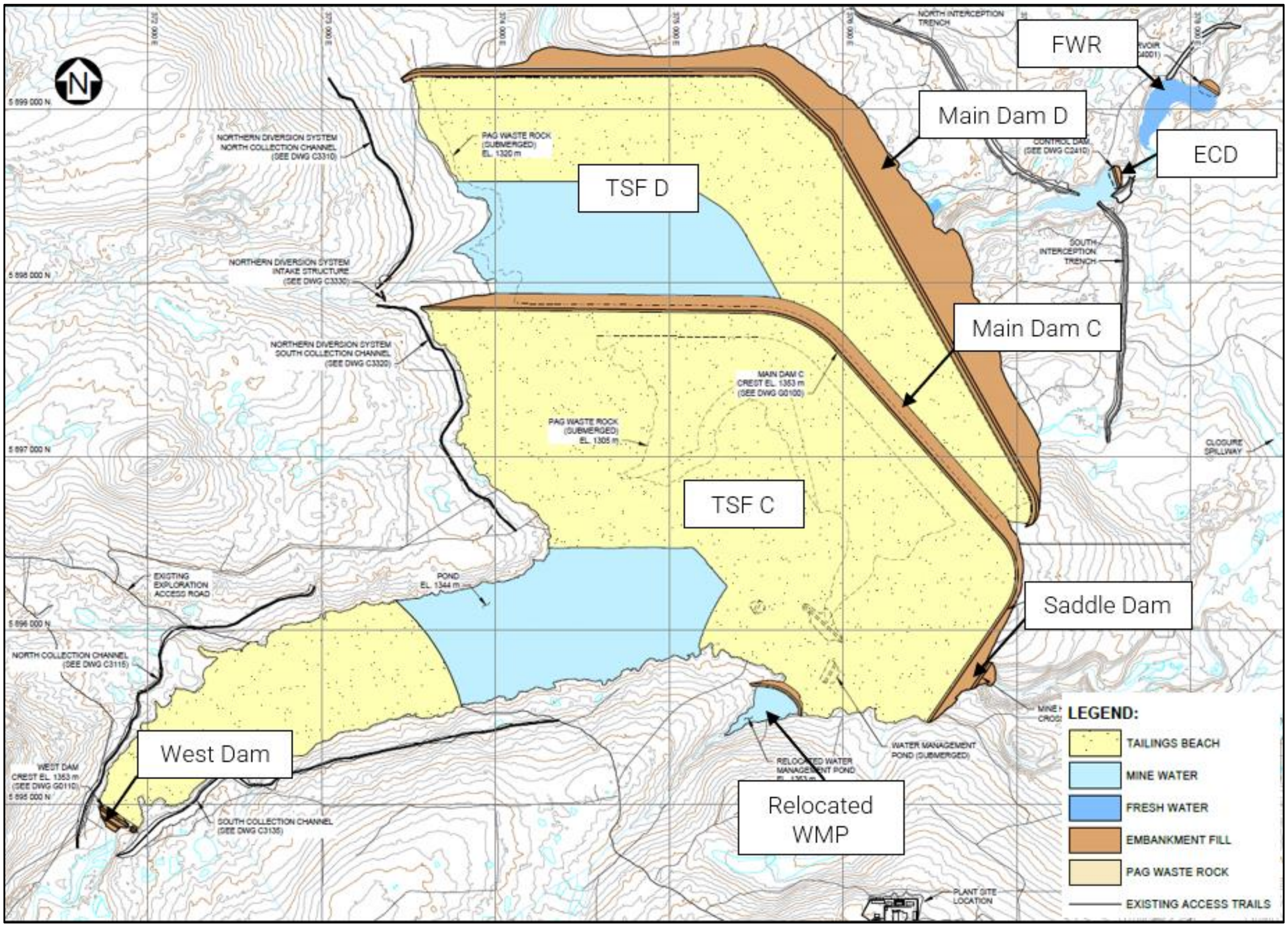
Figure 18-7: TSF General Arrangement – End of Year 8



Source: KP, 2021i
Note(s):
1. Tailings Storage Facility – General Arrangement - Year 8 from Drawing G0025, included in Appendix F of KP, 2021i.

The tailings distribution system will be directed to TSF D during Year 20 to allow for discharge of tailings from the dam crest beginning in Year 21 to cover the PAG/ML waste rock disposal area. 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. The general arrangement of the TSF at the end of mine life is shown on Figure 18-8.

Figure 18-8: TSF General Arrangement – End of Mine



Source: KP, 2021i
Note(s):
1. Tailings Storage Facility – General Arrangement – End of Mine from Drawing G0028, included in Appendix F of KP, 2021i.

18.5.7 Seepage Control Measures

18.5.7.1 TSF C

Main Dam C will utilize a cut-off trench excavation and glacial till (Zone S) backfill to connect the impervious earthfill seal zone (Zone S) of the dam to the existing low permeability subgrade (LPS) materials occurring in the dam foundation. The cut-off trench will be excavated through the surficial materials and keyed a minimum of 1 m into the LPS foundation materials. The LPS foundation generally comprises undifferentiated and lodgement type glacial tills with interbedded glaciolacustrine and glaciofluvial materials underlain by completely to highly weathered bedrock. The test pits and drillholes identified suitable glacial till for the base of the cut-off trench at depths of less than 5 m along most areas of the embankment centreline. The inferred depth to LPS materials was estimated based on the drilling, test pitting and lab testing work completed at the site, but a potential deeper LPS target was also identified in recognition of the uncertainty in the existing site investigate results and spacing between the investigation information. The deeper LPS target is greater than 10 m depth in some areas, such as the crossings of Davidson Creek and the mine area creek, which are impractical to investigation further until construction is underway. The final cut-off trench depth will be dependent on conditions encountered during construction.

TSF seepage will be controlled primarily by the low-permeability Zone S, the cut-off trench, and the LPS foundation materials. Seepage from the TSF will result from infiltration of ponded water directly through the embankment fill and the natural ground, and from expulsion of pore water as the tailings mass consolidates. The tailings dam design incorporates embankment drainage collection systems to control and collect seepage and direct the flow to the IECD.

The embankment drainage collection systems comprise the following:

- Longitudinal drains within the Zone T material to collect and convey seepage through the dam longitudinally.
- A vertical chimney drain spanning the length of the embankment up to elevation 1242 masl.
- Foundation drains in select locations downstream of the cut-off trench to collect seepage and control seepage gradients.
- Embankment outlet drains constructed in the Davidson Creek and mine area creek basins to convey seepage downstream to the IECD.
- Perimeter ditches along the downstream toe of the embankment to collect seepage and surface runoff.

TSF D will be constructed adjacent to and downstream of TSF C beginning in Year 6. The interstitial space within the waste rock in TSF D will be progressively saturated to limit oxidation and subsequent acid generation. The rising phreatic level within the saturated TSF D waste rock disposal area will reduce the hydraulic gradient between TSF C and TSF D, resulting in reduced seepage between the two facilities. The HDPE geomembrane facing will be the primary barrier to seepage above the Stage 6 crest elevation of approximately 1,312 masl.

18.5.7.2 TSF D

Main Dam D will also utilize a cut-off trench excavation and glacial till (Zone S) backfill to connect the impervious earthfill seal zone (Zone S) of the dam to the existing LPS materials occurring in the dam foundation. TSF D seepage will be controlled primarily by the low-permeability Zone S, the cut-off trench, and the LPS foundation materials. The tailings dam design incorporates embankment drainage collection systems to control and collect seepage and direct the flow to the ECD. The embankment drainage collection systems will include longitudinal drains, foundation and chimney drains, embankment outlet drains, and perimeter ditches along the downstream toe of the dam.

Groundwater monitoring wells will be installed at suitable locations downstream of the TSF and will be used to recover samples for groundwater quality monitoring. Additional monitoring wells will be installed as required before TSF Site D is commissioned.

18.5.8 Tailings Distribution

The tailings distribution system will transport tailings slurry to TSF C, and later to TSF D. There will be three phases of tailings distribution infrastructure sized to accommodate mill throughput capacities of up to 6 Mtpa, 12 Mtpa, and 20 Mtpa. All phases of operation will consist of gravity-fed pipelines with full-diameter discharge spigots located around the facility to accommodate tailings beach development to manage the supernatant pond location and PAG waste rock cover as required.

The initial stage of tailings distribution will consist of a single, gravity-fed pipeline to convey tailings from the plant site to the southwest side of the TSF (approximately 4,300 m in length). The pipeline will comprise normal diameter (ND) 500 mm HDPE pipe with full-diameter slurry knife gate valves as required to manage tailings deposition locations. The tailings distribution system was sized with sufficient capacity to convey a design flowrate of between 1,250 and 1,430 m^3/h, which is equivalent to a nominal tailings throughput of up to 6 Mtpa. Pipe specifications were selected to eliminate the need for energy dissipation infrastructure prior to discharge into TSF C.

The tailings distribution infrastructure will be expanded during the Phase 2 expansion with the addition of a secondary line to the southwest end of TSF C (approximately 4,300 m in length), as well as two pipelines to convey tailings along the crest of Main Dam C to discharge into the northeast side of the TSF (approximately 7,300 m long each). The additional pipelines are required to manage the supernatant pond location and cover the PAG/ML waste rock disposal area with an oxygen limiting barrier. The pipelines are sized to distribute tailings slurry flows from the plant of approximately 2,500 to 2,900 m^3/h.

The final stage of tailings distribution will require the addition of a third pipeline from the plant to both the southwest and northeast sides of the TSF, to accommodate a maximum mill throughput of 20 Mtpa. The pipeline alignments from the plant to the southwest side of the facility and to the northeast side of the facility are approximately 5,900 m and 8,600 m in length, respectively. The pipeline to the southwest side comprises approximately 5,900 m of ND750 mm DR11 HDPE pipe, while the pipeline to the northeast side comprises approximately 1,500 m (shared with the southwest discharge) ND750 mm DR11 HDPE and approximately 7,100 m ND750 mm DR7 HDPE. The total tailings flowrate will be approximately 4,180 to 4,760 m^3/h.

18.5.9 Monitoring

Geotechnical instrumentation will be installed along three planes through the Main Dam C and the IECD. The instrumentation will be installed during Stage 1 construction and will be supplemented with additional instrumentation over the life of the Project. The geotechnical instrumentation will comprise vibrating wire piezometers, slope inclinometers, and survey prisms, and will be installed in the foundations, embankment fill, and on the embankment crests. Other instrumentation to be installed include flow monitoring devices and water level meters.

Instrumentation details for the West Dam, Saddle Dam, Main Dam D, and the ECD will be developed during the detailed design of each dam; however, provisional allowances for instrumentation and monitoring for these facilities were included in the designs.

Piezometers, inclinometers, and survey prisms/total stations will be installed and connected to an automated data acquisition system that provides real-time access to the monitoring data. Measurements during construction will be taken and analyzed to monitor the response of the embankment fill and foundation materials. Flow monitoring devices, water level meters, and groundwater wells may be measured manually in non-critical areas as prescribed by the engineer. An operations maintenance and surveillance manual will be prepared following initial construction and prior to commissioning of the TSF to provide comprehensive operating instructions and monitoring frequencies for the TSF instrumentation.

18.6 Water Management Structures

18.6.1 Objectives and Concept

The principal design objectives for the water management structures described in this report are to manage surface water during mine operations and active closure. Surface water is to be managed in a manner that allows for the beneficial use of the water to support mine operations and to divert flows not needed to the downstream receiving environment. 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 flows by gravity into the TSF, following natural topographical drainages mapped for the Project; which simplifies water management, spill control, and mine closure. Mine affected runoff within the project area will be captured and recycled for us as process water. Surplus water not required to support mine operations will be sampled and analyzed, comparing to applicable water quality criteria, and if compliant, will be used to augment flow in lower Davidson Creek. Water management structures were designed to achieve the principal design objectives while taking into consideration the design requirements listed in the previous section.

Specific water management structures and systems described in this report and planned for the mine operations period are listed below.

- The FWR to store water and provide flows to lower Davidson Creek to meet instream flow needs (IFN) downstream of the mine and to provide water for mine operations when required.
- The WMP located downslope of the open pit and stockpiles area and within the ultimate footprint of TSF C to manage runoff from contributing areas and water pumped from collection points. The WMP will provide fresh make-up water to support ore processing. Water not needed to support mine operations will be used to augment flow in lower Davidson Creek (KP, 2021h).
- A discharge system to route water from the WMP to the FWR, and a raw water supply system to route water from the WMP to the plant site.
- Reclaim water system to route water from the TSF to the plant site.

- The CDS to divert fresh water around the TSF to a water transfer pond, from where it can be pumped to the WMP. The CDS includes two phases of development, with the first phase operational between Years -1 and 6 and the second phase between Years 7 and 23.

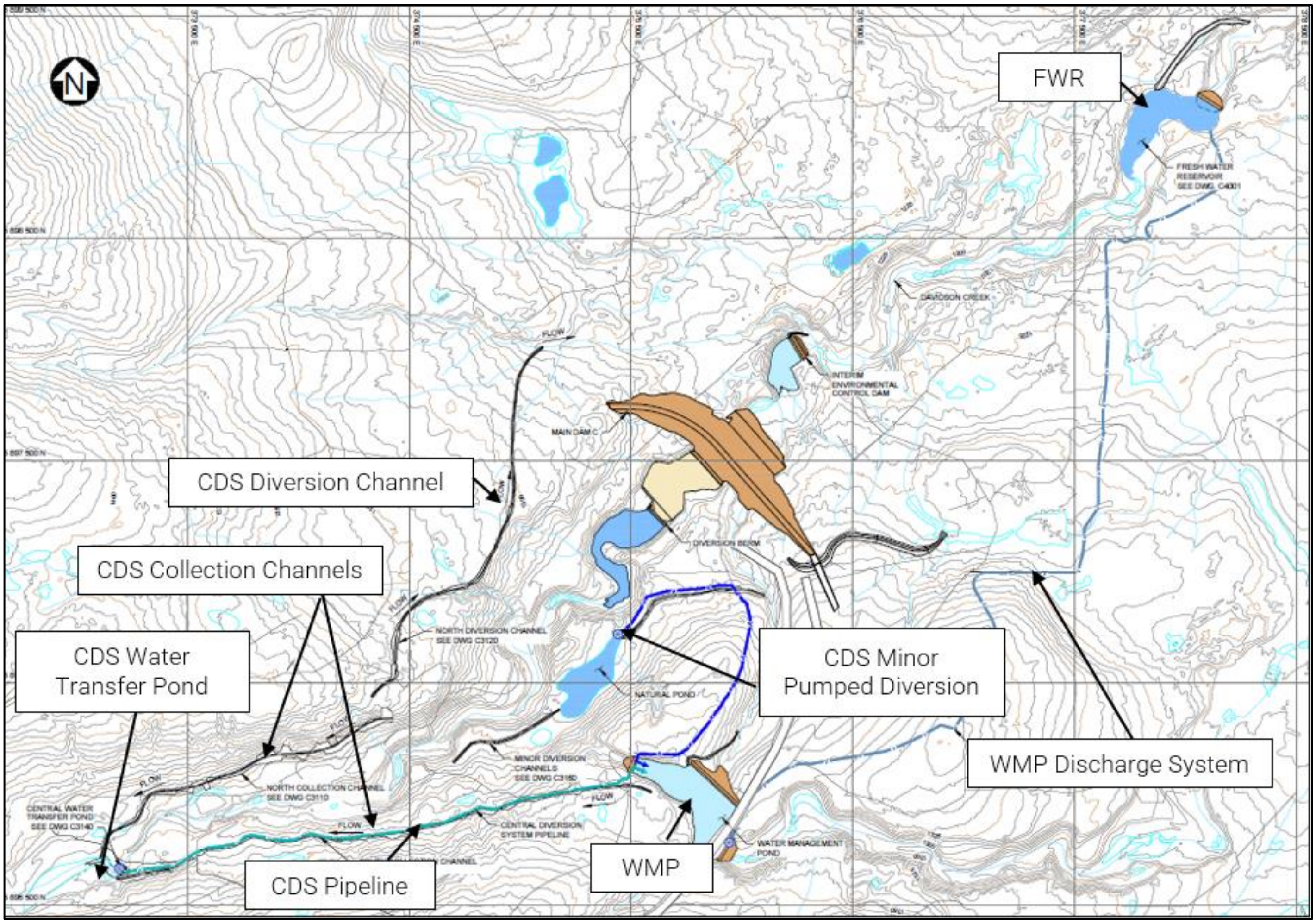
- The NDS to divert freshwater around TSF D to the FWR or allow it to bypass diversion and flow into TSF D, depending on the needs of the mine. The NDS will be required in approximately Year 6 following TSF D construction.

- Fresh water from Tatelkuz Lake supplied by the FWSS to the FWR beginning during the Phase 2 expansion (KP, 2021l).

- Stockpile water management structures to divert and contain seepage and surface water runoff from the LGO Stockpile and the stockpile.

A summary of each system is presented in the sections that follow with the initial water diversion systems required at the start of mine operations shown on Figure 18-9. The discussion of the WTPs planned for mine operations and closure are presented in Section 20.

Figure 18-9: Water Management – Diversion Systems – General Arrangement – Year 1



source: KP, 2021k
Note(s):
1. Diversion Systems General Arrangement for Year -1 from Drawing C3100, included in Appendix D of KP, 2021k.

18.6.2 Water Management Systems

18.6.2.1 Fresh Water Reservoir

The purpose of the FWR is to maintain a suitable source of fresh water to provide flows to lower Davidson Creek as required to reduce the potential environmental impacts of the project and to support mine operations when required. The FWR will be formed as an in-creek reservoir using natural topography enclosed by construction of an earthfill berm on the northeast side of the reservoir. The FWR embankment is designed with a crest elevation of 1,167.5 masl, and the base elevation at the highest section of the embankment is approximately 1,153 masl. Davidson Creek is incised in the vicinity of the reservoir, and the natural ground elevation is typically greater than 1,170 masl adjacent to the reservoir. The maximum embankment height is approximately 15.5 m, and most of the embankment is over 10 m high except locally at the abutments. The embankment will be approximately 125 m in length and will impound a total volume of around 370,000 m^3 from its foundation level to the spillway invert elevation. The specific design features of the FWR include the following:

- Water retaining zoned earthfill embankment;
- HDPE geomembrane liner on the upstream face of the embankment and select areas of the reservoir
- Low level outlet and surface level outlet pipes
- Temperature and flow control chambre
- Overflow spillway and stilling basin

18.6.2.2 Water Management Pond

The WMP will be formed using natural topography enclosed by construction of three geomembrane-lined earthfill berms on the west, north, and east sides of the pond. Each berm is designed with a crest elevation of 1,325 masl, with the pond basin invert at an average elevation of 1,308 masl. The basin and berms will be fully geomembrane-lined up to 1,324.5 masl providing a total water storage capacity of 825,000 m^3 at this elevation. Alternative lining methodologies may be evaluated following geotechnical investigations at the WMP during construction, which will be performed to verify that the ground conditions are consistent with design assumptions and to assess if natural ground conditions within the basin are more favorable than assumed in the design.

The berms will be constructed from locally borrowed earthfill materials meeting the Zone C material specification. The specified minimum crest width is 12 m, which includes an allowance for an access road running between two safety berms. The West Berm is designed with a 16 m crest width to accommodate the reclaim water pipeline corridor.

18.6.2.3 WMP Discharge System

The discharge system will convey surplus water from the WMP to the FWR beginning during pre-production. The system comprises three skid-mounted self-priming centrifugal pumps arranged in parallel near the WMP. The pumps are to be contained within weather-protected sea-can units. The pumps each provide approximately 40 m of total dynamic head (TDH) and flowrates of 940 m^3/h for a total system flowrate of approximately 2,819 m^3/h. The pumps will be controlled by a floating level control system in the WMP and equipped with variable frequency drives (VFDs) to balance the total system TDH from the individual units. One uninstalled pump will be available along with all the necessary control, check, drainage, and isolation valves. The pipeline will consist of a combination of ND 300 mm DR21 HDPE, ND 750 mm DR21 HDPE, and ND 500 mm DR9 HDPE. The pipeline corridor grade will be optimized to minimize high or low sections, which results in gravity-flow conveyance to the FWR after initial lifting from the pump system.

18.6.2.4 Raw Water System

Floating submersible pumps will be installed within the WMP. These pumps will be the same make and model as those installed at the TSF reclaim intake pump station within the supernatant pond. The pumps will feed a secondary shore-mounted centrifugal pump at the permanent booster station, also the same make and model as that used for the main reclaim system. This pump will feed an independent pipeline that will discharge to the raw water tank at the process plant site. The pump system from the WMP will be connected to the TSF reclaim system through a series of valves to allow for discharge from either pipeline between the permanent booster station and the process water tank or the raw water tank.

18.6.2.5 Water Reclaim System

Water reclaimed from the supernatant pond at TSF C will be delivered to the process water tank at the mill. The reclaimed water will consist of supernatant from the settled tailings and runoff from precipitation and snowmelt within the reporting catchment areas. The water reclaim system will initially comprise two pump stations and a pipeline. The intake pump station will comprise two low-head floating submersible pumps and one high-head shore-mounted centrifugal pump, and the booster station will contain a single shore-mounted centrifugal pump. The pipeline will consist of a combination of ND 450 mm DR11 and ND 500 mm DR7.3 HDPE pipe. The floating submersible pumps will be anchored on the southern side of the TSF C supernatant pond throughout operations, and tailings will be selectively discharge to the facility to maintain the location of the supernatant pond.

The initial system design flowrate will be approximately 1,066 m^3/h. The water reclaim system in TSF C will be twinned during the Phase 2 expansion, and a third parallel system will be subsequently added later, with each expansion coinciding with the two capital expansion periods for the mill facilities. A reclaim water system, consisting of a barge-mounted pump station and three 300 HP centrifugal pumps, will be added to TSF D in preparation for recycling process water resulting from tailings deposition within that facility.

18.6.2.6 Central Division System

The CDS will be constructed to divert freshwater around the TSF to the downstream receiving environment or to water transfer points where the captured flows can be pumped to the WMP. The CDS will comprise the following primary components as shown on Figure 18-7:

- Diversion and collection channels;
- Water transfer pond;
- Pipeline and pump station;

The design of the CDS was separated into two phases because the system components will need to be relocated in approximately Year 6 due to the expanding footprint of the TSF. The central water transfer pond will initially be located near the existing exploration access road and will be relocated in approximately Year 6 to the west of the West Dam following its construction. The berm will be less than approximately 5 m high and constructed of locally borrowed overburden materials and concrete lock blocks. A second flow-through berm will be constructed of screened gravel and cobble sized materials upstream of the water collection area to limit fish access from Upper Davidson Creek located to the west. The water conveyance pipeline will consist of a combination of ND 500 mm HDPE DR17 and ND300 mm HDPE DR21 pipe, which is sized to convey a flow of approximately 0.3 m^3/s (300 L/s).

18.6.2.7 Northern Division System

The NDS will be located up-gradient of TSF D and will be constructed in Year 6 to allow for diversion of upstream flows around the TSF. The NDS will comprise the following primary components:

- Collection channels;
- Water transfer pond and intake structure;
- Gravity flow pipeline;

The NDS intake structure will consist of a 3.5 m high concrete structure to provide submergence to the water conveyance pipeline. The pipeline will consist of a combination of ND 350 mm HDPE DR17 and ND 550 mm HDPE DR17, which is sized to convey a flow of approximately 0.3 m3/s (300 L/s). The intake structure will include a gated sluice pipe to clean out sediment accumulation. Flow will be conveyed for over 6 km around the Project facilities within the water conveyance pipe and discharged to FWR, via an energy dissipation structure. The intake structure will include a spillway sized to convey a 200-year design peak flow assuming the water conveyance pipeline and gated sluice pipe are inoperable. The spillway will comprise a 5 m wide broad-crested weir capable of passing the design storm while maintaining adequate freeboard.

18.6.2.8 Fresh Water Supply System

It is anticipated that the amount of available water from runoff to the FWR and flows conveyed to the FWR from the mine site area alone will be insufficient to meet the instream flow needs in Davidson Creek at some stage of mine development. Fresh water to offset flow reductions in lower Davidson Creek will be sourced from Tatelkuz Lake, which is located approximately 20 km northeast of the mine site. The FWSS will convey water from the lake to the Project site as required by pumping water from Tatelkuz Lake through a pipeline and into the FWR and/or directly into Davidson Creek at the FWR to provide additional water to address the shortfall. The FWSS consists of a land-based intake structure, an intake pump station, a pipeline, and a booster pump station.

The intake of water from Tatelkuz Lake will be conducted via a permanent, land-based concrete wet-well structure. The structure will consist of horizontal screened intake pipes, a cast-in place concrete wet-well, vertical turbine pumps, and a pre-fabricated steel superstructure to house the associated mechanical and electrical infrastructure.

The intake pump station will initially be outfitted with three vertical turbine pumps: two operating and one on standby. The pumps will be housed inside a prefabricated steel structure and will draw water from the wet-well chamber, approximately 9 m below the pump house floor. Each pump will provide the required TDH at a maximum flowrate of 335 L/s (1,200 m^3/h) to meet the total design flowrate of 670 L/s.

The pipeline will be constructed using ND 750 mm diameter HDPE of various thickness/pressure ratings ranging DR17 to DR9 depending on system operating pressure requirements. The pipeline will be buried and/or insulated and heat traced in various sections to prevent freezing. The pipeline will be approximately 13,740 m in length and the pipe alignment will be routed along existing access roads as much as practicable. In some cases, the pipeline alignment is routed along the best accessible route since no existing access roads exist between the proposed intake location and the FWR location; new access roads need to be constructed for this purpose.

The booster pump station will comprise a structural steel skid complete with a self-framing building equipped with all necessary access, insulation, heating, ventilation, and lighting. The skid will contain the pumps, piping, and valves in one room, and the electrical infrastructure in a separate room. The pump station will be equipped with three horizontal pumps: two operating, and one on standby. The pumps will be equipped with VFDs and will be fed directly from the main FWSS pipeline. Each horizontal pump will provide 155 m TDH at a flowrate of approximately 1,206 m^3/h, and be equipped with motors rated to 1,250 HP. The system will be controlled via input from pressure indicators and a flowmeter on the discharge of the pumps, all located within the enclosed skid. Discharge piping will be carbon steel prior to the connection to the main pipeline, which will convey water from the booster station to the FWR.

The water release conditions will be controlled by a temperature and flow control system consisting of temperature and flow measurement devices and associated control logic feedback loops on the discharge pipeline. A reservoir bypass line will connect directly to the water supply pipeline to allow for direct discharge of the required instream flow needs during reservoir maintenance. It can also be used to provide cooler water as required for fisheries in Davidson Creek.

18.6.2.9 Low-Grade Stockpile

The design of the LGO Stockpile water management system includes a liner system across the footprint area of the stockpiles and a series of non-contact water diversion channels and contact water collection channels. Non-contact water will be diverted around the facility and contact water will report to a collection pond. Contact water will be pumped to the process plant for neutralization before being discharged to the TSF.

18.6.2.10 Waste Storage Facility

Surface contact water will be managed in a manner that allows for safe containment and control. Collection channels will be constructed along the stockpile periphery to collect and convey contact surface runoff to a collection pond where the flows can be pumped to the metals WTP, if required, and eventually discharged to the WMP. Refer to section 20 for details of the water treatment systems.

18.6.3 Monitoring

Geotechnical instrumentation will be installed along one plane through the FWR and through each of the WMP berms. The instrumentation will be installed during initial construction and will be supplemented with replacement instrumentation over the life of the Project as required. The geotechnical instrumentation will comprise vibrating wire piezometers, slope inclinometers, and survey prisms, and will be installed in the foundations, embankment fill, and on the embankment crests. Measurements during construction will be taken and analyzed to monitor the response of the embankment fill and foundation materials.

Water from the FWR released through the outlets will be monitored using ultrasonic flow meters installed at each of the discharge outlets. In addition, an in-stream flow monitoring station downstream of the FWR within Davidson Creek may be required as a back-up station to confirm the IFN release, should the flowmeters require maintenance or are damaged. In addition to flow monitoring, water level monitoring within the reservoir is also required to inform the operation of the outlets. Water level monitoring will also provide information related to how much storage is available within the reservoir and will also be used to indicate when the reservoir capacity has been reached and flow through the spillway is occurring. Water levels will be monitored using a pressure transducer with an appropriate depth range. A baseline monitoring station is located in Davidson Creek between the FWR pipe outlets and the FWR spillway outlet and could be used in combination to determine the total flow from the FWR or a more permanent structure such as a weir could be installed further downstream below where the FWR spillway re-enters Davidson Creek.

Water level measurements will typically occur at all water retaining ponds to monitor water levels and compliance with freeboard requirements. Flow monitoring will typically occur at all water management systems to measure outflows from the ponds and/or pumped flows conveyed around the site to improve understanding of inflows/outflows and inform on-going water balance model calibration.

All monitoring instrumentation will incorporate a radio or telemetry system to facilitate automated data collection and to allow accessed to monitoring data in real time, if required. The instrumentation systems will include some degree of overlap and redundancy to enable verification of problems that may be detected. Flow monitoring devices, water level meters, and groundwater wells may be measured manually in non-critical areas as prescribed by the engineer. An operations maintenance and surveillance manual will be prepared following initial construction and prior to commissioning of the water management facilities to provide comprehensive operating instructions and monitoring frequencies for the instrumentation.

18.6.4 Water Balance

A LOM Water Balance Model (WBM) was prepared that explicitly considers the full life cycle of the Project (KP, 2021n). Key mine facilities and water management processes are represented in the model, in addition to modelling surface water runoff and groundwater recharge in the existing natural catchment areas, to simulate mine site and receiving environment water flows from baseline through post-closure. Water flow through key Project facilities was explicitly modelled, including the mill, TSF, waste and ore stockpiles, the open pit, water management structures and ponds, diversions, water treatment plants, and seepage collection systems. The model represents proposed Project development and simulates water management flows, surface water, and groundwater flows during all phases of mine development.

The key objectives of the LOM WBM were to:

- Develop surface water and groundwater flow estimates to use as inputs to a water quality model assessing potential impacts to water quality associated with development of the Project.
- Quantify potential changes to surface water and groundwater flows attributable to project development to support the assessment of potential impacts.
- Assess the Project water management strategies while considering the influence of potential climate variability, including strategies to obtain a source of reliable water to meet Project needs.

Water will be recycled within the Project areas to the maximum extent practicable to reduce consumptive uses from other freshwater sources. The plan is that the majority of process water will come from recycling water from the TSF. The remaining mill water demands are from other sources, prioritized as follows:

1. Open pit dewatering system (from within the pit sumps and pit depressurization wells)
2. Mine site treated water and upstream diversion water stored within the WMP
3. Upstream diversion water and Tatelkuz Lake water, stored within the FWR

Surplus water will be treated as needed before discharging water to Davidson Creek. Discharging treated water during the life of mine helps mitigate flow losses in Davidson Creek and reduces the volume of surplus water stored onsite.

Model results suggest the water inventory of the TSF supernatant ponds under average climate conditions can be maintained within the target operating range through the mine life, which increases during operations from 1 to 10 Mm^3 plus additional allowances for seasonal inflow. Under wetter than average conditions during Operations, model results suggest treatment and release of water from the TSF supernatant pond may be required to maintain the pond water inventory in the target operating range while under drier than average conditions a greater amount of non-contact water is needed to meet mill demand, especially later in the mine life. Mine years with a greater likelihood for treatment requirements (Years 3 and 4 and Years 8 through 10) occur immediately before the TSF C nominal operating water storage allowance specified in the model increases, preceding a ramp up in mine throughput. The start of each increase in throughput and the transition to tailings deposition in TSF D in Year 21 results in increased mine water needs.

Reliance on the FWSS system to provide freshwater to meet instream flow needs in Davidson Creek will increase with the increased demand for make-up water at the mine site as follows:

- There is a low likelihood the FWSS is needed during the first five years of mining, with an estimated 15 to 25% likelihood the FWSS will be needed to meet IFN in any non-freshet month during these early years. The FWSS is predicted to be needed in drier than average conditions during non-freshet months and is not predicted to be needed in average or wetter than average conditions.

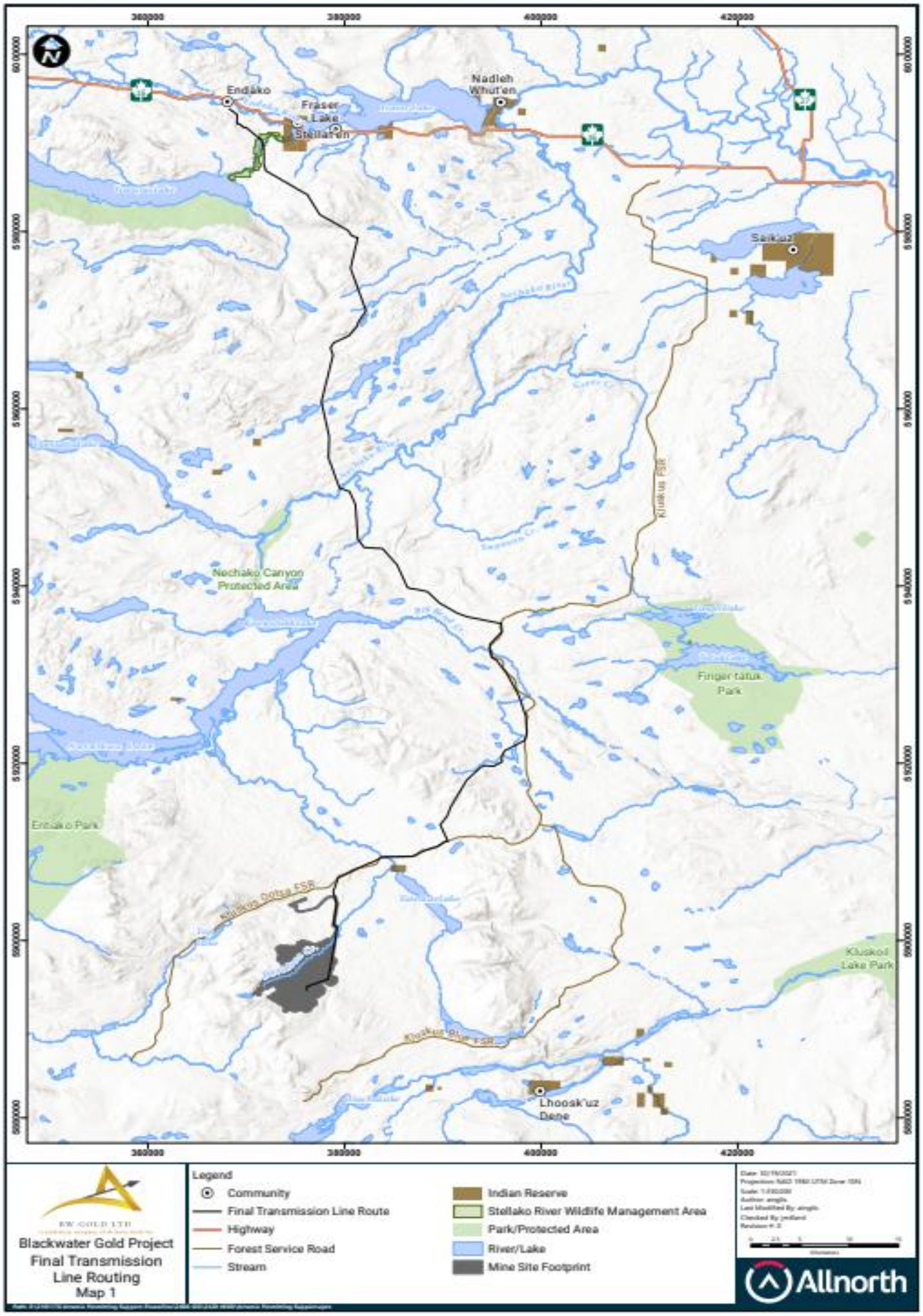
- The FWSS is predicted to be needed consistently after Year 5 once the mining throughput increases and construction of Main Dam D cuts off additional catchment contributing to Davidson Creek. The FWSS is estimated to provide 10% of total annual IFN flows in Years 6 through 17 and has up to a 60% likelihood of being needed during any given month. The FWSS is estimated to provide 40% of total annual IFN flows after Year 17 and has up to a 95% likelihood of being needed during any given month. The likelihood of needing flows from the FWSS to meet IFN increases in low flow months such as late winter and summer and is lower in freshet months.

18.7 Power and Electrical

The Project will require up to 110 MW of power once the full mine throughput is realized in Year 11. A 135 km, 230 kV overland transmission line will be constructed to connect to the BC Hydro grid at the Glenannan substation located near the existing Endako mine, 65 km west of Vanderhoof, BC. This point of interconnection has been assessed and was found to be technically viable in the System Impact Study and Facilities Studies completed by BC Hydro in 2014. A renewed System Impact Study was completed in June 2021 and a Concept Design phase is currently underway. The studies identified several upgrades to the substation and requirements for system reinforcement, which have been incorporated into the Project costs.

The transmission line was routed to make use of existing access and to cross recently logged areas as much as practicable along its alignment (Figure 18-10). The alignment was scrutinized to minimize impact on the environment and local stakeholders, and the design was optimized for reliability and constructability to reduce the effects of terrain on cost and construction. Stakeholder engagement and a comprehensive alternatives assessment has been completed for each of the alignment options. Different BC Hydro interconnection points were also contemplated throughout the design process before settling on the current arrangement

Figure 18-10: Proposed Power Transmission Line Routing



LiDAR terrain mapping, aerial photography, and site geotechnical assessments of the highest-risk terrain areas were used to complete the transmission line engineering along the preferred alignment. This information allowed the line designers to determine optimal structure locations that avoid areas of problematic soil conditions and other terrain hazards as much as possible. Geotechnical conditions were generally found to be favourable, and no significant problems are anticipated for the transmission line tower foundations.

Many sections of the proposed transmission line will be accessible by existing road infrastructure. A study of road access was completed and identified locations where new road access will be required for construction and maintenance during operations.

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 site protection scheme will interface with BC Hydro using "Point of Wave" control and load shedding as required and as identified in the BC Hydro System Impact Study.

The anticipated maximum connected electrical load for the Blackwater site is 110 MW for all three stages. The main substation will be in close vicinity to the mill grinding building, where the largest electrical loads are located, to minimize cabling costs and electric line losses. The main substation will consist of two power transformers, with the second added at Stage 3. The transformer secondaries will be connected to the primary distribution centre for power distribution around the site. Power will be distributed to the mine facilities at 25 kV, three-phase, 60 Hz through radial feeders originating at the primary distribution centre and routed around the site, depending on the distance, in cable and tray (short runs only), underground and on overhead power lines.

The primary power supply to the open pit will be a single 25 kV feed pole line running from the primary distribution centre at the main substation. Portable substations will transform the power to 4.16 kV for the mine shovels and drills. One 700 kW electric drill and one 34 m^3 1800 kW shovel will be used from mine start-up to Year 5. The remainder will be diesel. Additional electrical fleet will be added from Year 5.

A 25 kV feed and portable substations (25 kV/4.16 kV stepdown), will be used for the Tatelkuz Lake pump station, and the reclaim water system at the tailings facility. All other mine power will be supplied using pole-mounted transformers to step the voltage down from 25 kV to 600 V.

Emergency power 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 plant will consist of a minimum of two modular gensets rated at a combined nominal 3.5 MW. The temporary construction power generation equipment will be used as the source of backup power supply for the permanent camp.

18.8 Plant Site Earthworks

18.8.1 Introduction

Site preparation and bulk earthworks will consist of clearing and grubbing, removal of topsoil and grading of the site for the areas where the various facilities will be constructed. This is required to ensure that the surfaces at the various locations have been adequately prepared for the detailed construction, and to convey stormwater adequately away from the plant site.

The primary facilities at the plant site will include:

- Crushing area (primary, secondary and tertiary)
- Conveyor to the coarse ore stockpile
- Coarse ore stockpile
- Mill and process plant
- Collection pond
- Plant roads

18.8.2 Process Plant and Primary Facilities

The process plant area and primary facilities will be located as much as practical on in-situ material, i.e. constructed on cut rather than fill.

The general plant site area will be sloped at a minimum of 1% towards open channels. Areas around buildings will be sloped away from the buildings and foundations at a minimum of 2%. The minimum longitudinal slope of ditches and swales will be 0.3%.

The surface drainage around the plant site will be via open channels and culverts and reports to the contact water collection pond.

Diversion ditches around the plant site have been located in areas to minimize the catchment area for the contact water collection pond.

Sewage treatment facilities are included for the process plant facilities.

18.8.3 Collection Pond

The plant site is divided into sub catchments. Runoff from the sub-catchments will be conveyed away from the work area with drainage ditches and culverts and captured in the contact water collection pond.

The sedimentation that will be basin constructed during the site early works will be converted into a collection pond during construction activities at the site. The sedimentation basin was sized for the one in 200-year, 24-hour duration flood event and will have 1 vertical to 3 horizontal internal side slopes. The conversion to the collection pond will add a HDPE liner.

18.9 Buildings

18.9.1 Process Plant Buildings

A summary of the size of the process buildings is provided in Table 18-1. All buildings will be structural steel frame. Mill Building, Reagent Building, Gold Room and Truck Shop will be covered with insulated metal panels (IMP) roofing and, sidings and fitted with crane rails along their whole length and furnished with an overhead bridge crane. At least one large roll-up equipment door will be fitted with personnel doors for each building.

Table 18-1: Summary of Process Building Dimension and Crane Capacity

Building	Length (m)	Width (m)	Height top of side (even height)	Crane capacity (t)
Mill Building	52.5	30.0	26.5	70
Reagent Building	36.0	30.0	15.0	10
Gold Room	18.0	30.0	15.0	n/a
Truck Shop	55.0	30.5	17.0	50
Truck Shop Annex Building 1	14.5	15.0	5.0	n/a
Truck Shop Annex Building 2	28.0	15.0	5.0	n/a

18.9.2 Plant Offices

The plant offices will be adjacent to the main plant buildings and will house all plant operating and maintenance offices. The central control room will be in this complex, with closed circuit television coverage of all parts of the plant. The total area will be 401 m^2, with a change room and toilets. The construction will be modular, similar to the administration offices.

18.9.3 Laboratory

The laboratory building will be of modular construction, modified to allow solid floors where necessary for heavy equipment such as crushers or fire assay furnaces. The total area will be 640m^2 and will include toilets and a change room. Some area will be available for sample storage, but the main storage will be in a 100 m^2 unheated building adjacent to the main building.

18.9.4 Stores

Two fabric-covered buildings will be used as a plant store and a mining equipment store. They will be insulated. Heating using diesel heaters will be possible when required. Each will be 24 x 48 m (1,152 m^2). A small office (50 m^2) will be included for warehouse personnel.

18.9.5 Truck Shop Administrative Offices and Mine Offices

A staged approach is taken to construction and use of mine fleet maintenance facilities. Initially, a fabric covered 2-bay structure (30 x 44 m) will be built on the truck shop pad. In Year 1 of the Project a dedicated fabric covered wash bay will be added (23 x 32 m). Finally, in Year 4 of the Project, the truck shop pad will be expanded, and a steel covered 7-bay facility with additional warehousing and offices will be added (99 x 46 m). Both the fabric-covered and steel facilities will have overhead crane availability, with a 15 t capacity, and clearance in the bays for 230 t payload class rigid frame haul trucks.

A modular, combined mine office and change room complex will be built near the truck shop. The area of the complex will be 565 m^2.

18.9.6 Accommodation

The camp accommodations site (the camp) will be located approximately 2.8 km north east of the main mine processing building (approximately 5km by access road). Phase 1 of the camp will consist of buildings relocated from the existing exploration camp as well as seven new 38-person dormitory buildings and one kitchen/dinning/prep building, one gym and administration building. Phase 1 will accommodate 532 people. Each bunk room will include a bed, desk, closet and private bathroom. Following peak occupancy through construction, unused bunks will be decommissioned. Starting in Year 4, when camp population begins to rise, new 38-person dormitory buildings will replace the existing exploration camp buildings on an as needed basis.

Potable water will be supplied to the camp via TW13-02 which is approximately 240 m to the north. Water for fire protection and domestic use will be stored in two 63.5m^3 reservoirs on the camp pad. A package water treatment plant will treat the water to potable standards. Separate fire protection and potable water distribution systems will convey water from the reservoirs to the camp buildings.

Sanitary sewage will be collected from the individual camp buildings via a gravity system and a single lift station will transport sewage to the treatment plant located on the south west corner of the pad. Sewage from the kitchen will flow through a grease trap prior to entering the lift station. Treated effluent will be pumped to a disposal field located away from the camp pad.

Propane will be used to power both furnaces and hot water tanks. A 40,000 L propane tank located on the camp pad will provide 3-8.5 days of propane storage. Electricity will be supplied via the line power. A 1MW backup generator complete with a diesel fuel tank will also be located on the camp pad.

Storm water will be directed to ditches located at the edge of the pad. Ditches will convey stormwater to a sedimentation pond located to the north. Following settlement in the sedimentation pond, storm water will be released to the environment.

Vehicle access will be provided to the camp pad via a small access road off of the main haul road. A total of 40 parking spaces with electrical outlets are planned. Internal roads were designed to accommodate an intercity bus as well as food delivery vehicles.

18.10 Fuel

The fuel farm will be constructed in two phases to accommodate the different sized mine haul trucks and fleet sizes required for the mine operations. The Phase 1 fuel farm will have four 100,000 L double walled horizontal storage tanks, each with receiving/dispensing capabilities, and one diesel exhaust fluid system. The fuel and diesel exhaust fluid facilities will be located within a HDPE lined containment area, and the fuel farm will allow for filling of two Cat 789 rock trucks simultaneously. The storage tank capacity provides between 3.5 to 4.5 days of fuel usage for the first four years of operation until the fuel farm is expanded in Phase 2.

The Phase 2 fuel farm will be developed at the end of Phase 1 with an extension of the fuel farm pad. The expanded systems will have a diesel receiving/dispensing facility, a diesel exhaust fluid storage and dispensing module, a lube and oil module, four 100,000 L double-walled horizontal storage tanks from the Phase 1 system, and one new 488,000 L vertical storage tank. The fuel tanks will be located within a HDPE lined containment area that will be located on the new extension and the remaining modules will be in the original Phase 1 lined facilities. The system will allow for filling of two Cat 793 rock trucks simultaneously and will provide a minimum storage capacity of 4.5 days' worth of diesel fuel.

18.11 Fire and Potable Water Supply and Distribution

Water well TW13-02, within 240 m of the proposed camp accommodation site, will be the source of water supply for the temporary and operations camps and has sufficient flow to support the potable and fire water requirements for the camp and plant site. A test well in the area indicated that a single well would have the potential for a flow rate of more than 16

m^3/h, which meets the estimated average demand for the camps at their peak occupancy during construction. With both wells operating, the system will be able to refill the fire water tank within eight hours, in accordance with fire water requirements.

18.12 Security

The proposed mine site will employ a number of security systems 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 process control server room. Visual security will be provided by a network of IP-enabled 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 staff will be provided with radios for wireless communication and may be provided with tablets for access to the camera system.

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. This fiber aerial cable will land at the mine substation.

From the substation, the fiberoptic backbone cable will be distributed to the accommodations camp, main mine infrastructure area, and pit. Aerial all-dielectric self-supporting cable (along 25 kV power lines) will be supplied to the pit and camp areas, and the main mine infrastructure area will be fed with armoured fiber optic cable. In the main mine infrastructure area, fiber optic cable will be distributed to the various facilities and buildings by underground, tray, or conduit. The fiber optic architecture will use a ring configuration, whereby communication integrity will not be compromised when a part of the fiber backbone is damaged.

For the remaining mine areas not serviced by fiberoptic (fuel storage, reclaim and booster pumps, water transfer pond, central water transfer pond, interim and environmental control dam, well TW13-2, gatehouse, freshwater reservoir, and fresh water supply system) a wireless mesh communications network will be employed.

During the construction phase prior to the fiber optic line reaching site and being installed, a satellite system will provide internet service to the Project.

18.14 Comments on Sections 18

The QPs note:

- The overall Project facilities and major infrastructure include the mine site area, TSF, camp site, main access road, low grade ore and waste stockpiles, power line, and water pipeline from Tatelkuz Lake;
- Project infrastructure was designed to have a minimal footprint;
- WRSFs are planned to be located near the open pit and within the same or adjacent sub-drainages;

  The Blackwater site will be accessed via the FSR, which connects to Provincial Highway;16 near Vanderhoof. In addition, a new 15.6 km access road will be constructed from 124 km of the FSR to the plant site;

- Power will be supplied by connection to the BC Hydro grid. The line will follow existing resource roads and other previously-disturbed areas as much as practicable;
- A fibre-optic cable will be installed along with the main transmission line to provide high bandwidth telecommunications access to the site;
- Based on an extensive geochemical evaluation, some of the waste rock and the tailings will be classified as PAG and/or ML. A schedule was developed to place the different classes of material either within the TSF or in the WRSF, depending on the characterisation of the material;
- The TSF will comprise two adjacent sites, TSF C and TSF D, which are staged to support initial mine operations and the two subsequent expansion stages. The dam construction materials balance is integrated with the mine plan to minimize the need for additional external borrow material sources following initial site establishment and early TSF construction;
- Additional site investigation work was completed in 2020-2021 to further investigate the geotechnical and hydrogeological conditions at Main Dam C, the Interim Environmental Control Dam, the Fresh Water Reservoir, at the proposed Plant Site location, and in the vicinity of potential overburden and esker borrow source areas. These investigations have confirmed the feasibility of these locations for the proposed facilities;
- Drainage from the majority of the mine area flows by gravity into the TSF, following natural topographical drainages mapped for the Project; which simplifies water management, spill control, and mine closure. Mine affected runoff within the project area will be collected and stored within the TSF and used to inundate the PAG/ML waste rock and tailings solids to limit the potential for ARD and ML. Supernatant water will be recycled for use as process water.
- Surplus water not required to support mine operations will be sampled and analyzed, comparing to applicable water quality criteria, and if compliant, will be used to augment flow in lower Davidson Creek. Planned installation of the water treatment plants at the start of operations will enhance water management flexibility and allow for treatment of mine site contact water to meet discharge criteria, if required.
- The feasibility study is supported by the detailed design of the Stage 1 TSF and associated water management structures required for the start of mine operations;

19 MARKET STUDIES AND CONTRACTS

19.1 Market Studies

No formal marketing studies have been 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 particular time. The gold and silver market is very liquid with a large number of well-informed potential buyers and sellers active at any given time.

19.2 Commodity Price Projections

Commodity pricing for the 2021 FS economic analysis uses metal prices and exchange rates consistent with current consensus estimates provided by a consortium of banks in Q1 and Q2 2021.

19.3 Contracts

Artemis 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.

19.4 Gold Stream Agreement

Artemis have entered into a gold stream agreement with New Gold whereby New Gold will purchase 8.0% of the refined gold produced from the Blackwater Project. Once 279,908 ounces of refined gold have been delivered to New Gold, the gold stream will reduce to 4.0%. New Gold 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. In the event that commercial production at Blackwater is not achieved by the 7th, 8th, or 9th anniversary of closing of the acquisition of the Project by Artemis, New Gold will be entitled to receive additional cash payments of C$28 million on each of those dates. New Gold maintains a security interest over the Project in connection with the gold stream agreement.

19.5 Comments on Section 19

In the opinion of the QP, Artemis will be able to market gold and silver produced from the Project at the projected prices. Sales contracts that could be negotiated are expected to be within industry norms. The commodity pricing and exchange rate assumptions are suitable for use in the cash flow analysis in Section 22.

20 ENVIRONMENTAL STUDIES, PERMITTING, AND SOCIAL OR COMMUNITY IMPACT

20.1 Licensing and Permitting

The legislative and regulatory framework for metal mining in BC and Canada is well-established. In BC, major mine projects trigger a review under federal and provincial impact legislation to assess a project's potential environmental, social, economic, health, and cultural and heritage impacts. Mine projects in BC undergo a detailed permitting process and both provincial and federal agencies regulate all project phases. The regulatory requirements of the Project are well-defined as presented in the following subsections.

20.1.1 Federal/Provincial EA Processes

The Project was subject to review under the federal *Canadian Environmental Assessment Act, 2012* and the provincial *Environmental Assessment Act* (2002). Review of the Project's application for an Environmental Assessment Certificate / Environmental Impact Statement (Application/EIS; New Gold 2015) began in 2015. Numerous environmental and engineering studies were undertaken and reports prepared to support the Application/EIS. The BC Environmental Assessment Office (EAO) established a working group to participate in the environmental assessment (EA) review, which 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 (EMLI), Ministry of Environment and Climate Change Strategy (ENV), Ministry of Forests, Lands, Natural Resource Operations and Rural Development (FLNRORD) and Northern Health), Indigenous Groups and local governments. The public also had opportunities to review and provide input on the Application/EIS. Documentation produced during the EA review can be found online at the Environmental Assessment Office's ePIC website (https://www.projects.eao.gov.bc.ca/p/588511c0aaecd9001b82522a/project-details).

New Gold received Environmental Assessment Certificate #M19-01 on June 21, 2019 under the provincial *Environmental Assessment Act* (2002) (EAO 2019b) and a Decision Statement (DS) on April 15, 2019 under the federal *Canadian Environmental Assessment Act, 2012* (CEA Agency 2019b). The Project is supported by the Lhoosk'uz Dené Nation and Ulkatcho First Nations, who submitted letters of support for the Project following completion of the EA process. The mine site is located within the traditional territories of these two Indigenous groups.

In August 2020, Artemis acquired the mineral tenures, assets and rights in the Blackwater Project that were previously held by New Gold. On August 7, 2020, the Certificate was transferred to Artemis under the provincial *Environmental Assessment Act* (2018). On September 25, 2020, the Impact Assessment Agency of Canada verified that written notice had been provided within 30 days of the change of proponent as required in Condition 2.16 of the DS, and that a process had been initiated to amend the DS.

The Project's EAC 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 DS includes 172 binding conditions, which identify requirements for plans to offset project impacts, consultation requirements for offset plans and follow-up programs, and specific mitigation measures. Artemis is addressing these conditions in accordance with the timelines specified by the EAC and DS.

20.1.2 Existing Permits or Approvals

BW Gold holds a number of authorizations as presented in Table 20-1. Of note, MA Permit M-246 and Environmental Management Act (EMA) Permit #110602 were issued in June 2021, which authorizes site preparation and land-clearing, the first step required for mine development. Special Use Permit SP001 was issued in July 2021 to authorize construction of the Mine Access Road (located off the mine site).

Table 20-1: Blackwater Gold Project Existing Authorizations

Authorization (Issue Date)	Authorizing Legislation	Purpose
Permit Mineral Exploration (MX)-13-177 (September 14, 2005)	*Mines Act*, RSBC 1996, c. 293.	Authorizes access roads, trails, helipads, exploration surface drilling (valid to December 31, 2022).
Municipal Wastewater Authorization # 105882 (May 31, 2012)	*Municipal Wastewater Regulation*, BC Reg. 87/2012.	Authorizes discharge of treated effluent to a septic field.
Water System (Well 3 ID 31679; June 16, 2012)	*Drinking Water Protection Act*, SBC 2001, c 9.	Authorizes seasonal operation of groundwater well to supply potable water to Blackwater exploration camp.
Permit PE 106530 (May 9, 2013)	*Environmental Management Act*, SBC 2003, c.53	Authorizes discharge of air contaminants from camp incinerator.
Permit to Operate Food Service Establishment (August 1, 2018)	*Food Premises Regulation*, BC Reg. 210/99.	Authorizes operation of provision of food services at Blackwater exploration camp.
Occupant License to Cut L51102 (May 22, 2018)	*Forest Act*, RSBC 1996, c.157	Authorizes cutting and removal of timber (valid to December 31, 2022).
Free Use Permit 13-250 (June 13, 2018)	*Mines Act*, RSBC 1996, c. 293.	Authorizes cutting of 50 m^3 timber and use of timber to facilitate mining operations (valid to December 31, 2022).
MX-13-319 (November 13, 2020)	*Mines Act*, RSBC 1996, c. 293.	Authorizes mineral exploration in western area of Project area (November 12, 2025).
Environmental Assessment Decision Statement (April 15, 2019)	*Canadian Environmental Assessment Act, 2012*, SC 2012, c.19, s. 52.	Grants Decision Statement for the Project, subject to conditions.
Environmental Assessment Certificate (EAC) #M19-01 (June 21, 2019)	*Environmental Assessment Act*, SBC 2002, c. 43.	Grants EAC for the Project as described in the Certified Project Description and Table of Conditions.

Authorization (Issue Date)	Authorizing Legislation	Purpose
Permit M-246 (June 22, 2021)	*Mines Act*, RSBC 1996, c. 293.	Approves Blackwater early works program including surface disturbance and works, within Permitted Mine Area encompassing approximately 1,018.9 ha.
Permit 110602 (June 24, 2021)	*Environmental Management Act*, SBC 2003, c.53	Authorizes discharge of treated stormwater effluent to ground from early stage construction activities.
Occupant License to Cut L51102 Amendment (July 13, 2021)	*Forest Act*, RSBC 1996, c.157	Cancels and replaces existing Exhibit A May 22, 2018 issued for Occupant License to Cut L51102 to remove area to allow new Occupant License to Cut L51818 to be issued for mine site development without resulting tenure over tenure.
Occupant License to Cut L51817 (July 14, 2021)	*Forest Act*, RSBC 1996, c.157	Authorizes cutting and removal of 866 m^3 of Crown timber within Special Use Permit SP0001 license area (valid to July 13, 2026).
Occupant License to Cut L51818 (July 14, 2021)	*Forest Act*, RSBC 1996, c.157	Authorizes cutting and removal of 37, 982 m^3 of Crown timber within M-246 permitted area; valid to July 13, 2026).
Special Use Permit SP0001 (July 14, 2021)	*Provincial Forest Use Regulation*, BC Reg. 176/95	Authorizes construction and maintenance of the Mine Access Road (portion of road outside of mine site (valid to July 13, 2031).

20.1.3 Anticipated Federal Permits or Approvals

In order to construct and operate the Project, federal authorizations will be required as presented in Table 20-2. Preparation of key federal permit applications is underway with a target submission of the major permits allowing for construction on the mine site to relevant agencies in Q4 2021.

Table 20-2: **Anticipated Federal Permits, Licences and Authorizations**

Legislation	Authorization	Purpose
Explosives Act, RSC 985, c. E-17	License	Manufacture and use of explosives
Fisheries Act, RSC 1985, c. F-14	License	Death of fish, harmful, alteration, disruption of fish habitat (HADD), fish collection for scientific purposes
Nuclear Safety and Control Act, SC 1997, c.9	License for Nuclear Substance, License to use Radiation Device	Use, possession or importing of a prescribed substance in a device such as analyzers, chromatographs, calibrators, fixed and portable gauges, industrial radiography, logging, detectors, etc.
Radio Communications Act, RSC 1985, C, R-2	License	Issuance and operation of designated frequency
Transportation of Dangerous Goods Act, SC 1992, c. 34	Permit	Transport of dangerous goods

The Project also requires an amendment to Schedule 2 of the Metal and Diamond Mining Effluent Regulations to allow deposition of deleterious mine waste in waters frequented by fish. Pursuant to Environment and Climate Change Canada Guidelines for the Assessment of Alternatives for Mine Waste Disposal (ECCC 2016), Artemis prepared an assessment of alternatives to identify preferred candidate alternatives to store tailings, waste rock, and low grade ore generated during the Project. Particular to tailings management, the assessment also considered the best available technology and best available practices. The Project's alternatives assessment report was revised to incorporate ECCC comments. Artemis also prepared a plan to compensate or offset impacts to fish and fish habitat due to mine waste deposition. The updated alternatives assessment report and associated fish compensation plan were filed with ECCC in February 2021. Environment and Climate Change Canada held a public comment period on these documents and proposed Schedule 2 amendments from June 9, 2021 to August 30, 2021.

20.1.4 Anticipated Provincial Permits or Approvals

In order to construct and operate the Project, provincial authorizations are required as presented in Table 20-3. The primary provincial permits to proceed to the full mine construction and operations are issued under the MA (Permit approving Mine Plan and Reclamation Program) and the EMA (effluent, air discharge permits). A joint MA/EMA permits application is planned to be submitted in Q4 2021.

Table 20-3: Anticipated Provincial Permits, Licences and Authorizations

Legislation	Authorization	Authorization Purpose
Drinking Water Protection Act, SBC 2001, c 9.	Water System Construction Permit Water System Operating Permit	Construction and operation of potable water supply systems for operations camp and processing plant
Public Health Act, SBC 2008, c.28	Application for Health Approval	Building and operating a food service facility

Legislation	Authorization	Authorization Purpose
Environmental Management Act, SBC 2003, c.53	Effluent Discharge Permit Air Discharge Permit	Effluent (sediment control ponds, tailing storage facility, water treatment plants), fugitive dust and air contaminant (processing plant, assay lab) discharges
Hazardous Waste Regulation, BC Reg. 319/2004	Hazardous Waste Registration	Hazardous waste transfer facility, plant truck shop
Environmental Management Act, SBC 2003, c.53	Fuel Storage Registration	Fuel storage
Forest Act, RSBC 1996, c.157	Occupant Licences to Cut	Cutting and removal of timber on Crown land
Forest Act, RSBC 1996, c.157	Road Use Permit(s) (Kluskus Forest Road (FSR) and other FSRs)	Use of FSR
Heritage Conservation Act, RSBC 1996, c. 187	Archaeological Impact Assessment/Site Alteration Permits	Heritage site alteration and inspection
Land Act, RSBC 1996, c. 245	License of Occupation	Occupancy and use of Crown land offsite (transmission line, pumphouse site and waterline/electrical distribution for pump from Tatelkuz Lake to mine site)
Land Act, RSBC 1996, c. 245	Temporary use permit	Temporary access roads for offsite infrastructure (support transmission line construction)
Mineral Tenure Act, RSBC 1996, c.292	Mining lease	Production of minerals
Land Title Act, RSBC 1996, c. 250	Statutory Right of Way	Easement for transmission line
Mines Act, RSBC 1996, c. 293	Permit Approving Mine Plan and Reclamation Program, Ex	Approves mine plan and reclamation program
Mines Act, RSBC 1996, c. 293	Explosives Storage and Use Permit	Use, care and transport of explosives
Municipal Wastewater Regulation, BC Reg 87/2012	Municipal Waste Registration	Discharge of municipal wastewater
Safety Standards Act, SBC 2003, c. 39	Permit	Transmission line connection

Legislation	Authorization	Authorization Purpose
Water Sustainability Act, SBC 2014, c.15	License (Section 9/7)	Diversion, storage or use of surface water or groundwater for one or more purposes; construction of water storage dams[1], groundwater wells
Water Sustainability Act, SBC 2014, c.15	Approvals/Notifications of changes in or about a stream (Section 11)	Changes in or about a stream for the water line and crossings associated with the water line access road, transmission line
Wildlife Act Permit Regulation, BC Reg. 253/2000	Permit	Designate no shooting area, fish collection permits, relocate wildlife during construction

20.2 Environmental Settings and Studies

The Project's previous owners conducted comprehensive baseline studies to support the Application/EIS and BW Gold has continued to collect meteorology, surface water quality, groundwater and hydrology data. Table 20-4 summarizes the Project's baseline studies.

Table 20-4: Blackwater Project Baseline Studies

Project Baseline Studies	Years
Meteorology, Atmospherics and Noise	2011 - current
Air Quality	2012, 2013
Noise	2011 - 2013
Geochemistry	2013, 2020
Surface water quality	2011 - current
Surface water quantity	2011 - current
Groundwater quality	2011 - current
Groundwater quantity	2012 - current
Sediment quality	2011 - 2013
Fish and aquatic resources	2011 - 2013, 2017, 2021
Wildlife	2013, 2016, 2018
Soils, terrain and surficial geology	2013, 2017

[1] At the time of the writing of this document, the BC government had not confirmed if the freshwater reservoir and dam will be authorized under the *Mines Act* or the *Water Sustainability Act.*

Project Baseline Studies	Years
Wetlands	2011 - 2013, 2017, 2021
Vegetation and Ecosystems	2011 - 2013, 2017
Archaeology	2011 - 2013
Economic, Social and Cultural	2013, 2019
Land Use	2013

[1] Confidential studies are not included.

Conditions

20.2.1 Climate and Atmospheric

Climate and atmospheric conditions are summarized in Section 5.2. Meteorological data continues to be collected on site to support air quality and water management. The Project is located in an area with few sources of emission nearby.

20.2.2 Geochemistry

A geochemical characterization report was completed by AMEC (2014). All overburden samples from the open pit, plant, TSF and access road area were classified as NAG.

The waste rock from the open pit will be segregated and managed based on ML/ARD characteristics. Section 16.6.1 provides details of the geochemical characterization of waste rock.

The results of kinetic testing indicate time until acid generation will be longer under field conditions than in laboratory testing, metal loads under neutral pH conditions are lower than under acidic conditions, and underwater storage of waste rock significantly reduced metal leaching. Selenium and mercury were generally less than detection limit in all leaching tests and not expected to be of concern. The geochemical characterization results were used to develop source terms and predictive water quality models for several mine features. The results of testing and modelling were provided as input to the overall site water quality modelling efforts.

20.2.3 Geohazards

The Project is situated within central BC, where the level of recorded historical seismic activity has been low. A seismicity assessment was carried out, including a review of the regional seismicity and a probabilistic hazard analysis (KP, 2021d). Sections 5.6 and 18 details the incorporation of regional seismicity into project infrastructure.

The majority of the Project area is relatively flat to gently sloping, with slope angles of generally less than 50% (KP 2013) in conjunction with the slope stability assessment at the Project area, did not identify any clearly discernible, existing snow avalanche paths in local treed areas.

Subsidence is not anticipated to occur at the mine site through either natural causes or mining activities.

The closest volcano to the Project area is the dormant Nazko Cone, located approximately 90 km east of the Project. In the unexpected circumstance that the Nazco Cone or Satah Mountain became active, the distance of the proposed mine site from these volcanoes makes it unlikely that the proposed mine site and personnel would be in danger from the direct effects of a volcanic eruption.

20.2.4 Soil Development

Soil mapping was undertaken in support of the Application/EIS (New Gold, 2015) and additional soil information has been gathered during the course of geotechnical site investigations. In general, soil types vary according to elevation and drainage. The mine site is composed of well- to imperfectly-drained soils occurring on medium to moderately coarse-textured parent materials. In areas of wetter edaphic conditions, poorly developed soils due to saturation were observed. There are sufficient salvageable soil resources to facilitate reclamation of the mine site at closure.

20.2.5 Hydrology/ Surface Water Quantity

The Project is located in the sub-alpine areas north of Mount Davidson, primarily in tributaries of Chedakuz Creek, including Davidson Creek and Creek 661. Chedakuz Creek drains from Kuyakuz Lake through Tatelkuz Lake directly to the Nechako Reservoir. The confluence of Creek 661 and Chedakuz Creek is upstream of Tatelkuz Lake, whereas the confluence of Davidson Creek with Chedakuz Creek is downstream of Tatelkuz Lake. The discharge hydrographs for the flow monitoring stations are typically characterised by high spring snowmelt-driven flows, low summer flows sustained by groundwater inflows and periodic rainfall events, followed by smaller autumn rainfall events, and very low winter discharge as a result of cold temperatures and freezing conditions. Peak stream flows occur in May, with low flows in August and September as well as November through March.

20.2.6 Groundwater Quantity

Groundwater in the Project area flows from recharge zones located in topographic highs, Mount Davidson and surrounding highlands, and then discharges in the valleys of Davidson Creek, Creek 661, and Creek 705. Groundwater discharges into these creeks as baseflow, providing the majority of surface water flow in the winter and early spring months.

The main flow pathways in bedrock are through the highly fractured zones of rock. The upper 10 m to 20 m of bedrock is inferred to be highly fractured throughout the region and is expected to yield higher groundwater flow values than the underlying, more competent bedrock. A large zone of fractured bedrock, nearly circular in plan view, extends almost 500 m from the bedrock surface on the southeastern slope of Mount Davidson. This fractured rock, termed the broken zone, will be excavated by the planned open pit mine. Time-series water level measurements indicate water levels display some seasonal variability; some sites show high variability (2 m to 7 m), but most sites vary less than 1 m seasonally, increasing during the spring freshet (KP 2021c). Artesian conditions are present in the Davidson Creek mainly during the late spring and early summer.

20.2.7 Surface Water Quality

Surface water and sediment quality was collected for the watersheds of Davidson Creek, Creek 661, Turtle Creek, and Creek 705; tributaries flowing in to the south side of Tatelkuz Lake; Chedakuz Creek from confluence with Creek 661 to Tatelkuz Lake; Chedakuz Creek from Tatelkuz Lake to confluence with Turtle Creek (ERM 2021). Sampled lakes included Kuyakuz Lake, Tatelkuz Lake, and Snake Lake in the Chedakuz Creek area; Lake 1682 in the Davidson Creek headwaters; and Lake 1538 and Lake 1428 in the Creek 705 headwaters.

During freshet, stream and river waters have neutral to low alkalinity, low hardness, low total dissolved solids, and low concentration of nitrogen species; however, these increased during low flow periods. Mean metal concentrations were low, typically one to several orders below guidelines (BC and Canadian Council of Ministers of the Environment (CCME) Water Quality Guidelines). In general, lakes are well-mixed, with minimal difference among samples collected at different depths. Lakes were near-neutral to slightly alkaline with pH, alkalinity, hardness, and total dissolved solids slightly higher in the large downstream Kuyakuz Lake and Tatelkuz Lake, compared to the headwater lakes at higher elevations. Total suspended solids, turbidity, nutrients and anions were typically low with minor seasonal variability.

20.2.8 Groundwater Quality

Groundwater quality baseline data has been collected within the proposed mine site footprint and region from watersheds potentially connected with the mine activities. Wells were strategically placed to provide observations both up-gradient of the proposed mine site, and down-gradient to capture potential groundwater impacts (KP, 2013g).

Groundwater in the project area is generally characterized as neutral to basic pH, alkaline with strong buffering capacity and varied hardness from soft to hard. In general, water hardness is higher in the deep monitoring well compared to the shallow well at sites with paired wells. Hardness, alkalinity, and total dissolved solids TDS are variable in the wells, and no spatial pattern within the Davidson Creek watershed is observed.

There were no guideline exceedances for the major anions (bromide, chloride, fluoride, sulphate) for samples from any of the monitoring wells. Less than 10% of the samples in all wells had reportable concentrations of bromide and chloride. Sulphate concentrations were higher in the deeper well of paired shallow and deep wells, but no difference is seen in fluoride concentrations in the deeper and shallow paired wells. Nutrient concentrations were below drinking water and freshwater aquatic life guidelines in all wells but two; both samples exceeded the freshwater aquatic life long-term chronic guideline for nitrite. Baseline groundwater quality exhibited elevated concentrations of aluminum, arsenic, cadmium, chromium, cobalt, iron, manganese, mercury and zinc, with some parameters at some of the wells intermittently exceeding provincial water quality guidelines for the protection of freshwater aquatic life.

20.2.9 Fisheries and Aquatic Habitat

All watersheds in the fish and fish habitat in the mine site contain fish-bearing streams and lakes. Fish habitats are typical for central BC, steep, sub-alpine headwater tributaries of poor-quality habitat draining to lower gradient reaches of higher-quality habitat that flow into large, overwintering lakes (e.g., Tatelkuz Lake). Based on sampling, 12 species including kokanee, rainbow trout, mountain whitefish, northern pikeminnow, longnose sucker, largescale sucker, burbot, brassy minnow, lake chub, slimy sculpin, longnose dace and white sucker, were identified. Rainbow trout are the only species identified to use the mine site area streams. A Fish Habitat Offsetting Plan and a Fish Habitat Compensation Plan are required to offset impacts to fish and fish habitat (Section 20.6.2.1).

White sturgeon (Schedule 1 of the Species at Risk Act as Endangered and provincially red-listed (S2) was identified in the Nechako River, which will be crossed by the transmission line. There are no anticipated Project-related impacts to sturgeon.

20.2.10 Terrestrial Ecosystems

The majority of the mine site consists of sub-boreal spruce, Engelmann spruce, and subalpine fir, although there are also areas containing Lodgepole pine that have been severely affected by mountain pine beetle and have been subject to accelerated salvage logging. At higher elevations, forestry activity is limited and mountain pine beetle infestation is less predominant. The transmission line runs through a variety of ecosystems resulting in a greater diversity of vegetation.

There is whitebark pine (*Pinus albicaulis*) in the Project area, which is listed as endangered on Schedule 1 of the Species at Risk Act. This species occurs in dry high elevation sites, such as Boreal Alta-Fescue Alpine habitats on Mount Davidson. A Whitebark Management Plan is being developed (Section 20.6).

Sensitive ecosystems in the Project area include riparian, old growth forest, and sparsely vegetated areas. Wetland classes in the Project area include wet bogs, fen wetlands, marsh wetlands, swamp wetlands, shallow waters, and ponds. BW Gold is required to develop a Wetland Monitoring and Offsetting Plan (Section 20.6.2.2) to offset the Project's impacts on wetlands. No red- or blue-listed ecosystems were identified in the mine site during the baseline studies. Twelve ecosystems-at-risk were identified in the area of the transmission line and Kluskus FSR. Rare plant surveys did not identify any rare plants.

20.2.11 Wildlife Species

The landscape provides habitat for a variety of wildlife species such as ungulates, bears, furbearers, amphibians, migratory birds, other forest and grassland birds, raptors, and water birds. Locations with the highest diversity are typically associated with wetlands and a mixture of mature and old-growth forests. The wildlife habitat suitability ratings identified moderate and high value habitat of forest and grassland birds within the Project area.

The Project overlaps with the eastern boundary of the Tweedsmuir local population unit of Southern Mountain caribou, which is listed as Threatened under the Species at Risk Act. Artemis has developed a Caribou Mitigation and Monitoring Plan (CMMP) to offset the Project's impacts on caribou and its critical habitat (Section 20.6.2.3). Four amphibian species were observed in the Project area including Western toad (Anaxyrus boreas), Columbia spotted frog (Rana luteiventris), wood frog (Lithobates sylvatica), and long-toed salamander (Ambystoma anabystoma). A total of 97 forest and grassland bird species were detected within study areas, including five species of conservation concern. Eighteen species of raptors were detected, including two species of conservation concern. A total of 23 species of water birds were detected including one species of conservation concern.

Ten different mammals were observed during baseline surveys and the most frequently detected species were moose (Alces alces), followed by snowshoe hare (Lepus americanus), lynx (Lynx canadensis), and wolf (Canis lupus). Grizzly bear (Ursus arctos) and black bear (U. americanus) were detected within the Project area. Acoustic bat surveys detected nine species of bat, including two species of conservation concern, within the Project area:

A total of 85 species of butterfly and dragonfly were detected within the Project area, including three species of conservation concern.

20.2.12 Economic, Social, and Cultural Setting

Economic activity in the Project region includes the forestry sector (logging, wood products and by-product manufacturing), public sector (health care, social services, education and public administration), and to a lesser degree, agriculture and ranching, mining, and the service sector (AMEC, 2013a; Statistics Canada, 2016). The forestry sector contributes about 2% to the provincial gross domestic product (GDP) (Statistics Canada, 2021, FLNRORD, 2019). The mining and oil and gas sectors region provide <4% of the jobs in the Regional District of Bulkley-Nechako and 1% of the jobs in Prince George (Statistics Canada 2016).

While regional population has been declining over the past decade, the population of some communities in the region has been stable or increased. Housing in the region is considered to be affordable and house prices are below the provincial average. Regional and community infrastructure is in good condition and of suitable capacity to meet future growth. Local and regional services, including schools, health care, social and emergency services operate below capacity and are able to meet the current demand. However, local communities typically have lower income levels, higher crime, more at risk youth, and lower overall well-being compared to the provincial averages.

There is a variety of recreational opportunities in the region, including snowmobiling, skiing, sledding, fishing, resident hunting, hiking and horseback riding. Other land uses include guide outfitting, trapping, and eco-tourism (AMEC, 2013c).

Current use of land and resources for traditional purposes by Indigenous peoples includes fishing, hunting, trapping, gathering, and the use of habitations, trails, and cultural and spiritual sites (AMEC, 2013c). Mount Davidson is identified by UFN and LDN community members as a sacred site. The mountain holds traditional healing powers and historically has also been an important hunting and berry-picking site (Keefer Ecological Services Ltd., 2019). Historically and culturally significant trails within 20 km of the Project include the Messue Wagon Road, Messue Horse Trail/Kluskus Bypass, and the Alexander Mackenzie Heritage Trail. These trails were used in earlier periods as Grease trails by Indigenous nations (ERM, 2016).

Through Project employment and procurement, the Project is anticipated to provide significant economic benefits to regional, provincial and federal economies. These benefits include employment and income for direct Project workers and contractors, and those resulting from spin-off opportunities, such as employment in supplier industries (indirect employment) and employment further down in the supply chain (those resulting from workers spending their income, referred to as induced employment). Additional benefits will include procurement opportunities resulting from spin-off opportunities, higher provincial GDP, and an increase in local government and provincial tax revenue. Communities likely to experience direct and indirect socio-economic Project effects include Vanderhoof, Prince George, Fraser Lake, Burns Lake, Fort St James, and Quesnel, and Indigenous communities proximal to the Project.

20.2.13 Land Use Setting

The Project is located in central BC, a region home to 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.

Three guide-outfitter certificates overlap the proposed mine site:

- Batnuni Lake and Guide Outfitters (Certificate 601039) overlaps 7% of the planned mine site;
- Fawnie Mountain Outfitters (Certificate 601097) overlaps 0.7% of the planned mine site;
- Euchiniko Lakes Guest Ranch (Certificate 500929) overlaps 0.01% of the planned mine site.

Three provincially registered traplines overlap the proposed mine site. Trapline TR0512T027 overlaps 12% of the planned mine site, TR0512T014 overlaps 1%, and TR0601T003 overlaps 1.5 %.

No Agricultural Land Reserves overlap the proposed mine site. One range tenure (RAN075154) overlaps 7% of the planned mine site.

20.2.14 Land Use Planning Context

The Project is subject to the provisions of the Vanderhoof Land & Resource Management Plan (Government of BC 2007), approved by the Province of BC in January 1997 and reviewed in 2005. The LRMP divides 1.38 million hectares into six different Resource Management Zone categories. Mineral exploration and development is allowed in all Resource

Management Zones occupied by the Project area and any specific requirements are met through the Project's management plans.

20.3 Water Management

Water management objectives within the context of the receiving environment include meeting Instream Flow Needs (IFN) and water quality guidelines or approved Science-Based Environmental Benchmark (SBEBs) in Davidson Creek and Creek 661.

The purpose of the Mine Site Water and Discharge Monitoring and Management Plan (MSDP) is to:

- Provide operational management and monitoring procedures for each phase of the LOM as well as contingency measures for the effective interception, conveyance, diversion, storage, and discharge of contact and non-contact water on the mine site.

- Monitoring on site contained within this plan is to comply with relevant legislation and other requirements to identify, assess, evaluate and mitigate potential impacts to receiving surface and groundwater quality by continually monitoring process water and effluent discharge to reduce the likelihood of a potential non-compliance event as a result of the mining operations conducted at the Project.

- In parallel with the Aquatic Effects Monitoring Plan (AEMP), the MSDP provides 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 occurrences 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.

20.3.1 Potential Environmental Risks and Mitigations

The Project is predicted to alter existing surface water quantity and quality, and groundwater quantity and quality. Application of the water management system detailed in Section 18.6 is anticipated to mitigate water quality issues. Water quality is predicted to not exceed BC or Canadian Council of Ministers of the Environment water quality guidelines for the protection of aquatic life (WQG-AL) in the receiving environment. A SBEB, is proposed for dissolved aluminum in Davidson Creek and Creek 661 owing to naturally occurring conditions. Predicted water quality does not exceed the proposed dissolved aluminum SBEB in Davidson Creek and Creek 661.

20.3.2 Monitoring

The MSDP also provides the 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.

20.4 Waste Management

20.4.1 Tailings Storage Facility Management

The TSF will permanently store tailings from ore processing and PAG waste rock from stripping and open pit mining. The TSF will also provide water for processing and support mine site water balance management. The TSF design is detailed in Section 18.5.

20.4.1.1 Potential Environmental Risk Management and Monitoring

20.4.1.1.1 Water Quality and Quantity

Surface water and groundwater (seepage) effluent will be discharged from the TSF. During construction, predicted solute concentrations for the TSF C starter pond are low and near background values as the loading sources to this pond are limited mainly to discharges from background/non-contact flow terms. During operations, the predicted concentrations for ammonia, sulphate, and cyanide loadings are anticipated to be elevated above background values in the TSF C and D ponds, primarily due to loadings from process water. Metal concentrations (e.g., zinc) are anticipated to be elevated due to the placement of PAG/ML waste rock within the TSF.

In closure, ammonia and cyanide concentrations in the TSF C and D ponds are predicted to decrease rapidly to values close to background and well below screening guidelines. Specifically, the cessation of ore processing and cyanide usage (the dominant source of ammonia and cyanide), coupled with other removal processes (e.g., oxidation), will result in rapid concentration declines. Sulphate concentrations will also decrease notably but are expected to remain slightly elevated in closure due to loadings from tailings beach runoff, TSF D pump-back water, and the water from the waste stockpile. Applying a cover to the tailings beach and a reduction in pumping from TSF D pond will result in lower sulphate concentrations in post-closure.

To mitigate the effects of water quality and quantity, surplus water not required to support mine operations will be sampled and analyzed, comparing to applicable water quality criteria, and treated if required prior to discharge to the environment.

The following elements contribute to the environmental mitigation related to the TSF and associated structures:

- IECD – seepage from TSF C pumped back to TSF C (Phase 1 mine operations);
- ECD – seepage from TSF D is pumped back to TSF D and pit lake (Phase 2 and 3 mine operations, and closure);
- Membrane WTP – used to treat surplus TSF supernatant, if required, prior to discharge to the WMP (all phases of mine operations);
- Membrane WTP – used to treat ECD water, pit water, and TSF pond water, if required, prior to discharge to Davidson Creek (post-closure);
- Metal WTP – used to treat stockpile runoff and pit dewatering, if required, prior to use as raw water for ore processing or discharging to the WMP); and
- Lime neutralization – used to treat contact water flows from the low-grade ore stockpile.

Discharges from the TSF will be monitored through the MSDP.

20.4.1.1.2 Dust Generation

Air quality and fugitive dust from these sources will be managed through the Air Quality and Dust Management Plan (AQDMP; Section 20.5).

Dust generation from the TSF will be monitored through the Air Quality and Dust Monitoring Program has been integrated into the AQDMP. Visual monitoring of dust will be undertaken. These observations, together with meteorological conditions and mitigation efforts taken to deal with any problem, will be recorded and included in monthly and annual reports. Monitoring will be initiated during construction and continue to the end of operations. Monitoring obligations under the Environmental Management Act air discharge authorizations would continue in accordance with conditions.

20.4.1.1.3 Wildlife

Management of wildlife use of the TSF ponds will be guided by the Wildlife Mitigation and Monitoring Plan (WMMP; Section 20.6.1).

Monitoring of wildlife interactions with the TSF will be guided by the Wildlife Mitigation and Monitoring Plan (WMMP; Section 20.8). The results of the WMMP for the TSF will be reported annually from construction to operations.

20.4.1.2 Monitoring

20.4.1.2.1 Water Quality

Discharges from the TSF will be monitored through the MSDP. Water quality monitoring is detailed in Section 20.3.3.

20.4.1.2.2 Dust Generation

Dust generation from the TSF will be monitored through the Air Quality and Dust Monitoring Program has been integrated into the AQDMP. Visual monitoring of dust will be undertaken. These observations, together with meteorological conditions and mitigation efforts taken to deal with any problem, will be recorded and included in monthly and annual reports. Monitoring will be initiated during construction and continue to the end of operations. Monitoring obligations under the Environmental Management Act air discharge authorizations would continue in accordance with conditions.

20.4.1.2.3 Wildlife

Monitoring of wildlife interactions with the TSF will be guided by the Wildlife Mitigation and Monitoring Plan (WMMP; Section 20.8). The results of the WMMP for the TSF will be reported annually from construction to operations.

20.4.2 Tailings Alternative Assessment/Best Available Tailings Technology Assessment

Evaluation of Alternative Tailings Technologies for the Blackwater Project (ERM, 2015) was prepared in response to the BC EAO request that all mine proponents consider the implementation of Best Available Technology (BAT) and Best Available Practices (BAP) identified by the Report on Mount Polley Tailings Storage Facility Breach (Independent Expert Engineering

Investigation and Review Panel, 2015). The report assessed tailings and waste rock storage alternatives and BAT/BAP for tailings management, considering the safety, technical, water balance, and lifecycle costs aspects for all Project phases, as well as the implications for environmental, health, social and economic values. At the request of Environment and Climate Change Canada (ECCC), these reports were combined into one stand-alone report to incorporate the Project optimizations in the 2020 Pre-feasibility Study (Artemis, 2020). This report was prepared in accordance with 16 Guidelines for the Assessment of Alternatives for Mine Waste Disposal (ECCC, 2016).

Thirteen candidates passed the pre-screening step and were carried through to a high-level risk assessment. The risk assessment identified thickened slurry tailings with submerged PAG/NAG3 waste rock (refer to Section 16.6.1) as the preferred alternative TSF location/technology.

20.4.3 Overburden and Waste Rock Management

A single stockpile will store overburden and NAG waste rock from stripping and open pit mining. The overburden and NAG waste rock stockpile design is presented in Section 16.6.

20.4.3.1 Potential Environmental Risks and Mitigations

20.4.3.1.1 Water Quality

Discharges from the overburden and waste rock stockpiles will be monitored through the MSDP. Water quality monitoring is detailed in Section 20.3.2.

20.4.3.1.2 Dust Generation

Potential fugitive dust generation may occur from exposed surfaces on the waste stockpiles. Fugitive dust from these areas will be managed measures in the AQDMP. In the closure and post-closure phases, the waste stockpiles will be covered and revegetated, hence, controlling fugitive dust generation.

Dust generation from waste stockpiles will be monitored through the Air Quality and Dust Monitoring Program and has been integrated into the AQDMP. Visual monitoring of dust will be undertaken. These observations, together with meteorological conditions and mitigation efforts taken to deal with a problem, will be recorded and included in monthly and annual reports. Monitoring will be initiated during construction and continue to the end of operations. Monitoring obligations under the Environmental Management Act air discharge authorizations would continue in accordance with conditions. The results of visual dust monitoring will be reported in annual reports.

20.4.3.1.3 Wildlife

During the construction and operations, wildlife are not likely to interact with the waste dumps as they are not suitable habitat and subject to constant human disturbances. In the early years of closure, each waste stockpile will be re-contoured to facilitate wildlife access, covered, and planted to native plant communities. In post-closure, the stockpile will continue to be covered by a self-regenerating native plant community.

Monitoring of wildlife for the mine site including the waste stockpiles will be guided by the WMMP. In closure and post-closure, the reclamation to use for wildlife will be assessed through meeting end land use goals.

20.4.4 Domestic and Industrial Non-Hazardous and Hazardous Waste Management

20.4.4.1 Sewage Treatment and Disposal

The exploration camp domestic wastewater treatment and disposal is already permitted, under the Municipal Wastewater Regulation, and has operated since 2012. New domestic wastewater treatment and disposal facilities will be constructed in association with the operations camp, plant and contractor laydowns. Applications will be submitted to seek authorization for these systems under the Municipal Wastewater Regulation registration and/or Northern Health as appropriate.

20.4.4.2 Waste Management

Waste generated over the LOM will include food and other putrescible; combustible (non putrescible); non-combustible; recyclable; and hazardous. The Waste (Refuse and Emissions) Management Plan (Section 20.6) provides procedures for collection, handling, and disposal of wastes. Waste transfer areas (WTA) will be established to manage material destined for off-site disposal.

Artemis is permitted to operate a commercially available diesel-fuel fired, double chamber incinerator (Environmental Management Act Authorization #106530). Authorized waste for incineration include putrescible camp waste, paper, cardboard and lumber scraps that cannot be recycled.

Hazardous wastes will include waste oil, oily waste (e.g., contaminated soils, pads, absorbent mats, rags, personal protective equipment, waste paints, solvents, and batteries (nickel cadmium, lead acid, and lithium). Hazardous and non-hazardous wastes will be stored separately and will be segregated according to classifications in the Hazardous Waste Regulation (BC Reg. 63/88). Hazardous wastes and flammable hazardous wastes will be transported by licensed carrier for appropriate disposal or recycling offsite at designated disposal facilities.

20.5 Air Quality Management Plan (including Greenhouse Gases)

Potential sources of air quality discharges from the Project are anticipated to include:

- Crushing and grinding areas from the Processing Plant;
- Discharge from Processing Plant scrubbers;
- Scrubbers from assay laboratory; and
- Fugitive dust from disturbed areas including roads, stockpiles, tailings beaches, and open pit.

Project air discharge emissions will be authorized through an *Environmental Management Act* air discharge permit.

20.5.1 Potential Environmental Risks and Mitigations

Key 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.

20.5.2 Monitoring

Air quality monitoring program will be undertaken during construction and operations and will include:

- Meteorology – continued operation of two on-site meteorological stations;
- Particulate matter – airborne particulate matter including dust and other fine matter; and
- Nitrogen dioxide and sulfur dioxide – passively monitoring during construction, operations and early closure.

Measurements will be summarized daily and reported annually.

20.6 Environmental Management System

An Environmental Management System will be developed to provide the framework to organize and guide activities during all mine phases to ensure orderly, safe, compliant, and environmentally and socially responsible operations at the mine.

20.6.1 Management Plans

Individual management plans form the basis for the Environmental Management System to be implemented throughout the life of mine. Initial management plans that will be prepared prior to the start of construction are presented in Table 20-5. A number of plans have been developed for submission with the joint MA/EMA application to fulfill the requirements of that submission; plans required by a number of EAC conditions will be developed in accordance with timelines prescribed in the EAC.

Table 20-5: Management Plans

Plan	Source Requirement
Surface Erosion Prevention and Sediment Control Plan	MA/EMA
Soil Management Plan	MA/EMA
Construction Environmental Management Plan	MA/EMA
Metal Leaching/Acid Rock Drainage Management Plan	MA/EMA
Mine Site Water and Discharge Monitoring and Management Plan	MA/EMA
Vegetation Management Plan	MA/EMA
Invasive Plant Management Plan	MA/EMA
Wildlife Mitigation and Monitoring Plan	MA/EMA
Reclamation and Closure Plan	MA/EMA
Archaeological Management and Impact Mitigation Plan	MA/EMA

Plan	Source Requirement
Mine Emergency Response Plan	MA/EMA
Mine Site Traffic Control Plan	MA/EMA
Fuel Management and Spill Control Plan	MA/EMA
Chemicals and Materials Storage, Transfer, and Handling Plan and Cyanide Management Plan	MA/EMA
Waste (Refuse and Emissions) Management Plan	MA/EMA
Air Quality and Dust Management Plan	MA/EMA
Care and Maintenance Plan	MA/EMA/EAC
Aboriginal Group Engagement Plan	EAC
Aboriginal Group Monitor and Monitoring Plan	EAC
Cultural and Spiritual Resources Management Plan	EAC
Noise and Vibration Effects Monitoring and Mitigation Plan	EAC
Caribou Mitigation and Monitoring Plan	EAC
Wetland Management and Offsetting Plan	EAC
End Land Use Plan	MA/EMA/EAC
Chedakuz Creek and Tatelkuz Lake Surface Water Quality Monitoring Plan	EAC
Transmission Line Sedimentation Monitoring Plan	EAC
Aquatic Effects Monitoring Plan	EAC
Tatelkuz Lake Protection Plan	EAC
Mine Waste and Water Management Plan	MA/EMA/EAC
Closure and Post-Closure Water Quality Management Plan	MA/EMA/EAC
Tailings Dam Safety Transparency Plan	EAC
Accidents and Malfunctions Administration and Communication Plan	EAC
Community Liaison Committee and Community Effects Monitoring and Management Plan	EAC
Tenure Holder Communication and Mitigation Plan	EAC

Plan	Source Requirement
Final Transmission Line Routing Plan	EAC
Health and Medical Services Plan	EAC
Country Foods Monitoring Plan	EAC

20.6.2 Offset Plans

Compensation/offsetting plans are required to mitigate potential impacts to fish and fish habitat, wetlands and southern mountain caribou. Summaries of each plan are provided below.

20.6.2.1 Fish and Fish Habitat

The Project will require an Authorization under Paragraph 35(2)(b) of the *Fisheries Act*, and an amendment of Schedule 2 of the Metal and Diamond Mining Effluent Regulations (under Section 36 of the *Fisheries Act*). A conceptual fish habitat compensation/offsetting plan was developed during the EA phase. Two separate fish habitat compensation/offsetting plans will be developed to offset instream and riparian habitat loss in accordance with regulatory requirements.

20.6.2.2 Wetlands

Artemis requires a Wetland Monitoring and Offsetting Plan. The Plan is intended to offset the Project's impacts on wetland functions. The Mathews Creek wetland complex has been selected as the primary offsetting site. The wetland complex is located off the Project area on lands partially owned by Artemis (fee-simple) with the balance being Crown land.

20.6.2.3 Southern Mountain Caribou

The local population of mountain caribou (*Rangifer tarandus caribou*) is listed as threatened under Schedule 1 of the *Species at Risk Act,* a special concern by Committee on the Status of Endangered Wildlife in Canada, and blue-listed by the province (BC CDC 2020). The Project overlaps the eastern boundary of the Tweedsmuir local population unit (BC FLNRORD, 2020). A conceptual Caribou Mitigation and Monitoring Plan was developed during the EA phase to avoid, reduce and offset the Project's adverse effects on caribou and its critical habitat as defined in the Recovery Strategy for the Woodland Caribou, Southern Mountain population in Canada (ECCC, 2014). A final Caribou Mitigation and Monitoring Plan is being developed in accordance with EAC and DS conditions.

20.7 Closure and Reclamation

Reclamation of the Project area will conform to the requirements of the Health, Safety, and Reclamation Code for Mines in BC (BC EMLI 2021). As much as possible, disturbed areas will be reclaimed to native ecosystems and waterways restored to pre-disturbance flow patterns. In the extended Closure and Post-closure phases, activities will focus on monitoring vegetation and geotechnical stability of reclaimed areas, and water treatment, as required.

The following performance closure and reclamation objectives are required for the Project:

Objective 1. A mix of sustainable conditions supporting wildlife habitat, traditional and current use by Indigenous peoples, and recreational use of Crown land by the general public.

Objective 2: Self-sustaining vegetation that will progress to plant communities similar to pre-disturbance ecosystems as supported by the results of the ecohydrological modelling.

Objective 3: Physical stability of post-closure mining landforms designed to incorporate controls to minimize erosion of surficial materials.

Objective 4: Control of geochemical sources to achieve stable surface geochemical conditions.

Objective 5: Water quality and flow that support fish habitat downstream from the mine site and reclamation objectives.

Details of the closure and reclamation will be detailed in the Project's Reclamation and Closure Plan (RCP) which is a living document to be updated throughout the life of the Project.

The general scheduling approach to closure and reclamation for this Project is to:

- Conduct progressive reclamation and reclamation research where possible during construction and operations phases;
- Document the status of mine site disturbance, reclamation, reclamation materials inventory, research trials, monitoring programs and other related results in the Annual Reclamation Report;
- Provide updated 5-year detailed Mine Plans with updated closure cost estimates;
- Implement final reclamation and closure measures at the cessation of mine operations and after decommissioning mining infrastructure, beginning at closure and continuing into post-closure;
- Implement post-closure water treatment as required; and
- Complete post-closure monitoring to demonstrate that closure end land use objectives have been achieved.

20.8 Social Considerations

Artemis has engaged in consultation with local groups as discussed in Section 4.7.

21 CAPITAL AND OPERATING COSTS

21.1 Capital Cost Estimates

21.1.1 Introduction and Scope of Work

The objective of the Study was to develop a capital cost estimate with an accuracy of +15/-10% according to Ausenco's Class 3 estimate standards for Initial capital estimates. The estimate includes the cost to complete the design, procurement, construction and commissioning, of all the identified facilities.

In addition, two capital cost estimates with an accuracy of ±25% according to Ausenco's Class 4 estimate standards were prepared for the proposed Phase 2 and Phase 3 process plant expansions. The Class 4 cost estimates were developed at the same time as the 2021 FS, and benefited from the use of current market data, detailed cost build-ups and quotations rather than historical data.

The physical facilities and utilities for Blackwater Project include but are not limited to the following areas:

Initial Capital (Phase 1):

The process plant will consist of the following facilities:

- Site preparation
- ROM material receiving feed bin;
- MSE retaining wall at primary crusher station;
- Primary crushing;
- Secondary and tertiary crushing and screening;
- Open crushed ore stockpile with reclaim system;
- Dust collection and dust suppression;
- Ball milling with hydro-cyclones;
- Gravity concentration and intensive leaching;
- Agitated tank leaching and CIL adsorption;
- AARL elution and carbon reactivation;
- Electrowinning;

- Smelting to doré bars;
- Cyanide destruction (SO_2 / air method);
- Raw water circuit and process water circuit;
- Reagent storage and preparation facilities;
- Utilities including:
  - Air
  - Electricity
  - Potable water and treatment
  - Fire water
  - Sewage treatment
- Process buildings and structures will include:
  - Process building;
  - Reagent storage building c/w gold room;

Facilities beyond the plant footprint will include:

- Phased TSF;
- WRSFs;
- Tailings disposal pipeline from process facility to the TSF;
- Process water from the TSF including water reclaim pumps, including boosters, and pipeline to the process facility; and
- Site water management systems.

Additional site infrastructure will include:

- Access road improvements;
- BC Hydro overland power line feed and substation;
- Relocation and rehabilitation of onsite existing facilities for project use;
- Accommodations camp;

- Electrical power reticulation within the process facility (tie-in at low voltage side of HV Hydro substation at the process plant area) and to nominated site infrastructure;
- Electrical and control rooms for the process and crushing facilities;
- Laboratory building;
- Reserved location in the layout for a leased oxygen plant office;
- Central control room building;
- Combined truck shop, mine administration/mine dry building;
- Plant stores / warehouse building;
- Bulk fueling station; and
- Fiberoptic network.

Mining activities will include but are not limited to the following:

- Mining pre-stripping;
- Mine development;
- Mine equipment;
- Mine infrastructure and services;
- Mine dewatering;
- Explosive storage facilities; and
- Mine access and haul roads.

21.1.1.1 Phase 2 Expansion:

The Phase 2 expansion to 12Mtpa was simplified to be achieved with only minor modifications needed to the existing Phase 1 crushing, stockpile and ball mill feed system. The second ball mill will operate in series with the Phase 1 mill. The rest of the plant will be duplicated (gravity concentration, leaching, adsorption, elution and cyanide destruction). Minor upgrades will be carried out on some infrastructure to accommodate the increased throughput.

21.1.1.2 Phase 3 Expansion:

The Phase 3 expansion to 20 Mtpa will require a new process line, from primary crushing through to cyanide destruction, although a carbon elution circuit will not be needed as the Phase 1 and 2 units will have sufficient capacity with the lower ore grades expected in Phase 3.

No expansion of the process facilities is required for Phase 4 operations.

21.1.2 Project Execution

The estimate was based on a traditional engineering, procurement, and construction management (EPCM) approach where the EPCM contractor will oversee the delivery of the completed Project from detailed engineering, procurement and construction to handover of working facility. The EPCM contractor will engage and coordinate several subcontractors to complete all work within the given scopes.

In addition to the EPCM approach, the estimate is also based on Artemis self executing the mining activities and a portion of the earthworks components.

21.1.3 Estimate Summary

The capital cost estimates were summarized at the levels indicated by the following tables and stated in millions of Canadian Dollars (C$ M) with a base date of Q2-2021 with no provision for forward escalation. The estimate collectively presents the entire costs for the project including all Third-Party estimates, Owner's scope and Ausenco's scope. The estimate summary of major facilities and major disciplines are defined in Table 21-1 and Table 21-2 respectively.

Table 21-1: Estimate Summary Level 1 Major Facility

	Description	Initial Capital	Expansion / Growth Capital	Sustaining Capital	Deferred Capital	Total LOM Capital
1000	Mining	64.5	62.8	430.1	4.0	561.4
2000	Site development / tailings storage facility / waste rock facility	73.1	175.9	273.2	0.2	522.5
3000	Ore crushing and reclaim	55.6	45.2	0	0	100.8
4000	Process plant	141.7	254.6	0	0	396.3
5000	On-site infrastructure	30.5	31.5	17.0	26.1	105.2
6000	Off-site infrastructure	100.7	0	22.8	12.7	136.2
	Subtotal Direct Costs	466.2	570.0	743.2	43.0	1,822.40
7000	Indirects	16.5	26.2	2.6	0.1	45.4
8000	Engineering and project management	60.9	58.1	22.7	3.5	145.1
9000	Provisions and owners costs*	101.7	66.3	62.2	5.1	235.3
	Subtotal Indirect Costs	179.0	150.6	87.5	8.6	425.8
	PROJECT TOTAL (C$ M)	645.2*	720.6	830.7	51.6	2,248.2

* The impact of PST has been included in WBS 9000

Table 21-2: Estimate Summary by Major Discipline (Phase 1 only)

Disc.	WBS Description	Total Initial Capital (C$ M)	% of Total
B	Earthworks	10.6	1.6%
C	Concrete	19.1	3.0%
S	Structural steelwork	19.5	3.0%
F	Platework	21.4	3.3%
M	Mechanical equipment	89.4	13.9%
P	Piping	27.6	4.3%
E	Electrical equipment	17.4	2.7%
I	Instrumentation	3.2	0.5%
L	Electrical bulks	13.3	2.1%
A	Architectural	17.8	2.8%
R	Third party estimates	226.9	35.2%
U	Field indirects	6.1	1.0%
V	Other (spares, fills, vendors)	9.3	1.4%
T	Project delivery	61.9	9.6%
Y	Provisions	55.0	8.5%
O	Owners costs	46.6	7.2%
	Project Total (C$ M)	645.2	100.0%

21.1.4 Estimate by Consultant

Ausenco has completed battery limit and scope review meetings with a scope matrix assigned by WBS with each Third Party to ensure the scope of work for the project has been accounted for. The Estimate Summary by Consultant is defined in Table 21-3.

Table 21-3: Estimate Summary by Consultant

Consultant	WBS Description	Total Initial Capital	% of Total
MMTS	Pre-Stripping, Mine Development, Mine Equipment	C$69.1	10.7%
Ausenco	Site Development, Process Plant, On-Site Infrastructure, Pipelines, Electrical Distribution, Fire Protection, Field Indirects, Spares/Fills/Vendor Reps	C$350.0	54.2%
KP	Tailings Storage Facility, Water Management Structures, Site Water Systems	C$70.4	10.9%
Allnorth	Off-Site Infrastructure, Accommodations Camp, Bulk Fuel and Dispensing, Fiber Optics	C$16.6	2.6%
Owners	Powerline, Substation, Accommodations Camp Operations, Corporate Overheads, Fish Offsetting, Site Office, Environmental, Staff and Labour, Setup and Running Costs	C$139.2	21.6%
	PROJECT TOTAL (C$ M)	C$645.2	100.0%

21.1.5 Definition of Costs for the 2021 FS

Initial capital is defined as the capital expenditure required to start up a business to a standard where it is ready for initial production.

Sustaining capital is defined as the capital cost associated with the periodic addition of new plant equipment or services that are required to maintain production and operations at their existing levels. Expansion capital does not form part of the sustaining capital.

Direct costs are defined as those costs that pertain to the permanent equipment, materials and labour associated with the physical construction of the process facility, infrastructure, utilities, buildings, etc. Contractor's indirect costs are contained within each discipline's all-in rates which include construction equipment.

Indirect costs are defined as including all costs associated with implementation of the plant and incurred by the Owner, engineer or consultants in the design, procurement, construction, and commissioning of the Project.

21.1.6 General Methodology

The estimate was developed based on a mix of detailed material take-offs and factored quantities and costs. Detailed unit costs were supported by contractor bids and budgetary quotations for major equipment supply.

The structure of the estimate was a build-up of the direct and indirect cost of the current quantities; this included the installation/construction hours, unit labour rates and contractor distributable costs, bulk and miscellaneous material and equipment costs, any subcontractor costs, and freight and growth.

The methodology applied and source data used to develop the estimate was as follows:

- Define the scope of work
- Quantify the work in accordance with standard commodities;
- Organize the estimate structure in accordance with an agreed WBS;

- Calculate "all in" labour rates for construction work;
- Determine the purchase cost of equipment and bulk materials;
- Determine the installation cost for equipment and bulks;
- Establish requirements for freight;
- Determine the costs to carry out detailed engineering design and project management;
- Determine foreign exchange content and exchange rates;
- Determine the estimate contingency value;
- Undertake internal peer review, finalize the estimate, estimate basis and obtain sign off by the Project manager and/or Qualified Persons.

21.1.7 Source Data

Source data included:

- Scope of work;
- Design criteria;
- General arrangement drawings;
- Drawings and sketches;
- Structural models;
- Geotechnical investigation data;
- Process flow diagrams;
- Material take-offs and Equipment Lists;
- Execution plan;
- Equipment and bulks pricing;
- Contractor installation data
- Vendor material supply costs
- Historical data and Existing study data
- Project schedule.

21.1.8 Basic Information

The following basic information pertains to the estimate:

- The estimate base date is Q2-2021
- The estimate is expressed in Canadian Dollars (C$);
- Metric units of measure are used throughout the estimate

21.1.9 Exchange Rates

The estimate is prepared in the base currency. Pricing has been converted to CAD using the following exchange rates. Estimate exchange rates are defined in Table 21-4.

Table 21-4: Estimate Exchange Rates

Code	Currency	Exchange Rate
CAD	Canadian dollar	1.00 CAD = 1.0000 CAD
AUD	Australian dollar	1.00 CAD = 1.1008 AUD
USD	United States dollar	1.00 CAD = 0.7900 USD
ZAR	South African Rand	1.00 CAD = 11.7490 ZAR

21.1.10 Market Availability

The pricing and delivery information for quoted equipment, material and services was provided by suppliers based on the market conditions and expectations applicable at the time of developing the estimate.

Market conditions are susceptible to the impact of demand and availability at the time of purchase and could result in variations in the supply conditions. The estimate in this report is based on information provided by suppliers and assumes there are no problems associated with the supply and availability of equipment and services during the execution phase.

21.1.11 Basis of Mining Capital Cost Estimate

Mine capital costs are derived from vendor quotations and operational data collected by other Canadian open pit mining operations.

Pre-production mine operating costs, that is, all mine operating costs incurred before mill start-up, have been capitalized and are included in the capital cost estimate. These costs include grade control, drill and blast, load and haul, support and GME costs.

The mine equipment fleet is planned to be purchased via various lease arrangements, with quoted commercial terms from the equipment suppliers. Down payments and lease payments are capitalized through the initial and sustaining periods of the Project. The leased mining fleet is paid off over several years, so cost are mostly captured as sustaining capital.

Mine Operations site preparation costs have been capitalized including:

- Explosives and Contractor Pad preparation;
- Stockpile preparation;
- Clearing and grubbing
- Topsoil removal and stockpiling;
- Haul road construction;

The following items have also been capitalized:

- Pit Electrification;
- Explosive magazine and mixing plant;
- 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;
- Piping for open pit dewatering.

The capital cost estimate for mining scope of work was summarized at the levels indicated by the following tables and stated in the base currency for initial capital for the consolidated estimate. The mining initial capital costs are summarized Table 21-5.

Table 21-5: Mining Initial Capital Costs (C$ M)

WBS Lvl2	Description	Total Initial Capital
1100	Mine development	15.6
1200	Pre-stripping	37.5
1300	Mobile equipment	7.5
1400	Mine infrastructure area	3.9
	Sub-Total Direct Costs	64.5
7100	Indirects general	0.3
8100	Operations readiness	0.2
8200	Vendor representatives	0.2
9100	Contingency	4.0
	Sub-Total Indirect Costs	4.7
	PROJECT TOTAL (C$ M)	69.1

21.1.12 Basis of Process Plant and On-Site Infrastructure Capital Cost Estimate

21.1.12.1 Direct Costs

Direct costs are generally quantity based and include all permanent equipment and materials associated with the physical construction of the facility. Typically, these are included in the following list:

- Direct man-hours and labour;
- Contractors distributables;
- Construction equipment;
- Job materials (consumables);
- Permanent equipment and bulk materials; and
- Freight and subcontracts.

The capital cost estimate for the process plant and overall site infrastructure scope of work was summarized at the levels indicated by the following tables and stated in the base currency for initial capital for the consolidated estimate. The process plant and overall site infrastructure initial capital costs are summarized Table 21-6.

Table 21-6: **Process Plant and Overall Site Infrastructure Initial Capital Costs**

WBS Lvl2	Description	Total Initial Capital
2100	BULK EARTHWORKS	9.3
2200	PROCESS IN-PLANT ROADS	0.3
2400	Tailings storage facility (TSF)	5.4
3100	Crushing	48.8
3200	Crushed ore stockpile and reclaim	6.8
4100	Process plant buildings	22.7
4200	Grinding / gravity concentration / intensive leaching	38.5
4300	Gold leaching	13.0
4400	Carbon-in-leach / adsorption	17.8
4500	Carbon reactivation and regeneration	15.6
4600	Electrowinning and smelting	1.6
4700	Cyanide destruction and tailings	9.6
4800	Reagent make-up area	8.3
4900	Process plant utilities	14.2
5100	Oxygen plant	0.3
5200	Plant site ancillary buildings	3.3
5300	Truck shop ancillary buildings	8.1
5500	Site power systems	10.9
5600	Site water systems	6.2
5700	Waste management systems	1.1
5800	IT and communications	0.4
6300	Water wells	0.02
6400	Permanent camp	8.6
	Sub-Total Direct Costs	250.8
7100	Indirects general	5.9
7200	Operations readiness	1.5
7300	Vendor representatives	1.1
7400	Spares and first fills	7.7
8100	Engineering and procurement (EP) general	42.7
8200	Construction management (CM)	3.9
8300	General	8.9
9100	Contingency	44.0
	Sub-Total Indirect Costs	115.7
	PROJECT TOTAL (C$ M)	366.5

*Costs are the sum of Ausenco and Allnorth scope of works

21.1.13 Basis of Tailings Storage Facility and Water Management Estimates

The capital cost estimate for the tailings storage facilities and water management structures/systems was developed based on the current project understanding and level of detail available within the detailed designs supporting the *Mines Act and Environmental Management* Act Permit Application (the Permit Application).

Equipment and material quantities supporting the feasibility study cost estimate were developed from the material take-off data provided by the detailed designs supporting the Permit Application. Earthworks and mechanical equipment unit rates were developed from a combination of vendor quotations, manufacturers' information, industry standards and rates, and unit rates developed by KP based on estimated productivity rates, equipment fleets, and all-in contractor equipment rates.

The total initial capital cost of the TSF and Water Management infrastructure comprises the Site Water Management Structures, the Tailings Storage Facility, Site Water Systems, and indirects including KP construction QA/QC, field reviews, design office support, and contingency specific to each stage of TSF development. The estimated initial capital costs for the tailings storage facilities and water management structures/systems are summarized in Table 21-7.

Table 21-7: TSF and Water Management Structures Initial Capital Costs

WBS	Description	Total Initial Capital
2300	Site water management structures	23.4
2400	Tailings storage facility	34.7
5600	Site water systems	0.3
	Sub-total direct costs	58.4
8200	KP construction QA/QC, field reviews, design office support	5.0
9100	Contingency	7.0
		12.0
	Project Total (C$ M)	70.4

21.1.14 Indirect Capital Cost Estimate

21.1.14.1 Field Indirects and Temporary Construction Facilities

Indirect costs include all costs that are necessary for project completion that are not related to the direct construction costs.

The temporary construction facilities were developed from budget quotations, historical data and allowances and cover temporary signage, office consumables and janitorial services, medical and personal protective equipment supplies, snow removal, third-party surveying, security, light plants, rigging, rental equipment, additional scaffolding, washrooms, temporary generators, winterization, heating and hoarding, fuel bulks, heavy lift cranes, etc.

Costs were developed by each consultant to cover their scope of work and summarised in the estimate to cover the construction duration.

Indirect Costs include:

- Spares
- First Fills
- Vendor Representatives
- Pre-commissioning, Commissioning

21.1.15 Construction Camp and Catering

A construction camp was allowed for in the estimate, and the camp cost has been included in the consolidated estimate based on Allnorth's detailed estimate.

A cost for camp operations and maintenance during the construction has been included in the consolidated estimate based on Allnorth's detailed estimate.

Costs for accommodation and messing are 'Free Issue' by the Owner. As such, no accommodation and messing costs are included in the contractor rates.

21.1.16 Owners Cost Estimate

The capital cost estimate was summarized for the Owner's scope of work at the levels indicated by the following tables and stated in Canadian dollars for initial capital for the consolidated estimate. The Owner's Initial capital costs are defined in Table 21-8.

Table 21-8 Owners Initial Capital Costs (C$ M)

WBS Lvl2	Description	Total Initial Capital
4100	Process plant buildings	0.5
6200	230 kV HV power transmission line and substation	92.1
	Sub-Total Direct Costs	**92.5**
9900	Owners Costs (labour, training, expenses, etc)	46.6
	Sub-Total Indirect Costs	**46.6**
	PROJECT TOTAL (C$ M)	139.2

Ausenco has included the initial capital Owner's costs into the 2021 FS estimate.

Owner's costs include, but are not limited to, the following:

- Corporate overheads and office
- Environmental monitoring
- Fish offsetting
- Site office
- Setup and running costs
- Staff and labour
- Laboratory equipment and fit out

- Camp catering and accommodation operating costs
- Off-site infrastructure
  - 230kV powerline and substation
  - Glenannan substation upgrades(BC Hydro)

21.1.17 Contingency Cost Estimate

Estimate contingency was included to address anticipated variances between the specific items contained in the estimate and the final actual Project cost.

The estimate contingency will not allow for the following:

- Abnormal weather conditions
- Changes to market conditions affecting the cost of labour or materials
- Changes of scope within the general production and operating parameters
- Effects of industrial disputes.

The capital cost estimate was summarized for contingency at the levels indicated by the following tables and stated in Canadian dollars for initial capital for the consolidated estimate. The contingency allowances for the initial capital costs are defined in Table 21-9.

Table 21-9: Estimated Contingency

Type	Total Initial Cost (C$ M)	% of TDC (by scope)
Ausenco	41.9	13.6
MMTS	4.0	6.1
KP	7.0	11.1
Allnorth	2.1	14.3
Artemis	Included in costs	0
Total Contingency (Blended)	55.0	12.2

21.1.17.1 Management Reserve Analysis

No management reserve was allowed for in this estimate as it lies outside the estimate scope.

21.1.17.2 Escalation

No escalation was proportioned to any part of the estimate.

21.1.18 Estimate Exclusions

The following items were not considered in the 2021 FS capital cost estimate:

- Permitting and Environmental approvals;
- Force majeure issues;
- Scope changes;
- Special incentives (schedule, safety or others);
- Growth;
- No allowance has been made for loss of productivity and/or disruption due to religious, union, social and/or cultural activities;
- Owner's escalation costs;
- Concrete pours in winter conditions (Nov 01 – May 01);
- Demolition and salvage of any existing on-site structures.

21.2 Operating Cost Estimates

21.2.1 Introduction

The operating cost estimates were prepared by the following parties:

- MMTS: Mining cost estimates;
- Ausenco: Process cost estimates, including reagent and steel consumables consumption and costs, spare parts utilization, labour costs, freight, and electrical power; mill staffing plans,
- JAT Metco: G&A estimates including Allnorth estimates for main access road main and Kluskus Forestry Services Road maintenance, and bussing services for site workforce during operations.

21.2.2 Mine Operating Costs

The mine operating costs were built up from first principles and consisted of the following components:

- Equipment operating cost – the activities of grade control, drilling, blasting, loading, hauling, mining support, and equipment maintenance. Equipment operating costs were based on the total 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 was for the hauling function, and a significant portion of the planned hauls for Blackwater were downhill loaded, especially early in the Project life;

- Salary and hourly personnel – mine department salary staff and general mining labour; maintenance and operator labour were included as part of the equipment operating costs, so distributed into the direct mining functions (drill, load, haul, etc.). During peak production, the mine was expected to employ 35 salaried personnel and 348 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 WRFs, office costs and overheads for mine operations, mine maintenance and technical services departments.

The total mining cost once in commercial production is estimated to be approximately C$2.5 B to mine 1.0 Bt of material, and move 1.2 Bt of material, including ore and waste rehandle. The total unit cost for the Project was approximately C$2.60/t mined.

The total operating cost for all activities is approximately C$7.57/t of ore mined (Table 21-10) during the operations period.

Table 21-10: Mining Costs (Operations Period)

Mine Operating Cost Summary	C$/t Mined	C$/t Milled	Total $ (C$ M)
Grade Control	0.03	0.07	24
Drilling	0.14	0.42	140
Blasting	0.25	0.74	248
Loading	0.24	0.70	235
Hauling	1.42	4.12	1,376
Support	0.36	1.04	347
Unallocated Labour	0.03	0.10	32
Direct Costs - Subtotals	2.47	7.18	2,402
Mine Operations GME	0.06	0.18	61
Mine Maintenance GME	0.03	0.08	26
Mine Engineering GME	0.04	0.13	42
GME Costs - Subtotals	0.13	0.39	129
Total Mine Operating Cost	2.60	7.57	2,531

The mining costs (operations period) in the above table are presented net of reclassifications of certain mining costs to growth capital as it was considered to relate to the TSF construction. The above table also includes an estimated impact of Provincial Sales Tax in BC.

21.2.3 Overall Process Operating Costs

Process operating cost estimates for the multiple phases and the overall LOM process operating costs are summarized in

Table 21-11 below.

Table 21-11: Process Operating Cost Estimate Summary

Cost Centre	Phase 1	Phase 2	Phase 3	LOM Total
	C$/t milled	C$/t milled	C$/t milled	C$ M
Reagents and Consumables	5.54	5.44	5.21	1,766
Plant Maintenance	0.32	0.28	0.26	90
Power	1.69	1.70	1.56	535
Labour	1.49	1.08	0.88	327
G&A - Expenses	0.43	0.23	0.15	64
TOTAL	**9.47**	**8.73**	**8.06**	**2,782**

21.2.4 Process Operating Cost Estimate – Phase 1

The annual process operating cost for Phase 1 was estimated to be C$62.5 M. A breakdown of this value and the unit costs are presented in Table 21-12.

Table 21-12: Average Annual Process Operating Cost – Phase 1

Cost Centre	Average Annual Costs C$ M/a	C$/t milled	% of Total
Reagents and Consumables	36.5	5.54	58.5
Plant Maintenance	2.1	0.32	3.4
Power	11.2	1.69	17.8
Labour	9.8	1.49	15.7
G&A - Expenses	2.8	0.43	4.5
TOTAL	62.5	9.47	100

21.2.4.1 Reagents and Operating Consumables

Individual reagent consumption rates were estimated based on the metallurgical testwork results, Ausenco's in-house database and experience, industry practice, and peer-reviewed literature. Each reagent cost was obtained through vendor quotes.

Other consumables (e.g., liners for the crushers, screen deck panels ball mill liners, and ball media for the mills) were estimated using:

- Metallurgical testing results (abrasion)
- Ausenco's in-house calculation methods, including simulations
- Forecast nominal power consumption
- Supplier quotations

21.2.4.2 Plant Maintenance

General plant maintenance costs were C$2.1 M/a or C$0.32/t processed. Annual maintenance consumable costs were calculated based on a total installed mechanical cost by area using factors applied to the equipment cost.

21.2.4.3 Power

The processing power draw was based on the average power utilisation of each motor on the electrical load list for the process plant and services. Power will be supplied by BC Hydro to service the facilities at the site. The total process plant power cost was C$ 11.2 M/a or C$1.69/t.

21.2.4.4 Labour

Staffing was estimated by benchmarking against similar projects. The labour costs incorporate requirements for plant operation, such as management, administration, metallurgy, operations and maintenance. The total operational labour averaged 80 employees.

Individual personnel were divided into their respective positions and classified as either 8-hour or 12-hour shift employees. The rates were estimated as overall rates, including all burden costs (benefits) and were based on a published Canadian mine salary survey for 2020.

21.2.4.5 Phase 2 Process Operating Costs

The operating costs for the Phase 2 expansion case are shown in Table 21-13.

Total operational labour is expected to increase from 80 to 116 employees for the Phase 2 expansion.

Table 21-13: Phase 2 Average Annual Process Operating Cost

Cost Centre	Average Annual Costs C$ Ma	C$/t milled	% of Total
Reagents and Consumables	68.6	5.44	62.3
Plant Maintenance	3.5	0.28	3.2
Power	21.4	1.70	19.5
Labour	13.6	1.08	12.4
G&A - Expenses	2.9	0.23	2.6
TOTAL	110.0	8.73	100

21.2.4.6 Phase 3 Process Operating Costs

The operating costs for the Phase 3 expansion case are shown in Table 21-14.

Total operational labour is expected to increase from 116 to 155 employees for the Phase 3 expansion.

Table 21-14: Phase 3 Average Annual Process Operating Cost

Cost Centre	Total Annual Costs C$ M/a	C$/t milled	% of Total
Reagents and Consumables	103.5	5.21	64.6
Plant Maintenance	5.2	0.26	3.2
Power	31.0	1.56	19.4
Labour	17.5	0.88	10.9
G&A - Expenses	3.0	0.15	1.9
TOTAL	160.0	8.06	100

21.2.5 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 cost are, among others, power costs, water treatment costs and powerline/ road maintenance costs.

The G&A operating costs are summarized in Table 21-15.

Table 21-15: Average Annual G&A Operating Costs

Summary G&A	Phase 1 C$M	Phase 2 C$M	Phase 3 C$M	Phase 4 C$M
Personnel				
Administration	3.1	3.3	4.3	2.7
IT	0.1	0.1	0.2	0.1
Human Resources	0.4	0.4	0.4	0.4
Security	2.3	2.3	2.3	1.7
Public/Community Relations	0.3	0.4	0.3	0.3
Environmental	0.7	0.8	0.8	0.7
Safety	0.9	1.0	1.3	0.9
Site Services	1.7	1.7	1.7	1.0
Sub-total	9.5	10.0	11.4	7.9
Expenses				
Administration	4.0	4.8	6.2	4.9
IT	0.5	0.6	0.7	0.5
Human Resources	0.8	0.9	0.9	0.5
Security	0.4	0.4	0.5	0.4
Public/Community Relations	0.4	0.4	0.4	0.4
Environmental	1.3	1.4	1.4	1.3
Safety	0.3	0.4	0.5	0.3
Emergency Response	0.2	0.3	0.3	0.2
Site Services	4.8	7.2	9.0	6.1
Travel and Accommodation	4.9	6.3	6.4	3.1
Sub-total	17.7	22.7	26.3	17.8
Total	27.2	32.7	37.7	25.7

22 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;
- 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 Financial Model Parameters

The economic analysis assumes a 100% equity financed project. All dollar amounts in this analysis are expressed in Q2 2021 Canadian dollars, unless otherwise specified.

The economic analysis includes the entire Project life, comprising two years of construction and 22 of years of mining and milling.

Corporate sunk costs up to the start of construction, including costs for exploration, technical studies, and permitting, are excluded from initial capital. The cost of the Project acquisition was considered in the estimation of tax depreciation pools.

The net present value (NPV) at 5% is discounted to the start of project construction.

The economic analysis is presented as a Base Case, which assumes no leverage, and a Levered Case, which assumes debt financing. Financing of the Project is not a measure of the economic viability and technical feasibility of the Project, but a measure of the ability of Artemis to secure debt financing for the Project.

22.2.1 Metal Price and Selling Costs

The 2021 FS Base Case metal prices and selling costs are summarized in Table 22-1. Due to updated metal price forecasts these parameters are different than those used prepare the production schedule and the Mineral Reserve statement. Both metal prices are considered in the sensitivity tables in Section 22.4.

As discussed in Section 19.4, the economics include a gold stream agreement with New Gold.

Table 22-1: Inputs to Economic Analysis

Item	Units	Value
Gold price	US$/oz.	1,600
Silver price	US$/oz.	21.33
Currency exchange rate	US$:C$	0.79
Gold payable	%	99.9
Silver payable	%	95
Selling costs	C$/AuEq oz.	3

22.2.2 Royalties

The 2021 FS cashflow considers two private royalties at 1.0% and 1.5% over parts of the Mineral Reserve, and these have been applied to the economic cash flow model using life-of-mine average royalty rates. Estimated payments to Indigenous groups are also included in the economic cash flow model for the Project.

22.2.3 Taxation Considerations

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.

22.2.4 Closure Costs and Salvage Value

Closure costs are estimated at approximately C$175 million when discounted to Year 22, and a salvage value of C$42 M has been estimated for Project assets upon closure. Bonding of the reclamation and closure costs has been applied starting in Year -2 and is based on progressive disturbance.

22.2.5 Levered Case Assumptions

The Levered Case is based on the following assumptions, applied in addition to the Base Case assumptions:

- C$360 million (plus up to C$25 million in capitalized interest) in project debt financing;
- Annual interest rate of Canadian Dollar Offered Rate (assumed at 0.5% in the 2021 FS) plus a margin of 4.25% up to the date of completion, with the margin reducing to 3.75% once the Project is effectively in commercial production;
- Customary upfront and standby financing fees;
- Six-year term post commencement of commercial production with principal and capitalized interest repayable in quarterly installments over six years, commencing one year following achievement of commercial production, with a repayment holiday during years 4 and 5 of production while the company expects to undertake its Phase 2 expansion;
- Expansion capital is assumed to be funded through operating cashflow.

22.3 Financial Results

The production schedule on which the economic analysis is based is provided and discussed in Section 16.7 and is tabulated in Table 16-4.

Base Case cashflow for the overall Project is summarized in Table 22-2 on an annualized basis. The economic analysis for the overall Project is summarized in Table 22-3 for the Base Case and the Levered Case.

22.4 Sensitivity Analysis

An NPV sensitivity analysis was performed examining initial capital costs, operating costs, foreign exchange rate, gold grade and gold price as shown in Figure 22-1, Table 22-4, and Table 22-5. The Project 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.

Table 22-2: Projected Base Case Cashflow (years)

YEAR	UNITS	-2	-1	1	2	3	4	5	6	7	8	9	10	11	12	13	14	15	16	17	18	19	20	21	22	23	TOTAL
Project Stage				1	1	1	1	1	2	2	2	2	2	3	3	3	3	3	3	3	4	4	4	4	4	4	
Mill Throughput	Mt	-	-	6	6	6	6	9	12	12	12	12	15	20	20	20	20	20	20	20	20	20	20	20	18	-	334
Recovered Gold	('000) oz.	-	-	291	297	297	297	421	476	374	339	324	391	408	491	452	555	516	390	254	179	179	179	179	164	-	7,453
Recovered Silver	('000) oz.	-	-	1,120	927	946	934	1,223	2,183	2,566	1,296	1,722	1,443	1,290	1,491	1,966	2,256	2,467	2,042	2,430	2,462	2,462	2,462	2,462	2,250	-	40,398
Net Revenue	C$ (M)			579	583	583	581	825	956	765	663	648	770	826	982	914	1,123	1,046	789	539	401	404	404	405	368	(8)	15,140
Operating Cost	C$ (M)	-	-	175	178	191	202	216	292	285	303	340	360	393	369	390	387	354	311	240	207	205	211	206	189	-	6,005
Capital	C$ (M)	265	380	33	1	1	31	320	12	1	-	16	302	1	55	-	-	-	-	-	-	-	-	-	-	-	1,418
Sustaining Capital and Closure	C$ (M)	-	-	57	63	62	54	50	65	44	58	44	39	46	44	49	44	41	21	12	7	4	18	2	139	1	963
Working Capital and Bonding	C$ (M)	(7)	6	14	17	6	6	10	(11)	8	8	10	8	(25)	7	7	7	7	7	6	6	5	5	4	(4)	3	109
Pre-tax Cash Flow	C$ (M)	(262)	(388)	300	323	323	288	230	598	427	294	239	60	413	507	469	684	644	450	281	180	191	170	193	44	(12)	6,645
Taxes	C$ (M)	-	-	9	33	85	108	161	201	134	94	80	83	115	181	157	239	228	156	94	58	64	61	67	44	(14)	2,407
Post-tax Cash Flow	C$ (M)	(262)	(388)	291	290	238	179	69	396	293	200	159	(22)	298	326	311	445	417	295	187	122	127	109	126	-	2	4,239

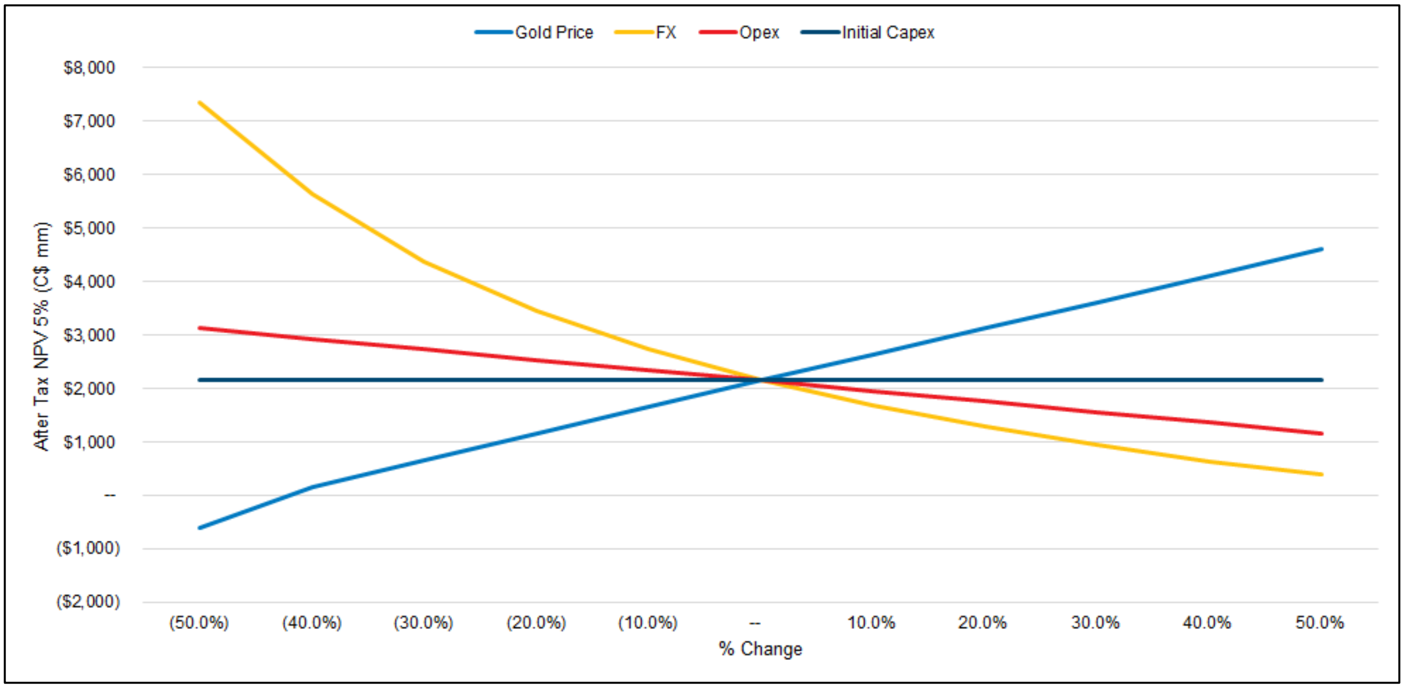
Note: numbers may not sum due to rounding.

Table 22-3: Economic Analysis

Item	Unit	Base Case	Leveraged Case
Pre-tax NPV (5%)	C$ M	3,449	3,441
Pre-tax IRR	%	40	52
Pre-tax Payback	Years	2.1	2.2
Post-tax NPV (5%)	C$ M	2,151	2,158
Post -tax IRR	%	32	43
Post -tax Payback	Years	2.3	2.4
Total LOM gold production	oz Au	7,453,000	7,453,000
Total LOM silver production	oz Ag	40,398,000	40,398,000
LOM strip ratio	Waste:ore	2.0:1	2.0:1
Average grade	g/t Au	0.75	0.75
Average grade	g/t Ag	5.8	5.8
Life of Mine Cash Cost*	C$/oz Au	720	720
Life of Mine All-in Sustaining Costs**	C$/oz Au	850	850

*Cash Costs include site operating costs and selling costs, less gross revenue generated from silver sales, divided by gold ounces recovered. Royalty payments and payments that are expected to be made to Indigenous groups are also included.
**All-in Sustaining Costs are Cash Costs (as described above) with the addition of sustaining capital and closure costs, all divided by the gold ounces recovered.

Figure 22-1: After Tax NPV 5% Sensitivity Analysis



Source: MMTS, 2021

Table 22-4: Base Case After Tax NPV 5% Sensitivity to Gold Price and Foreign Exchange

	US $ Gold Price								
US$/C$	$1,200	$1,300	$1,400	$1,500	$1,600	$1,700	$1,800	$1,900	$2,000
0.65	$1,785	$2,158	$2,530	$2,902	$3,275	$3,647	$4,019	$4,391	$4,763
0.70	$1,437	$1,784	$2,130	$2,476	$2,822	$3,168	$3,514	$3,859	$4,204
0.75	$1,133	$1,459	$1,783	$2,107	$2,429	$2,752	$3,075	$3,398	$3,720
0.79	$915	$1,228	$1,537	$1,844	$2,151	$2,458	$2,764	$3,070	$3,377
0.85	$626	$918	$1,209	$1,496	$1,782	$2,067	$2,352	$2,637	$2,922
0.90	$412	$691	$966	$1,241	$1,512	$1,782	$2,051	$2,320	$2,589
0.95	$218	$485	$749	$1,009	$1,269	$1,526	$1,781	$2,037	$2,291

Table 22-5: After Tax IRR Sensitivity to Gold Price and Foreign Exchange

	US $ Gold Price								
US$/C$	$1,200	$1,300	$1,400	$1,500	$1,600	$1,700	$1,800	$1,900	$2,000
0.65	28.1%	32.0%	35.7%	39.2%	42.6%	45.9%	49.2%	52.3%	55.3%
0.70	24.3%	28.2%	31.8%	35.2%	38.5%	41.7%	44.8%	47.8%	50.8%
0.75	20.9%	24.7%	28.2%	31.6%	34.8%	37.9%	40.9%	43.8%	46.7%
0.79	18.2%	22.0%	25.6%	28.9%	32.1%	35.1%	38.0%	40.9%	43.7%
0.85	14.5%	18.3%	21.8%	25.2%	28.3%	31.3%	34.1%	36.9%	39.6%
0.90	11.6%	15.4%	18.9%	22.3%	25.4%	28.3%	31.1%	33.8%	36.5%
0.95	8.6%	12.6%	16.2%	19.5%	22.6%	25.5%	28.3%	31.0%	33.6%

23 ADJACENT PROPERTIES

This section is not relevant to this Report

24 OTHER RELEVANT DATA AND INFORMATION

24.1 Risks and Opportunities Assessment

A comprehensive risk and opportunities register was developed.

24.1.1 Risks

Project risks that have been identified include:

- Economic risks: changes in metal prices and exchange rates; changes in input costs, primarily labour, fuel, and bulk materials for mining and processing; escalation in capital cost. Economic risks are almost completely out of the control of Artemis;
- Capital cost growth: costs for contractors, personnel, materials, and equipment are volatile; labour shortages may occur. Mitigation measures include provision for milestones in capital contracts let; provision of an attractive work site; employment of a cost engineer on the design team who will be responsible for tracking all major quantities and any design changes during detailed design and for providing insight into trends to minimize "cost creep"; employment of quantity surveyors during construction to check placed quantities against final design quantities; implementation of a rigorous Project controls system to provide progress, cost, and schedule monitoring and control;
- Operating costs: operating costs are sensitive to changes in the price of labour, consumables such as diesel fuel, and contractor services. The drill-and-blast study assumed the mine rock could be blasted using a "low energy" blast pattern design more than 70% of the time. If the rock proves to be more competent than assumed, then more drilling and blasting could be required, leading to higher mining operating costs. Mitigation measures for labour costs include offering competitive employment contracts;
- Productivity: the assumed equipment and labour productivities are based on good regional practice and a new operation will need to invest in training and hiring to achieve the same levels. Equipment productivities depend strongly on operator experience. Artemis will employ similar strategies for employee hiring and training as successfully used in regional operations;
- Dilution and ore loss: geological and mining conditions may be more challenging and complex than has been assumed, possibly resulting in lower revenues than anticipated due to lower ore tonnages or feed grades. Implementing effective grade-monitoring and grade-control procedures will be key in preventing higher dilution and/or ore losses than estimated in this study and in the ability to identify waste rock for TSF construction;
- High-grade silver in the mill feed: the presence of high-grade silver in the mill feed later in the mine life (approximately Year 18) could increase the frequency of elution because of faster carbon loading and cause soluble losses of silver, resulting in lower overall silver recovery. Plant feed may have to be carefully blended, possible with material from the high silver grade stockpile, through close coordination of the mine and mill departments. Appropriate operating action can be taken to mitigate the higher silver: gold ratio mill feed;
- Possible short term reductions in gold recovery due to ore containing deleterious elements.

- Possible short term peaks in silver content exceeding carbon loading capacity.
- Geological, geotechnical, and hydrogeological site conditions being less favourable than assumed during the design of the facilities.
- The cost of construction for the initial works associated with the TSF and water management will be heavily influenced by how the work is executed. The estimated costs are reliant on the plan to have the mine equipment and staff available to prepare haulage routes and transport, dump and spread a large portion of the dam materials. Managing the scope, contractual terms, and seasonality of the early contractor construction works to levels consistent with the estimate in the study and providing mining resources to support that work within a clearly defined execution plan is key to keeping costs low.
- Higher costs may be encountered in integrating mining operations with TSF construction:
  - Waste rock material types with low potential for ARD and ML (NAG4 and NAG5) and overburden will be required for TSF dam construction. If suitable waste rock and/or overburden is unavailable when required or in the areas of the pit planned for initial development, then the mine plan may need to target this better-quality material more specifically elsewhere, leading to alterations to the mine plan;
  - Waste characterization for the Project is based on a comprehensive geochemical database and model and the testing to date, which found that NAG4 and NAG5 waste rock behave in a similar fashion. Haulage costs and sustaining capital (e.g., more haul trucks) may be higher if more NAG waste rock is found to be ML and needs to be disposed of in the TSF;
  - When the detailed waste release schedule, the TSF dam construction schedule, and the material capture rates are developed, it may become apparent that additional trucks are required for longer hauls of waste materials;
- Project delays: project delays could result from uncertainty in the federal and provincial environmental permitting process, including the time required to prepare permit applications and the duration of the technical review of the permit applications by regulators and other stakeholders
- An amendment to Schedule 2 of the Metal and Diamond Mining Effluent Regulations is also required to authorize use of the proposed TSF for mine waste disposal;
- Camp accommodation construction: degradation in the quality of the existing camp due to unforeseen abuses or damage may delay the availability of beds coming online at the start of construction or require expenditure of capital to replace or refurbish;
- Camp accommodation construction: higher construction costs may be encountered as no specific geotechnical investigation has been carried out in this area;
- Airstrip: higher construction costs may be encountered as no specific geotechnical investigation has been carried out in this area;
- Infectious disease becomes more serious, additional variants emerge forcing lockdown measures and delaying site work;
- Heavy equipment vendors (mill, crushers etc) become overloaded and lead times blow out causing delay to schedule;

- Mineral construction industry becomes much busier, leading to shortages of skilled labour. Subcontractors cannot fully man-up and forecast productivities are not met; and

- International shipping becomes more difficult to arrange and costs escalate, delivery dates are not able to be maintained.

24.1.2 Opportunities

Project opportunities that were identified include:

- Mineral resources:

  - Mineralization remains open at depth under the planned open pit and may represent an upside opportunity for future pit expansion with additional drilling;

- Mining equipment:

  - A high-level study in early 2013 assessed the potential for implementing a trolley- assist system for the haul trucks. The assessment found that this option could possibly reduce costs in the later years of the mine life, depending upon diesel and electricity costs at the time;

  - Once drilling experience is gained at site, it may be possible to use a larger class of machine capable of drilling larger holes, thus reducing drilling and blasting costs;

  - Mine fleet and drill automation;

- Mine operations:

  - Waste production could also be reduced, and additional mineralization may be able to be recovered if pit slope angles could be increased;

- Process plant operation:

  - Process facility automation;

  - Oxygen can be used instead of compressed air for cyanide leaching and cyanide detoxification. Although high purity oxygen is not required for the Project, early-stage testing found it improved kinetics and potentially increased metal recovery; this could lead to a reduction in the number of leach tanks required and/or increased metal recoveries. Oxygen could be supplied by an on-site cryogenic plant (99% pure oxygen), a vacuum swing adsorption oxygen system (90% pure oxygen), or by trucking to site (liquid oxygen). The vacuum swing adsorption option is significantly less expensive than either of the other two alternatives. Further testing of oxygen sparging could be conducted in the operating plant or in a large-scale continuous pilot plant;

- Kluskus Forest Service Road

  - Ongoing maintenance of the road by existing users may reduce the effort and cost required to upgrade the road

- TSF and Water Management Structures construction:
    - The cost of construction for the initial works associated with the TSF and water management may be less than estimated depending on how the work is executed. The estimated costs are reliant on the plan to have the mine equipment and staff available to prepare haulage routes and transport, dump and spread a portion of the dam materials. Providing mining resources to support this work and managing the scope, contractual terms, and seasonality of the contractor construction works within a clearly defined execution plan that fits with the overall development plan for the project may be able to provide better value.
    - Alternative lining methodologies may be evaluated following geotechnical investigations at the WMP, which will be performed to verify that the ground conditions are consistent with design assumptions and to assess if natural ground conditions within the basin are more favourable than assumed in the design.
    - The actual rate of TSF filling and stage construction during operations will be affected by a variety of factors, including the mining rate and pit development sequence, waste rock quality, ore processing rates achieved at the mill, tailings beach slopes, the supernatant pond area and volume, variability of tailings density throughout the facility, and on-going consolidation of the tailings mass throughout operations. It may be possible to reduce TSF construction requirements and associated costs during the staged raises of the TSF.
- Value engineering:
    - Evaluate additional on-site borrow sources to reduce borrow haul distances;
    - Re-evaluate the water management system to simplify design;
    - Optimize the layout and construction of the low-grade stockpile to simplify low- grade ore placement and the drainage water collection system;
    - Optimize tailings and PAG waste rock deposition plans during mine operations to simplify closure plan;
    - Evaluate alternative power generation such as wind generation or similar;
    - For phased plant expansions, perform trade-off studies to evaluate the economics of different processing methods.

25 INTERPRETATION AND CONCLUSIONS

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

- Artemis holds 100% recorded interest in 329 mineral claims covering an area of 148,902 ha distributed among the Property and the Capoose, Auro, Key, Parlane and RJK claim blocks;
- The Blackwater Property claim block comprises 76 mineral cell claims totalling 30,791 ha. All Blackwater claims are 100% held in the name of Artemis. All claims expire in 2022. There are no other parties with beneficial interests in these mineral rights. None of the Blackwater cell claims are known to overlap any legacy or Crown granted mineral claims, or no-staking reserves;
- The Blackwater deposit spans the Davidson claim (509273), the Dave claim (515809); and the Jarrit claim (515810);
- A review of surface rights in the vicinity of the Property was undertaken in September 2020. The majority of the Blackwater mineral claims comprising the Property are located on Crown lands. The review identified an overlapping private parcel, land reserves/notations, a transfer of administration/control area, grazing tenures, forest recreation sites, forest tenures, trap lines, guide outfitter areas, and an Ungulate Winter Range. Sixteen (16) of the Capoose claims have minor portions overlapping onto Entiako Provincial Park;
- A review of surface rights in the vicinity of proposed electrical transmission line, water pipeline, and access roads (Linear Infrastructure) was undertaken in December 2013 and in September 2020. This review identified private parcels; a Land Act license, rights of way, reserves/notations and a transfer of administration/control area; grazing tenures; forest tenures; a forest recreation sites; traplines; guide outfitter areas; a wildlife management area; an agriculture land reserve; and third-party mineral tenures overlapping or in close proximity to the proposed electrical transmission line route. The review also identified grazing tenures, forest tenures, traplines, and guide outfitter areas overlapping all elements of the Linear Infrastructure; a forest recreation site overlapping the proposed water pipeline route; and third party mineral tenures overlapping the access road, and the water pipeline Linear Infrastructure route;
- Artemis 100% interest in the Blackwater claim block is subject to three net smelter return (NSR) agreements:
    - Dave Option: A 1.5% NSR royalty is payable on mineral claim 515809 (Dave Claim). The claim covers a portion of the Blackwater deposit.
    - Jarrit Option: A 1% NSR royalty is payable on mineral claim 515810 (Jarrit Claim). The claim covers a portion of the Blackwater deposit.
    - JR Option: The current agreement would allow Artemis to purchase two-thirds of three Blackwater Claims (637203, 637205, and 637206) NSR royalty for C$1,000,000 at any time, such that a 1% NSR royalty would remain.

Only the royalties with respect the Dave Option and the Jarrit Option exist within the current Mineral Reserves.

- Artemis' 100% interest in the property, assets and rights related to the Blackwater Project and six contiguous claim blocks (Blackwater, Capoose, Auro, Key, Parlance and RJK) is subject to the following considerations:
  - A secured gold stream participation in favor of New Gold, whereby New Gold will purchase 8.0% of the refined gold produced from the Project. Once 279,908 ounces of refined gold have been delivered to New Gold, the gold stream will reduce to 4.0%.
  - New Gold 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. In the event that commercial production at Blackwater is not achieved by the 7th, 8th, or 9th anniversary of Closing, being August 21, 2020, New Gold will be entitled to receive additional cash payments of $28 million on each of those dates;
- New Gold has a security interest over the Project in connection with the gold stream agreement.

25.2 All other material encumbrances within the Blackwater claim blocks are listed in Section 4.Property description and locationGeology and Mineralization

- Knowledge of the deposit settings, lithologies, and structural and alteration controls on mineralization, and the mineralization style and setting is sufficient to support Mineral Resource estimation.
- The deposit is considered to be an example of an intermediate sulphidation epithermal system.
- The deposit type used for exploration targeting is appropriate to the mineralization identified and the regional setting.

25.3 Exploration, and Drilling

The exploration programs completed to date are appropriate to the style of the known mineralization within the Project area;

- Given the lack of bedrock exposure, no detailed surface geologic mapping has been carried out over the main deposit or surrounding, and geologic information has been obtained primarily by core drilling. Areas of shallow overburden near the centre of the deposit are potential targets for bulk sampling or trench mapping/sampling programs.
- Geophysical surveys have proven useful to assist in interpreting deposit geology and identifying drill targets for future exploration.
- The resolution and accuracy of the surface topography as interpreted from the 2011 LiDAR survey are considered sufficient to support detailed Project studies.
- The total sample database for the Blackwater Gold Project contains results from 1,053 core holes totalling 324,839 m drilled between January 1987 and January 15, 2013. Due to lack of QA/QC and accurate survey information, holes drilled before 2009 were not used for statistical analysis, or grade estimation.
- Gold and silver mineralization occurs within an irregularly-shaped system of stockwork and disseminated sulphides that strikes approximately east–west and dips moderately to the north.

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

25.4 Sample Preparation and Analysis

- Sampling methods for the drillhole data used in the block model are acceptable, meet industry-standard practice, and are acceptable for Mineral Resource and Mineral Reserve estimation and mine planning purposes.
- Bulk density determination procedures are consistent with industry-standard procedures, and there are sufficient bulk density determinations to support tonnage estimates.
- Analysis is performed by accredited third-party laboratories.

25.5 Data Verification

- Verification has been performed on all digitally-collected data, and includes checks on surveys, collar co-ordinates, lithology data, and assay data. The checks are appropriate, and consistent with industry standards.
- The process of data verification performed by the QP indicates that the data collected from the Project during the 2009 to 2013 work programs adequately reflect deposit dimensions, true widths of mineralization, and the style of the deposit, and adequately supports the geological interpretations, and the analytical and database quality.
- QA/QC with respect to the results received to date for the 2009–2013 exploration programs is acceptable, and protocols have been well documented.

25.6 Metallurgical Testwork

- Extensive metallurgical testwork program was carried out over the period 2008 to 2013 on samples that were composited to represent process plant feed in the mine development plan. This testwork was performed by industry recognized metallurgical laboratories.
- An extensive program of testwork was carried out in 2019 by BaseMet. As the work was carried out as an integrated whole by one laboratory, with consistent laboratory techniques and analysis, this work was relied upon for generating the process design.
- The basic leach conditions were first determined using composites made up of samples representing expected grades over the first ten years of mining.
- A P80 grind of 150 µm was confirmed, as were the requirements for pre-aeration and a somewhat long leach time of 48 hours.
- 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.

- A further 48 samples were taken from drillholes distributed throughout the deposit. All of these were treated using the proposed flow sheet. In addition to these tests, some comminution testing was carried out and cyanide destruction was also tested, using SO_2/air.
- Carbon loading data and slurry rheological properties were also determined.
- Estimated recoveries over the LOM are gold recovery of 93% and silver recovery of 65%.

25.7 Mineral Resource Estimate

- The mineral resource estimate for the Project conforms to industry best practices, and meets the requirements of CIM (CIM, 2014) following the updated CIM guidelines (CIM,2019).
- The estimate is based upon a geologic block model that incorporates 288,738 individual assays from 309,293 m of core from 1,002 drillholes.
- Due to lack of QA/QC and accurate survey information, holes drilled before 2009 were not used for statistical analysis or grade estimation of the Mineral Resource but were used in forming the lithological wire frame construction.
- The Mineral Resource estimate is based on reasonable assumptions of eventual economic extraction and assuming open pit mining method. An AuEq cut-off value of 0.20g/t is the base case cut-off.
- Measured and Indicated Mineral Resources total 597 Mt at 0.61 g/t Au and 6.4 g/t Ag.
- Inferred Mineral Resources are estimated at 17 Mt grading 0.45 g/t Au and 12.8 g/t Ag.
- The following factors could affect the Mineral Resources: 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.

25.8 Mineral Reserve Estimates

Proven and Probable Mineral Reserves have been modified from Measured and Indicated Mineral Resources. Inferred Mineral Resources have been set to waste. The Mineral Reserves are supported by the 2021 Feasibility Study Mine Plan and classified in accordance with the 2014 CIM Definition Standards for Mineral Resources and Reserves.

Blackwater Mineral Reserves total 334.3 Mt at 0.75 g/t Au and 5.8 g/t Ag (0.78 g/t AuEq).

Factors that may affect the Mineral Reserve estimates include metal prices and foreign exchange rate, changes in interpretations of mineralisation geometry and continuity of mineralisation zones, geotechnical and hydrogeological assumptions, ability of the mining operation to meet the annual production rate, operating cost assumptions, process plant and mining recoveries, the ability to meet and maintain permitting and environmental licence conditions, and the ability to maintain the social licence to operate.

25.9 Mine Plan

A reasonable open pit mine plan, mine production schedule and mine capital and operating costs have been developed for Mineral Reserves at Blackwater.

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 Feasibility Study.

25.10 Process

The process plant will be constructed in three distinct phases, and the process for Phase 1 will consist of:

- Three stage crushing, consisting of a primary gyratory crusher, a secondary cone crusher and two tertiary cone crushers. Crushed ore will be stored on a covered stockpile.
- Crushed ore will be conveyed from the stockpile to a single, 7.3 x 12.5 m, 14 MW ball mill for grinding, with the circuit being closed by cyclones. Gravity concentration will be incorporated into the grinding circuit using centrifugal concentrators with an intensive cyanide leach unit for recovering gold from the gravity concentrate.
- The leach-adsorption circuit will consist of one pre-aeration tank, three leach tanks and seven CIL adsorption tanks fitted with mechanical agitators, with cyanide being added to the second and subsequent tanks. The combined leach and adsorption circuit residence time will be 24 hours.
- Carbon in leach adsorption of gold and silver will be carried out in a counter-current circuit, with barren carbon entering the adsorption circuit at CIL Tank 7. The carbon will advance counter current to the main slurry flow using carbon advance/transfer pumps from a downstream to upstream tank. Carbon will be retained in the CIL tanks by intertank screens. Loaded carbon will be periodically removed from CIL Tank 1.
- The loaded carbon will be treated in 12 ton batches in an AARL elution and electrowinning circuit consisting of an acid wash column and an elution column operating at 120 degrees Celsius. An electric heater will provide the necessary temperature and an additional heat exchanger will control the temperature around the circuit. A rotary kiln operating at 750 degrees Celsius will be used to maintain carbon activity. Electrowinning will be carried out to recover gold and silver from the elution solution and the resulting metallic precipitate will be dried and smelted to doré bullion.
- Cyanide destruction using an SO_2/O_2system will be carried out in the final tailings slurry, with the sulphur dioxide being produced by the combustion of elemental sulphur.
- The Phase 2 expansion will treat an additional 6.0 Mtpa (12 Mtpa total) through minor upgrades to the Phase 1 crushing circuit, and addition of new milling, leaching, adsorption, elution, and detox capacity.
- The Phase 3 expansion will include a new process line consisting of crushing, grinding, leaching, adsorption and detox circuits with a capacity of 8.0 Mtpa. The Phase 1 and 2 acid wash, elution, electrowinning and gold room facilities will be used, and combined throughput will increase from 12 Mtpa to 20 Mtpa.

25.11 Onsite Infrastructure

- Access to the Project from highway 37, west of Vanderhoof is via the Kluskus and Kluskus-Ootsa FSRs for approximately 124 km, then a new road will be built, 15.6 km long, to reach the mine plant site (Figure 18-2). Presently, the site is reached by another, longer route known as the exploration road, which will be partially decommissioned following completion of the new mine access road. The remaining portions of the exploration road within in the mine site boundary will be used for local construction access and mine operations. Sections of the exploration road located within the TSF will be inundated in approximately Year 6.

- An existing on site exploration camp is available and is in excellent condition. It has a capacity for 250 persons and has a dining hall, kitchen, recreation room etc. The location of this camp is too close to the ultimate pit for use for the entire LOM but can be used during construction and in the first years of operation. An additional permanent camp will be built to the northeast of the current camp facilities. This camp will be fully self-contained with dining facilities, a recreation area etc. and will initially have a capacity for 240 people. This will give a total capacity during the construction phase for 490 persons. Part or all of the present camp will be moved to the permanent camp area at the end of the first-stage construction phase to provide a total of 490 beds. The camp will be further expanded to accommodate the construction personnel for Stage 3. The accommodation will be supported in all phases with back-up power, potable water generation, fire water systems, and sewage collection and treatment systems.

- Power will be supplied to the Blackwater site by connection to the BC Hydro grid. A 135 km long 230 kV transmission line will be constructed from the BC Hydro Glenannan Substation to the Blackwater site.

- Wells will be developed near the new camp area to supply water for the temporary and operations camps. The water will be treated and distributed around the camp site for domestic use.

- Fresh water for the Project will be sourced from Tatelkuz Lake, approximately 20 km northeast of the mine site, to offset flow reductions in Davidson Creek downstream of the TSF.

25.12 Waste Characterization

Mine waste classification was based on material type, ARD potential and ML potential. Waste is defined as potentially acid generating (PAG) or non-potentially acid generating (NAG) based on neutralization potential ratio (NPR), with NPR values ≤ 2 being considered PAG. NAG waste rock is further classified based on ML potential. The waste classes NAG3, NAG4 and NAG5 are defined based on Zn content, with NAG3 being considered high ML potential and NAG4 and NAG5 having similarly low ML potential. The overall classification criteria for the Blackwater Project are as follows:

- Ore and Low-grade ore (PAG)
- Tailings (PAG)
- Overburden (NAG)
- Waste Rock
  - PAG1 – NPR ≤ 1
  - PAG2 – 1 < NPR ≤ 2

- o NAG3 – NPR > 2 and Zn >1000 mg/kg
- o NAG4 – NPR > 2 and 600 mg/kg < Zn < 1000 mg/kg
- o NAG5 - NPR > 2 and Zn < 600 mg/kg

Collectively, these mine rock units have been incorporated into the mine plan using the ARD Block Model and are used to define waste management. The onset to ARD from PAG mine waste at the Project is expected to be relatively rapid (<1 year). This ARD potential can be mitigated through submergence of PAG mine waste within the TSF impoundment. Special handling will also be required of NAG3 waste rock. This can be accomplished by placing NAG3 in internal zones of the TSF embankment or co-disposal within the TSF where it will ultimately be saturated. Low metal leaching mine waste is suitable for construction and consists of NAG4, NAG5 and overburden.

25.13 Tailings Storage Facility

- The TSF was designed to permanently store tailings, PAG waste rock, and potentially ML NAG waste rock that is generated during operation of the mine. The facility was designed to hold 469 Mm^3 of tailings and waste rock material and up to 12 Mm^3 of pond storage under normal operating conditions. The TSF comprises two adjacent sites, TSF C and TSF D.

- TSF C will be constructed first to provide storage capacity for start-up of the process plant. It was designed to contain tailings for approximately 21 years of mine operations and PAG/ML waste rock generated during the first six years of mining. TSF D will be formed adjacent to and downstream of TSF C beginning in Year 5, during the Phase 2 expansion, to provide additional storage capacity to contain PAG/ML waste rock generated between Year 6 and the end of mining and up to two years of tailings beginning in approximately Year 21 when TSF C reaches design capacity.

- TSF C will comprise a valley-fill style impoundment formed by construction of three embankments (Main Dam C, the West Dam, and the Saddle Dam) in the upper reaches of the Davidson Creek drainage basin. Main Dam C will be constructed during the preproduction phase to form TSF C and will be raised annually through Year 6 using centreline construction methods. Thereafter, the dam will be raised periodically in stages approximately 8 m high using downstream construction methods comprise zoned earth-rockfill dams complete with high density polyethylene (HDPE) geomembrane facing. The West Dam and Saddle Dam will be constructed in Years 6 and 12, respectively, and raised periodically in stages along with later stages of Main Dam C.

- TSF D will be formed by construction of one embankment (Main Dam D). The embankment will be engineered, water retaining, zoned earth-rockfill dams with a compacted low-permeability seal zone and appropriate filter/transition zones. Main Dam D will be raised annually using centreline construction methods.

- Local material borrows will be developed to provide fill materials as required for construction of the TSF and water management facilities. Previous and low-permeability materials for the seal zones and shell zones are naturally occurring and will be sourced from local glacial till and glaciofluvial borrow areas. Engineered fill materials, such as the dam filter materials, riprap, and road wearing course, will require processing and will be sourced from the local esker borrow areas. An esker complex was identified in close proximity to the Main Dam C Stage 1 construction area. A larger esker complex is also located further downstream along the planned Mine Access Road adjacent to the location of the FWR. The local esker complex described above is an extension of this larger complex further downstream.

- The feasibility study is supported by the detailed design of the Stage 1 TSF, and associated facilities required for the start of mine operations. These designs form the basis of the joint Mines Act / Environmental Management Act Permit Application.

- The designs are supported by extensive geotechnical and hydrogeological site investigations completed between 2012 and 2021. Uncertainty in the existing site investigation and in-situ testing results is primarily due to large footprint area of the facilities and spacing between the investigation information. Recommendations for additional supplemental investigations to be executed prior to and during the initial construction of the mine (Years -2 and -1) and extending over the first several years of operations (Years 1 to 3) are included in Section 26.

25.14 Water Management Structures

- The water management facilities planned for the Project provide the flexibility to manage water in a manner that allows for the beneficial use of the water to support mine operations and to divert flows not needed by the mine to the downstream receiving environment.

- The feasibility study is supported by the detailed design of the water management facilities required at the start of mine operations. These designs form the basis of the joint Mines Act / Environmental Management Act Permit Application.

- A water balance model was prepared that explicitly considers the full life cycle of the Project. Model results suggest the water inventory of the TSF supernatant ponds under average climate conditions can be maintained within the target operating range through the mine life. Under wetter than average conditions during Operations, model results suggest treatment and release of water from the TSF supernatant pond may be required to maintain the pond water inventory in the target operating range while under drier than average conditions a greater amount of non-contact water is needed to meet mill demand.

- Planned installation of water treatment plants at the start of operations will enhance water management flexibility and allow for treatment of mine site contact water to meet discharge criteria, if required.

25.15 Environmental Considerations

The Project is supported by a full set of environmental, social, economic, and cultural heritage baseline studies. The potential Project effects to environmental, social, economic, and cultural heritage components have been fully assessed. The Project has been granted an EAC #M19-01 on June 21, 2019 (EAO 2019c) under the 2002 *Environmental Assessment Act* and an Environmental Assessment DS on April 15, 2019 under the *Canadian Environmental Assessment Act, 2012* (CEA Agency 2019b). Assessment of environmental components to updates in the Project design have been considered in recent permits (Table 20-1) under permitting applications, currently in progress (Table 20-2 and Table 20-3). To manage potential effects of the Project, an Environmental Management System supported by a comprehensive set of management plans are in the progress of being developed for permitting phase of the Project (Section 20.6.1).

25.16 Closure and Reclamation Plan

The Closure and Reclamation Plan will take advantage of progressive reclamation opportunities through the life of the mine. In particular, mining operations will cease in the Open Pit in the later Operations phase. Closure of major mine infrastructure is anticipated to take one to three years after cessation of ore processing from stockpiled ore. Reclamation of the Project area will conform to the requirements of the Health, Safety, and Reclamation Code for Mines in BC (BC EMLI 2021). As much as possible, disturbed areas will be reclaimed to native ecosystems and waterways restored to pre-disturbance flow patterns. In the extended Closure and Post-closure phases, activities will focus on monitoring vegetation and geotechnical stability of reclaimed areas, and water treatment, as required.

25.17 Permitting

The Project is entering the permitting phase. Provincial and federal permits, licenses, and authorizations to approve the construction and operation of the Project have been identified and are in progress (Tables 20-2 and 20-3). Key federal approvals include: impact 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: permit approving mine plan and reclamation program (*Mines Act*) and effluent discharge permit and air discharge permit (*Environmental Management Act*). Mines Act permit M-246 issued in mid-2021 authorizes early works site clearing and initial mine site road construction. Approvals to construct the Mine Access Road have been granted.

25.18 Social License

Consultation with all First Nations, government, and other stakeholders to the Project is on-going. The intent of the consultation is to increase the mutual awareness and understanding of the Project and its potential effects, and to explore potential strategies to mitigate negative effects and enhance positive ones.

Moving forward engagement with Indigenous nations will be guided by agreements with Artemis Gold and conditions in the Project's EAC M#19-01 (EAO 2019b) and federal DS (CEA Agency 2019). Engagement with stakeholders and the public will be guided by the Project's EAC and DS conditions and permits.

Pursuant to EAC Condition 37, BW Gold has established a Community Liaison Committee (CLC) to provide information to BW Gold on Project effects in members' communities and mitigation measures to address potential social and economic effects. Committee members include the District of Vanderhoof, Village of Fraser Lake, UFN, LDN, NWFN, SFN, STFN, City of Quesnel, Electoral Area I of Cariboo Regional District, Electoral Area F of Regional District Bulkley Nechako, Northern Health, Nechako Environment and Water Stewardship Society and College of New Caledonia. The CLC will be in place throughout construction, operations and the first five years of closure.

25.19 Capital Costs

- The initial capital cost for Phase 1 at 6.0 Mtpa is estimated at $645.2 million.
- The expansion capital cost for Phase 2 (12 Mtpa) and Phase 3 (20 Mtpa) is estimated at C$347 million and C$374 million, respectively, for a total of C$721 million.
- The sustaining capital cost is estimated at C$830.7 million;
- The deferred capital cost is estimated at $51.6 million;
- Total LOM capital is estimated to be $2,248.2 million.

25.20 Operating Costs

- LOM operating costs are estimated at $17.96/t of ore milled;
- LOM all-in sustaining cash costs are estimated at C$850/oz Au recovered.

25.21 Financial Analysis

For the 22-year mine life and 334 Mt mill feed, the following after-tax Base Case financial parameters were calculated:

- $2,151 million NPV at 5.0% discount rate;
- 32% IRR;
- 2.3 year initial capital payback

For a leveraged case the following after-tax financial parameters were calculated:

- $2,158 million NPV at 5.0% discount rate;
- 43% IRR;
- 2.4 year initial capital payback.

The Leveraged Case is based on the following additional assumptions:

- C$360 million (plus up to C$25 million in capitalized interest) in project debt financing
- Annual interest rate of Canadian Dollar Offered Rate (assumed at 0.5% in the Study) plus a margin of 4.25% up to the date of completion, with the margin reducing to 3.75% once the Project is effectively in commercial production;
- Customary upfront and standby financing fees;
- Six-year term post commencement of commercial production with Principal and capitalized interest repayable in quarterly instalments over six years, commencing one year following achievement of commercial production, with a repayment holiday during years 4 and 5 of production while the company expects to undertake its Phase 2 expansion;
- Expansion capital is assumed to be funded through operating cashflow.

NPV sensitivity analysis was performed on the Project Base Case using gold price, exchange rate, operating costs and initial capital costs. The impacts of changes in the gold grade mirror the impact of changes in the gold price The Project is more sensitive to changes in the gold price (grade) and the USD:CAD exchange rate than to changes in capital or operating costs.

25.22 Risks and Opportunities

- The major risks to the Project were identified as:

  - Changes to metal prices and exchange rate assumptions;
  - Increases in capital costs;
  - Increases in operating costs;
  - Productivity assumptions;
  - Equipment and commodities delay due to supply chain constraints;
  - Increased dilution;
  - Presence of high-grade silver in the mill feed could affect leach times;
  - Integration of mining operations and the TSF construction
  - Permitting delays;
  - Lack of social license affecting permit grant
  - Escalation of infectious disease and related restrictions;
- Project opportunities included:
  - Delineation of additional mineralization that could potentially support higher-confidence resource categories through additional drilling;
  - Use of a trolley assist system later in the mine life;
  - Assessment of methods to reduce waste mining costs;
  - Mine fleet and drill automation;
  - Process facility automation;
  - Value engineering initiatives.

25.23 Conclusions

The QP considers that the scientific and technical information available on the Project can support further development of the Project. However, the decision to proceed with a mining operation on the Project is at the discretion of Artemis.

26 RECOMMENDATIONS

26.1 Introduction

The QPs propose the following recommendations as a single-phase work program. All recommendations can be conducted concurrently.

26.2 Exploration and Resource

- It is recommended to extend the grade control program and analyses to test the continuity of mineralization in additional areas of the deposit. The costs related to extending the grade control program are estimated to be $500,000 and are included in the estimated mining costs for the project, described in Section 21.2.2

- It is recommended to continue with structural interpretation of the deposit to better understanding the faulting and its potential influence on mineralization, particularly at depth.

26.3 Metallurgy

The following metallurgical testwork is recommended:

- Although the composite for the first 10 years of operation showed gold recoveries in excess of 93%, 6 samples out of 48 variability samples showed lower than 90% gold extraction. This warrants further investigation, and more testwork is proposed. The cost related to the testwork is estimated to be C$50,000.

26.4 Mineral Reserves and Mine Planning

The following recommendations are made with regard to future mining studies:

- Incorporate results of recent grade control drilling program and exploration holes into ongoing grade control strategy and mine planning.

- Incorporate updated open pit geotechnical interpretation (KP, 2021o) into updated set of open pit designs.

- Complete approximately two oriented core geomechanical drillholes in the Southeast and East pit design sectors to refine the boundaries between the Broken Zone and Competent Zone in this area for early phase pit development. Complete one to two oriented geomechanical drillholes in the Northwest pit design sector to refine the pit slope design in this area. Complete several vertical sonic drillholes 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.

- 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 trolley assist, crushing and conveying systems, or other electrification initiatives for transporting waste rock from the open pit to the TSF.

- Further refine existing design work to a construction level of engineering, including open pits, stockpiles, ex-pit haul roads, explosives storage facilities, and fleet maintenance facilities.

- Investigate other technologies to optimize material movement from the open pit to various mine infrastructures.

These field programs and studies are estimated to cost approximately C$1,500,000.

26.5 Geotechnical and Hydrological

Additional work was outlined in order to progressively refine the site geological model, verify conditions assumed for detailed design of the Stage 1 TSF and water management structures, and collected supplemental information supporting the detailed design of subsequent stages of the TSF. The recommendations are broken down into a phased five-year investigation plan to be executed prior to and during the initial construction of the mine (Years -2 and -1) and extending over the first several years of operations (Years 1 to 3).

The focus areas for further site investigation during initial construction of the mine include the following:

- Field verification of 2012-2013 drillhole logs, to the extent practicable, to confirm revised interpretation.

- Conduct seismic refraction surveys along the Main Dam C alignment to confirm the bedrock profile.

- Conduct geotechnical drilling investigations at the WMP to verify that the ground conditions are consistent with design assumptions, and to evaluate alternatives to the WMP lining strategy if natural ground conditions within the basin are more favorable than assumed in the design.

- Conduct geotechnical drilling investigations within the IECD footprint area to verify ground conditions at the maximum section of the dam.

- Conduct additional geotechnical and hydrogeological investigations between the MDC alignment and IECD to verify the extent and connectivity of the inferred glaciofluvial subglacial corridor and further investigate the nature and consistency of the glaciolacustrine units present in this area.

- Conduct in-situ testing of near surface glaciolacustrine materials at the FWR.

The TSF and water management structures investigation programs and studies are estimated to cost approximately $2,000,000 prior to and during the initial construction of the mine.

Additional site investigation work is recommended within the stockpiles area prior to construction, which will be performed to verify that the ground conditions are consistent with design assumptions. The recommended work program will include geotechnical drilling and in-situ testing at select locations within the stockpile footprints, and test pits to further investigate the nature and consistency of the surficial materials in the general vicinity of the proposed stockpiles. The stockpile area investigation programs and studies are estimated to cost approximately C$1,000,000.

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