Blackwater Gold Project British Columbia

NI 43-101 Technical Report on Pre-Feasibility Study

Submitted to:
Artemis Gold Inc.

Effective Date:
26 August 2020

Report Authors:	**Company:**
Sue Bird, P.Eng.	Moose Mountain Technical Services
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John Thomas, P.Eng.	JAT Met Consult Ltd.

Certificate of Qualified Person

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 British Columbia, NI 43-101 Technical Report on Pre-Feasibility Study” that has an effective date of August 26, 2020 (the “technical report”).

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

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

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

I visited the property on July 14, 2020.

I am responsible for Sections 1.1, 1.3, 1.4, 1.4.1, 1.4.2, 1.4.3, 1.4.4, 1.5, 1.5.1, 1.5.2, 1.5.3, 1.5.4, 1.6, 1.7, 1.8, 1.9, 1.10, 1.11, 1.12, 1.13, 1.14, 2.1, 2.2, 2.3, 2.4.1, 2.5, 2.6, 2.7, 3, 4, 5.1, 5.3, 5.4, 5.7, 6, 7, 8, 9, 10, 11, 12, 14, 23, 25.1, 25.2, 25.3, 25.4, 25.5, 25.6, 25.8, and 26.1.1 of the technical report related to the Mineral Resource.

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

I have 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 the technical report not misleading.

Dated: 18 September 2020

{Signed and Sealed}

Signature of Qualified Person

Sue Bird, P.Eng.

Certificate of Qualified Person

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

This certificate applies to 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 (the "technical report").

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

I have practiced my profession for 14 years since graduation from university. I have been directly involved in performing and overseeing geotechnical engineering design, tailings management and water management studies, environmental assessments, and monitoring construction activities for mining projects during this time.

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

I visited the Blackwater Gold Project site most recently on July 23-24, 2019 as well as on several other occasions previously between 2011 and 2016.

I am responsible for Sections 1.19, 1.20, 1.21, 1.22, 1.23, 1.24, 2.4.2, 5.2, 5.5, 5.6, 18.3, 18.4, 18.5, 18.6, 18.7, 18.8, 18.15, 20, 25.13, 25.14, 25.15, 25.16, 25.17, 25.18, 25.19, and 26.1.4 of the technical report related to climate conditions and seismicity, geotechnical investigations, mine waste characterization and management, the tailings storage facility, water management, fresh water supply system, and the transmission line.

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

I have not previously co-authored a technical report on 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 the technical report not misleading.

Dated: 18 September 2020

{Signed and Sealed}

Signature of Qualified Person

Daniel Fontaine, P.Eng.

Certificate of Qualified Person

I, Tracey Meintjes, P.Eng., of Oliver BC do hereby certify that:

I am a Metallurgical Engineer with Moose Mountain Technical Services with a business address at 1975 1st Avenue South, Cranbrook, BC, V1C 6Y3.

This certificate applies to 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 (the "technical report").

I am a graduate of the Technikon Witwatersrand, (NHD Extraction Metallurgy – 1996)

I am a member in good standing of the Association of Professional Engineers and Geoscientists of British Columbia (#37018).

My relevant experience includes metallurgy, process engineering, and mine planning in North America, South America, Africa, and Europe. My experience includes both operations and metallurgical process development including base metals, precious metals, industrial minerals, coal, uranium and rare earth metals, including project financial analysis. I have been working in my profession continuously since 1996.

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

I am a "Qualified Person" for the purposes of National Instrument 43-101 (the "Instrument").

I visited the Blackwater Gold Project on 07 May 2020.

I am responsible for Sections 1.2, 1.25, 1.26, 1.27, 1.28, 1.29, 2.4.3, 19, 21.1.2, 21.1.4, 21.1.5, 21.1.6, 21.1.7, 21.2.5, 21.3, 22, 24, 25.20, 25.21, 25.22, 25.23, 25.24, 26.2, and 27 of the technical report related to the economic analysis.

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

I have 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 the technical report not misleading.

Dated: 18 September 2020

{Signed and Sealed}

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Signature of Qualified Person

Tracey D. Meintjes, P.Eng.

Certificate of Qualified Person

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 British Columbia, NI 43-101 Technical Report on Pre-Feasibility Study" that has an effective date of August 26, 2020 (the "technical report").

I am a member of the self-regulating Association of Professional Engineers, Geologist and Geophysicists of Alberta (#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.

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

My most recent site visit to the Blackwater Gold Project was on July 14, 2020.

I am responsible for Sections 1.15, 1.17, 1.25, 1.26, 2.4.4, 15, 16, 18.1, 21.1.2, 21.1.3, 21.1.6, 21.1.7, 21.2.2, 25.9, 25.10, 25.20, 25.21, 26.1.3 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 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 the technical report not misleading.

Dated: 18 September 2020

{Signed and Sealed}

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Signature of Qualified Person

Marc Schulte, P.Eng.

Certificate of Qualified Person

I, John Alan Thomas, work as a process engineer, being the president of JAT Met Consult Ltd. of 5766 Goldenrod Crescent, Delta, BC V4L 2G6.

This certificate applies to the technical report titled "Blackwater Gold Project British Columbia, NI 43-101 Technical Report on Pre-Feasibility Study" that has an effective date of August 26, 2020 (the "technical report").

I am a member of the self-regulating Association of Professional Engineers and GeoScientists of British Columbia (#125986). I graduated with Chemical Engineering degrees (B.Sc. M.Sc. and Ph.D.) from the University of Manchester in 1969, 1971 and 1973 respectively.

I have worked as a process engineer for over 47 years since my graduation from university. I have worked on precious metals and base metals, including process development, engineering, project management and mine operation. Similar projects specifically include those done for Atlantic Gold, Infinito Gold, Bolivar Gold Corp., Bolivar Goldfields and Star Mining.

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

I visited the Blackwater Gold Project on May 7th, 2020.

I am responsible for Sections 1.16, 1.18, 1.25, 1.26, 2.4.5, 13, 17, 18.2, 18.9, 18.10, 18.11, 18.12, 18.13, 18.14, 18.15, 21.1.2, 21.1.6, 21.1.7, 21.2.3, 21.2.4, 25.7, 25.11, 25.12, 25.20, 25.21, and 26.1.2 of the technical report.

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

I have 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 the technical report not misleading.

Dated: 18 September 2020

{Signed and Sealed}

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Signature of Qualified Person

John A. Thomas P.Eng.

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1 SUMMARY

1.1 Introduction

Moose Mountain Technical Services (MMTS), Knight Piésold Ltd. (KP), and JAT Met Consult Ltd. (JAT Metco) have prepared a technical report (the Report) for Artemis Gold Inc. (Artemis) on a pre-feasibility study (PFS) evaluation (2020 PFS) of the Blackwater Gold Project (the Project), located in British Columbia, Canada.

BW Gold Inc. (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.

For the purpose of this report the property (the Property) comprises six contiguous claim blocks held by BW Gold Ltd. (Blackwater, Capoose, Auro, Key, Parlane and RJK).

The Blackwater Project (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 2020 PFS are:

- At a 0.20 g/t gold equivalent (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.
- Proven and Probable Mineral Reserves total 334.0 Mt at 0.75 g/t Au and 5.78 g/t Ag.
- Ore processing commences with a nominal milling rate of 15,000 t/d (5.5 Mtpa, Phase 1). The ore processing facilities will be expanded to achieve 33,000 tpd (12 Mtpa, Phase 2) starting in Year 6 with a final expansion to achieve 55,000 t/d (20 Mtpa, Phase 3) starting in Year 11 of operation.
- A combined gravity circuit and whole ore leaching (WOL) will be used for recovering gold and silver.
- The average gold feed grade will be 1.57 g/t Au over the first five years.
- The initial capital cost estimate is $592 million, including a 15% contingency.
- Expansion capital is $426 million for the Phase 2 expansion to 12.0 Mtpa and $398

million for the Phase 3 expansion to 20.0 Mtpa.

- The LOM operating costs are estimated at $17.65/t of ore milled. Total LOM all-in sustaining cash costs are estimated at $811/oz.
- After-tax net present value at a 5% discount rate is estimated at $2,247 million.
- After-tax internal rate of return is 34.8%.
- After-tax initial capital payback is estimated at 2.0 years
- Expansion capital is paid for from operating cashflow.

1.3 Terms of Reference

The Report supports disclosures in in Artemis' press release entitled "Artemis Announces Revised PFS for Blackwater Project" dated 26 August 2020.

All currencies are expressed in Canadian dollars ($CDN) 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 & 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: life-of-mine (LOM) 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 (Artemis, 2020)

1.4.2 Mineral Tenure

BW Gold holds a 100% recorded interest in 328 mineral claims covering an area of 148,688 ha distributed among the Blackwater, Capoose, Auro, Key, Parlane and RJK claim blocks. The Blackwater claim block comprises 75 mineral cell claims totaling 30,578 ha. 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

BW Gold's 100% interest in the Blackwater claim block is subject to four 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 BW Gold. to purchase two-thirds of three Blackwater Claims (637203, 637205, and 637206) NSR royalty for $1,000,000 at any time, such that a 1% NSR royalty would remain.

PS Claim

A 2% NSR royalty is payable on mineral claim 835014. The existing agreement would allow BW Gold to purchase half for $1,000,000.

Only the royalties with respect the Dave Option and the Jarrit Option exist within the current Mineral Reserves.

BW Gold'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, payable to New Gold:

- A cash payment of $50 million to be paid on or before August 21, 2021 (Second Payment); and
- 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 first ranking security interest over the Project until the Second Payment is made, and will thereafter maintain a security interest over the Project in connection with the gold stream agreement (subject to any security to be granted over the Project in respect of future project financing.

All other material encumbrances within the Blackwater claim blocks are listed in Section 4.6.

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). BW Gold will undertake road improvements over a small section of the FSR. BW Gold 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. BW Gold will upgrade part of the FSR to meet the 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, the fibre-optic cable and the 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 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.

1.5.3 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 Project area at the summit of Mt. Davidson, which is the highest peak in the Fawnie Range. Bedrock outcrops are limited and most of the area is covered 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 of 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.

1.5.4 Regional Tectonics and Seismicity

The Project is situated within central British Columbia, where the level of recorded historical seismic activity has been low. A seismicity assessment was carried out for the Project in 2013, including a review of the regional seismicity and a probabilistic hazard analysis. A design earthquake magnitude 8.5 was selected for earthquake return periods of 500, 5,000, and 10,000 years, based on the review of regional tectonics and historical seismicity, and the findings of deaggregation of the probabilistic seismic hazard. This represents large magnitude earthquakes along the Queen Charlotte fault system and Cascadia subduction zone. The potential for shallow crustal earthquakes closer to the Project site was also considered for longer return period events of 5,000 and 10,000 years, representing earthquakes of up to about magnitude 7.5 along coastal British Columbia.

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 2014 (2014 FS). 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 orebody and east–northeast-trending faults along UTM easting 375,600E. This 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 has been 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 Project area were done 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 DC resistivity and 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 have been conducted on selected drill samples. A two-phase alteration study was also 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 1,041 core drillholes totalling 317,718 m have been 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 has been predominantly HQ diameter (63.5 mm) diamond drill core except where a reduction to NQ diameter (47.6 mm) was required to attain target depths. Drilling for metallurgical has used PQ diameter (85 mm) core. Some of the condemnation drilling was undertaken using reverse circulation (RC) methods.

Geological logging includes geotechnical, magnetic susceptibility, and specific gravity measurements taken at regular intervals. Lithology is logged and the core 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 1,041 holes, 1,025 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 are 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.

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 is 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) were drilled, and 91 RC holes and 18 core holes comprised the condemnation drill program.

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 are inserted into the sample stream. The drillhole database is supported by over 40,000 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 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 have been completed by Sue Bird, Principal of MMTS. This QP has 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 are acceptable. The protocols were reviewed and have been well documented. The drillhole database is 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, the Ag is generally lognormally distributed at higher grades, and the Ag mineralization has much lower value to the Project. 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 1-1 are reported inclusive of Mineral Reserves. Mineral Resources that are not Mineral Reserves do not have demonstrated economic viability.

Table 1-1 includes a range of 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” pit is 0.20 g/t AuEq, as highlighted in Table 1-1.

As part of the model validation process, a comparison of the Au content in the 2020 model (which used MIK for the Au estimate) to that in the 2014 resource model (which used OK) was done. The comparison has been done using 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 Au content is within 2% for the measured and indicated classes.

Table 1-1 Mineral Resource Estimate (effective date of May 5, 2020)

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 has been prepared by Sue Bird, P.Eng., an independent Qualified Person.*
2. *Resources are reported using the 2014 CIM Definition Standards and were estimated using 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 "reasonable prospects of eventual economic extraction" pit using the following assumptions: US $2,000/oz. Au and US $21.43/oz Ag at a currency exchange rate of 0.75 US$ per $CDN; 99.9% payable Au; 95.0% payable Ag; $8.50/oz Au and $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. Pit slope angles are assumed at 40º.

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 LG pit shell, including metal recoveries, and mining and process cost assumptions.

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

1.15 Mineral Reserve Estimates

Proven and Probable Mineral Reserves are modified from 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 2020 PFS mine plan.

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, 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.

Table 1-2 Mineral Reserve Estimate

Reserve Class	Tonnage (Mt)	Gold Grade (Au, g/t)	Contained Metal (Au, Moz.)	Silver Grade (Ag, g/t)	Contained Metal (Ag, Moz.)	AuEq Grade (g/t)
Proven	325.0	0.74	7.8	5.8	60.5	0.78
Probable	9.1	0.80	0.2	5.5	1.6	0.84
Total Reserve	**334.0**	**0.75**	**8.0**	**5.8**	**62.1**	**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 August 18, 2020.*
2. *Mineral Reserves are based on the 2020 Pre-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.*
4. *Mineral Reserves are reported at an NSR cut-off of $13.00/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; $8.50/o. Au and $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 NSR cut off- covers processing costs of $10.00/t and administrative (G&A) costs of $3.00/t.*
7. *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).*
8. *Numbers have been rounded as required by reporting guidelines.*

1.16 Metallurgy and Processing

The process flowsheet was chosen based on historical test work and designed using the results of more recent test work carried out in 2019 for the previous owner of the Project, New Gold. The most recent metallurgical program, completed in 2019, was carried out with the primary objective of confirming and optimizing the flowsheet and design criteria using a combination of new test work, results from the historical and previous test work programs, and trade-off studies completed since the 2014 FS. Drill core from site was sent to Base Metallurgical Laboratories Ltd. (BaseMet) in Kamloops, BC for test work that included core splitting, sample preparation, interval assaying, mineralogy, gravity concentration, cyanide leach and cyanide destruction.

The test program included three larger composites for optimization test work 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 test work 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. A correlation between gold extraction and head grade was not observed. The results obtained from three composites representing the first 10 years of mining averaged 93.7% total gold extraction with gravity gold recovery of 34.2%

Based on the test results, a gold doré can be produced with a primary grind size of 80% passing (P_{80}) 150 µm followed by gravity concentration, two hour pre-oxidation, a 48 hour cyanide leach at an initial cyanide concentration of 500 ppm and a pH of 10.5, carbon-in-pulp (CIP) adsorption, desorption and refining process. The weighted average of the year composites, based on the mine plan, is estimated to achieve an overall average gold recovery in the range of 93% to 94%.

The initial design daily throughput is 15,000 t/d, with an availability of 75% used in designing the crushing circuit and 93% for the design of the rest of the plant.

The process will consist of:

- Three stage crushing, consisting of a primary jaw crusher with grizzly feeder, a secondary cone crusher and two tertiary cone crushers. The primary jaw crusher, the three cone crushers and the three vibrating screens will each be housed in steel-framed buildings, with covered conveyors transporting material between each stage. The crushed ore stockpile will be covered to prevent freezing;
- 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 circuit will consist of eight tanks fitted with mechanical agitators, an initial pre-oxidation tank with cyanide being added to the second and subsequent tanks. The leach residence time will be 48 hours;
- Carbon in pulp adsorption of gold and silver will be carried out in a “carousel” unit, with “pump cells” moving leached slurry between the six tank units while the carbon remains in the same tank until fully loaded;
- The loaded carbon will be treated in a Zadra elution and electrowinning circuit consisting of an acid wash column and two elution columns operating at 140 degrees Celsius. A propane heater will provide the necessary temperature and two additional heat exchangers will control the temperature around the circuit. A rotary kiln operating at 700 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 air system will be carried out in the final tailings slurry, with the sulphur dioxide being produced by the combustion of elemental sulphur.

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 18 months prior to mill start-up and are anticipated to run for 18 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 23 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 eight phases, with initial phases containing higher gold grade, lower strip ratio, and mineralization. The first phase will target suitable waste rock for construction whilst exposing near-surface, high-grade material. The second 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 5.5 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 \$13/t and \$16.50/t NSR will be stockpiled. Cut-off grade optimization on the mine production schedule also sends ore above

$16.50/t NSR to a high-grade ore stockpile in certain planned periods. The stockpiled Mineral Reserves are planned to be re-handled back to the crusher once the pits are 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 550 t class hydraulic shovels and 220 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 lease arrangements.

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

Ore will be hauled to a crusher 1 km north of the open pit limit, which feeds the process plant; and waste rock is generally used as fill for construction of the tailings storage facility (TSF) located 2.5 to 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, within 2 km west of the pit, will be used to store excess overburden and non-acid generating (NAG) waste rock. Ore stockpiles, within 1 km west of the open pit, are 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 $1.89–$2.99/t with a LOM average of $2.37/t mined. Mine operations will include ore control and 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 2020 PFS mine plan.

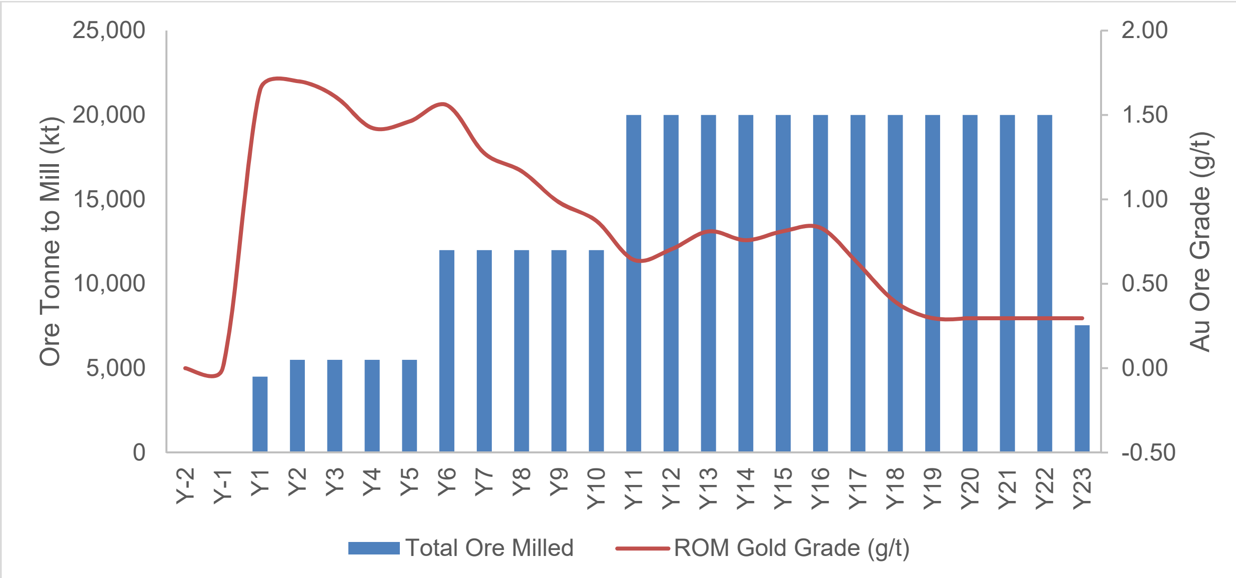

Figure 1-2 Mill Feed Tonnes and Grade (source: Moose Mountain, 2020)

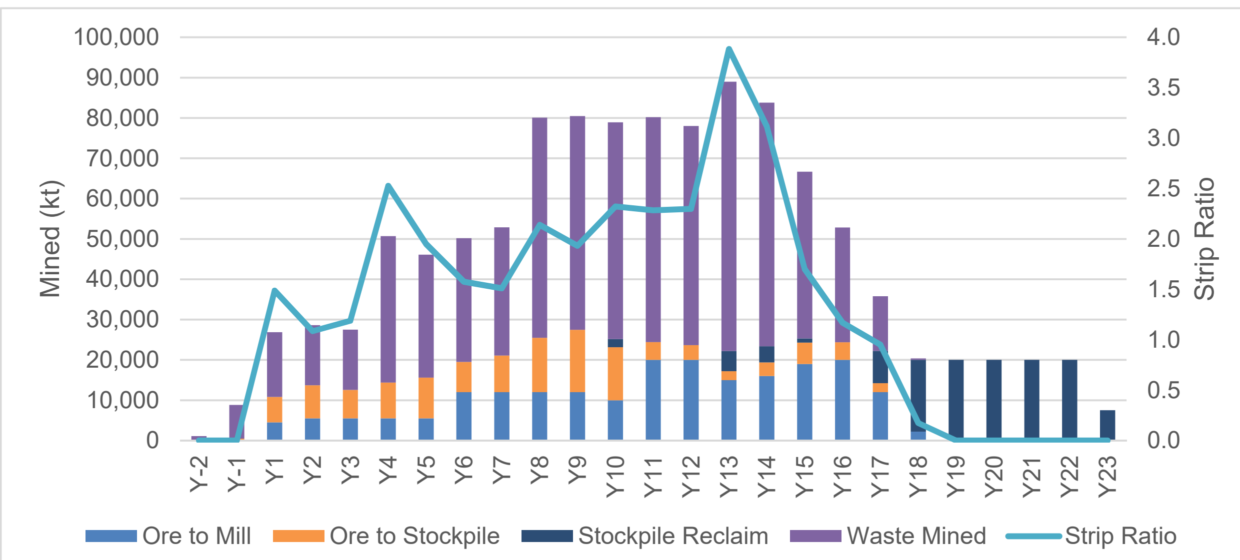

Figure 1-3 Material Mined and Strip Ratio (source: Moose Mountain 2020)

1.18 Onsite Infrastructure

A 250 bed camp already exists on site and a further 240 bed camp will be added. This will accommodate the construction personnel and the new camp will be used for operating personnel. Additions will be made as required for development and operation of Stages 2 and 3.

Onsite infrastructure to support mining and milling activities will include a primary crusher, reclaim conveyors, mill building, elution and refinery building, whole ore leach tanks, main truck shop, administration and emergency services buildings, explosives storage facility and fuel farm.

Power will be supplied to the Blackwater site by connection to the BC Hydro grid and construction of a 135 km long 230 kV transmission line from the BC Hydro Glenannan Substation to the Blackwater site.

- The incoming transmission line will terminate at the site main substation adjacent to the main process facilities. The power line has been designed to provide for a maximum load of 110 MW, sufficient for the fully expanded project.

Emergency power will be available from a standby power station that will consist of a minimum of two modular gensets rated at a nominal 3.0 MW.

A fibre optic cable will be installed along with the main transmission line to provide high bandwidth telecommunications access to the site.

Onsite power will be distributed from the mine site substation and will service all mine loads with the exception of mine pit equipment until Year 6 when electric drills and shovels will be deployed. Power to equipment, buildings, etc. will be supplied from the 25 kV line through pole mounted 25 kV to 600 V transformers or pad mounted 25 kV to 4.16 kV transformers.

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.

1.19 Waste Characterization

Mine waste was classified based on its predicted acid generation potential into PAG or NAG as shown by the calculated neutralization potential ratio (NPR). NAG waste rock will be further classified as to its metal leaching (ML) potential based on zinc content. Classification criteria will be as follows:

- Overburden (NAG)
- 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)
- Ore and tailings (PAG)
- Low-grade ore (PAG)

PAG1 and PAG2 waste rock will be stored in the TSF or selectively used in construction of the TSF dams (e.g. in the upstream zone) and submerged in the TSF within one year of mining to prevent the formation of acid rock drainage (ARD). NAG3 waste rock will be used in construction of the TSF dams or otherwise stored in the TSF and submerged in the TSF within three to five years of mining to reduce metal leaching. NAG4 and NAG5 waste rock will be used for construction on site and in the downstream shells of the TSF dams. Overburden is classified as NAG and will be used for construction or stockpiled in waste dumps adjacent to the open pit.

Sulphide and transition ore tailings are classified as PAG and previously deposited tailings will be covered routinely as tailings accrete in the TSF or otherwise kept saturated or submerged within the TSF during operations to prevent ARD. Oxide tailings exhibit lower ARD and ML potential.

1.20 Tailings Storage Facility

The TSF was designed to permanently store all tailings solids, PAG1 and PAG2 waste rock, and NAG3 waste rock generated during the operation of the mine. The TSF will comprise two adjacent sites, TSF Site C and TSF Site D. TSF Site C will be constructed first to provide storage capacity for start-up of the process plant. The Site C facility was designed to contain up to approximately 16 years of tailings and the first six years of PAG/NAG3 waste rock, and includes a storage allowance for the supernatant pond to provide a continuous source of process water to the mill operations. TSF Site D will be constructed adjacent to and downstream of TSF Site C beginning in Year 5 to provide additional storage capacity for PAG/NAG3 waste rock and tailings. The facility was designed to contain PAG/NAG3 waste rock generated during mining between Years 7 and 18 and up to approximately six years of tailings beginning in Year 17 when TSF Site C reaches design capacity.

The TSF was designed to contain 462 Mm^3 of tailings and waste rock material and will require approximately 83 Mm^3 of construction material with approximately 95% being supplied by waste rock and overburden from development of the open pit. A total of three embankments will be constructed across the two sites to form the ultimate facility. The TSF embankments will be engineered, water-retaining, zoned earthfill/rockfill dams with a compacted low-permeability core zone and appropriate filter/transition zones. The TSF Site C and Site D dams will be expanded using centreline construction methods. 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. Several borrow sources should be available in the vicinity of the TSF basin, if needed, including pit-run granular fill materials for the dam shell, fine-grained glacial till for the core zone, and aggregate materials that could be crushed and/or screened to produce desirable quantities and grain size distributions for engineered fill materials.

1.21 Water Management

All drainage from the mine will flow by gravity into the TSF to simplify water management, spill control, and mine closure. The following strategies are used in the tailings and mine water management plan:

- Manage sediment mobilization and erosion by installing sediment controls prior to land disturbance, limiting land disturbance to the minimum practicable extent. Install appropriate temporary erosion and sediment control measures or Best Management Practices (BMPs) prior to, and during, initiation of land disturbance

- Use the water within the proposed Project area to the maximum practicable extent by collecting and managing site runoff from disturbed areas, maximizing the recycle of process water, and storing water within the TSF

- Reduce potential flow reductions in lower Davidson Creek using staged TSF designs with the revised development approach and mine plan

- Inclusion of staged engineered diversions (Southern, Central, and Northern Diversions) to allow diversion of upstream flows from significant undisturbed catchment areas around the TSF to Davidson Creek. Flow diversions will be operated in a manner that allows for diversion, when not required to support ore processing, or collection of water from these areas in order to manage the mine site water balance within the target operating range and maintain flows in Davidson Creek at or above the defined instream flow needs.

- Planned installation of a water treatment plant at the start of operations to enhance water management flexibility and allow for treatment of mine site contact water to meet discharge criteria, if required

- Pump water as required from Tatelkuz Lake through a water supply pipeline to a water reservoir downstream of the TSF to provide fresh water to supplement flows in lower Davidson Creek to meet instream flow needs for fish

- Collect all recoverable TSF seepage downstream of the main dam during operations and post closure until the pit lake overflows or the water is acceptable for direct discharge to Davidson Creek

- Monitor surface water and groundwater quality, maintain fish habitat, develop compensatory fish habitat, and reclaim disturbed areas.

During operations, drainage from the low-grade and coarse ore stockpiles may become acidic with elevated metals content; the drainage will be collected and neutralized with lime to increase the pH and precipitate metals before disposal in the TSF. Pit water is predicted to be of neutral pH with relatively low metals content during operations; it will be pumped to a small holding/monitoring pond, which will overflow to the TSF or be treated/released to the downstream receiving environment depending on needs of the mine.

1.22 Closure Plan

The primary objective of the closure and reclamation initiatives will be to eventually return the Project site to a self-sustaining facility that satisfies end land use objectives. The facilities will be reclaimed according to accepted practices at the time of closure and in a manner that maintains long-term geochemical and physical stability. All buildings not needed beyond closure will be removed, disturbed lands rehabilitated, and the property will be returned to otherwise functional use according to approved reclamation plans. Site infrastructure required for water management following closure will be maintained and operated according to approved closure water management plans developed in consultation with First Nations, government, and other stakeholders to the Project.

1.23 Environmental Studies, Assessment, and Social or Community Impact

The Project has successfully completed the provincial and federal environmental assessment processes and has been awarded an Environmental Assessment Certificate (provincial) and positive Decision Statement (federal). Extensive environmental baseline and social studies were conducted in support of the provincial and federal environmental assessment processes. Many of the key environment baseline studies continued throughout and subsequent to the environmental assessment process, which provides a robust environmental baseline dataset supporting Project design and development.

As a result of the commitments made during the environmental assessment and in support of provincial and federal permitting, several monitoring and management plans will be developed to ensure appropriate mitigation of potential project effects during construction, operation, closure, and post-closure phases.

The proposed Blackwater mine site is primarily located within the asserted traditional territories of the Lhoosk'uz Dené Nation and Ulkatcho First Nation. New Gold entered into a trilateral Participation Agreement with these two Indigenous nations on April 18, 2019, who, following completion of the environmental assessment process, confirmed their support for the Project and consented to issuance of the BC EA Certificate, and any other permits or authorizations to be issued by or on behalf of the Environmental Assessments Office pertaining to the Project. On closing of the acquisition of the Blackwater Project, the Participation Agreement was assigned to BW Gold.

The Project could also interact with the Carrier Sekani First Nations and their Aboriginal title, rights, and interests as a result of the transmission line, use of road to access the site, and potential downstream water quality and other effects. During the environmental assessment, New Gold committed to continuing negotiations with the Carrier Sekani First Nations with the goal of reaching a mutually acceptable participation agreement that will include accommodative measures and other benefits. Artemis remains committed to this goal.

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.

1.24 Permitting

The primary provincial permits required for the Project to proceed to the construction and operations phases are issued under the Mines Act (Mine and Reclamation Permit) and the Environmental Management Act (Effluent Discharge Permit and Air Discharge Permit). Several permits of a more routine nature will also be required, including a lease or license of occupation under the Land Act, a Mining Lease under the Mineral Tenure Act, a Water License under the Water Sustainability Act, and several lesser permits.

Federal authorizations are required under the Fisheries Act and Explosives Act for the Project to proceed to construction and operations. Notably, a Schedule 2 Listing will be required for the TSF under the Metal and Diamond Mining Effluent Regulations of the Fisheries Act. This is a process administered by Environment Canada and adjudicated by an Order in Council. A detailed fish and fish habitat offsetting plan is required to support the Schedule 2 amendment application.

1.25 Capital Cost Estimate

The capital cost estimate for the Blackwater Project has been developed to provide an estimate suitable for the 2020 PFS. The cost estimate is based on a combination of material take-off (MTO) data, design drawings, vendor quotes, manufacturers' information, and industry standards and rates. All costs are expressed in Q3 2020 Canadian dollars with a +25%/-10% accuracy.

The initial capital cost is estimated to be $592 million for Phase 1 (5.5 Mtpa), with expansion capital of $426 million for the Phase 2 expansion to 12.0 Mtpa, and expansion capital of $398 million for the Phase 3 expansion to 20.0 Mtpa. Sustaining capital over the life of mine is estimated at $637 million. Closure costs are estimated at $117 million, partially offset by proceeds from equipment salvage values, estimated at $42 million. Capital costs summarized in Table 1-3 include a 15% contingency.

Table 1-3 Capital Cost Summary

	Phase 1 Initial Capital	Phase 2 Expansion Capital	Phase 3 Expansion Capital	Sustaining Capital	Total Capital
	$M	$M	$M	$M	$M
Directs					
Mining	68	89	68	337	562
Process Plant	109	130	143	-	382
Onsite Infrastructures	68	38	19	2	127
Offsite Infrastructure	81	9	12	6	108
Tailings and Water Management	37	29	33	190	290
Total Directs	364	294	274	536	1,469
Indirects and EPC	120	73	69	18	279
Owners Costs	31	3	3	-	37
Contingency	77	56	52	83	268
Total	**592**	**426**	**398**	**637**	**2,052**

1.26 Operating Cost Estimate

For operating cost estimation purposes, the Project has been divided in 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 +25%/-10%.

Average operating costs for the various phases of operation are summarized in Table 1-4.

Table 1-4 Operating Cost Summary

	Units	Pre-strip	Phase 1	Phase 2	Phase 3	LOM
Mining*	$/t mined	3.31	2.15	2.14	2.62	2.37
	$/t milled	-	14.61	12.12	4.98	7.03
Process	$/t milled	-	9.17	8.31	8.24	8.33
G&A	$/t milled	-	4.64	2.87	1.91	2.30
Total	$/t milled	-	28.42	23.30	15.13	17.65

**Mining costs includes stockpile re-handle, LOM mining costs exclude pre-stripping*

The LOM operating cost estimates for Blackwater peak in Phase 1 at $28.42/t, with economies of scale and driving down costs to $23.30/t in Phase 2 and $15.13/t in Phase 3. Over the LOM, the Project has estimated average operating costs of $17.65/t.

1.27 Economic Analysis

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.

1.27.1 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.

1.27.2 Cashflow Basis

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

- Gold – US$1,541/oz;
- Silver – US$19.60/oz;

- Exchange rate – 0.76 (US$/$CDN).

The economic analysis is presented as a Base Case, which assumes no leverage, and a Leverage 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.

1.27.3 Base-Case

For the 23-year mine life and 334 Mt mill feed, the following after-tax Base Case financial parameters were calculated as follows:

- $2,247 million NPV at 5.0% discount rate;
- 34.8% IRR;
- 2.0 year initial capital payback.

1.27.4 Leveraged-Case

For a leveraged case assuming initial capital is 60% debt financed 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:

- $2,249 million NPV at 5.0% discount rate;
- 49.7% IRR;
- 2.2 year initial capital payback.

1.27.5 Sensitivity Analysis

Sensitivity analysis was performed on the Project base case using metal price (grade), exchange rate, operating costs and initial capital costs. 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.

1.28 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;
- 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;
- Use of oxygen rather than compressed air for cyanide leaching and cyanide detoxification;
- Value engineering initiatives.

1.29 Recommendations

It is recommended that a feasibility study be completed for the Blackwater Project in two phases. The first phase of work will collect information that will be used to refine the feasibility study outcomes in Phase 2.

Phase 1 field work to support a feasibility study would include supplemental geotechnical and hydrogeological site investigation, a grade control drill program, and metallurgical test work. The estimated cost for this work is $8,320,000. The Phase 2 Feasibility Study will require an estimated $2,100,000 to complete.

2 INTRODUCTION

2.1 Introduction

Moose Mountain Technical Services (MMTS), Knight Piésold Ltd. (KP), and JAT Met Consult Ltd. (JAT Metco) have prepared a technical report (the Report) for Artemis Gold Inc (Artemis) on a pre-feasibility study (PFS) evaluation (2020 PFS) of the Blackwater Gold 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 Revised PFS for Blackwater Project" dated 26 August 2020.

All measurement units used in this Report are metric, and currency is expressed in Canadian dollars 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 using the 2019 edition of the Canadian Institute of Mining, Metallurgy and Exploration (CIM) Estimation of Mineral Resources & 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: life-of-mine (LOM) refers to the life of mine including the pre-production period; the operational period refers to the mine life excluding the pre-production duration.

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.

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:

- Sue Bird, P.Eng., Moose Mountain Technical Services;
- Daniel Fontaine, P.Eng., Knight Piésold Ltd., Vancouver;
- Tracey Meintjes, P.Eng., Moose Mountain Technical Services;
- Marc Schulte, P.Eng., Moose Mountain Technical Services;

- John 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 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.

2.4.2 Daniel Fontaine Site Visit

Daniel Fontaine visited the site most recently on July 23–24, 2019 as well as on several other occasions previously between 2011 and 2016. During his visits, Daniel viewed the proposed locations of the processing plant, open pit, waste rock storage facilities (WRSFs), haul roads, tailings storage facility (TSF) and associated dam sites, and select locations associated with water management components of the proposed Project.

2.4.3 Tracey Meintjes Site Visit

Tracey Meintjes visited the site on 07 May 2020. During his visit, Tracey viewed the proposed locations of the processing plant, pits, WRSFs, haul roads, and TSF, reviewed selected drill core samples, and reviewed the on-site and off-site infrastructure options.

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:

- Date of the Mineral Resource estimate: 5 May 2020;
- Date of the Mineral Reserve estimate: 18 August 2020;

- Date of the Technical Report: 26 August 2020.

The overall Report effective date is taken to be the date of the economic analysis and is 26 August 2020.

2.6 Information Sources

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

2.7 Previous Technical Reports

Artemis has not previously filed a technical report on the Project.

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.

3 RELIANCE ON OTHER EXPERTS

The QP authors of this Report state that they are qualified persons for those areas as 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 environmental status in sections of this Report as noted below.

3.1 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 have fully relied upon, and disclaim responsibility for, information supplied by Artemis experts and experts retained by Artemis for this information through the following documents:

- Blake, Cassels & Graydon, LLP, 2020: opinion letter addressed to Artemis from Mandev Mann dated 25 August 2020.

This information is used in Section 4.2, Section 4.3, Section 4.4 and Section 4.6 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.

3.2 Surface Rights

The QPs have fully relied upon, and disclaim responsibility for, information supplied by experts retained by Artemis for information relating to the status of the current Surface Rights as follows:

- Blake, Cassels & Graydon, LLP, 2020: opinion letter addressed to Artemis from Mandev Mann, dated 25 August 2020.

This information is used in Section 4.5 of the Report and in support of the Mineral Resource estimate in Section 14, the Mineral Reserve estimate in Section 15 and the economic analysis in Section 22.

4 PROPERTY DESCRIPTION AND LOCATION

4.1 Introduction

For the purpose of this report the property (the Property) comprises six contiguous claim blocks held by BW Gold Ltd. (Blackwater, Capoose, Auro, Key, Parlane and RJK) as shown in Figure 4-1.

The Blackwater Project (the Project) refers to exploration and development activity related to the Blackwater deposit which is contained within the Blackwater claim block. The Project lies in central British Columbia, approximately 112 km southwest of Vanderhoof and 446 km northeast of Vancouver. The Project is within NTS map sheet 93F/02 and is centred at 5893000 N and 375400 E (UTM NAD83).

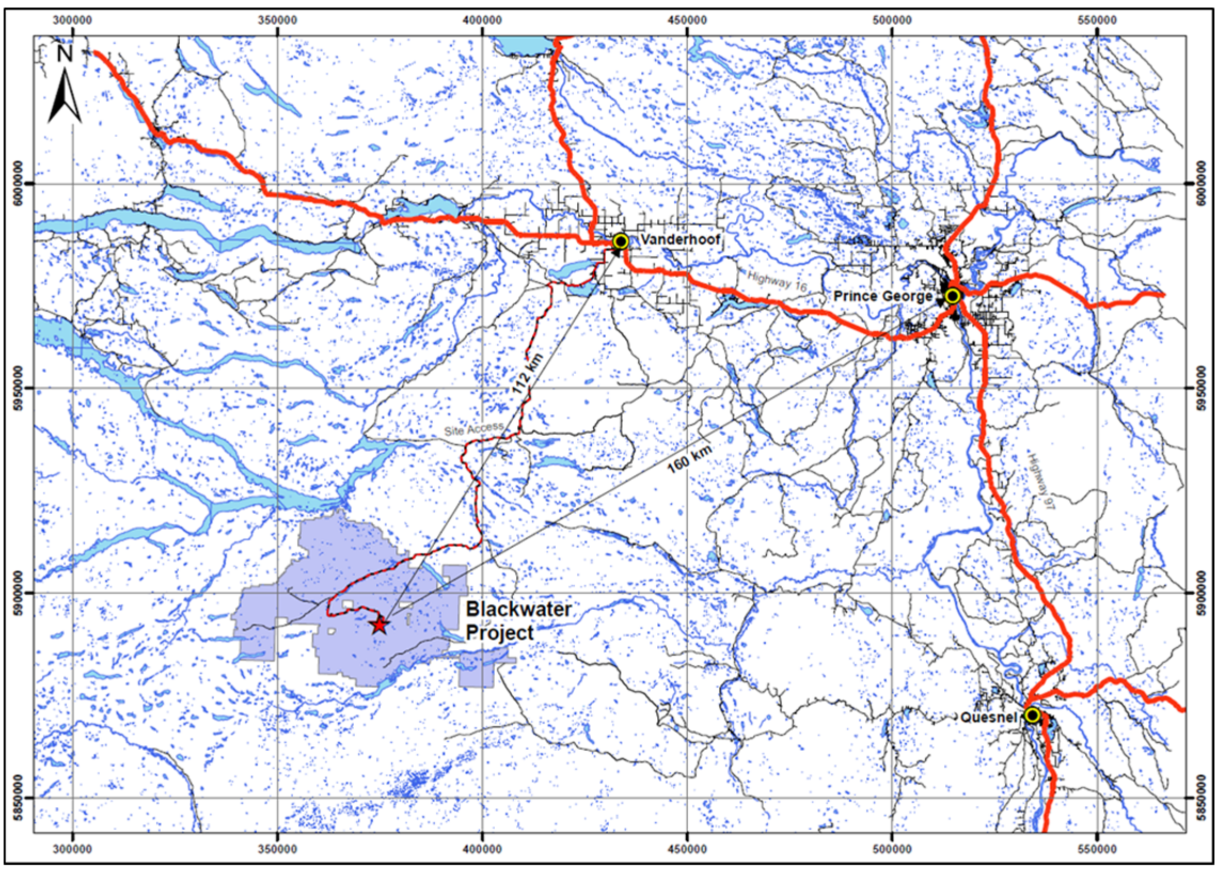

Figure 4-1 Location Plan Blackwater Project (source: MMTS, 2020)

4.2 Project Ownership History

The Project 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, BW Gold, a wholly-owned subsidiary of 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.

4.3 Mineral Tenure

Artemis, through BW Gold, holds a 100% recorded interest in 328 mineral claims covering an area of 148,688 ha (Figure 4-2).

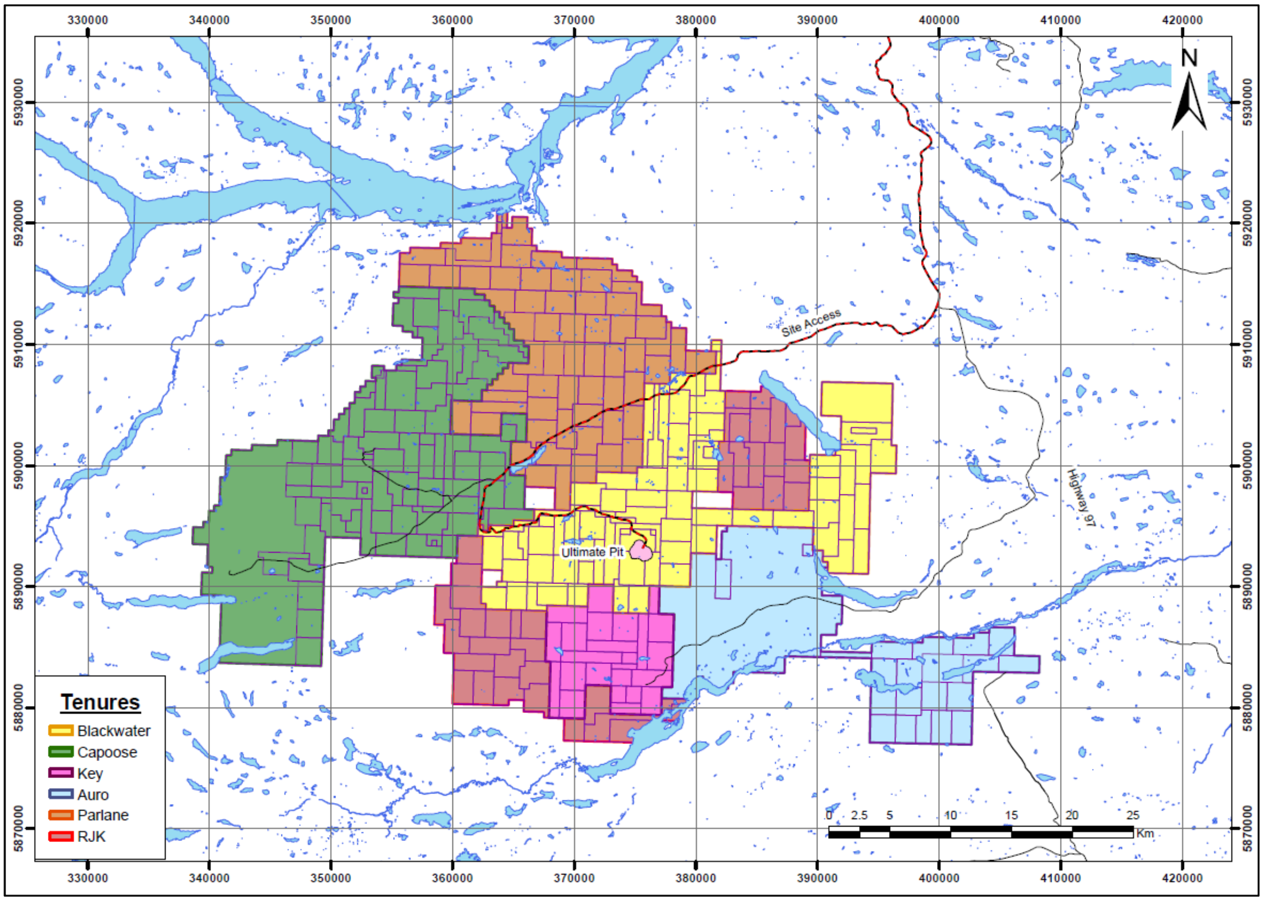

Figure 4-2 Mineral Claim Blocks of the Property (source: MMTS, 2020)

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

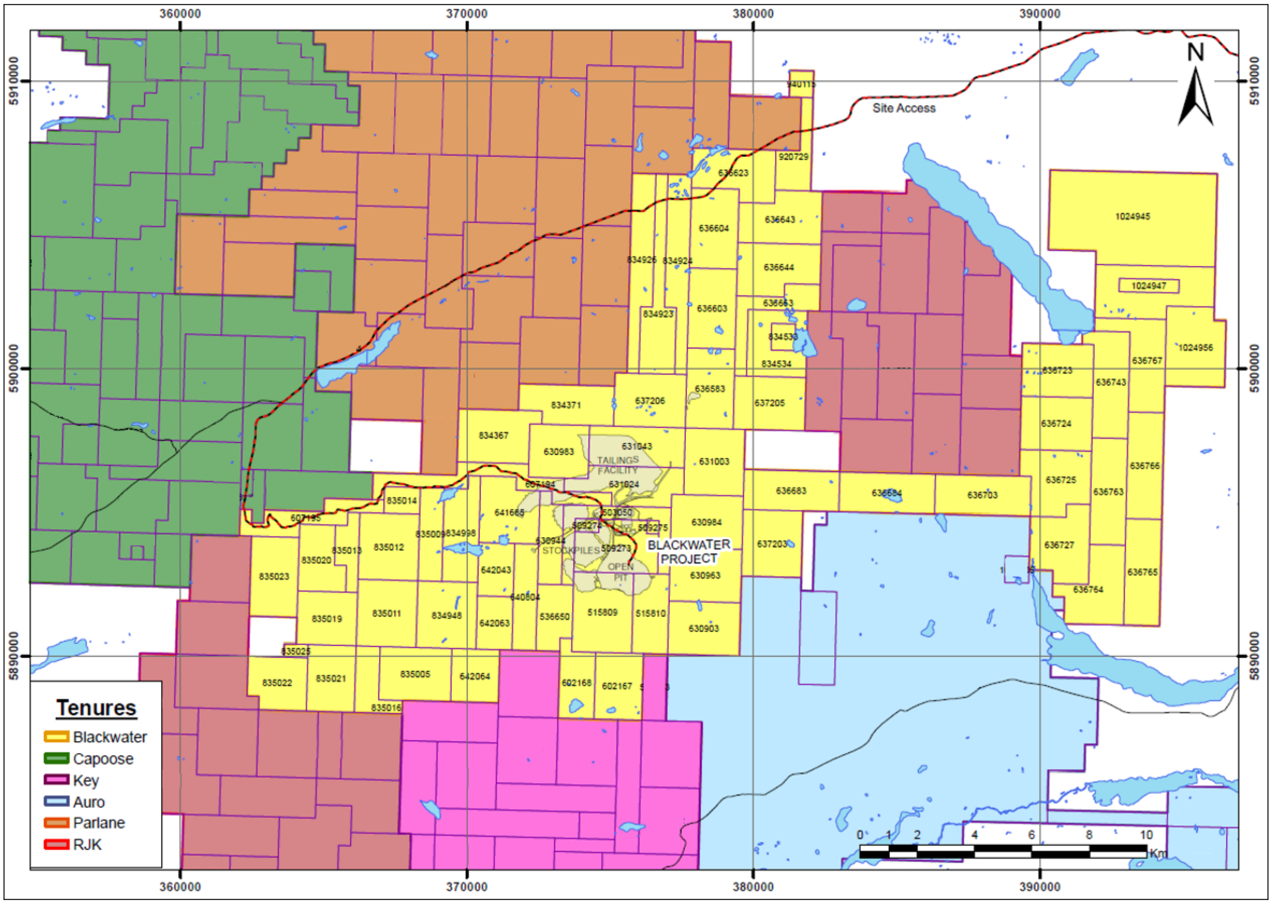

Figure 4-3 Mineral Claim Blocks of the Project (source: MMTS, 2020)

4.3.1 Blackwater Claim Block

The Blackwater claim block comprises 75 mineral cell claims totaling 30,578 ha (Table 4-1). All Blackwater claims are 100% held in the name of BW Gold. BW Gold 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 BW Gold. Capoose mineral claims are summarized in Table 4-1.

All claims in the Capoose claim block, excepting claim 238045, are cell claims. 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 BW Gold. 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 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 BW Gold. 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 mineral 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 BW Gold. 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 BW Gold. RJK mineral claims are summarized in Table 4-1.

No RJK claims are known to overlap any legacy or Crown Granted mineral claims. No RJK claims overlap any mineral 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/2	387.9	Blackwater

Title Number	Claim Name	Title Type	Title Sub Type	Map Number	Issue Date	Good To Date	Area (ha)	Block
						9		
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
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

Title Number	Claim Name	Title Type	Title Sub Type	Map Number	Issue Date	Good To Date	Area (ha)	Block
636766	RIGHT STUFF 12	Mineral	Claim	093F	2009/SEP/18	2022/AUG/29	464.8	Blackwater
636767	RIGHT STUFF 13	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	RICHFIELDADJACENTCC	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 WEST2	Mineral	Claim	093F	2010/OCT/04	2022/AUG/2	368.6	Blackwater

Title Number	Claim Name	Title Type	Title Sub Type	Map Number	Issue Date	Good To Date	Area (ha)	Block
						9		
835023		Mineral	Claim	093F	2010/OCT/04	2022/AUG/29	465.2	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
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

Title Number	Claim Name	Title Type	Title Sub Type	Map Number	Issue Date	Good To Date	Area (ha)	Block
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/2	310.0	Capoose

Title Number	Claim Name	Title Type	Title Sub Type	Map Number	Issue Date	Good To Date	Area (ha)	Block
						9		
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
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

Title Number	Claim Name	Title Type	Title Sub Type	Map Number	Issue Date	Good To Date	Area (ha)	Block
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/2	466.0	Key

Title Number	Claim Name	Title Type	Title Sub Type	Map Number	Issue Date	Good To Date	Area (ha)	Block
						9		
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	2008/JUL/30	2022/AUG/29	485.6	Key
589177	LOCK 2	Mineral	Claim	093F	2008/JUL/30	2022/AUG/29	485.8	Key
589183	LOCK 3	Mineral	Claim	093F	2008/JUL/30	2022/AUG/29	484.9	Key
589231	LOCK 4	Mineral	Claim	093F	2008/JUL/30	2022/AUG/29	485.2	Key
589232	LOCK 5	Mineral	Claim	093F	2008/JUL/30	2022/AUG/29	485.4	Key
589234	LOCK 6	Mineral	Claim	093F	2008/JUL/30	2022/AUG/29	388.4	Key
589236	LOCK 7	Mineral	Claim	093F	2008/JUL/30	2022/AUG/29	466.2	Key
589238	LOCK 8	Mineral	Claim	093F	2008/JUL/30	2022/AUG/29	233.1	Key
589241	LOCK 9	Mineral	Claim	093F	2008/JUL/30	2022/AUG/29	407.6	Key
589242	LOCK 10	Mineral	Claim	093F	2008/JUL/30	2022/AUG/29	465.7	Key
589243	LOCK 11	Mineral	Claim	093F	2008/JUL/30	2022/AUG/29	194.0	Key
589244	LOCK 12	Mineral	Claim	093F	2008/JUL/30	2022/AUG/29	388.6	Key
642003	YELLOW & BLACK	Mineral	Claim	093F	2009/SEP/27	2022/AUG/29	174.9	Key
642004	BLACK & YELLOW	Mineral	Claim	093F	2009/SEP/27	2022/AUG/29	174.9	Key
642023	BLACK	Mineral	Claim	093F	2009/SEP/27	2022/AUG/29	388.6	Key
642024	YELLOW	Mineral	Claim	093F	2009/SEP/27	2022/AUG/29	233.2	Key
694043		Mineral	Claim	093F	2010/JAN/04	2022/AUG/29	464.7	Parlane
694044		Mineral	Claim	093F	2010/JAN/04	2022/AUG/29	483.9	Parlane
694045		Mineral	Claim	093F	2010/JAN/04	2022/AUG/29	483.8	Parlane
694046		Mineral	Claim	093F	2010/JAN/04	2022/AUG/29	464.6	Parlane
694063		Mineral	Claim	093F	2010/JAN/04	2022/AUG/29	445.0	Parlane
694064		Mineral	Claim	093F	2010/JAN/04	2022/AUG/29	483.8	Parlane
694065		Mineral	Claim	093F	2010/JAN/04	2022/AUG/29	483.7	Parlane
694066		Mineral	Claim	093F	2010/JAN/04	2022/AUG/29	464.1	Parlane
694083		Mineral	Claim	093F	2010/JAN/04	2022/AUG/29	483.6	Parlane

Title Number	Claim Name	Title Type	Title Sub Type	Map Number	Issue Date	Good To Date	Area (ha)	Block
694084		Mineral	Claim	093F	2010/JAN/04	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	483.0	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/2	463.6	Parlane

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						9		
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
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/04	464.7	RJK
694188		Mineral	Claim	093F	2010/JAN/04	2023/FEB/04	483.8	RJK
694203		Mineral	Claim	093F	2010/JAN/04	2023/FEB/04	483.8	RJK
694204		Mineral	Claim	093F	2010/JAN/04	2023/FEB/04	484.1	RJK
694205		Mineral	Claim	093F	2010/JAN/04	2023/FEB/04	483.8	RJK
694206		Mineral	Claim	093F	2010/JAN/04	2023/FEB/04	464.7	RJK

Title Number	Claim Name	Title Type	Title Sub Type	Map Number	Issue Date	Good To Date	Area (ha)	Block
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/09	2023/FEB/04	290.0	RJK
1015566	RJ1	Mineral	Claim	093F	2012/DEC/31	2022/AUG/29	174.9	RJK
1015573	RJK4	Mineral	Claim	093F	2012/DEC/31	2022/AUG/2	311.0	RJK

Title Number	Claim Name	Title Type	Title Sub Type	Map Number	Issue Date	Good To Date	Area (ha)	Block
						9		
1015575	RJK5	Mineral	Claim	093F	2012/DEC/31	2022/AUG/29	563.5	RJK
1015577	RJK6	Mineral	Claim	093F	2012/DEC/31	2022/AUG/29	602.6	RJK
1015578	RJK8	Mineral	Claim	093F	2012/DEC/31	2022/AUG/29	466.6	RJK
1015579	RJK9	Mineral	Claim	093F	2012/DEC/31	2022/AUG/29	155.5	RJK

4.4 Surface Rights

BW Gold, 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.

If BW Gold 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 BW Gold 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 BW Gold could apply for certification by the Minister of Mines and Energy that the surface rights over that area are required by BW Gold for the purposes of a mining activity. If that certification is made, then BW Gold 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 BW Gold 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 utilized searches of the MTO system, the Integrated Land and Resources Registry (ILRR) system, 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; and third party mineral tenures overlapping the access road, and the 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 Property or the Linear Infrastructure.

4.5 Artemis/New Gold Purchase Agreement

The purchase agreement includes the following considerations:

- A cash payment of $50 million to be paid on or before August 21, 2021 (still to be paid); and
- 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 first ranking security interest over the Project until the Second Payment is made, and will thereafter maintain a security interest over the Project in connection with the gold stream agreement (subject to any security to be granted over the Project in respect of future project financing.

4.6 Royalties and Encumbrances

BW Gold’ 100% interest in the Project 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.

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 BW Gold to purchase two-thirds of three Blackwater Claims (637203, 637205, and 637206) NSR royalty for $1,000,000 at any time, such that a 1% NSR royalty would remain. This royalty does not affect the deposit or economics of the PFS.

PS Claim

A 2% NSR royalty is payable on mineral claim 835014. The existing agreement would allow Artemis to purchase half for $1,000,000. This royalty does not affect the deposit or economics of the PFS.

4.6.2 Capoose Claims Block Agreements and Encumbrances

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 BW Gold may purchase half for $1,000,000. BW Gold has an obligation to pay an advance royalty to the optionor of $30,000 per annum, to be credited against the NSR royalty.

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. BW Gold may purchase two-thirds of the Buck NSR royalty for $2,000,000, such that a 0.5% NSR royalty would remain.

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. BW Gold may purchase two-thirds of the NSR royalty for $2,000,000 such that a 0.5% NSR royalty would remain.

Capoose Option and Joint Venture

New Gold, through Silver Quest, acquired a 100% interest in claim no. 512838, 60% of which was obtained from Bearclaw Capital Corp. 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 BW Gold to purchase four-ninths of the NSR royalty for $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

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. BW Gold may purchase half of the Key NSR royalty for $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 $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 $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 $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 which may be bought down by a payment of $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 1st day of December 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 $1,000,000 resulting in a 1.0% NSR royalty remaining.

New Gold acquired through an Option Agreement dated 3rd day of December 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 $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 $2,000,000 resulting in a 1.5% NSR royalty remaining.

4.7 Environment, Environmental Liabilities and Social License

The Project is currently subject to liabilities limited to exploration activities conducted since 1973 by New Gold, its predecessors, and previous explorers. A reclamation security sufficient to reclaim the site has been posted in accordance with British Columbia *Mines Act*.

Environmental and social base line studies and potential environmental and social effects of development and operation of the Blackwater Project are discussed in Section 20 of this Report.

4.8 Permits

The BC Minister of Energy, Mines and Petroleum Resources and Minister of Environment and Climate Change Strategy issued Environmental Assessment (EA) Certificate # M19-01, which included the Certified Project Description and a Table of Conditions, on June 21, 2019. The proponent is required to have substantially started the Project within five years of receiving the Certificate, in accordance with Section 18(1) of the BCEAA 2002.

The federal Minister of the Environment concluded that the Project is not likely to cause significant adverse environmental effects referred to in both subsection 5(1) and subsection 5(2) of CEAA 2012, and established conditions related to the environmental effects with which the proponent must comply. The Decision Statement was issued on April 15, 2019 and does not contain an expiry date.

A number of additional permits and authorizations are required from both the provincial and federal governments prior to the Project proceeding to the construction and operation phase, primarily provincial permits under the *Mines Act* and the *Environmental Management Act* for discharge, and federal authorizations under the *Fisheries Act* and *Explosives Act*. Notably, a Schedule 2 Listing will be required for the tailings facility under the Metal and Diamond Mining Effluent Regulations of the *Fisheries Act*. Project permitting is addressed in Section 20 of this Report.

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 BW Gold 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 in December 2013 and September 2020 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.
- BW Gold 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 known significant factors and risks 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 Forest Service Road (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

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 mean annual precipitation for the site is estimated to be 564 mm with 62% falling as rain and 38% as snow (KP, 2018).

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, which is the highest peak in the Fawnie Range. 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 of 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 Forest Service Road (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 forming a portion of the flow of the Entiako River into the Nechako Reservoir.

5.6 Regional Tectonics and Seismicity

A seismicity assessment was carried out for the Project area in 2013, 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 WRSFs 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.

The Project is situated within central BC, where the level of recorded historical seismic activity has been low. The maximum earthquake magnitude for this region of central BC is estimated to be about magnitude 7.0, based on historical earthquake data and the regional tectonics (Adams and Halchuk, 2003). There is potential for larger earthquakes of up to about magnitude 7.5 along the coastal region of mainland western BC, including the Coast Mountains. The high seismicity offshore and west of the Project site is associated with the Queen Charlotte fault system, which defines the boundary between the Pacific and North American tectonic plates. There is a potential for earthquakes of up to about magnitude 8.5 along this fault system (Adams and Halchuk, 2003). The seismic hazard along the southwest coast of BC is significant because of subduction zone earthquakes along offshore faults and within the subducting oceanic tectonic plate (Juan de Fuca plate). There is potential for very large earthquakes of magnitude 8.0 to 9.0+ along the Cascadia subduction zone.

A design earthquake magnitude 8.5 was selected for earthquake return periods of 500, 5,000, and 10,000 years, based on the review of regional tectonics and historical seismicity, and the findings of deaggregation of the probabilistic seismic hazard. This represents large magnitude earthquakes along the Queen Charlotte fault system and Cascadia subduction zone. The potential for shallow crustal earthquakes closer to the Project site was also considered for longer return period events of 5,000 and 10,000 years, representing earthquakes of up to about magnitude 7.5 along coastal BC.

5.7 Comments on Section 5

In the opinion of the QP:

- Mining activities should be capable of being conducted year-round
- There is sufficient suitable land available for any 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 Work History

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).
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 totaling 207,333 m), geotechnical holes (13 totaling

Year	Operator	Work
		5,003 m), metallurgical test holes (20 totaling 1,816 m), waste rock characterization (14 holes totaling 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.
2014	New Gold	Feasibility study; estimation of Mineral Reserves based on 2013 resource estimate. 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.

7 GEOLOGICAL SETTING AND MINERALIZATION

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

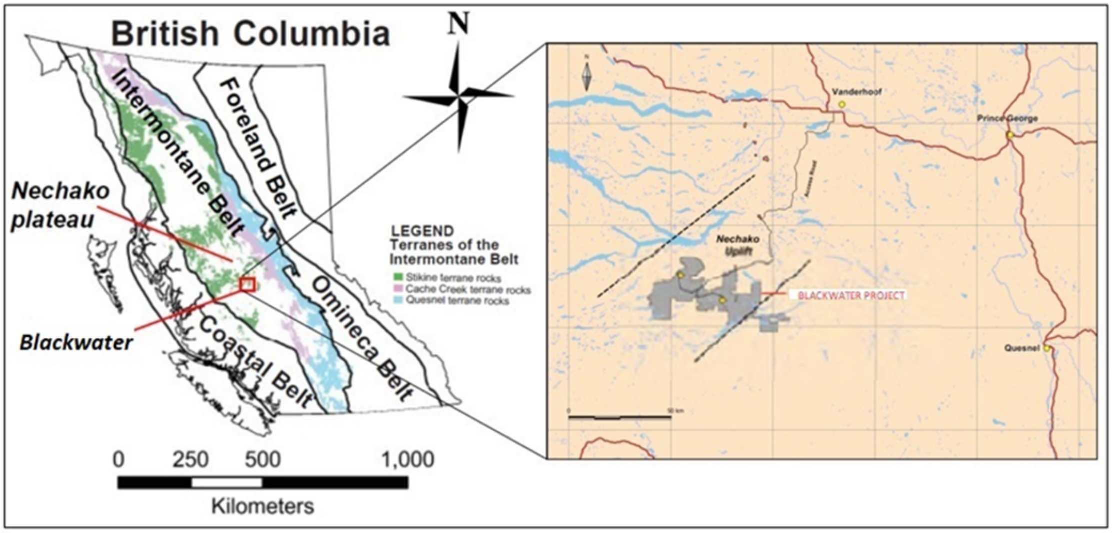

Figure 7-1 Blackwater Project Location and Tectono-Stratigraphic Setting (source: New Gold, 2014)

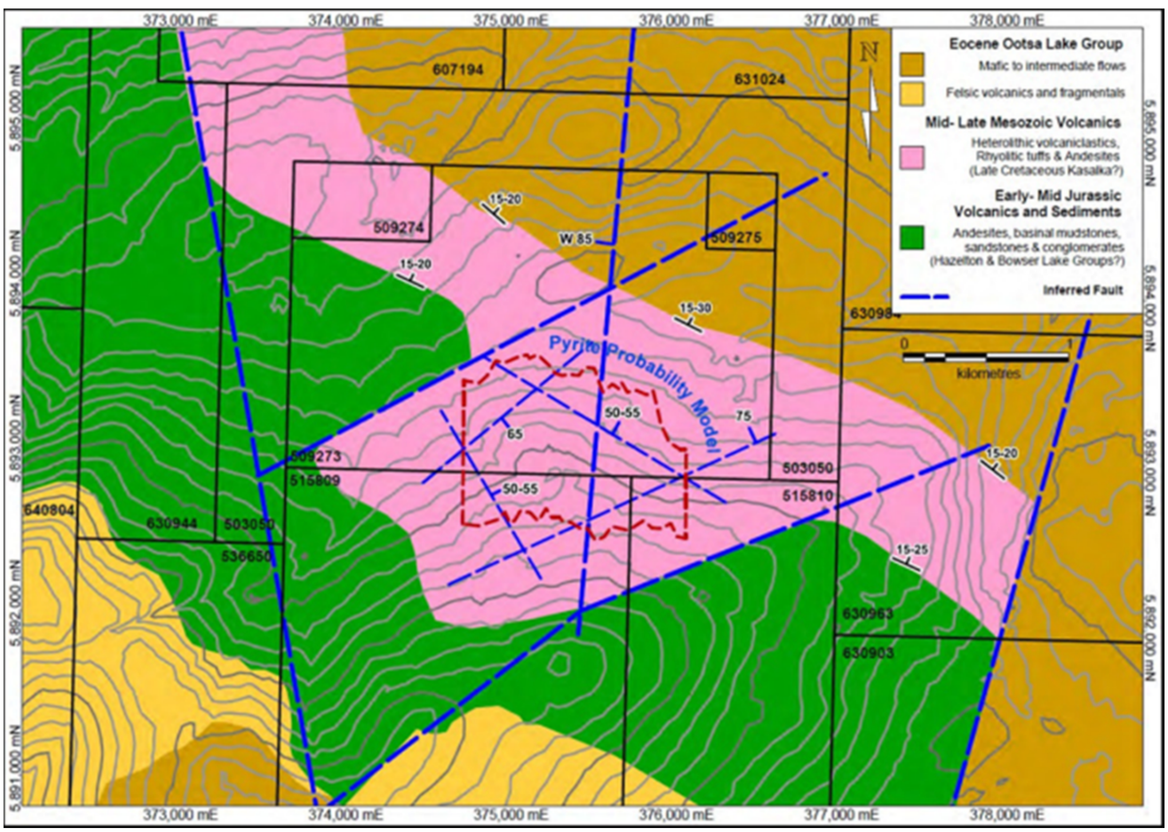

Figure 7-2 Top of Bedrock Geology in Vicinity of Blackwater Deposit (source: New Gold, 2014)

The Project site 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 and 74.1 ± 2.2 Ma (Mortensen, 2011). The adjacent andesites have been 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 Mn-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
OB	Overburden
AND	Andesite
FT	Felsic tuff
FLPT	Felsic lapilli tuff
VC	Volcaniclastic
EC	Epiclastic
SED	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 and east-northeast-trending faults along UTM easting 375,600E. This 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 (NH4) 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 NH4 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 – 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 holes 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.

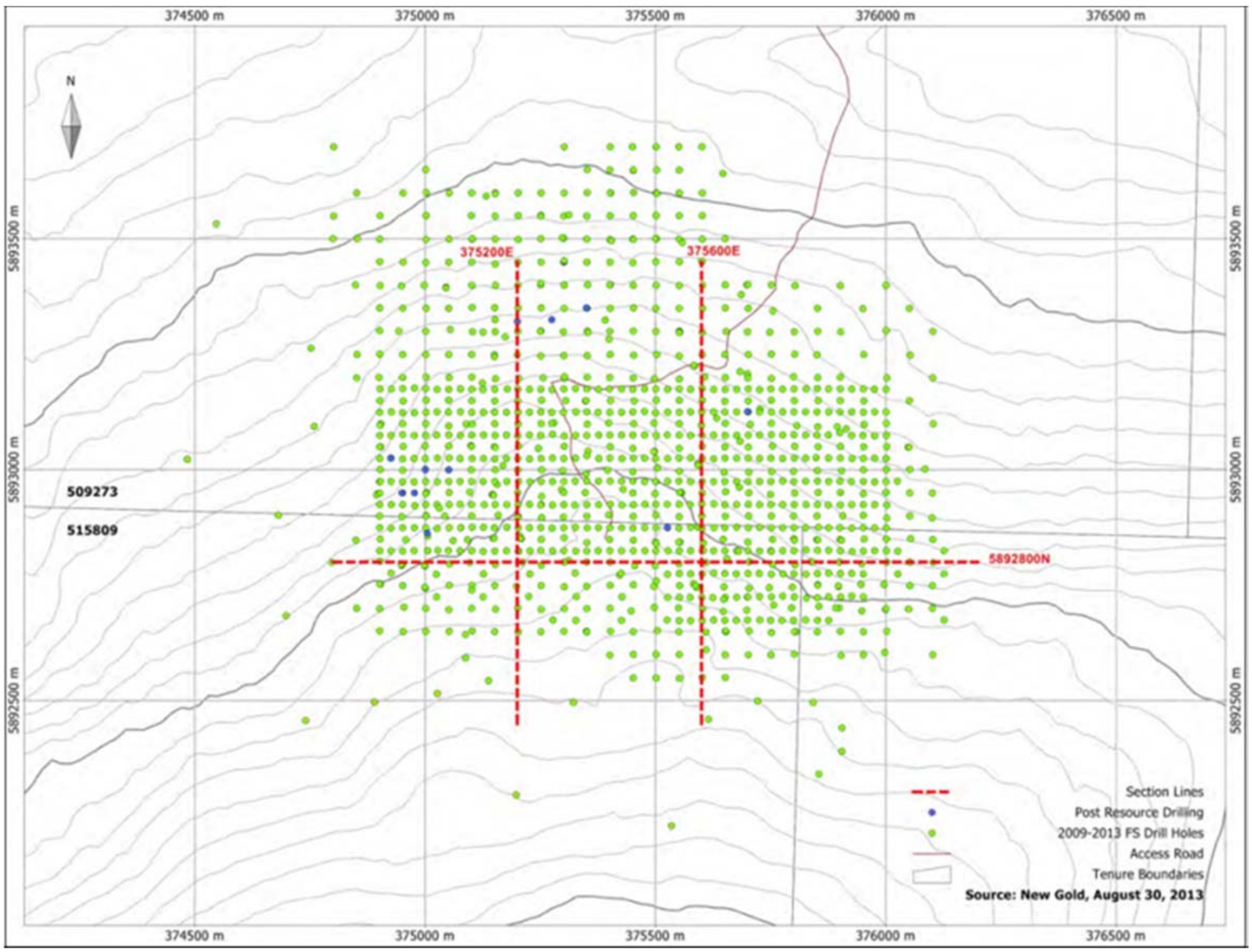

Figure 7-3 Drillhole Plan Showing Location of Referenced Cross-Sections

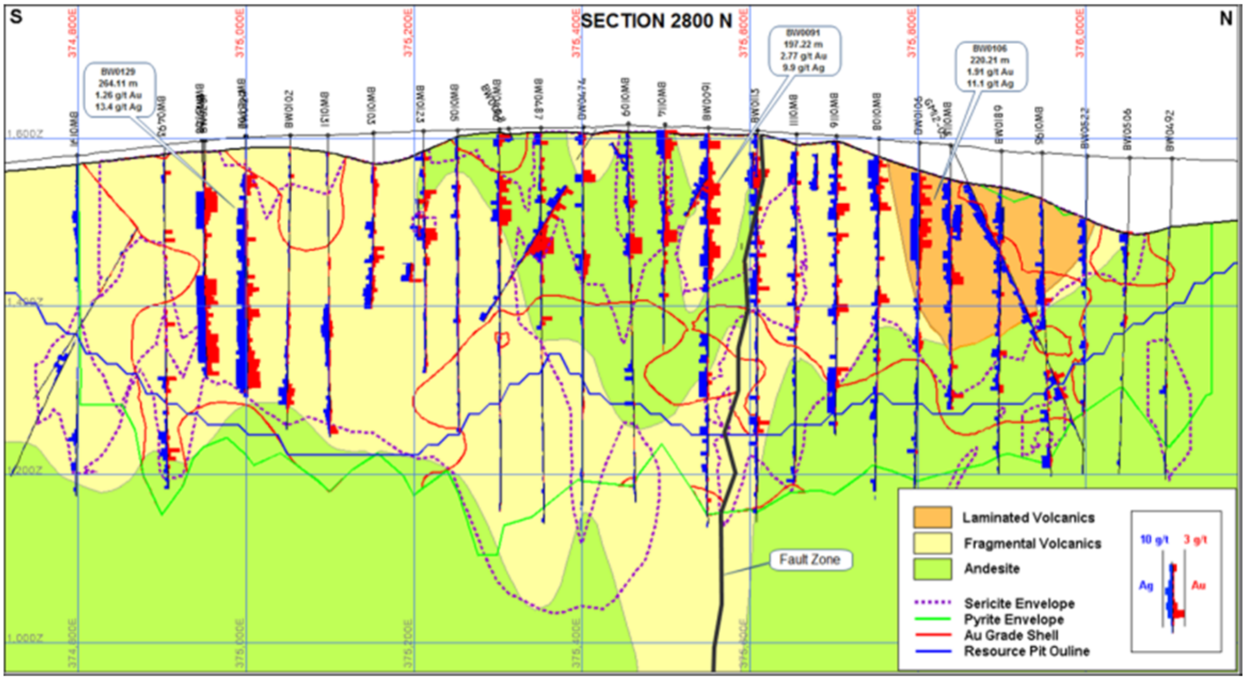

Figure 7-4 Cross-Section 2800 N (source: New Gold, 2014)

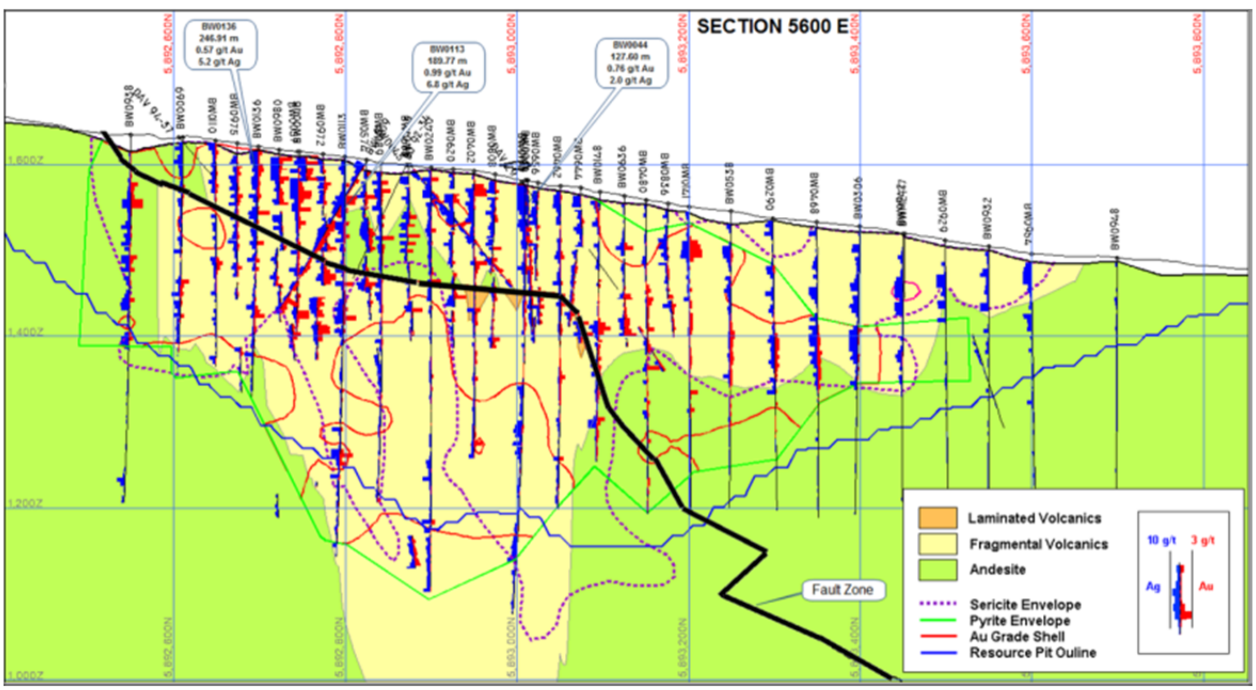

Figure 7-5 Cross-Section 5600 E (source: New Gold, 2014)

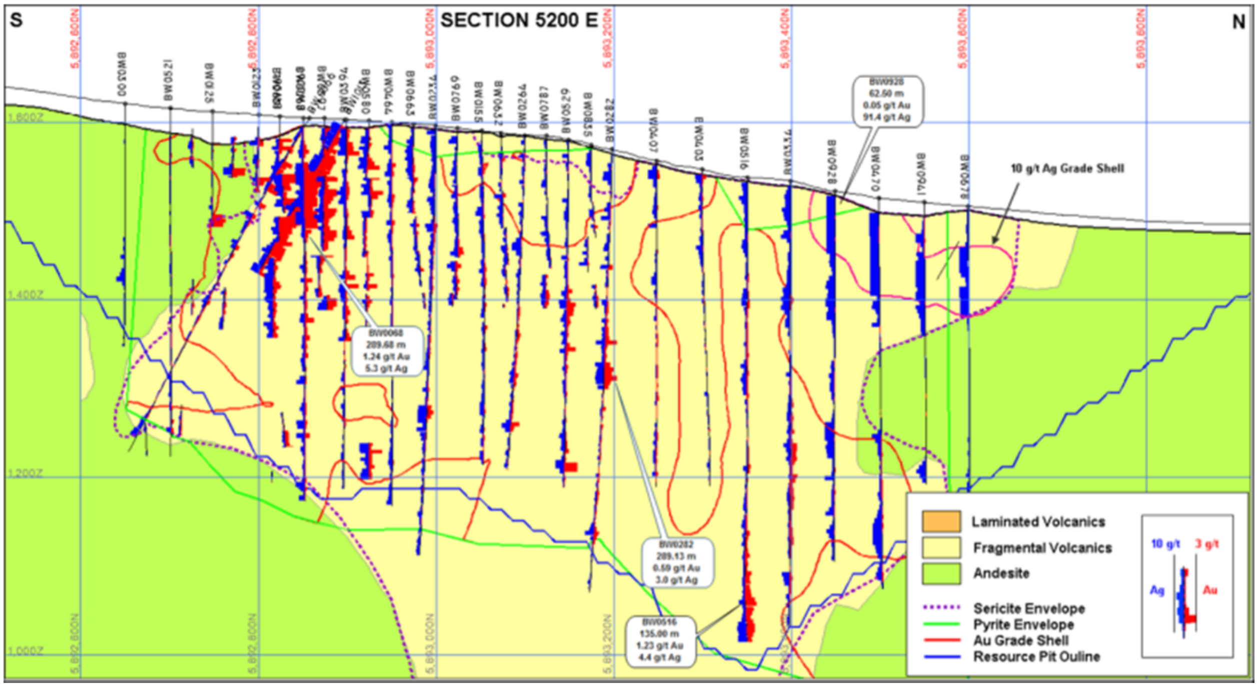

Figure 7-6 Cross-Section 5200 E (source: New Gold, 2014)

8 DEPOSIT TYPES

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 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).

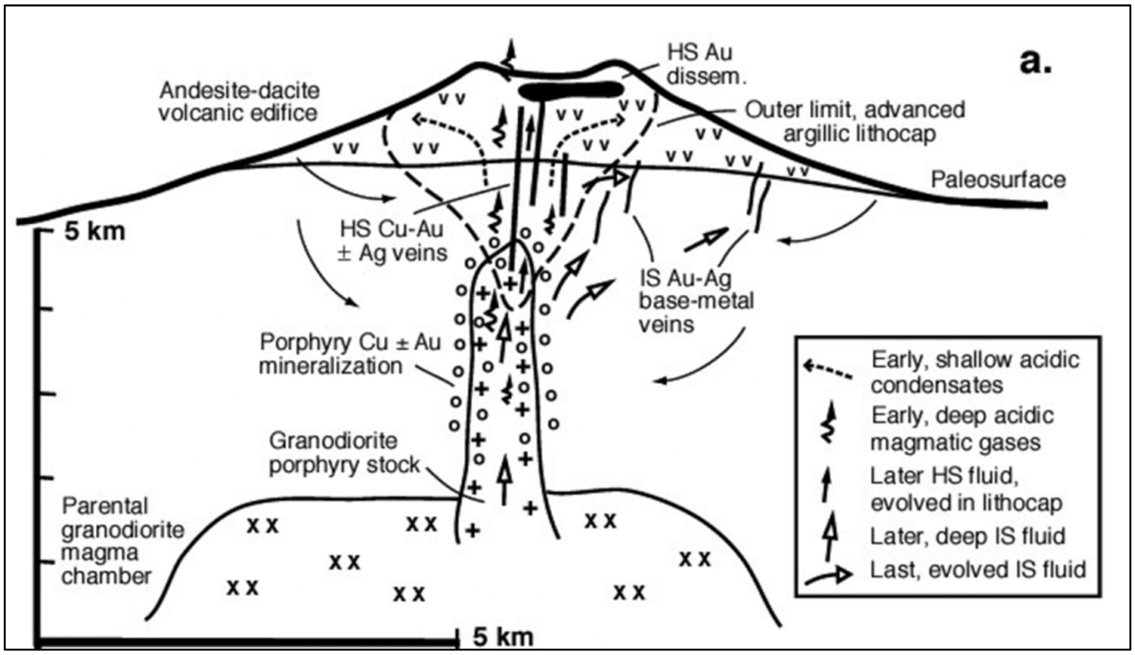

Figure 8-1 Schematic Section of Calc-Alkaline Volcanic Arc Setting and Associated Epithermal and Related Mineralization (source: New Gold, 2014)

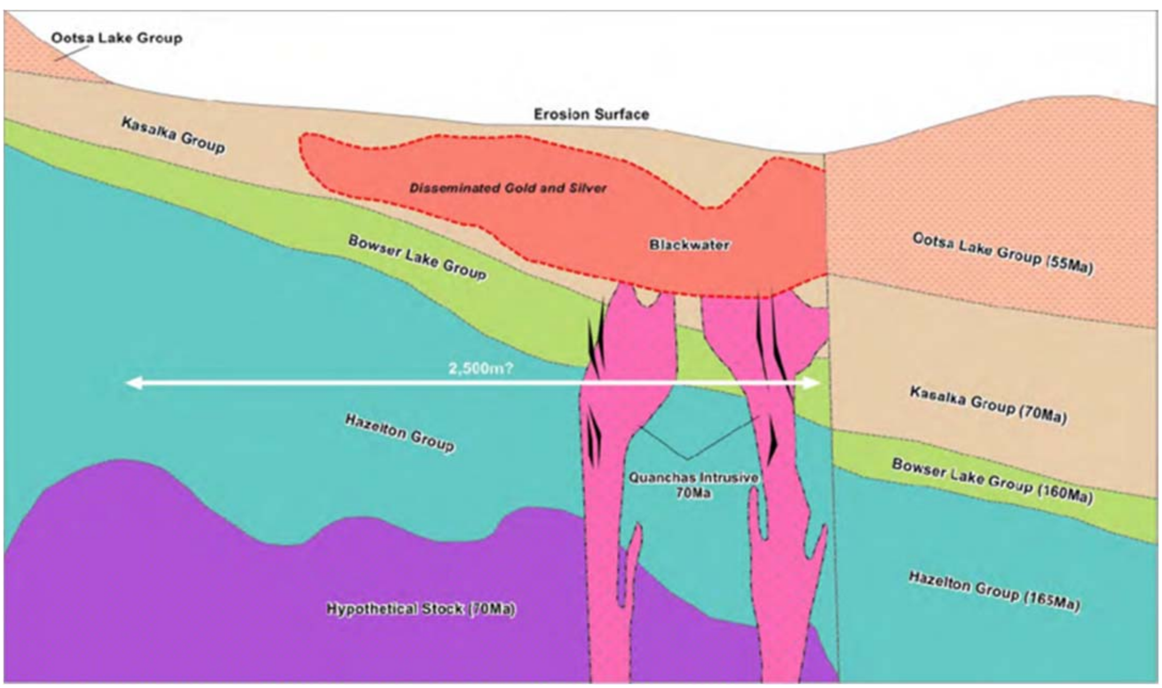

Figure 8-2 Cross-Section of Conceptual Blackwater Model (source: New Gold, 2014)

Table 8-1 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)	Potosi, 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 Undetermined	Tellurides common; selenides present locally	None known, but few data	Tellurides common locally; selenides uncommon	Selenides common; tellurides present locally	Tellurides abundant; selenides uncommon

APS, aluminum-phosphate-sulfate minerals

Characteristic of Blackwater Deposit

9 EXPLORATION

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

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

9.1 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 in the northwestern part of the resource area reached bedrock and returned an anomalous silver grade from a grab sample. Mapping of pits and road-cut exposures over the deposit confirmed the geologic interpretation of the deposit in the subsurface.

Results from drilling by Richfield and New Gold indicate areas of shallow overburden cover near the centre of the deposit that may be potential targets for future bulk sampling or trench mapping/sampling programs.

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) of Toronto to conduct a Titan 24 DC resistivity and IP chargeability geophysical survey. The objective of the study was to determine the relationship between IP chargeability and resistivity and zones of known gold mineralization within the mineral resource area to aid in geologic interpretation and drill targeting. The survey was carried out along five 3.5 km long north-south 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 correlate positively 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. 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 was 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 has been 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 spacing 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 1,041 core drillholes (317,718 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 907 by New Gold. The drilling of 109 condemnation holes has been discussed by others (New Gold, 2014). BW Gold has not conducted any drilling since acquiring the Project.

Table 10-1 2020 PFS 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	New Gold	2013	1	420
PH13 series and PH12-2-3 (pilot Holes)	New Gold	2013	4	1,645.74
Subtotal	New Gold		*907*	*273,033.90*
Grand Total			1,041	317,718.40

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.

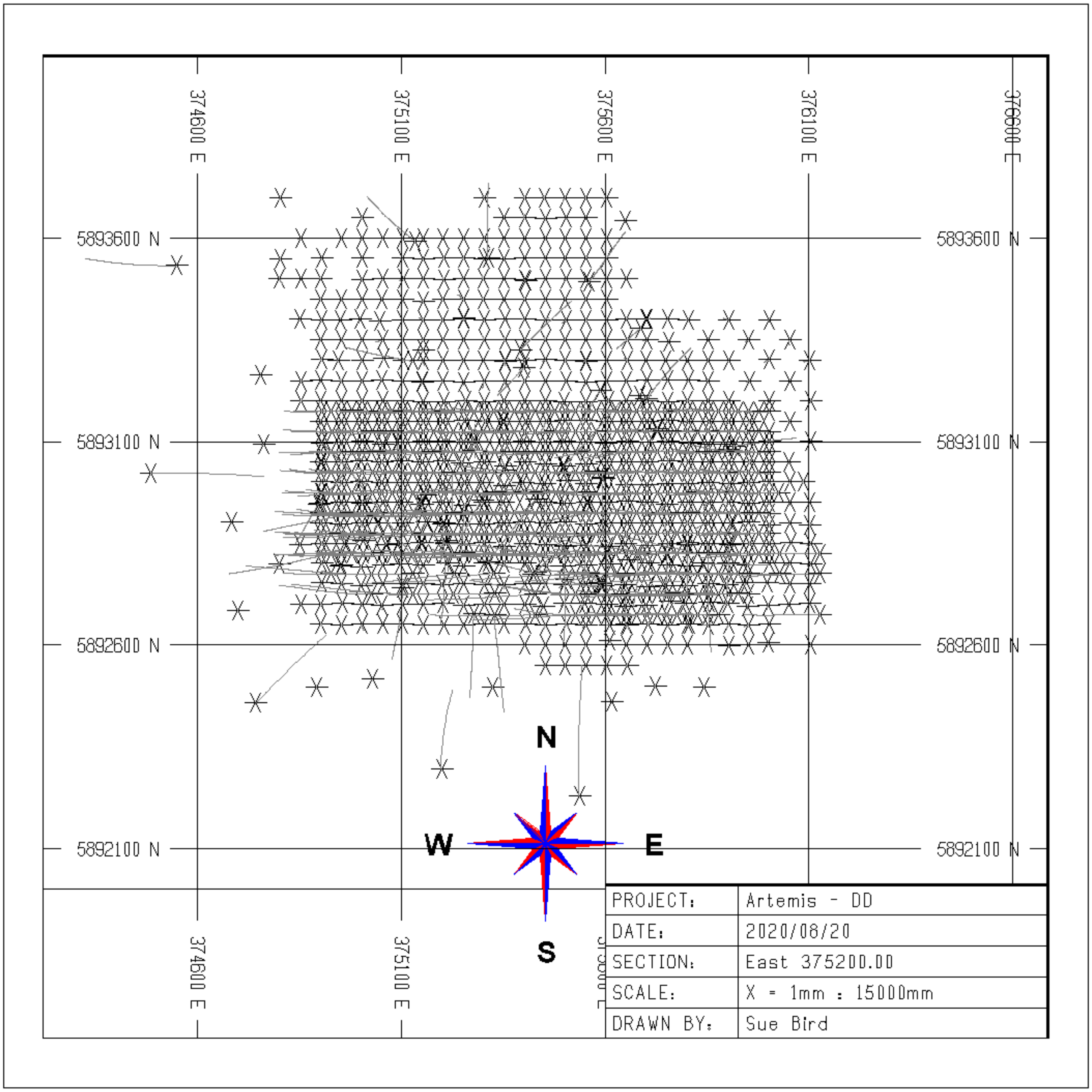

Figure 10-1 Project Drillhole Location Plan

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™. An RQD measurement is the cumulative length of core pieces longer than 10 cm in a given core run divided by the total length of that run. 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 have been 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 1,041 holes, 1,025 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 geomechanical (HQ) drillholes were drilled to collect information for the pit slope stability assessment for the 2014 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 hydrogeological observation wells (HQ), 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 HQ holes, BWMET01–04, and 23 PQ holes, BWMET05–27. The total length of these metallurgical holes was 4,098 m. Information gathered from these drill holes is used in support of the 2020 PFS.

10.9 Waste Rock Characterisation Drilling

Fourteen specific waste rock characterisation holes (HQ) were drilled, BWWR01–14. The total length of these holes was 2,952.5 m. Information from these drill programs is used in the 2020 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 2020 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 Drillhole Summary

YEAR	COMPANY	Holes	Meters Assayed	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 Sample Length/True Thickness

Typical drillhole orientations are as indicated in the example cross-sections in Section 7 (refer to Figure 7-4 to Figure 7-6). The sections show examples of summary assay (or mineralization intensity) values. The grade variations encountered in the drilling are illustrated by colour-coded down-hole histograms for gold and silver and indicate areas of higher grades, low-grades, and intervals of higher grades in lower-grade zones. The sections confirm that sampling is representative of the gold–silver grades in the deposits.

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. Depending on the inclination of an individual drillhole, and the local dip of mineralization, drill intercept widths are approximately equivalent to true widths.

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 estimation. There are no known sampling or recovery factors with these programs that could materially impact the accuracy and reliability of the results.

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 GeoSim (2011a, b; 2012), AMEC (2012c) and New Gold (2014).

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

Quality control procedures implemented in 2012 for silver analysis show acceptable levels of precision and accuracy for silver results. The 2012 QC results mitigate previous concerns about the accuracy and precision of pre-2012 silver results resulting from a lack of comprehensive silver QC.

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

11.1 Sampling Methods

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.

11.2 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 BW Gold.

11.3 Sample Preparation and Analysis

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

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

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

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 material (CRM) standards, blanks, field duplicates, and pulp duplicates. The drillhole database was verified by MMTS, who performed an analysis of more than 51,000 QA/QC assays, approximating 15% of the assay database. The analysis presented below shows the data is of sufficient quality for resource estimation and no significant problems have been identified.

11.6.1 Standards

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

A subset of 16,309 gold assays of CRM insertions was checked to confirm accuracy. These were selected to include the larger instances of a single CRM insertion and include expected values across a wide range of assays. The results of 29 gold CRMs are presented in Table 11-1. The results show few fails at the ± 3 SD from the expected value due to the previously described diligent identification and re-assay of failed samples. Of these fails, most appear to be likely mislabeled as the value is significantly different from the expected value. It appears in these cases the samples were not rerun or relabeled, but because the instance of them is low it does not present a problem. The mean of the assays compares closely to the expected value, in the cases where the error approaches 5%, the mean is lower than the expected value which is acceptable. The CV of the assays of the CRMs is reasonable and not indicative of any problems.

Table 11-1 Au CRM Checks

CRM (Au)	Samples	High Fail Au	Low Fail Au	Percent Fail	Expected Value Au (g/t)	Sample Average Au (g/t)	% Error	StdDev of Au (g/t)	CV
GLG310-3	145	0	1	0.7%	0.119	0.121	1.6%	0.008	7.0%
G911-6	239	0	1	0.4%	0.17	0.161	-5.5%	0.006	3.7%
G303-8	3,067	3	0	0.1%	0.26	0.247	-5.1%	0.026	10.5%
G308-7	245	0	0	0.0%	0.27	0.257	-4.9%	0.009	3.4%
G310-4	3,172	1	0	0.0%	0.43	0.414	-4.0%	0.015	3.5%
CGS-27	156	0	0	0.0%	0.432	0.447	3.3%	0.020	4.4%
GS-P4A	263	0	0	0.0%	0.438	0.446	1.9%	0.014	3.1%
PM449	311	0	0	0.0%	0.45	0.452	0.4%	0.012	2.7%
G311-1	247	0	0	0.0%	0.52	0.509	-2.2%	0.018	3.6%
CGS-22	295	0	1	0.3%	0.64	0.641	0.2%	0.038	5.9%
G310-6	2,191	1	4	0.2%	0.65	0.628	-3.6%	0.024	3.9%
GS-P7B	193	0	0	0.0%	0.71	0.724	1.9%	0.030	4.1%
ME-1	61	0	0	0.0%	0.87	0.876	0.7%	0.021	2.5%
G907-2	1,133	1	0	0.1%	0.89	0.876	-1.6%	0.066	7.5%
PM452	246	0	0	0.0%	0.952	0.976	2.4%	0.028	2.9%
GS-1H	148	0	0	0.0%	0.972	0.991	1.9%	0.044	4.4%
GS-1G	133	0	0	0.0%	1.14	1.159	1.7%	0.038	3.3%
G311-5	625	0	0	0.0%	1.32	1.308	-0.9%	0.035	2.7%
GBMS911-3	654	1	1	0.3%	1.33	1.329	-0.1%	0.098	7.4%
GS-1P5D	222	0	0	0.0%	1.47	1.470	0.0%	0.056	3.8%
PM440	147	0	0	0.0%	1.62	1.655	2.1%	0.030	1.8%
ME-2	523	0	0	0.0%	2.1	2.102	0.1%	0.053	2.5%
G308-8	328	0	0	0.0%	2.45	2.428	-0.9%	0.062	2.5%

CRM (Au)	Samples	High Fail Au	Low Fail Au	Percent Fail	Expected Value Au (g/t)	Sample Average Au (g/t)	% Error	StdDev of Au (g/t)	CV
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 which 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 Au 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.

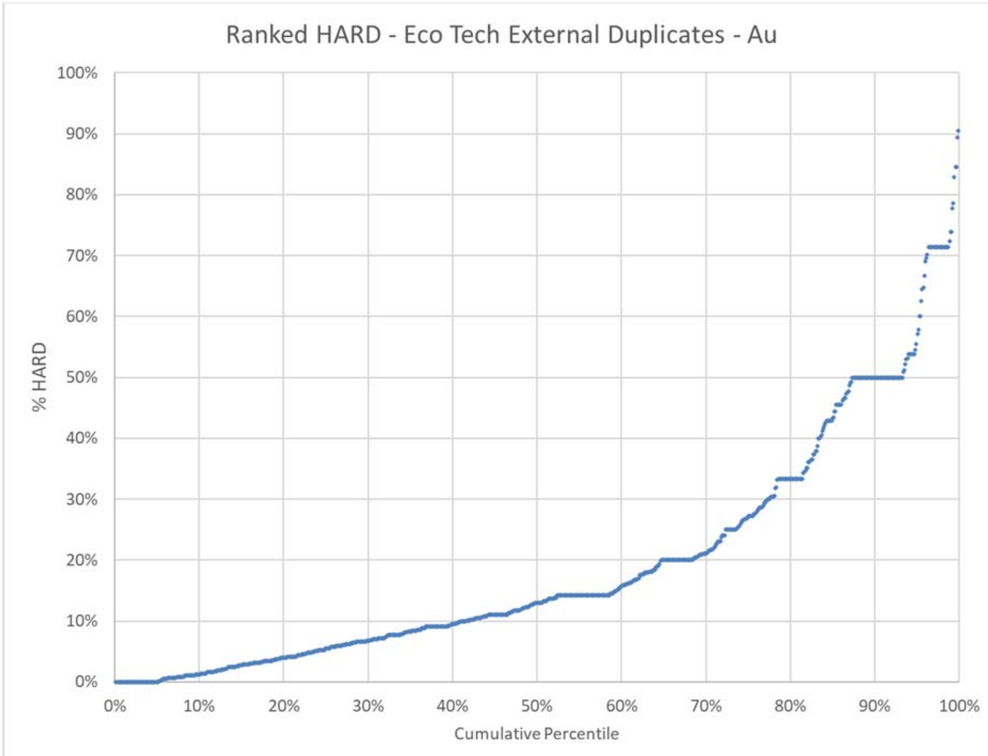

Figure 11-1 Ranked HARD Plot of Eco Tech External Duplicate Pairs – Au (source: MMTS, 2020)

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.

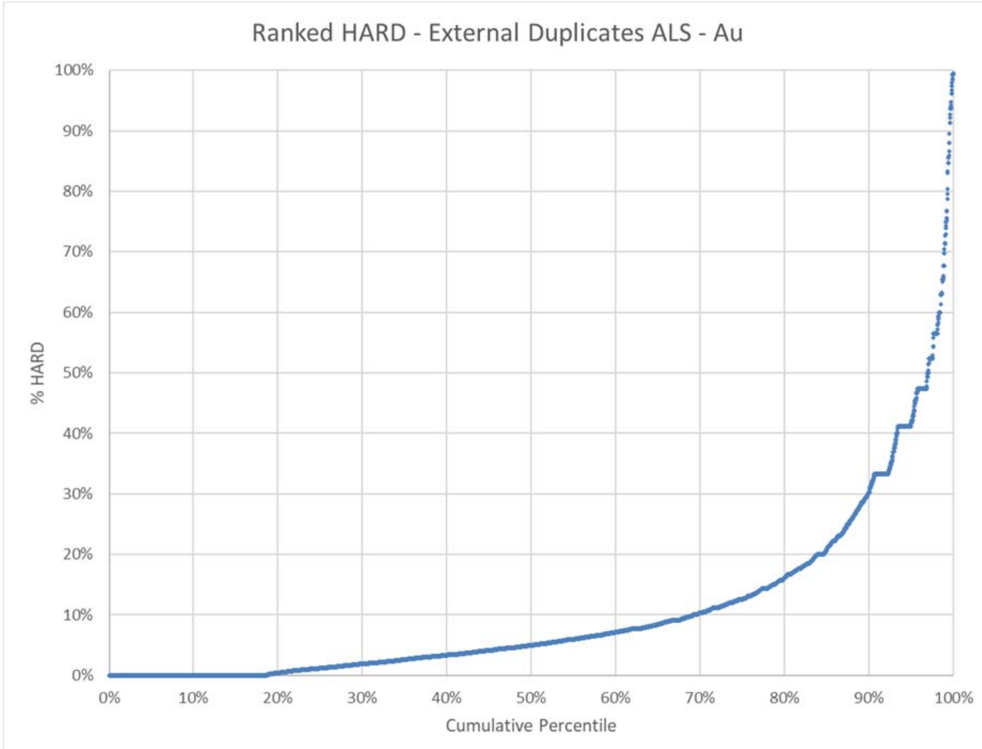

Figure 11-2 Ranked HARD Plot of ALS External Duplicate Pairs – Au (source: MMTS, 2020)

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

		Ag Grade	
Lab Comparison	**Number of duplicates**	**Mean (g/t)**	**Median (g/t)**
Eco Tech SGS Difference	845	3.92 3.88 1.0%	1.7 1.8 -5.9%
ALS SGS Difference	2938	3.7 3.56 3.8%	1.5 1.5 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.

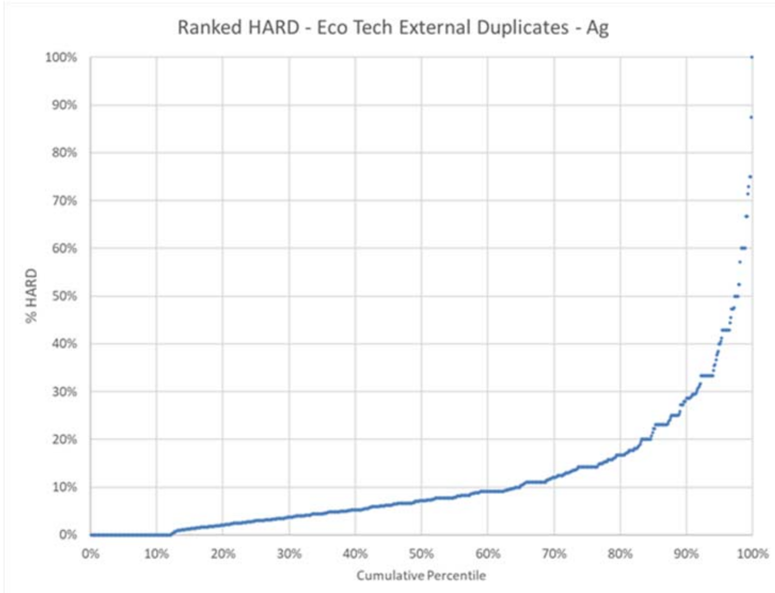

Figure 11-3 Ranked HARD Plot of Eco Tech External Duplicate Pairs – Ag (source: MMTS, 2020)

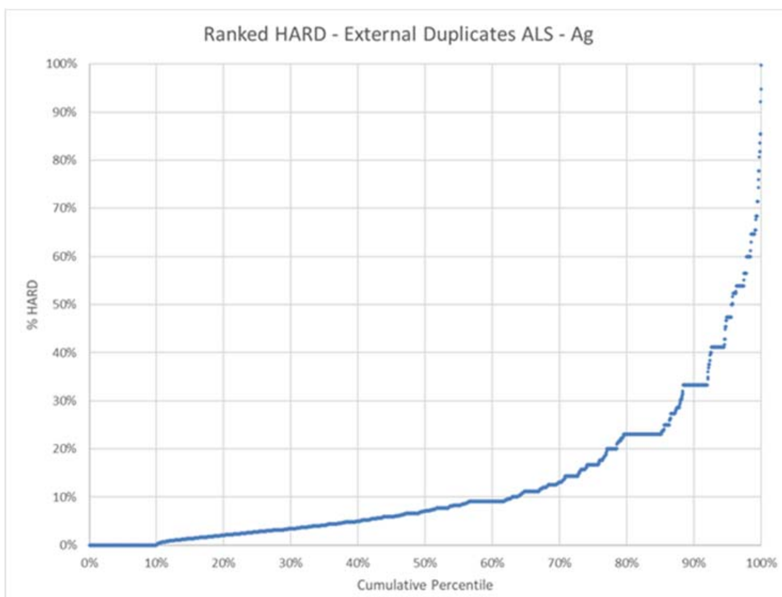

Figure 11-4 Ranked HARD Plot of ALS External Duplicate Pairs – Ag (source: MMTS, 2020)

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 Field Duplicates

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

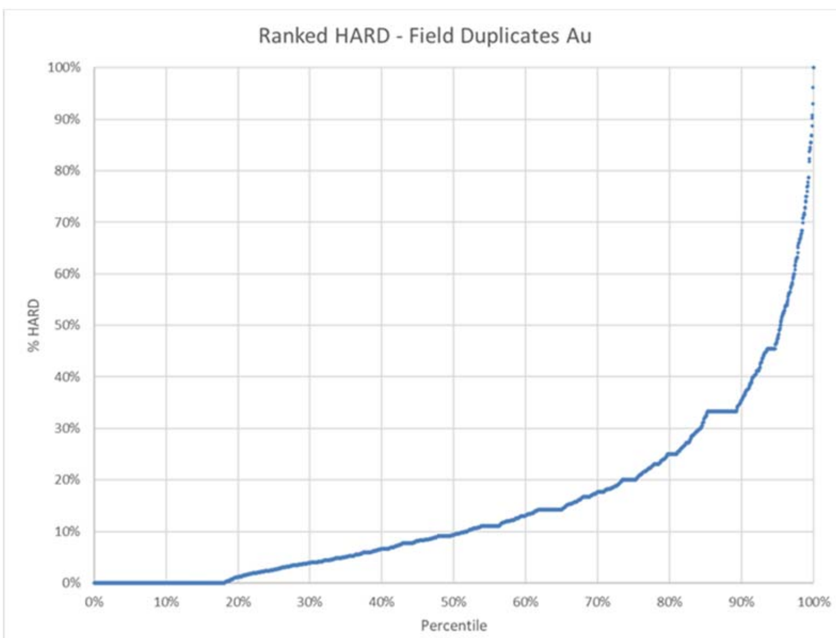

Figure 11-5 Field Duplicates Ranked HARD plot – Au (source: MMTS, 2020)

11.6.5 Sampling Procedure Optimisation

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.

11.7 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 2020 PFS.

12.4 Conclusion

In the opinion of the QP, the data verification is appropriate to support a resource estimate.

13 MINERAL PROCESSING AND METALLURGICAL TESTING

13.1 Introduction

AMEC (2014) presented metallurgical test data from extensive test work carried out from 2008 to 2013 by eight laboratories i.e. Inspectorate, G&T Laboratories, SGS (Lakefield and Vancouver), Dawson Metallurgical Laboratories, McClelland Laboratories, Pocock and MetSolve Laboratories. 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.

Comminution test work was carried out by G&T Laboratories and the average values obtained are shown in Table 13-1.

Table 13-1 Comminution Test Results

Ore Type		Axb SAG Mill Breakage Parameter	BMWi Bond Ball Mill Index (kWh/t)	Ai Abrasion Index
Oxide	Average	55.8	14.5	0.111
	Std. Dev.	13.0	2.16	0.042
Transition	Average	38.0	16.0	0.142
	Std. Dev.	8.7	2.71	0.099
Sulphide	Average	31.1	18.3	0.206
	Std. Dev.	5.3	1.8	0.114
All Types	Average	35.7	17.4	0.182
	Std. Dev.	11.1	2.45	0.110

Whole ore leaching test work was carried out by SGS, McClelland and MetSolve and showed that a grind to a P80 of 150 µm was optimum and pre-aeration was effective in reducing cyanide consumption. It was stated in the processing section of the AMEC (2014) report that:

- All samples responded well to direct cyanidation with extractions of about 90%;
- Silver extractions of 45% to 65%% were obtained;
- Leach times of 30 hours were recommended to ensure maximum extraction;
- The addition of lead nitrate had no effect on recovery;
- 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 Gold and Silver Extractions from 2014 FS

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 Recent Test Work

An extensive program of test work was carried out in 2019 by BaseMet Laboratories. 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. Fifteen drill core intervals were used for the first composite, 19 for the second and nine for the third. A P80 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 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.

13.3 Comminution Testing

Table 13-3 shows the results from four twin holes drilled specifically to provide sufficient samples for comminution testwork. Table 13-4 shows the results obtained from composites 1, 2 and 3. The results are similar to these obtained in the previous test work, but the Bond ball mill work index is somewhat higher for the composites than for the twinned holes. This highlights the variability in the hardness of the ore.

Table 13-3 Comminution Test Results from 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

Table 13-4 Bond Ball Mill Index Results from Composite Samples

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

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 2014FS, 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 samples to assess the effectiveness of gravity concentration and the results presented in Table 13-5 were obtained.

Table 13-5 Gravity Test Work 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, 95.4% with gravity. This, with similar results at finer grind sizes, led to the use of gravity concentration in all subsequent tests. This test work also demonstrated that finer grinds than a P_{80} of 150 µm did not give a significant increase in gold recovery.

13.5 Leach Testing

The analysis of the three one-year composites is shown in Table 13-6.

Table 13-6 Analyses of the Three Composites used for Leach Test Work

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

S(t) *– total sulphur*
S(s) *– sulfide sulphur*

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

Table 13-7 Sulphide Mineral Distribution in Composites

Mineral	Mineral Content % of total sulphur		
	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 mineral types in the three composites tested in 2019.

Table 13-8 Mineralogical Content of Composites

Minerals	Mineral Content (%)		
	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	1.0	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-limenite	0.1	0.1	<0.1
Apatite	0.3	0.3	<0.1
Others	0.1	0.2	0.3

Notes – Iron oxides include goethite, limonite and iron metals
Others include trace amounts of barite and unresolved mineral species

13.6 Oxygen Uptake Rate

The rate of oxygen uptake was measured for each of the composites. The results show some difference between the composites (Figure 13-1) but the main oxygen demand occurred in the first few hours and this should be accounted for when designing the oxygen addition system.

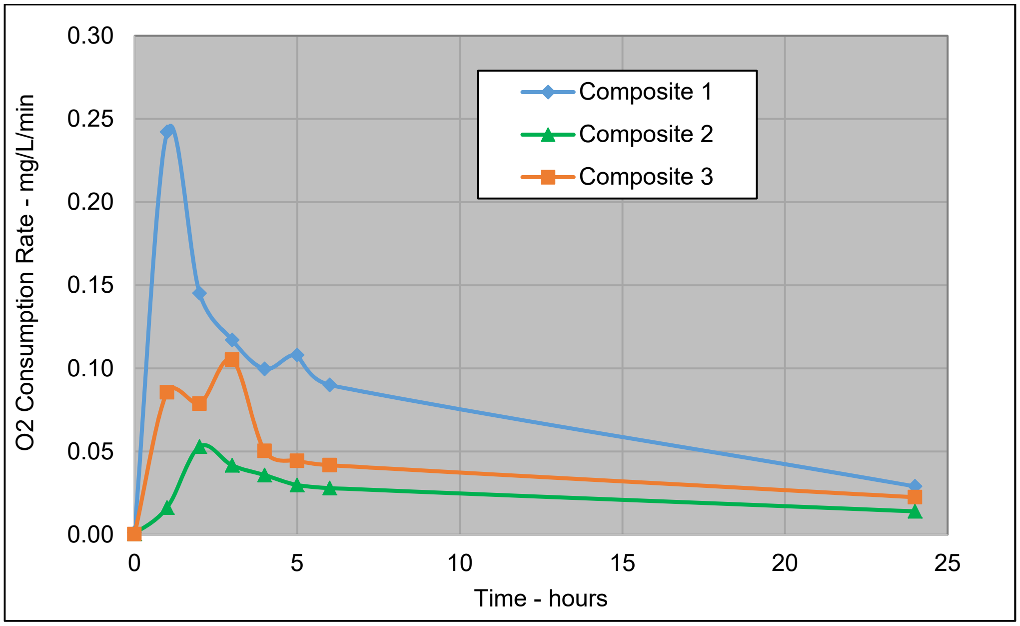

Figure 13-1 Oxygen Uptake Rates During Composite Leaching (source: BaseMet, 2019)

13.7 Leach Results

The results of the series of tests carried out on the three composites are shown in Table 13-9. First, 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 to 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 % solid appeared to reduce the overall recovery by about 2% but this requires confirmation.

In general, oxygen was sparged into the pulp during leaching. 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. 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 Tests Carried Out to Optimize Leach Parameters

Sample ID	Test	Conditions	Extraction - percent			Reagent Consumption	
		Grind,NaCN,Lead Nitrate	Au	Au	Ag	kg/tonne	
			Pan Con	48hrs	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	63.2	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-air, 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 pH11 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,O2,	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	0.61	1.58

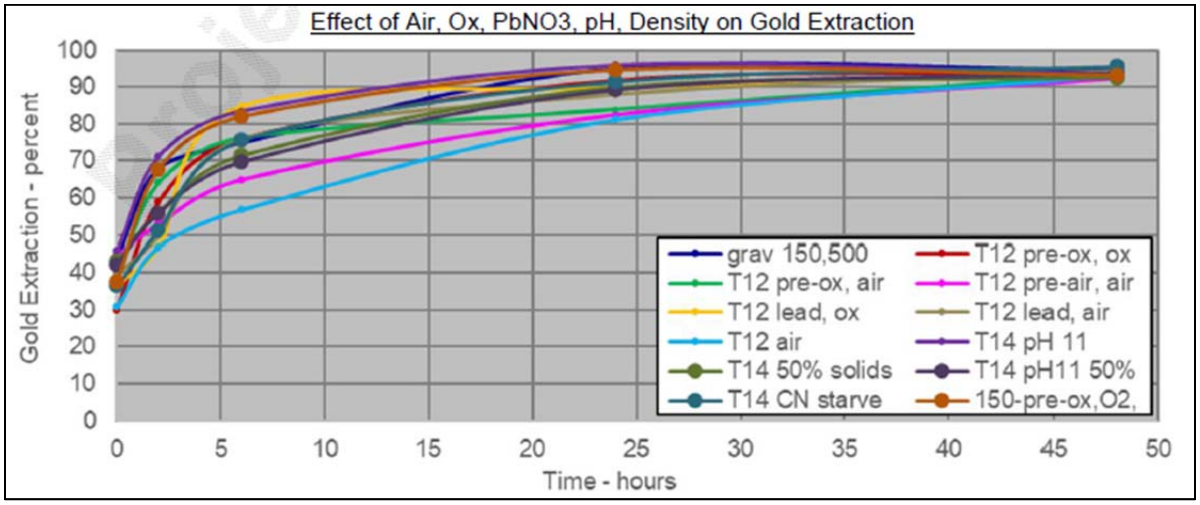

Figure 13-2 Effect of Various Parameters on Leaching Rate (source: BaseMet, 2019)

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%.

13.8 Variability Testing

To investigate the variability of the extraction over the orebody, as part of the 2019 test work carried out by BasMet Laboratories, 48 samples were chosen, distributed throughout the deposit. The extraction testing was carried out using the standard procedure: 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 Table 13-10.

.

Table 13-10 Results of Variability Testing

Sample Number VC	Head Grade g/t Gold	Head Grade g/t Silver	Au Extraction (%)	Ag Extraction (%)	Cu (ppm)	Reagent Consumption NaCN (kg/t)	Reagent Consumption Lime (kg/t)	BBWi kW (hr/t)
2	3.09	8	95.5	52.3	199	1.02	1.01	15.6
3	4.42	13	96.2	64	429	1.53	0.68	19.8
4	0.43	5	93.2	54.7	176	0.49	1.12	21.5
5	0.46	2	92.4	30.8	206	0.54	1.54	21.6
6	0.62	3	91.9	35.4	63	0.45	0.74	
7	0.79	4	99.2	75.6	2640	5.65	2.85	21.1
8	0.32	2	91.8	68.3	90	0.68	0.91	
9	2.10	22	98.4	61.4	115	0.21	0.86	14.0
10	6.21	10	94.3	76.1	789	1.38	0.80	16.0
11	0.61	6	91.4	96.9	63	0.17	1.50	11.8
12	4.71	49	94.2	87.9	131	0.28	0.25	12.1
13	3.66	15	92.9	78.6	135	0.22	0.76	16.8
14*								
15	1.79	8	96.9	65.9	133	0.51	0.80	15.1
16	1.47	4	83.0	36.4	40	1.00	1.04	18.8
17	1.63	3	92.1	69.7	182	0.70	0.99	19.7
18	11.95	15	98.3	65.3	70	1.11	3.22	16.1
19	4.00	9	98.9	78.2	529	0.05	0.06	17.7
20	0.65	13	95.3	87	172	1.12	1.72	19.7
21	0.94	3	94.6	69.1	232	0.82	0.98	24.6
22	0.75	10	97.7	97	43	0.79	0.63	14.4
23	1.18	5	97.4	48.7	167	0.82	1.25	20.5
24	0.90	2	93.1	87.7	98	0.95	1.35	20.2
25	0.59	8	51.1	48.1	101	1.12	0.70	
26	0.69	5	84.2	93.7	45	0.82	1.00	13.9
27	1.36	9	98.9	66.3	179	1.38	2.15	
28	2.22	3	93.6	69.1	206	0.75	0.76	22.1
29	0.81	13	97.7	90.9	76	0.69	0.71	20.4
30	1.20	7	96	56.6	175	0.93	1.04	21.1
31	0.75	5	94	83.5	89	0.99	2.01	21.4
32	0.57	2	98.6	78.3	75	1.15	0.92	18.2
33	1.60	40	78.6	52	296	1.42	1.00	18.2
34	0.92	4	96.9	65.3	60	0.98	1.10	
35	0.83	3	96.2	91	51	0.81	1.05	16.6
36	0.94	13	90.3	57.4	380	1.00	0.63	19.2

Sample Number VC	Head Grade g/t		Au Extraction (%)	Ag Extraction (%)	Cu (ppm)	Reagent Consumption		BBWi kW (hr/t)
	Gold	Silver				NaCN (kg/t)	Lime (kg/t)	
37	2.95	37	79.3	77.5	1940	4.15	0.59	19.0
38	1.74	5	97.2	94.6	363	1.09	1.58	17.7
39	0.30	6	99.2	57.9	292	0.89	0.30	
40	2.81	5	92.5	94.2	418	1.00	0.93	20.3
41	1.09	7	95.7	93.9	99	0.93	1.09	
42	1.41	10	82.0	52.7	1340	1.00	0.47	19.4
43	0.48	12	94.8	52.6	424	1.25	0.98	21.8
44	1.59	6	94.4	60.5	105	0.90	1.09	
45	2.33	8	96.9	85.6	335	0.74	1.24	22.1
46	1.07	3	85.4	35	54	0.43	0.60	
47	8.77	9	96.2	58.3	195	0.91	0.94	
48	1.11	25	92.3	88.3	58	0.59	0.41	
49	1.66	13	91.3	85.2	140	0.48	0.90	
50	0.79	7	91.8	74.5	255	0.99	1.65	
Mean	1.83	9.4	92.6	69.8	301	1.00	1.06	18.5

Note – test 14 was lost

Of the 48 drill composites tested, six (shown in yellow in Table 13-10) gave overall extractions (gravity + leach) of less than 90%, all the other samples gave results significantly greater than 90%. Of the samples which showed low extractions. VC7, VC37 and VC42 gave gold extractions of 64.8%, 79.3% and 82%% respectively and all had high copper contents, of 2,640, 1,940 and 1,340 ppm respectively, as shown in the table. In the case of sample VC7, repeating the test with a higher cyanide concentration raised the extraction to 99%, but the other two samples remained with rather low extractions when cyanide concentration was raised. The remaining samples had an average of 190 ppm copper if these three samples are ignored.

Another of the low recoveries (VC-25) occurred in drillhole BW 0979, which had an abnormally high arsenic content (8,950 ppm).

It is presumed that in all these four cases, either more cyanide was needed, or the gold contained was at least partially encapsulated in copper or arsenic sulphide minerals.

The low extractions obtained from samples VC-33 (hole BW 839) and VC46 (hole BW – 91) require further investigation.

No other deleterious elements other than those discussed above are known that could have a significant effect on potential economic extraction.

The average extraction for gold was 92.6% and 69.8% for silver. Taking into account the results from the composites (94.4 % extraction of gold, 61.5% for silver) and the result of the repeat test on one of the high copper samples, 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 test work was 1 kg/t and the lime consumption 1.06 kg/t. However, in the previous, extensive test work, 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 and the use of pH 10.5 in the test work may have increased the degree of association of the CN- and H+ and led to cyanide losses as HCN. The test procedure used by Basemet Laboratories in the 2019 test work 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 together with the values recorded in previous test work, carried out with fresh core and at a pH of 11, it is recommended that a cyanide consumption of 0.6 kg/t is used.

13.9 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 Table 13-11.

Table 13-11 Results of Cyanide Destruction Test Work

Sample	Detox Test	Target WADCN	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
T46 (Comp 1)	Feed	-	10.3	-	-	-	-	-	367.4	161	9	0.95	55.5
	C1	25	8.4	60	4.0	15	120	2	4.8				
	C2		8.6	60	3.0	15	180	3	26.3	21.6	<0.2	<0.1	<0.1
	C3	10	8.5	60	4.0	15	240	4	3.7				
	C4		8.6	60	3.5	15	270	5	7.5	5.4	0.4	<0.1	<0.1
	C5	1	8.5	60	4.0	15	240	4	3.2				
	C6		8.5	60	5.0	15	180	3	0.3	1.31	<0.2	<0.1	<0.1
	C7		8.5	60	4.5	15	180	3	0.2	0.79	0.60	<0.1	<0.1
	C8		8.5	60	4.5	0	180	3	14.4				
	C9		8.5	60	4.5	15	480	8	0.2				
T79 (Comp 2)	Feed	-	10.5	-	-	-	-	-	273.0	28.1	3.77	0.86	95.1
	C10	25	8.5	60	3.0	15	300	5	11.5	15.6	<0.2	<0.1	0.1
	C11	10	8.5	60	3.5	15	180	3	0.4	1.00	<0.2	<0.1	0.19
	C12	1	8.5	60	3.5	0	60	1	11.4	5.8	<0.2	0.18	0.2
	C13	1	8.5	60	3.5	7.5	240	4	0.2	1.09	<0.2	<0.1	0.23
T90 (Comp 3)	Feed	-	10.3	-	-	-	-	-	264.0	73.7	12.83	0.485	58.7
	C14	25	8.5	60	3.0	15	240	4	24.8	32.9	0.10	<0.1	<0.1
	C15	10	8.5	60	3.5	15	240	4	7.5	4.86	0.11	<0.1	0.10
	C16	1	8.5	60	4.0	15	180	3	0.3	1.33	<0.1	<0.1	<0.1

**C1 target WADCN was 25ppm*

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

14 MINERAL RESOURCE ESTIMATE

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.

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, the Ag is generally lognormally distributed at higher grades, and the Ag mineralization has much lower value to the Project.

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.

As part of the model validation process, a comparison of the Au content in the 2020 model (which used MIK for the Au estimate) to that in the 2014 resource model (which used OK) was done. The comparison has been done using 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 Au content is within 2% for the measured and indicated classes.

Table 14-1 Blackwater Mineral Resource Estimate – Effective date: May 5, 2020

			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 has been prepared by Sue Bird, P.Eng., an independent Qualified Person.*
2. *Resources are reported using the 2014 CIM Definition Standards and were estimated using 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 “reasonable prospects of eventual economic extraction” pit using the following assumptions: US $2,000/oz. Au and US $21.43/oz Ag at a currency exchange rate of 0.75 US$ per $CDN; 99.9% payable Au; 95.0% payable Ag; $8.50/oz Au and $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. Pit slope angles are assumed at 40º.*
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 LG 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**

14.3 Geologic Modelling

The geologic, alteration and structural models were defined 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 site. The domains and interpolation were clipped to the bottom of the overburden surface, and the resource 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 are not used as they do not represent hard boundaries to mineralization.

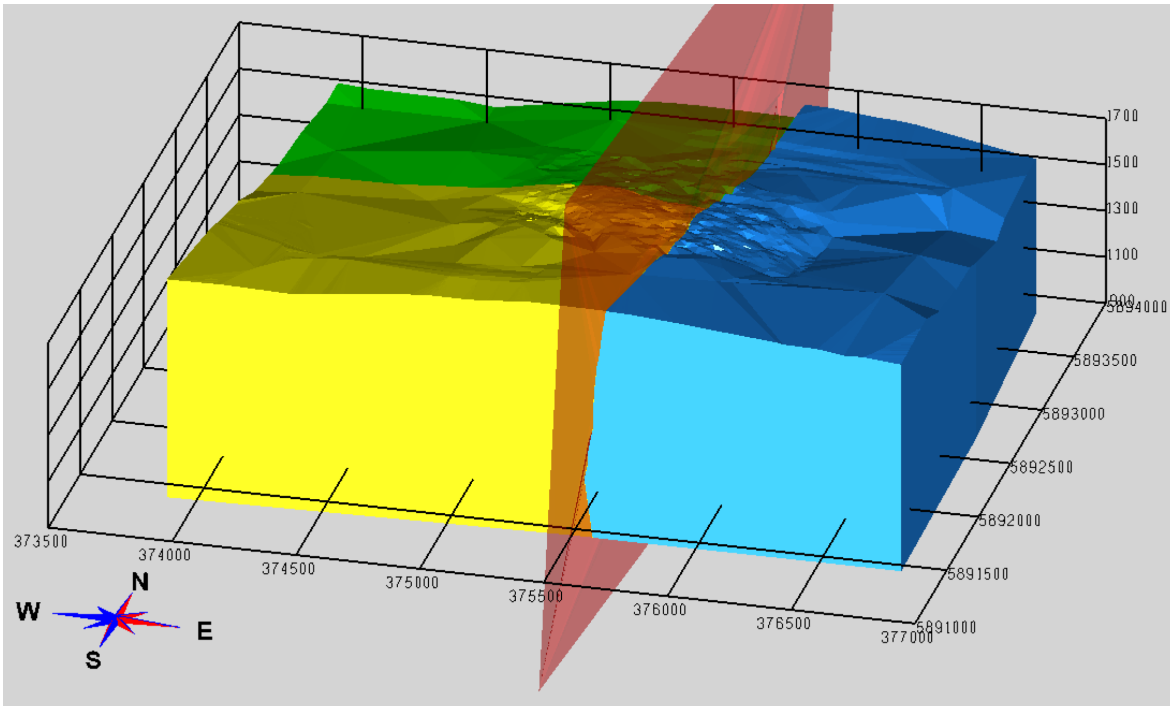

Figure 14-1 Domains used for Interpolation (green=1, yellow=2, blue=3, red=major N-S fault; source: MMTS, 2020)

14.4 Assay Statistics and Capping

The assay statistics were examined using boxplots, histograms, and cumulative probability plots (CPPs). The grade distribution for silver within the domains is generally lognormal. However, the distribution for gold contains inflections points above about the 90% of the data and as well as 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 multiple indicator kriging (MIK), and the interpolation method for silver is ordinary kriging (OK). Figure 14-2 and Figure 14-3 illustrate 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 grade and therefore compositing has not introduced a bias.

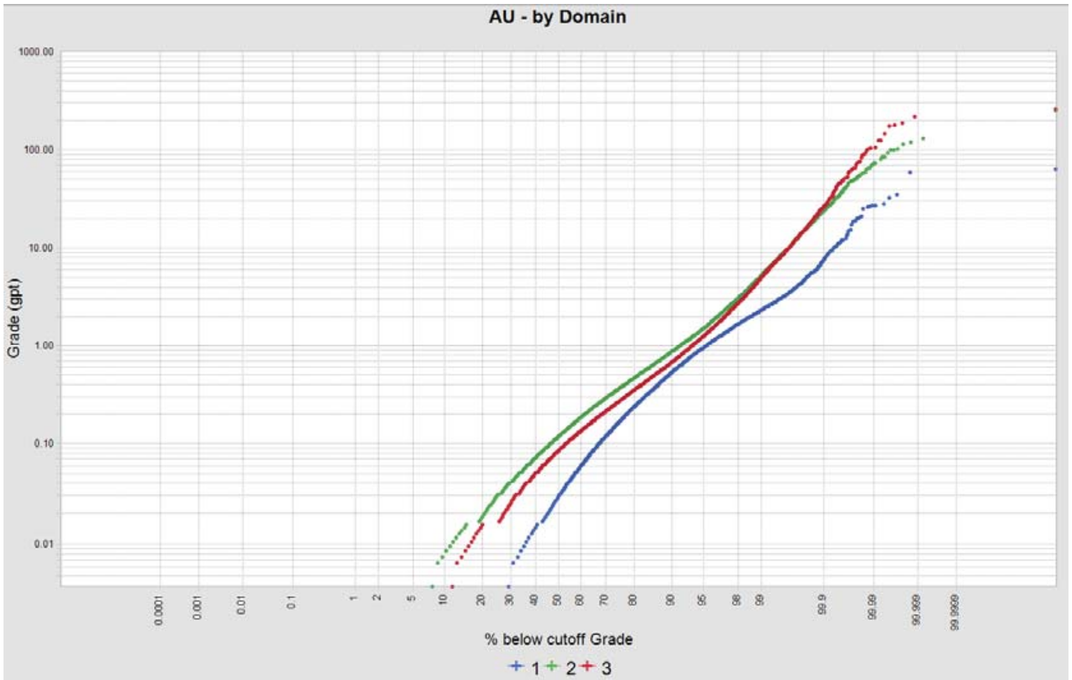

Figure 14-2 CPP of Au Assay Grades by Domain (source: MMTS, 2020)

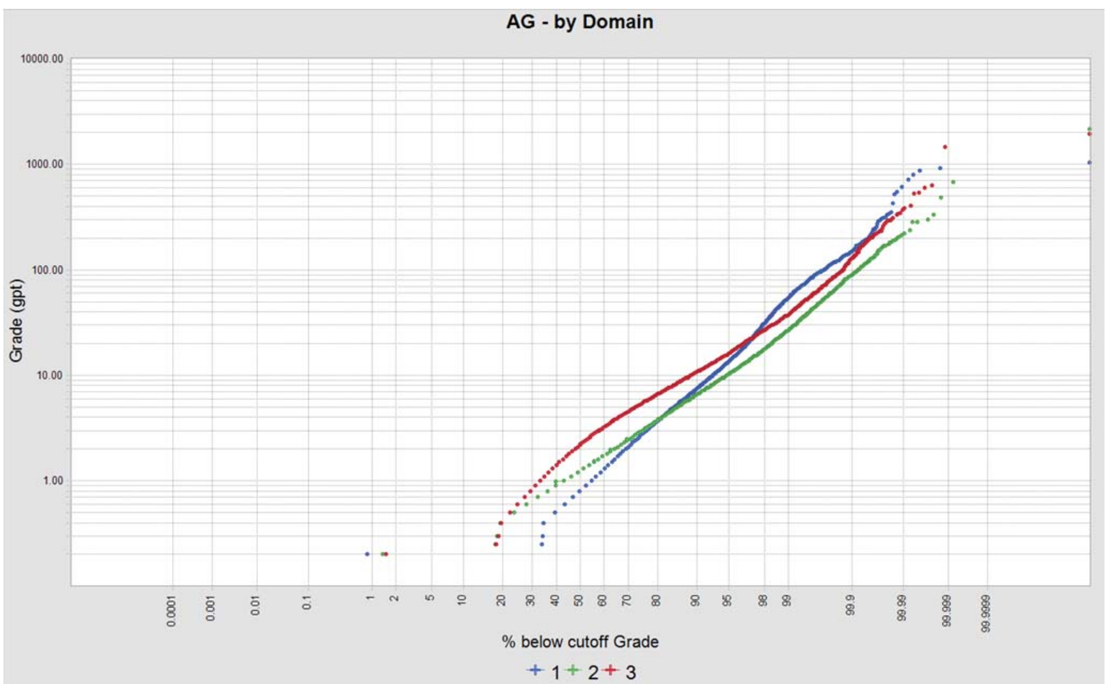

Figure 14-3 CPP of Ag Assay Grades by Domain (source: MMTS, 2020)

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	495	392	215	495	393
	Min (g/t)	0.003	0.003	0.003	0.10	0.10	0.10
	Max	63.70	252.00	262.00	1045.00	2170.00	1950.00
	Wtd mean (g/t)	0.208	0.446	0.401	3.94	3.04	4.90
	Weighted CV	3.609	4.480	6.538	3.96	3.09	2.85
Comps	Num Samples	33,295	69,629	43,355	33,295	69,629	43,355
	Num Missing Samples	241	1350	1708	241	1350	1708
	Min (g/t)	0.003	0.003	0.003	0.10	0.10	0.10
	Max	45.900	132.450	221.000	615.50	1087.90	1705.00
	Wtd mean (g/t)	0.208	0.446	0.401	3.94	3.04	4.90
	Weighted CV	3.083	3.664	5.554	3.44	2.45	2.58
Difference (1-Assay/comp)		0.0%	0.0%	0.1%	0.0%	0.0%	0.0%

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

14.5 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.

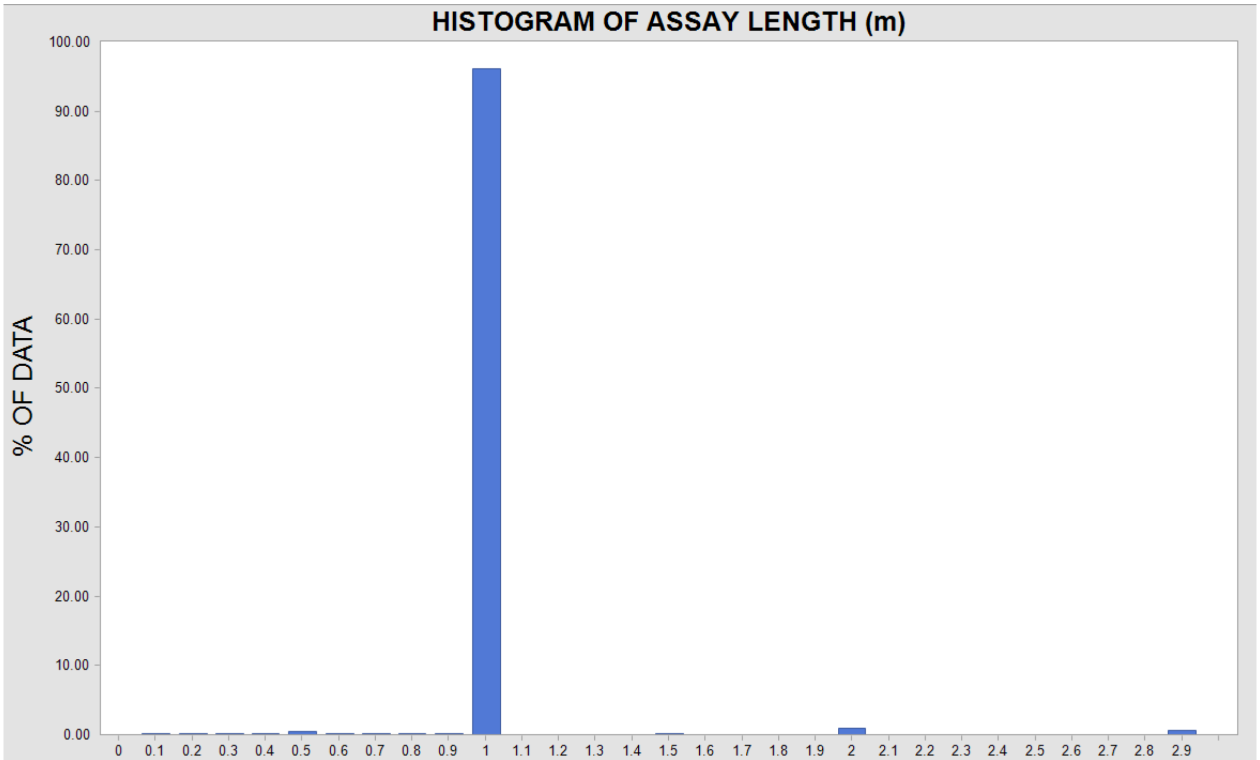

Figure 14-4 Histogram of Assay Lengths (source: MMTS, 2020)

Composite statistics, for the capped values are summarized in Table 14-4. Of note is that the C.V. for Au remains rather high, further pointing to MIK as an appropriate interpolation method for Au. 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

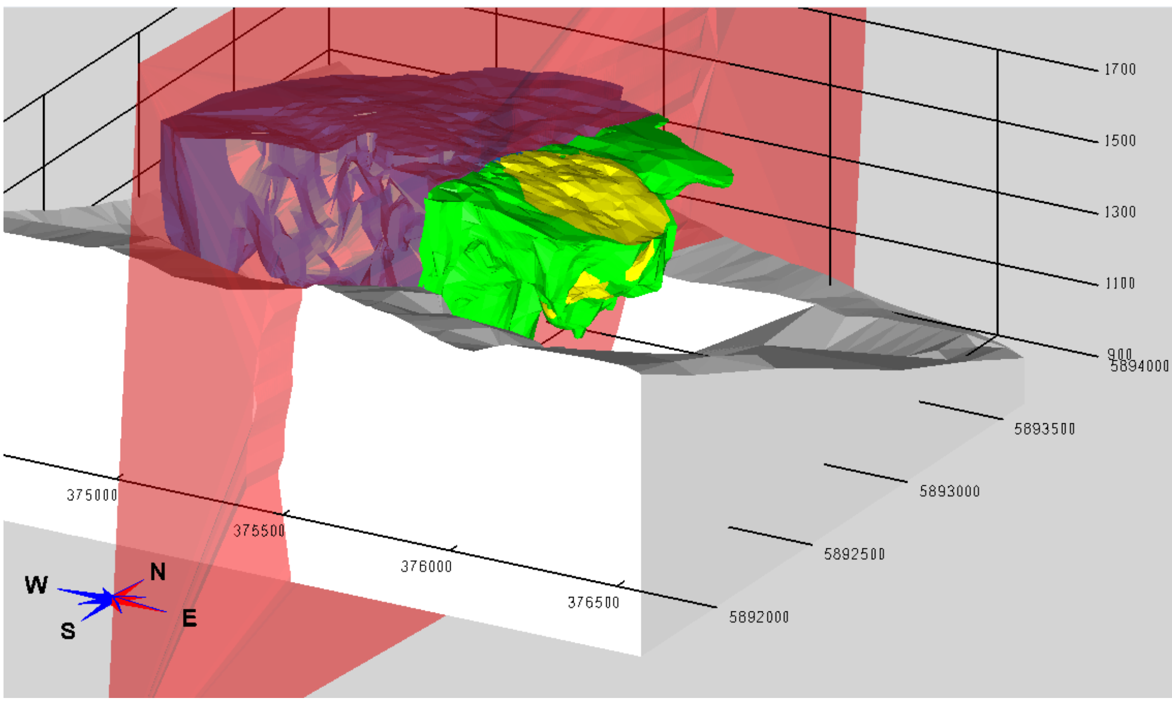

Figure 14-5 Sediments and Alteration Solids used for SG (white=sediments, yellow=central sericite, green=east sericite, blue=west sericite, red=fault; source: MMTS, 2020)

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

14.7.1 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

						Ranges - Spherical1			Ranges - Spherical2		
Dom	Ind	Cut-off	C0	C1	C2	Y ("Major")	X ("Minor")	Z ("Vert")	Y ("Major")	X ("Minor")	Z ("Vert")
1	1	0.003	0.2	0.35	0.45	75	50	40	720	300	270
	2	0.146	0.2	0.35	0.45	75	50	40	720	300	270
	3	0.323	0.2	0.35	0.45	75	50	40	720	300	270
	4	0.522	0.25	0.35	0.4	60	50	40	650	280	270
	5	0.765	0.3	0.4	0.3	60	50	40	650	280	220
	6	1.075	0.3	0.4	0.3	40	30	30	500	250	220
	7	1.496	0.55	0.25	0.2	40	30	20	400	220	180
	8	2.133	0.7	0.2	0.1	20	20	10	250	180	150
	9	3.786	0.9	0.1	0	30	10	5			
2	1	0.003	0.3	0.4	0.3	50	60	40	200	230	250
	2	0.257	0.3	0.4	0.3	50	60	40	200	230	250
	3	0.478	0.5	0.3	0.2	50	60	40	200	230	250
	4	0.765	0.5	0.3	0.2	30	60	40	160	200	250
	5	1.185	0.5	0.3	0.2	20	20	25	120	130	150
	6	1.893	0.7	0.2	0.1	30	30	35	120	130	140
	7	3.350	0.8	0.15	0.05	50	30	50	90	100	110
	8	6.241	0.9	0.1	0	45	25	60			
	9	14.244	0.95	0.05	0	30	25	60			
3	1	0.003	0.3	0.4	0.3	50	60	40	200	230	250
	2	0.212	0.3	0.4	0.3	50	60	40	200	230	250
	3	0.433	0.5	0.3	0.2	50	60	40	200	230	250
	4	0.766	0.5	0.3	0.2	30	60	40	160	200	250
	5	1.296	0.5	0.3	0.2	20	20	25	120	130	150
	6	2.244	0.7	0.2	0.1	30	30	35	120	130	140
	7	4.272	0.8	0.15	0.05	50	30	50	90	100	110
	8	9.143	0.9	0.1	0	45	25	60			
	9	25.165	0.9	0.1	0	45	25	60			

Table 14-9 Summary of Correlogram Parameters for Au and Ag

					Ranges - Spherical1			Ranges - Spherical2			Ranges - Spherical3		
Dom	**C0**	**C1**	**C2**	**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	250	220	200
2	0.3	0.6	0.1		30	30	20	180	150	220			
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

14.8 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 Measured was created with all blocks inside the shape given a final classification of Measured, and blocks outside the shape given Indicated or Inferred based on the distance criteria in Table 14-1.

Figure 14-6, which is a north–south section through the center of the deposit and Figure 14-7, which is a 3-dimensional (3D) image illustrate the final block classification 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.

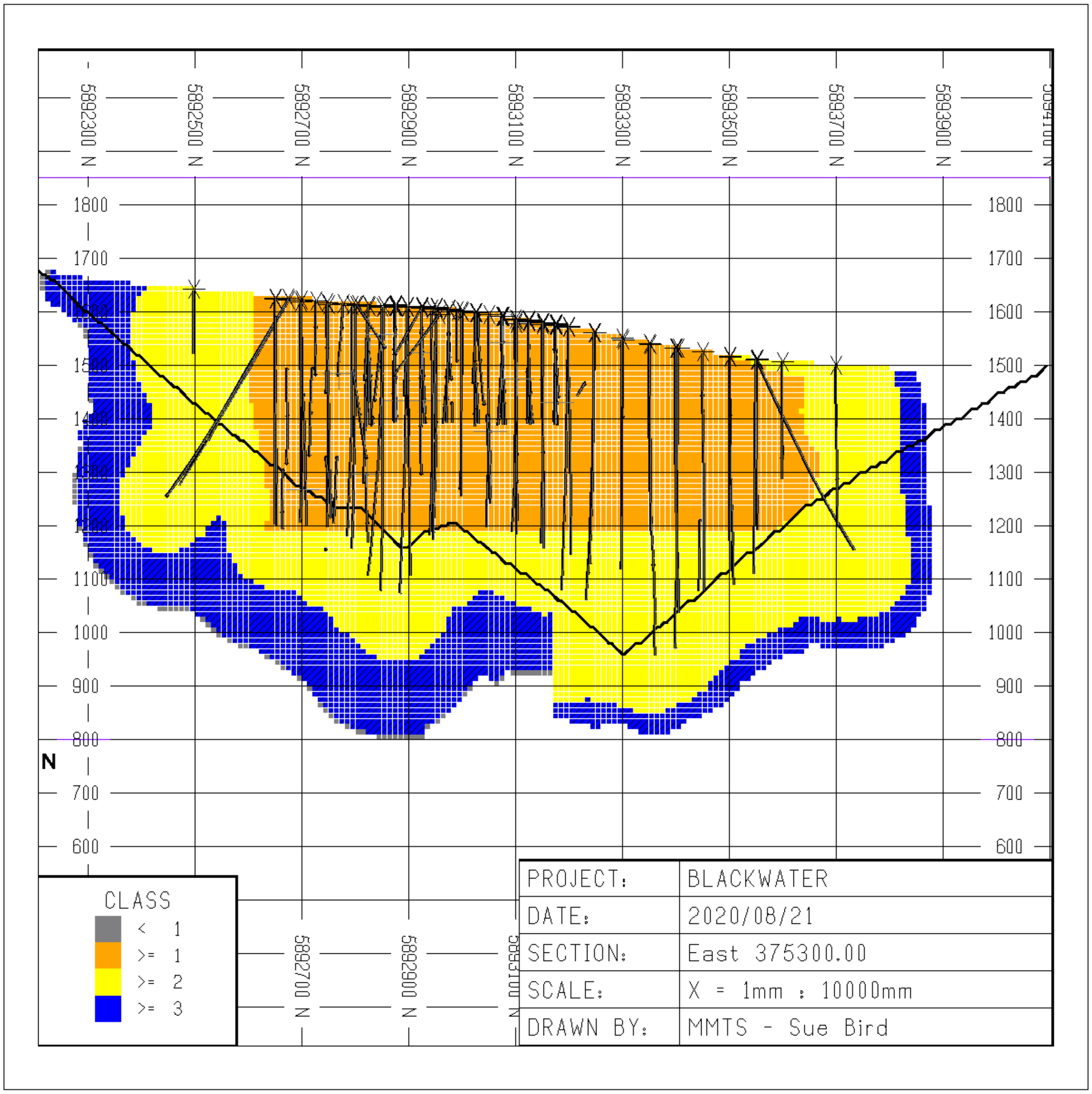

Figure 14-6 Illustration of Classification, DH Density and Resource Pit – 375300N (source: MMTS, 2020)

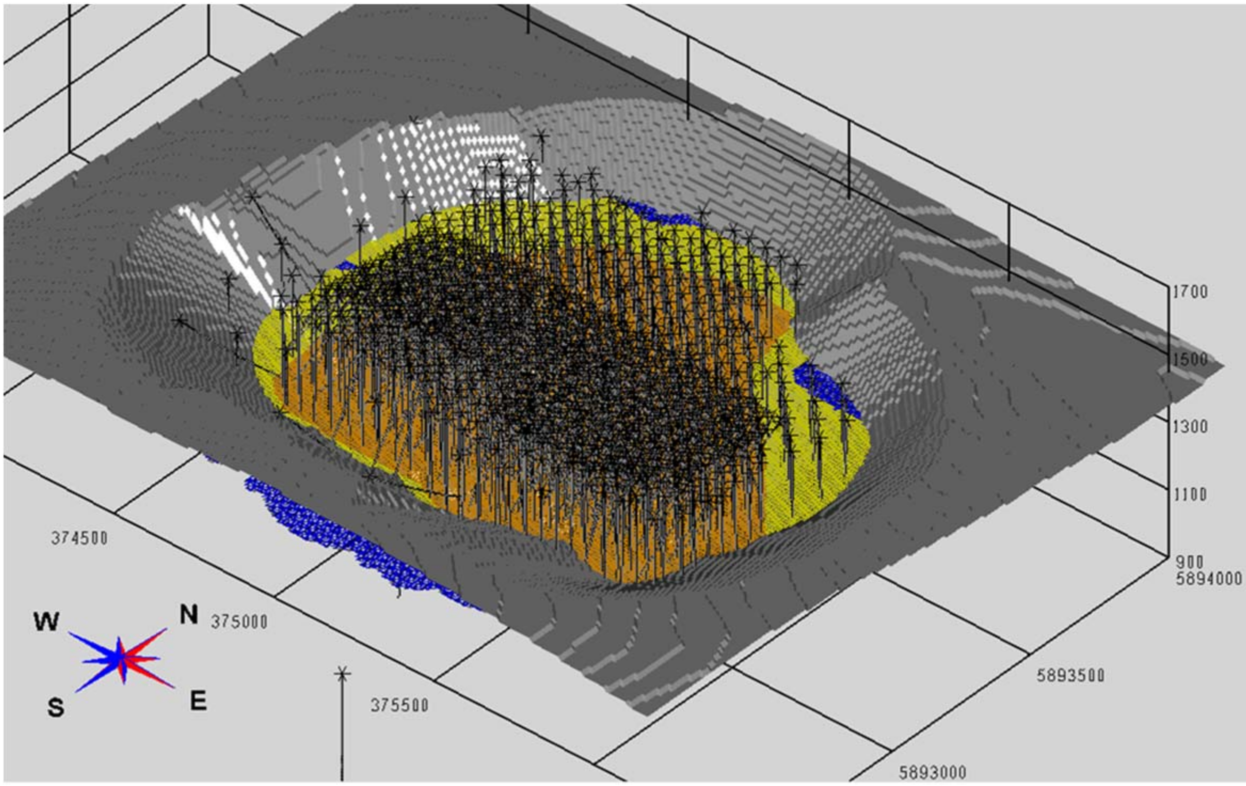

Figure 14-7 Three-dimensional View of the Classification at elev.=1350 (mid pit), the Drill Pattern, and the Resource Pit (source, MMTS, 2020)

14.9 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.

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

14.9.2 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.

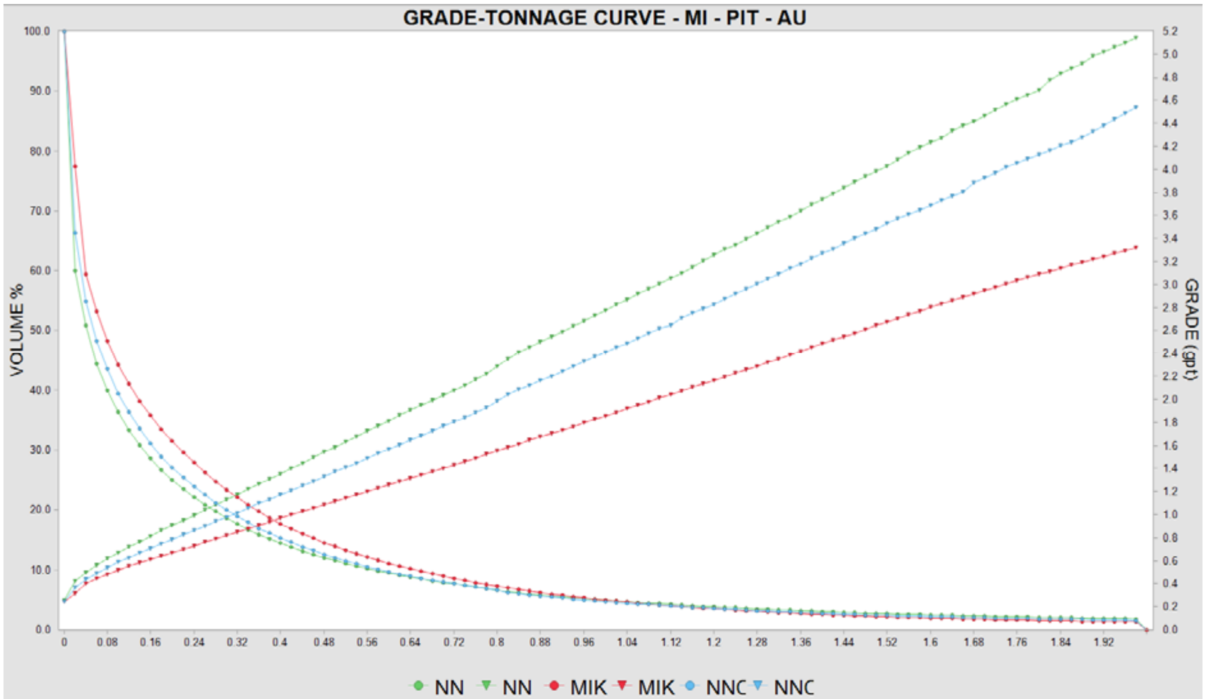

Figure 14-8 Grade-Tonnage Curve Comparison for Au – MI within the Resource Pit (source MMTS, 2020)

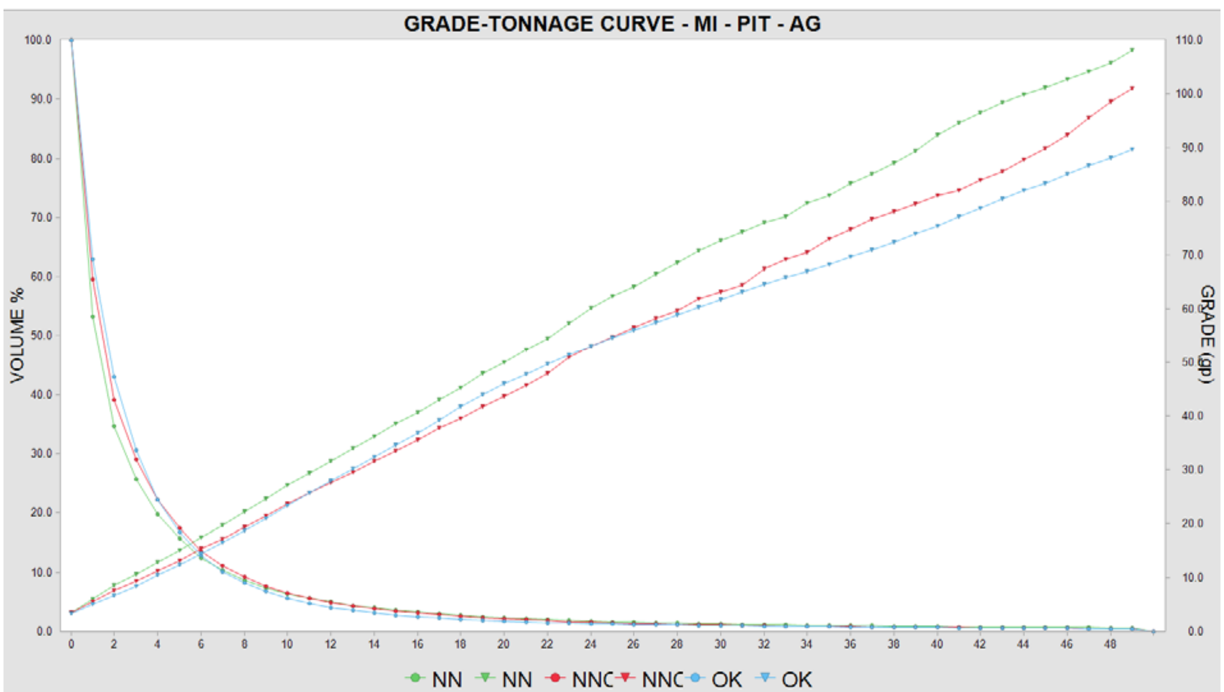

Figure 14-9 Grade-Tonnage Curve Comparison for Ag – MI within the Resource Pit (source MMTS, 2020)

14.9.3 Visual Comparisons

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

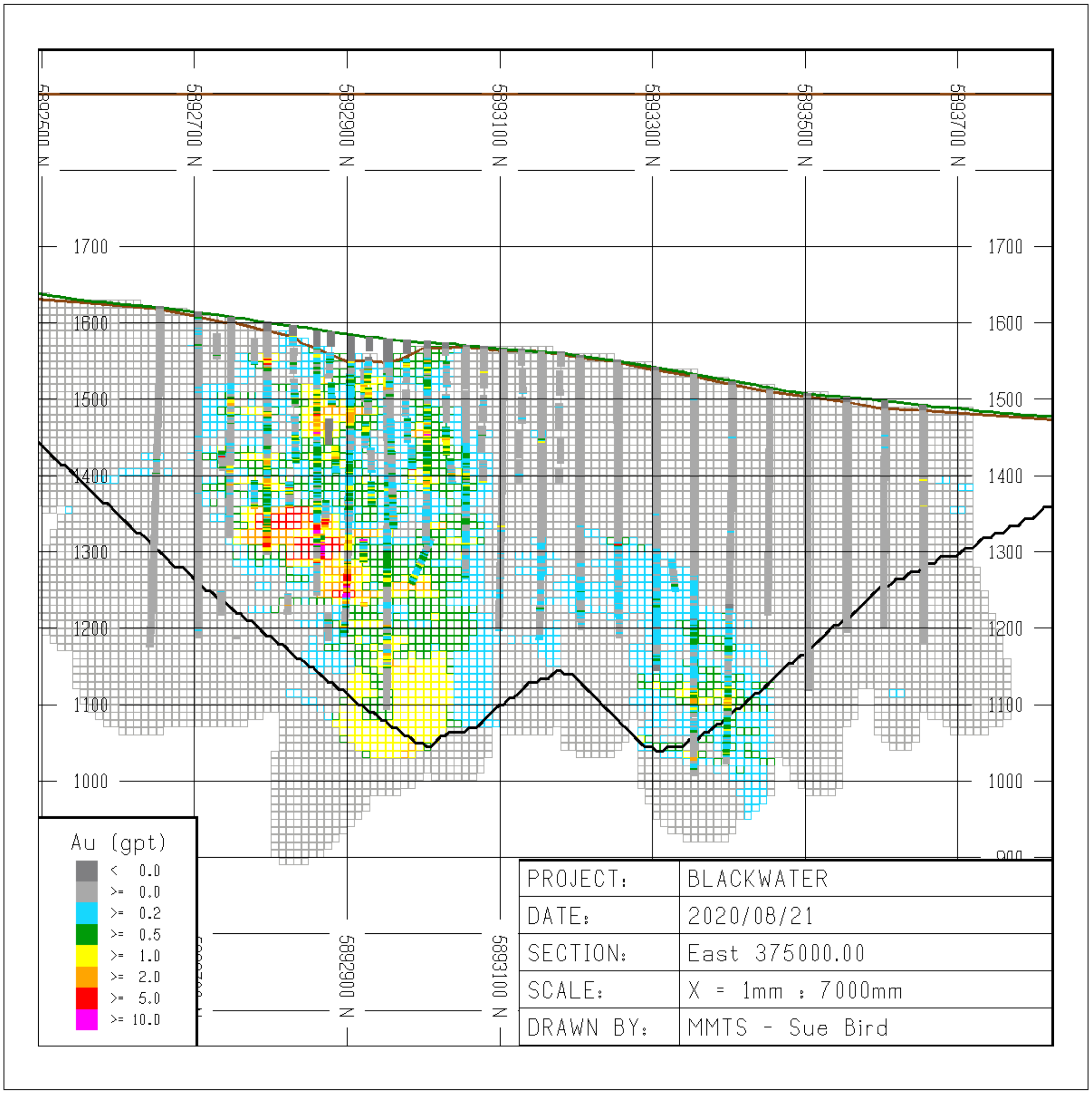

Figure 14-10 Au Grade - Model Compared to Composite – 375000E

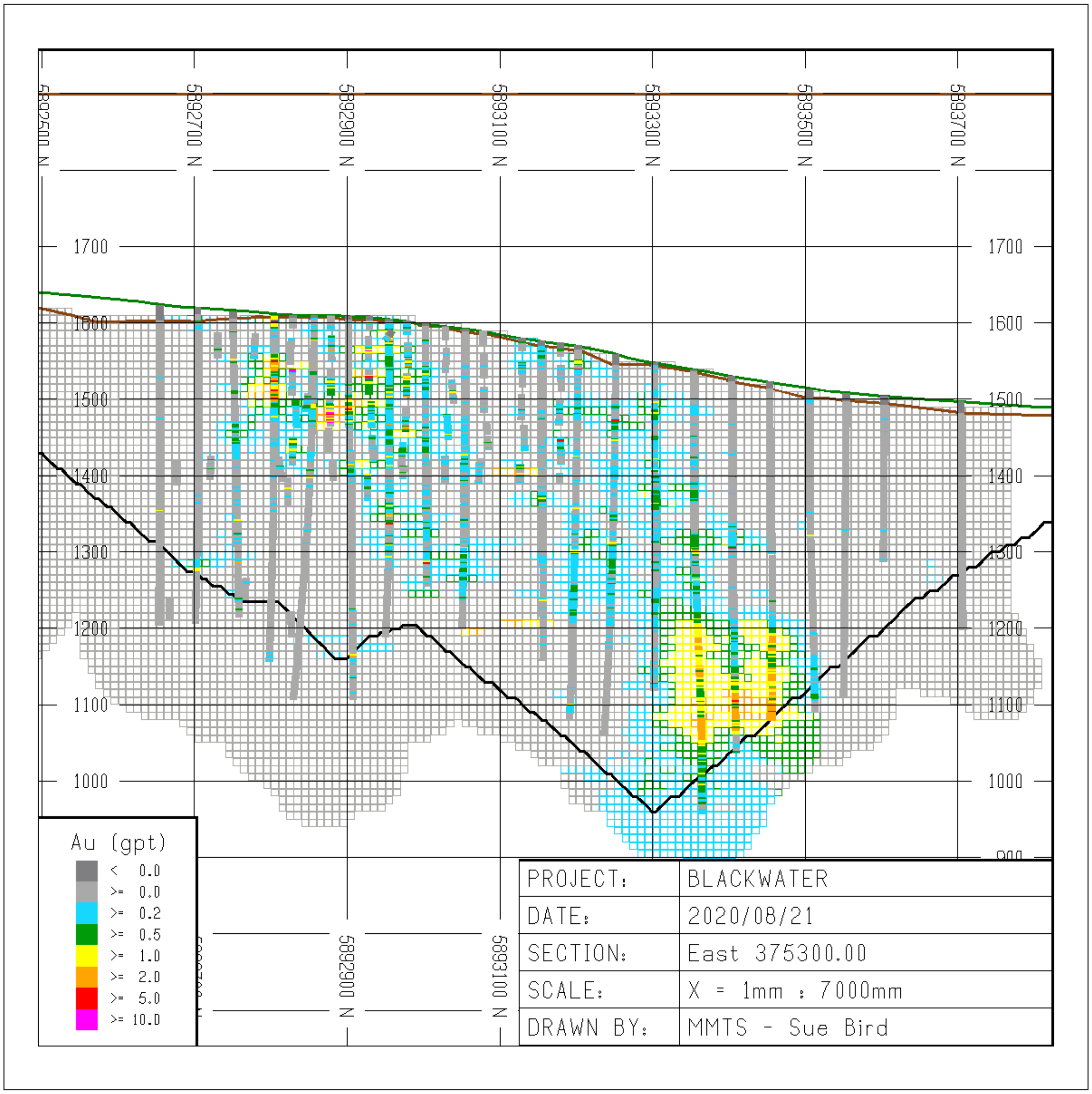

Figure 14-11 Au Grade - Model Compared to Composite – 375300E

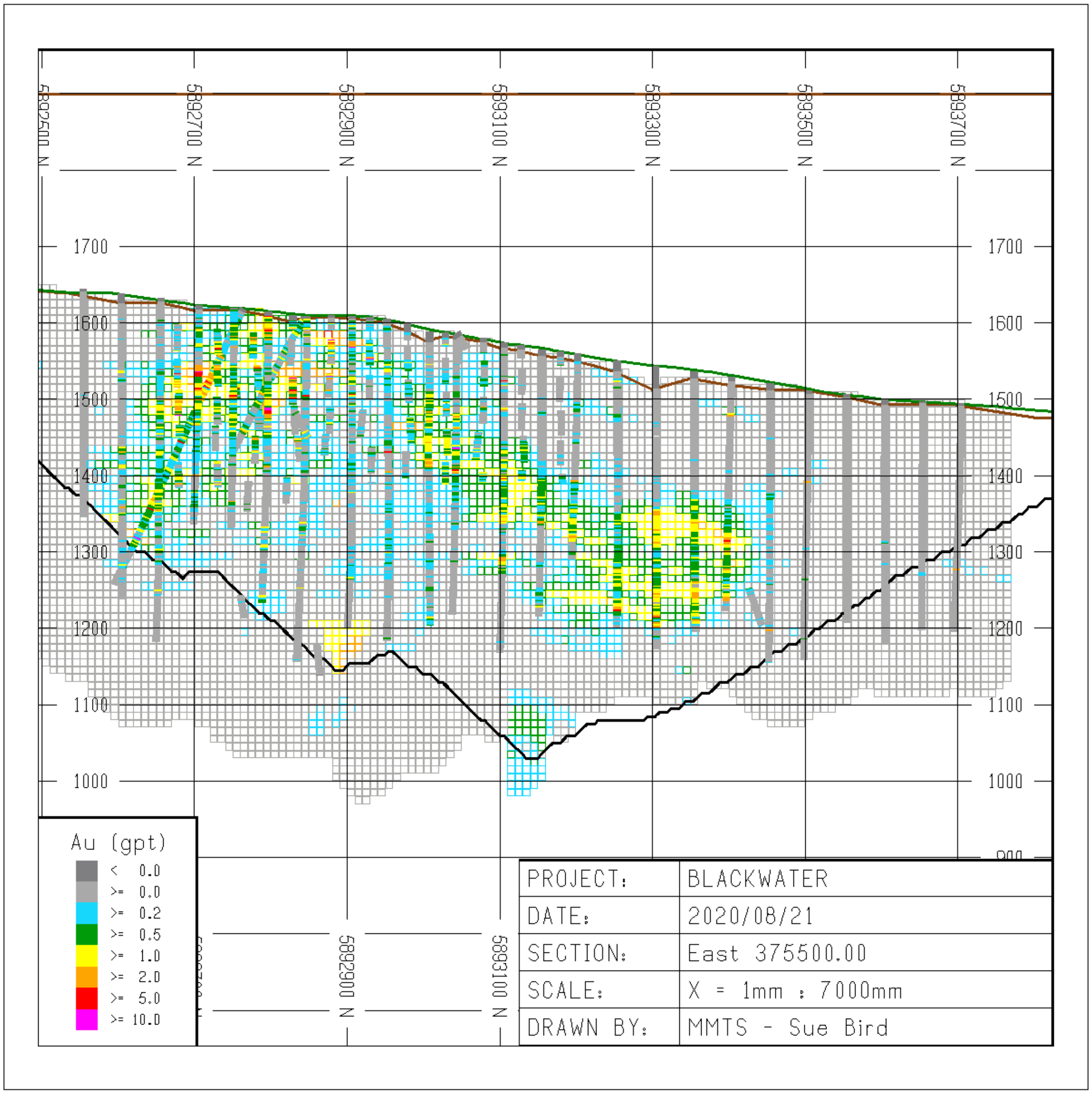

Figure 14-12 Au Grade - Model Compared to Composite – 375500E

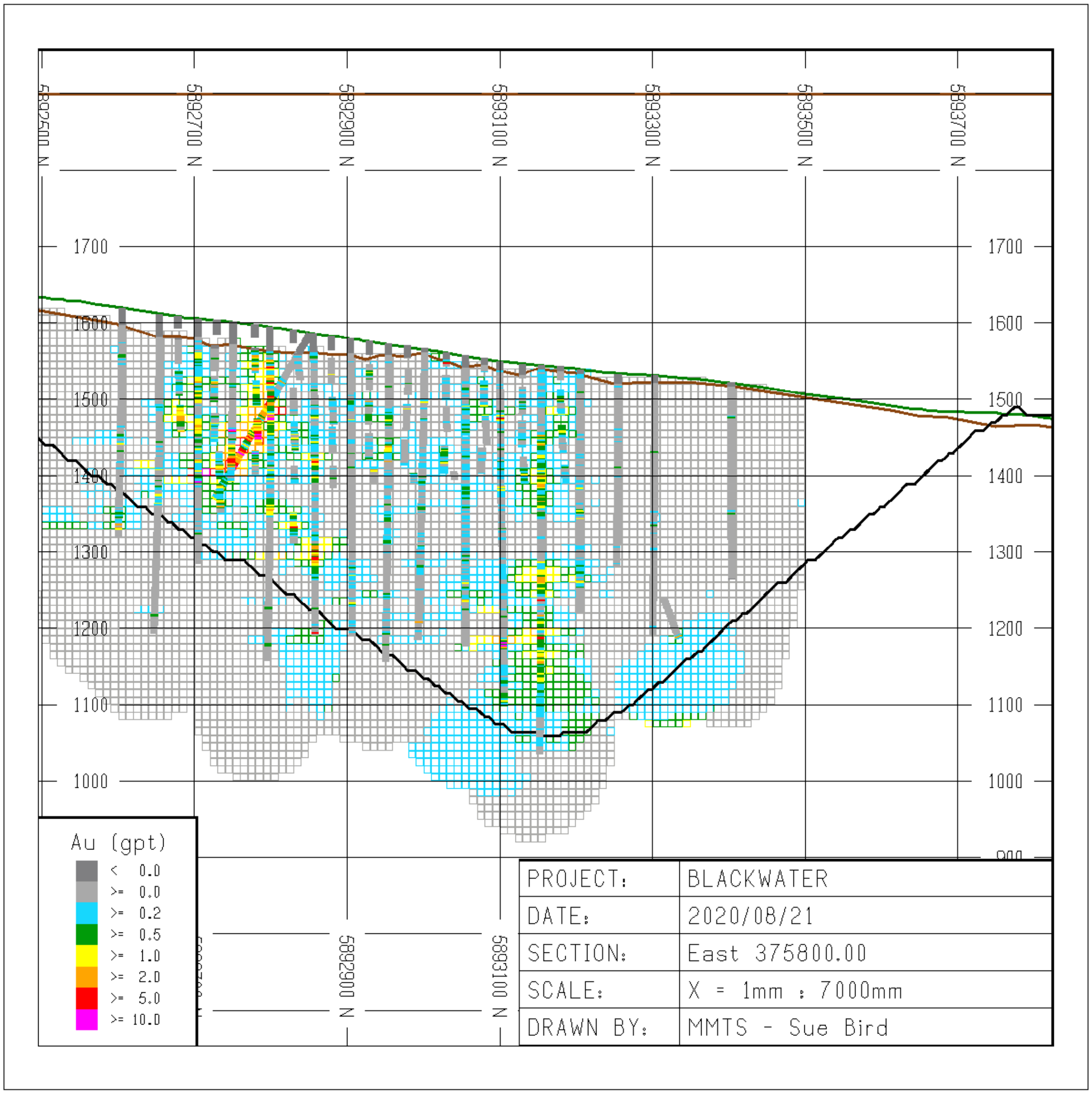

Figure 14-13 Au Grade - Model Compared to Composite – 375800E

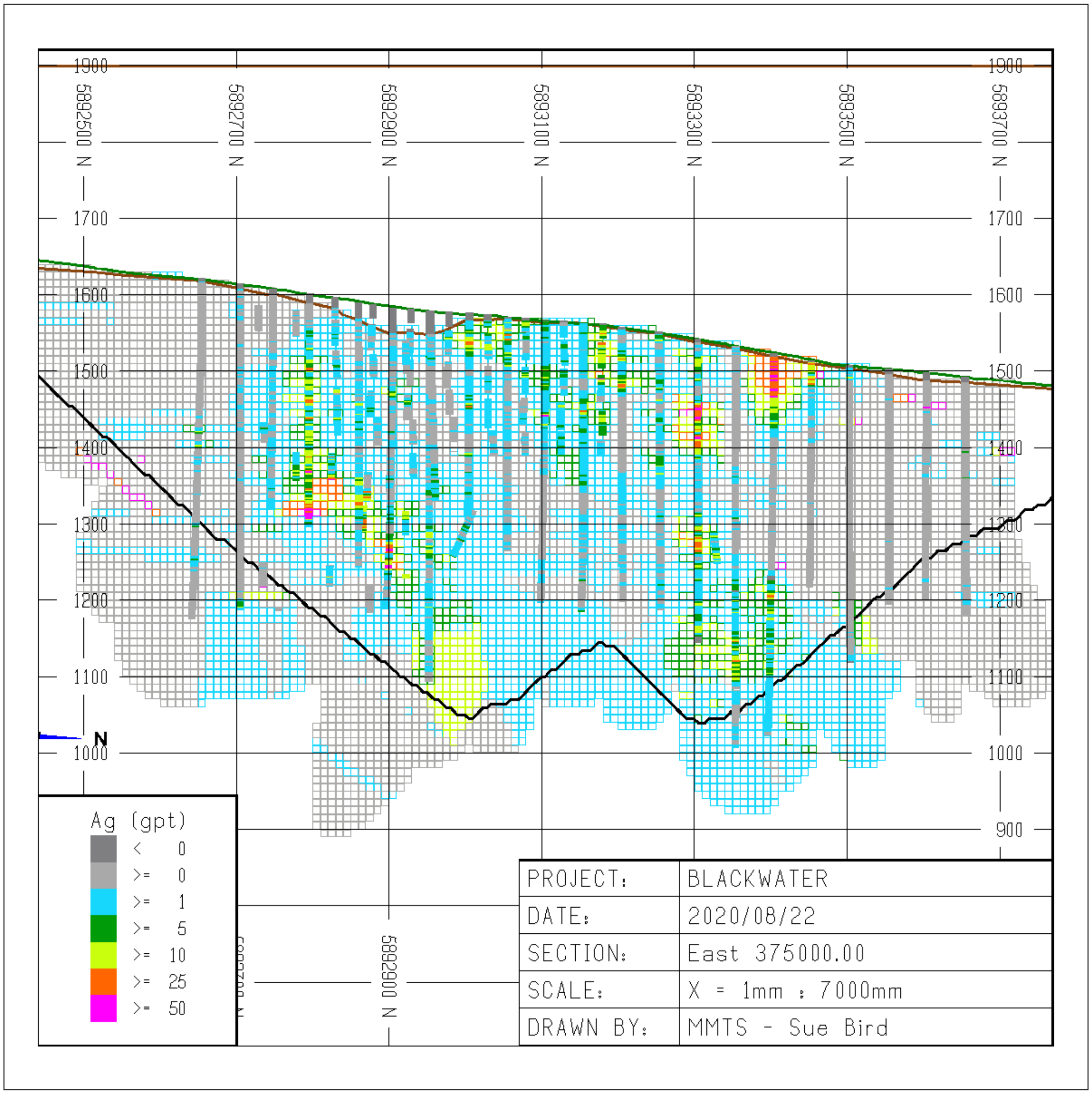

Figure 14-14 Ag Grade - Model Compared to Composite – 375000E

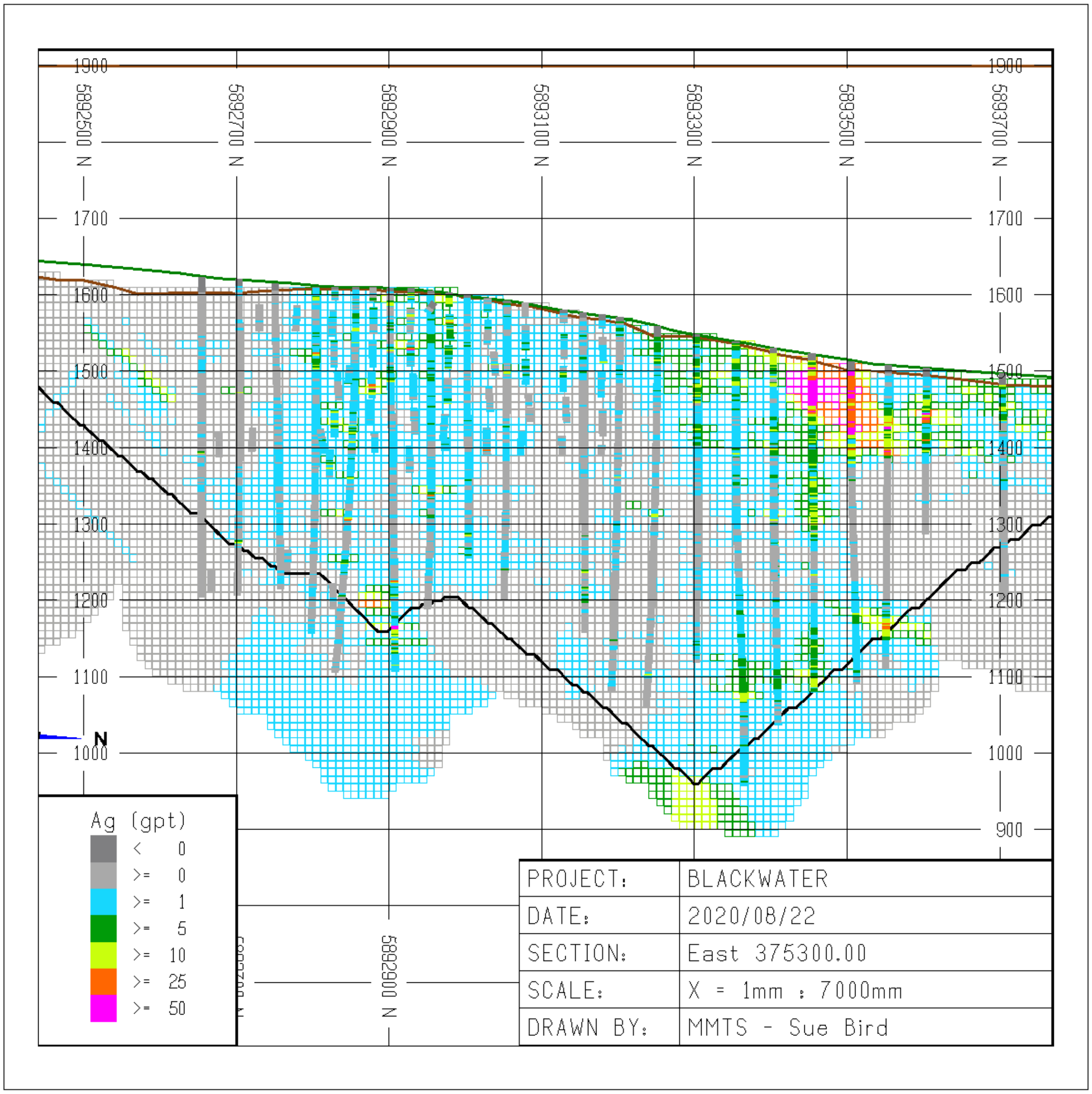

Figure 14-15 Ag Grade - Model Compared to Composite – 375300E

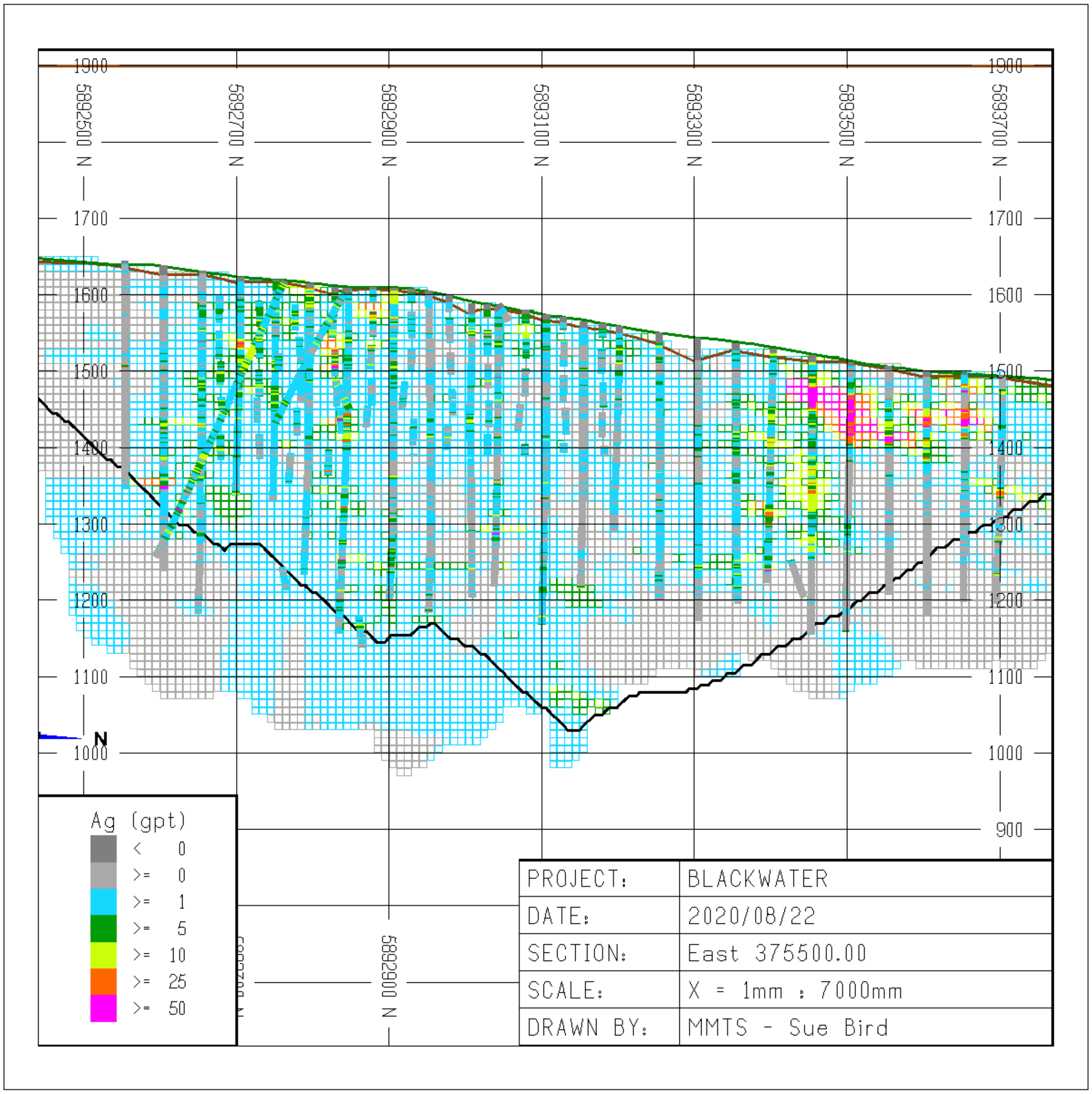

Figure 14-16 Ag Grade - Model Compared to Composite – 375500E

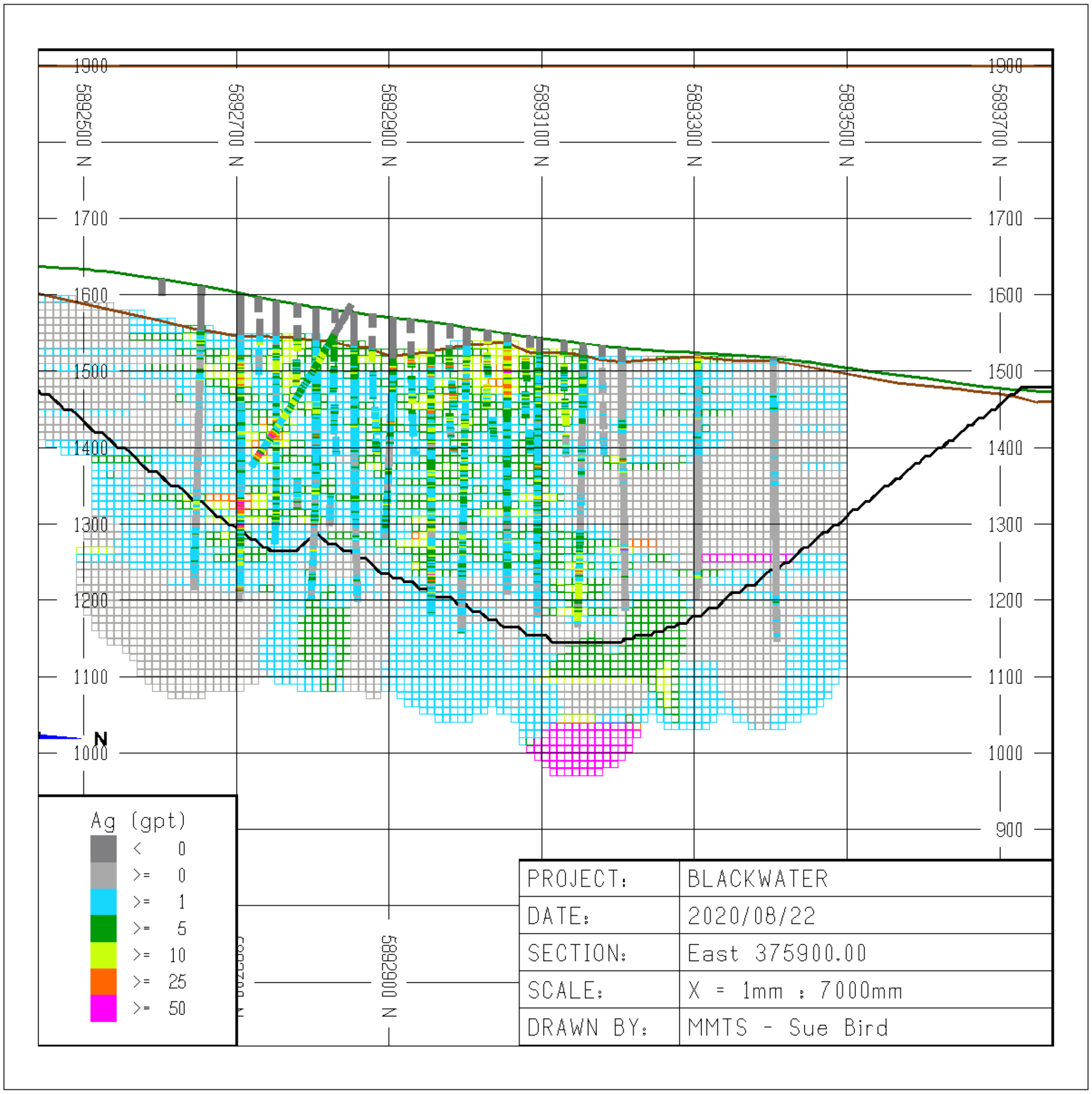

Figure 14-17 Ag Grade - Model Compared to Composite – 375900E

14.10 Reasonable Prospects of Eventual Economic Extraction

An open pit created using Lerchs–Grossmann (LG) pit optimization has been 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
Forex	0.75	($US:$CDN)
Gold Payable	99.9	%
Silver Payable	95.0	%
Gold Offsites	8.50	$/oz
Silver Offsites	0.25	$/oz
Royalty	1.5%	%
Net Smelter Gold Price	$58.79	$/g
Net Smelter Silver Price	$0.59	$/g
Gold Process Recovery	93	%
Silver Process recovery	55	%

The resulting NSR equation in Canadian dollars is:

$$NSR = (\$58.79 * AuGrade * 0.93) + (\$0.59 * AgGrade * 0.55)$$

And the resulting gold equivalent (AuEq) grade is:

$$AuEq = Au + 0.006 * AgGrade$$

A gold price of US$1,450/oz and a processing cost of $12.00/t require an AuEq grade of 0.20 g/t. Therefore, a cut-off of 0.20g/t AuEq is considered appropriate for current gold prices. The final resource pit has been based on the LG pit at $US2,000/oz Au and $US21.43/oz Ag 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.

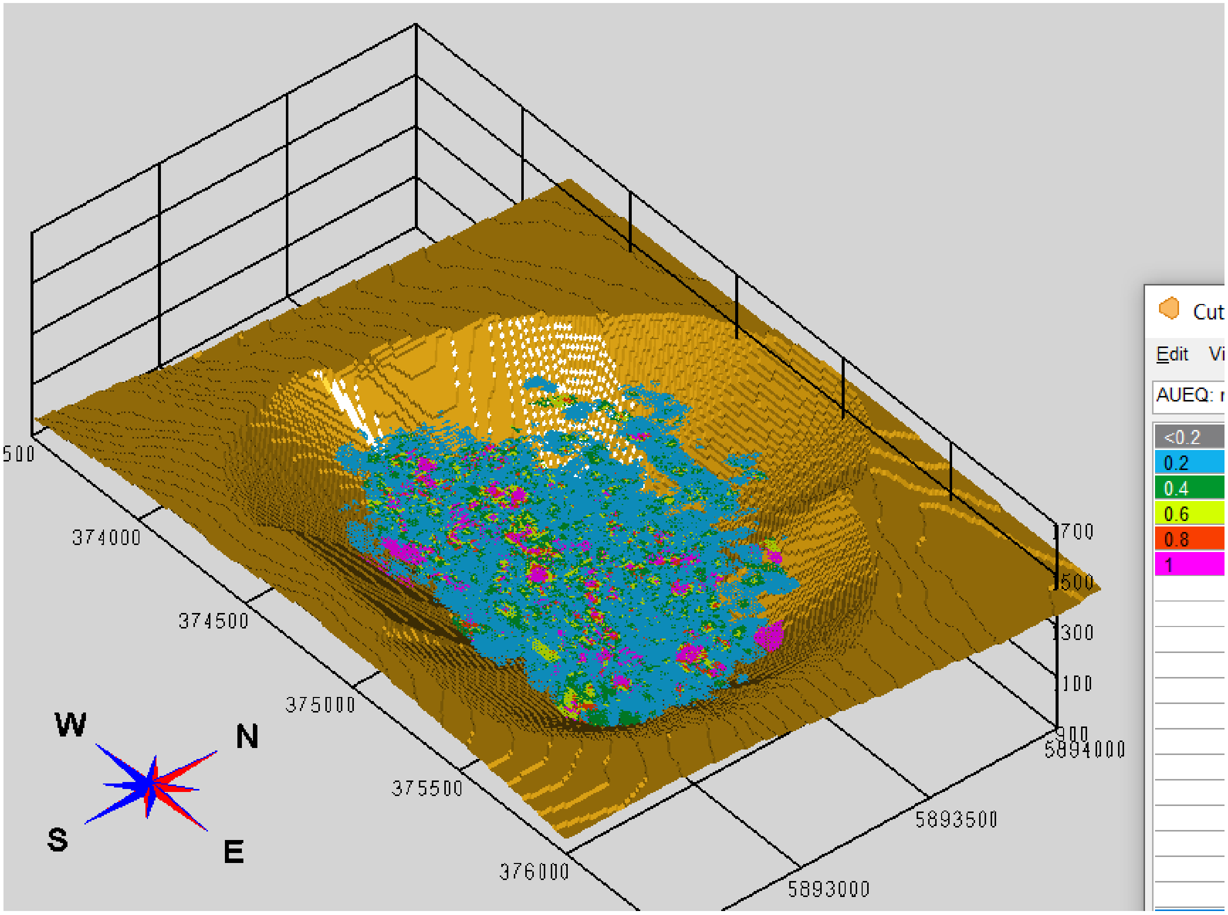

Figure 14-18 Three dimension View of the Resource Pit and AuEq blocks above 0.2g/t AuEq (source: MMTS, 2020)

For the LG pit optimizations the costs given in Table 14-4 were used. Constant pit slopes at 40º were used for the resource pit.

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

Cost	Value	Units
Mineralized Mining Costs	$2.50	/tonne
Waste Mining Costs	$2.30	/tonne
*Bench Incremental Mining Costs	$0.025	/tonne
*starting at 1,500 m: $0.025/t is added for every 10 m elevation drop		
Processing Costs	$12.00	/tonne of mineralization
G&A Costs	$4.50	/tonne of mineralization

14.11 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.

14.12 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	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 2020 PFS open pit life of mine 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	325.0	0.74	7.8	5.8	60.5	0.78
Probable	9.1	0.80	0.2	5.5	1.6	0.84
Total Reserve	**334.0**	**0.75**	**8.0**	**5.8**	**62.1**	**0.78**

Footnotes:

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 August 18, 2020.*
2) *Mineral Reserves are based on the 2020 Pre-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.*
4) *Mineral Reserves are reported at an NSR cut-off of $13.00/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; $8.50/oz Au and $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 NSR cut-off covers processing costs of $10.00/t and administrative (G&A) costs of $3.00/t.*
7) *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).*
8) *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	0.21	23.2	11.0	-
Starter Pit	P651i	24.8	1.06	6.1	23.0	0.9
East Pushback 1	P652i	16.3	0.78	5.9	12.5	0.8
East Pushback 2	P653i	33.6	0.94	7.1	55.4	1.6
West Pushback	P654i	37.3	0.93	4.6	64.7	1.7
North Pushback 1	P655i	53.3	0.65	3.8	61.7	1.2
North Pushback 2	P656i	83.9	0.58	8.2	190.3	2.3
South Pushback	P657i	84.8	0.71	4.6	248.3	2.9
Total Open Pit	**P657**	334.0	**0.75**	**5.8**	667.1	**2.0**

***Footnotes**:*

1. *An NSR cut-off of $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
- 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 Section 15 are supported by the open pit mine plan summarised in this section.

Open pit mine designs, mine production schedules and mine capital and operating costs were developed for the Blackwater deposit at a PFS 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 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$/$CDN
Payable Au	99.9%	
Payable Silver	95.0%	
Gold Offsite Costs	$8.50	$/oz.
Silver Offsite Costs	$0.25	$/oz.
Royalty	1.5%	

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

The NSR $/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. NSR in each block of the model is calculated using the following formula:

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

With the NSR value calculated for each mineralized block in the 3D block model, 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 positive cash flow if they have an insitu grade value 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, the block can contribute positively on an incremental basis to the cash flow.

An economic mine planning NSR cut-off grade of $13.00/t is chosen. This cut-off grade will cover the incremental production costs of processing and G&A.

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 Au grade compared to the de-clustered composite data. This comparison was done in Section 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 effects of dilution and loss from mining operations.

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 predominantly range from 5 to 20 m thick across the open pit. Surficial material thickness increases up to 110 m along the eastern side of the deposit;
- Broken zone – This domain is recognized as zones with RQD of <40% and was encountered in all deposit rock types;
- Competent rock – Defined as all zones with RQD >40%.

Slope stability analyses for each pit design sector found few geotechnical controls for the pit slope design, other than rock mass failure through the broken zone and associated slope depressurization requirements. The achievable slope geometry is 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.

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

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.

Plans for highwall depressurization are described in Section 16.8.1.

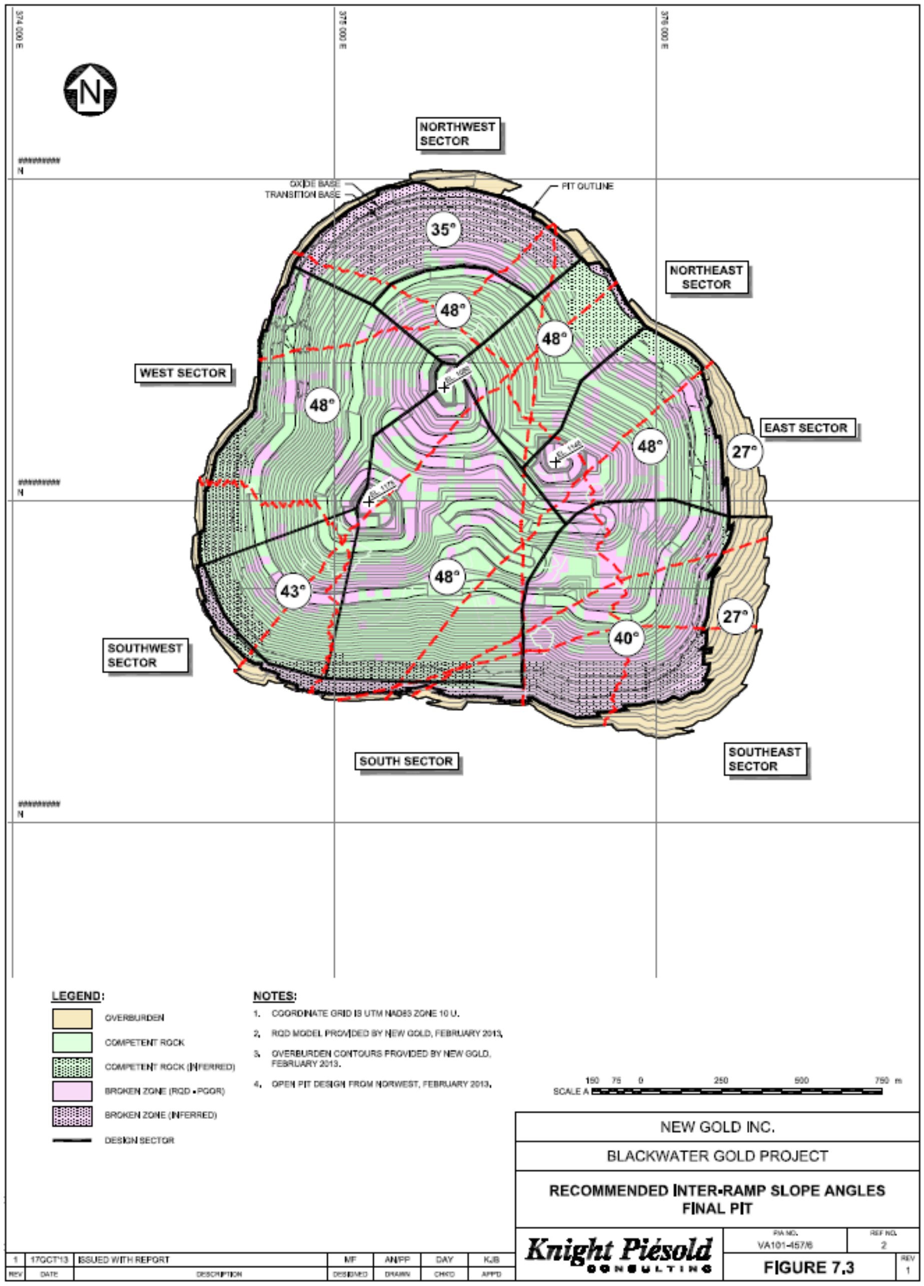

Figure 16-1 Open Pit Geotechnical Domains (source: KP, 2013b)

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 16-1 and Table 16-3. The UCFs for each case are compared to reinforce the selected point at which increased pit expansions do not 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 assumption and operating cost assumptions for the Pseudoflow runs are provided in Table 16-1 and Table 16-3 respectively.

Table 16-3 Inputs into Pseudoflow Pit Optimization

Item	Unit
Pit Rim ore Mining Cost	$2.50/t, pit rim of 1500 m
Pit Rim Waste Mining Cost	$2.30/t, pit rim of 1500 m
Incremental Bench Haulage Cost	$0.025 per every 10 m bench below pit rim
Processing Cost	$12.00/t
General/administration Cost	$4.50/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 25 to be a point at which larger pit shells will not produce significant increases to project value.

The pit shell generated from Case 25 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.

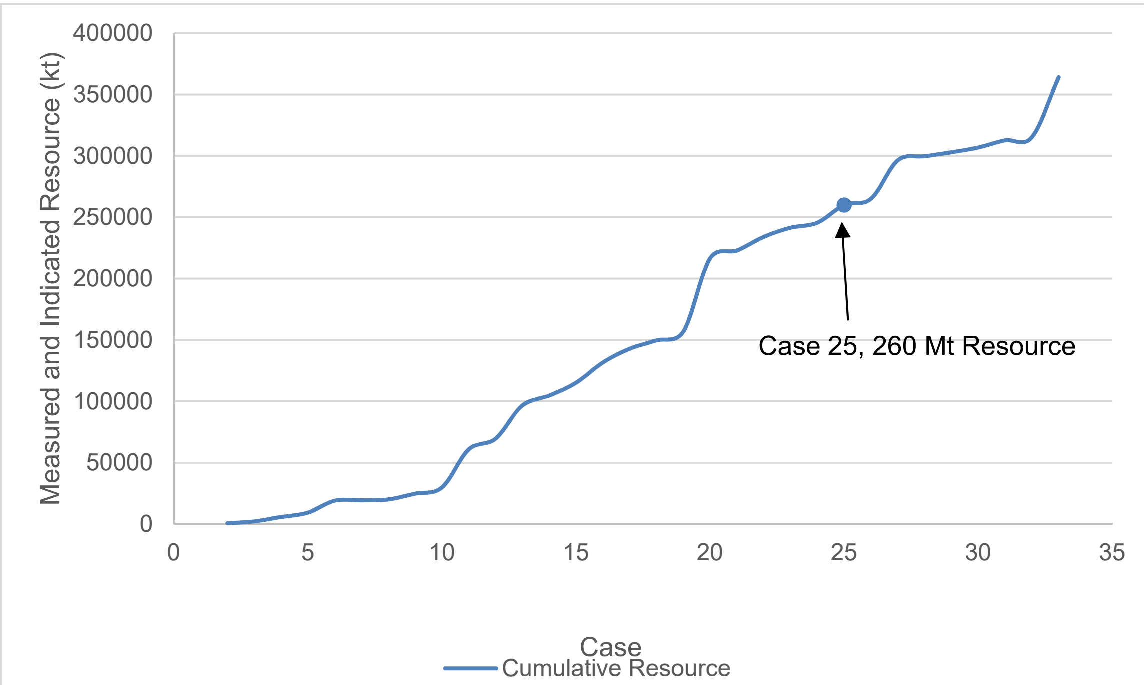

Figure 16-2 Pseudoflow Pit Shell Resource Contents by Case (source: MMTS, 2020)

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.

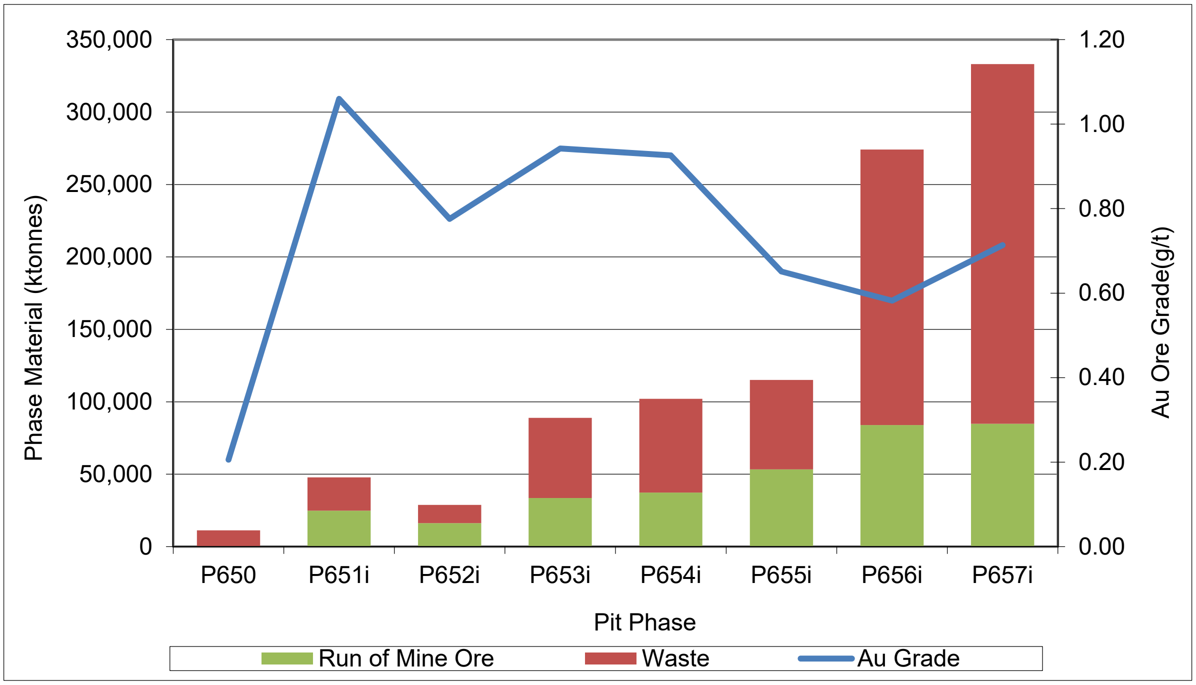

Figure 16-3 Designed Phase Pit Contents (source: MMTS, 2020)

16.3.1 In-Pit Haul Roads

Two-way in-pit haul roads of 29 m and 31 m widths are designed. The narrower roads are planned for the first five pit phases, as smaller haul trucks are planned to operate in those phases, and the wider roads are planned for the final three phases. 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 segment will be removed using some form of retreat mining. The bottom two ramped benches of the pit use one-way haul roads of 22 m width and 12% grade since bench volumes and traffic flow are reduced.

16.3.2 Pit Phasing

Ultimate pit limits are generally split up 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 50 m are honoured, with a vast majority of the bench pushbacks well over 100 m.

The Blackwater pit is split into eight phases. The first phase acts as a borrow pit targeting NAG waste rock, which is useful during the early year construction periods. The second phase targets higher-grade, lower-strip-ratio areas of the pit defined by Case 6 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 highest economic margin to lowest, targeting pit shells represented by inflection points on the curve shown in Figure 16-2.

16.3.3 Pit Designs

The phased Blackwater pit designs are discussed below and shown in Figure 16-4 to Figure 16-11. Sections through the deposit showing the whole block gold and silver grades are illustrated in Figure 16-12 to Figure 16-15.

- Construction Borrow Pit, P650 – This phase targets near surface non-acid generating (NAG) waste rock for project construction purposes, contained with three small pit areas within the ultimate pit limits. Some overburden soil and a small amount of potentially acid-generating (PAG) waste rock and resource is also contained within this initial phase. All three pits 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 1,450 m elevation. The ramp runs counter clockwise down from the pit exit at the 1,560 m elevation in the west. The eastern portion of the pit is accessed from the 1,580 m elevation down to the 1,520 m elevation from a secondary ramp running clockwise from the pit exit in the north. 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 1,640 m elevation, down to the pit bottom at the 1,460 m elevation. The main ramp runs clockwise from the pit exit at the 1,570 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 1,650 m elevation, down to the pit bottom at the 1,380 m elevation. The main ramp runs clockwise from the pit exit at the 1,550 m elevation in the northeast of the pit. 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 1,640 m elevation, down to the pit bottom on the 1,300 m elevation. The main ramp runs counter clockwise from the pit exit at the 1,550 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 1,560 m elevation, down to two pit bottoms on the 1,350 m and 1260 m elevation, bridging between the two bottoms on the 1,390 m bench. The main

ramp runs clockwise from the pit exit at the 1530 m elevation in the west of the pit, with switchbacks on the 1350 and 1320 m benches.

- North Pushback 2, P656 – This phase pushes out the pit to the ultimate limits in the north and east. The phase mines from the pit crest at the 1,610 m elevation, down to the pit bottom on the 1,140 m elevation. The main ramp runs clockwise from the pit exit at the 1,490 m elevation in the north of the pit. Upper benches of the pit will be accessed via ex-pit haul roads on the hillside. An extra wide geotechnical berm is left behind on the 1420 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.

- South Pushback, P657 – 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 1500 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.

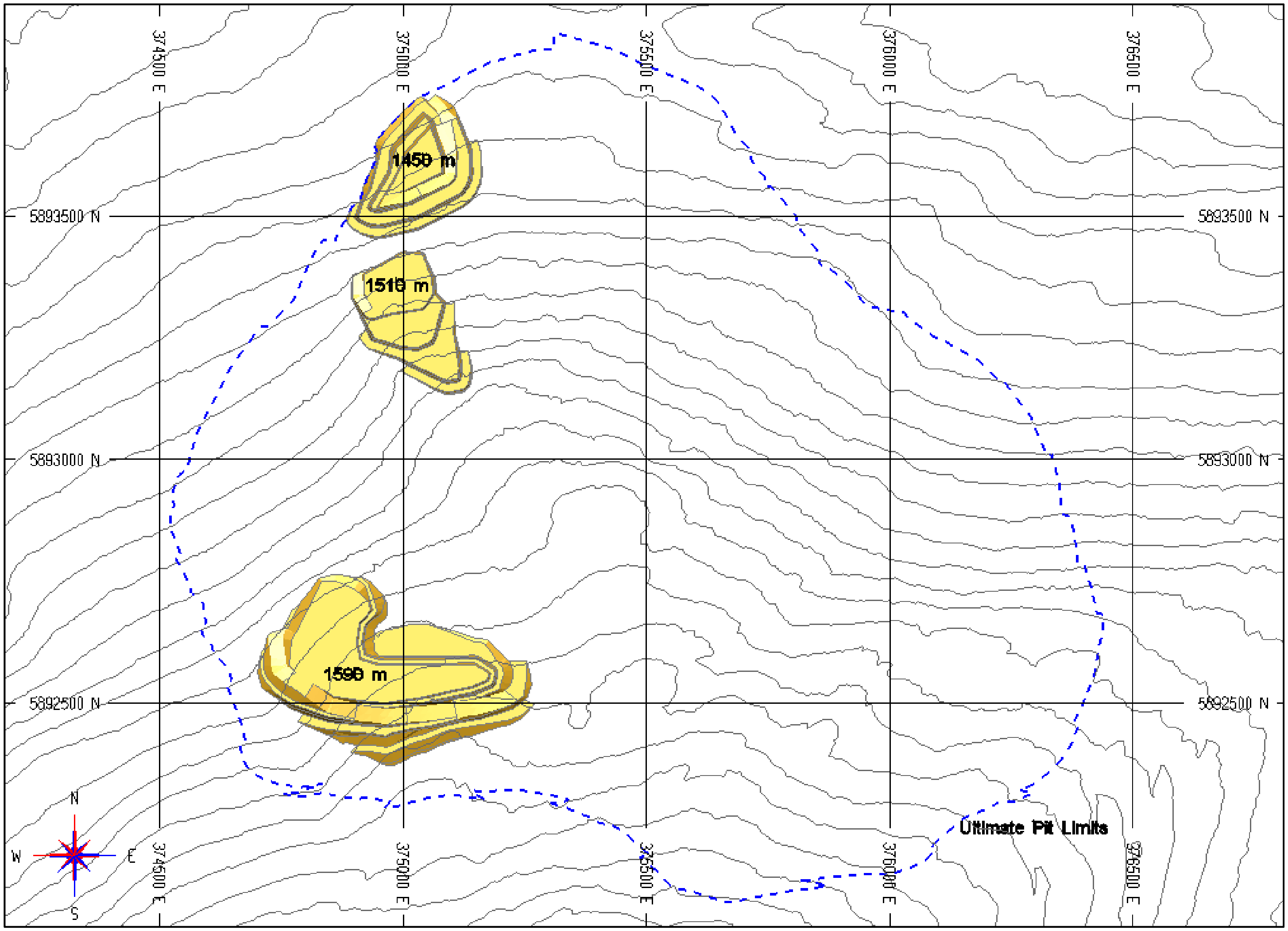

Figure 16-4 Construction Borrow Pit, P650 (source: MMTS, 2020)

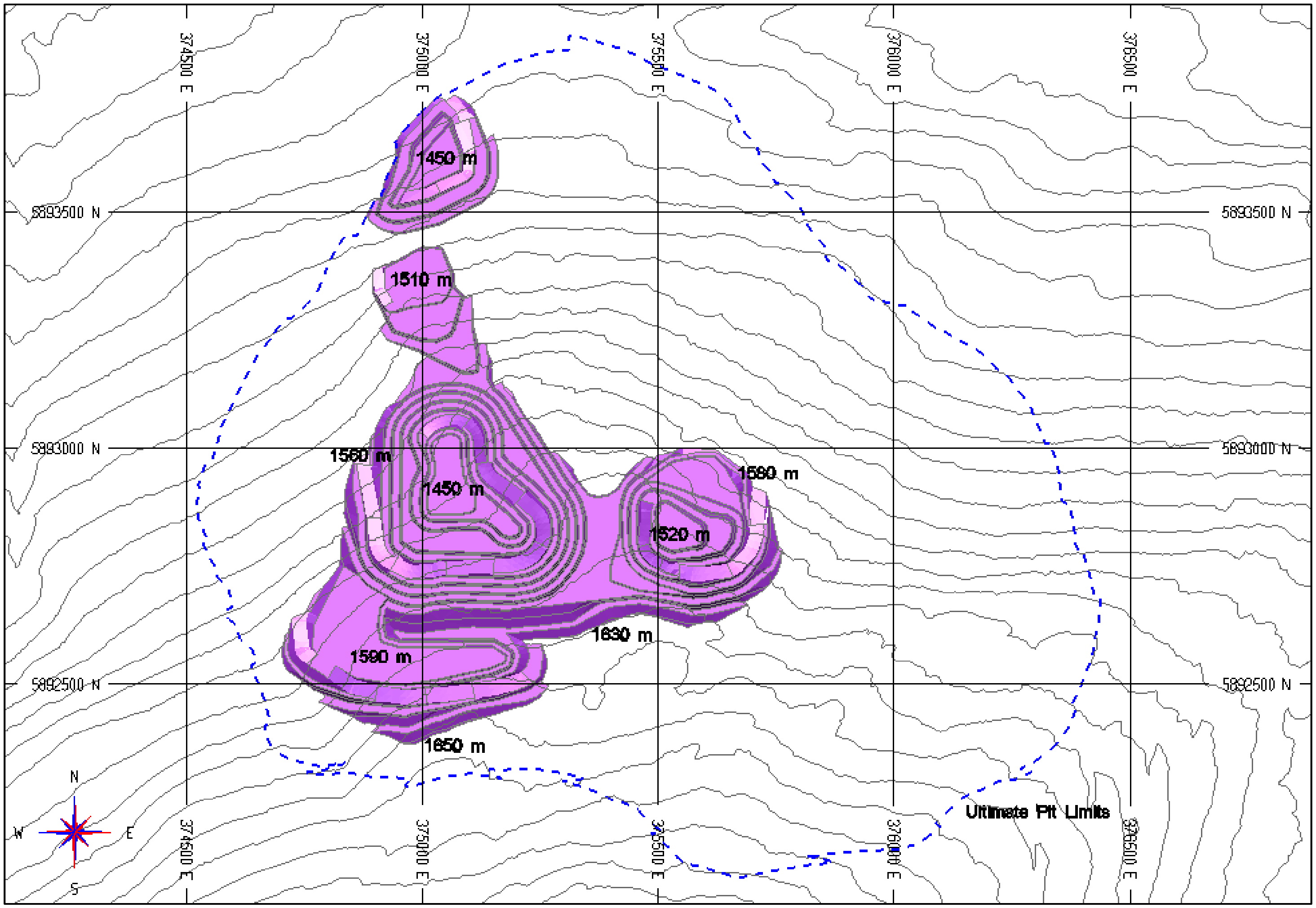

Figure 16-5 Starter Pit, P651 (source: MMTS, 2020)

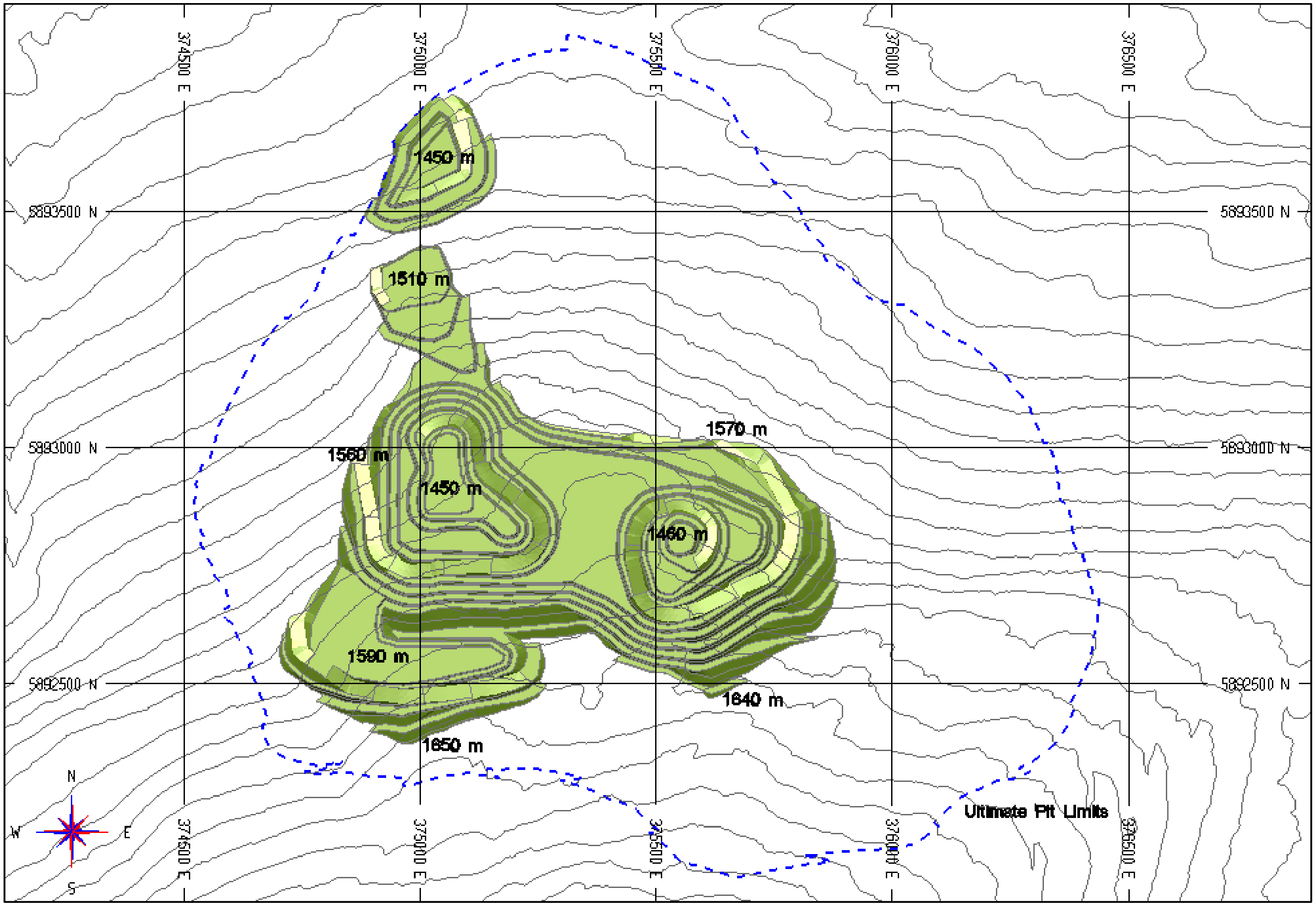

Figure 16-6 East Pushback 1, P652 (source: MMTS, 2020)

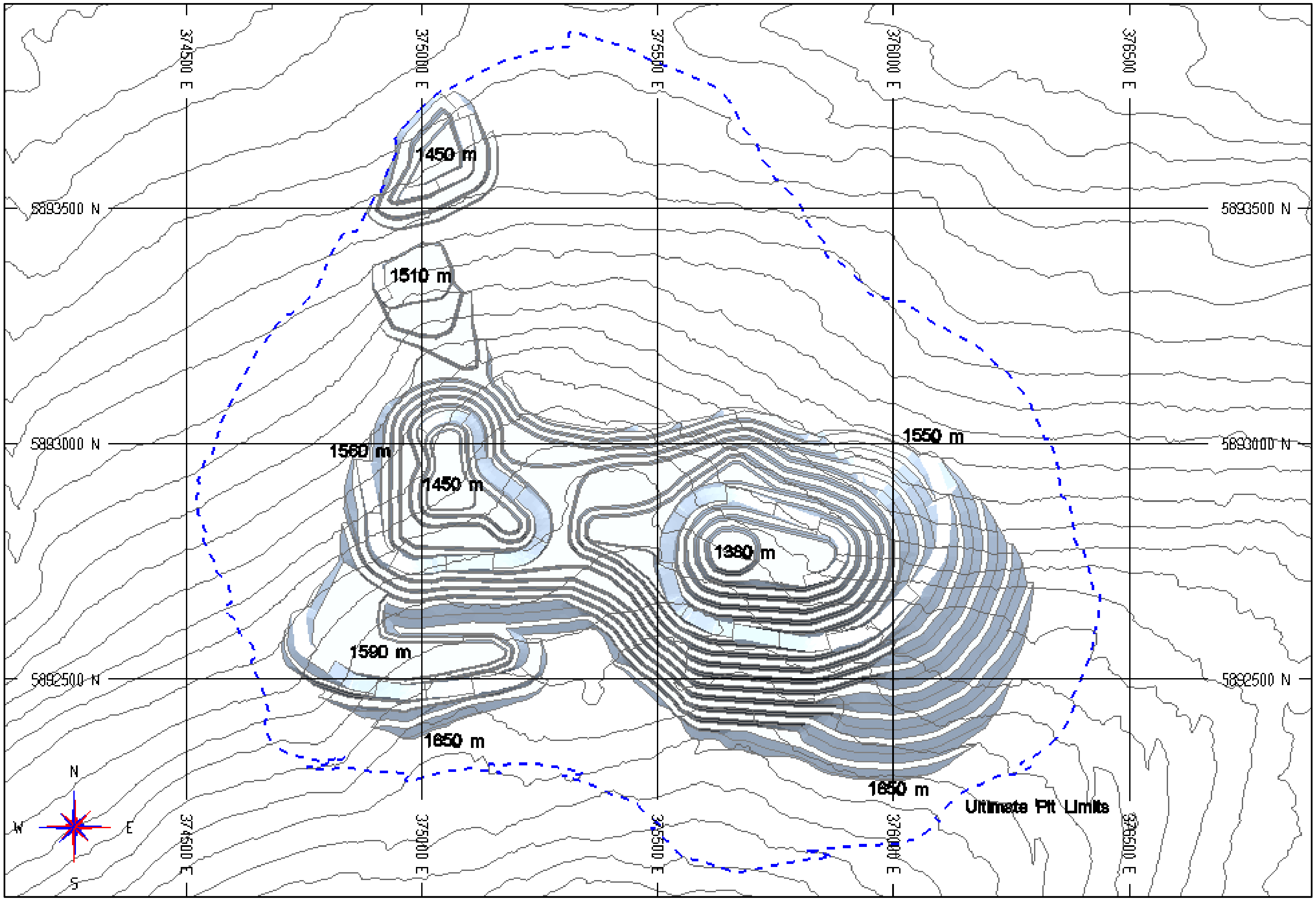

Figure 16-7 East Pushback 2, P653 (source: MMTS, 2020)

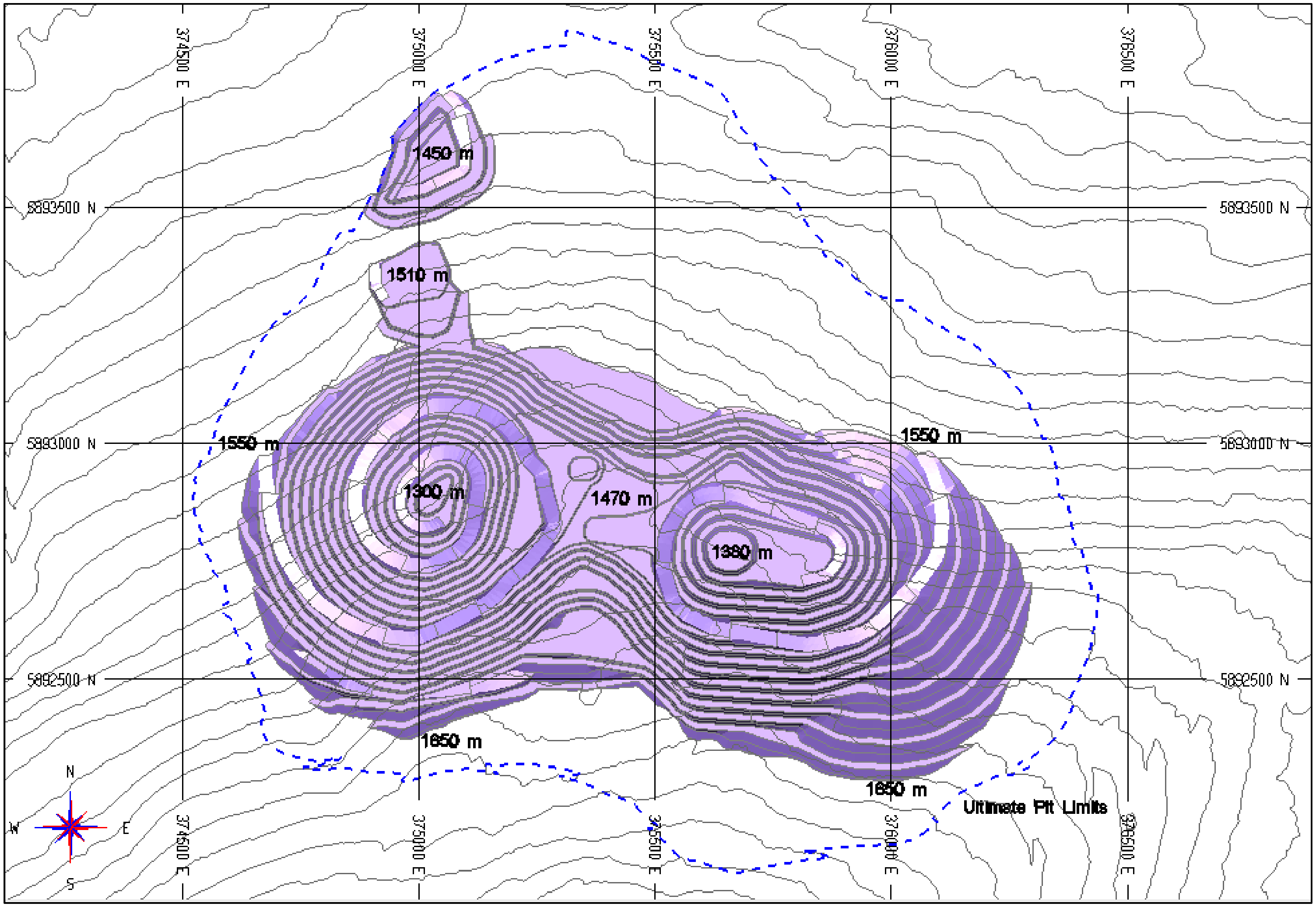

Figure 16-8 West Pushback, P654 (source: MMTS, 2020)

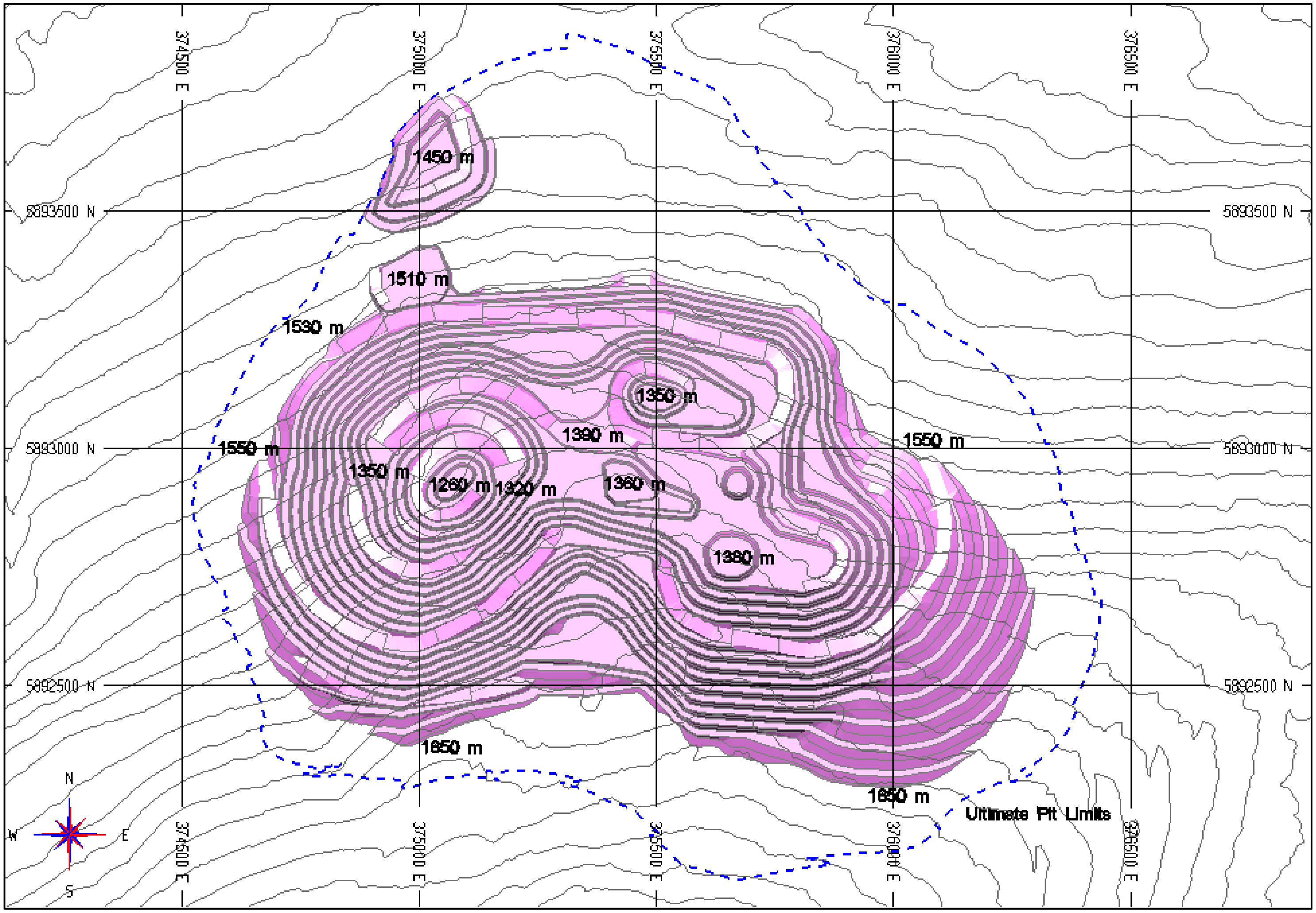

Figure 16-9 North Pushback, P655 (source: MMTS, 2020)

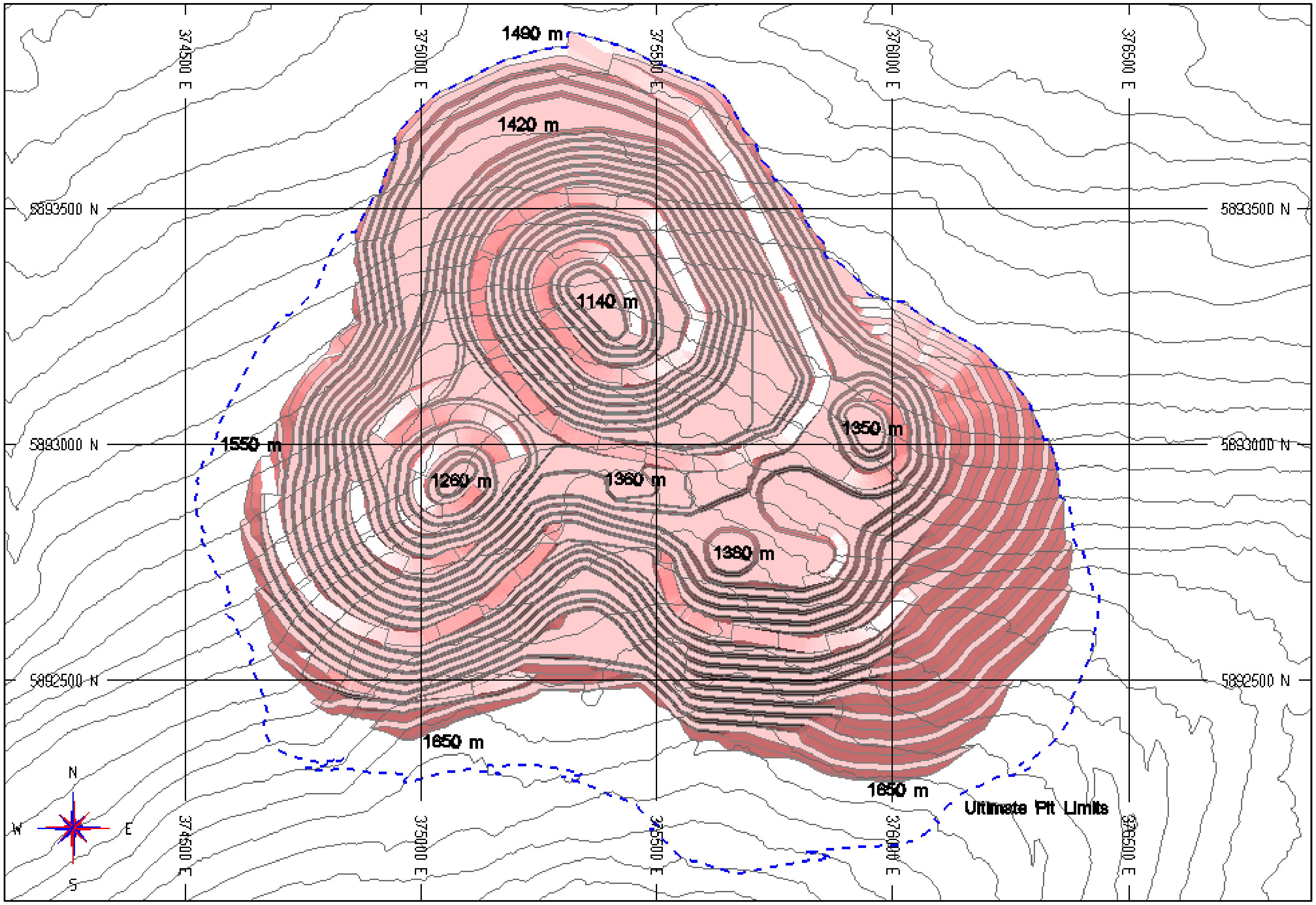

Figure 16-10 North Pushback 2, P656 (source: MMTS, 2020)

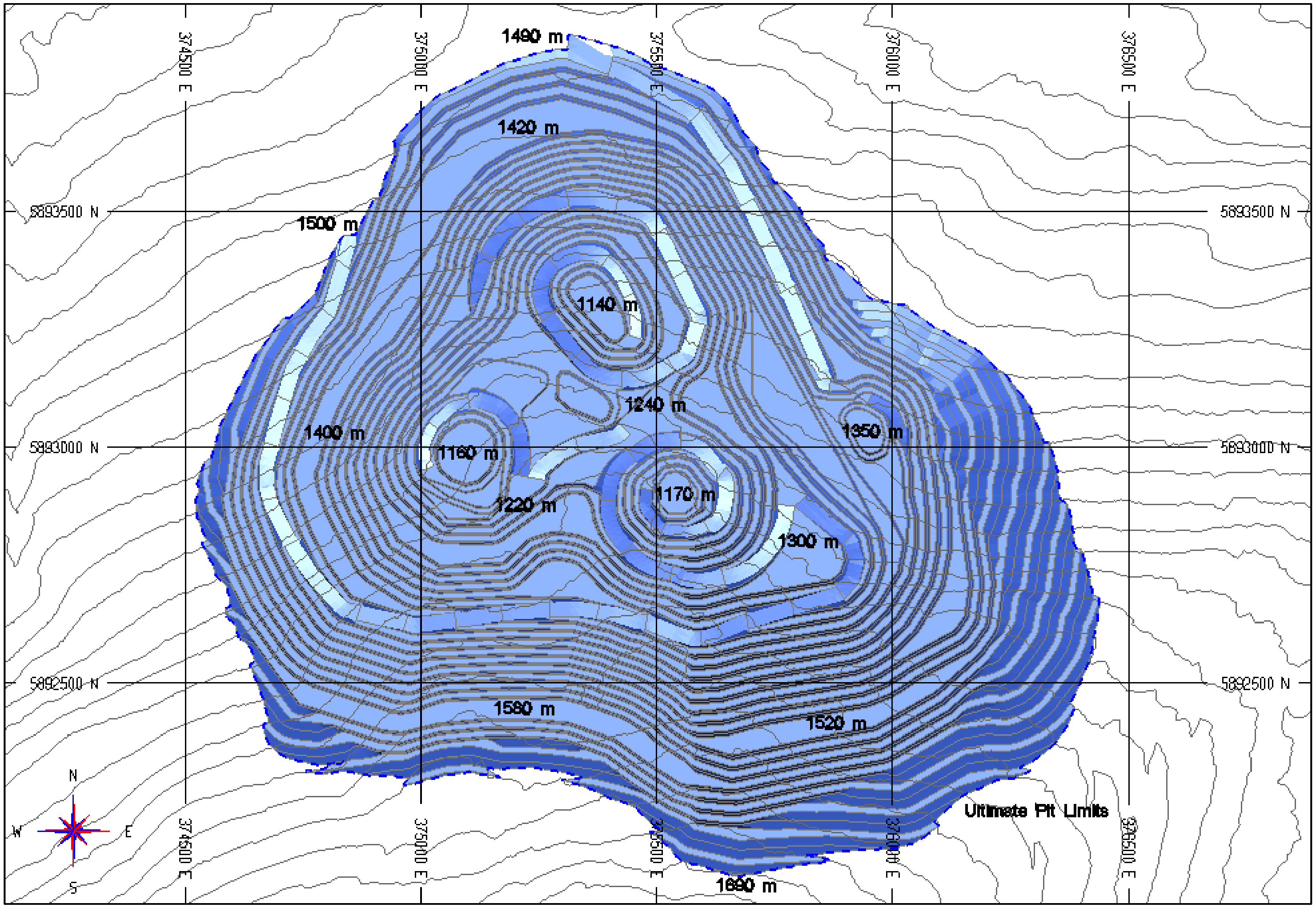

Figure 16-11 South Pushback, P657 (source: MMTS, 2020)

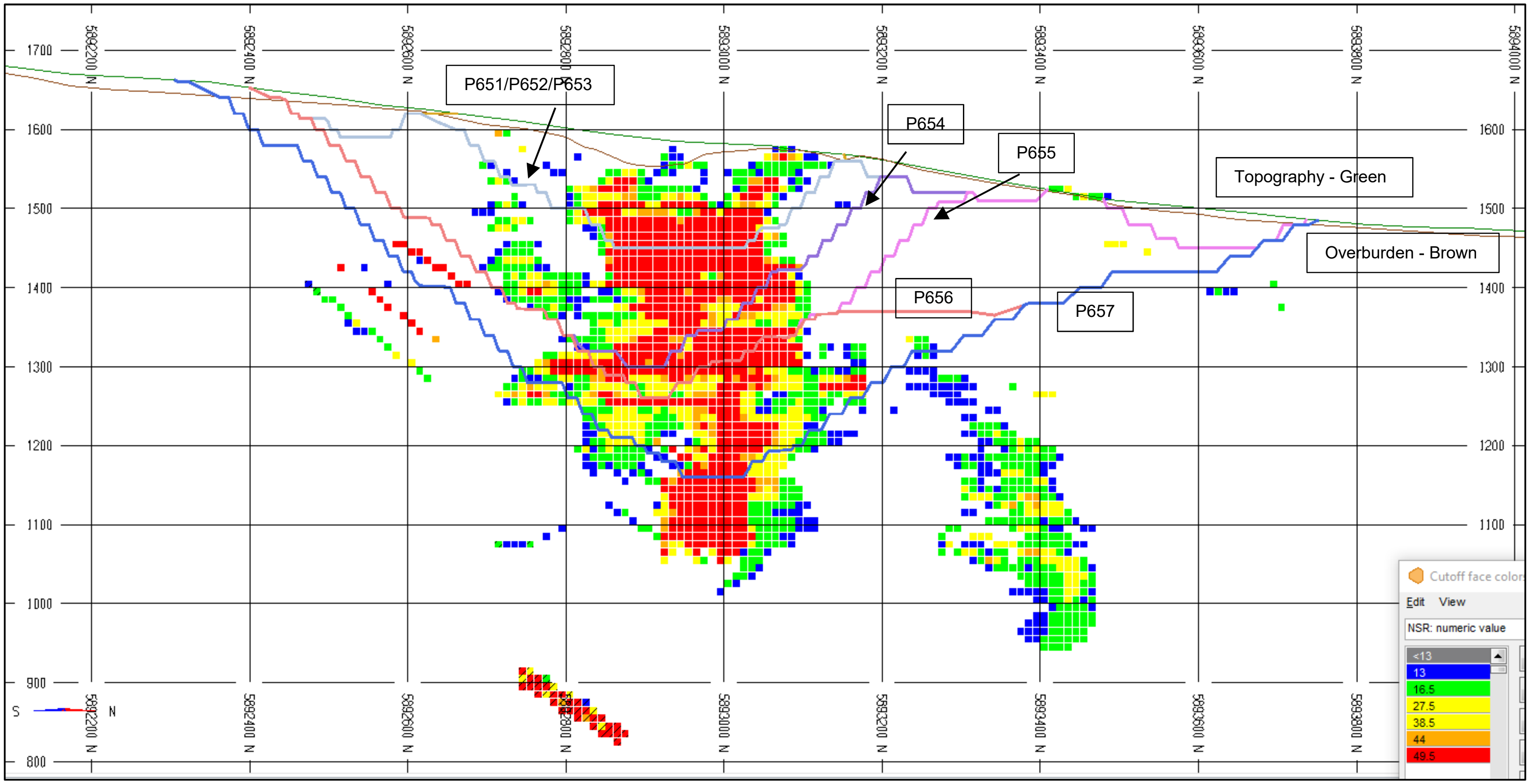

Figure 16-12 North South Section, 375,050E Looking West (source: MMTS, 2020)

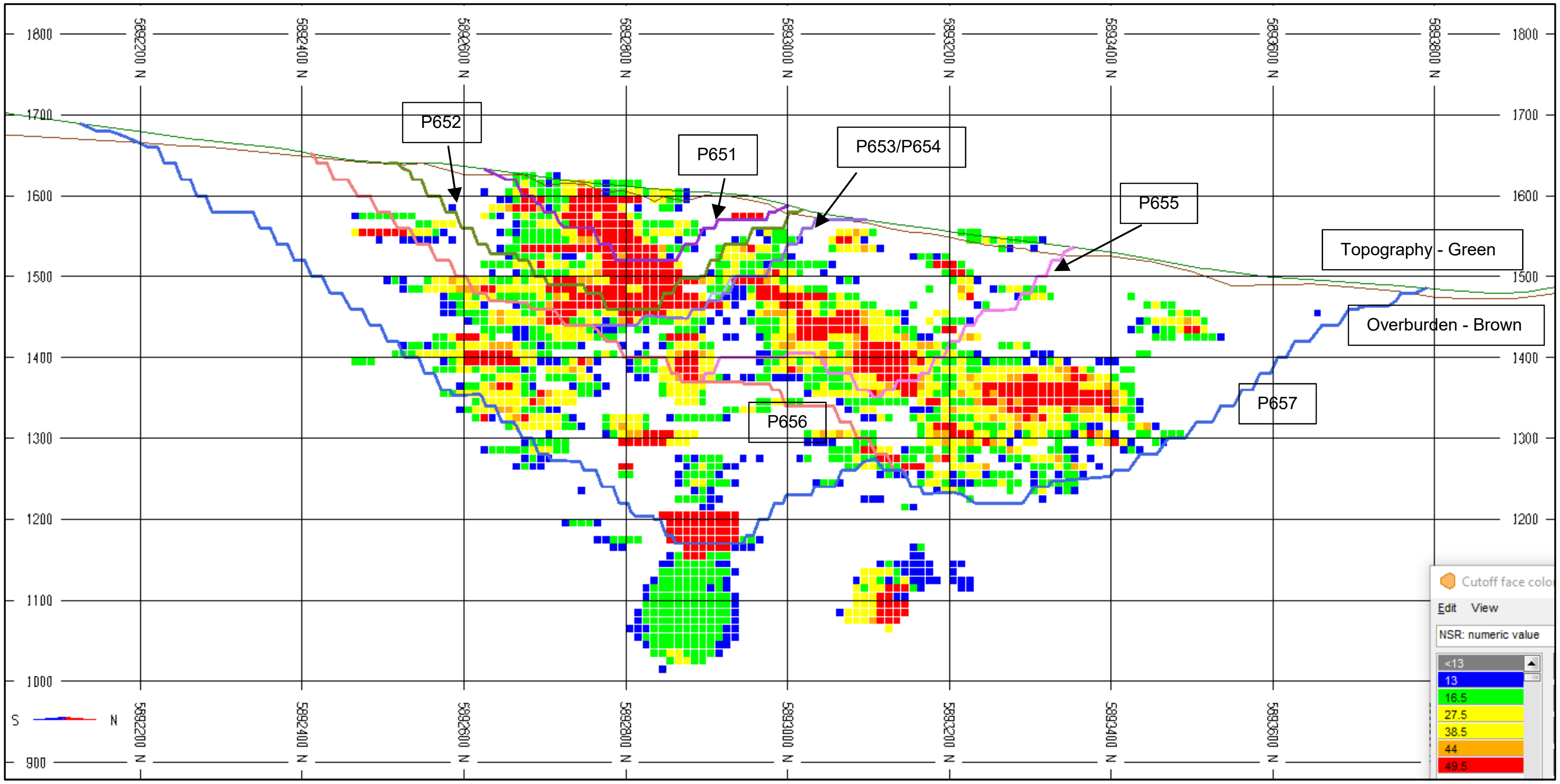

Figure 16-13 North South Section, 375,550E Looking West (source: MMTS, 2020)

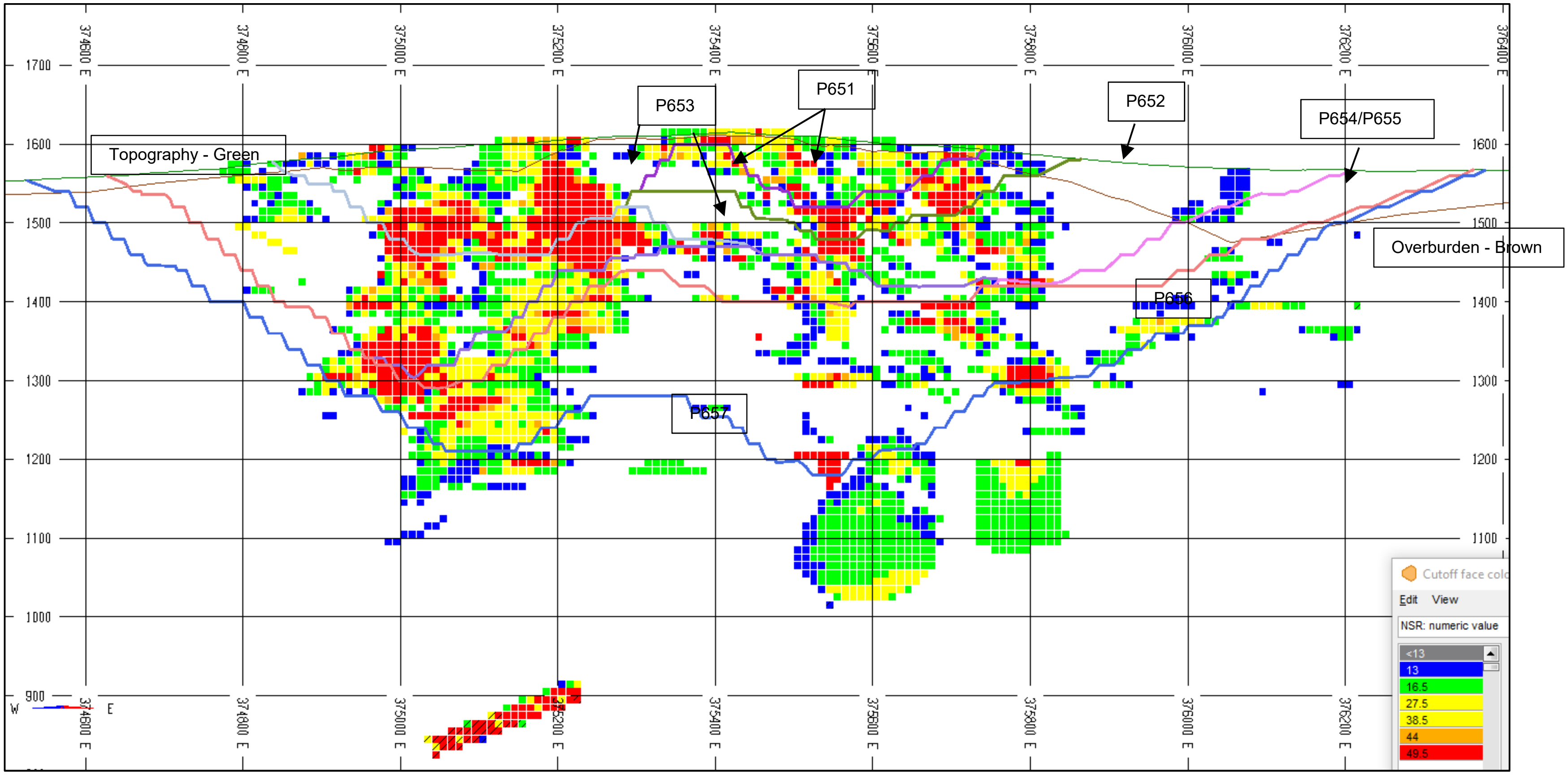

Figure 16-14 East West Section, 5,892,850N Looking North (source: MMTS, 2020)

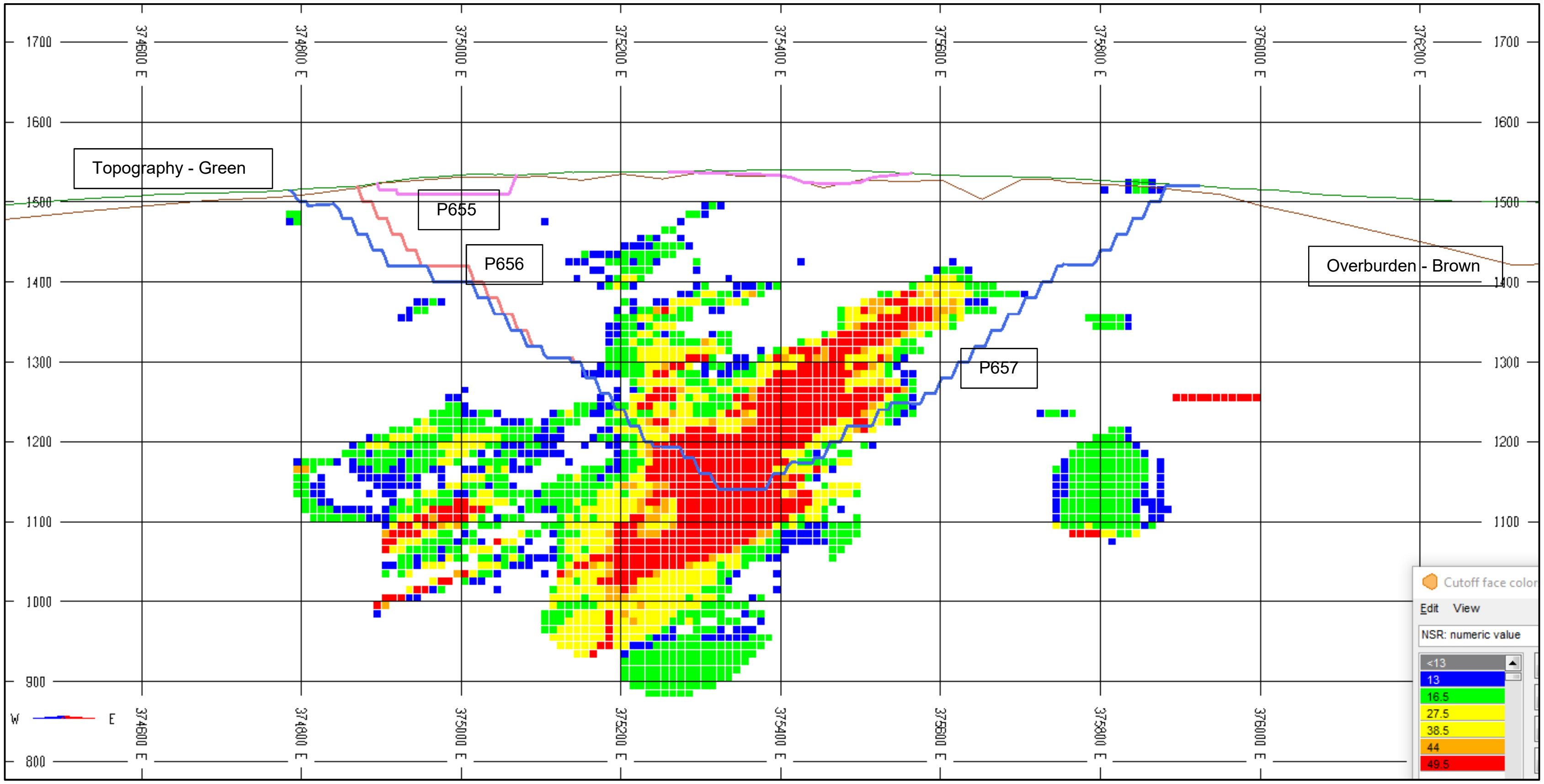

Figure 16-15 East West Section, 5,893,350N Looking North (source: MMTS, 2020)

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:

- 36 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 16-16.

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.

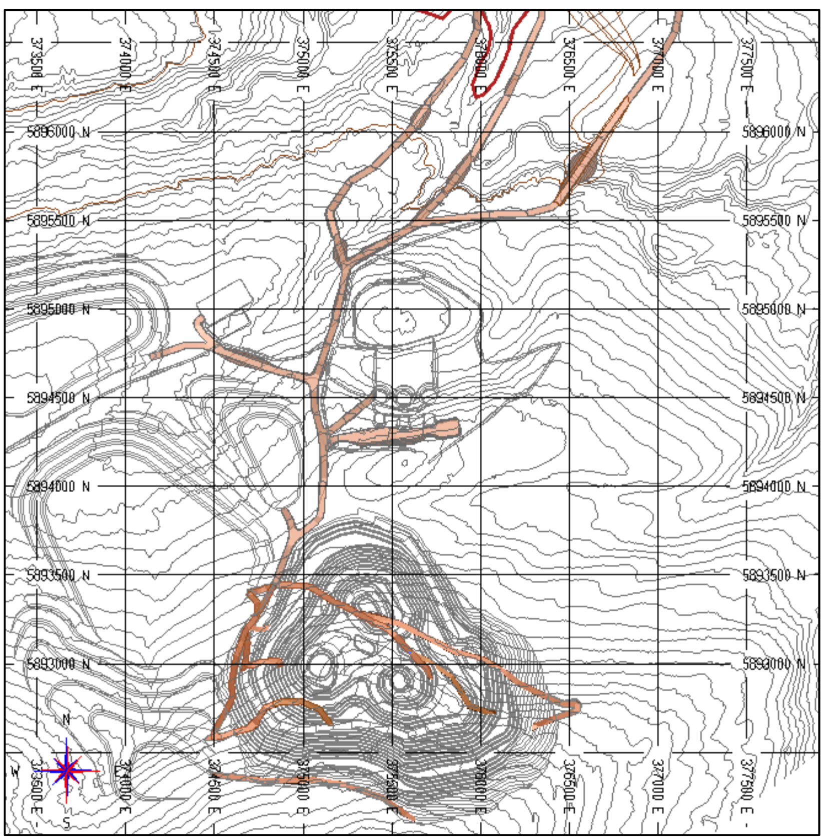

Figure 16-16 Ex-Pit Mine Haul Roads (source: MMTS, 2020)

16.5 Ore Storage Facilities

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

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

Throughout the life of operations, ore grading between $13.00/t and $27.50/t NSR will be stockpiled in low-grade ore stockpiles just outside the pit limits to the northwest.

Cut-off grade optimization on the mine production schedule also sends ore above $27.50/t NSR to a high-grade ore stockpile over the first five years of the mine life. This stockpile is also located just outside the pit limits to the northwest.

The stockpiled ore is planned to be re-handled back to the crusher during the mine life. The high-grade stockpile will be re-handled during pit operations and the low-grade stockpile once the pits are exhausted.

The ore stockpiles are shown in Figure 16-17.

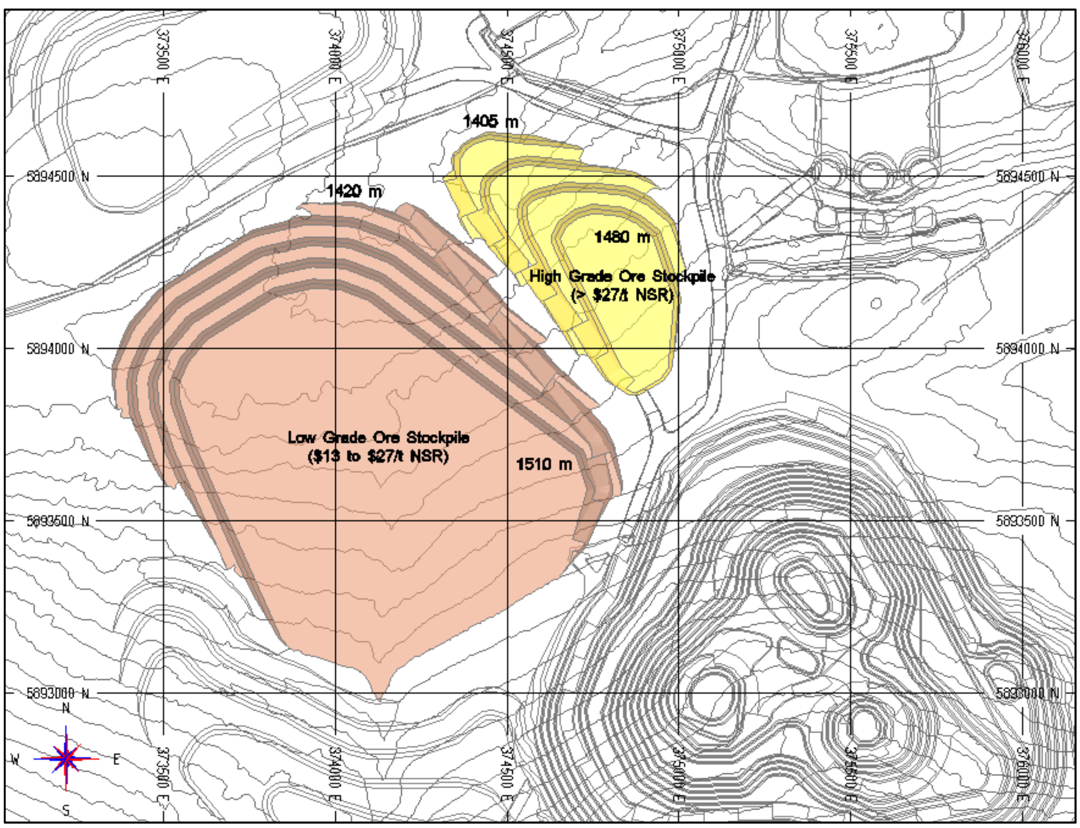

Figure 16-17 Ore Stockpiles (source: MMTS, 2020)

The ore stockpiles are designed to sit within existing EA certified boundaries for disturbance.

These stockpiles will be built on the hillside, each on 4 x 20 m lifts dumped out at angle of repose (1.3H:1V). Each facility is planned at a 3H:1V overall slope, the low-grade stockpile from the 1,420 m elevation to the 1,510 m elevation, the high-grade stockpile from the 1,405 m elevation to the 1,480 m elevation.

Ore is classified as PAG with a relatively short lag time, and the ore stockpiles are expected to generate acidic drainage with elevated metals until the ore is processed. The stockpiled ore will be placed on a compacted till liner with a drainage collection system. The drainage will be neutralized with lime prior to discharge to the TSF.

16.6 Waste Rock Storage Facilities

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 acid generation potential of the waste and ore was estimated using the calculated neutralization potential ratio (NPR) values derived from sulphur and neutralization potential (NP) assays and estimates.

Concentrations of sulphur were converted to acid potential (AP) values for use in the NPR calculation (where NPR = NP/AP). Values for NP were determined directly in ABA or estimated from the calcium assay from the larger geological metal dataset; the latter assumed calcium was only present as calcium carbonate and was the sole source of acid neutralization capacity. 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). The waste material is grouped based on whether it is predicted to be PAG or NAG, as shown by the calculated NPR. NAG waste rock was further classified as to its metal leaching potential based on zinc content, as that metal (and cadmium) is elevated in some of the mine waste. The classification criteria are 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)

Median NPR values for PAG waste rock (particularly PAG1) are relatively low, indicating a relatively short lag time to acid production; this was verified in laboratory kinetic (humidity cells) and field tests (field bins). The humidity cell tests showed that all the PAG1 and some of the PAG2 samples were acid generating with little to no lag time. The results of the field bin tests for waste rock were similar to those for the humidity cell tests, but typically had a longer lag time to acid generation. 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 significantly reduce metal leaching (ML) potential.

In contrast, median NPR values for NAG waste rock are relatively high, indicating little potential for acid production. Segregation of waste rock with zinc greater than 1,000 ppm (NAG3) results in high NPR and relatively low zinc and cadmium concentrations in the remaining NAG4 and NAG5 waste rock. NAG4 and NAG5 waste rock can be stored in out-of-pit stockpiles, while storage of NAG3 subaqueously in the TSF will limit the potential for neutral metal leaching from NAG waste rock.

All overburden samples from the open pit, plant, TSF, and access road area were classified as NAG based on ABA results and the net acid generation test. However, one sample from the overburden-bedrock interface had elevated sulphur and was classified as PAG, and three interface samples had an uncertain ARD classification. Except for material near the bedrock interface, overburden can be used for construction and reclamation purposes. Overburden near the interface will be tested and if found to be PAG or ML it will be managed like PAG waste rock.

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 achieve chemical stability were established during previous studies (AMEC, 2014a; KP, 2015) and remain unchanged for the 2020 PFS. These were:

- NAG4 and NAG5 waste rock and overburden (no expected ML or ARD): Used as a construction material with surplus and unsuitable (geotechnical) materials disposed of in on-land WRSFs, progressively reclaimed during operations and revegetated at closure

- NAG3 waste rock (potentially ML): Used to construct TSF embankments and storage of any surplus in the TSF in such a manner that it is saturated within 3-5 years by either water or tailings to limit ML potential

- PAG1 and PAG2 waste rock (expected ML and ARD): Stored within the TSF and used sparingly to construct TSF embankments in such a manner that it is saturated within one year by either water or tailings to limit oxidation and subsequent acid generation.

The vast majority of the waste materials will be used for construction of TSF or placed in the TSF itself. Stockpiles are 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 either the upper overburden stockpile or the lower NAG and overburden stockpile. These stockpiles are shown in Figure 16-18. They are designed to sit within existing EA certified boundaries for disturbance.

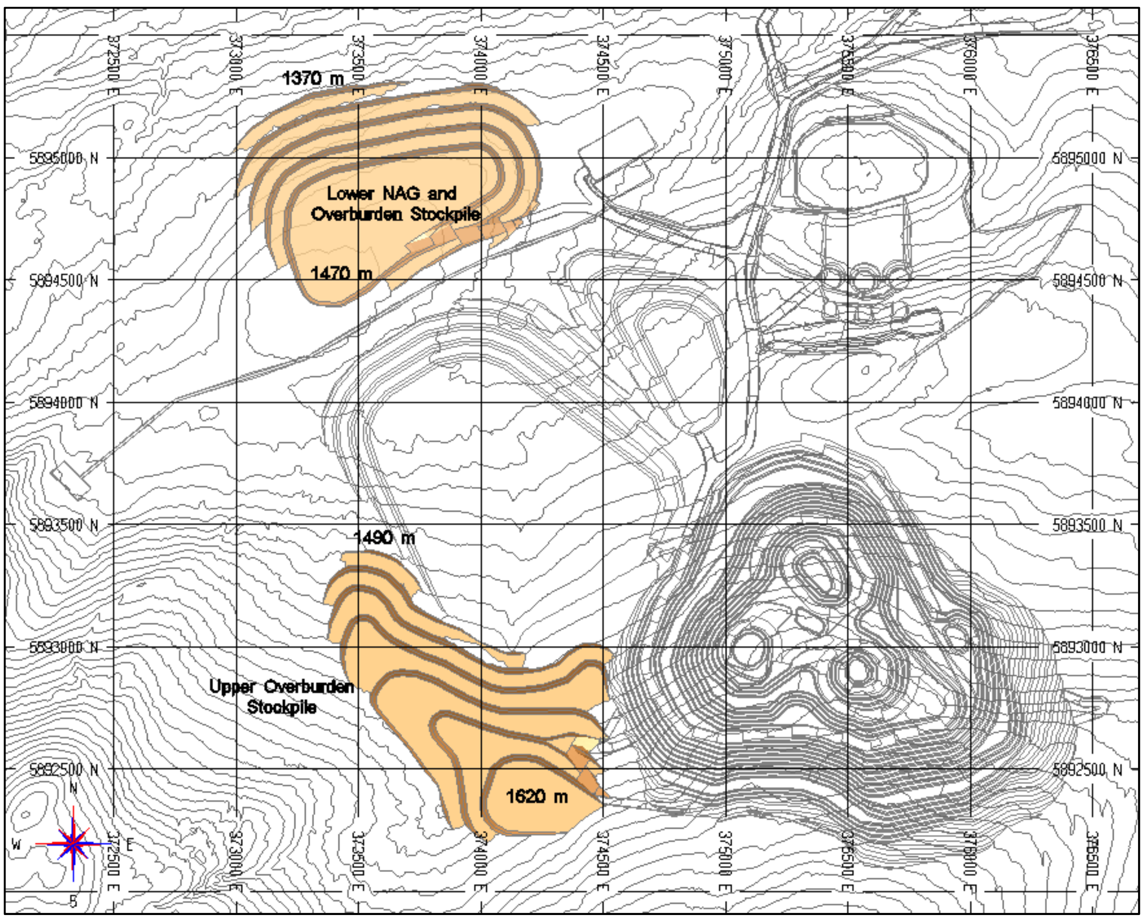

Figure 16-18 Waste Stockpiles (source: MMTS, 2020)

The upper overburden stockpile will be located directly west of the pit limits, is planned on 6 x 20 m lifts dumped out at angle of repose (1.3H:1V) and will store solely overburden waste materials. It is planned at a 4.5H:1V overall slope from the 1,490 to 1,620 m elevation. The upper lifts of the eastern side of this facility will be used as a haul road to access the upper benches of the final South Pushback pit highwall.

The lower NAG and overburden stockpile will be located 1.5 km northwest of the pit limits and is planned on five 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 m elevations.

Overburden and NAG waste rock will be used for construction of the downstream and some upstream portions of the TSF dams. PAG waste rock (including NAG3 classified materials) 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.

16.7 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 16-19 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;
- An annual mill feed rate of 5.5 Mt is targeted for the first five years of operation, increasing to 12 Mt for the next five years, and 20 Mt thereafter;
    - Year 1 throughput is targeted at 4.5 Mt (82% of nameplate capacity);
- 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 90 m in each year:
    - Average annual vertical phase progression is 55 m;
- Pre-production mining requirements are as follows:

  - 850 kt of overburden and NAG waste rock for haul road construction purposes;
  - 7,850 kt of overburden, PAG 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;
- 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	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**	**334,048**	**0**	**0**	**4,500**	**5,500**	**5,500**	**5,500**	**5,500**	**12,000**	**12,000**	**12,000**	**12,000**	**12,000**	**20,000**	**20,000**	**20,000**	**20,000**	**20,000**	**20,000**	**20,000**	**20,000**	**20,000**	**20,000**	**20,000**	**20,000**	**7,546**
Au	**g/t**	**0.75**	**0.00**	**0.00**	**1.65**	**1.70**	**1.61**	**1.42**	**1.46**	**1.56**	**1.27**	**1.17**	**0.98**	**0.87**	**0.64**	**0.70**	**0.81**	**0.76**	**0.81**	**0.83**	**0.62**	**0.39**	**0.30**	**0.30**	**0.30**	**0.30**	**0.30**
Ag	**g/t**	**5.78**	**0.00**	**0.00**	**8.83**	**7.12**	**7.52**	**8.04**	**7.22**	**6.46**	**6.49**	**7.31**	**7.90**	**10.60**	**3.69**	**2.63**	**3.53**	**4.27**	**5.10**	**5.73**	**5.08**	**6.12**	**6.16**	**6.16**	**6.16**	**6.16**	**6.16**
TOTAL Ore Mined from Pit	ktonnes	334,048	0	481	10,808	13,717	12,565	14,381	15,624	19,492	21,072	25,504	27,450	23,155	24,420	23,659	17,194	19,382	24,314	24,381	14,246	2,204	0	0	0	0	0
Au	g/t	0.75	0.00	0.40	0.92	0.92	0.92	0.77	0.77	1.09	0.87	0.72	0.59	0.56	0.57	0.63	0.81	0.71	0.70	0.73	0.74	1.17	0.00	0.00	0.00	0.00	0.00
Ag	g/t	5.78	0.00	5.73	6.54	5.73	6.30	7.47	5.72	5.29	5.12	6.33	9.90	12.35	3.64	2.56	2.38	3.53	4.62	5.31	4.23	5.77	0.00	0.00	0.00	0.00	0.00
Ore Mined Directly to Mill	ktonnes	208,706	0	0	4,500	5,500	5,500	5,500	5,500	12,000	12,000	12,000	12,000	10,000	20,000	20,000	15,000	16,000	19,000	20,000	12,000	2,204	0	0	0	0	0
Au	g/t	1.00	0.00	0.00	1.65	1.70	1.61	1.42	1.46	1.56	1.27	1.17	0.98	0.94	0.64	0.70	0.89	0.81	0.83	0.83	0.84	1.17	0.00	0.00	0.00	0.00	0.00
Ag	g/t	5.52	0.00	0.00	8.83	7.12	7.52	8.04	7.22	6.46	6.49	7.31	7.90	11.38	3.69	2.63	2.47	3.66	5.02	5.73	4.32	5.77	0.00	0.00	0.00	0.00	0.00
Ore Mined to Stockpile	ktonnes	125,342	0	481	6,308	8,217	7,065	8,881	10,124	7,492	9,072	13,504	15,450	13,155	4,419	3,659	2,194	3,382	5,314	4,381	2,246	0	0	0	0	0	0
Au	g/t	0.32	0.00	0.40	0.40	0.40	0.39	0.37	0.39	0.34	0.33	0.32	0.28	0.27	0.25	0.26	0.26	0.25	0.25	0.25	0.25	0.00	0.00	0.00	0.00	0.00	0.00
Ag	g/t	6.22	0.00	5.73	4.90	4.80	5.35	7.11	4.90	3.43	3.31	5.46	11.45	13.08	3.46	2.17	1.76	2.94	3.20	3.40	3.75	0.00	0.00	0.00	0.00	0.00	0.00
Stockpile Retrieval to Mill	ktonnes	125,342	0	0	0	0	0	0	0	0	0	0	0	2,000	0	0	5,000	4,000	999	0	8,000	17,796	20,000	20,000	20,000	20,000	7,546
Au	g/t	0.32	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.56	0.00	0.00	0.56	0.56	0.43	0.00	0.30	0.30	0.30	0.30	0.30	0.30	0.30
Ag	g/t	6.22	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	6.70	0.00	0.00	6.70	6.70	6.60	0.00	6.22	6.16	6.16	6.16	6.16	6.16	6.16
Stockpile Balance	ktonnes		0	481	6,789	15,005	22,070	30,951	41,075	48,567	57,639	71,143	86,593	97,747	102,167	105,825	103,020	102,402	106,716	111,096	105,342	87,546	67,546	47,546	27,546	7,546	0
Au	g/t		0.00	0.40	0.40	0.40	0.40	0.39	0.39	0.38	0.37	0.36	0.35	0.33	0.33	0.33	0.31	0.30	0.30	0.30	0.30	0.30	0.30	0.30	0.30	0.30	0.00
Ag	g/t		0.00	5.73	4.96	4.87	5.03	5.62	5.45	5.14	4.85	4.96	6.12	7.05	6.89	6.73	6.62	6.50	6.33	6.22	6.16	6.16	6.16	6.16	6.16	6.16	0.00
TOTAL Waste Mined	ktonnes	667,107	1,100	8,343	16,081	14,891	14,935	36,331	30,466	30,708	31,814	54,576	53,015	53,760	55,780	54,358	66,783	60,412	41,347	28,478	13,547	381	0	0	0	0	0
NAG Rock Waste (NAG4/NAG5)	ktonnes	116,873	204	4,224	1,487	3,257	1,277	3,603	10,820	4,246	1,211	5,401	5,194	4,133	10,369	21,923	24,064	13,337	1,989	91	43	0	0	0	0	0	0
PAG Rock Waste (PAG1, PAG2, NAG3)	ktonnes	467,287	120	1,123	8,759	9,394	6,987	15,596	16,269	25,457	24,487	35,981	43,767	49,030	37,877	27,306	36,458	47,046	39,359	28,387	13,504	381	0	0	0	0	0
Overburden Waste	ktonnes	82,927	776	2,996	5,835	2,240	6,669	17,132	3,376	1,006	6,116	13,194	4,055	597	7,535	5,129	6,242	29	0	0	0	0	0	0	0	0	0
Wasted Inferred	ktonnes	20	0	0	0	0	2	0	0	0	0	0	0	0	0	0	18	0	0	0	0	0	0	0	0	0	0
Au	g/t	0.34	0.00	0.00	0.00	0.00	0.23	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.35	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00
Ag	g/t	1.39	0.00	0.00	0.00	0.00	1.70	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	1.35	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00
Waste Destination Summary																											
OVB to Construction	ktonnes	3,300	250	250	0	0	0	2,800	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0
NAG to Construction	ktonnes	4,450	200	150	200	200	200	450	250	250	250	250	250	300	300	300	300	300	300	0	0	0	0	0	0	0	0
OVB to Tailing Dam	ktonnes	41,150	500	2,450	1,800	1,800	200	350	750	900	5,750	12,950	4,000	550	3,700	650	4,800	0	0	0	0	0	0	0	0	0	0
PAG to Tailing Dam	ktonnes	41,300	100	750	550	750	450	650	1,550	2,700	3,100	4,500	4,700	2,150	1,950	1,950	2,300	6,100	6,300	750	0	0	0	0	0	0	0
NAG to Tailing Dam	ktonnes	91,050	0	4,050	1,250	3,050	1,050	3,150	10,550	3,950	950	5,150	4,900	3,600	9,850	11,650	13,800	12,700	1,350	50	0	0	0	0	0	0	0
PAG to TSF	ktonnes	426,007	20	373	8,209	8,644	6,539	14,946	14,719	22,757	21,387	31,481	39,067	46,880	35,927	25,356	34,176	40,946	33,059	27,637	13,504	381	0	0	0	0	0
OVB Upper	ktonnes	29,149	30	320	4,072	447	6,496	13,985	2,647	152	377	244	99	280	0	0	0	0	0	0	0	0	0	0	0	0	0
NAG and OVB Lower	ktonnes	30,701	0	0	0	0	0	0	0	0	0	0	0	0	4,053	14,452	11,407	366	339	41	43	0	0	0	0	0	0
Strip Ratio (Waste/Resource Mined)		2.0	0.0	17.3	1.5	1.1	1.2	2.5	1.9	1.6	1.5	2.1	1.9	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0
Cumulative Strip Ratio			0.0	19.6	1.6	1.6	1.5	1.8	1.8	1.8	1.7	1.8	1.8	1.9	1.9	2.0	2.1	2.2	2.1	2.1	2.0	2.0	2.0	2.0	2.0	2.0	2.0
Total Material Mined	ktonnes	1,001,155	1,100	8,824	26,889	28,608	27,500	50,712	46,089	50,201	52,887	80,080	80,465	76,915	80,200	78,017	83,978	79,794	65,661	52,859	27,793	2,586	0	0	0	0	0
Cumulative Material Mined	ktonnes		1,100	8,824	27,989	56,597	84,096	134,809	180,898	231,098	283,985	364,065	444,530	521,444	601,644	679,661	763,638	843,432	909,093	961,952	989,745	992,331	992,331	992,331	992,331	992,331	992,331
Total Material Moved	ktonnes	1,126,496	1,100	8,824	26,889	28,608	27,500	50,712	46,089	50,201	52,887	80,080	80,465	78,915	80,200	78,017	88,978	83,794	66,660	52,859	35,793	20,382	20,000	20,000	20,000	20,000	7,546

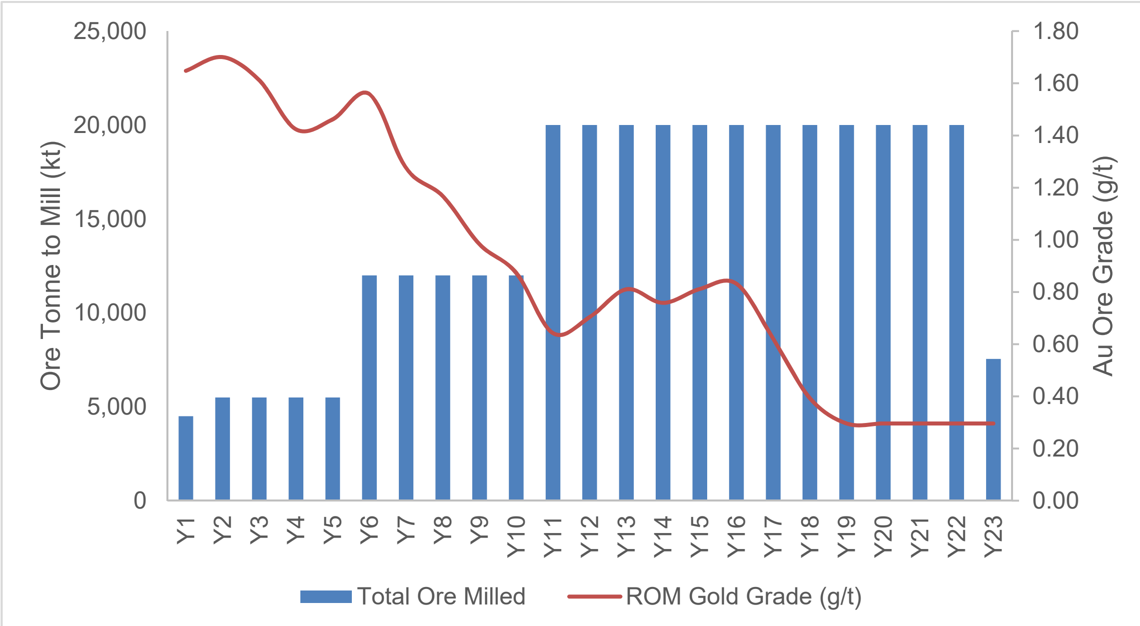

Figure 16-19 Mill Feed Tonnes and Grade (source: MMTS, 2020)

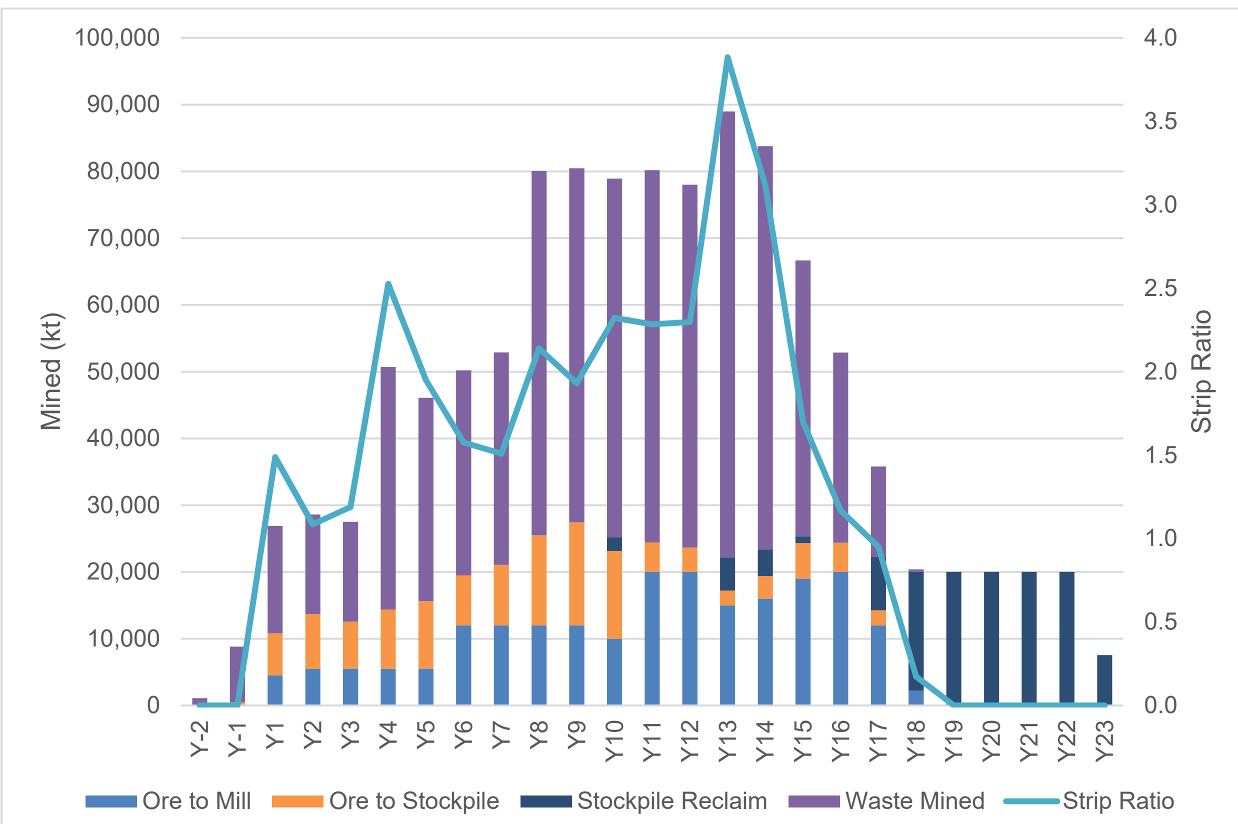

Figure 16-20 Material Mined and Strip Ratio (source: MMTS, 2020)

16.7.1 Mine Sequence

The pit operations will run for 20 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 10 and Year 18 are illustrated in Figure 16-21 to Figure 16-26.

Table 16-5 Annual Mine Operations

Year	Activity
Y-2 & Y-1	Clearing and grubbing the initial pit phases Clearing and grubbing of ex-pit haul road, and portions of the ore stockpiles and upper overburden piles. 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 1,510 bench. Mining of the starter pit down to the 1,610 bench. Delivery of construction rock to the process area, for use in the conveyor pads. Delivery of construction rock to Site C tailings dam. Stockpiling high-grade ore on the ROM pad and high-grade ore stockpile for use in mill commissioning. Stockpiling low-grade ore in the low-grade stockpile for storage until the end of mine life. Delivery of excess mined overburden to the upper overburden stockpile.
Y1	Continued clearing and grubbing of pit and stockpile areas (annually throughout mine life). Construction borrows pit mined down to 1,490 bench; starter pit mined down to 1,550 bench. Continued stockpiling of excess high-grade ore to the high-grade ore stockpile (continues until Y5). Continued delivery of low-grade ore to the low-grade ore stockpile (continues until Y17). Delivery of construction rockfill to Site C tailings dam (continues until Y16). Delivery of excess mined overburden to the upper overburden stockpile (continues until Y9).
Y2	Construction borrow pit mined to the pit bottom on the 1450 bench. Starter pit mined to the 1,480 bench and East Pushback 1 pit mined to the 1,580 bench.
Y3 To Y5	Starter pit mined to the pit bottom on the 1,450 bench (Y3). East Pushback 1 pit mined to the pit bottom on the 1460 bench (Y4). East Pushback 2 pit mined to the 1460 bench; West Pushback pit mined to the 1,540 bench. Y4 pit electrification and purchase of electric driven drills and shovels. Y5 final period of high-grade ore delivery to the high-grade ore stockpile. Y5 start of delivery of construction rockfill to the Site D tailings dam (continues until Y16).
Y6 To Y10	Mill feed increased to 12 Mtpa in Y6. East Pushback 2 pit mined to the pit bottom on the 1,380 bench (Y7). West Pushback pit mined to the pit bottom on the 1,300 bench (Y9). North Pushback 1 pit mined to the 1,310 bench; North Pushback 2 pit mined to the 1,410 bench. South pushback pit mined to the 1,670 bench. Y10 final period of use on the Upper Overburden stockpile. Re-handle to crusher of 2 Mt of stockpiled high-grade ore.
Y11 To Y18	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.
Y19 to Y23	Re-handle to crusher of remaining stockpiled low-grade ore (stockpiled depleted). Initiate work on closure plan for pits and overburden and NAG stockpiles.

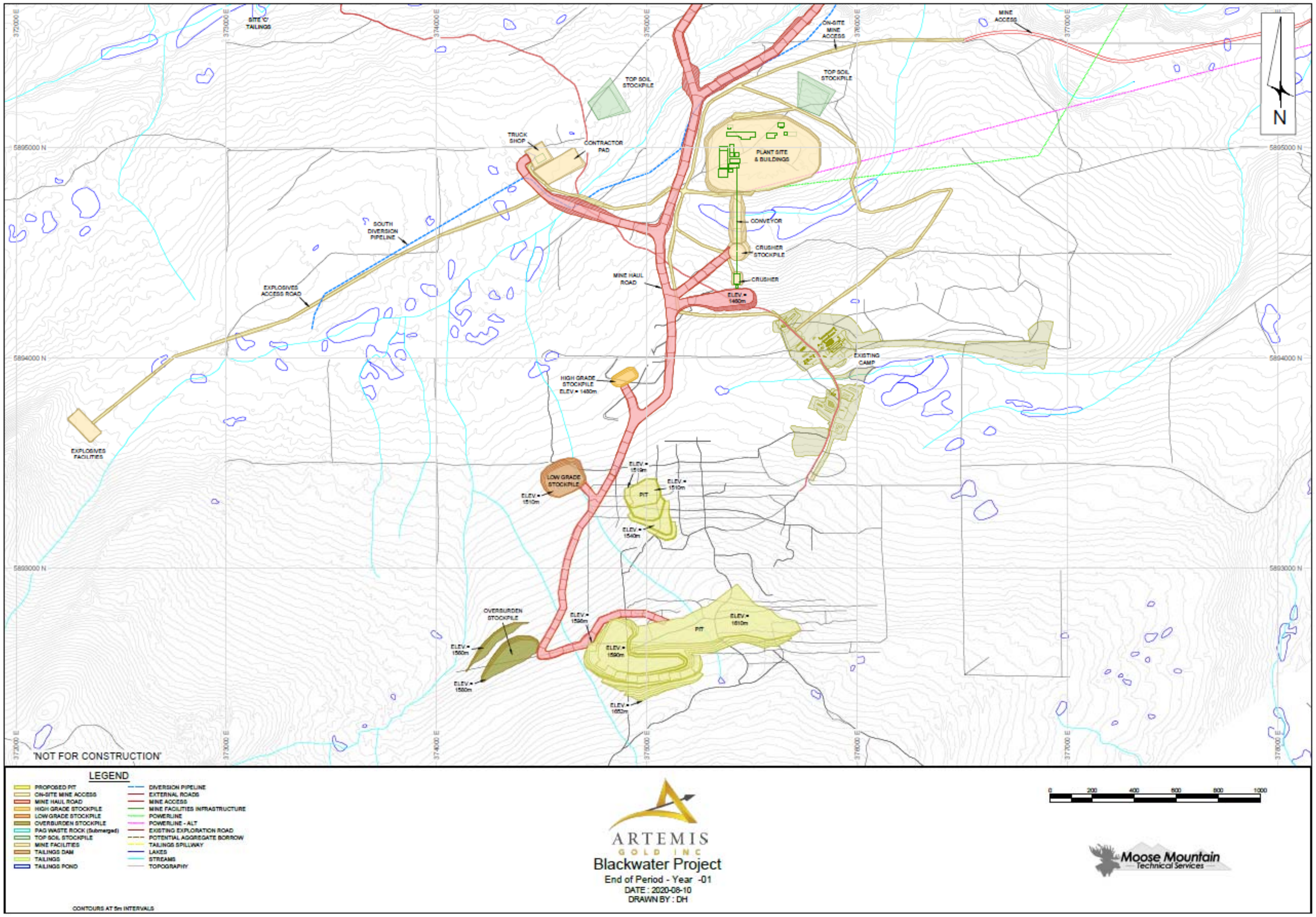

Figure 16-21 EOP Mine Operations, Y-1 (source: MMTS, 2020)

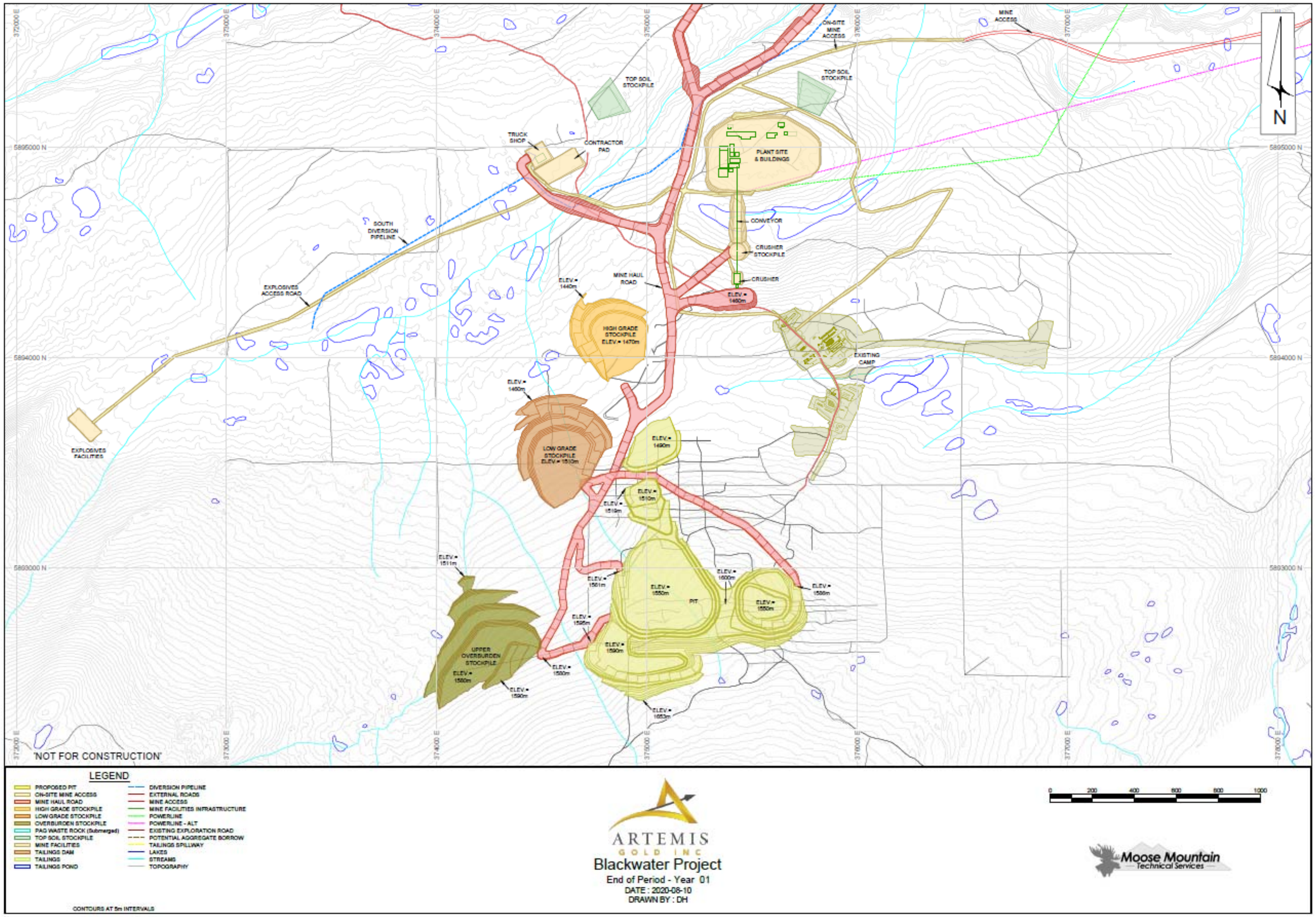

Figure 16-22 EOP Mine Operations, Y1 (source: MMTS, 2020)

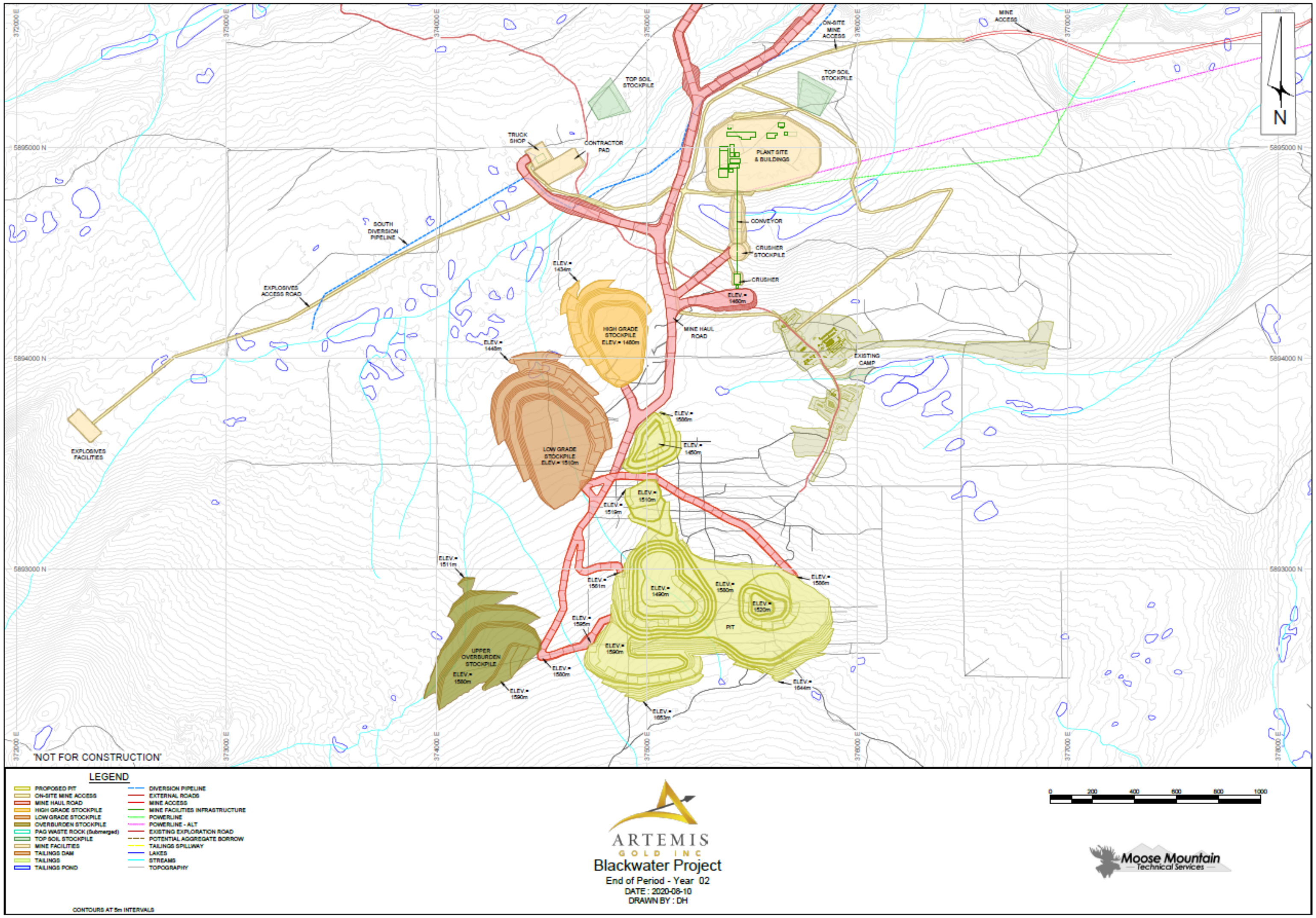

Figure 16-23 EOP Mine Operations, Y2 (source: MMTS, 2020)

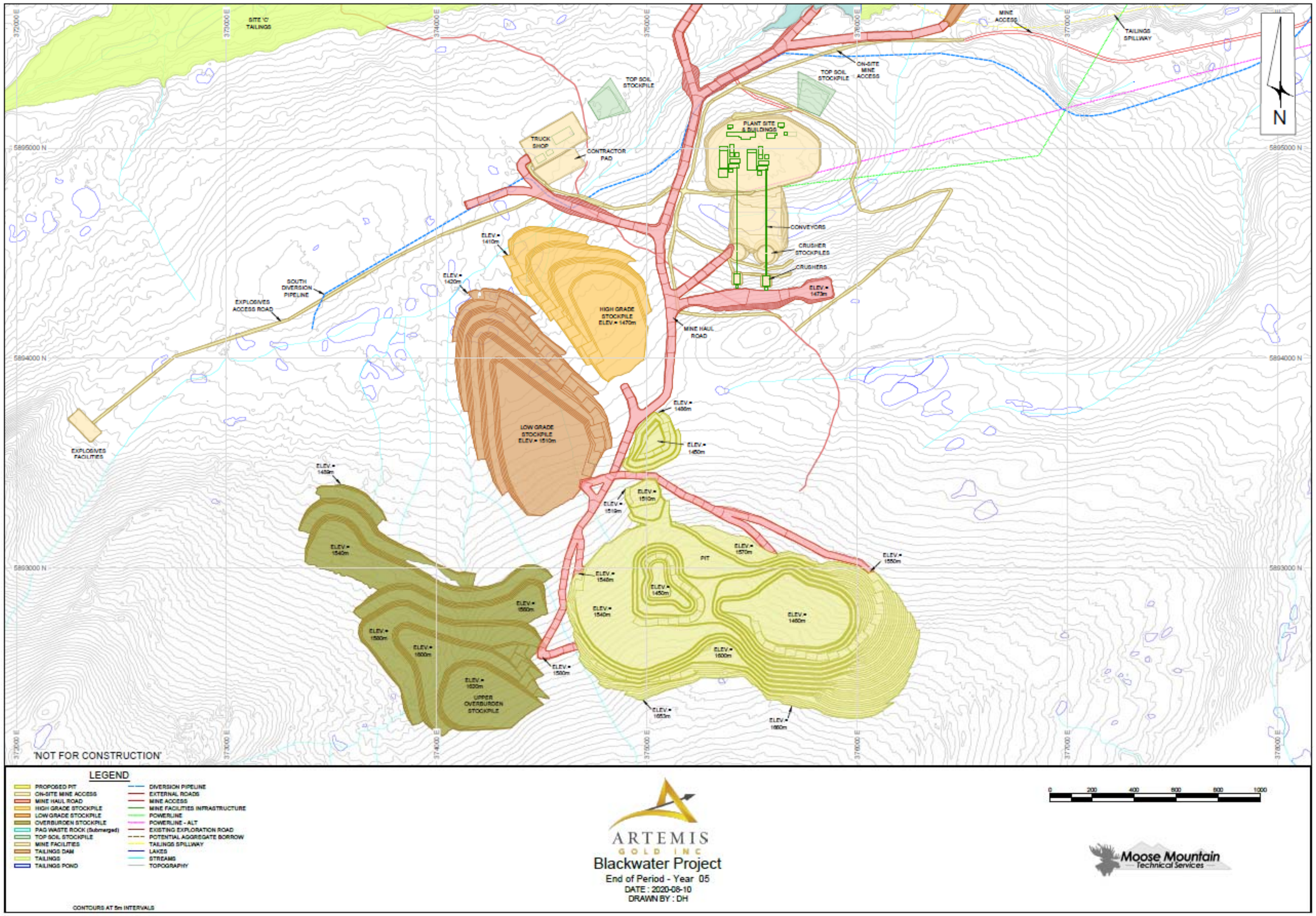

Figure 16-24 EOP Mine Operations, Y5 (source: MMTS, 2020)

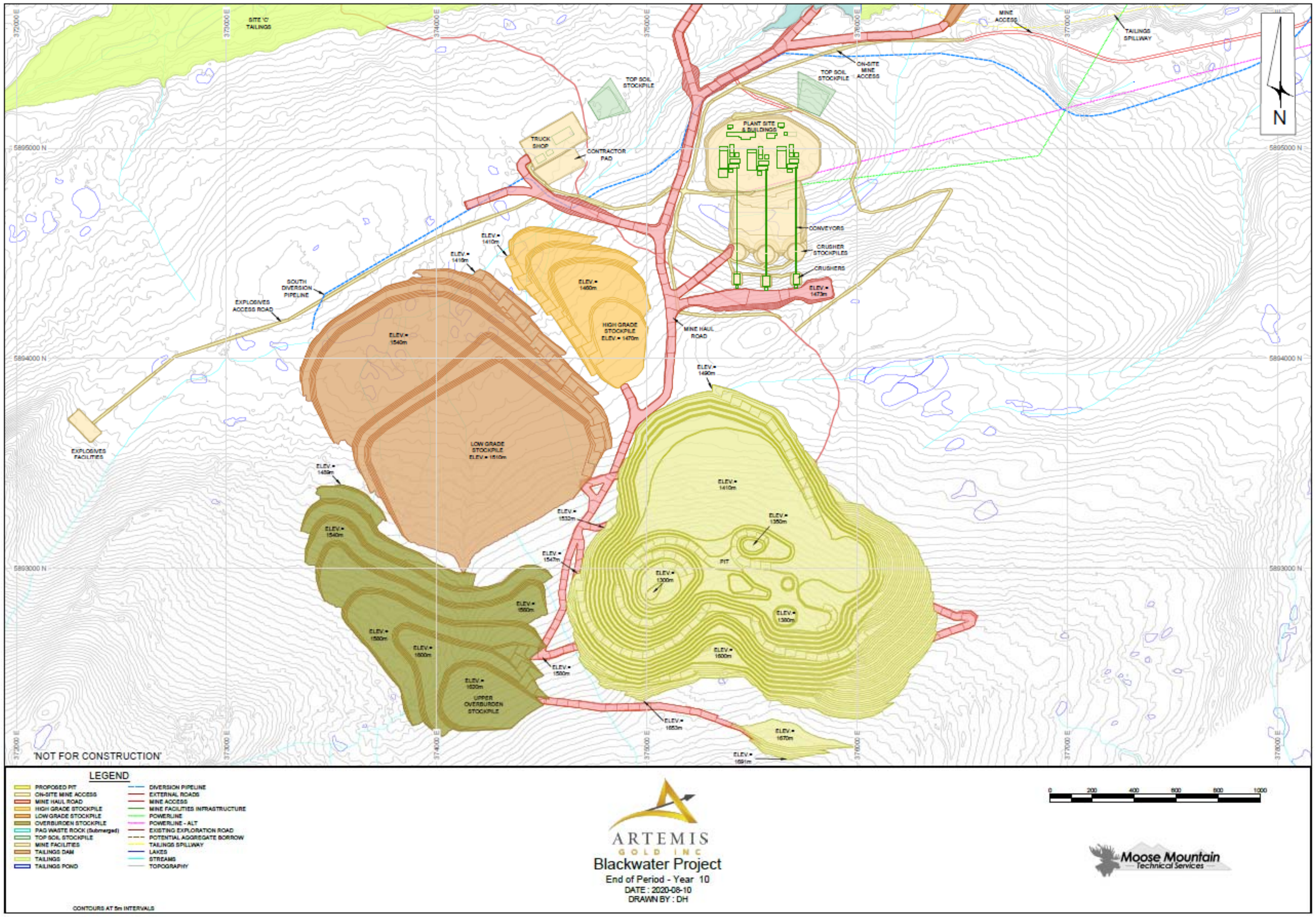

Figure 16-25 EOP Mine Operations, Y10 (source: MMTS, 2020)

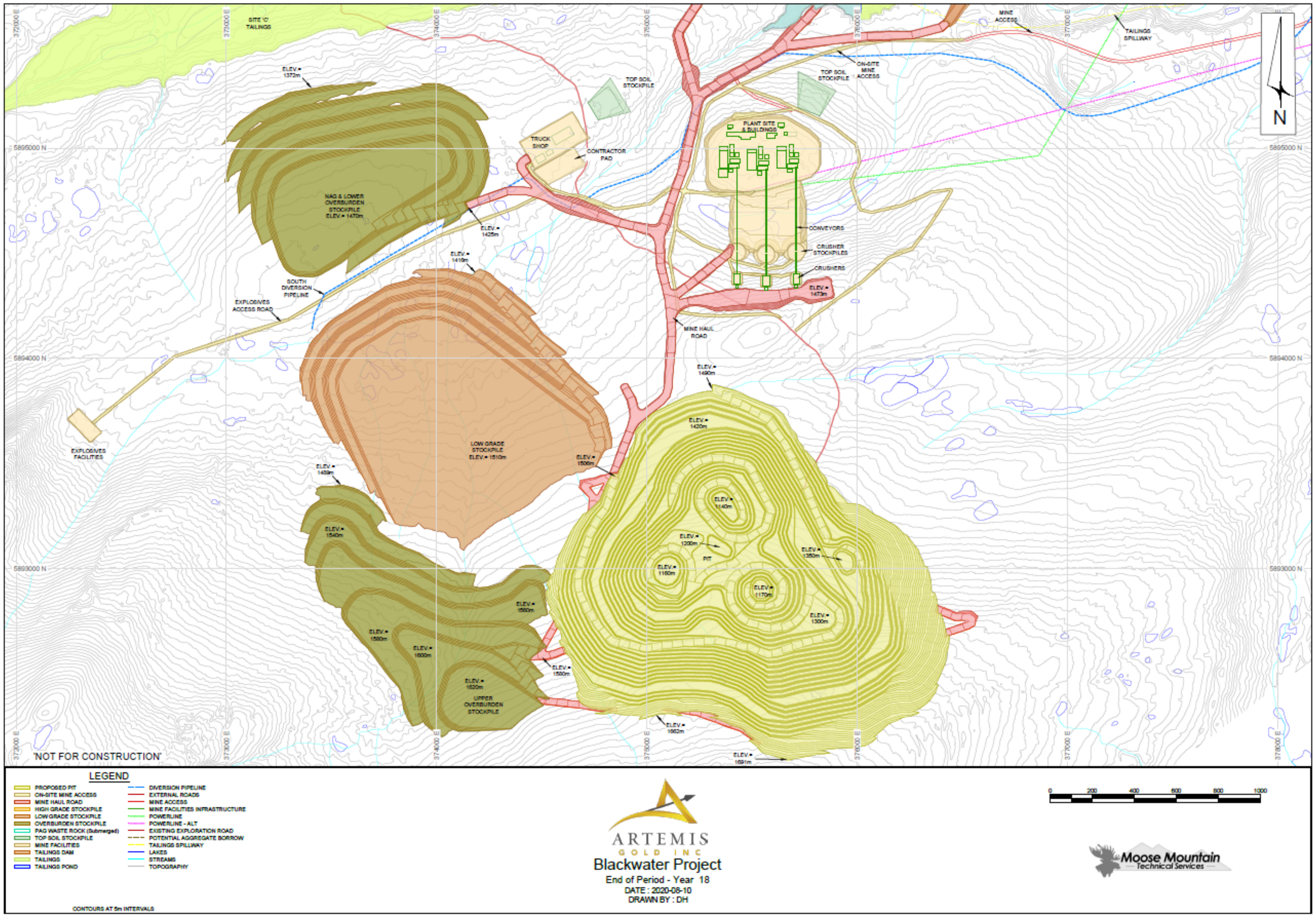

Figure 16-26 EOP Mine Operations, Y18 (source: MMTS, 2020)

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 0.32 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 plant and 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 315 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.

The one-in-100-year return period storm was used to size the pit surface water dewatering system and was estimated to be approximately 142,000 m^3. The estimated runoff coefficient inside the open pit surface area was conservatively assumed to be 100%.

The pit dewatering design is based on lowering the groundwater table within the highly-permeable zone to approximately 15 m below the pit base elevation, considering both removal of groundwater from storage and from recharge.

A combination of in-pit and perimeter pumping wells will be used for slope depressurization and pit dewatering. All pumping wells will be installed to a nominal depth of approximately 350 m below ground level.

The in-pit groundwater wells will remove water from storage in the highly permeable zone. This will also draw down water in the surrounding rock mass as the groundwater flows toward the highly-permeable zone. Perimeter dewatering wells will be established along the south high wall to lower and extend the cone of depression beyond the pit walls where drainage to the permeable zone is not adequate for slope depressurization. There will be approximately 10 dewatering wells spaced at 200 to 250 m intervals to achieve an adequate cone of depression to lower the groundwater level.

The site water management plan is described in Section 18.

16.9 Mine Equipment

Grade control drilling will be carried out with 144 mm (5.5") diesel RC drills, with sampling and assaying on 2 m intervals. Production drilling will be carried out with 254 mm (10") diesel rotary drills in waste and 203 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 (27 m^3 bucket) are proposed for waste loading based on their efficient pass match to the haulers and productivity on 10 m benches. Front end wheel loaders (19.0 m^3 bucket) are proposed based on their ability 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 4 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 (191 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 (231 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 6 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. The mine operations graders are also planned to maintain on-site roads. On-highway trucks that are outfitted with a water tank (150 kL) and gravel spreader are included for haul road maintenance.

Track dozers (450 kW) are included to handle waste rock and overburden to the various waste storage locations. Track dozers (325 kW) are included to support in pit mining activities. Front-end wheel loaders (13.5 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, 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 (pit bottom submersible pumps).

Mine fleet maintenance activities will generally be performed in the truck shop facilities that will be located 1.4 km north 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	Y19	Y20	Y21	Y22	Y23
Drilling																									
Diesel Rotary tracked drill 254 mm (10") holes	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	0	0	0	0	0
Electric Rotary tracked drill 254 mm (10") holes	0	0	0	0	0	0	1	1	1	1	2	2	2	2	2	2	2	2	2	2	0	0	0	0	0
Diesel Rotary tracked drill 203 mm (8") holes	1	1	2	2	2	3	3	3	4	4	4	4	4	4	4	4	4	4	2	2	0	0	0	0	0
Diesel RC tracked drill 144 mm (5.5") holes	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	0	0	0	0	0
Loading																									
Diesel Hydraulic front shovel 27 m^3 bucket	0	0	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	0	0	0	0	0	0	0
Electric Hydraulic front shovel 27 m^3 bucket	0	0	0	0	0	0	1	1	1	1	2	2	2	2	2	2	2	1	1	1	1	1	1	1	1
Diesel Hydraulic excavator 22 m^3 bucket	0	1	1	1	1	2	2	2	2	2	2	2	2	2	2	2	2	2	1	1	0	0	0	0	0
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	1	1	1	1	1
Hauling																									
Rigid frame haul truck 191 t payload	3	4	10	10	10	18	19	19	19	19	19	19	19	19	19	19	16	12	8	3	0	0	0	0	0
Rigid frame haul truck 231 t payload	0	0	0	0	0	0	0	4	4	11	11	16	16	16	16	16	16	16	10	3	3	3	3	3	3
Articulated haul truck 40 t payload	2	4	4	4	4	4	4	4	4	2	2	2	2	2	2	2	2	2	0	0	0	0	0	0	0

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	3
Water/Gravel Truck	Haul Road Maintenance	1
Track Dozer (450 kW)	Dump Maintenance	3
Track Dozer (325 kW)	Pit Maintenance, Shovel Support, Snow Clearing, Site Prep, Construction	3
Wheel Loader (19 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	2
Fuel and Lube Truck	Fuel/lube support of hydraulic shovel, wheel loader, drills and support equipment	3
Shuttle Bus (15 passenger)	Employee Transportation	4
Pickup Trucks (1/4 ton)	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

The process design criteria were developed based on the test work described in Section 13. Information from the 2014 FS was also used. The proposed process is similar to many gold plants around the world, and many details of the design have been used very successfully in Atlantic Gold's gold ore leach plant in Nova Scotia.

The plant will consist of:

- Three stage crushing;
- Grinding in a single ball mill with gravity concentration incorporated into the grinding circuit;
- Leaching with pre-oxidation;
- Carbon in pulp adsorption of gold and silver;
- Elution and electrowinning ;
- Smelting to gold silver doré;
- Cyanide destruction using SO_2 air system.

The process design criteria are shown in Table 17-1.

Table 17-1 Summary of Process Design Criteria

	Unit	Nominal Value	Source
General			
Daily Throughput	t/d	15,000	
Process Plant Availability		92%	
Design Feed Rate	t/hr	679	
LOM Head Grade Au	g/t	1.3	
LOM Head Grade Ag	g/t	5.1	
Overall Recovery Au	%	93%	Test work
Overall Recovery Ag	%	60%	Test work
Crushing			
Availability/Utilization	%	75%	
Crusher throughput	t/hr	833	
Crushing work index	kWh/t	0	From 20131 FS
Maximum feed size	mm	1,400	
Product size (P80)	mm	13	
Crusher Ore Stockpile	t	45,000	Design consideration
Live Capacity	t	15,000	Approximate calculation
Grinding			
Bond Ball Mill Work Index	kWh/t	21.1	75% percentile from 2019 test work
Bond Abrasion Index	g	0.25	75% percentile from 2019 test work
Ball Mill Diameter	m	7.3	
Ball Mill Length	m	12.5	
Installed Ball Mill Power	MW	14	
Circulating Load	%	250%	
Product size (P80)	µm	150	
Gravity Concentration			
Feed to gravity circuit	t/hr	800	
Feed to gravity concentrators	mm	-2.0	
% Feed <2 mm		70%	
Feed to gravity concentrators	t/hr	560	
Number of Gravity Concentrators		2	
Concentrate to Intensive Leach	t/day	3.4	
Gold Recovery by Gravity	%	30%	
Pre-Leach Thickener			
Not Required			
% Solids Feed to Leach		48%	
Leaching			
Pre-aeration Residence Time	h	2	Test work

	Unit	Nominal Value	Source
Leach Residence Time	h	48	Test work
Number of Leach Tanks	#	8	
Sodium Cyanide Consumption	kg/t	0.6	
Lime consumption (quick lime)	kg/t	0.8	
CIP Residence Time	h	3	
Number of leach tanks		8	
Carbon concentration	kg/m^3	30	
Carbon Loading (gold)	g/t	1,250	
Carbon Processing			
Gold recovered by Leaching	grams	12,285	
Carbon Handling Capacity	t/day	9.8	
Elution operating temperature	Celsius	145	
Carbon Consumption	g/t ore	70	
Cyanide Destruction			
Feed Solution CNWAD	mg/l	350	
Discharge Solution CNWAD		<1.0	
Residence Time	h	1	
Number of Tanks	#	2	
SO_2 Consumption	g/g CNWAD	0.67	
Lime consumption (quick lime)	g/g CNWAD	7.8	
Copper Sulfate Concentration	mg/l	15	

The overall plant flow sheet is summarized in Figure 17-1. Each unit of the process plant is described in turn.

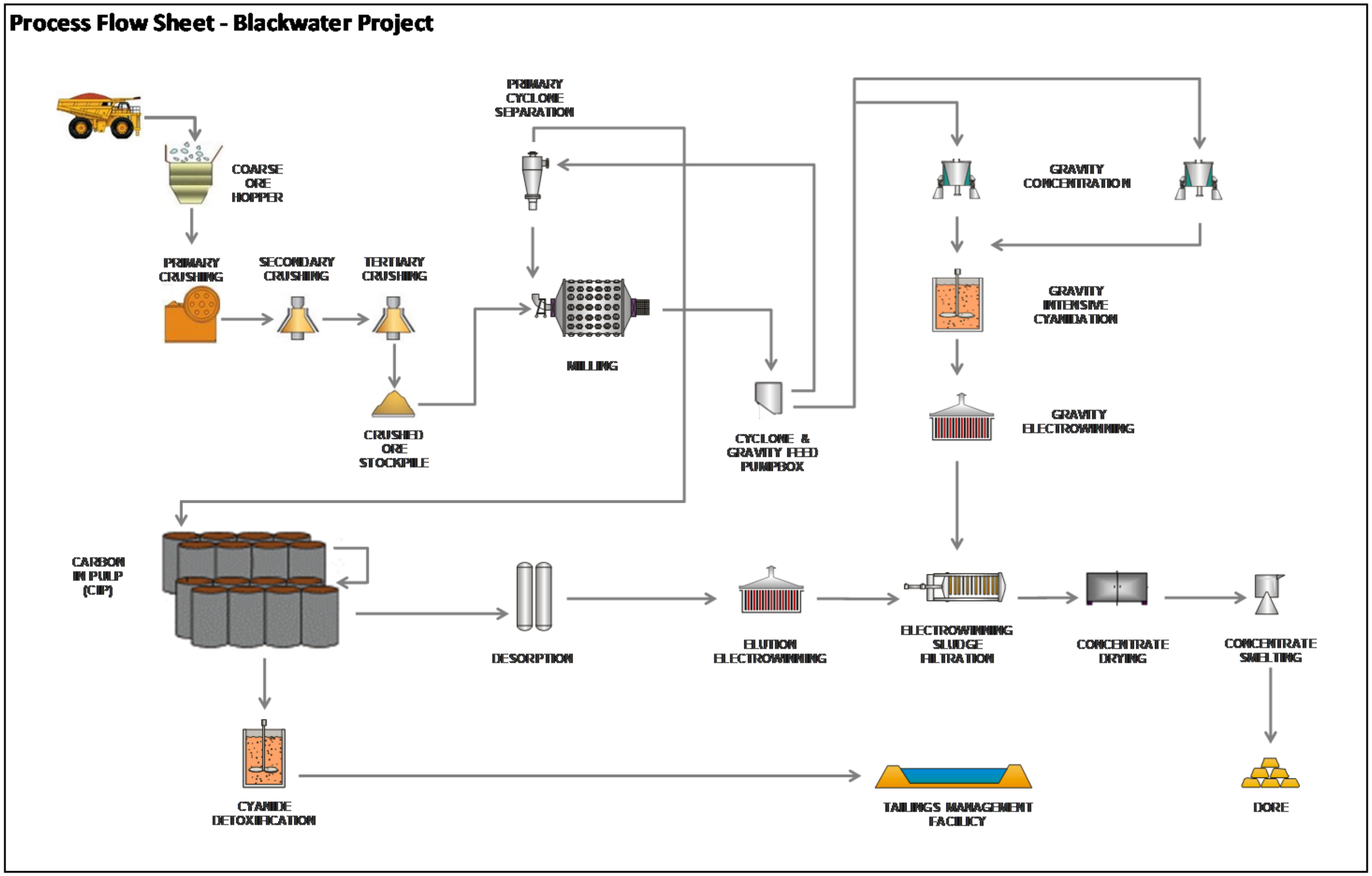

Figure 17-1 Proposed Blackwater Process Flow Sheet (source: BaseMet, 2019)

17.1 Primary Crushing

Ore will be loaded into the feed bin (360 t capacity) of the primary crusher either directly from the ore truck or from a stockpile using a wheel loader. An apron feeder, with a fines conveyor installed beneath it will feed the ore into a 250 kW jaw crusher (1.27 x 1.52 m) over an inclined vibrating grizzly to remove fines prior to the crusher. A hydraulic hammer will be installed to allow breaking of any oversize blocks.

The crushed ore will be fed to a 1.8 x 4.2 m vibrating screen and the -19 mm undersize material will report to the product conveyor. Oversize will be conveyed to the secondary cone crusher (300 kW) with a closed size setting of 44 mm and the product from the crusher will be conveyed to a 2.4 x 6 m vibrating screen where -19 mm material will report to the product conveyor. The oversize will be conveyed to a pair of tertiary cone crushers (300 kW each, the same model as the secondary crusher except for the liner) with a closed size setting of 19 mm. The crushed product will be conveyed to the final 2.4 x 6 m screen and the undersize will report to the product conveyor. Oversize will be conveyed back to the tertiary crushers. The product conveyor will transport the -19 mm ore to a covered stockpile where a tripper on the conveyor will deposit the material. A concrete tunnel under the stockpile will house the mill feed conveyor and the crushed ore will be fed onto the mill feed conveyor at a controlled rate by three vibrating feeders.

The retaining wall necessary to provide the change in height will be made using a reinforced earth retaining wall and an elevated control cabin will be installed to allow operator vision of the jaw crusher feed and the hydraulic breaker.

The crushers and screens were chosen for their proven reliability, and cone crushers and screens will be supplied as complete modules, with feed bins, feeders, product conveyors and all structural steel for the cone crushers and product conveyors, pans and structural steel for the screens. This will greatly reduce the civil construction and design work required for the circuit. The low temperatures and heavy snow fall at site will require that all crushers and screens are installed in closed buildings. The circuit layout was designed to accommodate the three screens in one building and all of the cone crushers in another. These buildings will allow the use of dry fog for dust control. All conveyors will be fitted with walkways and the belts will be covered.

17.2 Milling

Crushed ore will be fed at a design feed rate of 680 t/hr into a 7.3 x 12.5 m ball mill, with a double pinion drive (2 x 7 MW motors). The mill discharge will report to a pump box from which the slurry will be pumped to the cyclones and to the gravity recovery circuit. Both streams will have one duty pump and one standby. The ball mill will be fed with 75 mm balls and 0.5 kg/t of lime.

A cyclone cluster (four operating, two standby) will separate the P80 150 µm product and the cyclone underflow will flow directly to the ball mill. The cyclone overflow will report directly to a 20 m^2 linear screen for trash removal prior to leaching.

The gravity circuit feed pump will discharge the slurry to two vibrating screens fitted with a 2 mm deck. The +2 mm will join the cyclone underflow feeding the ball mill. The -2 mm material will feed one of two centrifugal gravity concentrators working in parallel with the tailings from the concentrators reporting to the cyclone feed pump box. Ball mill discharge will report to the cyclone feed pumps. The gravity concentrate, discharged approximately every hour, will be collected in the intensive cyanidation feed tank. This arrangement makes the gravity circuit completely independent of the milling circuit and the feed rate to the gravity circuit can be controlled to maximize recovery. It also has the advantage of minimizing the height required for installation of the cyclones.

A particle size analyser will be installed on the cyclone overflow to allow continuous control and monitoring of the grinding circuit.

The mill, cyclones, gravity separation and intensive cyanidation will be installed in one building with a 20 t overhead crane.

17.3 Intensive Cyanidation

An automated intensive leach system will be used to leach the gravity concentrate and the leach liquor will be transferred to a storage tank in the gold room prior to electrowinning. The washed residue from the leach will be returned to the ball mill feed.

17.4 Leaching

Traditionally, thickeners have been used to increase the solids concentration after milling, but experience has shown that this may be omitted provided the cyclones in the milling circuit are operated correctly.

The leach circuit will consist of eight equal sized tanks, approximately 20 m in diameter and 21 m high, with 1 m of freeboard. Each tank will be fitted with a mechanical agitator, baffles and a feed down comer. Air will be sparged into the bottom of each tank via a “witches hat”. Bypasses will be arranged to allow any tank to be taken off-line for maintenance. Lime slurry will be added to the first tank to raise the pH to 10.8 and will be used for pre-aeration to minimize cyanide usage. Cyanide will be added to the second tank and to whichever subsequent tank requires it. A cyanide analyser will be used to monitor cyanide concentrations in the tanks and dissolved oxygen meters will also be installed. Close monitoring and good instrumentation of the leach circuit is necessary to compensate for variations in the feed.

Both leach and carbon-in-pulp (CIP) tanks will be installed on a concrete containment area, in the open air with appropriate bund walls to contain 1.5 times the volume spilt from the largest tank. A small building of light, easily demountable construction will protect the agitators and the cyanide analyser from the weather.

17.5 Adsorption

Gold and silver will be recovered from the leach slurry in a self-contained carousel-type carbon adsorption plant. By using a much higher carbon concentration (50 g/L), the residence time is much shorter than the conventional adsorption circuit. The carousel-type plant occupies a much smaller footprint and with the cold temperatures experienced at site, it can be housed in a relatively small building. It will consist of six agitated tanks, 6.5 m in diameter by 10 m high with a pump cell-type inter-stage carbon screen that will allow slurry to move from one tank to the next without any change in height and will agitate the slurry. The carbon will remain in the tank until fully loaded with gold and silver, at which time the whole tank will be emptied and the carbon removed for final processing. The feed and discharge from each tank will be rotated as required to maintain the flow of slurry in counter current with the carbon via a manifold installed over the tanks that will be fitted with simple knife-edge valves.

An overhead crane system will be installed to allow easy removal of the inter-stage screens for cleaning and maintenance. The loaded carbon screen and the carbon safety screens after the adsorption system will be housed in the same heated building, thus avoiding problems with freezing.

17.6 Carbon Treatment

Carbon loaded with gold and silver will be removed from the adsorption unit, screened and washed on a vibrating screen fitted with 0.8 mm screen panels. The undersize slurry will return to the adsorption circuit. The carbon will be transferred to an acid leach column for removal of carbonates using dilute hydrochloric acid. Approximately 12.5 t of carbon will be produced per batch and the acid wash and elution column will be approximately 2 m in diameter and 8 m high for this amount (approximately 25 m^3).

After washing to remove the residual acid, the carbon will be transferred to the elution column, fabricated in carbon steel, insulated and approximately 1.8 m in diameter and 5 m high. A hot solution of sodium hydroxide and sodium cyanide (1% and 0.1% respectively) will be passed through the carbon at a temperature of 140–145º C (Celsius) to de-sorb the metal values from the carbon. The operating pressure in the column will be 0.4 MPa. Two heat exchangers and a propane-fired solution heater will be used to heat up the solution, control the temperature in the column, and the temperature of the solution exiting the column. The solution, containing gold and silver, will then flow to an electrowinning cell where gold and silver will deposit on the cathode. The solution will recycle to the elution column and this flow will continue until the gold and silver levels in the carbon have reached approximately 50 g/t each. This generally will take 12 hours and the system will then be cooled, all liquid will be transferred to the barren solution tank, and the carbon will then be pumped from the column as a water slurry and transferred to the feed hopper of the carbon regeneration kiln. This will be a rotary furnace, heated using propane, operating at 650–700ºC. The regenerated carbon will be quenched in water, passed over a sizing screen to remove any fines and then be transferred back to the carbon adsorption unit. Make-up carbon will be dumped from its container (a big bag) into an agitated tank fitted with an agitator, and after an hour or agitation in water will be passed over the sizing screen to remove any fines. It will join the main carbon inventory.

The carbon treatment unit will be installed in the same building as the adsorption system.

17.7 Electrowinning and Refining

Solution from the elution column will flow directly to two lines of two electrowinning cells, operating in parallel, where gold and silver will be deposited on stainless steel wire mesh cathodes. The solution exiting the cells will flow to the barren solution storage tank, from which it will be pumped back to the elution column. The barren solution tank will be of sufficient size to accommodate all the solution from the elution column and electrowinning cells.

Every two days, the first cells in line will be drained and the cathodes will be pressure washed to remove the precious metal sludge and this sludge, together with any sludge accumulated in the bottom of the cells, will be washed into a filter press. The filter cake will be dried overnight in a tray drier and then smelted in an induction furnace with the addition of silica and borax fluxes. The furnace will be fitted with an extraction hood and any fume will be collected in a bag filter unit. Excess slag will be poured into slag moulds and the molten metal will be poured into bar moulds. The doré bars will be cleaned, numbered and weighed prior to shipping. The slag will be broken up, any large pieces of metal recovered manually, and the rest will be recycled to the mill.

The electrowinning cells will be equipped with extraction hoods and fans and the air will be scrubbed for removal of gases prior to discharging.

17.8 Cyanide Destruction

Cyanide destruction will be carried out in two tanks in series, the piping will allow each tank to be bypassed if required, and each tank will provide a one-hour residence time. The tanks will be 11 m in diameter and 12 m high. The slurry will flow from the carbon safety screens in the adsorption unit directly to the first of the tanks. A high-power agitator will provide good gas dispersion of the air sparged into the tank.

Sulphur dioxide will be produced by burning sulphur in a burner unit which will include storage for sulphur, a melting system, a sulphur burner and quench tank to cool the combustion gases.

Some liquid from the quench vessel will be pumped into the cyanide destruction tanks with the major part of the SO_2 being sparged into the agitated tank in the gaseous form. The sulphur burning unit will be supplied as a complete, skid-mounted, turnkey unit.

A unit to prepare as sodium metabisulfite (SMBS) solution will be installed in the reagent building as a back-up. Copper sulphate solution to catalyse the reaction will be available to add to the reactor to ensure an adequate level of copper in solution (15 mg/L) but as most of the ore contains sufficient copper to provide this concentration, it will only be added when required (about 70% of the ore should provide enough copper). Lime slurry will be metered in under pH control. The anticipated consumption of sulphur will be 140 kg/hr, that of copper sulphate a maximum of 42 kg/hr, with an average addition over an extended time period of 12.5 kg/hr and lime addition of 700 kg/hr. The tanks will be installed in the open air, in the same containment area as the leach tanks. Instrumentation will include pH control, dissolved oxygen monitoring and HCN alarms and the final effluent will be monitored continuously for weakly acid-dissociable cyanide (CN_{WAD}). Upon discharge from the final tank, the slurry will be pumped to the tailings storage facility.

17.9 Reagents and Services

The makeup and storage systems for sodium cyanide, copper sulphate, SMBS and lime will be housed in a building with sufficient area to also provide storage. Lime will be stored in a silo next to the reagent building and the slaking and storage tanks will be housed inside the building. The sulphur dioxide unit, with a sulphur store, melting tank and burner will be housed in a separate building, adjacent to the cyanide destruction tanks. All reagents and materials will be delivered by road.

Compressors for instrument air and for general use and the blowers for the leach circuit, water pumps and fire pumps will also be installed in the reagent building, water tanks for process water, raw water, fire water and potable water, insulated and fitted with heaters to prevent freezing will be installed adjacent to the reagent and service building.

Electrical power will be obtained via a 240 kV power line as described in Section 18.

Process water will be reclaimed from the TSF using a floating reclaim barge. Seal water, water for use in the elution circuit and potable water will be obtained from wells.

Propane will be delivered to site as a compressed gas–liquid and stored in a pair of supplier-managed on-site tanks. Storage capacity on site will be 30 days.

17.10 Process Control Philosophy

Field instrumentation will input date into programmable logic controllers (PLC) which will be monitored by a process control system (PCS). The PCS will be configured to provide outputs to alarms, control the function of selected process equipment, and provide advisory comment to the plant operators and will be located in a central control room. The crushing plant will have its own dedicated control room, with duplicate displays in the central control room. As Stages 2 and 3 are developed, a single central control room will contain the PCS system for all three process lines.

18 PROJECT INFRASTRUCTURE

18.1 Project Layout

The overall Project facilities and major infrastructure cover the mine site area, TSF, camp site, main access road, and water supply system from Tatelkuz Lake. A layout plan is presented in Figure 18-1.

18.2 Road and Logistics

18.2.1 Access Roads

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.

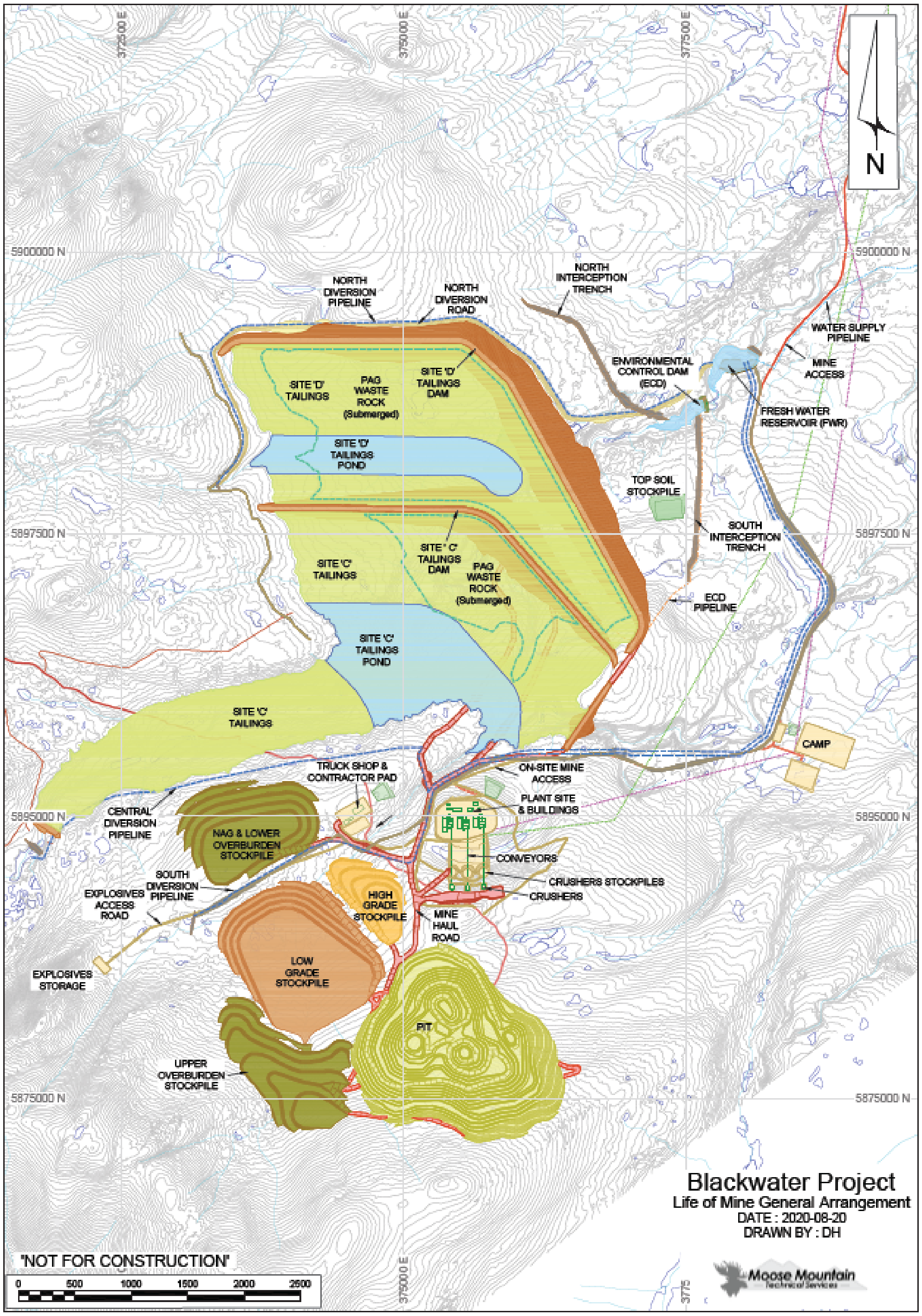

Figure 18-1 Project Site General Arrangement

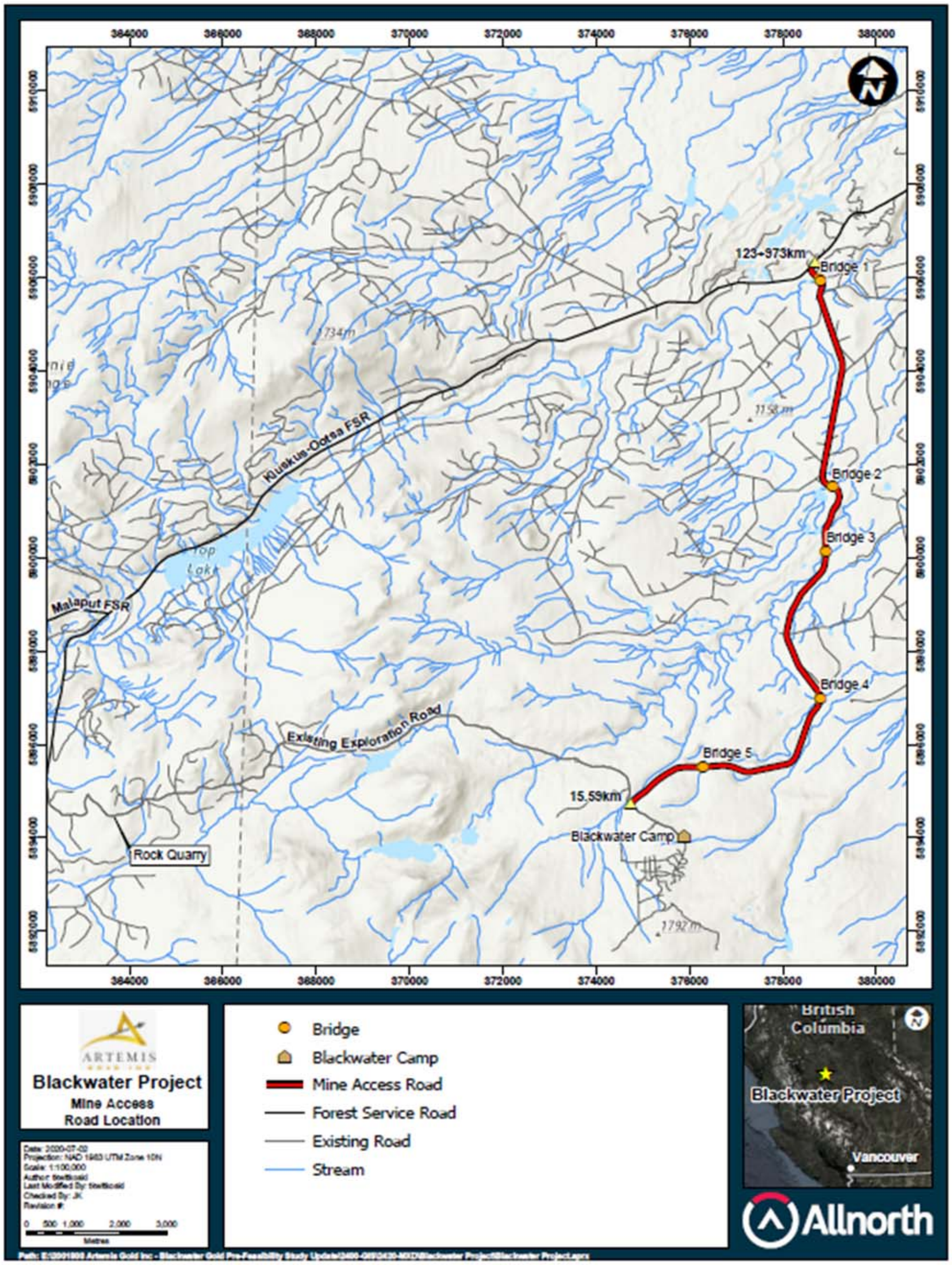

Figure 18-2 Blackwater Mine Access Road

The new road will be designed to give year-round access for heavy equipment both during construction and operation. For Stage 1, it will be 5 m wide with passing places every 500 m. When the mine is in operation, prior to Stage 2 constructions, the road will be widened to 10 m.

Five bridges will be needed, and these will be permanent structures, 5 m wide. They are relatively short spans (Table 18-1). Drainage will be provided where necessary using 600 mm culverts. Local borrow pits will be used to provide aggregate for the road surface, with high fines aggregate being used for the road surface.

Table 18-1 Bridge Required for New Access Road

	Location	Length	Description
Bridge 1	0.490 km	18.288 m	Steel concrete composite on precast spread footing
Bridge 2	5.190 km	13.0 m	Slab girder bridge on precast spread footing
Bridge 3	6.715 km	18.288 m	Steel concrete composite on precast spread footing
Bridge 4	10.320 km	14.0 m	Slab girder bridge on precast spread footing
Bridge 5	13.790 km	12.0 m	Slab girder bridge on precast spread footing

Some of the road passes through areas which that been clear felled; other areas will require that trees are felled. It is expected that the costs for cutting the trees will be met by the sale of the timber. It is estimated that the road can be built in one season, March to September.

Allnorth Consultancy prepared the road design and estimated the capital cost for the road.

18.2.2 On Site Roads

On site roads of approximately 10 km in length will be necessary to provide access to the plant, accommodation, truck-shop and explosives store. These will be at least 10 m wide to allow two-way traffic and wider where mine trucks will travel. An access road will also be needed to the pumping station on Tatelkuz Lake.

18.3 Geotechnical Investigations

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, 2013c; KP, 2013e). 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. Drillhole, ground geophysics, and test pit locations were adjusted as the program progressed and site conditions became better understood. These site investigation programs were supplemented with additional site investigation work completed in 2019 in the vicinity of the proposed TSF and mine infrastructure areas.

18.3.1 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 ODEX drilling techniques with standard penetration tests (SPTs) in the surficial materials and diamond drill coring (HQ3) 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 (SCPT) 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 PVC 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.

18.3.2 Site Stratigraphy

The stratigraphy of the surficial materials and bedrock from surface downward is as follows:

- Holocene deposits;
- Fraser glaciation deposits, including the follow sub-types:
    - Glaciofluvial deposits;
    - Glacial ablation till deposits;
    - Glacial lodgement till deposits;
    - Undifferentiated till deposits;
    - Glaciolacustrine deposits
- Interglacial fluvial deposits;
- Older glacial deposits (predominantly glacial till from an earlier period of glaciation);
- Reworked regolith (reworked completely weathered bedrock);
- In-situ regolith (completely weathered bedrock);
- Intact bedrock.

18.3.3 Tailings Storage Facility

The Site C Main Dam and Site D Main Dam alignments were established to optimize use of the natural topography of the Davidson Creek watershed, allowing for efficient and long-term storage of mine waste. The foundation conditions at the Site D Main Dam are characterized by a surficial glacial sequence ranging up to approximately 100 m thick overlying bedrock. Surficial materials at the north abutment are particularly thick. The completely weathered bedrock horizon ranges in thickness from approximately 2 to 30 m. Static groundwater levels range from 7 to 32 m below ground surface, mirroring the surface topography. The groundwater depth is consistent with the saturated zone identified by seismic refraction survey lines.

The dominant surficial material type at the Site C Main Dam site is low-permeability glacial till ranging in thickness from approximately 20 to 60 m with interbedded intervals of glaciofluvial and glaciolacustrine materials. A completely-weathered bedrock horizon was identified underlying the glacial deposits prior to the intact bedrock horizon. Static groundwater levels are relatively consistent in the vicinity of Davidson Creek with a piezometric elevation of approximately EL. 1225 m, coinciding with the approximate elevation of Davidson Creek in the area.

The Site C West Dam will be located at the west side of TSF Site C. Drillholes and seismic lines were completed in this area to investigate the subsurface conditions and optimize the dam alignment. The dam location was shifted downstream during the 2014 FS to its current position approximately 100 m to the northeast; seismic refraction surveys indicated this was a more favourable site. The surficial materials are approximately 5 m-thick at this location comprising dense fluvial sand and gravel deposits with more than 15% fines content. Bedrock consists of felsic tuff and felsic lapilli tuff. The completely weathered bedrock horizon was not encountered. The static groundwater level is shallow, at approximately 3 m.

The environmental control dam (ECD) and interception trenches will be located approximately 1 km downstream of the Site D Main Dam. Conditions in this area are characterized by a surficial material sequence ranging from 24 to 108 m thick that is deepest in the centre of the Davidson Creek valley. The completely weathered bedrock horizon in this area ranges from approximately 2 to 30 m thick. The static groundwater level is a reflection of the surficial topography and ranges from 21 to 53 m.

Evaluation of potential seepage pathways during 2013 identified that:

- Drilling and mapping of the surficial glaciofluvial channel deposits within the Site C and Site D basins found they were not continuous or hydraulically connected to inter-glacial fluvial deposits;
- Interglacial fluvial deposits were found to be localized, discontinuous, and absent or thin in drill core;
- Assessment of the hydraulic conductivities of the highly weathered bedrock yielded values of 3 x10-5 to 7 x10-8 m/s, indicating that the highly weathered bedrock has the potential to be a possible seepage pathway beneath the TSF and would daylight downstream of the Site D Main Dam. Subsequent seepage control measures were

incorporated into the design to mitigate this using the ECD and seepage interception trenches;

- The primary cut-off trench for seepage control will be constructed beneath the TSF dams. Seepage from the Site C Main Dam will be captured and contained in TSF Site D. The Site C West Dam will require a 5 m cut-off trench to key the dam into bedrock.

18.3.4 Overburden and Waste Rock Stockpile Facilities, Low-Grade Ore Stockpile

Geotechnical conditions in the area proposed for the overburden and waste rock and low-grade ore stockpiles are characterized by surficial materials ranging from 18 to 75 m in thickness, being thickest at lower elevations. Several eskers and localized kames and ablation till were identified in the footprint area. Static groundwater levels range from 3 to 4 m below surface and mirror the surficial topography.

18.3.5 High-Grade Stockpile

Sand and gravel deposits comprising a meltwater corridor and kame complex were identified in the proposed stockpile footprint. The foundation materials are characterized by glaciofluvial surficial deposits up to 18 m thick overlying the highly weathered andesitic bedrock. The static groundwater level is at approximately 13 m depth below ground surface.

18.3.6 Plant Site

The plant site will be located on a topographic high between the open pit and the TSF at approximately elevation 1,433 m.

The surficial materials at the plant site are primarily glaciofluvial deposits composed of sand and gravel and interbedded silty sand and silty gravel materials. The deposits range in thickness from 1 to 44 m, being thinnest across the topographic high of the plant site and increasing on the flanks of the hill. The material is dense to very dense below 5 m depth. The static groundwater table was encountered at approximately 7 m below ground surface in the sand and gravel unit.

Andesite bedrock is highly weathered for approximately the first 5 m above moderately to slightly weathered bedrock. The RQD is greater than 60% and RMR is classified as FAIR to GOOD. Rock compressive strength testing ranged from 70 to 275 MPa indicating very strong rock. Hydraulic conductivity of the rock mass was low and ranged from 1×10^{-7} m/s to 3×10^{-8} m/s.

18.4 Borrow Sources

Potential borrow source locations were identified as follows:

- Approximately 500,000 m^3 of material could be generated from excavations at the planned plant site. The excavated material would be suitable for use as backfill but may require some processing to be used as structural fill or in MSE wall construction;
- Several suitable borrow sources should be available within 2 km of the Site C Main Dam, included pit-run granular fill materials for the dam shell, fine-grained glacial till for the core zone, and aggregate materials that could be crushed and/or screened to produce desirable quantities and grain size distributions for engineered fill materials;
- In excess of 3 Mm^3 of materials suitable for use as sub-base material or for processing to produce other well-graded materials could be sourced from a site approximately 5 km down the proposed mine access road from the plant site, just beyond the new camp location;
- In excess of 3 Mm^3 of sand and gravel materials are available from an esker deposit located approximately 10 km from the plant site area. These materials will need to be crushed and/or screened to produce the desired quantities and grain size distributions.

Concrete aggregate suitability testing was performed on two samples, one from the plant site, and the second from the esker deposit. 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.

18.5 Tailings Storage Facility

18.5.1 Site Selection

A Tailings Alternatives Assessment (TAA) for the Blackwater Project was completed in 2015 (ERM, 2015) in response to requests from the BC Environmental Assessment Office (EAO) and Canadian Environmental Assessment Agency (CEAA). The study included evaluation of the best available technology (BAT) and best available practices (BAP) for tailings and waste rock management 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 for the Project, 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. 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.

18.5.2 Dam Hazard Classification

The Canadian Dam Association Dam Safety Guidelines (CDA, 2019) and the Part 10 Guidance Document for the Health, Safety and Reclamation Code for Mines in British Columbia 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 Project tailings dams. The tailings dams were classified by considering the potential incremental consequences of a failure.

The following minimum target design flood and earthquake levels were adopted from the CDA guidelines for a ‘VERY HIGH’ dam hazard classification for the construction and operational phases of the Project:

- IDF – 2/3 between one-in-1,000 year return period flood and the probable maximum flood (PMF);
- EDGM – 1/2 between one-in-2,475 year return period and the one-in-10,000 year return period or maximum credible earthquake (MCE).

The following minimum target design flood and earthquake levels were adopted for the closure (passive care) phase of the Project:

- IDF – PMF;
- EDGM – MCE (or 1-in-10,000-year event).

18.5.3 Tailings Characteristics

A laboratory testing program was conducted in 2013 to determine the geotechnical and physical characteristics of the tailings (KP, 2013d). The SG of the tailings solids was determined to be 2.79, and the material can be described as non-plastic sandy-silt with trace clay. The particle size distribution of the tailings sample was approximately 44% fine sand, 46% silt, and 10% clay.

Undrained settling, drained settling, and air-drying tests were carried out to provide information on the effect of initial slurry solids content on the settling and permeability characteristics of the material and the effect on water recovery and achieved density. Tests were performed for a target solids content equal to 50%.

Laboratory tests conducted to determine the consolidation and permeability characteristics of the tailings included slurry consolidometer testing, a low-stress slurry consolidation test, and a falling head permeability test (conducted on settled tailings after completion of drained settling). The tailings rheology characterized the slurry as non-Newtonian, Bingham plastic slurry typical of tailings slurries. It is fairly coarse and clean with good packing density and low rheology.

18.5.4 Facility Design

The principal design objectives for the TSF 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 Project life.

The TSF was designed to permanently store tailings, PAG1 and PAG2 waste rock, and NAG3 potentially ML waste rock that will be generated during mine operations. The TSF will comprise two adjacent sites, TSF Site C and TSF Site D. The general arrangement of the TSF is included as Figure 18-3.

The facility was designed to contain 462 Mm^3 of tailings and waste rock material and will require approximately 83 Mm^3 of construction material with approximately 95% being supplied by waste rock and overburden from development of the open pit. The dam construction materials balance is integrated with the mine plan to minimize the need for additional external bulk borrow material sources following initial site establishment and early TSF construction. Several borrow sources should be available in the vicinity of the TSF basin, included pit-run granular fill materials for the dam shell, fine-grained glacial till for the core zone, and aggregate materials that could be crushed and/or screened to produce desirable quantities and grain size distributions for engineered fill materials.

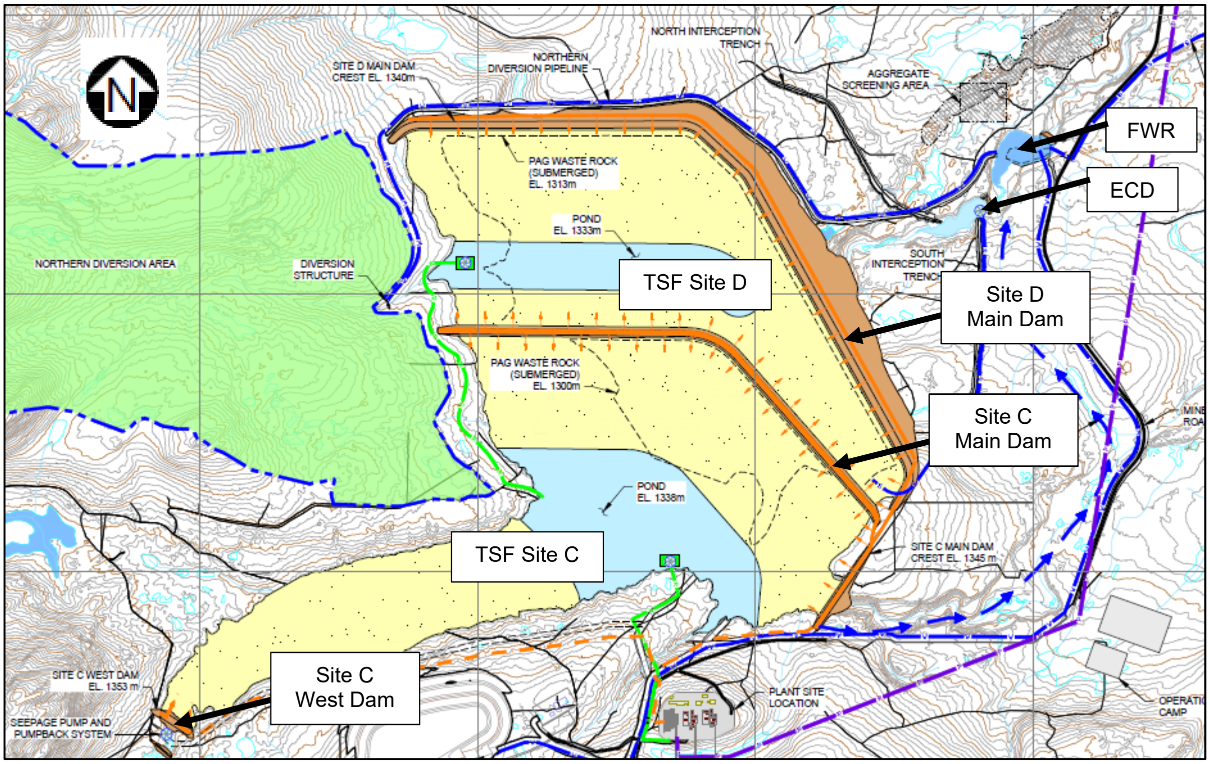

Figure 18-3 TSF General Arrangement – Year 23 (source: KP, 2020)

The TSF Site C Main Dam will be constructed to elevation 1,345 m and the Site C West Dam will be constructed to elevation 1,353 m. The TSF Site D Main Dam will be constructed to elevation 1,340 m. Specific overall features of the TSF are:

- Cofferdams, sediment control ponds, and surface water management ditches to manage water during construction by either routing water around the TSF or directing water to the TSF for collection;
- Three zoned water-retaining earth-rockfill dams referred to as the Site C Main Dam, Site C West Dam, and Site D Main Dam;
- An interim environmental control dam to capture seepage from the Site C Main Dam prior to construction of TSF Site D;
- The environmental control dam (ECD) and interception trenches to capture seepage downstream of the Site D Main Dam and direct water to the TSF;
- The fresh water reservoir (FWR) located downstream of the ECD to store fresh water for release to Davidson Creek to offset flow reductions downstream;

- Staged non-contact water diversion systems to route run-off from undisturbed areas around the TSF to Davidson Creek or the FWR;
- Collection channels that route contact water to the TSF;
- Tailings distribution system;
- Reclaim water system;
- Tailings beaches;
- Supernatant water pond;
- Designated PAG/NAG3 WRSFs within the TSF.

TSF Site C

TSF Site C will be constructed first to provide storage capacity for start-up of the process plant. The facility was designed to contain up to approximately 16 years of tailings and the first six years of PAG/NAG3 waste rock, and includes a storage allowance for the supernatant pond to provide a continuous source of process water to the mill operations.

Davidson Creek will be diverted around the initial construction area in a diversion channel constructed along the left (north) bank of Davidson Creek. Construction of the diversion channel will require excavation and temporary stockpiling of approximately 250,000 m^3 of material, which will be used during initial dam construction if the excavated materials meet fill material specifications. A diversion berm, with a height of approximately 10 m, will be located approximately 500 m upstream of the Site C Main Dam centerline to facilitate diversion of Davidson Creek into the diversion channel. A cofferdam and downstream sediment control pond will be constructed nearer to the initial construction area prior to site preparation and initial embankment construction.

Construction of the Stage 1A dam 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. The dam will be built to elevation 1,250 m by the end of Year -2, requiring placement of approximately 0.8 Mm^3 of fill material sourced from nearby external borrow sources. Construction of the Stage 1B dam will begin in early Year -1, immediately following completion of Stage 1A. The dam will be raised to elevation 1,274 m to provide sufficient capacity for a start-up pond up to 2 Mm^3 and to impound tailings and PAG/NAG3 waste rock generated during the first year of operations, with additional capacity to contain the IDF. Construction of Stage 1B requires placement of approximately 3.6 Mm^3 of fill material, the majority of which will be supplied from stripping of the open pit.

The Site C Main Dam will be raised annually through Year 15 using centerline construction methods to an ultimate elevation of 1,345 m to maintain the storage capacity of the TSF. Sustaining embankment construction will require placement of approximately 25 Mm^3 of fill material. The dam raise schedule includes construction of three significant downstream step-outs of the shell zone (Zone C), which are designed to support several staged vertical raises of the embankment. Each raise is designed to provide enough storage for the following year of operations, a sufficient supernatant pond allowance ranging from 2 to 10 Mm^3 (which is aligned with the staged capital expansion of the mill facilities), and additional capacity for storage of the IDF. The Site C Main Dam is designed as a zoned earthfill-core rockfill dam with appropriate filter zones to prevent piping and internal erosion from developing within the adjacent core zone as shown on Figure 18-4.

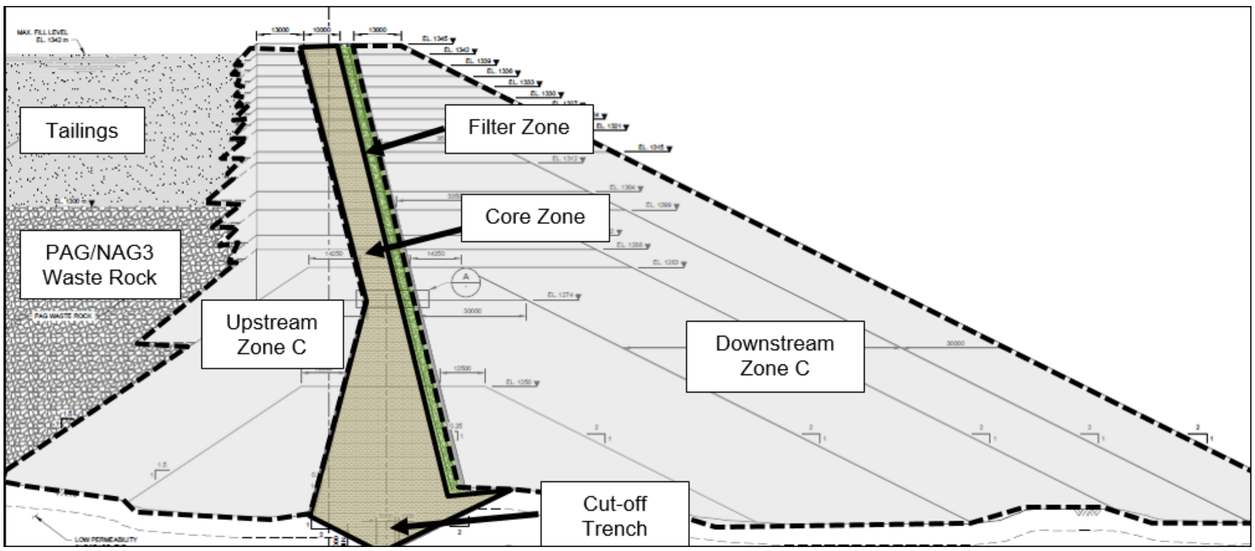

Figure 18-4 Site C Main Dam Typical Section (source: KP, 2020)

Tailings will be discharged into TSF Site C from one or more points on the west side of the facility with PAG/NAG3 waste rock disposed of directly upstream of the Site C Main Dam during the first six years of operations up to an elevation of approximately 1,300 m to enhance stability on the upstream side of the dam. The supernatant pond will form near the interface of the two disposal areas to allow for efficient saturation of the waste rock voids to meet geochemical objectives. The Site C West Dam will be constructed in a single stage to elevation 1,353 m during Year 6 to constrain the western extent of TSF Site C. Construction will require placement of approximately 0.2 Mm^3 of fill material sourced from nearby external borrow sources. The Site C West Dam is designed as a zoned earthfill-core rockfill dam with similar filter zones to prevent piping and internal erosion from developing within the adjacent core zone as shown on Figure 18-5.

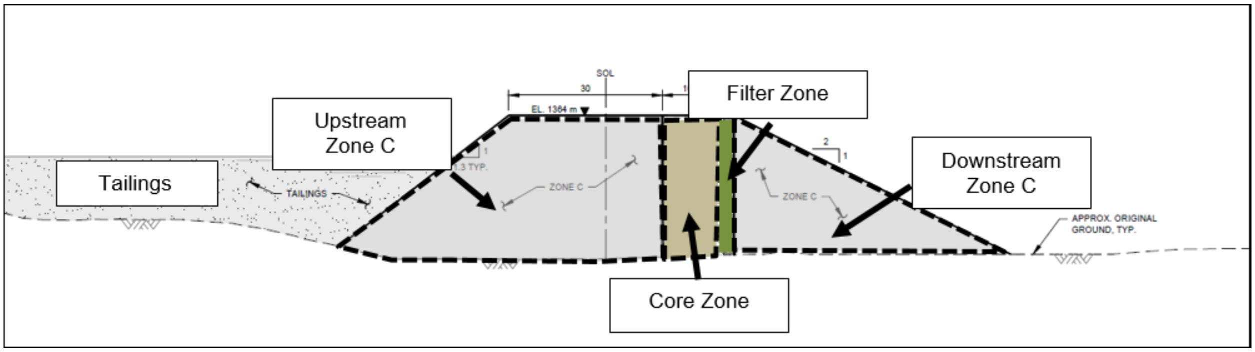

Figure 18-5 Site C West Dam Typical Section (source: KP, 2020)

The tailings distribution system will be extended along the crest of the Site C Main Dam during Year 6 to allow for discharge of tailings from the dam crest beginning in Year 7 to cover the submerged PAG/NAG3 WRSF and manage the location of the supernatant pond as shown on Figure 18-6.

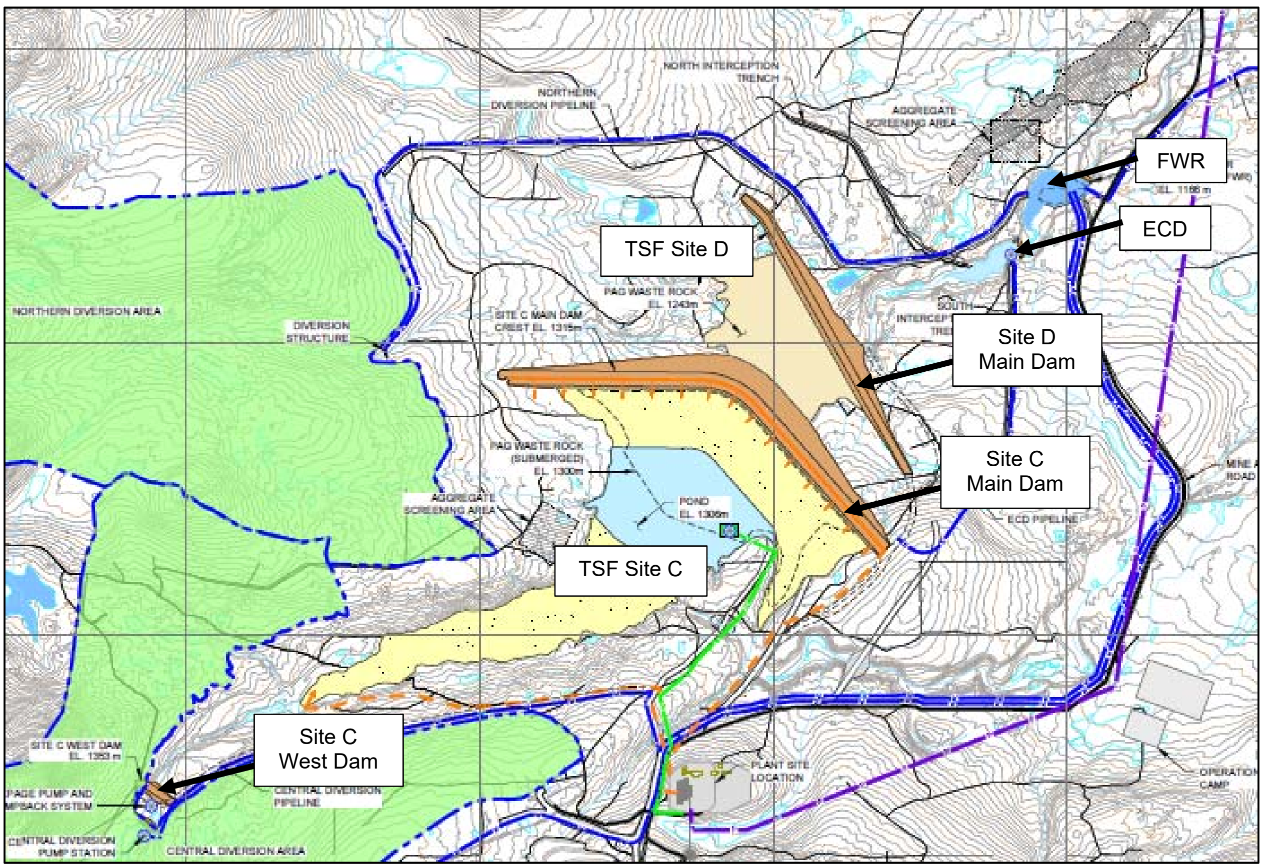

Figure 18-6 TSF General Arrangement – Year 7 (source: KP, 2020)

Tailings will be discharged into TSF Site C from the Site C West Dam throughout Years 7 to 16 to form a long sloping beach from the west side of the facility, designed to manage the position of the supernatant pond located approximately 3 km to the east as shown on Figure 18-7.

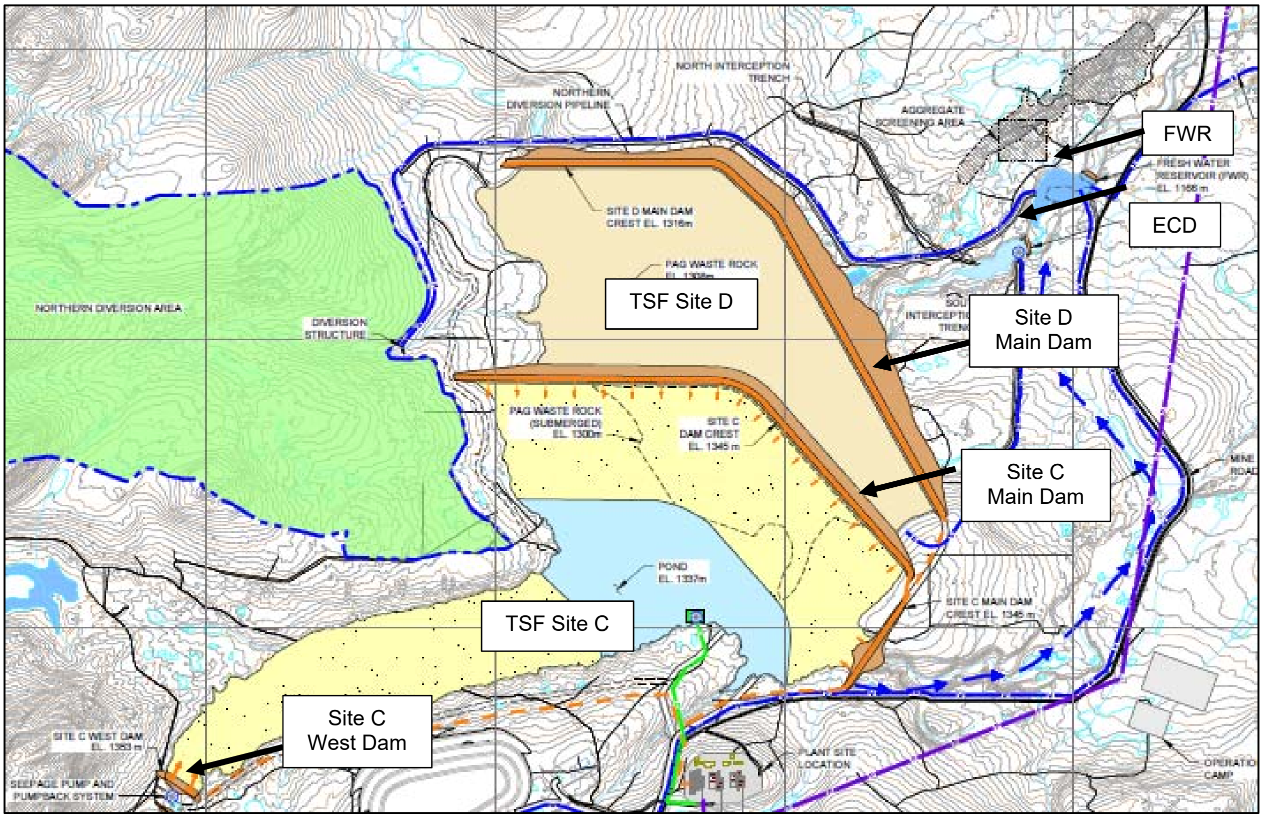

Figure 18-7 TSF General Arrangement – Year 16 (source: KP, 2020)

TSF Site D

TSF Site D will be constructed adjacent to and downstream of TSF Site C to provide additional storage capacity for PAG/NAG3 waste rock and tailings. The facility was designed to contain PAG/NAG3 waste rock generated during mining between Years 7 and 18 and up to approximately six years of tailings beginning in Year 17 when TSF Site C reaches design capacity. The design includes a nominal pond storage allowance beginning in Year 17 to allow for recycling of process water to TSF Site C and sufficient capacity for storage of the IDF.

Smaller upstream tributaries will be diverted around the initial construction area for the Site D Main Dam prior to construction of a cofferdam and downstream sediment control pond nearer to the initial construction area. Construction of the Stage 1 dam will commence in Year 5 following completion of the site establishment and construction of the necessary water management design measures. The embankment foundations will be cleared and stripped in preparation for fill placement for each stage. The dam will be built to elevation 1,240 m by the end of Year 6, requiring placement of at least 3.7 Mm^3 of fill material sourced from nearby external borrow sources and supplied from the open pit.

Filling of TSF Site D will begin in Year 7 following two years of initial construction as shown on Figure 18-6. TSF Site D will be used between Years 7 and 16 for disposal of PAG/NAG3 waste rock generated during mining. The Site D Main Dam will be raised annually by the centreline method reaching an elevation of 1,316 m in Year 13 to provide sufficient capacity for the following year of operations and capacity for storage of the IDF. The PAG/NAG3 waste rock will be disposed of directly upstream of the Site D Main Dam and downstream of the Site C Main Dam to form a WRSF between the two up to an elevation of approximately 1,315 m as shown on Figure 18-7. The waste rock disposal area in TSF Site D will gradually buttress the Site C Main Dam, enhancing stability, and the waste rock material contained in TSF Site D will be gradually saturated on an annual basis, by direct precipitation and other inflows, to achieve geochemical objectives.

The Site D Main Dam will be raised by the downstream method to a final design elevation of 1,340 m during Years 14 and 15 to provide sufficient capacity for storage of tailings and PAG/NAG3 waste rock between Years 17 and 23. Sustaining embankment construction will require placement of approximately 50 Mm^3 of fill material. The dam raise schedule includes construction of three significant downstream step-outs of the shell zone (Zone C), which are designed to support several staged vertical raises of the embankment. The Site D Main Dam is designed as a zoned earthfill-core rockfill dam with appropriate filter zones to prevent piping and internal erosion from developing within the adjacent core zone as shown on Figure 18-8.

The tailings distribution system will be extended along the crest of the Site D Main Dam during Year 16 to allow for discharge of tailings from the dam crest beginning in Year 17 to cover the submerged PAG/NAG3 WRSF. Process water recovered following discharge of tailings to TSF Site D will be pumped to the supernatant pond in TSF Site C for reuse in ore processing.

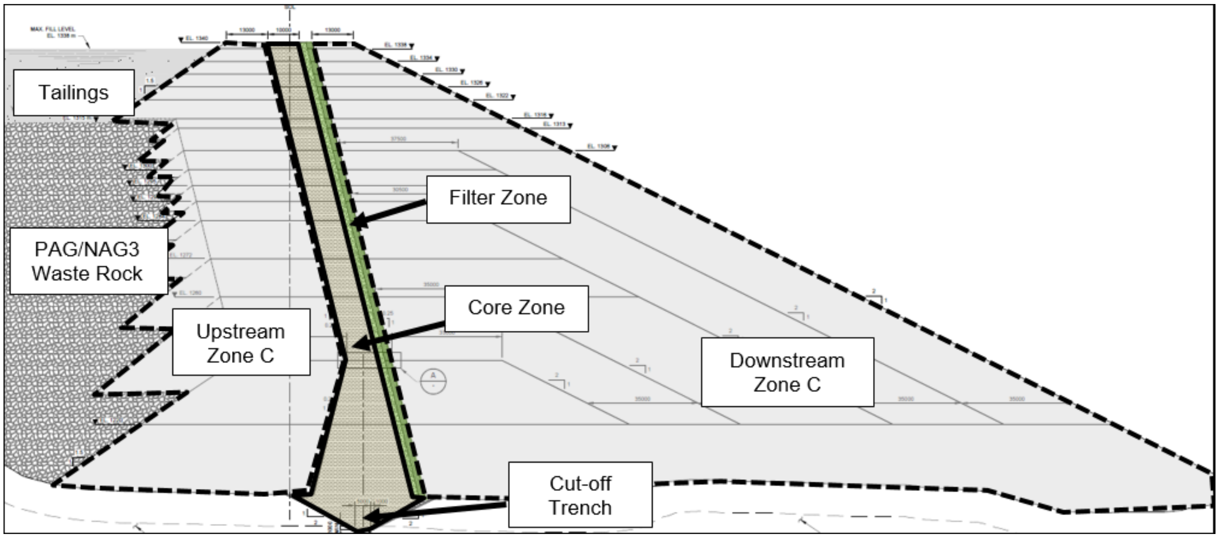

Figure 18-8 Site D Main Dam Typical Section (source: KP, 2020)

18.5.5 Seepage

Seepage will be controlled primarily by the low-permeability core zone constructed prior to the development of the tailings beach, the cut-off trenches, and the low-permeability 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.

Special design provisions incorporated into the tailings dam design to minimize seepage losses include the development of extensive tailings beaches to isolate the supernatant pond from the dam, embankment drainage collection systems, and toe drains at the downstream toe of the dams to reduce seepage gradients. Additional seepage collection ditches constructed along the toe of the embankments will collect seepage and surface runoff and direct the flow to the pumpback systems.

Secondary seepage collection at the ECD will be achieved by constructing a collection dam approximately 1 km downstream at a topographic low point in Davidson Creek. A pumpback system will manage seepage and storm water inflows. Recovered water will be pumped to TSF Site D, and the collection pond will be kept dewatered to the maximum extent practicable. Seepage through this dam will be captured in an embankment drain system and sump and be pumped back to the ECD pond.

Two seepage interception trenches, one on each side of Davidson Creek, will be excavated through the surficial sand and gravel terraces downstream of the Site D Main Dam and will report to the ECD pond. The locations of the seepage interception trenches are based on the results of geotechnical drilling along the proposed alignments. The trenches will be excavated and keyed into the low-permeability overburden horizon and will be approximately 3.3 km long and typically 5 to 15 m deep.

Groundwater monitoring wells have been installed in the downstream areas below Site D and could be used to locate recovery wells, if required, to recover any foundation seepage. Additional monitoring wells will be installed as required before TSF Site D is commissioned.

The Site C West Dam seepage will be controlled in the long-term by selectively discharging tailings to hydraulically separate the supernatant pond from the West Dam. Any seepage from the dam will be collected and recycled back to the TSF.

18.5.6 Tailings Distribution

Tailings from the process plant will be thickened and delivered by gravity through a pipeline from the thickener to either TSF Site C or TSF Site D. The tailings pipelines are sized to ensure gravity flow for the entire duration of the mine life while dissipating as much energy head through friction as possible. Expansions to the tailings distribution system will occur coinciding with capital expansions to the mill facilities and provide sufficient tailings distribution capacity at each stage of mine development. An additional pipeline extending to TSF Site C will be constructed to allow for emergency discharge of tailings to the TSF in a location that is a sufficient distance from the reclaim water intakes to avoid issues with suspended sediment in water reclaimed to the mill.

18.5.7 PAG and NAG 3 Disposal Area

A filling schedule was developed for each site that takes into account the storage characteristics of the facility and includes the approximate rate of rise of the tailings and waste rock horizon, supernatant pond allowance, and IDF freeboard.

The PAG disposal area will be developed at the same or similar rate of rise as TSF filling level but will be several metres higher to provide a dry, stable placement surface for truck traffic. The design objective for the PAG area is to flood the waste rock within one year of placement. The maximum elevation of the WRSF will remain at an elevation where it can be flooded by the supernatant pond in the case of premature closure. At closure, this WRSF will be covered with overburden (in the case of premature closure) or submerged below the final closure tailings elevation.

The PAG disposal area from preproduction through Year 6 of operations will be within TSF Site C and adjacent to the Site C Main Dam. The disposal area will expand as a fill platform with overall slopes at angle of repose. The PAG disposal area beginning in Year 7 will be within TSF Site D. Waste placement will commence near the upstream zone of the TSF Site D Main Dam below an elevation of approximately 1,240 m and will extend northwest up the valley as a lobe. The fill platform will rise slightly below the Site D Main Dam elevation until Year 17 when tailings discharge to TSF Site D commences. The surface of the PAG disposal area will be progressively covered by tailings during low-grade ore processing when waste rock production has ceased.

18.5.8 Monitoring

Geotechnical instrumentation will be installed along representative instrumentation planes within the Site C West Dam, Site C Main Dam, Site D Main Dam, and ECD. The instrumentation will be installed during construction and maintained or replaced over the life of the Project. The geotechnical instrumentation will consist of vibrating wire piezometers, slope inclinometers, settlement and movement monitoring points, and will be installed within the foundations, embankment fill, and on embankment crests.

Instrumentation monitoring will be carried out routinely during construction and operations. Daily measurements will be taken and analyzed during construction to monitor the response of the embankment fill and the foundation from the loading of the embankment fill. The operational monitoring systems will be connected to an automated data acquisition system that provides real-time access to the monitoring data.

18.6 Water Management

18.6.1 Objective

Water within the Project area will be used by collecting runoff from the mine site area and recycling process water to the maximum practicable extent. Site runoff water will be collected and stored within the TSF and used to inundate the PAG/NAG3 waste rock and tailings solids to limit the potential for ARD and ML. Water will be stored in the supernatant ponds within the TSF and recycled to the mill for use in the process. The water supply sources for the Project are as follows:

- Runoff from the catchment areas above the TSF;
- Direct precipitation onto the TSF and runoff from the mine site facilities;
- Water recycled from the TSF supernatant ponds;
- Groundwater from open pit dewatering and depressurization;
- Water extracted from two wells east of the camp area for potable and firewater use;
- Fresh water pumped from Tatelkuz Lake to mitigate flow reductions in lower Davidson Creek for downstream fisheries;
- Runoff water from undisturbed areas diverted around the mine facilities to mitigate flow

reductions in lower Davidson Creek.

18.6.2 Operational and Closure Water Management

The construction, operations, and closure water management strategies for the Project 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. All site drainage during operations and closure will drain by gravity to the TSF. Virtually all seepage from the TSF and WRSFs will also be collected and directed to the TSF. This simplifies water management, spill control, and closure in addition to providing water for the process.

The water stored in the TSF Site C start-up pond will serve as the primary process water source at the start of mill operations and an adequate volume of water storage within TSF Site C will be maintained throughout operations to provide a continuous source of water for mill operations. Runoff water that accumulates in TSF Site D beginning in Year 5 will be conveyed to TSF Site C (via the pump system at the interim Environmental Control Dam) as necessary to control the rate of inundation of PAG/NAG3 waste rock and to maintain sufficient freeboard to manage the IDF. Once tailings deposition in TSF Site D commences in Year 17, process water conveyed with the tailings slurry will be transferred from TSF Site D to TSF Site C pond prior to being reclaimed to the mill to support ore processing.

Additional makeup water, if required, will be provided from undisturbed areas up-gradient of Project facilities. The Southern, Central, and Northern Diversions will be operated in a manner that allows for diversion or collection of water from these large undisturbed catchment areas in order to manage the mine site water balance with the target range and maintain flows in Davidson Creek at or above the defined instream flow needs.

Groundwater inflow and surface runoff to the open pit, including water from the vertical depressurization wells, will be collected and conveyed to TSF Site C or treated to remove elevated metal concentrations and discharged from site.

The pit dewatering system will be decommissioned in Year 18 and the pit will begin to fill with water while the low-grade ore is processed through the mill from Year 18 to 23. Once mill operations cease in Year 23, the surplus inflow to TSF Site D (inflow minus losses) will be pumped to the open pit to accelerate pit filling and associated flooding of PAG rock exposed in the ultimate pit walls. Once the open pit is full, excess flows will be conveyed to the downstream receiving environment either through the TSF closure pond or via treatment systems, if required. Subsequently, the TSF Site C pond will overflow via the closure spillway and discharge channel to a plunge pool in Davidson Creek downstream of the ECD.

18.6.3 Water Management Systems

Environmental Control Dam

The primary seepage collection point downstream of the TSF will be located approximately 1 km downstream at a topographic low point in Davidson Creek. The collection pond will be created by constructing an approximately 12 m high dam (ECD) across Davidson Creek. The dam is designed to contain continuous seepage and runoff from events up to the one-in-100-year, 24-hour storm. A spillway is designed to pass the one-in-200-year, 24-hour storm. The pond will be fed by two interception trenches. The primary pumpback system at the ECD is designed to maintain the pond at a minimum water level. The ECD will have an embankment drain system, seepage collection sump and monitoring device, and secondary pumpback system to collect and recycle seepage.

Water Reclaim System

Water reclaimed from the supernatant pond at TSF Site C will be delivered to the reclaim 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 reclaim water system will initially comprise a barge-mounted pump station and reclaim water pipeline. The pipeline will initially consist of a combination of 20-inch (500 mm) diameter HDPE (DR7.3 and DR11) pipe and 18-inch (450 mm) diameter HDPE DR 17 pipe. The initial barge-mounted pump station will be equipped with two 300 HP vertical turbine pumps sized to deliver approximately 900 m^3/h of reclaim water. The reclaim barge will be anchored on the southern side of the TSF Site C supernatant pond throughout operations, and tailings will be selectively discharge to the facility to maintain the location of the supernatant pond.

The reclaim water system in TSF Site C will be twinned in Year 5 and a third parallel system will be added in Year 10, 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 Site D in Year 16 in preparation for recycling process water resulting from tailings deposition within that facility. The reclaim pipeline will comprise a combination of 36-inch (900 mm) diameter HDPE DR11 pipe and 26-inch (700 mm) diameter HDPE DR21 pipe to convey flow from TSF Site D to TSF Site C.

Low-Grade and High-Grade Stockpiles

The low-grade and high-grade stockpiles will be developed over a prepared low-permeability foundation with surface water and seepage collection and monitoring systems. Drainage from these stockpiles is expected to be acidic and contain elevated metals; therefore, the drainage will be collected, neutralized, and discharged to the TSF.

Overburden and NAG Disposal Areas

The Overburden and NAG WRSF layouts were refined to minimize surface water control requirements. Foundation drains will be installed in areas of existing drainage lines or when excessive seeps or springs are encountered during clearing and grubbing. Non-contact surface water will be diverted around the WRSFs during operations and closure and will be field-fit with the advancing fill platforms. Water that infiltrates through the WRSFs will be collected in ditches near the toe of the WRSFs and routed to a sediment basin before discharge to the TSF.

Southern Diversion

The Southern Diversion will be located up-gradient of TSF Site D and will be constructed during preproduction to allow for diversion of upstream flows around the mine infrastructure area and the TSF. The Southern Diversion intake structure will consist of a 3.5 m high concrete structure to provide submergence to the water conveyance pipeline. The 10-inch (250 mm) diameter HDPE DR21 water conveyance pipeline is sized to convey the estimated maximum mean monthly flow of 0.1 m^3/s (100 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 Davidson Creek or the FWR, via an energy dissipation structure. The pipeline will be relocated and extended in Year 6 during construction of TSF Site D. The intake structure will include a spillway sized to convey a 200-year design peak flow of 4.1 m^3/s; assuming the water conveyance pipeline and gated sluice pipe are inoperable. The spillway will consist of a 3 m wide broad-crested weir capable of passing the design storm while maintaining 0.5 m of freeboard. The collection ditches will be at least 1 m-deep and trapezoidal shaped with a 1 m-base width and 2H:1V side slopes. Erosion resistant material will be placed over a non-woven geotextile, which will help prevent erosion of any underlying fine soils.

Central Diversion

The Central Diversion will consist of a small berm to impound water within Davidson Creek upstream of the active TSF area, skid-mounted pump systems, and water conveyance pipeline to route flows around the TSF area to Davidson Creek or the FWR. The Central Diversion intake infrastructure will initially be located near the existing exploration access road and will be relocated in approximately Year 6 to the west of the Site C West Dam following its construction. The berm will be less than approximately 5 m high and constructed of locally borrowed overburden materials. 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 16-inch (400 mm) diameter HDPE DR11 and 14-inch (350 mm) diameter HDPE DR21 pipe, which is sized to convey a flow of approximately 0.2 m^3/s (200 L/s).

Northern Diversion

The Northern Diversion will be located up-gradient of TSF Site D and will be constructed in Year 6 to allow for diversion of upstream flows around the TSF. The Northern Diversion intake structure will consist of a 3.5 m high concrete structure to provide submergence to the water conveyance pipeline. The 16-inch (400 mm) diameter HDPE DR21 water conveyance pipeline is sized to convey the estimated maximum mean monthly flow of 0.2 m^3/s (200 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 Davidson Creek or the FWR, via an energy dissipation structure. The intake structure includes a spillway sized to convey a 200-year design peak flow of 6.5 m^3/s; 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 0.5 m of freeboard. The collection ditches will be at least 1 m-deep and trapezoidal shaped with a 1 m-base width and 2H:1V side slopes. Erosion resistant material will be placed over a non-woven geotextile, which will help prevent erosion of any underlying fine soils.

18.6.4 Water Balance

A site-wide water balance model (WBM) was developed to provide input information to the study. The objectives of the WBM included:

- Determining and quantifying whether the Project will operate under a deficit or surplus condition;
- Identifying potential deficit or surplus mitigation measures;
- Determining whether the PAG waste rock voids in the TSF will be saturated within one year after placement throughout the life of the Project.

The WBM was developed in GoldSim (version 12.1), a dynamic simulation modelling software used extensively for mine water management applications. The study is based on the average monthly climate conditions summarized in Table 18-2 and did not include climatic variability at this stage of design.

Table 18-2 Average Monthly Climate Input Data

Parameter	Month												Annual
	Jan	Feb	Mar	Apr	May	Jun	Jul	Aug	Sep	Oct	Nov	Dec	
Precipitation	51	32	26	35	40	64	60	52	48	60	52	44	564
Potential Evapotranspiration (mm)	0	0	0	23	64	88	103	91	58	19	0	0	446
Temperature (C)	-7.3	-5.5	-2.7	1.6	5.6	8.6	10.8	10.3	6.7	1.7	-4.0	-7.6	-

A target operating volume for the TSF Site C supernatant pond was implemented in the model to allow for adequate reclaim water, while reducing the likelihood of surplus conditions developing, and maintaining sufficient space to meet the storm storage requirements. The target pond volume increases coinciding with planned capital expansion of the mill facilities as follows:

- 2 Mm^3 from Year 1 to 5;
- 5 Mm^3 from Year 6 to 10;
- 10 Mm^3 from Year 11 onward.

The water management logic incorporated in the WBM considers the staged development of the mine, including the expanding footprints of the various mine facilities over time, rate of PAG/NAG3 waste rock placement into the TSF, and the changes in tailings production rate and deposition location over the LOM. A high-level summary of the WBM logic and assumptions are summarized below for each of the facilities is as follows:

- TSF Site C:
  - Tailings slurry is deposited to TSF Site C throughout operations, except from Years 17 through 22 when the slurry is deposited to TSF D;
  - Reclaim water is drawn from TSF C throughout operations;
  - PAG/NAG3 waste rock is deposited to TSF C from Years 1 through 6 and water within TSF C saturates the waste rock voids before being available for reclaim as process water;
  - The interim environmental control dam is located downstream of TSF Site C to collect seepage and downstream embankment runoff from construction through Year 6, when TSF Site D is constructed.
  - Throughout production, TSF Site C is managed within a target operating range that ensures sufficient water for reclaim while reserving sufficient capacity for storm storage;
- TSF Site D:
  - TSF Site D is constructed by Year 6 and collects runoff water for one year prior to PAG waste rock deposition in the facility;
  - The ECD is located downstream of TSF Site D to collect seepage from the TSF as well as downstream embankment runoff. Flow is pumped back to TSF Site D;

- Waste rock is disposed of in TSF D from Years 7 through 18. Water previously stored in TSF Site D and any subsequent inflows first fill the waste rock voids prior to being available as surplus water;
- Tailings slurry is deposited in TSF Site D from Years 17 through 22 and water not entrained in the tailings voids is available to fill waste rock voids that are not yet saturated prior to being available for recycling as process water;
- When the TSF Site D waste rock voids are saturated, surplus water is pumped to TSF Site C;

- Open pit/plant site/stockpile areas:
  - Open pit inflows and runoff from plant site, stockpile areas, and the undisturbed areas between these facilities are directed to TSF Site C, TSF Site D, or a water treatment plant. Water is managed to meet specific objectives, including limiting a surplus or deficit conditions from developing and saturating waste rock voids within one year of deposition.

The results of the WBM indicate that the mine is expected to operate with a surplus of approximately 2 to 3 Mm^3 per year under average conditions during the first five years of operations. A surplus condition of approximately 4.5 Mm^3 is predicted beginning in Year 17; however, these excess flows can be conveyed to the open pit beginning in Year 18 to commence development of the closure pit lake.

The potential surplus conditions predicted under average conditions could be effectively mitigated through diversion of non-contact water and treatment/release of contact water from within the mine infrastructure areas (plant site, stockpiles, open pit). Flows from these areas are needed to maintain the water balance within the target operating range beginning in Year 6 due to the increase rate of mining and ore processing. The water balance model estimates indicate that the operating pond volume in TSF Site C can be managed under average conditions at approximately the target volumes described previously with seasonal fluctuations, as shown on Figure 18-9.

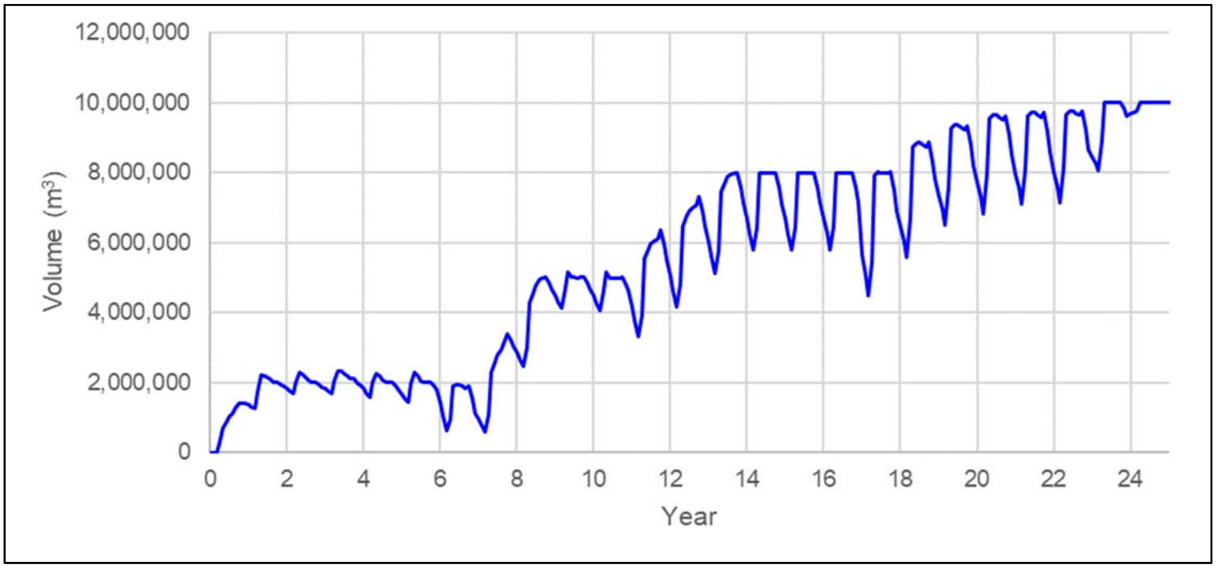

Figure 18-9 TSF Site C Pond Volume Results (source: KP, 2020)

The water balance model indicates that the PAG/NAG3 waste rock voids within TSF Site C will be rapidly flooded due to the availability of water within the adjacent supernatant pond throughout early operations. The modelling also indicates that the waste rock voids within TSF Site D can be saturated within one year of placement, but will require that runoff from mine infrastructure area or TSF Site C is conveyed to TSF Site D to supplement direct precipitation and runoff reporting directly to the facility.

18.7 Fresh Water Supply System

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. Tatelkuz Lake is fed by a watershed of approximately 395 km^2.The water will be conveyed via buried pipeline from the lake to the Project site. The fresh water supply system will be designed to supply instream flow needs (IFN) to offset flow reductions in lower Davidson Creek. The fresh water supply system includes the following components:

- Tatelkuz Lake intake;
- Fresh water supply pipeline;
- Booster pump station;
- FWR;
- Temperature and flow control system.

Water intake will be via a land-based wet-well structure, which will be a permanent two-level concrete structure constructed on the shoreline of Tatelkuz Lake. The structure is designed with a steel superstructure, an installed crane, and removable roof hatches for pump installation, removal, and maintenance. The intake pump station will initially be outfitted with one 400 hp vertical turbine pump to convey a design flow of 125 L/s (450 m^3/hr) to the FWR.

The pipeline to the FWR will initially be a combination of 14-inch (350 mm) diameter HDPE (DR7.3, DR11, and DR13.5) and 12-inch (300 mm) diameter HDPE DR17 pipe. The pipeline from the lake to the FWR will be approximately 14 km long with a total static head of 243 m.

One booster pump station, outfitted with the same 400 hp vertical turbine pump as the intake pump station, will be required to reach the reservoir. One uninstalled spare pump will be maintained on site to facilitate rapid repairs to either the intake pump station or booster pump station. The booster pump station includes a holding tank, with a five minute capacity adjacent to the booster station, as a buffer for smooth pump operation. The structure is designed with a concrete foundation, steel superstructure, an installed crane, and removable roof hatches for pump installation, removal, and maintenance.

The pipelines and pump systems will be expanded in Year 10 coinciding with capital expansion of the mill facilities. The intake pump station and booster station will be outfitted with two additional 700 hp vertical turbine pumps each to convey a design flow of 500 L/s (1,800 m^3/hr) to the FWR. The pipeline to the FWR will be twinned with a combination of 24-inch (600 mm) diameter HDPE (DR9, DR11, and DR17) pipe. One uninstalled spare pump will be maintained on site to facilitate rapid repairs to either the intake pump station or booster pump station.

The FWR will be an in-creek water body downstream of the final seepage collection point at the ECD. The reservoir will be created by constructing a dam approximately 14 m high across Davidson Creek. The embankment is designed as a water-retaining structure and will have a low-permeability core with appropriate filter zones and random fill shell zones. The reservoir will have a storage capacity of approximately 400,000 m^3 of fresh water. The size of the reservoir was based on an estimate of contingency storage required for a reasonable range of possible malfunctions in the fresh water supply system.

A spillway will route storm flows through the reservoir and around the dam. The reservoir slopes will be covered with riprap or otherwise protected to prevent sediment-laden water from being released downstream. Appropriate sediment control BMPs will be implemented on natural drainages entering the reservoir to ensure the success of the plan.

Water will be discharged from the reservoir to Davidson Creek through two concrete-enclosed pipelines near the downstream toe of the dam: a 150 mm diameter pipe to supply instream flow needs and 610 mm diameter steel pipe for flushing flows (channel maintenance). The discharge outlet will be equipped with a fish screen and will discharge to a riprap-lined outfall channel for energy dissipation and to assist with aeration to raise dissolved oxygen levels in the discharged water.

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.

18.8 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 had been assessed and was found to be technically viable in the Systems Impact Study and Facilities Studies completed by BC Hydro in 2014. The studies identified a number of 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. Alternative line alignments and BC Hydro interconnection points were contemplated throughout the design process before settling on the current arrangement.

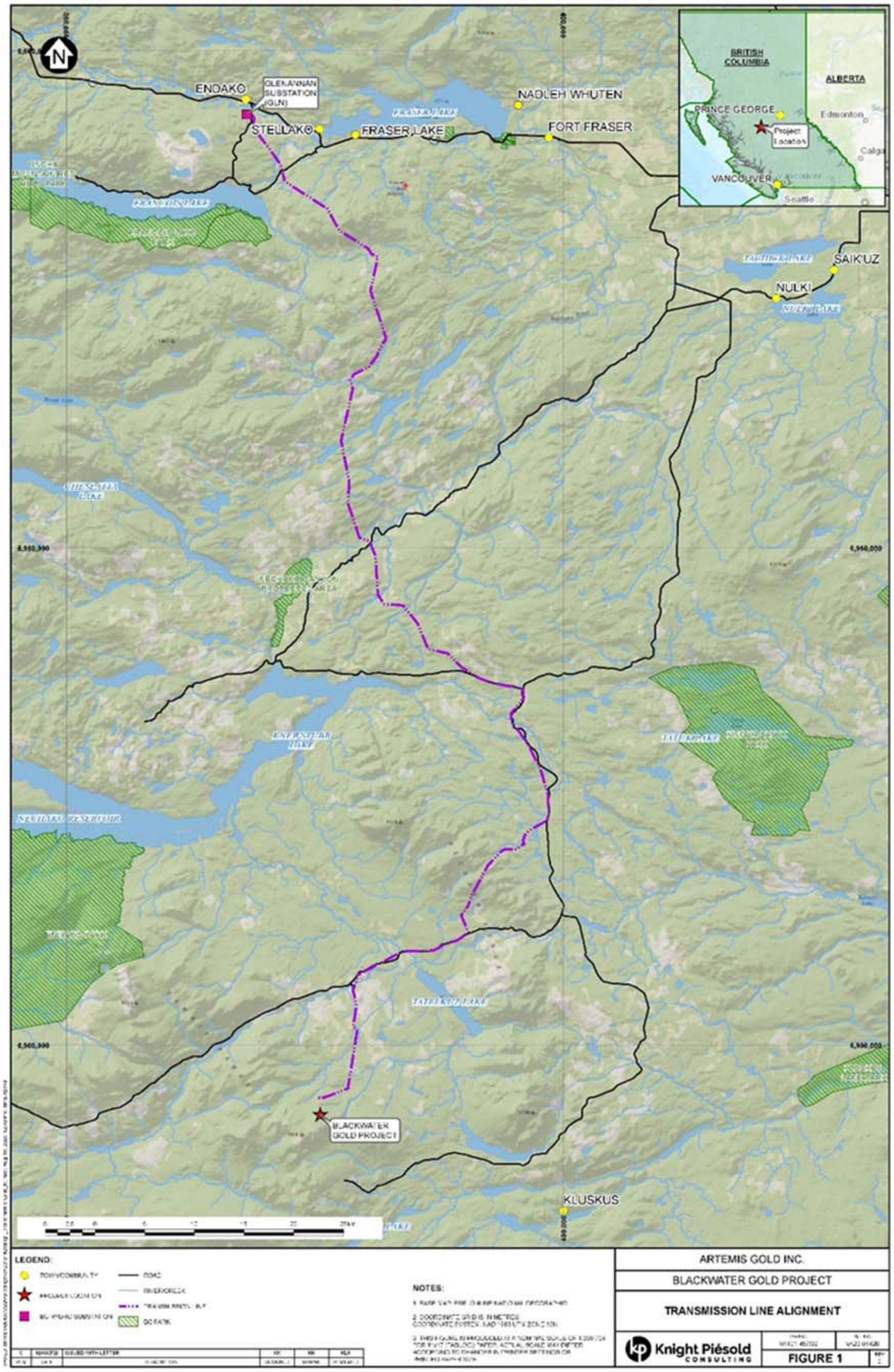

Figure 18-10 Proposed Power Transmission Line Routing

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 transformation 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 adjacent 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. The transformer secondaries will be connected to the primary distribution centre (PDC) 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 PDC and routed around the site in cable and tray (short runs only) 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 PDC at the main substation. Portable substations will transform the power to 4.16 kV for the mine shovels and drills. Diesel drills and shovels will be used from mine start-up to the end of Year 5 with electric drills and shovels coming online in Year 6. One set of electric drilling equipment (550 kW) and one 27 m^3 electric shovel (1,450 kW) will be required in Year 5 with a second drill and shovel in Year 9. Each system will consist of a single portable substation. Additional portable substations (25 kV/4.16 kV stepdown), also powered from the mine 25 kV pole line, will be used for mine water supply from Tatelkuz Lake and the reclaim water system barge(s) 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 a standby power station 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 nominal 3.0 MW. The temporary construction power generation equipment will be used as the source of backup power supply for the permanent camp.

18.9 Buildings

18.9.1 Process Plant Buildings

A summary of the size of the process buildings is provided in Table 18-3. All will be steel frame, with the exception of the gold room, with sandwich panel covering, fitted with crane rails along their whole length and furnished with a travelling crane. At least one large equipment door will be fitted with three personnel doors. All the main plant buildings will be in close proximity and will be joined by covered corridors. Although further apart, the crusher buildings will also be joined by covered corridors.

Table 18-3 Summary of Process Building Dimension and Crane Capacity

Building	Length (m)	Width (m)	Height to top of side (eve height)	Crane Capacity (t)
Mill Building	60	24	25	20
Reagent Building	48	24	15	10
Adsorption Building	60	22	25	20
Primary Crusher Building	30	20	19	20
Cone Crusher Building	42	18	19	20
Screen Building	36	16	19	10
Gold Room	16	16	15	none

The gold room will be constructed with tilt up concrete to provide a more secure environment.

18.9.2 Administration Offices

The administration offices will be of modular construction, 350 m^2 in area. They will include a meeting room, toilets, closed offices and an open work area equipped with work stations.

18.9.3 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 TV coverage of all parts of the plant. The total area will be 400 m^2, with a change room and toilets. The construction will be modular, similar to the administration offices.

18.9.4 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

400 m^2 and 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 building adjacent to the main one (100 m^2)

18.9.5 Stores

Two fabric-covered buildings will be used as a plant store and a mining equipment store. They will be insulated and 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.

18.9.6 Truck shop 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 240 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.

18.9.7 Accommodation

A camp on site that used during exploration 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 new camp will comprise of five “Jack and Jill” units (one bathroom between two rooms) of 42 bedrooms each and one unit for 30 persons with individual bathrooms. This new accommodation will include a dining hall and recreation facilities, a storage room for the belongings of persons on days off, a backup generator, water storage for three days and sewage treatment facilities.

18.10 Fuel

A diesel storage and dispensing unit will be supplied by Petro-Canada, through their wholesalers, Jepson Petroleum. They will supply skid-mounted storage tanks, pumps and meters. Payment will be made through a surcharge on the fuel purchased.

Propane-storage will be provided by the supplier.

18.11 Fire and Potable Water Supply and Distribution

Two wells approximately 1 km east of the exploration camp area will be the source of water supply for the temporary and operations camps. 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.

Water from the wells will be treated in modular portable water treatment plants (PWTPs) then pumped to a fire/fresh water storage tank at each of the camp sites. The existing exploration camp already has a PWTP and a new unit will be installed at the new permanent camp with capacity to supply the 240 bed camp. This will be increased for the expansions. These tanks will provide more than 24 hours of storage. A similar but smaller PWTP will be installed in the process plant area to provide potable water for the offices, process plant and laboratory. A further unit will be installed at the mine offices/change rooms.

18.12 Sewage and Waste

The exploration camp will continue to use of the existing sewage treatment system, consisting of two rotating bacterial contact units packaged treatment plants and an infiltration field. A similar system will be installed at the new permanent camp. Similar units will be installed in the process plant area and the area of the truck shop and mine offices/change rooms.

Recyclable materials such as scrap metal, unusable machinery, aluminum cans, drained oil filters, and antifreeze will be segregated at the point of generation to avoid becoming mixed with other wastes. These materials will be collected and stored in a centralized location where they will be compacted, packaged, and palletized for shipping to recycling facilities in BC or further afield if necessary.

Depending on the type of material, general refuse (i.e., pallets, cardboard, non-recyclable containers, construction waste, putrescible and non-putrescible refuse) will either be placed directly in the solid waste facility or incinerated and then placed in the solid waste facility. The solid waste facility would be co-located with the overburden portion of the West WRSF for ultimate burial.

Hazardous waste will be safely stored in a curbed and secured area until it can be transported off site for disposal to a permitted hazardous waste facility.

18.13 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 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 (CCTV) 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 CCTV camera system.

18.14 Communications

Telus currently provides telecommunications service from an existing microwave tower at the Blackwater site. The bandwidth is 150 Mbps, which will be expanded to 250 Mbps for Project construction and normal operations. At present, a 3G (third generation) wireless communication system with full cellular or data service is available on site.

Data service will be brought to site at a later date via a permanent fibre optic cable running along the power line. The in-plant communications system will then transition to the fibre optic cable as the primary communication connection; the microwave system will be retained for secondary backup.

The site-wide communications system will be based on a fibre optic backbone connected to the following systems:

- IT network (VoIP telephone and local area network);
- Control system network;
- Security network;
- CCTV network;
- Fire alarm network.

18.15 Comments on Section 18

The QPs note:

- The overall Project facilities and major infrastructure cover the mine site area, TSF, camp site, main access road, and water pipeline from Tatelkuz Lake;

- Project infrastructure has been designed to have a minimal footprint. The TSF and WRSFs are 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;
- A helipad will be included for emergencies;
- 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 Site C and TSF Site D, which are staged to support initial mine operations and the two subsequent expansion stages. The starter dam for TSF Site C was relocated slightly downstream to optimize initial capacity and haulage distances, improve constructability by following existing access trails in an area of gentler terrain, and simplify water management during early operations;
- 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;
- The staging of the TSF delays some environmental impacts of the facility and capital expenditures to the extent practicable, while not changing the final footprint of the facility that successfully completed a coordinated provincial and federal environmental assessment in 2019;
- All drainage from the mine will flow by gravity into the TSF to simplify water management, spill control, and mine closure;
- Water within the Project area will be used by collecting runoff from the mine site area and recycling process water to the maximum practicable extent. Site runoff water will be collected and stored within the TSF and used to inundate the PAG/NAG3 waste rock and tailings solids to limit the potential for ARD and ML. Water will be stored in the supernatant ponds within the TSF and recycled to the mill for use in the process;

- Runoff water from undisturbed areas not needed for ore processing will be diverted around the mine facilities and fresh water will be pumped from Tatelkuz Lake to offset flow reductions in lower Davidson Creek and meet instream flow needs for downstream fisheries;
- Planned installation of a water treatment plant 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.

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 is bought and sold, and it is not difficult to obtain a market price at any particular time. The gold 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 PFS economic analysis uses metal prices and exchange rates consistent with current consensus estimates.

19.3 Contracts

BW Gold expects that terms contained within any sales contract that could be entered into would be typical of, and consistent with, standard industry practices, and be similar to contracts for the supply of doré elsewhere in the world.

19.4 Gold Stream Agreement

Artemis and BW Gold 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 $28 million on each of those dates.

19.5 Comments on Section 19

In the opinion of the QP, BW Gold will be able to market gold produced from the Project. Sales contracts that could be negotiated would be expected to be within industry norms.

20 ENVIRONMENTAL STUDIES, PERMITTING, AND SOCIAL OR COMMUNITY IMPACT

20.1 Baseline Studies

The Project underwent a coordinated provincial and federal environmental assessment that was initiated in 2012 and ended successfully in 2019. Environmental baseline study areas were defined to characterize the current conditions in the areas potentially affected by Project components or activities. Results of work completed to date are included as summaries in the following sub-sections. The information is largely sourced from documents developed during the environmental assessment process.

20.1.1 Atmospheric Environment

Air quality

Background levels of total suspended particulates (TSP), particulate matter (PM) with a diameter less than 10 µm (PM_{10}), particulate matter with a diameter less than 2.5 µm ($PM_{2.5}$), nitrogen dioxide, carbon monoxide, and sulphur dioxide in the Project area were estimated using three public domain air quality databases from similar remote areas and on-site particulate monitoring for $PM_{2.5}$ and PM_{10} from August 2012 to December 2013. The background concentrations of most of the common air contaminants (CACs) were low; however, TSP, PM_{10}, and $PM_{2.5}$ concentrations exceed Ambient Air Quality Objectives. Emissions of the CACs are predicted to increase compared to background levels during construction and operation of the Project as a result of machinery and equipment use (combustion), vehicle travel, and materials handling. Mitigation measures will be contained in an Air Quality and Emissions Management Plan, and will include meeting stringent emission standards, the use of ultra-low sulphur diesel, control of vehicle speeds, and using water or a chemical dust suppressant on unpaved road surfaces and materials to reduce PM emissions.

Noise and Vibration

The baseline noise survey indicated that both the ambient sound levels and sound characteristics in the Blackwater area are low (approximately 31 decibels) and below all relevant ambient noise standards and criteria. There are currently no anthropogenic sources of vibration in the vicinity of Blackwater. Construction and operation activities will increase noise and vibration levels. Proposed mitigation measures include selecting vehicles and equipment with industry standard abatement technology for noise, maintaining equipment in good working condition, and positioning noisy equipment in sheltered or enclosed locations. A Noise and Vibration Effects Monitoring and Mitigation Plan will also be developed to monitor Blackwater-related noise and implement mitigation measures to minimize adverse effects.

20.1.2 Terrestrial Environment

Topography and Soils

Topography includes landforms such as mountains, plateaus, hills and valleys, as well as ridges, slopes, terraces, and gullies. Soil and soil quality relate to overburden, sediments, and soils located overtop of bedrock. Potential adverse effects from the transmission line, mine access road, and mine site facilities include the alteration of landforms, terrain instability, and accelerated erosion due to surface disturbance. One of the major considerations in the soil quality assessment is the availability and/or suitability of soils for reclamation purposes: the majority of soils in the study area are considered “Fair” to “Poor” in terms of reclamation suitability, due to the relatively high coarse fragment content and coarse textures of the soil matrix as well as the soil pH. The primary measure to mitigate adverse effects of the Project on topography is to minimize the mine footprint.

Wildlife and Ecosystems

The mine site is in the Davidson Creek Vanderhoof Land and Resource Management Plan Resource Management Zone (RMZ) and has important habitat for grizzly bear, marten, and moose. The Chedakuz Creek and Laidman Lake RMZs border the planned mine site and provide important habitat for moose, mule deer, grizzly bear, and black bear. A total of 28 federally-and provincially-listed wildlife species at risk have the potential to occur within the Project area, of which 18 were confirmed present. A total of 159 plant species-at-risk potentially occur in the Project area, and 341 ha of whitebark pine, which is listed as endangered under the federal Species at Risk Act, is present in the mine site study area. The Project is also located at the eastern edge of the Tweedsmuir-Entiako caribou herd’s winter range, with the easternmost extent of the herd’s core winter range being about 10 km from the mine site. A key concern during the EA related to wildlife was the predicted impact from the Project on the caribou population, which is listed as threatened under the federal Species at Risk Act.

The Project has the potential to cause loss and degradation of vegetation and wildlife habitat and ecosystems, increase the risk of mortality, and impact movement patterns, population dynamics, and survival and reproduction. Approximately 65% of the Project footprint would have permanent effects, while approximately 35% would be decommissioned and potentially restored to baseline conditions over time.

Project mitigation for wildlife and ecosystems includes minimization of the Project footprint and co-location of project components with existing disturbances where practicable. The implementation of best practices during Project phases will be outlined in several management plans, including the Caribou Mitigation and Monitoring Plan, the Wetlands Management Plan; Landscape, Soils, and Vegetation Management and Restoration Plan; Invasive Species Management Plan; Wildlife Management and Monitoring Plan; and Whitebark Pine Management Plan.

20.1.3 Aquatic Environment

Water Quality and Quantity

The Blackwater mine will be located in the sub-alpine areas north of Mt. 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.

Groundwater in the Project area flows from recharge zones located in topographic highs like Mount Davidson (in the area of the proposed open pit) and then discharges in the valleys of Davidson Creek, Creek 661, and Creek 705. Groundwater discharges provide most of the surface water flow in these creeks in the winter and early spring months. The main groundwater flow pathways in bedrock are through highly fractured zones of rock, which are found in the upper 10 to 20 m of bedrock, and in a nearly circular zone extending almost 500 m from the top of the bedrock on the southeastern slope of Mt. Davidson. This fractured rock will be excavated for the mine.

Surface flows in the study area are influenced by a climate characterized by brief warm summers and long cold winters, with roughly half of annual precipitation falling as snow, with rain predominant from May to September. Peak stream flows occur in May, with low flows in February or March.

Local surface water has low alkalinity, and low concentration of nitrogen species. Total and dissolved aluminum, total cadmium, total copper, total iron, and total zinc exceed provincial and/or federal water quality guidelines at some locations as a result of naturally elevated concentrations in the bedrock and soils. Arsenic, iron, and manganese exceeded sediment quality guidelines in stream samples. Mercury concentrations were slightly above guidelines in sediment samples from all lakes except for Tatelkuz Lake.

Construction of the Project will eliminate stream reaches in the Upper Davidson Creek watershed under the TSF and would alter surface and groundwater flows downstream. The open pit will be within the Davidson Creek and Creek 661 watersheds and will alter groundwater and surface water flows downstream. Mitigation to address changes in downstream flows include diverting fresh water around the mine site, obtaining process water from the TSF, construction of a FWR and approximately 14 km-long pipeline that will convey freshwater from Tatelkuz Lake to the FWR for release into Davidson Creek during the operations and closure phases, and treating and discharging pit water to Davidson Creek during operations.

Adverse changes to surface water quality will be mitigated through, among other measures, the construction and operation of a metals removal water treatment plant; collection of pit lake seepage in an open pit seepage collection system and conveyance by pipeline to the TSF following closure; an ion exchange and nanofiltration water treatment plant to treat TSF Site D supernatant pond water during late closure, and seepage and non-contact groundwater captured at the ECD post-closure.

Fish and Fish Habitat

Twelve fish species were captured or observed in streams and lakes within the Project area; rainbow trout and kokanee were the most numerous species and identified as the species of concern. Kokanee are primarily lake resident, migrating to spawning areas in tributary streams in mid-to-late summer, and spawning in late summer and fall. Eggs incubate over winter, and fry emerge from the gravel in the spring after ice break-up. Fry immediately migrate to their residence lakes. Adult rainbow trout inhabit Chedakuz Creek or Tatelkuz and Kuyakuz Lakes. They emigrate from these areas into Project area streams in early June to spawn and return to the lakes before the end of June. Rainbow trout fry emerge from the gravel and spend at least one year rearing in Project area streams before moving downstream to mature in Chedakuz Creek or the area lakes.

Fish and fish habitat have the potential to be directly affected by the Project footprint or the Project facilities (mortality or injury, habitat loss) and indirectly affected by changes in water quality and quantity and sediment quality, resulting in changes in health, growth, reproduction, and behaviour. Fish habitat losses will occur within the TSF footprint, isolated streams above the TSF, tributaries to Upper Creek 661, and linear corridor footprints. A permanent loss of 108 ha fish habitat is predicted in the upper reaches of Davidson Creek, and habitat isolation of 41 ha is expected in the upstream portions of Davidson Creek and its tributaries.

In addition to the measures to minimize impacts on water quality and quantity and sediment quality, mitigation measures specific to fish and fish habitat include minimizing the Project footprint limiting the disturbance area and avoiding sensitive kokanee spawning habitat. Development and implementation of a Fisheries Mitigation and Offsetting Plan, Aquatic Effects Monitoring Plan, and Tatelkuz Lake Protection Plan will further ensure appropriate and effective mitigation. The conceptual offsetting measures include flow augmentation in Davidson Creek to maintain instream flow needs for fish and several offsite habitat creation and restoration projects. Flow augmentation inputs through the fresh water supply pipeline include water pumped from Tatelkuz Lake, non-contact runoff water diverted around the TSF, and treated mine contact water. A final Fisheries Offsetting Plan will be required for the Fisheries Act Authorization for the loss of fish habitat. The offsetting plan will be completed to the satisfaction of DFO and Environment and Climate Change Canada and in consultation with Indigenous groups.

20.1.4 Socio-Economic Environment

The Project is located in the Cariboo Regional District of Central BC. The nearest large towns are Vanderhoof, Prince George, Fraser Lake, and Burns Lake. The closest Indian Reserve to the mine site is Tatelkus Lake 28, approximately 15 km away.

The proposed Blackwater mine site is located primarily within the asserted traditional territories of Lhoosk'uz Dené Nation and Ulkatcho First Nation, and overlaps with the far southeast portion of Skin Tyee Nation's asserted traditional territory. Other components of the Project cross the asserted traditional territories of Nadleh Whut'en First Nation, Saik'uz First Nation and Stellat'en First Nation (together referred to as the Carrier Sekani First Nations) and Nazko First Nation.

The majority of the work force is expected to be drawn from residents in the region. Activities associated with the proposed operations could affect the regional social environment through changes to demographics, infrastructure, services, family and community well-being, current use of lands and resources for traditional purposes, Non-traditional land and resource use, and visual aesthetics. Potential demographic changes during operations, primarily in Prince George and Vanderhoof, are assumed to benefit local communities, as they would be within current capacities for local communities and align with local government policies on attracting new residents. The increased population (population increase predicted of 0.3% in Prince George and 1.3% in Vanderhoof) would increase the demand for utilities, housing, and recreation; however, the local communities have the capacity to accommodate these needs. The planned operations have the potential to cause both temporary and long-term restrictions to tenured and non-tenured land and resources in the vicinity, including mining exploration and mineral tenures; forestry and timber resource use; traplines, guide outfitters, and fishing; and agriculture, grazing and range use.

The operations are forecast to generate positive economic impacts through direct expenditures on goods and services, creation of employment opportunities and generation of tax revenues for federal, provincial, and local governments. Sufficient capacity exists in regional infrastructure and services to accommodate the incremental demands of the Project-related population increase. Closure is anticipated to have a negative but not significant effect on the provincial economy and government revenues.

20.1.5 Heritage

The Blackwater development area was determined to have low to moderate potential to contain protected archaeological resources and historical heritage sites, and a moderate to high potential to contain cultural heritage resources such as trail blazes, traps, and traplines that postdate 1846 and are not protected under the Heritage Conservation Act. The transmission line alignment has the potential to impact paleontological sites.

Mitigation measures to be employed will include avoiding known archaeological and historic sites to the extent possible and developing and implementing a Cultural and Spiritual Resources Management Plan in consultation with Indigenous groups. The Cultural and Spiritual

Resources Management Plan would identify areas of cultural or spiritual importance, and develop a chance find procedure to deal with cultural and archaeological finds.

20.2 Social or Community Related Requirements

20.2.1 First Nations

The Blackwater mine site is located primarily within the asserted traditional territories of the Lhoosk'uz Dené Nation and Ulkatcho First Nation and overlaps with the far southeast portion of Skin Tyee Nation's asserted traditional territory. Other components of the project cross the asserted traditional territories of the following Indigenous groups:

- Nadleh Whut'en First Nation, Saik'uz First Nation, and Stellat'en First Nation (the Carrier Sekani First Nations)
- Nazko First Nation

The Environmental Assessment Office (EAO) notified the following Indigenous groups of key milestones during the environmental assessment and invited their participation in the review, 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:

- Tsilhqot'in Nation;
- Nee Tahi Buhn Band;
- Cheslatta Carrier Nation;
- Yekooche First Nation.

Lhoosk'uz Dené Nation and Ulkatcho First Nation

A trilateral Participation Agreement was signed with Lhoosk'uz Dené Nation and Ulkatcho First Nation on April 18, 2019. Ulkatcho First Nation and Lhoosk'uz Dené Nation provided a detailed report on their perspectives of the impacts of the Project on their Aboriginal Interests: this report was included with the EAO Assessment Report in May 2019 that was referred to the provincial Ministers in the environmental assessment process. Ulkatcho First Nation and Lhoosk'uz Dené Nation also submitted letters stating their consent to the issuance of the BC EA Certificate, and indicated that New Gold had adequately consulted and accommodated Lhoosk'uz Dené Nation and Ulkatcho First Nation with respect to their asserted Aboriginal rights and title. UFN and Lhoosk'uz Dené Nation both submitted letters to the EAO in October 2019 re-stating their support for the Project and consenting to the BC EA Certificate, and any other permits or authorizations to be issued by or on behalf of the EAO pertaining to the Project.

The Carrier Sekani First Nations

Project interactions with the Carrier Sekani First Nations, and their Aboriginal title, rights, and interests relate to the transmission line, use of road to access the site, and potential downstream water quality and other effects. The Carrier Sekani First Nations' asserted traditional territories do not overlap the mine site. The EAO and Carrier Sekani First Nations established a Blackwater Collaboration Plan, as part of the environmental assessment process, to meet the objectives of the April 2, 2015, Collaboration Agreement between the province and Carrier Sekani First Nations, including addressing the potential adverse effects of the Project on any Carrier Sekani First Nations' Aboriginal title, rights and interests. On May 2, 2019, the Carrier Sekani First Nations communicated to the EAO that they had reached consensus on proposed conditions; however, the Carrier Sekani First Nations raised additional concerns regarding economic accommodation and compensation. BW Gold has committed to continuing negotiations with Carrier Sekani First Nations 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.

General mitigation measures to address concerns related to Aboriginal title, rights, and interests are encompassed by several EA Certificate conditions, including Indigenous Cultural Awareness and Recognition, Aboriginal Group Monitor and Monitoring Plans (that would require the Certificate Holder to retain or provide funding for Aboriginal Monitors for each of the Carrier Sekani First Nations), and a Cultural and Spiritual Resources Management Plan. Conditions were also developed to address concerns related to wildlife, water quality and mine waste management.

20.2.2 Public and Other Stakeholders

The key mitigation measures identified to minimize adverse social effects are the development and implementation of several management plans, including a Community Effects Monitoring and Mitigation Plan and the forming of a Community Liaison Committee. The Project will also be required to retain a qualified person to develop a Tenure Holder Communication and Mitigation Plan to address outstanding concerns of the holder of a guide outfitter tenure that overlaps the Project area.

20.3 Mine Closure Requirements and Costs

Under the Mines Act and accompanying Health, Safety and Reclamation Code, proponents are required to post a reclamation security bond to cover the default cost of site reclamation, maintenance and closure, and to provide for protection of, and mitigation of damage to, watercourses and cultural heritage resources affected by the mine. The amount and installment schedule will be detailed in the Mines Act Permit Application.

The Application includes a conceptual Reclamation and Closure Plan that describes proposed reclamation measures that address reclamation standards as outlined in Section 10 of the Health, Safety, and Reclamation Code for Mines in BC, and is expected to form the basis of the Reclamation and Closure Plan detailed in the Mines Act Permit. The reclamation objectives conform to land and resource management objectives and strategies presented in the Vanderhoof Land and Resource Management Plan. During development of the Reclamation and Closure Plan, applicable legislation, criteria, and guidelines were considered. Methods to achieve these objectives include soil management and use, landform design, decommissioning and site preparation, revegetation prescriptions for specified ecotype targets, and seeding and planting densities.

The Reclamation and Closure Plan and Follow-up Monitoring and Compliance Reporting sections include proposed performance standards, management, and monitoring strategies to verify reclamation success, and a timeline for reclamation and monitoring activities, along with reclamation research programs. The plan includes strategies for temporary closure and premature closure. The plan emphasizes soil, vegetation, and wildlife habitat reclamation, and provides a cross-reference to relevant management plans. The provincial EA conditions also require that a Closure and Post-Closure Water Quality Management Plan be developed.

Progressive bonding for closure costs will be required. Closure costs, consistent with the conceptual plan included as part of the environmental assessment, have been incorporated within the Project cash flow model as described in Section 22.

20.4 Permitting Requirements

20.4.1 Environmental Assessment

Provincial Environmental Assessment Certificate #M19-01 is subject to 43 conditions set out in Schedule B, which are legally binding and subject to compliance and enforcement oversight. The holder of the Certificate must have substantially started the Project within five years from the issuance date of June 21, 2019.

The federal Decision Statement was issued under Section 54 of the Canadian Environmental Assessment Act, 2012 on April 15, 2019. The federal decision statement contains 102 legally-binding conditions.

The issuance of the provincial and federal environmental assessment was the pre-requisite for government ministries to initiate individual permitting processes. The permitting process is planned for initiation in late 2020 and 2021.

20.4.2 Provincial Permits

The primary provincial permits required for the Project to proceed to the construction and operations phases are issued under the Mines Act (Mine and Reclamation Permit) and the Environmental Management Act (Effluent Discharge Permit and Air Discharge Permit). Additional provincial permits that may be required for one or more phases of the Project include:

- Inspection, investigative, and site alteration permits under the Heritage Conservation Act;
- Lease(s) or license(s) of occupation under the Land Act;
- A Mining Lease under the Mineral Tenure Act;
- A Water License(s) under the Water Sustainability Act.

Several permits of a more routine nature will also be required.

20.4.3 Federal Permits

Federal authorizations are required under the Fisheries Act and Explosives Act for the Project to proceed to construction and operations. Notably, a Schedule 2 Listing will be required for the tailings facility under the Metal and Diamond Mining Effluent Regulations of the Fisheries Act. This is a process administered by Environment Canada and adjudicated by an Order in Council. A detailed fish and fish habitat offsetting plan is required to support the Schedule 2 amendment application.

A financial guarantee is required when submitting an application for authorization to cover the cost of implementing the offsetting plan and must be sufficient to cover the cost for implementing all elements of the offsetting plan, including monitoring measures. The amount of the financial guarantee is determined by the cost estimates described in the offsetting plan.

20.5 Comments on Section 20

The proposed Blackwater mine site is primarily located within the asserted traditional territories of the Lhoosk'uz Dené Nation and Ulkatcho First Nation. New Gold entered into a trilateral Participation Agreement with these two Indigenous nations on April 18, 2019, who, following completion of the environmental assessment process, confirmed their support for the Project and consented to issuance of the BC EA Certificate, and any other permits or authorizations to be issued by or on behalf of the Environmental Assessments Office pertaining to the Project.

On closing of the acquisition of the Blackwater Project, the Participation Agreement was assigned to BW Gold.

The Project could also interact with the Carrier Sekani First Nations and their Aboriginal title, rights, and interests as a result of the transmission line, use of road to access the site, and potential downstream water quality and other effects. Towards the end of the EA, New Gold committed to continuing negotiations with the Carrier Sekani First Nations with the goal of reaching a mutually acceptable participation agreement that will include accommodative measures and other benefits. Artemis remains committed to this goal.

21 CAPITAL AND OPERATING COSTS

21.1 Capital Cost Estimates

21.1.1 Introduction

Capital cost estimates were prepared by the following parties as part of the 2020 PFS:

- MMTS:
  - All costs related to mining;
- JAT Metco:
  - All cost estimates related to process, on site infrastructure, and access road;
- KP:
  - Design, quantities, and cost estimates for staged development of the TSF over the life of the mine, including development of the design and filling plan for TSF Site C and TSF Site D;
  - Design, staging, quantities, and cost estimates for water management components to accommodate the mine development schedule, including non-contact water diversions, contact water collection, reclaim water systems, water balance and surplus water management, and staging of the ECD;
  - Design, quantities, and cost estimates for tailings distribution systems;
  - Design, quantities, and cost estimates for off-site facilities, including the fresh water supply system, transmission line, and fish offsetting measures;
  - Input on estimated closure costs.

21.1.2 Basis of Estimate

The capital cost estimate was developed to provide an estimate suitable for the 2020 PFS, including costs to design, procure, construct, and commission the facilities. The expected accuracy range of the capital cost estimate is +25/-10%.

The cost estimate is based on a combination of material take-off (MTO) data, design drawings, vendor quotes, manufacturers' information, and industry standards and rates. Key assumptions used in generating the estimates were:

- Diesel: $CDN0.98/L;

- Electrical power: $CDN0.052/kWh;
- United States dollar: USD$0.76 to $CDN1.

The estimate is based on the following:

- Process flow diagrams;
- Equipment list;
- Design criteria;
- General arrangement drawings and site layout plans;
- Equipment specifications for major equipment;
- Budget quotations from vendors for major equipment;
- Budget pricing for bulk materials;
- Geotechnical and hydrogeological reports;
- Regional climactic data;
- Project execution plan;
- Project work breakdown structure (WBS).

The capital cost of the process plant has been estimated with a quote or multiple quotes being obtained for all the major equipment. Where possible, an installation cost has been included by the supplier, or guidance on the number of man hours or the time required to install has been obtained. Where installation information was not supplied, experience from similar jobs was used. Wherever possible, pricing was obtained for complete units, thus the crushers, feeders, and screens needed for the crushing circuit were all supplied by one manufacturer that also confirmed the flow sheet and throughput. Multiple quotes were obtained for the crushing circuit as were quotes for the ball mill. The gravity circuit was also quoted as a package, as were all the leach and cyanide destruction tanks. The adsorption unit, a carousel unit was quoted as a complete package, as was the carbon elution/regeneration/electrowinning and smelting unit. The system for preparing sodium cyanide, SMBS and copper sulphate solution were also quoted as a complete unit. Care was taken to only select suppliers known to produce equipment of a high quality.

The quantities of concrete needed were estimated from the building layouts and the size and weight of the equipment. In the case of the ball mill foundation, which is the largest single structure, the ball mill supplier provided a drawing of the foundation from which the volume of concrete was estimated. Distinction was made between slab on grade, plinths and pedestals, walls etc. but a single blended rate was used for estimating the cost of the concrete work, which was supplied by Ausenco. The rate used was \$1,680 per m^3.

The weight of structural steel needed was estimate from quantities used in a similar, smaller plant and increase by the scale up factor using the “0.6 rule”. Quotes were obtained from a well-known steel supplier and prices for erection were obtained from a company specializing in this work. Only one quote was obtained and they are expected to be lower-priced suppliers.

The man hours needed for installation of equipment were given by some suppliers, in some cases these were estimated from experience. The basic man-hour rate used was \$100 per hour, with an allowance of \$70 per day treated as an indirect cost for lodging and catering.

It is planned to purchase a 30 t and a 150 t rough terrain crane and other mobile construction equipment which will be available to the contactors and after construction will be used by operations.

The costs for electrical and piping installation were estimated by applying a factor to the cost of the equipment installed. A factor of 30% was used for electrical installation and 20% of piping. In some cases, such as the “package” carbon treatment system, the factors were reduced to take into account the electrical and piping supplied as part of the package.

The price of the process buildings was estimated from quotes for supply and installation and the cost of the concrete foundations were estimated as a cost per metre of perimeter, using a typical foundation drawing.

21.1.3 Mining

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 drilling, production drill and blast, load and haul, support and GME costs.

The mine equipment fleet is planned to be purchased through either finance or lease agreements with the vendors. Down payments and monthly lease payments are capitalized through the initial and sustaining periods of the Project.

Open pit site preparation costs have been capitalized including:

- Pad preparation;

- Stockpile preparation;
- Clearing, grubbing and topsoil removal;
- Haul road construction;
- Pit dewatering piping, in-pit structures and pumping.

The following items have also been capitalized:

- Explosives magazine and mixing plant;
- Site GPS and machine guidance systems;
- Mine survey gear and supplies;
- Geology, grade control and mine planning software licenses;
- Maintenance tooling and supplies;
- Mine rescue gear;
- Mine communications systems;
- Pit geotechnical instrumentation;
- Initial fabric structure truck shop.

21.1.4 Owner's Costs

BW Gold provided Owner's costs for inclusion in the capital cost estimate. These include personnel-related costs (salaries, travelling, offices) for the Owner's team, permitting, insurances, camp costs (catering and camp maintenance), and safety and training for Operations personnel.

21.1.5 Contingency

An overall contingency of 15% was applied to initial, expansion and sustaining capital costs.

21.1.6 Expansion and Sustaining Capital

Sustaining capital costs are the ongoing capital expenditures required to sustain operations. Expansion capital is the capital required to increase the processing throughput rate in Years 6 and 11.

21.1.7 Capital Cost Summary

The initial capital cost estimate for the facilities described in this Report is approximately $592 million (US$450 million). Total LOM sustaining capital is estimated to be $637 million (US$484 million). Reclamation and closure costs of $117 million and salvage value of $42 million have been included in the economic model.

The capital cost estimate and financial model exclude allowances for cost escalation over the Project duration.

Capital costs are summarized in Table 21-1 below.

Table 21-1 Capital Cost Estimate Summary – by Major Area

	Phase 1 Initial Capital	Phase 2 Expansion Capital	Phase 3 Expansion Capital	Sustaining Capital	Total Capital
	$M	$M	$M	$M	$M
Directs					
Mining	68	89	68	337	562
Process Plant	109	130	143	-	382
Onsite Infrastructures	68	38	19	2	127
Offsite Infrastructure	81	9	12	6	108
Tailings and Water Management	37	29	33	190	290
Total Directs	364	294	274	536	1,469
Indirects and EPC	120	73	69	18	279
Owners Costs	31	3	3	-	37
Contingency	77	56	52	83	268
Total	**592**	**426**	**398**	**637**	**2,052**

21.2 Operating Cost Estimates

21.2.1.1 Introduction

The operating cost estimates were prepared by the following parties:

- MMTS: Mining cost estimates;
- JAT Metco: Estimates for reagent and steel consumables consumption and costs, spare parts utilization, labour costs, freight, and electrical power; mill staffing plans, G&A;
- KP: Tailings and waste management area operations, including tailings pipelines

and water management systems.

21.2.2 Mine Operating Costs

The mine operating costs consist of the following components:

- Equipment operating cost – the activities of drilling, blasting, loading, hauling, mining support, and equipment maintenance. Equipment operating costs are calculated from the total annual operating hours based on the equipment productivities and the cost per SMU hour to operate the equipment. The largest component of the estimated mine operating costs is for the hauling function, and a significant portion of the planned hauls for Blackwater are downhill loaded, especially early in the project life;

- Salary and hourly personnel – mine department salary staff and general mining labour; maintenance and operator labour included as part of the equipment operating cost. During peak production, the mine is expected to employ 35 salaried personnel and 310 hourly personnel;

- General mine expense costs – miscellaneous tools and equipment necessary to support mine operation, such as surveying, mine planning software, geotechnical instrumentation for pit and dumps, office costs and overheads for mine operations, mine maintenance and technical services.

The total cost, including support, is approximately $2.3 billion to mine 1.1 Bt of material, including ore rehandle. The total unit cost for the Project is approximately $2.37/t mined. The unit cost is higher in the early years because of the small amount of material being moved and the higher cost of pre-stripping and site development. The costs near the end of mine life increase again due to the long uphill hauls to get out of the bottom of the pit.

The total operating cost for all activities is approximately $7.03/t of ore mined (Table 21-2) during the operations period.

Table 21-2 Mining Costs (Operations Period)

Mine Operating Cost Summary	$/t Mined	$/t Milled	Total $ (M)
Drilling	0.19	0.57	191
Blasting	0.25	0.75	249
Loading	0.29	0.86	289
Hauling	1.17	3.48	1,164
Support	0.30	0.89	299
Unallocated Labour	0.03	0.10	32
DIRECT COSTS - Subtotals	2.24	6.66	2,224
Mine Operations GME	0.05	0.16	53
Mine Maintenance GME	0.03	0.07	25
Mine Engineering GME	0.05	0.14	46
GME COSTS - Subtotals	0.13	0.37	124
TOTAL MINE OPERATING COST	**2.37**	**7.03**	**2,348**

21.2.3 Process Operating Costs

The process operating costs are based on the equipment included in the process flow diagrams.

Electrical power costs were estimated from a detailed list of equipment and the BC Hydro industrial rate.

Quotes were obtained for all major reagents and the expected consumption obtained from test work, vendors or from a data base.

Maintenance costs were estimated as a percentage of the capital cost of plant equipment, 5% being used in this case.

Labor costs were estimated from a manning table prepared using experience of operating similar plants, and the annual salaries, with appropriate overheads were obtained from a similar mining operation on the region.

Process operating costs are summarized in Table 21-3.

Table 21-3 Process Operating Costs

	Phase 1	Phase 2	Phase 3
	5.5 Mtpa	12 Mtpa	20 Mtpa
	$/t milled	$/t milled	$/t milled
Electrical Power	**1.46**	**1.46**	**1.46**
Consumables			
Crusher wear parts	0.37	0.37	0.37
Mill balls	1.27	1.27	1.27
Mill liners	0.10	0.10	0.10
Sodium cyanide	2.09	2.09	2.09
Lime (quick lime)	0.25	0.25	0.25
Sulphur	0.04	0.04	0.04
Copper sulphate	0.07	0.07	0.07
Lime for cyanide destruction	0.26	0.26	0.26
Carbon	0.13	0.13	0.13
Sodium hydroxide	0.02	0.02	0.02
Hydrochloric acid	0.04	0.04	0.04
Propane (elution & regeneration)	0.20	0.20	0.20
Others including lab supplies	0.07	0.07	0.07
Total Consumables	**4.91**	**4.91**	**4.91**
Labor	**2.20**	**1.36**	**1.29**
Plant G & A	**0.07**	**0.04**	**0.04**
Maintenance	**0.54**	**0.54**	**0.54**
Total	**9.17**	**8.31**	**8.24**

21.2.4 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. The G&A operating costs are summarized in Table 21-4.

Table 21-4 Summary of G&A Operating Costs

	Phase 1	Phase 2	Phase 3
	5.5 Mtpa	12 Mtpa	20 Mtpa
	$ M per Year	$ M per Year	$ M per Year
PERSONNEL			
Administration	1.3	1.6	1.7
IT	0.1	0.1	0.2
Human Resources	0.4	0.4	0.4
Security	0.7	0.7	0.7
Public/Community Relations	0.2	0.2	0.2
Environmental	0.7	0.8	0.8
Safety	0.4	0.5	0.6
Site Services	1.2	1.5	1.8
TOTAL PERSONNEL	**4.9**	**5.8**	**6.2**
EXPENSES			
Administration	3.3	5.0	5.6
IT	0.5	0.8	0.9
Human Resources	0.8	1.2	1.4
Security	0.2	0.4	0.4
Public/Community Relations	0.4	0.6	0.7
Environmental	1.4	2.1	2.3
Safety	0.3	0.5	0.6
Emergency Response	0.2	0.2	0.3
Site Services	5.6	8.5	9.8
Travel & Accommodation	6.9	9.4	8.2
TOTAL EXPENSES	**19.7**	**28.6**	**30.1**
TOTAL PERSONNEL + EXPENSES	**24.6**	**34.4**	**36.3**

21.2.5 Operating Cost Summary

The average LOM operating cost estimate is summarized by major area in Table 21-5.

Table 21-5 Summary of Blackwater LOM Operating Cost

	Units	Phase 1	Phase 2	Phase 3	LOM
Mining*	$/t Mined	2.15	2.14	2.62	2.37
	$/t Milled	14.61	12.12	4.98	7.03
Process	$/t Milled	9.17	8.31	8.24	8.33
G&A	$/t Milled	4.64	2.87	1.91	2.30
Total	$/t Milled	28.42	23.30	15.13	17.65

**Mining costs includes stockpile re-handle, LOM mining costs exclude pre-stripping*

The LOM operating cost estimates for peak in Phase 1 at $28.42/t, with economies of scale driving down costs to $23.30/t in Phase 2 and $15.13/t in Phase 3. Over the LOM, the Project has estimated average operating costs of $17.65/t.

21.3 Comments on Section 21

The QPs note:

- The initial capital cost for Phase 1 at 5,5 Mtpa is estimate is $592 million;
- The expansion capital cost for Phase 2 to achieve 12 Mtpa is estimate is $426 million;
- The expansion capital cost for Phase 3 to achieve 20 Mtpa is estimate is $398 million;
- Total LOM sustaining capital is estimated to be $636.5 million;
- Reclamation and closure costs net of salvage value are included in the financial model in Section 22;
- LOM operating costs are estimated at $17.65/t of ore milled;
- AISC is $668/oz in Stage 1, $696/oz in Stage 2, and $911/oz in Stage 3. The higher AISC in Phase 3 is mainly attributed to the inclusion of closure costs at the end of the life of mine;
- LOM AISC is $811 per ounce (or US$616/oz).

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 Q3 2020 Canadian dollars, unless otherwise specified.

The economic analysis includes the entire Project life, comprising two years of construction and 23 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 has been considered in the estimation of tax depreciation pools.

The economic analysis is presented as a Base Case, which assumes no leverage, and a Leverage 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.

Note that while the economic analysis and life of mine plan assumes that all stockpiled material is processed during the mine life, Artemis may be required to provide bonding for stockpiled material.

22.2.1 Metal Price and Selling Costs

The 2020 PFS Base Case metal prices and selling costs are summarized in Table 22-1.

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$/oz.	1,541
Silver price	US$/oz.	19.60
Currency exchange rate	US$:$CDN	0.76
Gold payable	%	99.9
Silver payable	%	95
Selling costs	$/oz.	3

22.2.2 Royalties

The 2020 PFS economics consider two private royalties at 1.0% and 1.5% over parts of the Mineral Reserve. 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 operating profits immediately upon the start of production 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 Levered Case Assumptions

The levered case is based on the following assumptions:

- Initial capital 60% debt financed;
- Annual interest rate of 5.5%;
- Upfront financing fee of 3%;
- 7-year term post commencement of commercial production with a balloon payment of 30% of the principal at maturity;
- Expansion capital is assumed funded through operating cashflow.

22.2.5 Closure Costs and Salvage Value

Closure costs are estimated at approximately $117 million. A salvage value of $42 million 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.

22.3 Financial Results

The production schedule on which the economic analysis is based is provided and discussed in Section 16 and is tabulated in Table 16-4.

Base Case cashflow for the overall Project attributable to Artemis 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 Leveraged Case.

The net present value (NPV) at 5% is discounted to the start of project construction.

22.4 Sensitivity Analysis

A sensitivity analysis was performed examining capital costs, operating costs, foreign exchange rate and gold price as shown in Figure 22-1, Table 22-4, and Table 22-5. The Project is most sensitive to fluctuations in gold price 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
Recovered gold	('000 oz.)	-	-	222	280	265	234	240	559	457	419	353	313	384	421	485	453	485	497	371	235	177	177	177	177	67	7,450
Recovered silver	('000 oz.)	-	-	830	818	865	925	829	1,619	1,628	1,834	1,981	2,659	1,540	1,100	1,476	1,783	2,130	2,396	2,123	2,558	2,577	2,577	2,577	2,577	972	40,374
Net Revenue	C$ (M)	(4)	(1)	439	547	519	460	474	1,102	896	821	701	649	775	837	968	909	982	1,009	753	503	397	401	401	401	146	15,081
Operating Cost	C$ (M)	-	-	120	144	141	171	177	259	250	294	303	292	398	386	392	384	376	335	287	236	214	213	220	215	91	5,897
Capital Cost	C$ (M)	247	345	52	42	39	168	258	60	68	71	134	263	47	49	41	49	37	35	15	4	9	7	2	3	82	2,127
Working Capital and Bonding	C$ (M)	7	1	6	1	1	1	2	(4)	1	1	2	2	(5)	1	1	1	1	1	1	1	1	1	1	2	8	41
Pre-tax cash flow	C$ (M)	(257)	(348)	261	360	338	120	37	786	577	454	262	92	335	400	534	475	568	637	450	262	172	180	177	181	(34)	7,016
Taxes	C$ (M)	-	-	7	20	101	65	50	283	203	161	104	66	100	133	184	168	201	229	157	88	58	62	61	64	6	2,571
Post-tax cash flow	C$ (M)	(257)	(348)	254	340	237	54	(12)	503	374	294	158	26	235	267	350	307	367	408	292	174	114	118	116	117	(41)	4,445

Note: numbers may not sum due to rounding.

Table 22-3 Economic Analysis

Item	Unit	Base Case	Leveraged Case
Pre-tax NPV (5%)	$CDN M	3,598	3,579
Pre-tax IRR	%	43	59
Pre-tax Payback	Years	2.0	2.1
Post-tax NPV (5%)	$CDN M	2,247	2,249
Post -tax IRR	%	35	50
Post -tax Payback	Years	2.0	2.2
Total LOM gold production	oz Au	7,450,000	7,450,000
Total LOM silver production	oz Ag	40,374,000	40,374,000
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	5.78	5.78

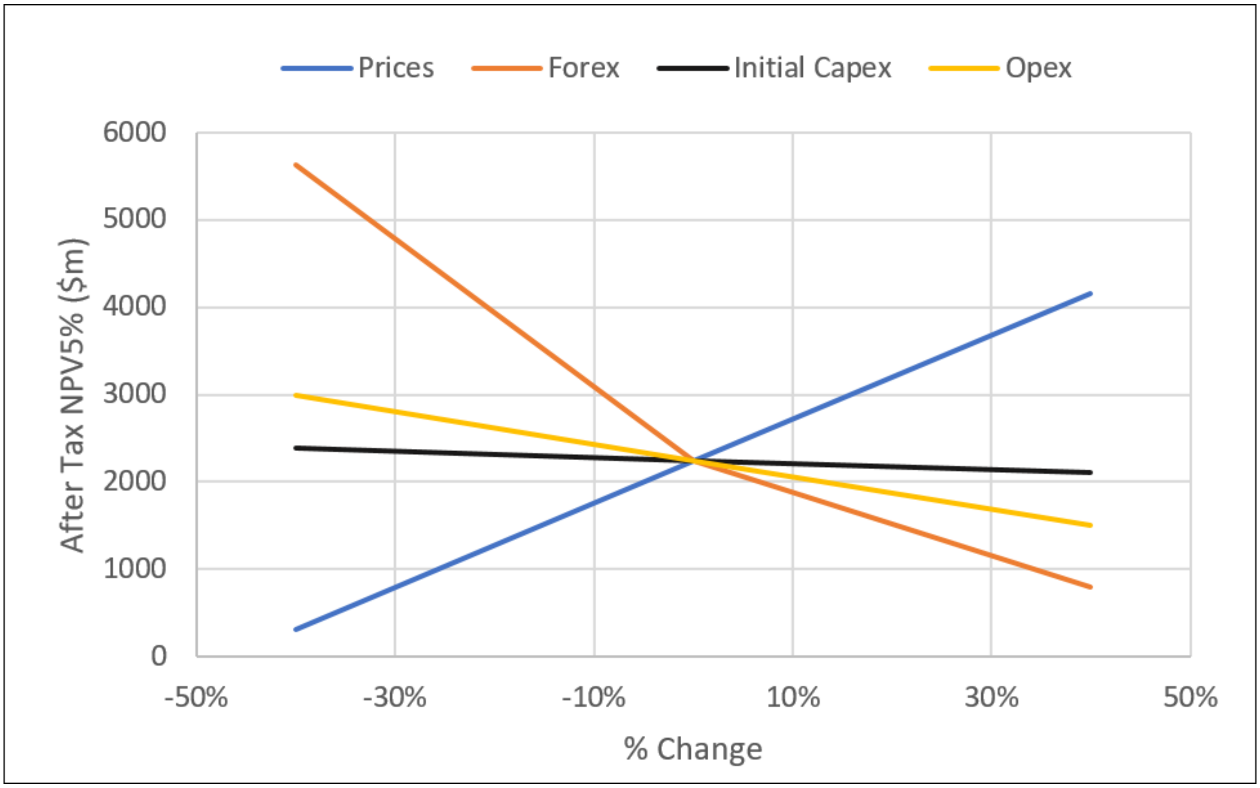

Figure 22-1 After Tax NPV5% Sensitivity Analysis (source: MMTS, 2020)

Table 22-4 After Tax NPV5% Sensitivity to Gold Price and Foreign Exchange

		Gold Price US$/Oz											
		$1,000	$1,100	$1,200	$1,300	$1,400	$1,541	$1,600	$1,700	$1,800	$1,900	$2,000	$2,100
Forex (US$ per C$)	0.60	1,475	1,869	2,262	2,654	3,047	3,600	3,831	4,224	4,616	5,008	5,400	5,792
	0.65	1,142	1,507	1,870	2,232	2,595	3,106	3,319	3,682	4,044	4,406	4,768	5,130
	0.70	860	1,195	1,533	1,870	2,207	2,682	2,880	3,217	3,553	3,890	4,226	4,562
	0.76	564	879	1,187	1,499	1,809	**2,247**	2,430	2,740	3,050	3,360	3,669	3,979
	0.80	383	691	984	1,281	1,577	1,993	2,167	2,461	2,756	3,050	3,345	3,639
	0.85	182	477	762	1,038	1,317	1,709	1,873	2,150	2,427	2,704	2,982	3,259
	0.90	1	281	560	825	1,085	1,456	1,611	1,873	2,135	2,397	2,659	2,921
	0.95	(189)	106	369	631	877	1,229	1,376	1,626	1,874	2,122	2,370	2,618

Table 22-5 After Tax IRR Sensitivity to Gold Price and Foreign Exchange

		Gold Price US$/Oz											
		$1,000	$1,100	$1,200	$1,300	$1,400	$1,541	$1,600	$1,700	$1,800	$1,900	$2,000	$2,100
Forex (US$ per C$)	0.60	27%	31%	35%	39%	42%	47%	49%	52%	56%	59%	62%	64%
	0.65	23%	27%	31%	35%	38%	43%	45%	48%	51%	54%	57%	60%
	0.70	19%	23%	27%	31%	34%	39%	41%	44%	47%	50%	52%	55%
	0.76	15%	19%	23%	27%	30%	**35%**	37%	39%	42%	45%	48%	50%
	0.80	12%	17%	21%	24%	28%	32%	34%	37%	40%	42%	45%	47%
	0.85	9%	13%	18%	21%	25%	29%	31%	34%	37%	39%	42%	44%
	0.90	5%	10%	15%	19%	22%	26%	28%	31%	34%	36%	39%	41%
	0.95	1%	7%	12%	16%	19%	24%	25%	28%	31%	34%	36%	38%

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 has been 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 employ 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;

- 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 test work is planned;
- TSF construction: higher costs may be encountered in integrating mining operations with TSF construction:
  - The bulk of the waste rock material types with low potential for ARD and ML (NAG4 and NAG5) will be required for TSF dam construction. If less low ARD/ML potential waste is available than anticipated, then the mine plan may need to target this better-quality material more specifically or to accept more overburden for dam construction;
  - 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;
- Notably, an amendment to Schedule 2 of the Metal and Diamond Mining Effluent Regulations is required to authorize use of the proposed TSF for mine waste disposal;

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 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:
  - 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 VSA oxygen system (90% pure oxygen), or by trucking to site (liquid oxygen). The VSA 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;
- Value engineering:
  - Assess the potential for increased mill throughput due to softer oxide/transition ores in early years to maximize existing plant capacity;
  - Optimize plant site layout to reduce earthworks;
  - Evaluate additional on-site borrow sources to reduce borrow haul distances;
  - Reassess single versus multiple process buildings as a potential cost reduction;
  - Re-evaluate the water supply system to simplify design;
  - Assess potential integration of the construction and operations camps;
  - Optimize the layout and construction of the low-grade stockpile to simplify low- grade ore placement and the drainage water collection system;
  - Develop detailed tailings and PAG waste rock deposition plans to simplify closure plan.

25 INTERPRETATION AND CONCLUSIONS

The QPs have reached the following conclusions and made the following interpretations as a result of the review of the 2014 FS document:

25.1 Project Setting

- Mining activities should be capable of being conducted year-round;
- There is sufficient suitable land area and surface rights available within the mineral claims for any future tailings disposal, mine waste disposal, and installations such as a processing plant, and related mine infrastructure.

25.2 Mineral Tenure, Surface Rights and Royalties

- BW Gold holds 100% recorded interest in 328 mineral claims covering an area of 148,688 ha distributed among the Property and the Capoose, Auro, Key, Parlane and RJK claim blocks;
- The Blackwater Property claim block comprises 75 mineral cell claims totalling 30,578 ha. All Blackwater claims are 100% held in the name of BW Gold. 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;

- BW Gold' 100% interest in the Blackwater claim block is subject to four 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 BW Gold to purchase two-thirds of three Blackwater Claims (637203, 637205, and 637206) NSR royalty for $1,000,000 at any time, such that a 1% NSR royalty would remain.
  - PS Claim: A 2% NSR royalty is payable on mineral claim 835014. The existing agreement would allow BW Gold to purchase half for $1,000,000.

  Only the royalties with respect the Dave Option and the Jarrit Option exist within the current Mineral Reserves.

- BW Gold acquisition of a 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 cash payment of $50 million to be paid on or before August 21, 2021 (still to be paid); and
  - 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 first ranking security interest over the Project until the Second Payment is made, and will thereafter maintain a security interest over the Project in connection with the gold stream agreement (subject to any security to be granted over the Project in respect of future project financing.
- All other material encumbrances within the Blackwater claim blocks are listed in Section 4.6.

25.3 Geology 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 a low to intermediate sulphidation epithermal system;

- The deposit type used for exploration targeting is appropriate to the mineralization identified and the regional setting.

25.4 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 1,041 core holes totalling 317,718 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.

25.5 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.

25.6 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.

25.7 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 test work 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;

- Estimated recoveries over the LOM are gold recovery of 93% and silver recovery of 65%.

25.8 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.

25.9 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 2020 PFS Mine Plan and classified in accordance with the 2014 CIM Definition Standards for Mineral Resources and Reserves;
- Blackwater Mineral Reserves total 334.0 Mt at 0.75 g/t Au and 5.8 g/t Ag;
- Factors that may affect the Mineral Reserve estimates include metal prices, 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 license conditions, and the ability to maintain the social license to operate.

25.10 Mine Plan

- Reasonable open pit mine plans, mine production schedules and mine capital and operating costs have been developed for Mineral Reserves at Blackwater;
- Pit layouts and mine operations are typical of other open pit gold operations in western 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 PFS.

25.11 Process

The process will consist of:

- Three stage crushing, consisting of a primary jaw crusher with grizzly feeder, a secondary cone crusher and two tertiary cone crushers. The primary jaw crusher, the three cone crushers and the three vibrating screens will each be housed in steel-framed buildings, with covered conveyors transporting material between each stage. The crushed ore stockpile will be covered to prevent freezing;
- 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 circuit will consist of eight tanks fitted with mechanical agitators, an initial pre-oxidation tank with cyanide being added to the second and subsequent tanks. The leach residence time will be 48 hours;
- Carbon in pulp adsorption of gold and silver will be carried out in a “carousel” unit, with “pump cells” moving leached slurry between the six tank units while the carbon remains in the same tank until fully loaded;
- The loaded carbon will be treated in a Zadra elution and electrowinning circuit consisting of an acid wash column and two elution columns operating at 140 degrees Celsius. A propane heater will provide the necessary temperature and two additional heat exchangers will control the temperature around the circuit. A rotary kiln operating at 700 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 air system will be carried out in the final tailings slurry, with the sulphur dioxide being produced by the combustion of elemental sulphur.

25.12 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;

- A camp on site that used during exploration 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;

- 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.

25.13 Waste Characterization

- Waste rock was classified based on its Neutralizing Potential Ratio (NPR) and metal leaching (ML) potential as follows:
  - 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);

- PAG1 and PAG2 waste rock will be stored in the TSF or used in construction of upstream zones of the dams as required in a manner that provides submergence in the TSF within one year of mining to prevent the formation of acid rock drainage (ARD). NAG3 waste rock will be stored in the TSF or used in construction of the dams as required in a manner that provides submergence within three to five years of mining to reduce metal leaching potential. NAG4 waste rock will be used for construction on site and in the downstream shell of the TSF dams. NAG5 will be used for construction on site and in the downstream shell of the TSF dams. Overburden is classified as non-acid generating and will be used for construction on and off site. Sulphide and transition ore tailings are classified as potentially acid generating and will be kept saturated or submerged within the TSF during operations to prevent ARD. Oxide tailings exhibit low ARD and ML potential.

25.14 Tailings Storage Facility

- The TSF comprises two adjacent sites, TSF Site C and TSF Site D, which are staged to support initial mine operations and the two subsequent expansion stages. The starter dam for TSF Site C was relocated slightly downstream to optimize initial capacity and haulage distances, improve constructability by following existing access trails in an area of gentler terrain, and simplify water management during early operations;

- The TSF has been designed permanently store 462 Mm3 of tailings and PAG/ML waste rock material and will require approximately 83 Mm3 of construction material with approximately 95% being supplied by NAG waste rock and overburden from development of the open pit. The dam construction materials balance is integrated with the mine plan to minimize the need for additional external borrow material sources following for initial site establishment and early TSF construction;

- Several borrow sources should be available in the vicinity of the TSF basin, included pit-run granular fill materials for the dam shell, fine-grained glacial till for the core zone, and aggregate materials that could be crushed and/or screened to produce desirable quantities and grain size distributions for engineered fill materials;

- The TSF embankments will be engineered, water-retaining, zoned earthfill/rockfill dams with a compacted low-permeability core and appropriate filter/transition zones. A total of three embankments will be constructed across the two sites. The TSF Site C and Site D dams will be expanded using centreline construction methods;

- The staging of the TSF delays some environmental impacts of the facility and capital expenditures to the extent practicable while not changing the final footprint of the facility that completed a coordinated provincial and federal environmental assessment successfully in 2019;

- The design of the TSF is supported by extensive geotechnical site investigations completed in 2012, 2013, and 2019. Recommendations for additional supplemental geotechnical and hydrogeological site investigations are included in Section 26.

25.15 Water Management

- All drainage from the mine will flow by gravity into the TSF to simplify water management, spill control, and mine closure;
- Water within the Project area will be used by collecting runoff from the mine site area and recycling process water to the maximum practicable extent. Site runoff water will be collected and stored within the TSF and used to inundate the PAG/NAG3 waste rock and tailings solids to limit the potential for ARD and ML. Water will be stored in the supernatant ponds within the TSF and recycled to the mill for use in the process;
- Runoff water from undisturbed areas not needed for ore processing will be diverted around the mine facilities and fresh water will be pumped from Tatelkuz Lake to offset flow reductions in lower Davidson Creek and meet instream flow needs for downstream fisheries;
- Planned installation of a water treatment plant 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.

25.16 Environmental Considerations

- The Project has successfully completed the provincial and federal environmental assessment processes and has been awarded a Environmental Assessment Certificate (provincial) and positive Decision Statement (federal);
- Extensive environmental baseline and social studies were conducted in support of the provincial and federal environmental assessment process. Most of the key environment baseline studies continued throughout and subsequent to the environmental assessment process, which provides a robust environmental baseline data set supporting project design and development;
- As a result of the commitments made during the environmental assessment and in support of provincial and federal permitting, several monitoring and management plans will be developed to ensure appropriate mitigation of potential project effects during construction, operation, closure, and post-closure phases;

- The proposed Blackwater mine site is primarily located within the asserted traditional territories of Lhoosk'uz Dené Nation and Ulkatcho First Nation. New Gold entered into a trilateral Participation Agreement with these two Indigenous nations on April 18, 2019, who, following completion of the environmental assessment process, confirmed their support for the Project and consented to the BC EA Certificate. On closing of the acquisition of the Blackwater Project, the Participation Agreement was assigned to BW Gold.

- The Project could also interact with the Carrier Sekani First Nations and their Aboriginal title, rights, and interests as a result of the transmission line, use of road to access the site, and potential downstream water quality and other effects. New Gold has committed to continuing negotiations with the CSFNs with the goal of reaching a mutually acceptable participation agreement that will include accommodative measures and other benefits. Artemis remains committed to this goal;

25.17 Closure Plan

- The primary objective of the closure and reclamation initiatives will be to eventually return the Project site to a self-sustaining facility that satisfies end land use objectives. The facilities will be reclaimed according to accepted practices at the time of closure and in a manner that maintains long-term geochemical and physical stability;

- All buildings not needed beyond closure will be removed, disturbed lands rehabilitated, and the property will be returned to otherwise functional use according to approved reclamation plans;

- Site infrastructure required for water management following closure will be maintained and operated according to approved closure water management plans developed in consultation with First Nations, government, and other stakeholders to the Project.

- Cost of closure has been incorporated into the economic analysis;

25.18 Permitting

- A large number of federal or provincial permits are required for mine construction and subsequent operations. The process of obtaining the provincial permits is partially coordinated through a Mine Development Review Committee. Some of the provincial permits have legislated timelines, while others do not. The federal permits and authorizations do not have legislated timelines, and some permits originating from federal agencies may not be issued within the review period set out in the BC legislation.

- Key permits, licenses, and authorizations required to construct the mine include:
  - BC Mines Act permit;

- BC Environmental Management Act permits for discharges from the Project to surface waters and for air emissions;
- Schedule 2 amendment under the federal Fisheries Act, which is required to place mine waste into a natural water body that is frequented by fish;
- Section 35(2) authorization under the federal Fisheries Act for a harmful alteration, disruption, or destruction of fish habitat;
- Section 23 application or exemption under the federal Navigable Waters Protection Act for effects on navigable waters;
- Licenses or approvals under the BC Water Act for the use of water;
- Mining Lease.

25.19 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.

25.20 Capital Costs

- The initial capital cost for Phase 1 at 5.5 Mtpa is estimate is $592 million;
- The expansion capital cost for Phase 2 to achieve 12 Mtpa is estimate is $426 million;
- The expansion capital cost for Phase 3 to achieve 20 Mtpa is estimate is $398 million;
- Total LOM sustaining capital is estimated to be $636.5 million.

25.21 Operating Costs

- LOM operating costs are estimated at $17.65/t of ore milled;
- AISC is $668/oz in Stage 1, $696/oz in Stage 2, and $911/oz in Stage 3. The higher AISC in Phase 3 is mainly attributed to the inclusion of closure costs at the end of the life of mine.

25.22 Financial Analysis

- For the 23-year mine life and 334 Mt mill feed, the following after-tax Base Case financial parameters were calculated:
  - $2,247 million NPV at 5.0% discount rate;

  - 35% IRR;
  - 2.0 year initial capital payback

- For a leveraged case assuming initial capital is 60% debt financed at an annual interest rate of 5.5%, an upfront financing fee of 3%, and a 7-year term post commencement of commercial production with a balloon payment of 30% of the principal at maturity, the following after-tax Base Case financial parameters were calculated:
  - $2,249 million NPV at 5.0% discount rate;
  - 50% IRR;
  - 2.2 year initial capital payback
- Sensitivity analysis was performed on the Project using metal price (grade), exchange rate, operating costs and capital costs. 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.

25.23 Risks and Opportunities

- The major risks to the Project were identified as:
  - Changes to metal prices and exchange rate assumptions;
  - Capital cost growth;
  - Increases in operating costs;
  - Productivity assumptions;
  - Dilution control;
  - Presence of high-grade silver in the mill feed;
  - Integration of mining operations and the TSF construction
  - Permitting delays;
  - Lack of social license affecting permit grant;
- Project opportunities included:
  - 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;

- o Assessment of methods to reduce waste mining costs;
- o Use of oxygen rather than compressed air for cyanide leaching and cyanide detoxification;
- o Pre-crushing the mill feed to a finer feed size;
- o Value engineering initiatives.

25.24 Conclusions

MMTS considers that the scientific and technical information available on the Project can support proceeding with additional data collection, trade-off and engineering work and preparation of more detailed studies. However, the decision to proceed with a mining operation on the Project is at the discretion of Artemis.

26 RECOMMENDATIONS

It is recommended that a feasibility study be conducted on the Project, through a two-phase work program. The first phase of work will collect information that will be used to refine the feasibility study outcomes in Phase 2.

26.1 Phase 1, Field Work to prepare for a Feasibility Study

26.1.1 Exploration and Resource

- The following exploration budget is recommended to test the potential for the extension of a high-grade zone in the northwest of the deposit at depth. This should consist of two drillholes of approximately 1,200 m total length. The costs related to the exploration program include drilling, mobilization, camp, crew transport, logging, and assay charges are estimated at $225/m all-in. The total exploration drill program is therefore estimated at $270,000.

- 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.

26.1.2 Metallurgy

The following metallurgical testwork is recommended:

- Some samples contain higher amounts of silver which must be adsorbed by carbon together with gold and recovered by elution and electrowinning. Carbon adsorption test work is required to confirm that the adequate capacity is available on the carbon based on the silver grades of the ore as defined by the mine plan.

- Although excellent recoveries were obtained from composites representative of the first 10 years of operation, some six samples from 48 variability samples gave lower recoveries and these should be re-tested and their position in the deposit identified to allow testing of adjacent samples.

- Cyanide consumption was higher in later test work, possibly due to oxidation of the samples or experimental technique and confirmation is required.

The metallurgical test work is expected to cost $50,000.

26.1.3 Mine Planning

The following recommendations are made with regard to future mining studies:

- Execute a grade control drilling and interpretation program in selected areas of the Phase 1 open pit that are planned to be mined for initial mill feed. The resultant tonnes and grade from this interpretation should be compared to the equivalent area resource modelled tonnes and grade. Results to be incorporated in ongoing grade control strategy and mine planning.

- 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.

- Re-analyze existing geotechnical core and re-interpretations with additional data collected from 2012 to 2020.

- Detail out opportunities to mine on larger benches in waste zones, using a larger equipment fleet.

- Run a trade-off study to investigate the merits of implementing a trolley assist system for the haul trucks, as well as a crushing and conveying system for transporting waste rock to the tailings facility.

- Detail out existing design work to a feasibility level of engineering, including open pits, stockpiles, ex-pit haul roads, explosives storage facilities, and fleet maintenance facilities.

These field programs and studies are estimated to cost approximately $5,000,000 The majority of this will be required for the grade control program.

26.1.4 Geotechnical and Hydrological

- Perform supplemental geotechnical and hydrogeological site investigations focused on dam foundations and borrow areas required during the first five years of mine operations, including geophysics and pumping tests at existing instrumentation and completion of approximately 30 sonic drillholes at the adjusted Site C Main Dam, Interim Environmental Control Dam, and FWR with co-located Cone Penetration Testing and low-disturbance sampling at select locations.

These programs and studies are estimated to cost approximately $3,000,000,

26.2 Phase 2, Feasibility Study

A feasibility study is recommended as Phase 2 at an estimated cost of $2,100,000. Phase 2 will incorporate the results of Phase 1.

27 REFERENCES

The following references were used in the compilation of the 2014 FS and support this Report. Additional references specific to the Report are also included.

Adams, J., and S. Halchuk, 2003: Fourth Generation Seismic Hazard Maps of Canada: Values for Over 650 Canadian Localities Intended For the 2005 National Building Code of Canada: Geological Survey of Canada, Open File 4459

AMEC, 2012a: P00169229-01 Blackwater Database Verification Review: report prepared for New Gold, 31 January 2012 Final

AMEC, 2012b: P00169229-02 Blackwater Database Verification Review: report prepared for New Gold, 24 February 2012

AMEC, 2012c: P00169229-04 Blackwater Site Visit 20-21: report prepared for New Gold, September 2012 Draft

AMEC, 2012d: P00169229-05 Blackwater Database Verification: report prepared for New Gold, September 2012 Draft v1

AMEC, 2012e: P00169229-06 Blackwater Database Verification: report prepared for New Gold, 19 November 2012 Draft v3

AMEC, 2013a: P00171193-03 Blackwater YE 2012 Database Verification Review: report prepared for New Gold, 23 January 2013 Final

AMEC, 2013b: P00171193-04 Blackwater YE to FS Resource Estimate Drillhole Database Review: report prepared for New Gold, March 2013 Final

AMEC, 2013c: P00171193-01 Blackwater 2013 FS Resource Estimate Review: report prepared for New Gold, March 2013 Draft v3

AMEC, 2013d: Instream Flow – Switch Decision for Chedakuz/Davidson/Tatelkuz Triangle: report prepared for New Gold, AMEC File No. VE52277, September 27, 2013

AMEC, 2013f: Blackwater Project, BC Feasibility Study: unpublished report prepared by AMEC, Knight Piésold, Geosim, Norwest Corp, and Allnorth, December 6, 2013, 17 vols

AMEC, 2014a: Blackwater Gold Project, British Columbia, NI 43-101 Technical Report on Feasibility Study. January 2014.

Atkinson, G.M., and D.M. Boore, 2003: Empirical Ground-Motion Relations for Subduction-Zone Earthquakes and Their Application to Cascadia and Other

Regions: Bulletin of the Seismological Society of America, Vol. 93, No. 4, p 1703–1729

BC Hydro, 2012: RS 1289 Tariff Amendment:
http://www.bchydro.com/etc/medialib/internet/documents/appcontent/your_account/BC_Hydro_Electric_Tariff.Par.0001.File.BC-Hydro-Tariff.pdf Accessed May 11, 2012

BC Ministry of Environment, Lands, and Parks (BC MOELP), 1996: Guidelines for Assessing the Design, Size and Operation of Sedimentation Ponds Used in Mining: November 1996

Blast Dynamics, Inc., 2012: Drilling and Blasting Feasibility Study – Blackwater Gold Project, December 2012.

British Columbia Mine Waste Rock Pile Research Committee (BCMWRPRC), 1991: Mine Rock and Overburden Piles: Investigation and Design Manual: Interim Guidelines, May 1991

British Columbia Ministry of Energy and Mines (BCMEM), 1998: Policy for Metal Leaching and Acid Rock Drainage at Mine Sites in British Columbia

Canadian Dam Association (CDA), 2007: Dam Safety Guidelines

Canadian Securities Administrators (CSA), 2011: National Instrument 43-101, Standards of Disclosure for Mineral Projects, Canadian Securities Administrators.

Canadian Institute of Mining, Metallurgy and Petroleum (CIM), 2019: Estimation of Mineral Resources and Mineral Reserves, Best Practice Guidelines: Canadian Institute of Mining, Metallurgy and Petroleum, November 29, 2019.

Canadian Institute of Mining, Metallurgy and Petroleum (CIM), 2014: CIM Standards for Mineral Resources and Mineral Reserves, Definitions and Guidelines: Canadian Institute of Mining, Metallurgy and Petroleum, May, 2014.

Caterpillar Inc., 2016: Caterpillar Performance Handbook: 46th edition, January 2016.

Cornell, C.A., 1968: Engineering Seismic Risk Analysis: Bulletin of the Seismological Society of America, Vol. 58, p 1583–1606

Diakow, L.J., and V.M. Levson, 1997: Bedrock and Surfical Geology of the Southern Nechako Plateau, Central British Columbia (93F/2,3,6,7): Geoscience Maps 1997- 2 (map)

Diakow, L.J., Webster, I.C.L., Richards, T.A., and H.W. Tipper, 1997: Geology of the Fawnie and Nechako Ranges, Southern Nechako Plateau, Central British Columbia (96F/2,3,6,7), Paper 1997-2: *in* Interior Plateau Geoscience Project: Summary of Geological, Geochemical and Geophysical Studies

Diakow, L.J., Webster, I.C.L., Levson, V.M., and T.R. Giles, 1994: Bedrock and Surficial Geology of the Fawnie Creek Map Area (93F/3): Open File 1994-2, BCMEMPR (Geological Survey of Canada)

Earthquake Spectra, 2008: Special Issue on the Next Generation Attenuation Project: Vol. 24, No. 1

ERM Consultants Canada Ltd. (ERM), 2015. Blackwater Gold Project: Evaluation of Alternative Tailings Technologies for the Blackwater Project. Prepared for New Gold Inc., Vancouver BC November 2015.

EZ-FRISK, 2008: Software for Earthquake Ground Motion Estimation, Version 7.32: Risk Engineering Inc., Boulder, Colorado, USA

Freeze, A.R., and R.A. Cherry, 1979: Groundwater: Prentice-Hall, Englewood Cliffs, New Jersey

Friedman, R.M, L.J. Diakow, R.A. Lane, and J.K. Mortensen, 2001: New U–Pb age Constraints on Latest Cretaceous Magmatism and Associated Mineralization in the Fawnie Range, Nechako Plateau, Central British Columbia: Canadian Journal of Earth Sciences, Vol. 38, No. 4, p 619-637

International Commission on Large Dams (ICOLD), 1995: Tailings Dams and Seismicity: Review and Recommendations: Bulletin 98

International Network for Acid Prevention (INAP), 2009: Global Acid Rock Drainage (GARD) Guide: www.gardguide.com

Knight Piésold Ltd., 2011a: Blackwater Gold Project – Tailings Storage Facility Site Alternative Assessment: report prepared for New Gold, VA10-00235, January 2011

Knight Piésold Ltd., 2011b: Blackwater Gold Project – Tailings Disposal Options – Conceptual Planning and Cost Estimates: report prepared for New Gold, VA11-01276, September 2011

Knight Piésold Ltd., 2012a: Blackwater Gold Project Updated Mine Development Concept Evaluation Considering Potentially Acid Generating (PAG) Disposal On-land and Subaqueously in the Tailings Facility: report prepared for New Gold, VA11-01960, April 2012

Knight Piésold Ltd., 2012b: Blackwater Gold Project – Fresh Water Supply Options Trade-off Study: report prepared for New Gold, VA12-00758 Rev. B, August 2, 2012

Knight Piésold Ltd., 2013a. 2012 Site Investigation Report (VA101-457/6-1), dated September 23, 2013.

Knight Piésold Ltd., 2013b. Feasibility Open Pit Slope Design Report (VA101-457/6-2), dated November 4, 2013.

Knight Piésold Ltd., 2013c. 2013 Site Investigation Report (VA101-457/6-4), dated November 7, 2013.

Knight Piésold Ltd., 2013d. Tailings Lab Testing Report (VA101-457/6-7), dated November 4, 2013.

Knight Piésold Ltd., 2013e. Geotechnical Characterization Report (VA101-457/6-8), dated November 29, 2013.

Knight Piésold Ltd., 2013f. Open Pit Water Management Plan Report (VA101-457/6-9), dated November 29, 2013.

Knight Piésold Ltd., 2013g. Construction Sediment and Erosion Control Plan Report (VA101-457/6-10), dated November 29, 2013.

Knight Piésold Ltd., 2013h. 2013 Hydrometeorology Report (VA101-457/6-12), dated November 4, 2013.

Knight Piésold Ltd., 2013i. Revised Baseline Tatelkuz Lake Levels (VA101-457/6-A.01, Continuity No. VA13-01604). July 22, 2013.

Knight Piésold Ltd., 2013j. Tatelkuz Lake IFN Withdrawal Model (VA101-457/6-A.01, Continuity No. VA13-02066). November 22, 2013.

Knight Piésold Ltd., 2013k. Blackwater Project – Feasibility Study Water Balance Model (VA101-457/6-A.01, Continuity No. VA13-00972). September 19, 2013.

Knight Piésold Ltd., 2013l. Blackwater Gold Project – Tailings Pipeline Design (VA101- 457/6-A.01, Continuity No. VA13-01900). October 15, 2013.

Knight Piésold Ltd., 2013n. Transmission Line Feasibility Study (VA101-457/6-3), dated September 3, 2013.

Knight Piésold Ltd., 2013o. Plant Site Geotechnical Design Report (VA101-457/6-5), dated November 27, 2013.

Knight Piésold Ltd., 2014: Blackwater Gold Project – Mine Waste and Water Management Design Report. Ref. No. VA101-00457/6-11, Rev. 2, September 2014.

Knight Piésold Ltd., 2015: Blackwater Gold Project – Alternative Technology Options Study. Ref. No. VA101-00457/15-1, Rev. 0, December 2015.

Knight Piésold Ltd., 2018: Blackwater Gold Project – 2016 Hydrometeorology Report. Ref. No. VA101-00457/22-3, Rev. 2, August 29, 2018.

Levson, V. M., Weary, G. F., and T.R. Giles, 1995: Surficial Geology and Drift Exploration Studies in the Tsacha Lake and Chedakuz Creek Areas: BCGS Geological Fieldwork

Mate, D. J., and V.M. Levson, 2000: Quaternary stratigraphy and history of the Ootsa Lake-Cheslatte River area, Nechako Plateau, central British Columbia: Canadian Journal of Earth Science.

McClelland Laboratories, 2012: MLI3597 Interim Results: report prepared for New Gold, 2012

Mine Environment Neutral Drainage (MEND), 2009: Prediction Manual for Drainage Chemistry from Sulphidic Geologic Materials: MEND Report 1.20.1

Mortensen, J.K., 2011: U-Pb Zircon Dating of Volcanic and Intrusive Rock Samples from the Vicinity of The Blackwater-Davidson Deposit By Laser Ablation ICP- MS: internal report to Richfield Ventures Inc., 10 p

New Gold, 2012a: 201202 PEA QAQC Report: internal company report New Gold, 2012b: 201203 QAQC Report: internal company report

New Gold, 2012c: 201204 QAQC Report: internal company report New Gold, 2012d: 201205 QAQC Report: internal company report New Gold, 2012e: 201206 QAQC Report: internal company report New Gold, 2012f: 201207 QAQC Report: internal company report New Gold, 2012g: 201208 QAQC Report: internal company report New Gold, 2012h: 201209 QAQC Report: internal company report

New Gold, 2012i: 201209 to 201211 YE QAQC Report: internal company report New Gold, 2012j: 201211 YE to 201302 FS QAQC Report: internal company report

Norwest Corporation, 2013: Blackwater Project Feasibility Level Mining Study, December 2013.

Price, 1997: Draft Guidelines and Recommended Methods for the Prediction of Metal Leaching and Acid Rock Drainage at Mine Sites in British Columbia: BCMEM0

Riddell, J., 2011: Lithostratigraphic and Tectonic Framework of Jurassic and Cretaceous Intermontane Sedimentary Basins of South-Central British Columbia: Canadian Journal of Earth Sciences, 48, p 870-896.

SGS Canada, 2012: 50179-001 Interim Results: report prepared for New Gold, 2012

Sillitoe, R.H., and Hedenquist, J.W., 2003: Linkages between Volcanotectonic Settings, Ore-Fluid Compositions, and Epithermal Precious Metal Deposits: Society of Economic Geologists Special Publication 10, p. 315-343

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, effective date 28 August 2012.

Stanley, C.R., and Lawie, D., 2007: Average Relative Error in Geochemical Determinations: Clarification, Calculation, and a Plea for Consistency: Exploration and Mining Geology, Vol. 16 No.3–4, p 265–274

Todd, D.K., 1959: Ground Water Hydrology: sixth printing, 1967, John Wiley & Sons, Inc. New York

Youngs, R.R., Chiou, S.J., Silva, W.J., and J. R. Humphrey, 1997: Strong Ground Motion Attenuation Relationships for Subduction Zone Earthquakes: Seismological Society of America, Seismological Research Letters, Vol. 68, No. 1, p 58–73