Skeena Resources Limited

Eskay Creek Project

British Columbia, Canada

NI 43-101 Technical Report on Preliminary Economic Assessment

Report Effective Date: 7 November 2019

Prepared for: Skeena Resources Limited

Prepared by:

Mr Robin Kalanchey, P.Eng. (Ausenco)
Mr Scott Elfen, P.E. (Ausenco)
Mr Scott Weston, P.Geo (Hemmera)
Ms Sheila Ulansky, P.Geo. (SRK)
Mr Adrian Dance, P.Eng. (SRK)
Mr Gordon Zurowski, P.Eng. (AGP)
Mr Willie Hamilton, P.Eng. (AGP)

CERTIFICATE OF QUALIFIED PERSON

I, Robin Kalanchey, P.Eng., am employed as a Director, Minerals & Metals – Western Canada with Ausenco Engineering Canada Inc. (Ausenco), with an address at 1050 West Pender Street, Vancouver, BC V6E 3S7.

This certificate applies to the technical report titled “Eskay Creek Project, British Columbia, Canada, NI 43-101 Technical Report on Preliminary Economic Assessment” dated 7 November 2019 (the “technical report”), prepared for Skeena Resources Limited (Skeena).

I am a Professional Engineer registered with the Association of Professional Engineers and Geoscientists of Alberta, member number 61986. I graduated from University of British Columbia with a Bachelor of Applied Science degree in metals and materials engineering in 1996.

I have practiced my profession for 23 years. I have been directly involved in mineral processing and metallurgical testing, metallurgical process plant design and engineering, and metallurgical project evaluations for gold, nickel, cobalt, copper, zinc and molybdenum projects in numerous countries.

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

I have not visited the Eskay Creek property.

I am responsible or co-responsible for Sections 1.1, 1.2, 1.13, 1.16, 1.17.1, 1.17.3, 1,18.1, 1.18.3, 1,19, 1.20, 1.21.1.3, 1.21.2.4, 1.21.2.5, 1.22, 1.23; Sections 2.1, 2.2, 2.3, 2.5, 2.6; Section 3; Section 17; Section 19; Sections 21.1, 21.2.1, 21.2.6, 21.3.1, 21.3.3, 21.3.4, 21.4; Section 22; Section 24; Sections 25.1, 25.8, 25.11, 25.12, 25.13, 25.14, 25.15.1.3, 25.15.2.4, 25.15.2.5, 25.16; Sections 26.1, 26.2.3, 26.2.4, 26.3; and Section 27 of the technical report.

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

I have been involved with the Eskay Creek property since 2019, during preparation of the Preliminary Economic Assessment report.

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

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

Dated: 20 December 2019

“Signed and sealed”

Robin Kalanchey, P.Eng.

CERTIFICATE OF QUALIFIED PERSON

I, Scott Elfen, P.E., am employed as a VP Global Lead Geotechnical Service with Ausenco Engineering Canada Inc., with an address at 855 Homer Street, Vancouver BC, Canada V6B 2W2.

This certificate applies to the technical report titled "Eskay Creek Project, British Columbia, Canada, NI 43-101 Technical Report on Preliminary Economic Assessment" dated 7 November, 2019 (the "technical report"), prepared for Skeena Resources Limited (Skeena).

I am a Registered Civil Engineer in the State of California (No. C56527) since 1996.

I graduated from University of California, Davis with a Bachelor of Science degree in Civil Engineering (Geotechnical Emphasis) in 1991.

I have practiced my profession for 28 years since graduation. I have been directly involved in geotechnical, civil, hydrology and environmental aspects for the development of mining projects, including preliminary economic assessments to construction of numerous underground and open pit base metal and precious metal deposits in the Americans, Russia, Africa, and Australia.

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

I have not visited the Eskay Creek property.

I am responsible or co-responsible for Sections 1.1, 1.2, 1.14, 1.21.1.4, 1.23; Sections 2.1 to 2.3, 2.5, 2.6; Section 3.1, Sections 18.1, 18.2, 18.5 to 18.11; Sections 25.1, 25.9, 25.15.1.4; Sections 26.2.7, 26.2.8, 26.2.9, 26.3; and Section 27 of the technical report.

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

I have been involved with the Eskay Creek property since 2019, during preparation of the Preliminary Economic Assessment report.

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

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

Dated: 20 December, 2019

“Signed and sealed”

Scott Elfen

CERTIFICATE OF QUALIFIED PERSON

I, Scott Weston, P.Geo., am employed as the Vice President, Business Development, with Hemmera Envirochem Inc., an Ausenco company, with an address at 18th Floor, 4730 Kingsway, Burnaby, BC, V5H 0C6.

This certificate applies to the technical report titled "Eskay Creek Project, British Columbia, Canada, NI 43-101 Technical Report on Preliminary Economic Assessment" dated 7 November, 2019 (the "technical report"), prepared for Skeena Resources Limited (Skeena).

I am registered as a Professional Geoscientist (P.Geo.) with Engineers and Geoscientists British Columbia (EGBC).

I graduated from the University of British Columbia in 1995 with a Bachelor of Science degree and from the Royal Roads University in 2001 with a Master of Science degree.

I have practiced my profession for 24 years, and have experience in geomorphic and landscape genesis interpretation, soil science, terrain stability analysis, natural hazard assessment, hydrology, and environmental impact assessment related to mining projects in Canada and Brazil.

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

I have not visited the Eskay Creek property.

I am responsible or co-responsible for Sections 1.1, 1.2, 1.12.2, 1.15, 1.21.1.5, 1.23; Sections 2.1 to 2.3, 2.5, 2.6; Section 16.4; Section 20; Sections 25.1, 25.7.2, 25.10, 25.15.1.5; Sections 26.1, 26.2.7, 26.2.8, 26.3; and Section 27 of the technical report.

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

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

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

Dated: 20 December, 2019

“Signed and sealed”

Scott Weston, P.Eng.

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CERTIFICATE OF QUALIFIED PERSON

I, Sheila Ulansky, P.Geo., am employed as a Senior Resource Consultant with SRK Consulting (Canada) Inc., with an address at 2200-1066 West Hastings Street, Vancouver, BC, V6E 3X2

This certificate applies to the technical report titled "Eskay Creek Project, British Columbia, Canada, NI 43-101 Technical Report on Preliminary Economic Assessment" dated 7 November, 2019 (the "technical report"), prepared for Skeena Resources Limited (Skeena).

I am a Professional Geoscientist of Engineers and Geoscientists British Columbia Association; EGBC#36085.

I graduated from the University of Victoria, British Columbia, with a B.Sc. in 2007, and from Laurentian University, Ontario, with a M.Sc. in 2019.

I have practiced my profession for twelve years since graduation. I have been directly involved in previous NI 43-101 studies, including: Preliminary Economic Assessment of the Re-opening of the Au-Ag-Cu Aranzazu Mine Zacatecas, Mexico, and two Technical Reports on the Eskay Creek Au-Ag Project in British Columbia, Canada.

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 Eskay Creek property from 27–28 June, 2018.

I am responsible or co-responsible for Sections 1.1 to 1.8, 1.10, 1.11, 1.21.1.1, 1.21.2.1, 1.21.2.2, 1.23; Section 2; Section 4; Section 5; Section 6; Section 7; Section 8; Section 9; Section 10; Section 11; Section 12; Section 14; Section 23; Sections 25.1 to 25.4, 25.6, 25.15.1.1; 25.15.2.1, 25.15.2.2; Sections 26.2.1, 26.2.2; and Section 27 of the technical report.

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

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I have been involved with the Eskay Creek property since 2018, and have co-authored the following reports on the property:

- Ulansky, S., Uken, R., and Carlson, G., 2019: Independent Technical Report on the Eskay Creek Au-Ag Project, Canada: report prepared by SRK Consulting (Canada) Inc. for Skeena, effective date 28 February, 2019.
- Ulansky, S., Uken, R., and Carlson, G., 2018: Independent Technical Report on the Eskay Creek Au-Ag Project, Canada: report prepared by SRK Consulting (Canada) Inc. for Skeena, effective date 6 July, 2018.

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

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

Dated: 20 December, 2019

“Signed and sealed”

Sheila Ulansky, P.Geo.

CERTIFICATE OF QUALIFIED PERSON

I, Adrian Dance, P.Eng., am employed as a Principal Consultant with SRK Consulting Canada) Inc., with an address at 2200-1066 West Hastings St., Vancouver, BC Canada.

This certificate applies to the technical report titled “Eskay Creek Project, British Columbia, Canada, NI 43-101 Technical Report on Preliminary Economic Assessment” dated 7 November, 2019 (the “technical report”), prepared for Skeena Resources Limited (Skeena).

I am a Professional Engineer registered with the Association of Professional Engineers & Geoscientists of British Columbia, license # 37151. I am a graduate of the University of British Columbia in 1987 where I obtained a Bachelor of Applied Science and a graduate of the University of Queensland in 1992 where I obtained a Doctorate.

I have practiced my profession for 27 years since graduation including 17 years as a consultant and have experience working in a number of gold operations around the world.

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

I have not visited the Eskay Creek property.

I am responsible or co-responsible for Sections 1.1, 1.2, 1.9; Sections 2.1, 2.2, 2.3, 2.5, 2.6; Section 13; Sections 25.1, 25.5; and Section 27 of the technical report.

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

I have been involved with the Eskay Creek property since 2019 during the preparation of the Preliminary Economic Assessment report.

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

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

Dated: 20 December, 2019

“Signed”

Adrian Dance, P.Eng.

CERTIFICATE OF QUALIFIED PERSON[1]

I, Gordon Zurowski, P.Eng., am employed as a Principal Mine Engineer with AGP Mining Consultants Inc., with an address at #246-132 Commerce Park Dr., Unit K, Barrie, ON, L4N 0Z7.

This certificate applies to the technical report titled "Eskay Creek Project, British Columbia, Canada, NI 43-101 Technical Report on Preliminary Economic Assessment" dated 7 November, 2019 (the "technical report"), prepared for Skeena Resources Limited (Skeena).

I am a Professional Engineer of the Professional Engineers of Ontario, member #100077750. I graduated from the University of Saskatchewan, with a Bachelor of Applied Science in Geological Engineering in 1989.

I have practiced my profession for 30 years. I have been directly involved in open pit mining including operating, design and evaluation in Canada and worldwide.

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

I have not visited the Eskay Creek property.

I am responsible or co-responsible for Sections 1.1, 1.2, 1.17.1, 1.17.2, 1.18.1, 1.18.2, 1.21.1.2, 1.21.2.3, 1.23; Sections 2.1 to 2.6; Section 16.3; Sections 21.1, 21.2.1, 21.2.2, 21.2.3, 21.2.4, 21.2.5, 21.3.1, 21.3.2, 21.4; Sections 25.1, 25.12, 25.13, 25.15.1.2, 25.15.2.3; Sections 26.2.5 to 26.2.6; and Section 27 of the technical report.

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

I have been involved with the Eskay Creek property since 2019, during preparation of the Preliminary Economic Assessment report.

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

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

Dated: 20 December, 2019

“Signed and sealed”

Gordon Zurowski

CERTIFICATE OF QUALIFIED PERSON

I, Willie Hamilton, P.Eng., am employed as a Senior Associate Mining Engineer with AGP Mining Consultants Inc, with an address at #246-132 Commerce Park Dr., Unit K, Barrie, ON, L4N 0Z7.

This certificate applies to the technical report titled "Eskay Creek Project, British Columbia, Canada, NI 43-101 Technical Report on Preliminary Economic Assessment" dated 7 November, 2019 (the "technical report"), prepared for Skeena Resources Limited (Skeena).

I am a Professional Engineer of the Association of Professional Engineers and Geoscientists of Alberta, member #47481. I graduated from the University of Alberta with a Bachelor of science in Mining Engineering in 1988 and a Master of Science in Mining Engineering in 1990.

I have practiced my profession for 30 years. I have been directly involved in open pit and underground mines in both operational and mine planning roles in Canada and the United States.

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 Eskay Creek property most recently from 21–22 August, 2019.

I am responsible or co-responsible for Sections 1.1, 1.2, 1.12.1, 1.12.3, 1.21.1.2, 1.21.2.3, 1.23; Sections 2.1 to 2.6; Section 15; Section 16 except for Sections 16.3 and 16.4; Sections 18.3, 18.4; Sections 25.1, 25.7.1, 25.7.3, 25.15.1.2, 25.15.2.3; Sections 26.2.5, 26.2.6; and Section 27 of the technical report.

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

Dated: 20 December, 2019.

“Signed and sealed”

Willie Hamilton, P.Eng

Important Notice

This report was prepared as National Instrument 43-101 Technical Report for Skeena Resources Limited (Skeena) by Ausenco Engineering Canada Inc. (Ausenco), Hemmera Envirochem Inc., an Ausenco company (Hemmera), SRK Consulting Canada Inc. (SRK), and AGP Mining Consultants Inc. (AGP), collectively the Report Authors. The quality of information, conclusions, and estimates contained herein is consistent with the level of effort involved in the Report Authors' services, based on i) information available at the time of preparation, ii) data supplied by outside sources, and iii) the assumptions, conditions, and qualifications set forth in this report. This report is intended for use by Skeena subject to terms and conditions of the Report Authors' respective contracts with Skeena. Except for the purposed legislated under Canadian provincial and territorial securities law, any other uses of this report by any third party is at that party's sole risk.

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1 SUMMARY

1.1 Introduction

Ausenco Engineering Canada Inc. (Ausenco), Hemmera Envirochem Inc., an Ausenco company (Ausenco), SRK Consulting (Canada) Inc. (SRK), and AGP Mining Consultants Inc. (AGP), have prepared an updated preliminary economic assessment (PEA) report for Skeena Resources Limited (Skeena) on the volcanogenic massive sulphide (VMS) Eskay Creek Project (the Project) located in British Columbia.

Skeena currently holds the Project through an option agreement with Barrick Gold Inc. (Barrick).

The Project hosts the previously-mined Eskay Creek deposit, which was in operation as an underground mine from 1995–2008.

1.2 Terms of Reference

The Report supports disclosures by Skeena in a news release dated 7 November 2019 entitled "Skeena Delivers Robust Project Economics for Eskay Creek: After-Tax NPV5% of C$638M, 51% IRR and 1.2 Year Payback".

All measurement units used in this Report are metric unless otherwise noted. Currency is expressed in Canadian (C) dollars (C$). The Report uses Canadian English. United States dollars, where referenced, are termed US$.

Mineral Resources and Mineral Reserves are reported in accordance with the Canadian Institute of Mining, Metallurgy and Petroleum (CIM) Definition Standards for Mineral Resources and Mineral Reserves (May 2014; the 2014 CIM Definition Standards) and the CIM Estimation of Mineral Resources and Mineral Reserves Best Practice Guidelines (November 2003; 2003 CIM Best Practice Guidelines).

As the ownership of, and ownership interests in, historical mining operations has changed hands numerous times during the production history, the Report uses the term "previous operator" to refer to work done from 1988 to 2017. The term "legacy" is used for data generated by the previous operator. Skeena obtained its option interest in December 2018.

1.3 Project Setting

The Eskay Creek Project is located in the Golden Triangle region of British Columbia, Canada, 83 km northwest of Stewart. Support services for mining and other resource sector industries in the region are provided primarily by the communities of Smithers (pop. 5,400) and Terrace (pop. 11,500). Both communities are accessible by commercial airlines with daily flights to and from Vancouver.

Access to the Eskay Creek Project is via Highway 37 (Stewart Cassiar Highway). The Eskay Mine Road is an all-season gravel road that connects to Highway 37 approximately 135 km north of Meziadin Junction. The Eskay Mine Road is a 54.5 km private industrial road that is operated by Altagas Ltd. (0 km to 43.5 km) and Skeena Resources Ltd. (43.5 km to 54.5 km). There are two nearby gravel air strips: Bronson Strip which is about 40 km west of the mine site and Bob Quinn, roughly 37 km northeast of the Eskay Creek Project.

The mean annual total precipitation at the former mine site is estimated to be 2,500 ± 500 mm. About 55–71% of precipitation falls as snow. The average temperature range is from -10.4°C in January to +15°C in July. Exploration activities can be curtailed by winter conditions. The previous mining operation was conducted on a year-round basis, and it is expected that any future operations will also be year-round.

The Eskay Creek Project lies in the Prout Plateau, a rolling subalpine upland with an average elevation of 1,100 m (amsl), located on the eastern flank of the Boundary Ranges. The plateau is characterized by northeast-trending ridges with gently-sloping meadows occupying valleys between the ridges. Relief over the plateau area ranges from 500 m in the existing Tom MacKay tailings storage facility (TMSF) area to over 1,000 m in the Unuk River and Ketchum Creek valleys. The plateau is drained by tributaries of the Stikine–Iskut and Unuk Rivers. The former Eskay Creek mine site is at approximately 800 m elevation. Mountain slopes are heavily forested. There are no known federal, provincial or regional parks, wilderness or conservancy areas, ecological reserves, or recreational areas near the Eskay Creek Project.

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

On December 18, 2017, Skeena and Barrick Gold Inc. entered into an Option Agreement on the Eskay Creek Property. This agreement affects all mineral claims and mineral leases that comprise the Eskay Creek Property, except for the single mineral claim registered to Skeena Resources Ltd. Skeena has the option to acquire all of Barrick's rights, title and interest in and to the Eskay Creek Assets (property and all facilities and portions of the Coast Road, an access road connecting the mine site to the public highway system), the permits (including the Coast Road Special Use Permit), and the Eskay Creek contracts by incurring $3.5 million of exploration expenditures in the Project area by December 18, 2020. In addition, to exercise the option Skeena must reimburse Barrick the aggregate amount of Barrick's reclamation expenditures during the Option Period plus pay a $10 million cash purchase price. Skeena must also post an environmental bond which in December 2017 was estimated at $7.7 million. If the reclamation expenditures and the bond requirement are in aggregate greater than $7.7 million, then the cash purchase payment will decrease by a corresponding amount. The purchase price, the reclamation expenditures reimbursement and the bond amount are collectively not to exceed $17.7 million. After closing Barrick will retain a 12 month back-in right to the Esaky project for 51% by paying Skeena three times its cumulative expense on the project and reimbursing the purchase price and 51% of the bond amount.

The Eskay Creek Project covers 5,093.81 ha, consisting of 40 mineral claims (3,263.55 ha), and eight mineral leases (1,830.26 ha). Where on-ground work commitments have not been met, Skeena has made cash-in-lieu payments as stipulated under BC regulations. All statutory annual reporting obligations have been met.

Royalties are payable on a number of the claims including a 1% NSR payable to Euro-Nevada Mining Corporation Limited (now Franco-Nevada Corp.); a 2% NSR payable to ARC Resource Group Ltd. (Option Agreement dated 4 November 1988 between ARC Resource Group Ltd. and Canarc Resources Corp.), a 2% NSR payable to ARC Resource Group Ltd. (Royalty Deed dated 1 August 1990 between Adrian Resources Ltd. and ARC Resource Group Ltd.), and a 1% NSR payable to David A. Javorsky. Should Skeena elect to purchase the project, a 1% royalty will be payable to Barrick on all of the claims, which will be in addition to the existing royalties. Should Skeena elect to purchase the project, a 1% royalty will be payable to Barrick on all of the claims. Should Barrick elect to exercise the claw-back interest clause in the option agreement, Barrick will obtain a 51% Project interest, and the 1% Barrick royalty will extinguish.

Skeena holds an interest in two surface leases and the Eskay Road access. Skeena will need to acquire surface rights in support of any future mining operations. No water rights are currently held. Skeena's current environmental liabilities are related to activities undertaken by Skeena, and activities arising from permitting. The key liabilities would be remediation of drill pads and drill access roads. Skeena has posted an environmental bond with the relevant BC authorities in relation to the work programs that have been conducted.

1.5 Geology and Mineralization

The Eskay Creek deposit is generally classified as an example of a high-grade, precious metals-rich epithermal volcanogenic massive sulphide (VMS) deposit; however, it has also been suggested to be an example of a subaqueous hot spring gold–silver deposit.

The Eskay Creek Project is located along the western margin of the Stikine Terrane, within the Intermontane Tectonic Belt of the Northern Cordillera. It is hosted within the Jurassic rocks of the Stikinia Assemblage at the stratigraphic transition from volcanic rocks of the uppermost Hazelton Group to the marine sediments of the Bowser Lake Group.

The Project area is underlain by volcanic and sedimentary rocks of the regionally extensive Lower to Middle Jurassic Hazelton Group. The Hazelton Group can be further subdivided into the Jack, Betty Creek, Spatsizi, Iskut River, Mt. Dilworth and Quock Formations (arranged from oldest to youngest). The stratigraphy in the immediate area of the property consists of an upright succession of andesite, marine sediments, intermediate to felsic volcaniclastic rocks, rhyolite, contact mudstone (host to the main Eskay Creek deposits), and basaltic/andesitic sills and flows. This sequence is overlain by mudstones and conglomerates of the Bowser Lake Group. These rocks are folded into a gently, northeast-plunging fold, the Eskay Anticline, and are cut by north-, northwest- and northeast-trending fault structures.

Regional metamorphic grade in the area is lower greenschist facies. Alteration in the footwall volcanic units is characterized by a combination of pervasive quartz–sericite–pyrite, potassium feldspar, chlorite and silica. Intense alteration zones are locally associated with sulphide veins that contain pyrite, sphalerite, galena, and chalcopyrite. An intense, tabular-shaped blanket of chlorite–sericite alteration, up to 20 m thick, occurs in the Eskay Rhyolite member, immediately below the contact with the main stratiform sulphide mineralization.

Several styles of stratiform and discordant mineralization are present at the Eskay Creek Project, defined over an area approximately 1,400 m long and as much as 300 m wide. Distinct zones have been defined by variations in location, mineralogy, texture, and precious metal grades.

Stratiform-style mineralization is hosted in black carbonaceous mudstone and sericitic tuffaceous mudstone of the Contact Mudstone (Iskut River Formation), located between the footwall Eskay Rhyolite member and the hanging wall Willow Ridge andesite unit. The stratiform hosted zones include the 21B Zone, the NEX Zone, the 21A Zone (characterized by arsenic–antimony–mercury sulphides), the 21C Zone, the 21Be Zone and the 21E Zone. Stratigraphically above the Contact Mudstone, and usually above the first basaltic sill, the mudstones also host a localized body of base metal-rich, relatively precious metal-poor, massive sulphides referred to as the Hanging Wall or HW Zone.

Stockwork and discordant style mineralization at Eskay Creek is hosted in the rhyolite footwall within the PMP Zone, the 109 Zone, the 21A Zone the 21B Zone, the 21C Zone, the 21E Zone, the NEX Zone and 22 Zone. The PMP Zone is characterized by pyrite, sphalerite, galena, and chalcopyrite-rich veins and veinlets hosted in strongly sericitized and chloritized rhyolite. The 109 Zone consists of gold-rich quartz veins with sphalerite, galena, pyrite, and chalcopyrite associated with abundant carbonaceous material hosted predominantly in siliceous rhyolite. The 21A, 21B, 21C, NEX and 21E Zones consist of very fine-grained cryptic pyrite with rare sphalerite and galena in sericitized rhyolite. The 22 Zone consists of cross-cutting arsenopyrite, stibnite and tetrahedrite veins hosted in massive to pyroclastic facies rhyolite.

There is significant remaining exploration potential in the Eskay Creek deposit and environs. Exploration targets include syn-volcanic feeder structures at depth and along strike; mineralization hosted within the largely unexplored Lower Mudstone horizon; and the in the vicinity of the 22 Zone, which remains open along strike and at depth. Due to limited legacy exploratory drilling in the area between the 21A and 22 Zones, additional opportunities exist to discover and delineate near-surface, rhyolite-hosted feeder mineralization.

1.6 History

The Project area has a long exploration history, dating back to initial prospecting activities in 1932. Companies with Project interests prior to Skeena's involvement include Premier Gold Mining Co. Ltd., MacKay Gold Mines Ltd., Canadian Exploration Ltd., American Standard Mines Ltd., Pioneer Gold Mines of B.C. Ltd., New York-Alaska Gold Dredging Corp., Western Resources Ltd., Stikine Silver Ltd., Canex Aerial Exploration Ltd., Mount Washington Copper Co., Newmont Mining Corp., Kalco Valley Mines Ltd., Texasgulf Canada Ltd., May-Ralph Resources Ltd., Ryan Exploration Ltd. (U.S. Borax), Kerrisdale Resources Ltd., Consolidated Stikine Silver Ltd., International Corona Corp., Homestake Canada Inc., and Barrick Gold Inc. Work conducted during this period included prospecting, geological mapping and reconnaissance, rock, stream, sediment, and soil geochemical sampling, trenching, surface geophysical surveys (electromagnetic (EM), very low frequency (VLF), ground magnetic/VLF-EM, induced polarization (IP), seismic refraction, University of Toronto electro-magnetic system (UTEM)), borehole geophysics (frequency domain EM (FEM)) core drilling, exploration adit and underground development, petrography, and mining studies. Underground mining operations were conducted from 1994 to 2008. From 1994–1997, ore was direct-shipped after blending and primary crushing. From 1997 to closure in 2008, ore was milled on site to produce a shipping concentrate.

Skeena has completed core drilling, an airborne light detection and ranging (LiDAR) and photo acquisition survey, Mineral Resource estimation, and preliminary technical studies since the option agreement was executed in late 2017.

1.7 Drilling and Sampling

Data collected prior to Skeena's project interest is referred to as legacy data. Legacy drilling consists of 1,522 surface diamond drill holes totalling 342,119 m and 6,061 underground drill holes totalling 309,213 m. In 2018, Skeena completed 46 drill holes from surface totalling 7,737.45 m. The 2019 surface drill program, as at 8 December 2019, included 209 drill holes totalling 14,267.27 m.

The underground areas are drilled at an average spacing of 10 m using BGM (~40 mm) core diameters. In highly complex areas where mining was active, drill spacing was locally reduced to 5 m. Underground drill holes are generally less than 100 m in length.

Limited information is available for procedures used during the exploration programs carried out before 2004. The drill core was logged using DLOG computer programs for data entry as well as for drill log printing. Information collected included lithology, mineralisation, textural descriptions, rock colour, structure, core recovery, and rock quality designation (RQD). Skeena currently does not have access to the legacy RQD and recovery data. Underground collar location surveys were performed by the mine surveyors. These provided accurate collar locations for the holes, and a check on the initial azimuth and dip was recorded for each drill hole. Prior to 2004, most of the underground drill holes in the database were surveyed downhole using a Sperry Sun Single Shot instrument, with readings taken every 60 m, or by acid tubes, with readings every 30 m. In early 2004, downhole surveying used an Icefield Tools M13 instrument. This provided azimuths and dips for each hole every 3 m down the hole. Readings were reviewed by staff and inaccurate entries were removed from the database. All collar and survey information were tabulated in master files within the DLOG computer program. Completed logs were printed and the information was exported into ACAD and Vulcan software to facilitate plotting drill hole location maps and cross-sections.

During the Skeena drill program, core was geologically logged for lithology, alteration, veining, mineralization and structural features. Geotechnical data such as recovery, RQD, longest stick, and magnetic susceptibility were recorded. Skeena recorded geological and geotechnical information into a GeoSpark database. Core was photographed wet. Surface drill hole collars were initially located using hand-held global positioning system (GPS) units and surveyed at the end of the drill program using a Trimble differential GPS (DGPS). Down hole orientation surveys for surface drill holes were taken approximately every 30 m down the hole using a multi-shot Reflex orientation tool.

Drill hole spacing throughout the deposit varies from 5 m, where underground production drilling encountered complex areas, to 25 m at the surface. The average drill hole spacing is approximately 10–15 m throughout the deposit. For surface drill holes, mineralisation true width approximates 80–100% of drilled width; for underground drill holes positioned on single platforms and drilled in radiating fans, true drilling widths are more variable.

Historically, sampling at Eskay Creek was selective and primarily based on visual estimations of sulphide percent. All sample intervals sent to the laboratory were tested for gold and silver; however, lead, copper, zinc, mercury, antimony and arsenic were inconsistently sampled from one drilling campaign to the next. For underground drilling, lead, copper, zinc, mercury, antimony and arsenic were assayed when samples exceeded 8 g/t gold equivalent (AuEq, where AuEq equaled Au + (Ag/68)). Legacy sampling intervals were variable. Prior to 2003, sample intervals varied from about 0.25 m up to 1.5 m though the optimum sample interval was 1.0 m. Sample intervals were always contained within one geological unit and did not straddle contacts. During 2004, sample intervals were typically on 1 m intervals, but smaller increments were applied where necessary to honour geological contacts.

During Skeena's drill programs, 1 m assay intervals were established when visible mineralization was first observed, and then uniform intervals were continued down the drill length until there is no evidence of mineralization. Assay intervals honoured geological contacts to a minimum of 0.5 m and a maximum of 1.5 m.

Specific gravity (SG) measurements collected during legacy programs were collected from diamond drill core in 1996 (250 measurements from 20 drill holes) and 1997 (84 measurements from seven drill holes), using the water displacement method. SG models were subsequently created using a formula that was experimentally derived based on comparisons between actual measurements and analyses. The following formula was used:

- SG = (Pb + Zn + Cu) x 0.03491 + 2.67

Where all metals are reported in percent.

A default SG value of 2.67 was applied to samples for which base metals were not reported. This is the average value of unmineralized rhyolite and mudstone host rocks combined. The measured SG values from the early drill programs were primarily from relatively low base metal, 21B-style mineralization. The formula is therefore likely biased on the low side for rocks with higher base metal content. During the Skeena programs, SG samples were collected one in every 20 m down the hole and measured using the water displacement method.

Laboratories used for sample preparation and analysis during legacy programs, where known, include: Independent Plasma Laboratories (IPL; independent, accreditations not known) and the Eskay Mine laboratory (not independent, not accredited). Skeena used the ALS sample preparation facility in Kamloops (ALS Kamloops). which is independent and accredited. Analysis was completed at the ALS facility in Vancouver (ALS Vancouver), which holds ISO17025 accreditation for selected analytical methods. Both laboratories are independent of Skeena. SGS Canada, located in Burnaby, BC, was used to independently test pulp duplicates and a select number of standards. SGS holds ISO 17025 accreditations for selected analytical techniques.

Legacy sample preparation and analytical methods included:

- IPL: crushed to -10 mesh, riffle split and 250 g pulverized to -15 mesh. Gold was assayed by fire assay (30 g) with an atomic absorption (AA) finish. All gold values >1.00 g/t were re-assayed by fire assay (30 g) and finished gravimetrically. Silver was assayed by fire assay (30 g) with an AA finish. Analysis for lead, zinc, copper, arsenic and antimony was done by an ore grade assay method using AA. Mercury analysis consisted of an aqua regia digestion and inductively-coupled plasma (ICP) finish;
- Eskay Mine laboratory: jaw-crushed to -⅛ inch, riffle split and pulverisation of 250–300 g. Gold was assayed by fire assay (10 g) with an AA finish. For analysis for zinc, antimony, copper, and lead, a 0.20g sample was digested in a heated solution of tartaric, nitric, perchloric and hydrochloric acids, and finished by AA. For mercury and arsenic, a 1.00 g sample was digested in a heated solution of nitric, perchloric and hydrochloric acids and finished by AA.

During the Skeena programs, all samples were initially sent and prepared at ALS Kamloops after which the pulp samples were split and shipped for analysis to ALS Vancouver. Sample preparation involved crushing to better than 70% passing 2 mm 10 mesh screen, and pulverizing to better than 85% passing a 75 µm 200 mesh screen.

Gold assays were performed on 50 g samples by fire assay and atomic absorption (ALS code: Au-AA26) with a lower and upper detection limit of 0.01 g/t and 100 g/t, respectively. For assays above the upper detection limit then samples were analysed by fire assay with a gravimetric finish (ALS code: Au-GRA22) with lower and upper detection limits of 0.05 g/t and 10,000 g/t Au, respectively. Silver assays were performed on 50 g samples by fire assay and gravimetric finish (ALS code: Ag-GRA22) with lower and upper detection limits of 5 g/t and 10,000 g/t, respectively. For assays above the upper detection limit, a concentrate and bullion grade fire assay and gravimetric finish were performed (ALS code: Ag-CON01) with lower and upper detection limits of 0.7 g/t Ag and 995,000 g/t Ag, respectively.

Multi-element assays were performed using a combination of digest and finish methods: a 0.25 g sample using a four-acid digest followed by an ICP atomic emission spectroscopy (AES) finish (ALS code: ME-ICP61), and a 0.1 g sample using lithium borate fusion followed by an ICP-MS finish (ALS code: ME-MS81). This combination in assay methods for the multi-elements ensured that the range of concentrations for all elements of interest, particularly for antimony, were covered. In the Skeena database, the ICP-AES finish method took precedence. A limited number of samples exceeded the upper limits for silver, arsenic, copper, lead and zinc. For these samples, the laboratory was instructed to apply overlimit methods on a 0.4 g sample (ALS code: OG62) using a four-acid digest and ICP or AAS finish. Sulphur overlimits were re-analyzed using the total sulphur Leco furnace method using a 0.1 g sample (ALS code: S-IR08) with a lower detection limit of 0.01% and upper detection limit of 50%. Mercury was separately analysed using low temperature aqua regia digestion followed by an ICP-AES finish (ALSO code: Hg-ICP42) with a lower detection limit of 1 ppm and an upper detection limit of 100,000 ppm.

Eskay Creek mine initiated QA/QC measures into their sample stream in 1997. With progressive years the QA/QC protocol became more comprehensive and detailed. Prior to 2002, there was no formal QA/QC program in place; however, the Eskay Creek mine laboratory and IPL were regularly monitored using pulp duplicates. In 2003, the Eskay mine laboratory started to implement QA/QC procedures into the sampling process. Control blanks and SRMs were added to the sample stream. Acme inserted their own in-house SRMs, blanks and pulp repeats into the sample stream. Acme also routinely used preparation, pulp and reject duplicates. An official QA/QC program was undertaken in 2004 whereby the Eskay Creek exploration team added SRMs, blanks and field duplicates to the sample stream and submitted them to Acme for checking. Sample repeatability at Eskay Creek was closely monitored during the 2004 drilling campaign by the regular insertion of field duplicates into the sample stream. Field duplicates at the Eskay mine laboratory performed well with the duplicate sample set. An audit was conducted on the 2004 QA/QC results and procedures by Dr. Barry Smee, of Smee & Associates Consulting Ltd. The findings from the analysis identified a low bias in relation to Acme's internal SRMs for both aqua regia and fire assay methods. Acme corrected the inconsistencies with batch repeats. The sampling precision by means of using duplicate preparation and pulp samples was found to be within acceptable limits.

Skeena implemented a formal QA/QC program from the inception of their 2018 Phase 1 drilling program, consisting of blanks, duplicates and SRMs. SRMs and blanks were monitored when batches of assay data were first received. If analyses were outside of the acceptable range after checking for data entry errors, then repeat assay were requested. Where two or more consecutive SRMs were both biased high or low (more than 105% of the expected value or less than 95% of the expected value) repeat assays were requested. The laboratory was instructed to retrieve five pulp samples before and after the QC failure. Duplicate data were also monitored, with Skeena reporting any concerns to the laboratory manager.

In early 2018, Skeena obtained access to the legacy database. The database files, assay certificates, drill hole logs, and report files were stored in various locations and in various states of order. No single complete data set was located. Between May and July 2018 Skeena personnel compiled and reviewed all available drilling and assay data to rebuild and produce a validated database in Microsoft Access format. The legacy database originated as a Vulcan file that was extracted and used as the building block for the final Skeena legacy database. Once the Skeena legacy database had been rebuilt, it was validated for gaps, overlapping intervals, duplicates, and lower detection limits. Surface drill hole collar locations were checked against the topographic surface for accuracy, and underground drill hole collar locations were checked against underground development wireframes. Where available, drill holes collar locations were confirmed from the original drill logs.

1.8 Data Verification

SRK conducted an independent review of the Skeena database, which consisted of review of the available legacy data in 2018, and review of the data from the Skeena 2018 Phase 1 drilling program in 2019. In addition, SRK reviewed the QA/QC programs. Aspects reviewed included:

- Verified assays in the Skeena legacy database against Eskay Mine laboratory and IML assay certificates, where assay certificates were available; however, the large number of missing assay certificates was a limitation on the validation effort;
- Checked for missing values, duplicate records, overlapping intervals, sample intervals exceeding maximum collar depths, borehole deviations, drill holes collars versus topography, laboratory certificate vs database values and special values (i.e. non-numeric or less than zero); any errors were reviewed with Skeena personnel;
- Viewed the collar locations of underground drill holes by means of 50 m sections with drill hole volume projections of 25 m; there was no obvious discrepancy between collar location and underground workings;
- Cross-checked the UTM and mine grid coordinates from 2004 with the Skeena legacy database. The checks confirmed that the imposed UTM-mine grid shift was acceptably accurate.

SRK inspected the 2018 Skeena data for collar survey discrepancies, erroneous downhole deviation paths, and overlapping or missing assay and lithology intervals. All errors found were corrected and the dataset used for resource estimation included the correct values.

SRK also reviewed all available legacy QA/QC data. SRK performed the following data validation steps on the legacy data:

- SRK independently compiled and merged all available laboratory assay certificates. The total number of certificates matched the compiled Skeena database to within 6%;
- Approximately 5% random samples were selected and checked against the original assay certificates. No apparent errors or omissions were discovered;
- SRK viewed the samples in 3D to identify for collar and survey discrepancies in relation to the available topographic surface; all errors were addressed and corrected;
- Mine grid coordinates and rotations were validated;
- Sections were viewed to check for discrepancies between underground collar locations and underground working solids;
- Lithology intervals were checked for overlapping intervals and were resolved when discovered.

SRK concluded that the results of the QA/QC analysis indicate that the historical data are unbiased. A large number of assays in the database were validated against the original digital assay certificates. These assays ranged from the years 1999 to 2004, and less than 1% errors were found.

In addition, the data analysed for the Skeena 2018 Phase 1 drilling program was collected and analysed in a systematic and unbiased manner. The data verification of this data did not identify any material issues and the QP is satisfied that the assay data is of suitable quality to be used as the basis for the resource estimate.

1.9 Metallurgical Testwork

1.9.1 Legacy Testwork

In 1991 and 1992, metallurgical testwork for the feasibility study had defined a complex hydrometallurgical flowsheet for the recovery of gold and silver, as well as copper and zinc. This process required a large capital outlay with high unit operating costs. The original operating plan was to construct the mining infrastructure at the mine site and transport ore to a processing facility located close to Placer Dome's Equity Silver mine, near Houston, B.C. In late 1994, mining operations commenced at Eskay Creek. In 1996, a testwork program was initiated at Process Research Associates with follow up locked-cycle testing at International Metallurgical and Environmental Inc. to evaluate the potential of a gravity/flotation process for upgrading ore from the NEX and 109 Zones into marketable concentrates. The work indicated that the mineralisation could be economically upgraded to a saleable concentrate. In 1997, Prime completed the engineering and construction of a 150 t/d mill to concentrate the gold and silver values for the NEX and 109 Zones. Over the next several years, the mill was steadily upgraded and expanded to its final production capacity of 350 t/d. Since 2008, the mine area has been under a state of reclamation, care and maintenance.

1.9.2 Current Testwork

As part of this PEA update, recent testwork has been completed by BlueCoast Research (BlueCoast) in Parksville BC, including comminution, whole ore leaching, gravity and flotation recovery methods. The process plant flowsheet assumed for this PEA includes only flotation recovery of a precious metal concentrate, for transport and shipment overseas. To further investigate to generate doré as a saleable product, a number of concentrate treatment alternatives are being evaluated. Concentrate treatment is an opportunity to transform the deleterious minerals into a safe form rather than incur higher treatment charges and penalties by including them in the concentrate.

Six metallurgical samples were collected including a "hot" sample that was elevated in silver, arsenic, antimony and mercury, significantly higher sulphur and sulphide content together with organic carbon (C_{org}). Zones 21A, 21C and 22 represent a significant portion of the life-of-mine (LOM) plant feed but Zone 21B was not sampled in the 2019 testwork program. Overall, the samples included a reasonable range in gold grade; however, they were lower in copper, lead, and zinc compared with the expected LOM average and future samples should be collected with higher base metal values. The samples selected for metallurgical testing were representative of various mineralisation forms present within the different zones. Samples were selected from a range of locations within the zones and sufficient mass and testing was performed to support this level of study.

Comminution or hardness testing on each sample consisted of semi-autogenous grind (SAG) mill comminution (DWi), Bond rod mill work index (RWi) and Bond ball mill work index (BWi) tests at a closing screen size of 150 µm. The test results indicated a range of material hardness.

Bottle roll cyanidation tests were performed to evaluate potential for whole ore leaching compared with the historical testwork on much higher-grade samples. Leaching under a range of 80% passing (P_{80}) grind sizes (80 µm, 50 µm and 30 µm) did not show any significant effect. Low gold extractions were attributed to a number of possible factors: fine-grained gold particles, the presence of preg-robbing sulphides and/or organic carbon and possible passivation of gold surfaces by antimony.

Based on an extended gravity recoverable gold (E-GRG) procedure with a three-pass grind and recovery sequence, gravity recovery was not recommended in the process flowsheet as part of this PEA.

A considerable number of open-circuit, rougher and rougher/cleaner float tests were conducted. A range of primary P_{80} grind sizes were tested (from 338 µm down to 39 µm) with ~60 µm used as the target P_{80} grind size for further float work. Rougher concentrate was also reground prior to cleaning, with a target P_{80} size of ~25 µm used as the base case. It was noted that the grind and regrind times were quite long (up to 40 minutes being required for the 25 µm regrind size); however, an investigation into possible overgrinding of phyllosilicate minerals did not reveal anything significant. Blue Coast noted that the flotation concentrate was very slow to pressure filter, and this remains a concern to be investigated and possibly addressed in solid/liquid separation testing in the future. The use of dispersants (sodium silicate and carboxymethyl cellulose, or CMC) was investigated as well as collector dosage. Samples exhibited relatively slow float kinetics with 80% Au recovery after 20 minutes of rougher flotation and 90% recovery after 40 minutes. An investigation into possible sliming did not reveal any explanation for the slow-floating nature of the samples.

Overall, the flotation testwork was able to produce a bulk concentrate with gold recoveries of 80–95% at grades of 40–50 g/t Au. Silver recoveries were in the range of 84–97% with grades from 1,000–1,300 g/t Ag. For the <3.5 g/t Au samples, the final concentrate contained around 1% As and Sb with ~200 ppm Hg. The LOM composite generated concentrates with much higher impurity levels due to the blend of Hot sample. As the expected mine plan calls for material at 4 g/t Au and below, the lower-grade sample results were used to generate the forecasted concentrate quality and quantity.

Automated mineralogical analysis was performed on both the final concentrate and tailings from the LOM sample float testing.

Based on the 2019 testwork results on samples with a range of head grades, a flotation concentrate of saleable precious metal content can be produced at high recoveries of both gold and silver. This concentrate will contain impurities of arsenic, antimony and mercury that will be subject to penalties. Depending on the concentrate customer, the antimony content may be included as a payable metal, provided the level is above a threshold value (e.g. 3% Sb). The open-circuit rougher and cleaner float test results were used to generate relationships between the gold and silver recovery versus head grade as well as the expected mass pull to concentrate. The concentrate impurity levels were well established from the testwork results. These relationships were done for 50 g/t, 40 g/t and 25 g/t Au concentrate to assist the marketing review completed as part of this PEA. The lower-grade concentrate required few stages of cleaner flotation. Across the proposed nine-year mine life, 60% of the plant feed anticipated to be rhyolite with 20% mudstone and 20% hanging wall andesite material. In Year 1, almost 60% of plant feed will be from the 21A Zone with higher precious metal grades and impurity levels. As the percentage of the 21A material decreases over time, the gold head grade will fall from almost 5 g/t Au to around 3 g/t Au. Similarly, silver grade will be higher in years 1–6 at 100 g/t Ag, and will fall to around half this value in Year 7.

While the generation of a precious metal concentrate was demonstrated for all metallurgical samples tested in 2019, supplementary testwork is ongoing into options for concentrate treatment. These treatments involve hydrometallurgical or pyrometallurgical oxidation of the sulphide content prior to cyanide leaching with/without carbon to minimise the impact of preg-robbing agents. For this PEA, concentrate treatment is considered an opportunity.

1.10 Mineral Resource Estimation

The Mineral Resource estimate is primarily based upon legacy diamond drilling completed by the previous operator; however, additional holes drilled by Skeena in 2018 have been included. The database used in estimation contains 7,583 legacy surface and underground diamond drill holes totalling 651,332 m, and 46 additional surface holes drilled in 2018 by Skeena (7,737 m).

A litho-structural model was constructed in Leapfrog Geo^{TM} software with three main lithologies (rhyolite, contact mudstone, and hanging wall andesite) and five faults recognized as meaningful for modelling purposes. Mineralization domains were subsequently defined using geologically realistic radial basis function (RBF) grade interpolants within major fault blocks. Mineralization domains were created using a 50% probability of a nominal gold equivalent cut-off grade >0.5 g/t AuEq. Ten mineralization domains were created to constrain the estimate: seven of which occurred exclusively in the open pit, and three domains that contained shared open pit and underground resource estimates. The 10 domains were interpolated separately based on presiding lithology types: either rhyolite, or mudstone and andesite combined.

Two block models were created:

- An open pit model using 9 x 9 x 9 m parent block sizes, with sub-block sizes of 3 x 3 x 2 m;
- An underground model using 3 x 3 x 2 m parent block sizes, with 1 x 1 x 1 m sub-block sizes.

One-metre composites were generated for the underground block model, and 2 m composites were created for the open pit block model. Grades within each domain were then capped within hard-domain boundaries. Gold capping values ranged from 45–900 g/t Au and silver capping values ranged from 600–30,000 g/t Ag.

Gold and silver variograms were used to determine the nugget, sills and ranges used during Kriging, however a dynamic surface, modelled along the Contact Mudstone basal contact, was used to incrementally modify the anisotropic search orientation during interpolation.

Ordinary kriging (OK) was used for the estimation of gold and silver in all domains, except for the low-grade shell which captured mineralization outside the mineralization domains. The Mineral Resources were estimated using two passes with increasing search radii based on variogram ranges. Indicated and Inferred resources were categorized during gold interpolation passes 1 and 2, respectively. The Indicated category (Pass 1) was defined by blocks interpolated using a minimum of three drill holes and a maximum distance of 43 m to a drill hole showing reasonable geological and grade continuity. In areas where blocks were interpolated during Pass 1, but continuity was insufficient or blocks were isolated, the blocks were reclassified to Inferred on a visual basis. In addition, all blocks located within a 3 m buffer around the underground workings were classified as Indicated. Inferred Mineral Resources (Pass 2) were interpolated using a minimum of two drill holes and a maximum distance to a drill hole composite of 95 m. SRK is of the opinion that the current Mineral Resource estimate is a reasonable representation of the global gold grade and tonnage at the current level of sampling and can be categorized as Indicated and Inferred based on quality data, data density and geological understanding.

Block tonnage was estimated from volumes using a density formula that was applied using interpolated lead, zinc, copper and antimony grades. This density formula was derived from the historical operator, based on comparisons between actual measurements and analysis at the Eskay Creek mine where:

- $SG = (Pb + Zn + Cu + Sb)\ x\ 0.03491 + 2.67$ (where all metals are reported in percent).

The 21A and 21B Domains have elevated levels of arsenic, mercury and antimony as compared to the rest of the mineralization domains at the Eskay Creek Project. The 21A Domain is geologically and geochemically equivalent to the 21B Domain which accounted for the bulk of mineralization historically mined at Eskay Creek. Blending of the 21B mineralized material with less deleterious material from other domains diluted these penalty elements thus reducing smelter penalties which allowed a profitable head grade to be maintained. A blending scenario similar to the one historically adopted is the expected approach for future mine and process planning.

SRK considers mineralization at the Eskay Creek Project to have reasonable prospects for economic extraction, in both open pit domains (22, 21A, 21C, 21B, 21Be, 21E, HW, NEX, PMP, and 109) and remaining underground domains (22, HW, and NEX). All Mineral Resources potentially amenable to underground mining methods occur immediately adjacent to, or within 100 m of existing underground infrastructure, of which all lifts and stopes have been duly backfilled. In addition to the required resource depletion applied to all historical workings, the mineralized material resources within 1 m of any historical working were excluded from the Mineral Resource estimate considered amenable to open pit mining methods. Similarly, any mineralization within 3 m of any historical working was excluded from the estimate of Mineral Resources potentially amenable to underground mining methods.

The cut-off grade for the open pit model was determined to be 0.56 g/t AuEq. The underground cut-off grade was determined to be 4.2 g/t AuEq. At Skeena's request, the cut-off grades applied for the resource statement were increased to 0.7 g/t AuEq for the open pit and 5.0 g/t AuEq for the underground resource.

1.11 Mineral Resource Statement

Mineral Resources are reported using the 2014 CIM Definition Standards in Table 1-1 and Table 1-2. Ms. S. Ulansky, Senior Resource Geologist, PGeo (EGBC#36085), an employee of SRK. (Canada) Inc.

Factors that may affect the estimate include: changes to long-term metal price assumptions; changes in local interpretations of mineralization geometry and continuity of mineralized zones; changes to the density values applied to the mineralized zones; changes to geological shape and continuity assumptions; potential for unrecognized bias in the assay results from legacy drilling where there was limited documentation of the QA/QC procedures; changes to the input values used to generate the AuEq cut-off grade; changes to metallurgical recovery assumptions; changes in assumptions of marketability of final product; changes to the conceptual input assumptions for assumed open pit operation; variations in geotechnical, hydrogeological and mining assumptions; changes to environmental, permitting and social license assumptions.

Table 1-1: Open Pit Mineral Resource Statement Reported at 0.7 g/t AuEq Cut-Off Grade

Classification	Domain	Tonnes (000)	Grade AuEq g/t	Au g/t	Ag g/t	Contained Ounces AuEq oz (000)	Au oz (000)	Ag oz (000)
Indicated	22	270	3.0	2.0	74	30	20	640
	21A	3,530	4.0	3.2	62	450	360	6,990
	21C	2,800	4.5	3.7	65	410	330	5,850
	21B	2,510	8.4	6.0	175	680	490	14,120
	21Be	860	9.7	6.5	241	270	180	6,660
	21E	200	4.1	2.6	112	30	20	720
	HW	880	6.0	3.8	170	170	110	4,820
	NEX	720	6.8	4.5	171	160	100	3,960
	PMP	180	5.9	4.5	106	30	30	620
	109	710	5.2	5.0	13	120	110	300
	Total	***12,650***	***5.8***	***4.3***	***110***	***2,340***	***1,740***	***44,660***
Inferred	ENV	3,110	2.2	1.4	57	220	140	5,740
	22	1,350	2.1	1.9	15	90	80	660
	21A	1,330	5.7	5.0	51	240	210	2,190
	21C	2,080	2.6	2.2	32	180	150	2,160
	21B	3,220	2.5	2.0	32	250	210	3,290
	21Be	720	4.0	2.9	85	90	70	1,960
	21E	900	2.9	2.0	61	80	60	1,750
	HW	740	3.8	2.4	105	90	60	2,500
	NEX	800	2.8	2.2	48	70	60	1,240
	PMP	100	4.9	3.9	70	20	10	220
	109	80	2.7	2.6	10	10	10	20
	Total	***14,420***	***2.9***	***2.3***	***47***	***1,340***	***1,050***	***21,720***

Table 1-2: Underground* Mineral Resource Statement Reported at a 5.0 g/t AuEq Cut-Off Grade

Classification	Tonnes (000)	Grade AuEQ g/t	Au g/t	Ag g/t	Contained Ounces AuEQ oz (000)	Au oz (000)	Ag oz (000)
Indicated	819	8.2	6.4	139	218	169	3,657
Inferred	295	8.2	7.1	82	78	68	778

Notes to accompany the Mineral Resource estimate:

1. These Mineral Resources are not Mineral Reserves as they do not have demonstrated economic viability. Results are reported in-situ and undiluted and are considered to have reasonable prospects for economic extraction.

2. As defined by NI 43-101, the Independent and Qualified Person is Ms. S Ulansky, PGeo of SRK Consulting (Canada) Inc., who has reviewed and validated the Mineral Resource Estimate.

3. The open pit block model was regularized to 9 m x 9 m x 4 m whole blocks using mineralization > 0.5 g/t gold equivalent (AuEq) within a single mineralisation percent field. AuEq is calculated using the formula AuEQ = Au (g/t) + (Ag (g/t)/75).

4. The effective date of the Mineral Resource estimate is February 28, 2019.

5. The number of metric tonnes and ounces were rounded to the nearest thousand. Any discrepancies in the totals are due to rounding.

6. Pit constrained Mineral Resources are reported in relation to a conceptual pit shell.

7. Block tonnage was estimated from volumes using a density formula that applied using interpolated Pb, Zn, Cu, and Sb whereby SG = (Pb + Zn + Cu + Sb) * 0.03491 + 2.67 (where all metals are reported in %).

8. All composites have been capped where appropriate.

9. Mineral Resources potentially amenable to open pit mining methods are reported at a cut-off grade of 0.7 g/t AuEq and Mineral Resources potentially amenable to underground mining methods are reported at a cut-off grade of 5.0 g/t AuEq.

10. Cut-off grades are based on a price of US$1,275 per ounce of gold, US$17 per ounce silver, and gold recoveries of 80%, silver recoveries of 90% and without considering revenues from other metals.

11. Estimates use metric units (metres, tonnes and g/t). Metals are reported in troy ounces (metric tonne * grade / 31.10348).

12. 2014 CIM definitions were followed for the classification of mineral resources.

13. Neither Skeena nor SRK is aware of any known environmental, permitted, legal, title-related, taxation, socio-political, marketing or other relevant issue that could materially affect this Mineral Resource estimate.

1.12 Mining Methods

The mine plan is partly based on Inferred Mineral Resources that are considered too speculative geologically to have the economic considerations applied to them that would enable them to be categorized as Mineral Reserves, and there is no certainty that the PEA based on these Mineral Resources will be realized.

1.12.1 Geotechnical Considerations

Pit slope angle assessments were primarily based on resource drilling data and core photographs, simple RQD data, economic pit shells, geologic models, and relevant background reports. No material geotechnical drilling, logging, mapping, sampling, or laboratory testing was completed for the PEA. Overall, the data indicate generally ‘fair’ to ’good’ rock mass conditions throughout the planned mining zone.

The pit slopes are expected to consist primarily of hanging wall andesite along the upper pit walls with rhyolite being more prevalent at lower pit elevations. The contact mudstone is expected to only affect narrow zones between the hanging wall andesite and rhyolite. The parameters developed for the north pit were also applied to the south pit due to limited information available and the small size of the south pit.

To allow steeper slope angles in areas with better quality rock and to minimize stripping to the greatest extent possible, AGP divided the pit into individual slope design sectors, based on slope height and dominant geology. Estimates of suitable overall slope angles were then developed for each of the individual sectors. The inter-ramp slope recommendations ranged between 32º and 42º.

1.12.2 Hydrological Considerations

The regional groundwater regime is most likely controlled by the regional groundwater flow system, and from seasonal snow melt. The regional faults likely provide high permeability recharge pathways and groundwater storage areas; however, the rock units themselves are highly fractured and even away from major faults constitute fractured aquifers. Faulted andesite most likely provides the highest permeability and highest storage capacity of all the rock units. Historically, three high-permeability zones with large areal extents, and six hydrostratigraphic units were identified.

The planned ultimate pit bottom will be at 714 masl, and therefore only about 50 m of flooded working is likely to require dewatering. The andesite and mudstone lithologies will likely dewater easily compared to the rhyolite, which reportedly has high fines content and drains poorly (significantly lower hydraulic conductivity than the andesite). The rhyolite will generally occupy lower elevations in the final pit extent; however, rhyolite would be present on the south and east pit highwall and may be susceptible to failure if pore-water pressure builds up on fault planes. Horizontal boreholes drilled from pit benches may be a more efficient and effective means of depressurizing this material than vertical dewatering wells.

Groundwater interaction with surface water may be exacerbated by dewatering of the underground workings; however, historic mine inflow records do not suggest a significant flow path for creek water to enter the mine. Pit stability can be managed by progressive dewatering of the ground behind the pit slope with vertical or horizontal boreholes. The mudstones may require special attention as matrix pore pressures could remain elevated despite successful dewatering.

1.12.3 Mine Plan

The mine plan is partly based on Inferred Mineral Resources that are considered too speculative geologically to have the economic considerations applied to them that would enable them to be categorized as Mineral Reserves, and there is no certainty that the PEA based on these Mineral Resources will be realized.

The PEA is based on open pit only mining of the Eskay Creek deposit. AGP's opinion is that with current metal pricing levels and knowledge of the mineralization and previous mining activities, open pit mining offers the most reasonable approach for development.

The Eskay Creek PEA has two pit designs: the north pit and the south pit. The north pit will have four phases with Phase 3 split into three parts for access. The south pit will be a small single-phase pit that will be mined at the end of the mine life. These pits will provide a total of 21.3 Mt of mill feed grading 3.23 g/t Au and 78 g/t Ag. Waste movement from these phases will amount to 154.0 Mt giving a strip ratio of 7.2:1 (waste:mill feed).

The mill feed cut-off is based on a value per tonne which is often referred to as the milling cut-off. This was determined to be 1 g/t Au, and considers all the penalties, transportation costs and smelting charges for the bulk concentrate.

The feed to the plant was diluted. The calculation is based on a 1.25 m dilution skin on contacting blocks. This higher level of dilution skin was assumed considering the requirement to work around old underground mine workings which could result in mixing of waste and feed material. The result of the dilution calculation was a 20.8% increase in feed tonnage and a 16.6% lower feed grade. A mining recovery of 98% was also applied.

The phases are scheduled to provide 2.5 Mt/a of feed to the mill over a nine-year operating mine life, after two years of pre-production stripping. The pits are sequenced to minimize initial stripping and provide higher feed grades in the early years of the mine life. This is accomplished with stockpiling of lower-grade material.

The pits will be built on 8 m benches with safety berm placement each 16 m. Minimum mining widths of 35–40 m were maintained in the design. Ramps will be at 10% gradient and will vary in width from 23.3 m (single lane width) to 30.2 m (double lane width). They have been designed for 142 t haulage trucks.

The mine equipment fleet is anticipated to be leased to lower capital requirements. The fleet will be consist of six 140 mm rotary drills, two 22 m^3 hydraulic shovels and one 13 m^3 front-end loader. The truck fleet will peak at nine trucks in Year 4. This is due to the long hauls anticipated from the pit bottom to the higher waste rock storage facility (WRSF) elevations. Dozers, graders, small backhoes and other support equipment are considered in the equipment costing. Additional support equipment in the form of snowplows and small excavators will be part of the fleet to maintain operations year-round with the expected annual snowfall. An additional front-end loader (13 m^3) will be at the primary crusher full time and tramming material from the stockpile as required. The pit front end loader will be the backup for crusher loader.

The WRSF will fill the valley from the primary crusher towards the plant on the western side of the pits. The WRSF will have a top elevation of 1122 masl and the toe will be near the primary crusher at 902 masl for a total height of 220 m. A total volume of 70.4 Mm^3 has been designed, which is sufficient for the mine needs with a total of 6.8 Mm^3 of in-pit backfill.

Material from the mine has been assumed to be potentially acid-generating (PAG). All drainage from the WRSF will be collected in ditches, pumped to the settling ponds and treated as required. Additional work on the exact nature of the material from a PAG perspective should be defined during more detailed studies.

1.13 Recovery Methods

The process plant design is based on a robust metallurgical flowsheet developed for optimum recovery while minimizing capital expenditure and life-of-mine operating costs. The plant is anticipated to process material at a rate of 2.5 Mt/a with an average head grade of 3.2 g/t Au and 78 g/t Ag to produce a flotation concentrate. Design criteria for the flotation plant were determined from metallurgical test work. The majority of the flotation testwork was conducted at P80 of 60 µm. Grind sensitivity tests conducted at 39, 57 and 83 µm demonstrated low impact on recovery; therefore, given the low grind sensitivity, 75 µm was selected for design purposes. An overall flotation residence time of 40 minutes was selected without a scale-up factor to remain consistent with testwork results.

The processing plant will consist of the following areas:

- Primary crushing: a vibrating grizzly feeder and jaw crusher;
- Crushed material storage and reclaim: stockpile with two reclaimers;
- Grinding: semi-autogenous grind (SAG)/ball mill circuit;

- Rougher flotation: rougher flotation cells;
- Regrind and cleaner flotation: fine grinding and final cleaner flotation cells;
- Concentrate dewatering and filtration: concentrate thickener and filtration;
- Concentrate load-out: storage shed to allow front-end loader filling of concentrate transportation;
- Final tailings disposal: tailings slurry pumped to TMSF.

1.14 Project Infrastructure

Infrastructure to support the Eskay Creek project will consist of site civil work, site facilities/building, a water system, and site electrical. Site facilities will include both mine and process facilities:

- Mine: administration offices, truckshop and warehouse, tire repair shop, mine workshop, mine dry, fuel storage and distribution, permanent camp facility and miscellaneous facilities;
- Process: process plant, crusher facility, process plant workshop and assay laboratory;
- Services: potable water, fire water, compressed air, power, diesel, communication, and sanitary systems.

The WRSF assumption is that there will be a major facility, WD-01, developed on the west side of the open pit. The remainder of the waste will be placed into the mined-out north pit as backfill.

The existing TMSF was selected as the preferred tailings storage option since it is permitted as a tailings storage facility (TSF) and has sufficient capacity to contain 19.5 Mt of tailings. The TMSF only requires a small embankment to contain the required volume of tailings with the majority of the tailings located below the existing outlet. The TMSF is approximately 3.4 km long and 0.3 km wide. The facility ranges in depth from 10 m at the south end to 42 m in the north–central section of the lake. The existing volume of the TMSF is around 12.9 Mm^3 at elevation 1082 masl, which is the current spill elevation of the basin. Tailings would be slurried from the process plant to the TMSF by way of a pipeline, which would extend onto the TMSF to a floating barge. The end of the pipeline would be positioned close to the base of the TMSF to maximise settling, and minimize entrainment of fine particles to the surface of the TMSF. The minimum water depth would be 7 m to prevent both wind and ice remobilization of the tailings. The barge would move around the TMSF to develop an even tailings distribution across the TMSF floor. Tailings are planned to be discharged at 35% solids and will have an overall dry bulk density of 1.4 t/m^3. The TMSF has sufficient capacity to store tailings without an embankment during the initial years of operations while maintaining 7 m (6–8 Mm^3) of water cover over the tailings bed. In year 4 of operations, a single embankment will be required to be constructed, so as to store the balance of the LOM tailings while maintaining 7 m of water cover. The TMSF will also provide the water for the process plant.

A projected site-wide water balance was constructed for PEA purposes. Tailings slurry and treated wastewater will be discharged subaqueously into the TMSF. No diversion works are anticipated. There will be inflow of water into the TMSF from direct rainfall and snow and runoff from the surrounding catchment into the TMSF. Pit dewater will be sent directly to the water treatment plant (WTP), then to D7 polishing ponds, and finally to Ketchum Creek. Since the water treatment plant's maximum capacity will be approximately 150 L/s, the overflow, i.e. portion of the flow greater than 150 L/s will be sent to the TMSF. Estimated pit dewatering flow rates are <150 L/s during the initial years of operations, therefore no water will be sent to the TMSF. As the open pit becomes larger toward the end of the project, pit dewatering flow rates are estimated to surpass 150 L/s between late spring and fall. During this period, the overflow portion sent to the TMSF will range from 4.5–286.4 L/s. The overflow will be pumped to the tailings mixing tank and sent with the tailings in the tailings transportation pipeline to the TMSF. The industrial water requirements will come from the TMSF, which are estimated to be 113 L/s to be used in mineral processing. The balance of the waste (tailings) and process water will be pumped to the TMSF and discharged subaqueously. The approximate discharge of water, along with the tailings, is projected to be 114 L/s. The project does not need to take into account evaporation or seepage from the TMSF, since it is a natural, water-retaining catchment. The tailings deposition does not significantly affect the net evaporation or seepage losses from the TSF. For the planned operations there is almost no net loss of water from mineral processing, i.e. <1 L/s.

The permanent camp will be housed in portable modular units comprising of 200 jack-and-jill-type dormitories. The planned camp will be supplied for all its water needs from a local well. It is estimated that the average consumption of water, based on the size of the camp, is 1 L/s. Any effluent coming from the camp will be treated and discharged into the TSF.

The project power will come from the local and recently commissioned 195 MW hydroelectric facilities and leverage on the existing power grid. The assumed required supply is 18.8 MW. A new 14 km power transmission line will tie-in to the existing transmission line and feed a high voltage substation at the project site. The tie-in point will be close to the hydroelectric facilities.

1.15 Environmental, Permitting and Social Considerations

1.15.1 Environmental Considerations

Several environmental studies were completed at the Eskay Creek mine under various owners. A limited number of reports were available for review, the key reports reviewed are discussed in this sub-section. The environmental baseline data were mostly collected between 1990 and 1993 by Hallam Knight and Piésold for Prime Resources Ltd to support their application for a Mine Development Certificate. Updates were made in 1997 to support a proposed mill expansion, and again in 2000 to amend the environmental assessment (EA) to deposit tailings and waste rock in the TMSF.

Due to the age of the baseline assessment, additional environmental, social, economic, heritage, and health studies are expected to update the baseline data set to meet current standards for environmental studies, to address refinement of the project design, and reflect current regulatory requirements in support of provincial and federal EA submissions.

The project will be designed, constructed, operated, and decommissioned to meet all applicable BC environmental and safety standards and practices. Skeena will develop and implement an Environmental Management System (EMS) that defines the processes by which compliance will be met and demonstrated. The EMS will include ongoing monitoring and reporting to relevant parties at the various project stages.

The main waste management issue for the project is the prevention and control of metal leaching/acid rock drainage (ML/ARD) from the tailings, and any acid generating or PAG waste rock that is produced during mine development or operations. Non acid-generating (NAG) waste rock will be deposited in two locations: approximately 90% will be stored in the WD-01 facility that will be located to the south of the open pit. The remaining 10% of the total waste rock will be backfilled in the north pit. PAG waste rock, if encountered, will be deposited in the WRSF and effluent managed to reduce any environmental impacts. Tailings will be deposited sub-aqueously in the permitted TMSF.

Site water management will be a critical component of project design. Mine water can be divided into two categories depending on the potential for contamination:

- Non-contact water from upstream catchments that has not been in contact with mine workings will be kept separate from water that has been in contact with mine workings and discharged to the environment with no treatment;
- Contact water that has been in contact with potential sources of contamination, includes seepage from the WRSF, process water, and pit dewatering. Contact water from the WRSF will be collected and sent to a water treatment plant for treatment prior to discharge if required. If contact water quality from the WRSF is within permitted parameter limits, and is confirmed with regular testing, this water will be discharged without treatment. Water from pit dewatering will be pumped to a water treatment plant for treatment prior to discharge to the existing mine water polishing ponds and ultimate discharge through permitted effluent discharge point D7 (identification number E219595) to Ketchum Creek. Process water will be discharged to the TMSF.

Strategies for water management include collecting surface water from disturbed areas (mine-contact) to manage surface water erosion; recycle mine-contact water whenever possible; treat mine-contact water as required, and monitor water quality to meet discharge standards prior to discharge.

1.15.2 Closure and Reclamation Planning

A Closure and Reclamation Plan will be developed as part of the EA and refined for the permitting process. In summary, the mine closure concept is to meet water quality objectives without ongoing treatment for ARD. Closure planning will include dialogue with First Nations and stakeholders to determine post mining land use objectives and necessary investigations required to achieve and monitor those objectives.

1.15.3 Permitting Considerations

Major mining projects in BC are subject to EA and review prior to certification and issuance of permits to authorize construction and operations. The project will require a provincial EA Certificate (EAC) before the issuance of any permits to construct or operate. It will also require a federal decision statement before the issuance of any permits to construct or operate.

Skeena has not filed a federal or provincial EA application. Once an application is filed, the BC Environmental Assessment Office (EAO) and IAAC will issue their decision for the project. Once the project has a provincial EAC and a federal decision statement, Skeena can apply for the necessary statutory permits and authorizations to commence project construction. No technical or policy issues are anticipated for obtaining the required project permits and approvals, given its long mining history.

Skeena will apply for synchronous permitting within the environmental review process for all permits. Synchronous permitting will expedite the permitting process and reduce the time to start construction. Skeena has prepared a preliminary list of the key provincial and federal authorizations, licences, and permits that will be required to develop the project.

1.15.4 Social Considerations

Northwestern BC is a sparsely populated and relatively undeveloped region of the province. Many of the smaller communities have predominantly Indigenous populations that are isolated from one another as well as from the main regional centres of Smithers and Terrace. Mining and forestry are the main sources of income. Community and socio-economic impacts of the project can potentially be very favourable for the region, as new long-term opportunities are created for local and regional workers.

Both the BC *Environmental Assessment Act* (BCEAA) and the federal *Impact Assessment Act* (IAA) 2019 contain provisions for consultation with First Nations, and the public as a component of the EA process. Future engagement and consultation measures will comply with federal and provincial regulations, best practices, and Skeena's internal company policies.

The Tahltan Nation has asserted Indigenous title and rights to this area in the Declaration of the Tahltan Tribe in 1910 (https://tahltan.org/central-government/). Previous operators have established formal agreements with the Tahltan Central Government regarding their ongoing participation at the mine site. More recently, Skii km Lax Ha Nation has produced maps indicating that the mine falls within their area of traditional land use (Rescan, 2009).

Skeena will engage and collaborate with federal, provincial, regional, and municipal government agencies and representatives as required with respect to topics such as land and resource management, protected areas, official community plans, environmental and social baseline studies, and effects assessments. Skeena will consult with the public and relevant stakeholder groups, including land tenure holders, businesses, economic development organizations, businesses and contractors (e.g., suppliers and service providers), and special interest groups (e.g. environmental, labour, social, health, and recreation groups), as appropriate.

1.16 Markets and Contracts

The concentrate as proposed is a complex gold concentrate with relatively low gold content and elevated levels of arsenic, mercury and antimony. Deleterious element assays are notably elevated in the first few years of mine life (arsenic in Years 1 and 2 and mercury in Years 1 to 3) before dropping to values which fall within typical industry expectations. Given the complexity of the Eskay Creek concentrate, in combination with the historical production of relatively difficult to market concentrates from the mine during its previous operational period, two independent, preliminary market studies were completed to support the payabilities used in the PEA.

The relatively high levels of deleterious elements, particularly mercury in the initial years of operation, may require that concentrate sales be spread across a number of buyers as individual smelters are likely to need to blend small volumes of concentrate with cleaner concentrates to remain within acceptable effluent limits. An alternative option could be to sell the concentrate to traders who may be able to buy it all and spread distribution across a range of end customers. Expectations of payability vary but if the concentrate can be spread across enough buyers then favourable payabilities may be achieved and penalties for deleterious elements may be minimised.

The PEA assumes that concentrates will be sent to an Asian port for smelting and refining. Smelter term and transport cost assumptions are included in the PEA. The Chinese market offers the best payable terms and does not penalize mercury at the expected amounts in the Eskay Creek concentrate. The Chinese market is more than capable of absorbing the volumes under consideration and, Chinese smelters are expected to actively be looking for feedstock in future to keep their utilisation high. Other smelters around the world, such as the Horne smelter in Canada, may also be interested in purchasing some of the concentrate, although the mercury levels could be a challenge.

No contracts have been entered into at the Report effective date for mining, concentrating, smelting, refining, transportation, handling, sales and hedging, and forward sales contracts or arrangements. It is expected that the sale of concentrate will include a mixture of long-term and spot contracts.

Ausenco and Skeena established metal price projections for use in the PEA, based on recent metal market information, in combination with two year trailing actual metal prices and bank analyst forward price projections. The PEA assumptions are:

- Gold: US$1,325/oz Au;
- Silver: US$16.00/oz Ag.

An exchange rate of 0.77:1 US$:C$ was used.

1.17 Capital Cost Estimates

1.17.1 Summary

The capital cost estimate is presented in Table 1-3 at a ±50% accuracy, using a base date of Q3, 2019, and an exchange rate assumption of US$0.77:C$1.00.

1.17.2 Mining Capital Cost Estimate

The mining capital cost estimate is grouped into three main categories: pre-production stripping costs; mining equipment capital; and miscellaneous mine capital. Pre-production stripping costs cover all associated management, dewatering, drilling, blasting, loading, hauling, support, engineering and geology departments labour, grade control costs and financing costs. The mining equipment capital costs reflect the use of financing of the major equipment and some support equipment. Equipment prices used current quotations from local vendors. A 20% down payment is included in the capital cost for those units financed. The remaining cost was included in operating costs. The miscellaneous mine capital includes various separate line items in the costing, such as office, despatch, communication, and dewatering equipment and software, road development, and clearing/grubbing.

1.17.3 Process Capital Cost Estimate

Process equipment costs were derived using recent similar projects, recent and historical budget quotes on file from vendors. Delivery and installation of process equipment is a factored cost relative to the total purchase price of equipment.

Bulk earthworks for the plant site, camp and mine ancillary buildings were developed using semi-detailed cut and fill volumes based on process plant layout and site topographical information. The indirect cost estimate was factored based on previous Ausenco experience with similar-sized projects. EPCM was estimated at 13% of total direct costs, and field indirect costs at 6% of total direct costs. Owner's costs were estimated at 4% of total direct and indirect costs.

1.18 Operating Cost Estimates

1.18.1 Summary

The operating cost estimate is presented in Table 1-4 at a ±50% accuracy, using a base date of Q3, 2019, and an exchange rate assumption of US$0.77:C$1.00.

Table 1-3: Capital Cost Estimate Summary (C$)

	Initial ($ M)	Sustaining ($ M)	LOM Total ($ M)
Mine			
Pre-stripping	62	—	62
Mining equipment	14	6	20
Mine capital	7	3	9
Sub-total mine	*83*	*9*	*91*
Processing			
Bulk earthworks	7	—	7
Processing	74	7	81
Reagents & plant services	7	1	8
Tailings & water treatment	19	2	21
Onsite infrastructure	22	2	23
Sub-total processing	129	12	141
Infrastructure			
Power	13	—	13
TSF, water supply & treatment	2	4	6
Sub-total infrastructure	*15*	*4*	*19*
Total directs	***226***	***24***	***250***
Indirects	27		27
Total directs + indirects	***253***	***24***	***277***
Owner's costs	10		10
Total excluding contingency	***263***	***24***	***287***
Project contingency	40	3	43
Sub-total	*303*	*27*	*330*
Closure	–	52	52
Total	***303***	***79***	***382***

Table 1-4: Operating Cost Estimate Summary (C$)

Operating Cost	Annual Cost ($M)	Annual Cost ($/t Processed)
Mining	65.8	26.32
Processing	43.3	17.31
Contingency on process	6.5	2.60
Water treatment	4.4	1.74
Site G&A	15.15	6.06
Total	***135.1***	***54.02***

1.18.2 Mining Operating Cost Estimate

Costs were estimated from base principals with vendor quotations for repair and maintenance costs and other suppliers for consumables. Key inputs to the mine cost are fuel and labour. The price provided for the project was $1.04/L delivered to the site. The mine fleet will be entirely diesel powered. The dewatering pumps will be electric powered and a price of $0.06 per kilowatt hour was used.

The life of mine operating cost was determined to be $3.44 /t mined or $26.32 /t mill feed.

Labour costs for the various job classifications were obtained from salary surveys in British Columbia and other operations. A burden rate between 39% and 44% was applied to the various rates. Labour was estimated for both staff and hourly on a 12-hour shift basis using a rotation of either two weeks on/two weeks off or 4x3.

All of the major mine equipment, and the majority of the support equipment, where it was considered reasonable, was assumed to be leased. The operating cost would vary annually depending on the equipment replacement schedule and timing of the leases. Using the leasing option adds $0.32/t to the mine operating cost over the life of the mine. On a cost per tonne of feed basis, leasing was $2.47/t mill feed.

Vendors provided repair and maintenance (R&M) costs for each piece of equipment selected for the Eskay Creek PEA. Fuel consumption rates were estimated from the supplied information and knowledge of the working conditions. Drilling in the open pit will use down the hole hammers drill rigs with 140 mm bits. The pattern size varies between mill feed and waste and is blasted in recognition of the equipment being used. An emulsion product will be used for blasting to provide water protection. The blasting cost is estimated using quotations from a local explosives vendor. Loading costs for both mill feed and waste are based on the use of hydraulic shovels and front-end loaders. Haulage profiles were determined for each pit phase for the primary crusher or the waste rock facility destinations. Cycle times were generated for the appropriate period tonnage by destination and phase to estimate the haulage costs. Support equipment hours and costs were determined on factors applied to various major pieces of equipment. Grade control will be completed with a separate fleet of RC drill rigs. Over the life of the mine, a total of 169,000 m of drilling are expected to be completed for grade control work.

The dewatering is planned to be completed with a set of four pumps in the pit and two pumps on the surface to push the water to the settling ponds. Additional dewatering in the form of horizontal drill holes is included as part of the dewatering costs. These holes will be campaigned and will be part of the sustaining mine capital.

1.18.3 Process and General & Administration Cost Estimate

Power costs were calculated from an estimate of annual power consumption and using a unit cost of $0.06/kWh. Power consumption was derived from calculated power draw of the ball and SAG mills, plus an allowance for the remainder of the plant, based on typical flotation plants. The average on-line power draw is estimated at 19 MW. Annual energy consumption is estimated at 127,564 MWh, or about $7.65 M.

Processing reagent and consumable costs were estimated based on the throughput. Annual maintenance spares and consumable costs were estimated at 3% of total installed costs for mechanical equipment, plate work, support steel and electrics. Labour costs include all processing and maintenance costs. Costs were estimated from a breakdown of staffing positions, excluding G&A manpower. An allowance of 15% of all other operating costs was made, to include fuel costs, laboratory chemicals and similar sundry items. The G&A operating costs were estimated based on benchmarked data from similar projects in BC Canada. Costs include camp operations, G&A personnel, off-site offices, contracts, and vehicle maintenance, as well as miscellaneous project costs.

1.19 Economic Analysis

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 estimates;
- Assumed commodity prices and exchange rates;
- The proposed mine production plan;
- Projected mining and process recovery rates;
- Assumptions as to mining dilution and ability to mine in areas previously exploited using underground mining methods as envisaged;
- Sustaining costs and proposed operating costs;
- Interpretations and assumptions as to joint venture and agreement terms;
- 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 or hydrogeological considerations during mining being different from what was assumed;
- Failure of mining methods to operate as anticipated;
- Failure of plant, equipment or processes to operate as anticipated;
- Changes to assumptions as to the availability of electrical power, and the power rates used in the operating cost estimates and financial analysis;
- Ability to maintain the social licence to operate;
- Accidents, labour disputes and other risks of the mining industry;
- Changes to interest rates;
- Changes to tax rates.

The mine plan is partly based on Inferred Mineral Resources that are considered too speculative geologically to have the economic considerations applied to them that would enable them to be categorized as Mineral Reserves, and there is no certainty that the PEA based on these Mineral Resources will be realized. Mineral Resources that are not Mineral Reserves do not have demonstrated economic viability.

Calendar years used in the financial analysis are provided for conceptual purposes only. Permits still have to be obtained in support of operations, and approval for development to be provided by Skeena's Board.

An engineering economic model was developed to estimate annual pre-tax and post-tax cash flows and sensitivities of the Project based on a 5% discount rate. It must be noted, however, that tax estimates involve many complex variables that can only be accurately calculated during operations and, as such, the after-tax results are only approximations. Sensitivity analysis was performed to assess impact of variations in metal prices, head grades, operating costs and capital costs. The economic analysis has been run with no inflation (constant dollar basis).

The economic analysis was performed using the following assumptions:

- Commercial production start-up in 2023;
- Construction period of two years;
- Mine life of 8.6 years;
- Base case gold price of US$1,325/oz and silver price of US$16/oz was based on consensus analyst estimates and recently published economic studies. The forecasts used are meant to reflect the average metal price expectation over the life of the Project. No price inflation or escalation factors were taken into account. Commodity prices can be volatile, and there is the potential for deviation from the forecast;
- United States to Canadian dollar exchange rate assumption of 0.77 (US$/C$)
- Cost estimates in constant Q3 2019 C$ with no inflation or escalation factors considered;
- Results are based on 100% ownership with 1% NSR;

- Capital costs funded with 100% equity (i.e. no financing costs assumed);
- All cash flows discounted to December 31, 2019;
- All metal products are assumed sold in the same year they are produced;
- Project revenue is derived from the sale of gold concentrate into the international marketplace;
- No contractual arrangements for smelting or refining currently exist.

At the effective date of the cashflow, the Project was assumed to be subject to the following tax regime:

- The Canadian Corporate Income Tax system consists of the federal income tax (15%) and the provincial income tax (12%);
- The BC Minerals Tax was modelled using a net current proceeds rate of 2% and a net revenue tax rate of 13%.

Total tax payments are estimated to be C$514 M over the LOM.

The economic analysis was performed assuming a 5% discount rate. The pre-tax net present value discounted at 5% (NPV5%) is C$993 M, the internal rate of return IRR is 63.3%, and payback is 1.1 years. On an after-tax basis, the NPV5% is C$638 M, the IRR is 50.5%, and the payback period is 1.2 years.

A summary of the Project economics is included in Table 1-5 and shown graphically in Figure 1-1.

1.20 Sensitivity Analysis

A sensitivity analysis was conducted on the base case pre-tax and after-tax NPV and IRR of the Project, using the following variables: metal price, discount rate, grade, capital costs, and operating costs. Results are presented in Table 1-6. Analysis revealed that the Project is most sensitive to changes in metal prices and head grade and then, to a lesser extent, to operating costs and capital costs.

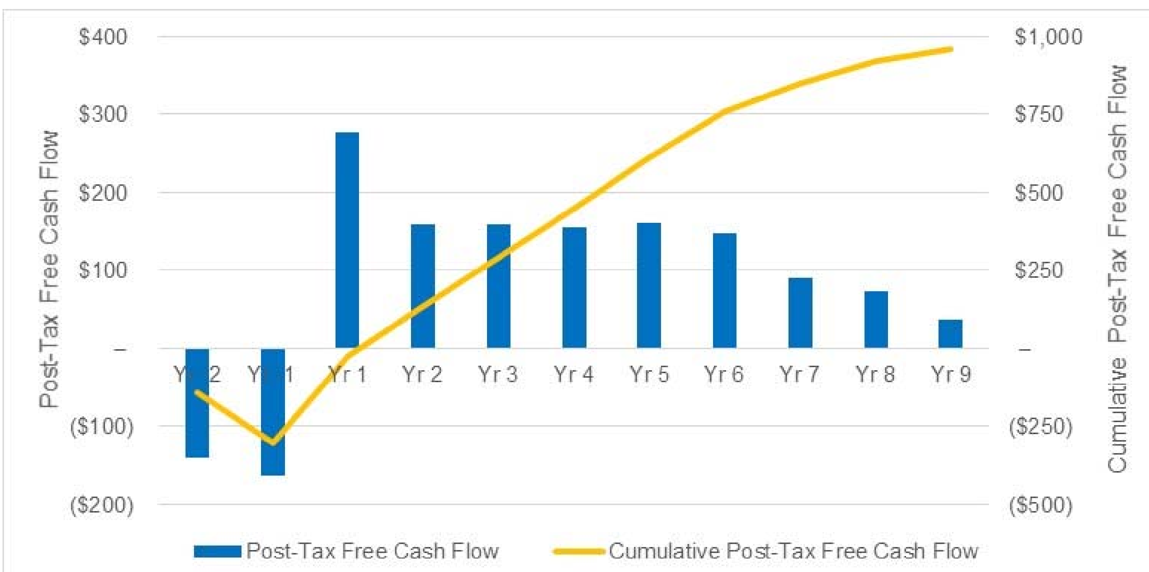
Table 1-5: Summary, Projected LOM Cashflow Assumptions and Results

	Units	Values
General Assumptions		
Gold price	(US$)	1,325
Silver price	(US$)	16
Exchange rate	(US$/C$)	0.77
Fuel cost	(C$/litre)	1.04
Power cost	(C$/kwh)	0.06
Discount rate	(%)	5
Net smelter royalty	(%)	1%
Contained Metals		
Contained gold ounces	(koz)	2,212
Contained silver ounces	(koz)	53,404
Contained gold equivalent ounces	(koz)	2,857
Production		
Gold recovery	(%)	91.1
Silver recovery	(%)	92.4
LOM gold production	(koz)	2,022
LOM silver production	(koz)	49,872
LOM gold equiv. production	(koz)	2,624
LOM avg. annual gold production	(koz)	236
LOM avg. annual silver production	(koz)	5,812
LOM avg. annual gold equiv. production	(koz)	306
Operating Costs Per Tonne		
Mining cost	(C$/t mined)	3.44
Mining cost	(C$/t milled)	26.32
Processing cost	(C$/t milled)	21.64
G&A cost	(C$/t milled)	6.06
Total operating costs	(C$/t milled)	54.03
NSR Parameters		
Gold payability	(%)	95%
Silver payability	(%)	80%
Treatment charges	(US$/dmt)	$180
Gold refining charges	(US$/oz)	$15
Silver refining charges	(US$/oz)	$1
Transport to smelter	(C$/wmt)	$118
Cash Costs and All-in Sustaining Costs		
LOM cash cost net of silver by-product	(US$/oz Au)	$582

	Units	Values
LOM cash cost co-product	(US$/oz AuEq)	$731
LOM AISC net of silver by-product	(US$/oz Au)	$615
LOM AISC co-product	(US$/oz AuEq)	$757
Capital Expenditures		
Pre-production capital expenditures	(C$M)	$303
Sustaining capital expenditures	(C$M)	$27
Reclamation cost	(C$M)	$52
Economics		
Pre-tax NPV (5%)	(C$M)	$993
Pre-tax IRR	(%)	63.3%
Pre-tax payback period	(years)	1.1
After-tax NPV (5%)	(C$M)	$638
After-tax IRR	(%)	50.5%
After-tax payback period	(years)	1.2
Average annual after-tax free cash flow (Year 1–9)	(C$M)	$147
LOM after-tax free cash flow	(C$M)	$959

Notes: Cash costs are inclusive of mining costs, processing costs, site G&A, treatment and refining charges and royalties. AISC includes cash costs plus corporate G&A, sustaining capital and closure costs. Gold equivalent (AuEq) calculated using the formula: Au (g/t) + [Ag (g/t) / 82.8].

Figure 1-1: Projected LOM Cashflow



Note: Figure prepared by Ausenco, 2019.

Table 1-6: Sensitivity Analysis Summary

Sensitivity Summary	Units	Lower Case	Base Case	Higher Case
Gold price	US$/oz	1200	1325	1500
Silver price	US$/oz	14	16	18
After-tax NPV	C$ M	453	638	878
After-tax IRR	%	39.7	50.5	62.5
After-tax payback	years	1.6	1.2	0.9
Average annual after-tax free cashflow, Year 1–9	C$ M	117	147	187

1.21 Risks and Opportunities

1.21.1 Risks

1.21.1.1 Geology and Resource Modelling

The current understanding of the distribution variability of elements that can be deleterious in concentrates is based on incomplete data, as epithermal and base metal elements were only selectively sampled in the legacy drill programs. It is expected that information obtained from the planned drill programs will provide more complete data on elemental distributions within key lithologies and domains, which in turn is likely to affect the domain and grade-shell outlines as interpreted in the current Mineral Resource estimate. The risk is that the variability is much higher than currently estimated, and that the model underestimates the deleterious elemental tonnages and grades that the PEA mine plan and concentrate marketability assumptions are based on.

1.21.1.2 Mining

Mining through voids during open pit operations is a generally manageable risk where such voids are known to exist. However, unidentified voids may exist, and present a risk to mine and production plans if alternate schedules have to be derived, or new safety measures implemented.

1.21.1.3 Process

Solid/liquid separation issues could increase process costs due to larger thickeners and filters and use of flocculant.

Higher mass pull to final concentrate might result without careful control on grinding pulp chemistry (e.g. stainless-steel media).

1.21.1.4 Infrastructure

A portion of the access road passes through topography which is known to have an elevated geohazard (e.g. avalanche) risk. There is potential for geohazard events to temporarily halt movement along the access corridor.

1.21.1.5 Environmental, Permitting and Social

The current permits for the Eskay Mine do not consider operations at the scale contemplated in this PEA. Additional work will be required to support permit updates and amendment applications, which will include environmental baseline data collection and environmental assessment.

The project is within the territories of Indigenous groups. Agreements with such groups that may be affected by the envisaged project remain to be negotiated.

1.21.2 Opportunities

1.21.2.1 Exploration

Exploration activities are likely to identify additional mineralization, and these efforts could result in changes to the style of mineralization to that currently identified, the scale of the Project, and the deleterious elemental issues identified.

1.21.2.2 Resource Modelling

There is upside Project potential if mineralisation currently classified as Inferred can be upgraded to higher confidence categories.

1.21.2.3 Mining

Material within the 1 m buffer around old stopes is currently classified and modelled as waste in the open pit model, and in the underground model, a 3 m buffer is assumed. With additional sampling, some or all of the buffer zone materials may be able to be brought into the mill feed, and may contain grade.

With detailed metallurgical testwork information on lithologies and zones, the mining sequence may be altered to provide higher value initially

There is potential for improved slope design, when additional geotechnical data such as waste rock strength and joint orientations, are available from drill testing.

1.21.2.4 Process

Higher gold and silver recoveries may be obtained from lower head grade samples with optimised flotation conditions.

Pre-concentration by screening and/or bulk sorting might reject waste material and increase plant feed grade.

1.21.2.5 Marketability

There is upside potential for the project if the planned drill programs more comprehensively document deleterious elemental distributions such that the levels of these elements, in particular arsenic and mercury, can be minimised in the concentrate to below smelter penalty thresholds.

1.22 Interpretation and Conclusions

Based on the assumptions and parameters presented in this Report, the PEA shows positive economics. The PEA supports that additional more detailed studies are warranted.

1.23 Recommendations

The recommended work program is divided into two phases. The phases can be conducted concurrently, but some portions of the phase 1 work plan would be incorporated into the phase 2 recommendations.

The first recommendations phase totals about $11.49 M. Recommendations consist of drilling; determination of whether bulk ore-sorting could potentially be implemented at the pre-mining stage; a study to determine if a relationship between rock mass structure and head grade exists; additional metallurgical testwork; materials handling tests; mine geotechnical data collection, data reviews in support of geotechnical and hydrological assumptions; additional hydrological data gathering; water treatment testwork; review of cost assumptions for grade control; additional mine studies, reviews of available climate data; collection of additional climate-related information, and geotechnical data collection in support of infrastructure locations and designs, and data collection on potential borrow pit sources.

The second phase is estimated at about $4.6 M, and will consist of project environmental, permitting, and social de-risking activities, including baseline and targeted environmental studies, negotiation of agreements with Indigenous groups, stakeholder engagement, an environmental assessment, application for operating permits, and an updated water balance to better understand makeup requirements, distribution of site flows, site water quality and water treatment requirements.

2 INTRODUCTION

2.1 Introduction

Ausenco Engineering Canada Inc. (Ausenco), Hemmera Envirochem Inc., an Ausenco company (Hemmera), SRK Consulting (Canada) Inc. (SRK), and AGP Mining Consultants Inc. (AGP), have prepared an updated preliminary economic assessment (PEA) report for Skeena Resources Limited (Skeena) on the volcanogenic massive sulphide (VMS) Eskay Creek Project (the Project) located in British Columbia (Figure 2-1).

Skeena currently holds the Project through an option agreement with Barrick Gold Inc. (Barrick); refer to Section 4.

The Project hosts the previously-mined Eskay Creek deposit, which was in operation as an underground mine from 1995–2008.

2.2 Terms of Reference

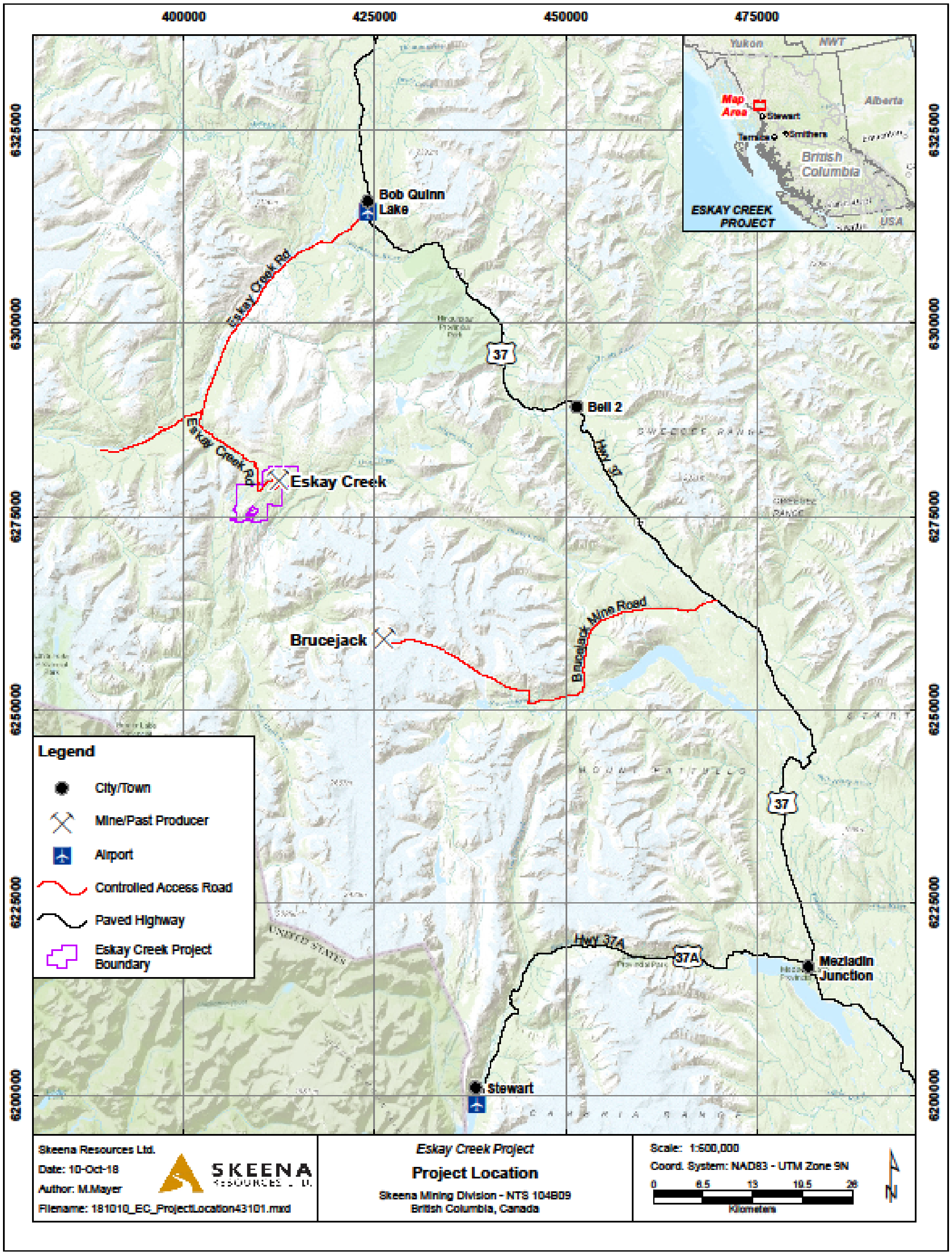
The Report supports disclosures by Skeena in a news release dated 7 November, 2019 entitled "Skeena Delivers Robust Project Economics for Eskay Creek: After-Tax NPV5% of C$638M, 51% IRR and 1.2 Year Payback".

All measurement units used in this Report are metric unless otherwise noted. Currency is expressed in Canadian (C) dollars (C$). The Report uses Canadian English.

Mineral Resources and Mineral Reserves are reported in accordance with the Canadian Institute of Mining, Metallurgy and Petroleum (CIM) Definition Standards for Mineral Resources and Mineral Reserves (May 2014; the 2014 CIM Definition Standards) and the CIM Estimation of Mineral Resources and Mineral Reserves Best Practice Guidelines (November 2003; 2003 CIM Best Practice Guidelines).

As the ownership of, and ownership interests in, the historical mining operations changed hands numerous times during the production history (refer to Section 6), the Report uses the term "previous operator" to refer to work done from 1988 to 2017. The term "legacy" is used for data generated by the previous operator. Skeena obtained its option interest in December 2018.

Figure 2-1: Project Location Plan



Note: Brucejack Mine is owned by third parties.

2.3 Qualified Persons

This Report was prepared by the following Qualified Persons (QPs):

- Mr Robin Kalanchey, P.Eng., Director, Minerals & Metals – Western Canada, Ausenco;
- Mr Scott Elfen, P.E., Global Lead Geotechnical Service, Ausenco;
- Mr Scott Weston, P.Geo., Vice President, Business Development, Hemmera;
- Ms Sheila Ulansky, P.Geo. Senior Resource Consultant, SRK;
- Dr Adrian Dance, P.Eng., Principal Metallurgist, SRK;
- Mr Gordon Zurowski, P.Eng., Principal Mine Engineer, AGP;
- Mr Willie Hamilton, P.Eng., Senior Mining Engineer, AGP.

2.4 Site Visits and Scope of Personal Inspection

Ms. Ulansky visited the Eskay Creek property from 27–28 June 2018. During that visit she viewed the general topography, independently located and surveyed 50 surface drill hole collars, and inspected the existing mine infrastructure.

Mr Hamilton visited the Eskay Creek site from 21–22 August 2019. On 21 August, he travelled by vehicle and observed the existing site facilities, active exploration drilling sites, the Tom Mackay tailings storage facility (TMSF) and the Albino Lake storage facility (SF) site. On August 22, Mr Hamilton toured the site via helicopter and also spent time reviewing drill core in the core shed.

2.5 Effective Dates

The Report has a number of effective dates as follows:

- Date of supply of last information on mineral tenure, surface rights and agreements: 27 June 2019;
- Date of supply of most recent information on ongoing drill program: 8 December 2019;
- Mineral Resource estimate: 26 November 2018;
- Date of PEA financial analysis: 7 November 2019.

The overall effective date of this Report is the effective date of the financial analysis which is 7 November 2019.

2.6 Information Sources and References

The Report is primarily based on a preliminary economic assessment prepared for Skeena in June–November 2019, and supporting memoranda and trade-off studies. This Report is also based in part on internal company reports, maps, published government reports, and public information, as listed in Section 27 of this Report. It is also based on the information cited in Section 3.

Additional information was sought from Skeena employees in their areas of expertise as required.

2.7 Previous Technical Reports

Skeena has filed the following technical reports on the Project:

- Ulansky, S., Uken, R., and Carlson, G., 2019: Independent Technical Report on the Eskay Creek Au-Ag Project, Canada: report prepared by SRK Consulting (Canada) Inc. for Skeena, effective date 28 February 2019.
- Ulansky, S., Uken, R., and Carlson, G., 2018: Independent Technical Report on the Eskay Creek Au-Ag Project, Canada: report prepared by SRK Consulting (Canada) Inc. for Skeena, effective date 6 July 2018.

3 RELIANCE ON OTHER EXPERTS

3.1 Introduction

Mr Kalanchey has relied upon the following other expert reports, which provided information regarding marketing in sections of this Report.

3.2 Marketing

Mr Kalanchey has not independently reviewed the marketing, smelter terms, or metal price forecast information. Mr Kalanchey has fully relied upon, and disclaims responsibility for, information derived from experts retained by Skeena for this information through the following documents:

- Open Mineral, 2019: Market Study and Contracts: report prepared by Open Mineral AG for Skeena, 4 November, 2019, 6 p.;
- Wood Mackenzie, 2019: Eskay Creek Concentrate Marketability Assessment: report prepared by Wood Mackenzie for Skeena, 10 December, 2019, 7 p.

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

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

Mr Kalanchey considers it reasonable to rely on Open Mineral for such information as the company is a specialist in commodities trading, and provides a web-based platform for trading base metal raw materials and secondary products. Within this platform, Open Minerals provides a calculator outlining all payables, penalties, deductions and charges, to arrive at estimated value of concentrate. It also provides information for sellers, buyers, and finance providers to understand the value of each material based on the current market terms. Open Mineral offers its clients customized trading solutions utilizing long-term relationships with hundreds of buyers and sellers in the base metal industry worldwide. Open Mineral is currently involved in the placement of gold and silver-bearing concentrates directly with toll refining facilities and has provided their report on the basis of dynamic trading activities.

Mr Kalanchey considers it reasonable to rely on Wood Mackenzie for marketing information because the company is a global leader in forecasting global and regional market fundamentals, and specializes in providing detailed commodity market studies spanning supply, demand and pricing outlooks, and industry trends analysis.

4 PROPERTY DESCRIPTION AND LOCATION

4.1 Introduction

The Eskay Creek Project is located in the Golden Triangle region of British Columbia, Canada, 83 km northwest of Stewart, with approximate centroids at 56° 39’ 13.9968” N and 130° 25’ 44.0004” W.

4.2 Project Ownership

Skeena entered into an option agreement with Barrick Gold Inc. (Barrick) in December 2017. Under the terms of the agreement (see Section 4.4), Skeena may earn a100% interest in the Project.

4.3 Mineral Tenure

The Eskay Creek Project covers 5,093.81 ha, consisting of 40 mineral claims and eight mineral leases (Table 4-1). Tenure locations are shown in Figure 4-1. This figure also outlines the area of mineralization within the Eskay Creek deposit.

Where on-ground work commitments have not been met, Skeena has made cash-in-lieu payments as stipulated under the BC regulations.

All statutory annual reporting obligations had been met as at 12 December 2019.

4.4 Property Agreements

On December 18, 2017, Skeena and Barrick Gold Inc. entered into an Option Agreement on the Eskay Creek Property. This agreement affects all mineral claims and mineral leases that comprise the Eskay Creek Property, except for the single mineral claim registered to Skeena Resources Ltd.

Skeena has the option to acquire all of Barrick’s rights, title and interest in and to the Eskay Creek Assets (property and all facilities and portions of the Coast Road, an access road connecting the mine site to the public highway system), the permits (including the Coast Road Special Use Permit), and the Eskay Creek contracts by incurring $3.5 million of exploration expenditures in the Project area by December 18, 2020.

In addition, to exercise the option Skeena must reimburse Barrick the aggregate amount of Barrick’s reclamation expenditures during the Option Period plus pay a $10 million cash purchase price. Skeena must also post an environmental bond which in December 2017 was estimated at $7.7 million. If the reclamation expenditures and the bond requirement are in aggregate greater than $7.7 million, then the cash purchase payment will decrease by a corresponding amount. The purchase price, the reclamation expenditures reimbursement and the bond amount are collectively not to exceed $17.7 million.

After closing Barrick will retain a 12 month back in right to the Esaky project for 51% by paying Skeena three times its cumulative expense on the project and reimbursing the purchase price and 51% of the bond amount.

Table 4-1: Mineral Tenure Summary Table

Tenure #	Claim Name	Owner 1	Owner 2	Title Sub Type	Map Number	Issue Date	Good to Date	Status	Area (ha)	Royalty Holder 1	Royalty 1	Royalty Holder 2	Royalty 2	Royalty Vendor(s)
316357		141657 (66.6666%)	104202 (33.3333%)	Lease	104B068	30-Apr-1994	30-Apr-2020	Good	276.7	ARC Resource Group Ltd.	2%	Fred Schomig	2%	Barrick Gold Inc.
316358		141657 (66.6666%)	104202 (33.3333%)	Lease	104B068	30-Apr-1994	30-Apr-2020	Good	367.7	ARC Resource Group Ltd.	2%	Fred Schomig	2%	Barrick Gold Inc.
316359		141657 (66.6666%)	104202 (33.3333%)	Lease	104B068	30-Apr-1994	30-Apr-2020	Good	278.7	ARC Resource Group Ltd.	2%	Fred Schomig	2%	Barrick Gold Inc.
300298	P-1	141657 (100%)		Claim	104B088	11-Jun-1991	20-May-2020	Good	25					
300299	P-2	141657 (100%)		Claim	104B088	11-Jun-1991	20-May-2020	Good	25					
300300	P-3	141657 (100%)		Claim	104B088	11-Jun-1991	20-May-2020	Good	25					
300301	P-4	141657 (100%)		Claim	104B088	11-Jun-1991	20-May-2020	Good	25					
306611		141657 (100%)		Lease	104B068	1-Jun-1992	1-Jun-2020	Good	41.8	Franco-Nevada Corp.	1%			Barrick Gold Inc.
306627		141657 (100%)		Lease	104B068	1-Jun-1992	1-Jun-2020	Good	355	Franco-Nevada Corp.	1%			Barrick Gold Inc.
252967	CAL #3	141657 (66.67%)	104202 (33.33%)	Claim	104B068	6-Aug-1989	22-Jun-2020	Good	400					
352974	STAR 21	141657 (100%)		Claim	104B068	7-Dec-1996	22-Jun-2020	Good	250	David A. Javorsky	1%			Barrick Gold Inc.
329241	MACK 23	141657 (100%)		Claim	104B068	21-Jul-1994	24-Jun-2020	Good	500					
329244	MACK 1	141657 (100%)		Claim	104B068	21-Jul-1994	24-Jun-2020	Good	25					

Tenure #	Claim Name	Owner 1	Owner 2	Title Sub Type	Map Number	Issue Date	Good to Date	Status	Area (ha)	Royalty Holder 1	Royalty 1	Royalty Holder 2	Royalty 2	Royalty Vendor(s)
329245	MACK 2	141657 (100%)		Claim	104B068	21-Jul-1994	24-Jun-2020	Good	25					
329246	MACK 3	141657 (100%)		Claim	104B068	21-Jul-1994	24-Jun-2020	Good	25					
329247	MACK 4	141657 (100%)		Claim	104B068	21-Jul-1994	24-Jun-2020	Good	25					
329248	MACK 5	141657 (100%)		Claim	104B068	21-Jul-1994	24-Jun-2020	Good	25					
329249	MACK 6	141657 (100%)		Claim	104B068	21-Jul-1994	24-Jun-2020	Good	25					
329252	MACK 9	141657 (100%)		Claim	104B068	21-Jul-1994	24-Jun-2020	Good	25					
329253	MACK 10	141657 (100%)		Claim	104B068	21-Jul-1994	24-Jun-2020	Good	25					
329254	MACK 11	141657 (100%)		Claim	104B068	21-Jul-1994	24-Jun-2020	Good	25					
329255	MACK 12	141657 (100%)		Claim	104B068	21-Jul-1994	24-Jun-2020	Good	25					
329256	MACK 13	141657 (100%)		Claim	104B068	21-Jul-1994	24-Jun-2020	Good	25					
329257	MACK 14	141657 (100%)		Claim	104B068	21-Jul-1994	24-Jun-2020	Good	25					
329258	MACK 15	141657 (100%)		Claim	104B068	21-Jul-1994	24-Jun-2020	Good	25					
329259	MACK 16	141657 (100%)		Claim	104B068	21-Jul-1994	24-Jun-2020	Good	25					
329260	MACK 17	141657 (100%)		Claim	104B068	21-Jul-1994	24-Jun-2020	Good	25					
329261	MACK 18	141657 (100%)		Claim	104B068	21-Jul-1994	24-Jun-2020	Good	25					
329262	MACK 19	141657 (100%)		Claim	104B068	21-Jul-1994	24-Jun-2020	Good	25					
329263	MACK 20	141657 (100%)		Claim	104B068	21-Jul-1994	24-Jun-2020	Good	25					

Tenure #	Claim Name	Owner 1	Owner 2	Title Sub Type	Map Number	Issue Date	Good to Date	Status	Area (ha)	Royalty Holder 1	Royalty 1	Royalty Holder 2	Royalty 2	Royalty Vendor(s)
329264	MACK 21	141657 (100%)		Claim	104B068	21-Jul-1994	24-Jun-2020	Good	25					
329265	MACK 22	141657 (100%)		Claim	104B068	21-Jul-1994	24-Jun-2020	Good	25					
329363	MACK 26 FR.	141657 (100%)		Claim	104B068	3-Aug-1994	24-Jun-2020	Good	25					
352975	STAR 22	141657 (100%)		Claim	104B068	7-Dec-1996	24-Jun-2020	Good	150	David A. Javorsky	1%			Barrick Gold Inc.
512867		141657 (100%)		Claim	104B	17-May-2005	24-Jun-2020	Good	106.808					
512881		141657 (100%)		Claim	104B	18-May-2005	24-Jun-2020	Good	17.804					
252976	IKS 2	141657 (100%)		Claim	104B068	2-Aug-1989	12-Jul-2020	Good	500	ARC Resource Group Ltd.	2%			Barrick Gold Inc.
252966	CAL #2	141657 (66.67%)	104202 (33.33%)	Claim	104B068	5-Aug-1989	15-Jul-2020	Good	500					
306286		141657 (100%)		Lease	104B068	13-Aug-1991	13-Aug-2020	Good	73.56	Franco-Nevada Corp.	1%			Barrick Gold Inc.
365539	KAY 1	141657 (100%)		Claim	104B068	12-Sep-1998	6-Oct-2020	Good	25					
365541	KAY 3	141657 (100%)		Claim	104B068	12-Sep-1998	6-Oct-2020	Good	25					
365542	KAY 4	141657 (100%)		Claim	104B068	12-Sep-1998	6-Oct-2020	Good	25					
365543	KAY 5	141657 (100%)		Claim	104B068	12-Sep-1998	6-Oct-2020	Good	25					
365544	KAY 6	141657 (100%)		Claim	104B068	12-Sep-1998	6-Oct-2020	Good	25					
365545	KAY 7	141657 (100%)		Claim	104B068	12-Sep-1998	6-Oct-2020	Good	25					
365546	KAY 8	141657 (100%)		Claim	104B068	12-Sep-1998	6-Oct-2020	Good	25					

Tenure #	Claim Name	Owner 1	Owner 2	Title Sub Type	Map Number	Issue Date	Good to Date	Status	Area (ha)	Royalty Holder 1	Royalty 1	Royalty Holder 2	Royalty 2	Royalty Vendor(s)
365547	KAY 9	141657 (100%)		Claim	104B068	12-Sep-1998	6-Oct-2020	Good	25					
365548	KAY 10	141657 (100%)		Claim	104B068	12-Sep-1998	6-Oct-2020	Good	25					
512879		141657 (100%)		Claim	104B	18-May-2005	6-Oct-2020	Good	35.58					
1056639	MELISSA	124845 (100%)		Claim	104B	24-Nov-2017	6-Oct-2020	Good	53.3585					
329944		141657 (100%)		Lease	104B068	6-Dec-1994	6-Dec-2020	Good	395	ARC Resource Group Ltd.	2%			Barrick Gold Inc.
254580		141657 (100%)		Lease	104B068	17-Dec-1990	17-Dec-2020	Good	41.8	Franco-Nevada Corp.	1%			Barrick Gold Inc.

Notes:

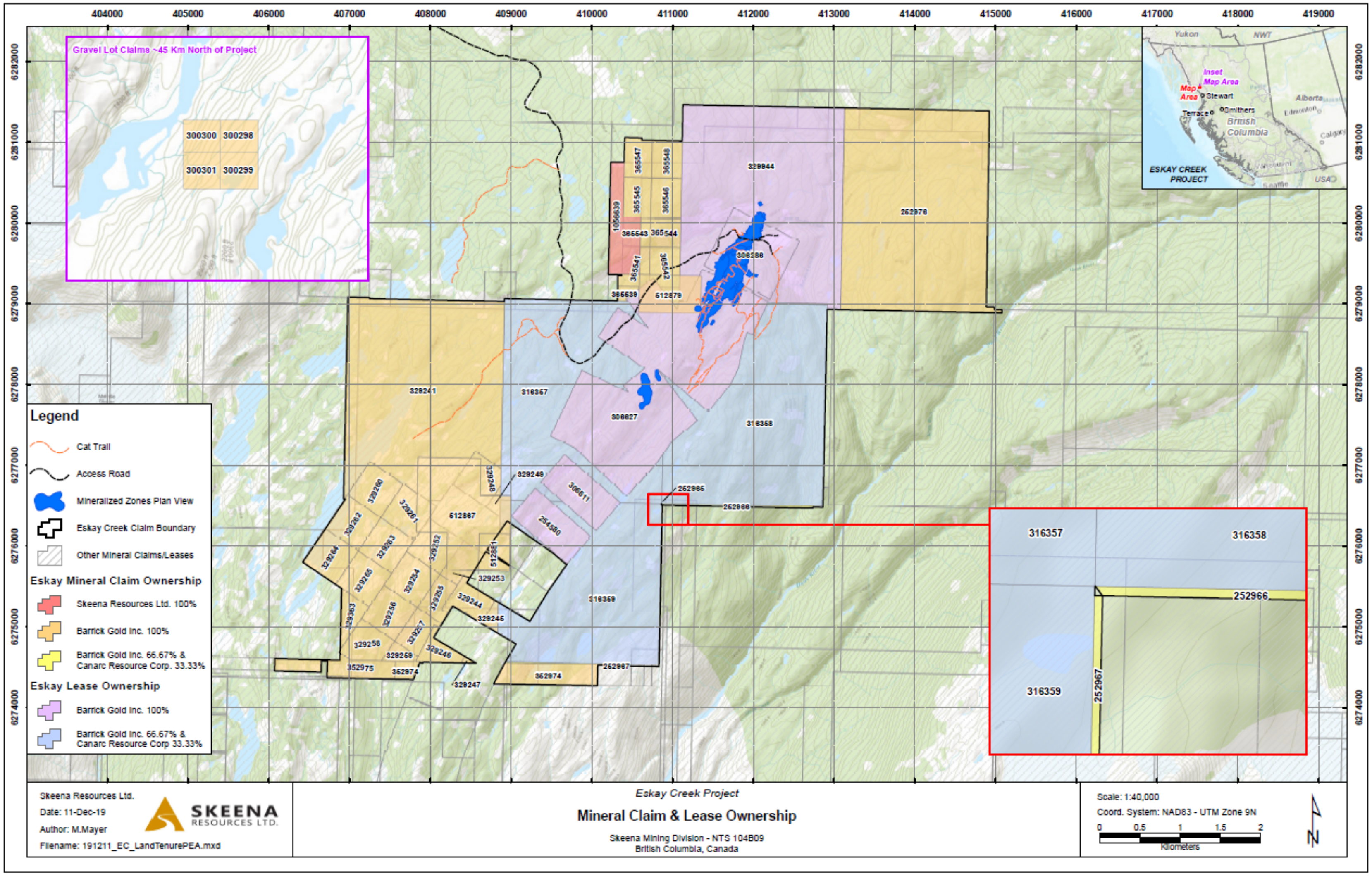
1 1% NSR payable to Euro-Nevada Mining Corporation Limited (now Franco-Nevada Corp.)

2 2% NSR payable to ARC Resource Group Ltd. (Option Agreement dated 04 November1988 between ARC Resource Group Ltd. and Canarc Resources Corp.)

3 2% NSR payable to ARC Resource Group Ltd. (Royalty Deed dated 01 August 1990 between Adrian Resources Ltd. and ARC Resource Group Ltd.)

4 1% NSR payable to David A. Javorsky.

Figure 4-1: Mineral Tenure Location Plan



4.5 Surface Rights

By way of the option agreement, Skeena holds the following surface rights interests:

- Surface lease number 634309: dated 24 December 1994 between the Province of BC and Prime Resources Group Inc.; interest assigned to Skeena;
- Surface lease number 740715: dated 25 July 2004 between the Province of BC and Optionor; interest assigned to Skeena;
- Special Use Permit S17635: for the use of the Eskay Creek road.

The locations of the surface leases are provided in Figure 4-2.

District Lots underly the Eskay Creek tenures, and a title search indicates that there are no mineral or surface rights associated with the District Lots. Skeena will need to acquire surface rights in support of any future mining and processing activities.

4.6 Water Rights

Skeena currently holds no water rights in relation to the Eskay Creek Project.

4.7 Royalties and Encumbrances

Prior to entering into the option agreement with Barrick, the Project was subject to the royalties summarized in Table 4-1. Figure 4-2 is a layout plan showing the tenures that have attached royalties.

Should Skeena elect to purchase the project, a 1% royalty will be payable to Barrick on all of the claims. Should Barrick elect to exercise the claw-back interest clause in the option agreement, Barrick will obtain a 51% Project interest, and the 1% Barrick royalty will extinguish.

4.8 Permitting Considerations

Permitting is discussed in Section 20.

4.9 Environmental Considerations

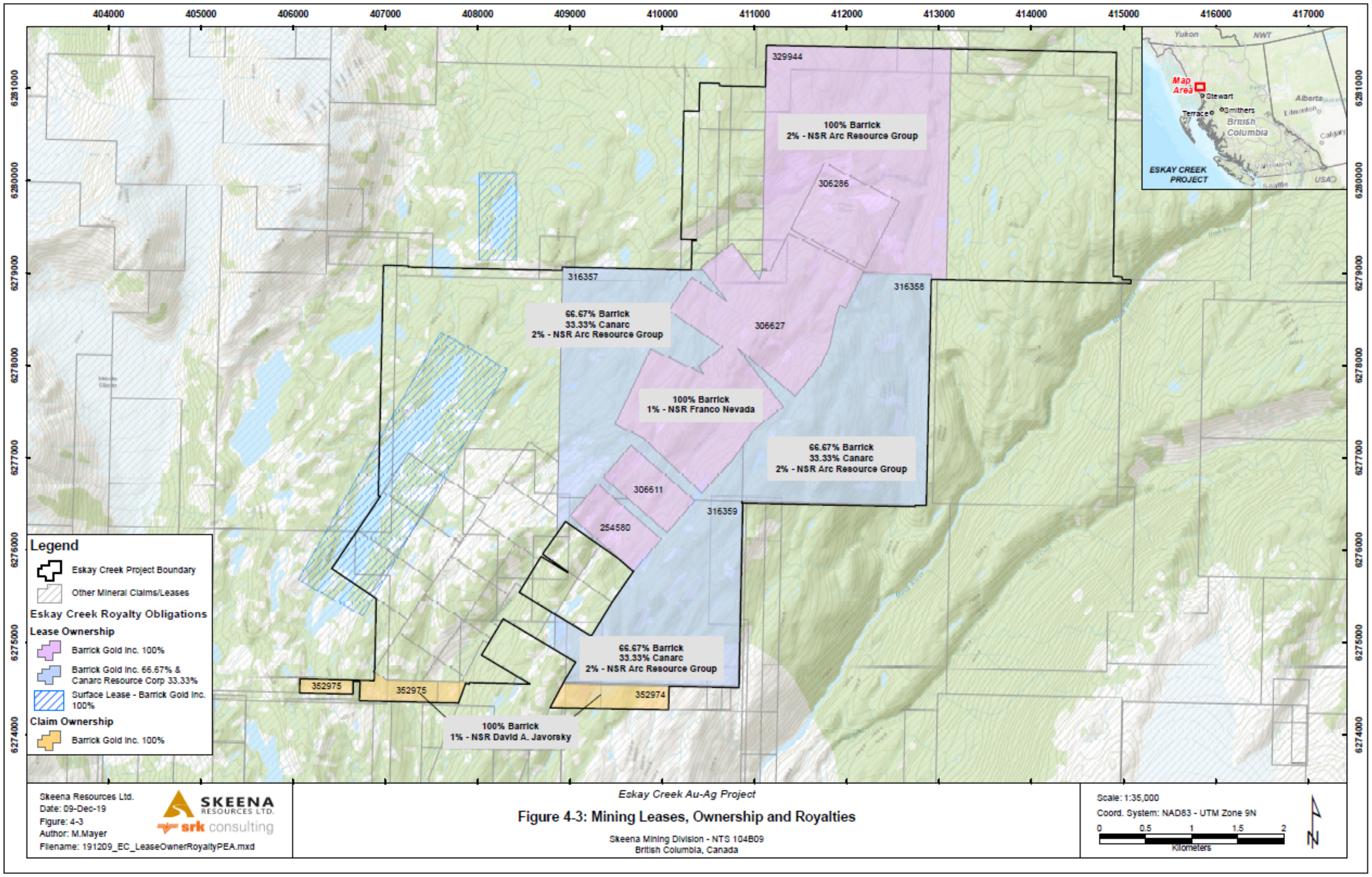
Environmental considerations are discussed in Section 20.

Skeena's current environmental liabilities are related to activities undertaken by Skeena, and activities arising from permitting. The key liabilities would be remediation of drill pads and drill access roads. Skeena has posted an environmental bond with the relevant BC authorities in relation to the work programs that have been conducted.

4.10 Social License Considerations

Social license considerations are discussed in Section 20.

Figure 4-2: Tenure Layout Plan Showing Royalty Interests



4.11 QP Comments on "Item 4; Property Description and Location"

The QP notes:

- Skeena has an option to earn a 100% interest in the Project;
- Mineral concessions are valid and in good standing;
- Barrick retains a 1% NSR royalty on tenements otherwise not subject to royalty payments;
- There are underlying royalties payable to third parties on some of the tenures;
- Skeena will need to acquire surface rights in support of any future mining and processing activities;
- Skeena currently holds no water rights in relation to the Eskay Creek Project.

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

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

5.1 Accessibility

Access to the Eskay Creek Project is via Highway 37 (Stewart Cassiar Highway). The Eskay Mine Road is an all-season gravel road that connects to Highway 37 approximately 135 km north of Meziadin Junction (refer to Figure 2-1). The Eskay Mine Road is a 54.5 km private industrial road that is operated by Altagas Ltd. (0 km to 43.5 km) and Skeena Resources Ltd. (43.5 km to 54.5 km).

There are two nearby gravel air strips: Bronson Strip which is about 40 km west of the mine site and Bob Quinn, roughly 37 km northeast of the Eskay Creek Project. Bronson Strip is a private air strip operated by Snip Gold Corporation. It is 1,500 m long and in fair condition. The Bob Quinn Strip is managed by the Bob Quinn Lake Airport Society, a not-for-profit organization consisting of residents and local business interests. The airstrip is about 1,300 m long and is in good condition.

5.2 Climate

The mean annual total precipitation at the former mine site is estimated to be 2,500 ± 500 mm. Data collection at the site from 1989–1993 indicated that about 55–71% of precipitation falls as snow. Snowpack data collected between 1990–1993 indicated peak snowpack (April) of 1,425 ±567 mm. Cumulative snowfall data at the mine site collected between 1999 and 2006 indicates a range of about 7.5–17.5 m of snow can fall between September and May.

The average temperature range is from -10.4°C in January to +15°C in July (Environment Canada, 2013).

Exploration activities can be curtailed by winter conditions. The previous mining operation was conducted on a year-round basis, and it is expected that any future operations will also be year-round.

5.3 Local Resources and Infrastructure

Support services for mining and other resource sector industries in the region are provided primarily by the communities of Smithers (pop. 5,400) and Terrace (pop. 11,500). Both communities are accessible by commercial airlines with regular flights to and from Vancouver.

Labour in support of exploration activities can be locally sourced. British Columbia has a long mining history and experienced mining personnel can be found within the Province.

Volume freight service in the region is supported by rail connections that extend from tidewater ports in Prince Rupert and Vancouver. The closest tidewater port to the project is in Stewart, approximately 260 km from the Project. Stewart is an ice-free shipping location and provides year-round access for bulk shipping.

The Project is in proximity to the new 287-kilovolt Northwest Transmission Line.

Additional information on local resources and infrastructure is provided in Section 18.

5.4 Physiography

The Eskay Creek Project lies in the Prout Plateau, a rolling subalpine upland with an average elevation of 1,100 m (amsl), located on the eastern flank of the Boundary Ranges. The plateau is characterized by northeast-trending ridges with gently-sloping meadows occupying valleys between the ridges. Relief over the plateau area ranges from 500 m in the existing Tom MacKay tailings storage facility (TSF) area to over 1,000 m in the Unuk River and Ketchum Creek valleys. The former Eskay Creek mine site is at approximately 800 m elevation.

Mountain slopes are heavily forested. Additional information on vegetation is included in Section 20.

Glacial features such as cirques, hanging valleys and over-steepened slopes, are present throughout the Project area. The plateau is surrounded by high serrate peaks containing cirque and mountain glaciers. The surficial geology in the area is varied, and includes till, colluvium at the base of bedrock outcrops and on steep slopes, organics in poorly-drained depressions, and alluvium along streams and the lake shorelines (Ulansky et al., 2018).

The Prout Plateau is drained by tributaries of the Stikine–Iskut and Unuk Rivers. Volcano Creek drains to the north into the Iskut River, a major tributary to the Stikine River system. The remainder of the plateau is drained almost exclusively by the Unuk River and its tributaries: the Tom MacKay, Argillite, Ketchum, Eskay and Coulter Creeks. The gradient of these drainages increases as the creeks descend from the moderate relief of the Prout Plateau into the deeply incised Unuk River valley. The plateau is occupied by the Tom MacKay, Little Tom MacKay and several smaller lakes as well as Argillite Creek, which collectively form the headwaters of the Tom MacKay Creek drainage system.

There are no known federal, provincial or regional parks, wilderness or conservancy areas, ecological reserves, or recreational areas near the Eskay Creek Project.

The Tahltan Nation has asserted Indigenous title and rights to this area in the Declaration of the Tahltan Tribe in 1910 (https://tahltan.org/central-government/). Previous operators have established formal agreements with the Tahltan Central Government regarding their ongoing participation at the mine site. More recently, Skii km Lax Ha Nation has produced maps indicating that the mine falls within their area of traditional land use (Rescan, 2009).

5.5 QP Comments on "Item 5; Accessibility, Climate, Local Resources, Infrastructure, And Physiography"

The QP notes:

- The existing local infrastructure, availability of staff, and methods whereby goods could be transported to the Project area to support exploration activities are well understood by Skeena, and can support the declaration of Mineral Resources and support evaluation at the PEA stage.
- There is sufficient area within the Project tenure holdings to allow for construction of all mine- and process-related facilities.

Surface rights for the Project are discussed in Section 4.5.

Former mining operations were conducted year-round, and it is expected that any operation conducted by Skeena would also be year-round.

6 HISTORY

6.1 Exploration History

The known exploration history of the Project area is summarized in Table 6-1.

Underground mining operations were conducted from 1994 to 2008. From 1994–1997, ore was direct-shipped after blending and primary crushing. From 1997 to closure in 2008, ore was milled on site to produce a shipping concentrate.

The Eskay Creek process plant began commercial production on 1 January 1998 at a 150 t/d rate. Production rates were incrementally increased from 1999–2004.

6.2 Production

The Eskay Creek mine production is summarized in Table 6-2.

Table 6-1: Project History

Year	Owner	Work Area	Description
1932	Unuk Gold/Unuk Valley Gold Syndicate	Unuk and Barbara Group claims (Core Property)	Prospecting
1933	Mackay Syndicate	Unuk and Barbara Claims	Trenching
1934	Mackay Syndicate/Unuk Valley Gold Syndicate	Unuk, Barbara and Verna D. Group Claims	Prospecting; core drilling (261.21 m)
1935–1938	Premier Gold Mining Co. Ltd.	Core Property	Optioned property and conducted prospecting, trenching, core drilling (1,825.95 m). Defined and named over 30 mineralised showings.
1939	MacKay Gold Mines Ltd.	#13 O.C./Mackay Adit	Financed by Selukwe Gold Mining and Finance Company Ltd. and acquired property. Conducted data review. Underground development of the MacKay Adit (84.12 m), which is about 3 km south of the Eskay Creek mine site.
1946	Canadian Exploration Ltd.	Mackay Adit	Optioned property. Conducted mapping, trenching. Underground development - extended the Mackay Adit to 109.73 m and put raise to surface at 46 m.
1947–1952	American Standard Mines Ltd./Pioneer Gold Mines of B.C. Ltd./New York-Alaska Gold Dredging Corp.	Canab Group (36 claims of the Mackay Group)	Optioned and conducted property examination.
1953	American Standard	Canab Group/Mackay Group 36 claims (No. 21, No. 22 and No. 5 areas)	Trenching (2,655.32 m). Open cutting in the 5, 21 and 22 zones. Core drilling (22 boreholes)
1954–1962	Western Resources Ltd.	Kay 1–18	Unknown – no work reported
1963	Western Resources Ltd.	Kay 1–18; Kay 19–36; Emma Adit	Underground development of the Emma Adit (111.25 m). Road building (13 km) from Tom Mackay Lake to property
1964	Stikine Silver Ltd. (Stikine Silver)/Canex Aerial Exploration Ltd.	Kay Group; Emma Adit	Optioned from Western Resources Ltd. Mapping, rock, stream sediment, and soil sampling. Underground core drilling (224.64 m)
1965	Stikine Silver	Kay Group (40 claims); Emma Adit	Trenching (1457.20 m in 18 trenches); core drilling (15.85 m). Underground development (extended Emma Adit to 178.61 m)
1967	Mount Washington Copper Co./Stikine Silver	Kay 1–36, (Core Property)	Electromagnetic (EM 16) and magnetometer geophysical surveys; petrography
1968–1970	Newmont Mining Corp.	Kay 1–8; Au 1–4; Kay 3–4	Surface and underground geological mapping; trenching (137.16 m)
1971–1972	Stikine Silver	22 Zone	Trenching. Surface bulk sample (1,515 kg grading 6.06 g/t Au, 4,451.56 g/t Ag, 2.8% Zn, 1.9% Pb).
1973	Kalco Valley Mines Ltd.	22 Zone	Surface geological mapping. Core drilling (299.62 m)
1975–1976	Texasgulf Canada Ltd.	#5 O.C.; #6 O.C.; (Kay 11–18, Tok 1–22 & Sib 1–16 claims)	Mapping (1:5,000, Donnelly, 1976 B.Sc. Thesis, UBC); line cutting; rock sampling; EM and magnetic geophysical surveys. Core drilling (373.38 m)

Year	Owner	Work Area	Description
1979	May-Ralph Resources Ltd.	22 Zone	Hand-cobbed bulk sample
1980–1982	Ryan Exploration Ltd. (U.S. Borax)	22 Zone; #6 Zone; Mackay Adit	Mapping; rock, stream sediment and soil sampling. Core drilling (452.32 m)
1985	Kerrisdale Resources Ltd.	#5 Zone; 21 Zone; 22 Zone	Mapping; rock and soil sampling. Core drilling (622.10 m)
1986	Consolidated Stikine Silver Ltd. (Consolidated Stikine)		Stikine Silver renamed to Consolidated Stikine.
1987	Consolidated Stikine	#3 Bluff; 5, 21 and 23 Zones	Stream sediment and soil sampling; core (all Kerrisdale) sampling; trench sampling
1988	Calpine Resources Inc. (Calpine)/Consolidated Stikine	21A; 21B Zones	Mapping; rock and soil sampling, core drilling (2,875.5 m). Discovery drill hole CA88-6 for 21A Zone
1989	Calpine/Consolidated Stikine	21A;21B Zones; 22 Zone	Prime Resources acquired a controlling interest in Calpine in 1989 and took over managing the Eskay Creek project. Prime Resources merged with Calpine in April 1990. Homestake Canada Inc. (Homestake) acquired an equity position in Consolidated Stikine. Mapping; rock and soil sampling; airborne magnetic, EM, and very low frequency (VLF) geophysical surveys; ground magnetic/VLF-EM, induced polarization (IP) geophysical surveys. Core drilling (44,338.9 m). Legal tenure surveys.
1990	Calpine/Consolidated Stikine	21B/21C Zones; PMP; Mack; proposed mill site; proposed mine site; GNC; Adrian	Mapping; rock and soil sampling; University of Toronto electro-magnetic system (UTEM) geophysical survey. Core drilling (141,412.86 m). Environmental and terrane studies. Geotechnical and metallurgical studies. Underground development (21B Zone). Bulk sample
1991	International Corona Corp. (Corona)	21B Zone; GNC	Mapping; rock and soil sampling; UTEM, seismic refraction and borehole frequency domain EM (FEM) geophysical surveys. Core drilling (2,791 m). Core relogging. Start of underground core drilling
1992	Corona	21B Zone; GNC	Mapping; rock and soil sampling; seismic refraction, gradient IP, transient EM, and borehole FEM geophysical surveys. Core drilling (3,342 m). Homestake acquired Corona.
1993	Homestake	21B Zone; GNC	Mapping; rock sampling; resistivity, borehole FEM geophysical surveys. Core drilling (1,606.6 m). Feasibility study. Completion of Eskay mine road. T. Roth MSc. thesis completed. R. Bartsch MSc. thesis completed.
1994	Homestake	21B Zone; Adrian; Albino Lake	Mapping, rock sampling; borehole EM geophysical surveys. Core drilling (4,080.95 m)

Year	Owner	Work Area	Description
1995	Homestake	21B Zone; NEX; Bonsai	Mapping; rock sampling. Core drilling (3,468.1 m). Start of production on 21B Zone.
1996	Homestake	21B Zone; NEX; HW; Adrian; Bonsai	Mapping; rock sampling; trenching. Core drilling (21,280.8 m). Orthophoto survey.
1997	Homestake	21B Zone; 21C/21E; Adrian; GNC; Mack; Star	Prospecting; silt sampling. Core drilling (16,220.47 m).
1998	Homestake	21C/21A; PMP; 5; 23; 22; 28; Mackay Adit; GNC; Mack; SIB Gaps; Star/Coulter	Mapping and prospecting; test gravity geophysical survey. Core drilling (21,909.63 m). Orthophoto survey
1999	Homestake	21C; 21A; PMP; Deep Adrian; West Limb; East Limb	Mapping and prospecting; structural study; geophysical compilation. Core drilling (17,363.96 m)
2000	Homestake	21C; 21A; PMP; Deep Adrian; West Limb; East Limb	Mapping and prospecting. Core drilling (25,893.93 m)
2001	Homestake	21C; 21A; PMP; Deep Adrian; West Limb; East Limb; Felsite Bluffs; SIB Gaps; Pillow Basalt Ridge	Mapping and prospecting. Core drilling (22,035.48 m)
2002	Barrick	21C; 21A; PMP; Deep Adrian; West Limb; 22 Zone; Mackay Adit	Acquired Homestake. Mapping and prospecting. Core drilling (15,115.69 m). T. Roth PhD. thesis completed
2003	Barrick	21C; 21A; PMP; Deep Adrian; West Limb; 22 Zone; Mackay Adit	Mapping and prospecting; IP and gravity geophysical surveys; line cutting. Core drilling (18,323.28 m)
2004	Barrick	22 Zone; Deep Adrian; West Limb; Ridge Block; Footwall	Mapping and prospecting; rock, soil, silt and vegetation sampling; topographic survey; borehole TEM geophysical survey. Core drilling (18,404.88 m)
2005	Barrick		Core drilling (16,000 m)
2008	Barrick		Mine closed in April. Reclamation commences.
2009–2016	Barrick		Mine reclaimed. Continuous care and maintenance
2017	Barrick/Skeena.		Skeena secures option
2018	Skeena.		Skeena files Notice of Work, commences Phase 1 diamond drill program consisting of 45 surface core drill holes on the 21A, 21C and 22 Zones (7,737.45 m), light detection and ranging (LiDAR) and photographic surveys. Mineral Resource estimate
2019	Skeena.		Updated Mineral Resource estimate; 209 surface core holes on the 21A, 21E and HW zones (14,267.27 m), metallurgical leaching testwork, PEA study

Table 6-2: Production History

Year	Gold Produced (oz)	Gold Produced (kg)	Silver Produced (kg)	Silver Produced (oz)	Ore Tonnes Milled	Ore Tonnes Shipped Direct
1995	196,550	6,113	309,480	9,950,401	—	100,470
1996	211,276	6,570	375,000	12,057,000	—	102,395
1997	244,722	7,612	367,000	11,799,784	—	110,191
1998	282,088	8,774	364,638	11,723,841	55,690	91,660
1999	308,985	9,934	422,627	13,588,303	71,867	102,853
2000	333,167	10,363	458,408	14,738,734	87,527	105,150
2001	320,784	9,977	480,685	15,454,984	98,080	109,949
2002	358,718	11,157	552,487	17,763,562	116,013	116,581
2003	352,069	10,951	527,775	16,969,022	115,032	134,850
2004	283,738	8,825	504,602	16,223,964	110,000	135,000
2005	190,221	5,917	323,350	10,396,349	103,492	78,377
2006	106,880	3,324	216,235	6,952,388	123,649	18,128
2007	68,000	2,115	108,978	3,503,861	138,772	0
2008	15,430	480	27,800	893,826	31,750	0
Totals	***3,272,628***	***102,112***	***5,039,065***	***162,016,018***	***1,051,892***	***1,205,604***

7 GEOLOGICAL SETTING AND MINERALIZATION

7.1 Regional Geology

The Iskut River region is located along the western margin of the Stikine Terrane, within the Intermontane Tectonic Belt of the Northern Cordillera (Figure 7-1). Anderson (1989) divides this area of the Stikine Terrane into four unconformity-bounded, tectonostratigraphic elements. Deformed and metamorphosed sedimentary and volcanic rocks of the Paleozoic Stikine Assemblage are overlain by volcano-sedimentary arc complexes of the Stikinia Assemblage (Triassic Stuhini Group and Lower to Middle Jurassic Hazelton Group). These units are subsequently overlain by Upper Jurassic to Lower Cretaceous siliciclastic sedimentary rocks of the Bowser Lake Group that formed an overlap assemblage following the amalgamation of the Stikine and Cache Creek Terranes. Six distinct plutonic suites have been recognized in the area and commonly intrude all assemblages.

Lower greenschist facies metamorphism is common throughout the area and is likely related to the Cretaceous deformation that formed the Skeena fold and thrust belt (Rubin et al., 1990; Evenchick, 1991). Deformation in the Iskut River area is characterized by regional upright anticlinoria and synclinoria, related thrust faults, mesoscopic folds and normal faults, and cleavage development. The regional-scale McTagg anticlinorium is the dominant structural feature, located in the eastern part of the Iskut River area.

The Iskut River region hosts many significant porphyry, precious-metal vein and volcanogenic massive sulphide deposits, the majority of which exhibit a close spatial relationship to Hazelton Group rocks (latest Triassic to Middle Jurassic) and their associated intrusions (Macdonald et al., 1996; Nelson et al., 2018).

7.2 Project Geology

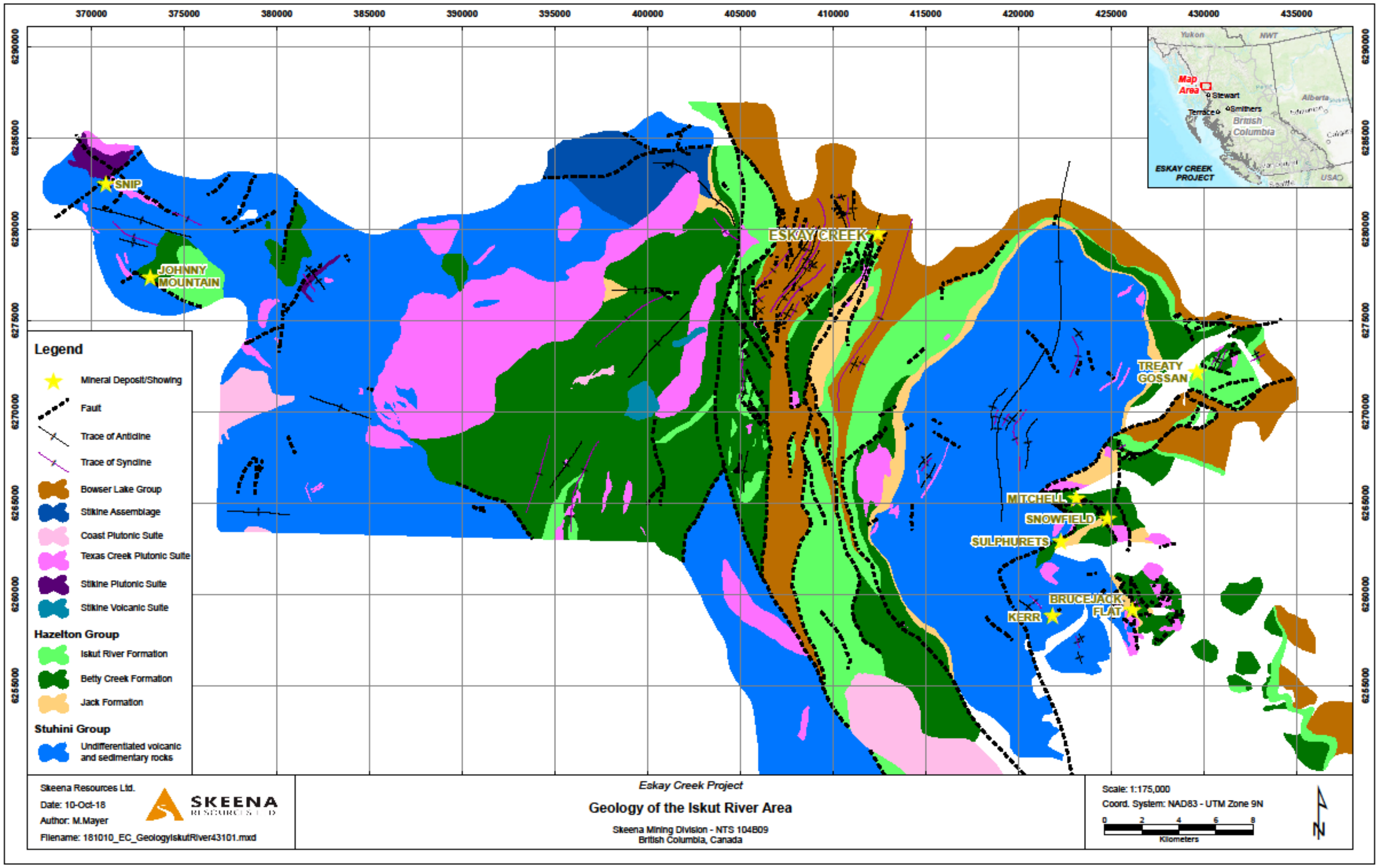
7.2.1 Lithologies

The Eskay Creek deposit is located near the northern margin of the Eskay Anticline, just below the stratigraphic transition from volcanic rocks of the uppermost Hazelton Group to marine sediments of the Bowser Lake Group (Figure 7-2).

Descriptions of units from the local mine stratigraphy have been compiled in Table 7-1 from Roth et al. (1999) with stratigraphic nomenclature taken from Nelson et al. (2018). A stratigraphic section through the Project area is included as Figure 7-3.

Intrusive units known in the Project area are summarized in Table 7-2.

Figure 7-1: Regional Geology



Note: Eskay Creek deposit is held by Skeena. Other mines and deposits shown are owned by third parties

Figure 7-2: Eskay Creek Local Geology Map

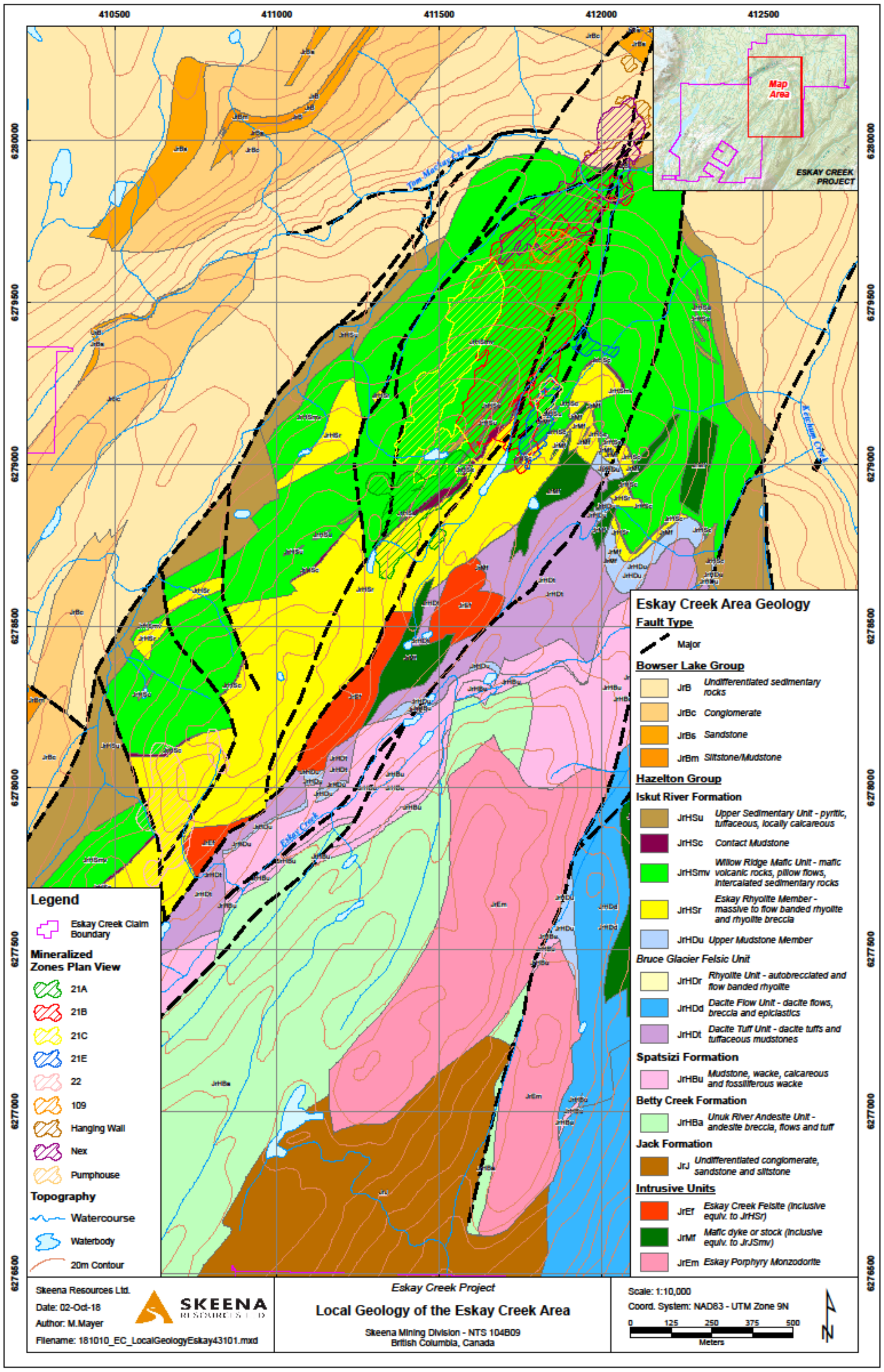
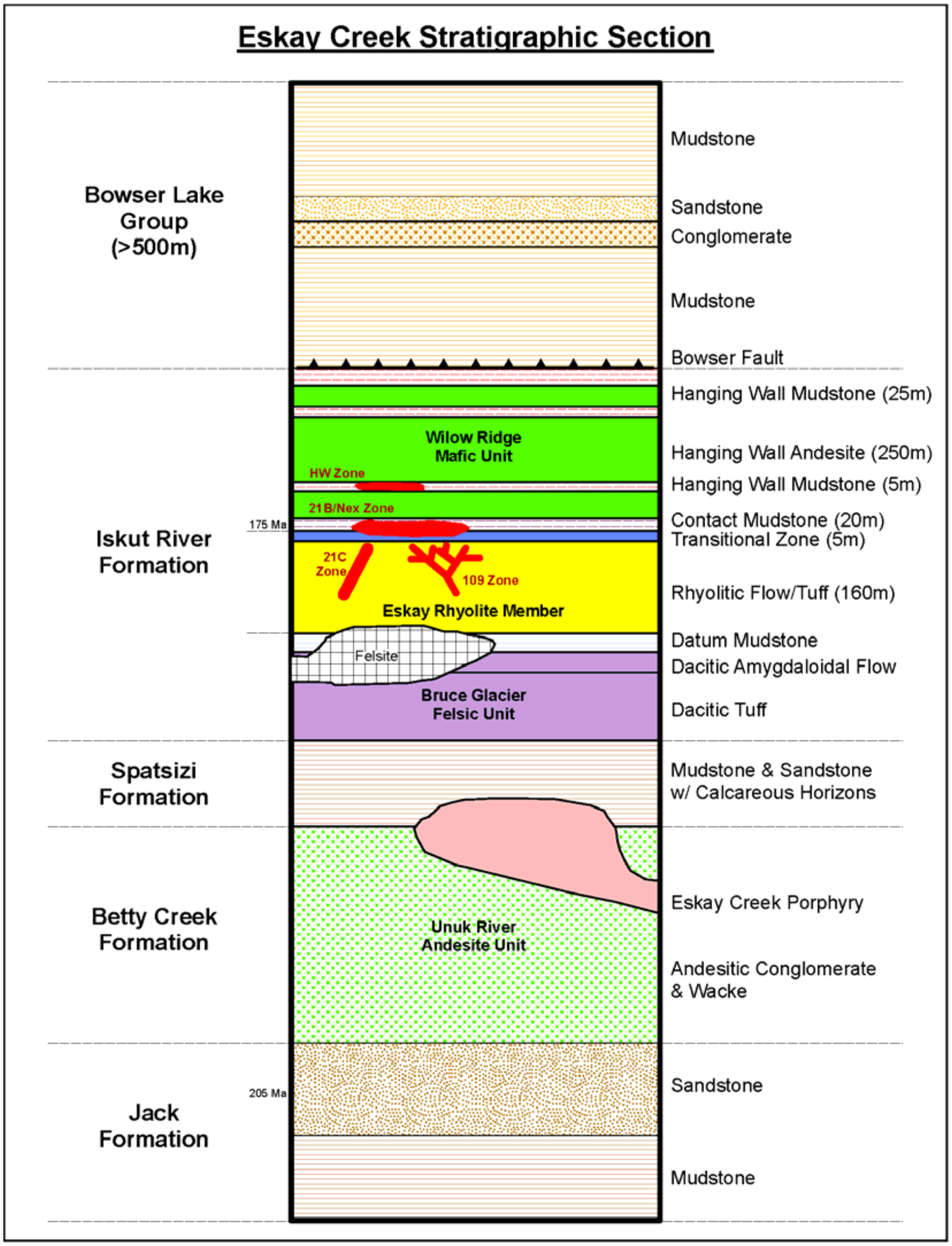


Table 7-1: Stratigraphic Units

Unit	Note
Recent	In-situ soils and transported tills
Bowser Lake Group	Mudstones and conglomerates
Willow Ridge mafic unit	Has both extrusive and intrusive phases, ranges from aphanitic to medium-grained with local feldspar phenocrysts, and in places exceeds 150 m thickness. Near the top of the sequence, well-preserved pillow flows and breccias, hyaloclastite, and basaltic debris flows containing minor mudstone and rhyolite clasts interspersed with thin argillite beds occur.
Contact Mudstone	Basal contact consists of a black-matrix breccia, comprising matrix-supported white rhyolite fragments set in a siliceous black matrix. Overlying the rhyolite and black matrix breccia are black mudstone and intercalated graded volcaniclastic sedimentary rocks. Within these volcaniclastic intervals, the presence of coarser rhyolite breccia fragments is interpreted to represent debris flows. The Contact Mudstone is the host unit for stratiform mineralization in the 21A, B, C, E, NEX and Hanging Wall (HW) Zones. It is characterized by laterally extensive, well-laminated, carbonaceous mudstone that is variably calcareous and siliceous and ranges from less than 1 m to more than 60 m in thickness.
Eskay Rhyolite member	Up to 200 m thick. Linear set of flow-dome complexes, with locally preserved flow bands, flow lobes, breccias, hyaloclastite, spherulites, and perlitic textures. Located in the immediate footwall to the economically significant stratiform mineralized bodies, and also hosts stringer-style discordant mineralization.
Datum Mudstone	Thin (5–15 m thick) black mudstone horizon
Datum Dacite	Amygdaloidal, aphanitic dacite flow or sill
Bruce Glacier felsic unit	Characterized by pumice-rich block and lapilli tuffs and heterogeneous epiclastic rocks that are locally fossiliferous
Spatsizi Formation	Marine shales and interbedded coarse clastic sedimentary, volcaniclastic, and calcareous rocks
Betty Creek Formation	Exposed in the core of the Eskay Anticline. Characterized by a thick sequence of coarse, monolithic andesite breccias and heterolithic volcaniclastic rocks.

Figure 7-3: Stratigraphic Section



Note: Figure courtesy Skeena, 2019.

Table 7-2: Intrusive Units

Unit	Note
Willow Ridge mafic unit	Basaltic dykes and sills. Where they cut the Contact Mudstone, their contacts are frequently brecciated and peperitic, suggesting the mudstone was still wet at the time of intrusion.
Felsic intrusive rocks	Chemically indistinguishable from the Eskay Rhyolite; display strong quartz, pyrite, and potassium feldspar alteration with minor sericite; may represent sub-volcanic portions of, or feeders to, the Eskay Rhyolite. Form a series of prominent gossanous bluffs which extend for 7 km to the southwest of the Eskay Creek deposit.
Eskay monzodiorite porphyry	Exposed in the core of the Eskay Anticline just south of the 21 Zone deposits. Predates the Eskay Rhyolite and mineralization located in the 21 Zone deposits.

7.2.2 Structure

Two structural events are recognized:

- D1: A mid-Cretaceous north-northwest compression event that formed northeast-trending, syncline–anticline couples and a spaced pressure solution cleavage. Bedding defines the Eskay Creek Anticline. The pressure solution cleavage is axial planar to the Eskay Creek Anticline, and is pervasive within the phyllosilicate-rich lithologies and even through the massive sulphide horizons. Faulting late in the D1 event resulted in the development of east-dipping thrust sheets, such as the Coulter Creek Fault, south of Eskay Creek. Regional metamorphism during the D1 event also resulted in the formation of porphyroblastic prehnite and calcite;
- D2: North–northeast-directed compression event, locally re-oriented the D1 cleavage planes and formed prominent north and northeast-trending, steeply-dipping faults. Crosscutting relationships suggest that the north set of faults are early with apparently consistent sinistral displacement (Edmunds and Kuran, 1992). The later northeast-trending set of faults commonly display oblique normal displacement. These faults form strong topographic lineaments and displace both stratigraphic contacts and mineralized zones.

7.2.3 Alteration

Alteration in the footwall volcanic units is characterized by a combination of pervasive quartz–sericite–pyrite, potassium feldspar, chlorite and silica. Intense alteration zones are locally associated with sulphide veins that contain pyrite, sphalerite, galena, and chalcopyrite (Roth et al., 1999).

Alteration zonation is common in the Eskay Rhyolite member (Roth et al., 1999), and is closely associated with the 21 Zone deposits. Rhyolite located lateral to and at deeper levels beneath the area of stratiform mineralization is commonly moderately silicified and potassium feldspar-altered. Silica alteration occurs as extremely fine-grained quartz flooding and densely developed quartz-filled micro veinlets. Potassium feldspar occurs as fine-grained replacement of plagioclase phenocrysts (Gale et al., 2004). Fractures that cut potassium feldspar–silica-altered rhyolite typically have sericitic alteration envelopes and contain very fine-grained pyrite. Where alteration is most intense, chlorite replaces sericite.

An intense, tabular-shaped blanket of chlorite–sericite alteration, up to 20 m thick, occurs in the Eskay Rhyolite member, immediately below the contact with the main stratiform sulphide mineralization. In these areas, magnesian chlorite has completely replaced the rhyolite to form a dark green, waxy rock consisting of clinochlore (Roth et al., 1999). This blanket coincides spatially with an area of greater rhyolite thickness and where extensive brecciation has developed in the upper part of the rhyolite unit. The brecciated zone likely created more pathways for hydrothermal fluids, and therefore greater surface area for fluid–rock interaction, resulting in development of the stronger alteration zone.

7.3 Deposit Descriptions

Several distinct styles of stratiform and discordant mineralization are present at the Eskay Creek Project, defined over an area approximately 1,400 m long and as much as 300 m wide (Figure 7-4). Early exploration efforts focused on discordant-style, precious metal mineralization hosted in sulphide veins within the rhyolite, felsic intrusions, and the footwall volcanic units. Following recognition of more significant stratiform mineralization, exploration expanded further to the north, defining the 21 Zone deposits. Distinct zones have been defined by variations in location, mineralogy, texture, and precious metal grades (Edmunds et al, 1994).

The main characteristics and stratigraphic locations of the mineralized zones, after Roth et al. (1999), are summarized in Table 7-3.

7.3.1 Stratiform Mineralization Zones

Stratiform-style mineralization is hosted in black carbonaceous mudstone and sericitic tuffaceous mudstone of the Contact Mudstone (Iskut River Formation), located between the footwall Eskay Rhyolite member and the hanging wall Willow Ridge mafic unit. The stratiform hosted zones include the 21B Zone, the NEX Zone, the 21A Zone (characterized by arsenic–antimony–mercury sulphides), the 21CZone, the 21Be Zone and the 21E Zone. Stratigraphically above the Contact Mudstone mineralization, and usually above the first basaltic sill, the mudstones also host a localized body of base metal-rich, relatively precious metal-poor, massive sulphides referred to as the Hanging Wall or HW Zone.

7.3.1.1 21A Zone

The 21A Zone is a gold–silver-rich sulphide lens that sits on the western flank of a small depression at the Eskay Rhyolite-mudstone contact, located 200 m south of the 21B Zone. Stratiform-style, mudstone-hosted mineralization averages 10 m thickness, and locally up to 35m. It is approximately 240 m wide, 250 m long and is bound to the east by the Pumphouse fault. It is underlain by a discontinuous zone of intense magnesian chlorite alteration and stockwork veining in the Eskay Rhyolite.

The sulphide lens consists of semi-massive to massive stibnite–realgar ± cinnabar ± arsenopyrite and local angular mudstone fragments. Disseminated stibnite, arsenopyrite, and tetrahedrite also occur in the immediate footwall of the sulphide lens within the intensely-sericitized rhyolite. Cinnabar is found in late fractures that cut the sulphide lens, the surrounding mudstone, and locally, the rhyolite. Realgar–calcite veinlets locally cut the mudstone in a restricted area adjacent to the sulphide lens.

Figure 7-4: Mineralized Zones

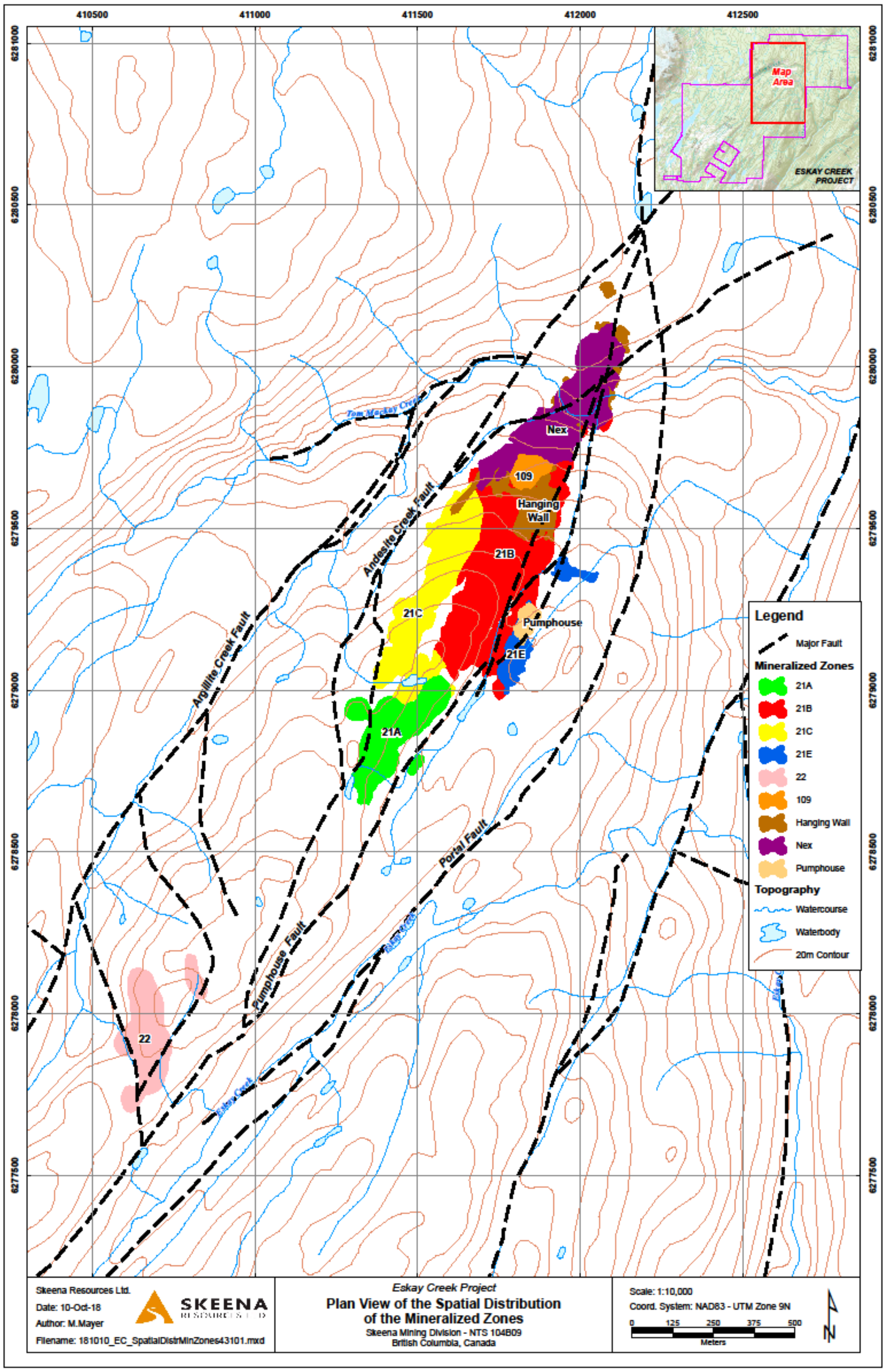


Table 7-3: Mineralized Zones

Zone	Associated Elements	Characteristics	Stratigraphic Position
21A	As–Sb–Hg–Au–Ag	Stratiform lenses of massive to semi-massive sulphides (realgar, stibnite, cinnabar, arsenopyrite).	At the base of the Contact Mudstone
		Disseminated stibnite, arsenopyrite, tetrahedrite, and veinlets of pyrite, sphalerite, galena, tetrahedrite ± chalcopyrite.	Hosted within the underlying rhyolite
21B	Au–Ag–Zn–Pb–Cu–Sb	Stratiform, bedded clastic sulphides and sulfosalts including sphalerite, tetrahedrite-freibergite, Pb sulfosalts (including boulangerite, bournonite, jamesonite), stibnite, galena, pyrite, electrum, and amalgam.	At the base of the Contact Mudstone
21C	Ba (Pb–Zn–Au–Ag)	Bedded massive to bladed barite associated with very fine-grained disseminated sulphides including pyrite, tetrahedrite, sphalerite and galena. Located sub-parallel to and down-dip of the 21B zone.	Within the Contact Mudstone
		Localized zones of cryptic, disseminated, precious metal bearing mineralization.	Hosted within the underlying rhyolite
21Be	Ag–Au–Zn–Pb–Cu	Fine-grained massive to locally clastic sulphides and sulfosalts. Massive pyrite flooding in rhyolite grading upwards into massive sulphides and sulfosalts.	Within a fault bounded block, mainly at the contact between mudstone and rhyolite
NEX	Au–Ag–Zn–Pb–Cu	The North Extension Zone (NEX) stratiform mineralization is similar to the 21E, and locally the 21Be zone. Contains fewer sulfosalts and has a local overprint of chalcopyrite stringers.	At the base of the Contact Mudstone
HW	Pb–Zn–Cu	Massive, fine-grained stratabound sulphide lens dominated by pyrite, sphalerite, galena, and chalcopyrite (mainly as stringers). This zone has generally lower gold-silver grades and higher base metals relative to the 21 zones.	Within the Contact Mudstone but at a higher stratigraphic level than the 21 zone deposits
PMP	Fe–Zn–Pb–Cu	Veins of pyrite, sphalerite, galena, and tetrahedrite. Commonly banded; locally with colloform textures. Local zones of very fine-grained mineralization in rhyolite.	Discordant, within the rhyolite; spatially underlying the 21B zone
109	Au–Zn–Pb–Fe	Veins of quartz, sphalerite, galena, pyrite, and visible gold associated with silica flooding and fine-grained amorphous carbon. Underlies the north end of the 21B and HW zones.	Discordant, within the rhyolite
21E	Ag-Au-Sb	Fine-grained stratabound sulphide lens dominated by stibnite, pyrite, sphalerite, galena, and chalcopyrite. This zone has generally lower gold–silver grades relative to the 21 zones. Disseminated stibnite, pyrite, sphalerite, galena ± tetrahedite associated with veinlets.	Within hanging wall sediments at a higher stratigraphic level (similar to HW Zone but with higher antimony) than the 21 zone deposit and contact mudstone. Hosted with rhyolite breccia

7.3.1.2 21B Zone

The main body of mineralization, the 21B Zone, is a moderately, westward dipping, stratiform tabular body of gold–silver-rich mineralization about 700 m long, 200 m wide, and locally exceeding 20 m thick. Individual clastic sulphide beds range from 1–100 cm thick and become progressively thinner up sequence. Mineralization consists of beds of clastic sulphides and sulphosalts containing variable amounts of barite, rhyolite, and mudstone clasts. Imbricated, laminated mudstone rip-up clasts have been observed locally at the base of the clastic sulphide–sulfosalt beds, indicating turbiditic emplacement of some beds.

In the thickest part of the zone, pebble- to cobble-sized clasts occur in a northward-trending channel overlying the Eskay Rhyolite. The beds grade laterally over short distances into thinner, finer-grained, clastic beds and laminations.

Gold and silver occur as electrum and amalgam while silver mainly occurs within sulfosalts. Precious metal grades generally decrease proportionally with the decrease in total sulphides and sulfosalts. Clastic sulphide beds contain fragments of coarse-grained sphalerite, tetrahedrite, lead sulfosalts with lesser freibergite, galena, pyrite, electrum, amalgam, and minor arsenopyrite. Stibnite occurs locally in late veins, as a replacement of clastic sulphides, and appears to be confined to the central, thickest part of the deposit, suggesting a locus for late hydrothermal activity. Cinnabar is rare and is found associated with the most abundant accumulations of stibnite. Barite occurs as isolated clasts, in the matrix of bedded sulphides and sulfosalts, and also as rare clastic or massive accumulations of limited extent. Barite is more common towards the north end of the zone.

7.3.1.3 21C Zone

The 21C Zone is roughly 650 m long, 60–150 m wide and about 8–15 m thick. It is dominantly characterized by stratabound to stratiform barite-rich mineralization associated disseminated base and precious metal-rich mineralization in the rhyolite footwall. It occurs at the same stratigraphic horizon as the 21B Zone, but is located down-dip towards the west. The 21C is separated from the 21B zone by 40–60 m of barren Contact Mudstone.

Mineralization is associated with mottled barite-calcite ± tetrahedrite beds in and near the base of the Contact Mudstone. Precious metal grades are variable. Local areas of brecciation are infilled with sulphides including sphalerite, pyrite, galena, and tetrahedrite. Mineralization in the underlying footwall forms a cryptic, tabular body, sub concordant to stratigraphy. Aside from containing 1–2% very fine-grained pyrite and trace sphalerite, tetrahedrite, and galena, the rhyolite appears similar to adjacent unmineralized areas.

7.3.1.4 21Be Zone

Precious-metal mineralization near the north end of the 21B Zone extends over top of the anticline into a block bound by segments of the north–south-oriented Pumphouse Fault. Mineralization is hosted within a steeply-dipping, fault-bounded slab of mudstone approximately 60–100 m wide and 500 m long that is complexly folded and faulted.

While some of the mineralization within the 21Be Zone appears similar to the 21B Zone, the majority is steeply-dipping and dominated by fine-grained, massive sulfosalts that grade downward into massive pyrite. There is a direct correlation of sulfosalts with higher-grade precious metal concentrations. The silver:gold ratio for the zone is approximately 100 times greater than in the 21B Zone. Stringers of chalcopyrite and chalcopyrite–galena–sphalerite overprint the mineralization. Fine-grained pyrargyrite occurs locally in hairline fractures cutting the mudstone and hosts higher-grade mineralization. Many of the textures observed in this zone suggest that the sulphides were introduced by replacement processes, perhaps along early faults.

7.3.1.5 North Extension Zone (NEX) Zone

The approximately 600 m long and 150 m wide North Extension Zone is geometrically complicated by numerous faults that cut the nose of the Eskay Anticline. Textures, mineralogy, and precious-metal grades are somewhat variable and show similar characteristics to parts of the 21Be Zone and distal parts of the 21B Zone, suggesting synchronous deposition. Pyrite and chalcopyrite are more common whereas Sb-Hg bearing minerals are less common. Chalcopyrite occurs in stringers that overprint earlier clastic mineralization and may be related to the formation of the HW Zone. Much of the contained pyrite may also have been introduced during this later event.

7.3.2 Discordant Style Mineralization

Stockwork and discordant style mineralization at Eskay Creek are hosted in the rhyolite footwall within the PMP, 109, 21A\, 21B, 21C\, 21E and 22 Zones. The PMP Zone is characterized by pyrite, sphalerite, galena, and chalcopyrite-rich veins and veinlets hosted in strongly sericitized and chloritized rhyolite. The 109 Zone consists of gold-rich quartz veins with sphalerite, galena, pyrite, and chalcopyrite associated with abundant carbonaceous material hosted predominantly in siliceous rhyolite. The 21A, 21B and 21C Zones consist of very fine-grained cryptic pyrite with rare sphalerite and galena in sericitized rhyolite. The 22 Zone consists of cross-cutting arsenopyrite, stibnite and tetrahedrite veins hosted in massive to pyroclastic facies rhyolite.

Descriptions of the following discordant mineralized zones are modified after Roth et al. (1999).

7.3.2.1 HW Zone

The HW Zone forms a second massive sulphide horizon hosted in Mudstone, but at a stratigraphic level above the Contact Mudstone. Its geometry is disrupted by fault structures associated with the fold closure. Sulphides are typically fine-grained, finely banded, and consist of semi-massive to massive pyrite, sphalerite, galena, chalcopyrite, and tetrahedrite. Sphalerite is reddish-brown, suggesting a higher iron content compared to sphalerite encountered in other zones. The HW Zone has a higher base metal content compared to other zones, except where tetrahedrite ± sulfosalts are observed, which are associated with significantly higher precious metal grades.

7.3.2.2 PMP Zone

The PMP Zone is a discordant zone of diffuse vein and disseminated sulphide mineralization hosted in the rhyolite unit beneath the 21B Zone. Precious metal grades are generally lower than in other zones. Patchy sulphide mineralization is observed locally through the rhyolite in the form of veins containing pyrite, sphalerite, galena and lesser sulfosalts such as tetrahedrite. Chalcopyrite content increases with depth. Sphalerite is generally darker (more iron-rich) than in the overlying 21B Zone. Locally, areas of very fine-grained disseminated sulphide mineralization enriched in precious metals occur; these are similar to footwall hosted mineralization observed beneath the 21C Zone.

7.3.2.3 109 Zone

The zone is characterized by a distinct siliceous stockwork of crustiform quartz veins with coarse-grained sphalerite, galena, minor pyrite, and chalcopyrite. The 109 Zone is hosted entirely within the Eskay Rhyolite, beneath the north end of the 21B and the HW Zones. Gold and silver occur in electrum and sulfosalts.

7.3.2.4 22 Zone

The zone is characterized by localized zones of cryptic, disseminated, precious metal bearing mineralization associated with disseminated sulphides, including pyrite, sphalerite and galena; having an association with quartz microveinlets.

7.4 Prospects/Exploration Targets

Exploration potential is discussed in Section 9.

7.5 QP Comments on “Item 7: Geological Setting and Mineralization”

In the opinion of the QP, the understanding of the Eskay Creek deposit setting, lithologies, and geological, structural, and alteration controls on mineralization is sufficient to support estimation of Mineral Resources.

8 DEPOSIT TYPES

8.1 Overview

The Eskay Creek deposit is generally classified as an example of a high-grade, precious metals-rich epithermal volcanogenic massive sulphide (VMS) deposit; however, it has also been suggested to be an example of a subaqueous hot spring gold–silver deposit.

Table 8-1 summarizes the key features of each deposit type.

Features that would classify Eskay Creek as a VMS deposit (Roth et al., 1999) include:

- Formed on the seafloor in an active volcanic environment with a rhyolite footwall and basalt hanging wall;
- Chlorite–sericite alteration in the footwall and sulphide formation within a mudstone unit at the seafloor interface.

Unlike many VMS deposits, Eskay Creek has high concentrations of gold and silver, and an associated suite of antimony, mercury and arsenic. These mineralization features, along with the high incidence of clastic sulphides and sulfosalts, are more typical of an epithermal environment with low formation temperatures.

Features that would classify Eskay Creek as a subaqueous hot spring gold–silver deposit (Alldrick, 1995) include:

- Broad hydrothermal systems marked by widespread sericite–pyrite alteration;
- Evidence of a volcanic crater or caldera setting;
- Accumulations of felsic volcanic strata.

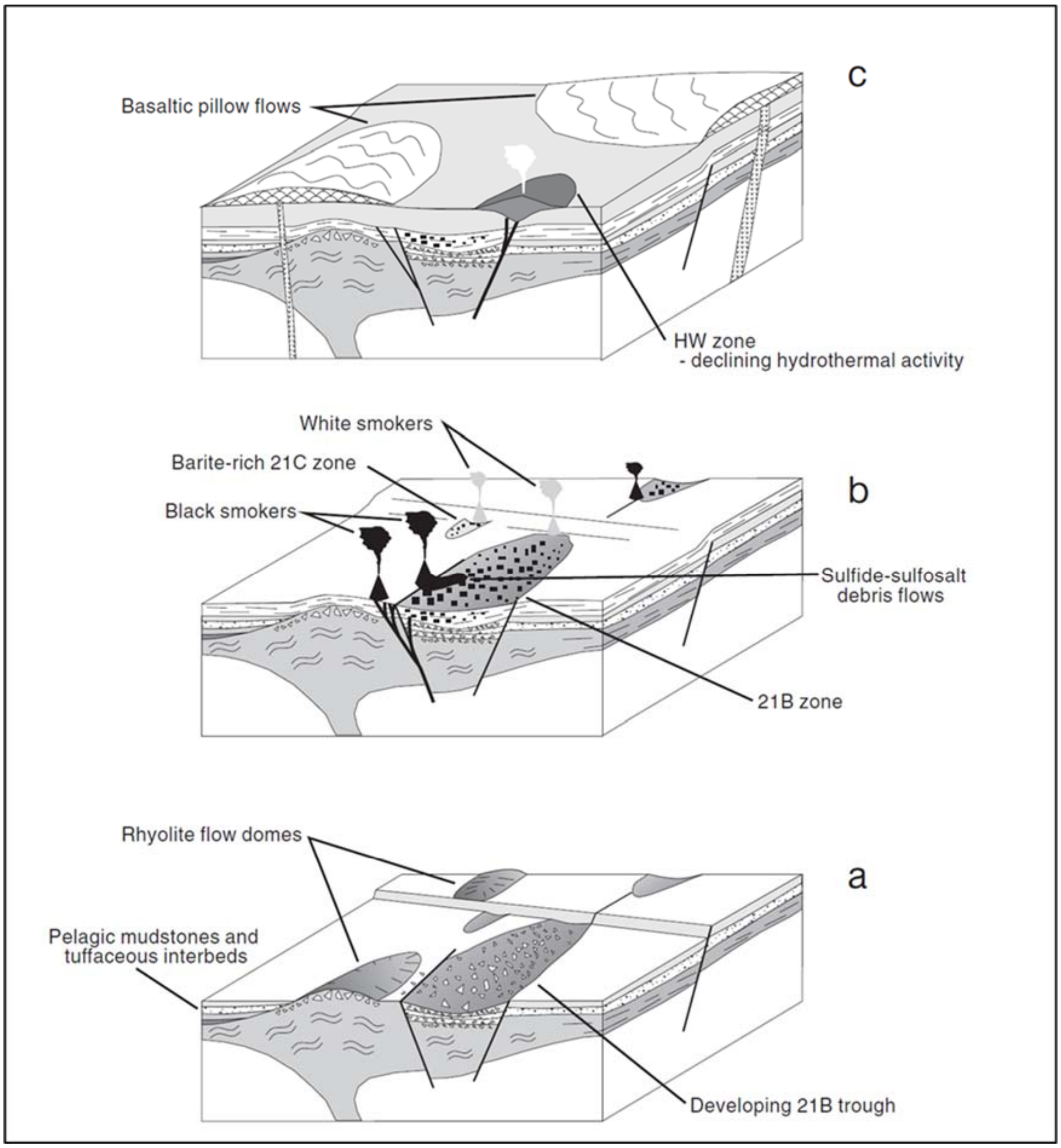
Roth et al., (1999) developed a deposit genesis model for the 21 Zones, that included the following phases

- Rifting, basin development and intrusion and extrusion of rhyolite flow domes. Coarse volcaniclastic debris from extrusive portions of the rhyolite domes are deposited along the developing 21B Zone trough (part a of Figure 8-1);
- Hydrothermal activity is focused through rift faults forming chimneys and mounds on the seafloor. Collapse or disruption of these mounds forms clastic sulphide-sulfosalt debris which is redeposited in the 21B Zone trough. Other smaller basins provide the sites for similar mineralization and barite-rich zones (21C Zone) related to white smokers (part b of Figure 8-1);

Table 8-1: Deposit Type Features

	VMS	Hot Spring Au–Ag
Tectonic setting:	Oceanic extensional environments, such back-arc basins, oceanic ridges close to continental margins, or rift basins in the early stages of continental separation. Form at, or near, the seafloor through the focused discharge of hot, metal-rich hydrothermal fluids.	Active volcanic arcs (both oceanic island arcs and continental margin arcs) are likely setting
Host/Associated rock types:	Terrigenous clastic rocks associated with marine volcanic rocks and sometimes carbonate rocks; these may overlie platformal carbonate or clastic rocks. Typically, a concordant sheet of massive sulfides up to a few meters thick and up to kilometers in strike length and down dip; can be stacked lenses	Mineralization hosted by intermediate to felsic flows and tuffs and minor intercalated sedimentary rocks. Pillow lavas, coarse epiclastic debris flows, and assorted subvolcanic feeder dikes are all part of the local stratigraphic package.
Deposit form:	Deposits typically comprise thin sheets of massive to well layered pyrrhotite, chalcopyrite, sphalerite, pyrite and minor galena within interlayered, terrigenous clastic rocks and calcalkaline basaltic to andesitic tuffs and flows. There is typically a mound-shaped to tabular, stratabound body composed principally of massive (>40%) sulphide, quartz and subordinate phyllosilicates, and iron oxide minerals and altered silicate wall-rock. These stratabound bodies are typically underlain by discordant to semiconcordant stockwork veins and disseminated sulphides. The stockwork vein systems, or “pipes”, are enveloped in distinctive alteration halos, which may extend into the hanging-wall strata above the VMS deposit.	Highly variable. Footwall stockwork or stringer-style vein networks. Large, textureless massive sulphide pods, finely laminated stratiform sulphide layers and lenses, reworked clastic sulphide sedimentary beds, and epithermal-style breccia veins with large vugs, coarse sulphides and chalcedonic silica. All types may coexist in a single deposit.
Ore mineralogy (principal and subordinate):	Pyrite, pyrrhotite, chalcopyrite, sphalerite, cobaltite, magnetite, galena, bornite, tetrahedrite, cubanite, stannite, molybdenite, arsenopyrite, marcasite	Sphalerite, tetrahedrite, boulangerite, bournonite, native gold, native silver, amalgam, galena, chalcopyrite, enargite, pyrite, stibnite, realgar, arsenopyrite orpiment; metallic arsenic, Hg-wurtzite, cinnabar, aktashite, unnamed Ag-Pb-As-S minerals, jordanite, wurtzite, krennerite, coloradoite, marcasite, magnetite, scorodite, jarosite, limonite, anglesite, native sulphur.
Gangue mineralogy (principal and subordinate):	Quartz, calcite, ankerite, siderite, albite, tourmaline, graphite, biotite	Magnesian chlorite, muscovite (sericite), chalcedonic silica, amorphous silica, calcite, dolomite, pyrobitumen, gypsum, barite, potassium feldspar, alunite with minor carbon, graphite, halite and cristobalite.

Figure 8-1: Genetic Model



Note: Figure from Roth et al., (1999).

- The HW zone of massive sulphide forms higher in the mudstone stratigraphy and basaltic magmatism begins (dykes and flows) during the waning stages of hydrothermal activity (part c of Figure 8-1).

8.2 QP Comments on "Item 8: Deposit Types"

The QP is of the opinion that exploration programs that use either a VMS or a hot-spring deposit model are applicable to the Project area.

9 EXPLORATION

9.1 Grids and Surveys

McElhanney Consulting Services Ltd. (McElhanney) of Vancouver, B.C. flew an airborne light detection and ranging (LiDAR) and photo acquisition survey in December 2018. The resulting topography map was compiled to 1 m accuracy.

LiDAR and photo acquisition were collected simultaneously with equipment co-mounted on the same aircraft. Sixty flight lines comprising 539-line kilometers were completed, covering the 100 km^2 survey area. Post-processing of the acquired data was completed in McElhanney's Vancouver office.

9.2 Exploration Potential

There is significant remaining exploration potential in the Eskay Creek deposit and environs.

Skeena considers that well-defined, mineralized syn-volcanic feeder structures that propagate through the volcanic pile have not been sufficiently explored at depth and along strike. Examples of this well-documented mineralization style include the 22 Zone, Water Tower Zone, 21C Zone and HW Zone.

In addition, the largely unexplored Lower Mudstone that is situated ~100 m stratigraphically below the more well-known Contact Mudstone, represents a second exhalative event horizon and has geological and mineralogical similarities to the Contact Mudstone-hosted mineralisation. Exploratory target ranking will be influenced by areas where known synvolcanic feeder structures intersect the Lower Mudstone, as these locales will offer the highest potential for development of additional exhalative mineralization.

The 22 Zone has potential to be expanded as this body of rhyolite-hosted feeder mineralization remains open for drill expansion along strike as well as at depth. Due to limited legacy exploratory drilling in the area between the 21A and 22 Zones, additional opportunities exist to discover and delineate near-surface, rhyolite-hosted feeder mineralization.

9.3 QP Comments on "Item 9: Exploration"

The exploration programs completed to date are appropriate to the style of the deposit and prospects. Additional exploration has a likelihood of generating further exploration successes.

10 DRILLING

10.1 Introduction

Data collected prior to Skeena's project interest is referred to as legacy data.

Legacy drilling consists of 1522 surface diamond drill holes totalling 342,119 m and 6,061 underground drill holes totalling 309,213 m. Since 2018, Skeena has drilled 255 surface core drill holes totalling 22,004.72 m. Drilling is summarized in Table 10-1 and shown in Figure 10-1 and Figure 10-2. Note that all of the underground drilling is legacy.

The underground areas are drilled at an average spacing of 10 m using BGM (~40 mm) core diameters. In highly complex areas where mining was active, drill spacing was locally reduced to 5 m. Underground drill holes are generally less than 100 m in length.

10.2 Drill Methods

Limited details are available regarding drilling contractors and drilling procedures specific to each campaign prior to 1995. Table 10-2 summarizes the known drill contractors and methods.

10.3 Logging Procedures

10.3.1 Legacy

Limited information is available for procedures used during the exploration programs carried out before 2004. The drill core was logged using DLOG computer programs for data entry as well as for drill log printing. The data were entered directly into laptop computers and the rock units coded with four-digit geology codes. Mineralized sections were logged separately as nested units or primary units depending on quantities. Textural descriptions, rock colour and structure were also coded with two-character fields. Remarks were typed into separate fields to characterize unique geology, structure or mineralization features.

All collar and survey information were tabulated in master files within the DLOG computer program. Completed logs were printed and the information was exported into ACAD and Vulcan software to facilitate plotting drill hole location maps and cross-sections.

Table 10-1: Drill Summary Table

Year	Operator	Prospect/ Zone	Number of Holes	Drill Hole ID	Metres Drilled
1932–1934	Unuk Gold/Unuk Valley Gold		11	Unuk 1-11	261.21
1935–1938	Premier Gold Mining Co. Ltd.		38	P 12-49	1,825.95
1964	Stikine Silver/Canex Aerial Exploration Ltd.	Emma Adit	6	C-1 to C-6	224.64
1965	Stikine Silver Ltd.	Emma Adit	3	?	15.85
1973	Kalco Valley Mines Ltd.	22 Zone	7	KV-1 to KV-7	299.62
1975–1976	Texasgulf Canada Ltd.	#5 O.C. #6 O.C.	7	K76-1 to K76-7	373.38
1980–1982	Ryan Exploration Ltd. (U.S. Borax)	22 Zone 6 Zone	7	MR-1 to MR-7	452.32
1985	Kerrisdale Resources Ltd.		5	KDL 85-1 to 85-5	622.1
1988	Calpine/Consolidated Stikine	21A 21B	16	CA88-01 to CA88-16	2,875.50
1989	Calpine/Consolidated Stikine	21A 21B 22 Zone	179	CA 89-17 to CA 89-196	43,017.90
				CA 89-198 to CA 89-205	
			7	CA 8922-01 to CA 8922-07	1,321.00
1990	Calpine/Consolidated Stikine	21B 21C	513	CA 90-197	115,272.26
		PMP		CA 90-206 to CA 90-691	
		Mack		MK 90-01 to MK 90-04	
		Proposed mill site		PMS 90-01 to PMS 90-06	
				KP-1 to KP-16	
			3	CA 90-692, 693, 696	1,036.60
		GNC	19	GNC 90-01 to GNC 90-19	3,318.00
		Adrian	35	AD 90-01 to AD 90-35	21,786.00
1991	International Corona	21B	12	C 91-700 to C 91-711	2,791.00
		GNC	5	GNC 91-20 to GNC 91-24	
1992	International Corona	21B	1	C 92-712	3,342.00
		GNC	7	GNC 92-25 to GNC 92-31	
1993	Homestake	21B	2	C 93-713- to C 93-714	1,606.60
		GNC	3	GNC 93-32 to GNC 93-34	
1994	Homestake	Adrian	6	AD 94-35 to AD 94-40	3,531.70
		21B	5	KP 94-1 to KP 94-5	549.25
1995	Homestake	21B NEX	21	C 95-715 to C 95-735 (formerly labelled NEX 95-1 to 18 and QZ 95-1 to 3)	3,468.10
		Bonsai	5	BZ 95-1 to BZ 95-5	

Year	Operator	Prospect/ Zone	Number of Holes	Drill Hole ID	Metres Drilled
1996	Homestake	21B NEX HW	94	C 96-736 to C 96-829	21,280.80
		Adrian	19	AD 96-41 to AD 96-59	
		Bonsai	1	BZ 96-06	
1997	Homestake	21B 21C 21E	42	C 97-830 to C 97-871	16,220.47
		Adrian	14	AD 97-60 to AD 97-73	
		GNC	1	GNC 97-30X	
		Mack Star	2	MP 97-01 to MP 97-02	
1998	Homestake	Core Property	79	C 98-872 to C 98-950	21,909.63
		GNC	2	GNC 98-35 to GNC 98-36	
		Mack	8	MP 98-03 to MP 98-09	
		Star	1	SP 98-01	
1999	Homestake	Core Property	64	C 99-951 to C 99-1014	17,363.96
2000	Homestake	Core Property	77	C001012W C001015 to C001088	25,893.93
2001	Homestake	22 Zone 21C Zone	61	C011089 to C011145	22,035.48
2002	Barrick	21C Zone 21A Zone Deep Adrian	47	C02-1146 to C02-1178 C02-920X, C02-975X	15,115.69
2003	Barrick	22 Zone 21A Zone 21C Zone	71	C03-1179 to C03-1245 C03-919X	18,323.28
2004	Barrick	22 Zone Ridge Block 21C/21E Deep Adrian	55	C04-1261 to C04-1298	18,404.88
				C04-1020X, C04-1196X	
				C04-1206X	
				5702, 6461, 6464	
2018	Skeena	21A Zone 21C Zone 22 Zone	46	SK-18-001 to SK-18-036	7,737.45
				SK-18-037 to SK-18-040	
				SK-18-42 and SK-18-43	
				SK-18-048 to SK-18-051	
2019	Skeena	21A Zone 21E Zone HW Zone	209	SK-19-052 to SK-19-247	14,267.27

Figure 10-1: Surface Drill Hole Location Plan

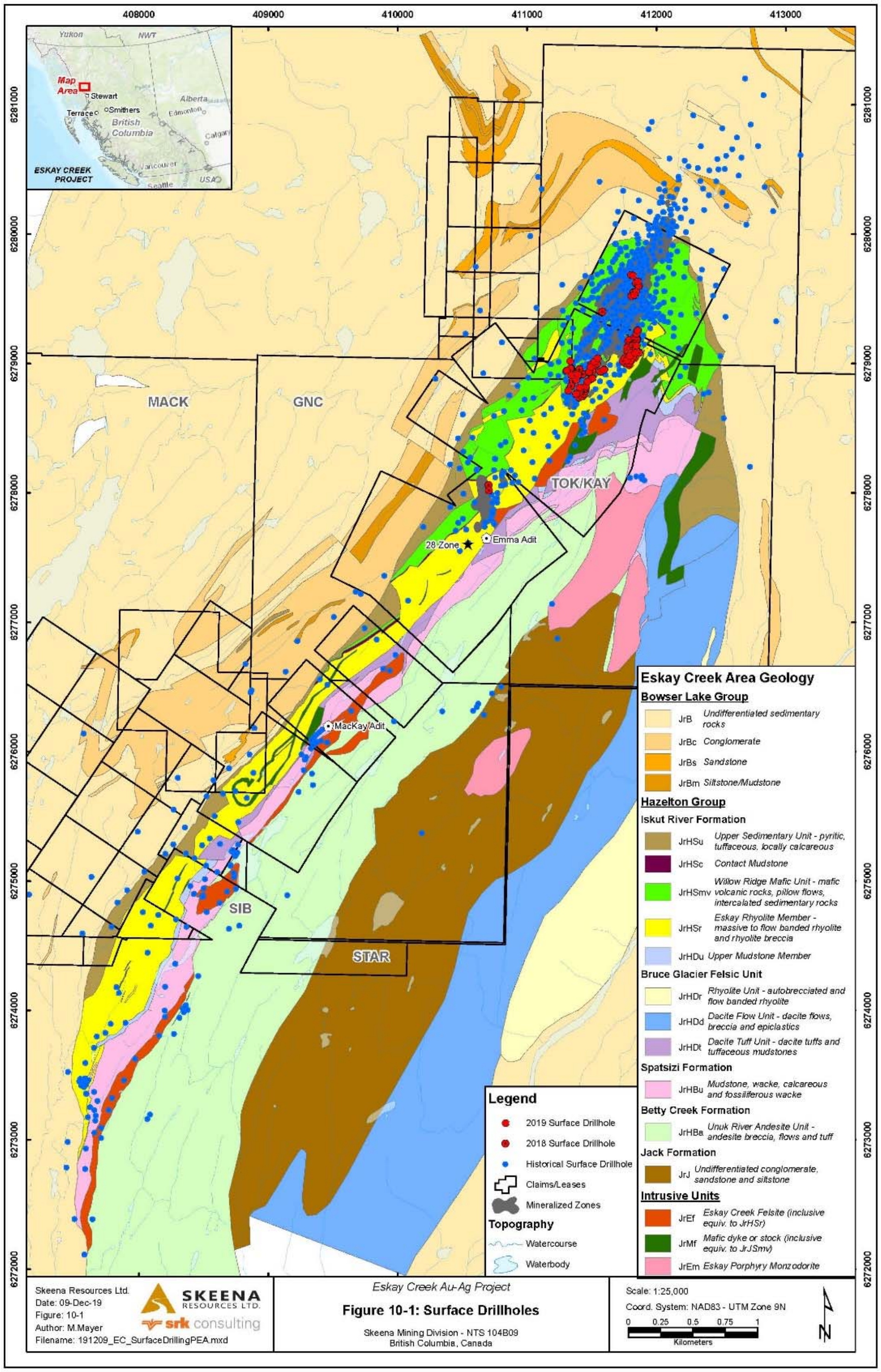


Figure 10-2: Underground Drill Hole Location Plan

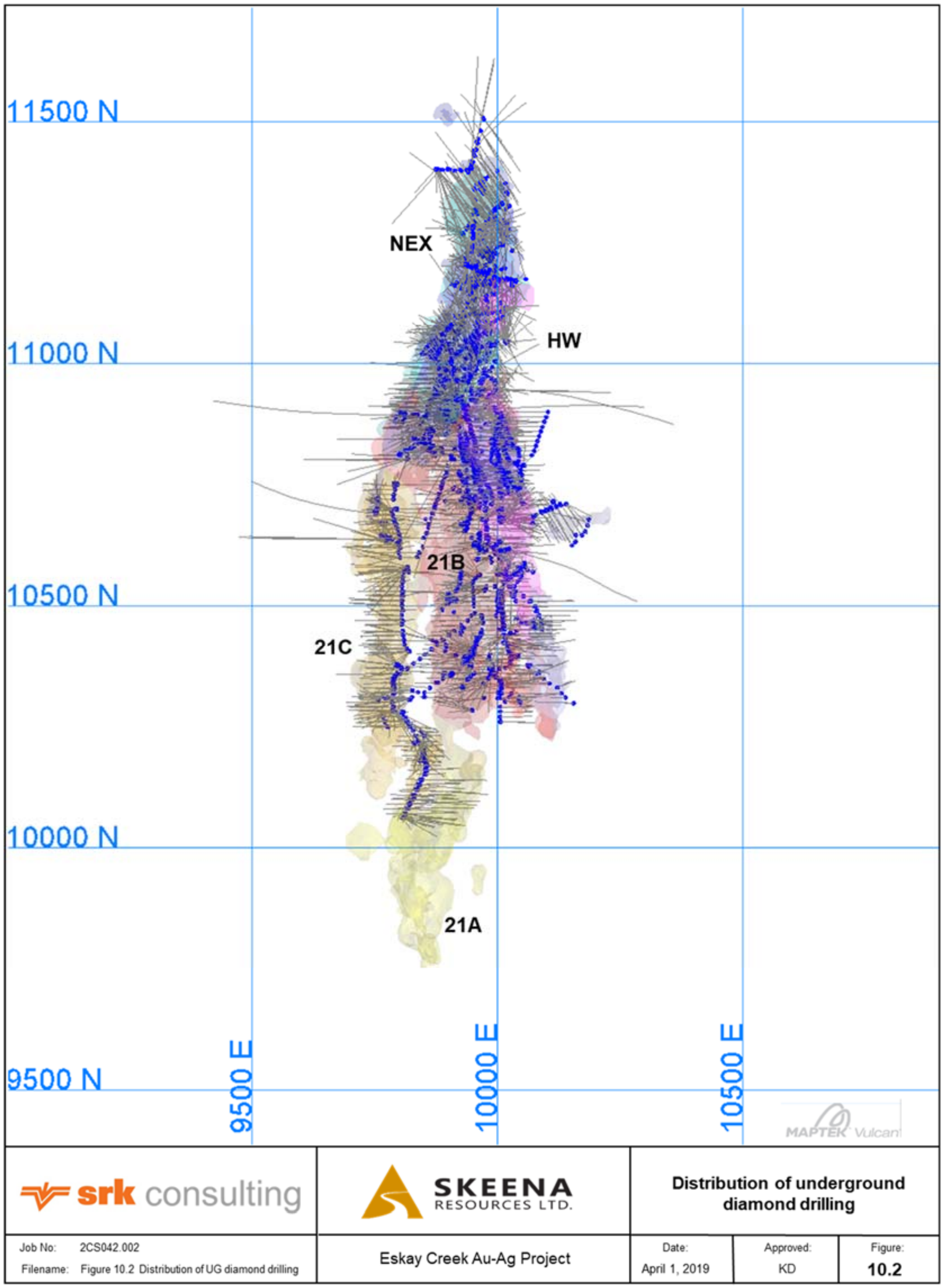


Table 10-2: Drill Contractors and Methods

Year	Contractor	Rig Type	Core Size and Core Diameters
1996	Advanced Drilling of Vancouver	Boyles 56	
1996–1997	Hy-Tech Drilling of Smithers, B.C. (Hy-Tech)	JKS-300 F-15	BQTK (40.7 mm) NQTK (50.6 mm) NQ2 (50.6 mm)
1998	Hy-Tech	JKS-300 F-15	BQTK (40.7 mm) BQTK (40.7 mm)
2004	Hy-Tech	JKS-300 F-15	BQTK (40.7 mm) BQTK (40.7 mm)
2018	Dmac Drilling Ltd. of Aldergrove, B.C. Hy-Tech	Hydracore 2000 Tech 5000	NQ2 (50.6 mm) NQ (47.6 mm)
2019	Tahltech Drilling Services Ltd.	Hydracore 2000	NQ (47.6 mm)

As part of the drill core processing procedures, all core was geotechnically logged. Two parameters were routinely measured and recorded:

- Core recovery (the percentage of drill core recovered in every 3.05 m run);
- Rock quality designation (RQD; the percentage of core within a run exceeding 10 cm in length.

10.3.2 Skeena

During the 2018 program, Skeena initially undertook logging and sampling at core logging facilities located just inside the Eskay Creek Mine site gate, proximal to Argillite Creek. However, as winter set in, core logging and sampling was moved to Colorado Resources' core facilities located at the McLymont Creek staging area in the Iskut Valley.

During the 2019 program, Skeena undertook all logging and sampling exclusively at a core shack set up at the McLymont staging area.

Core is geologically logged for lithology, alteration, veining, mineralization and structural features. Geotechnical data such as recovery, RQD, longest stick, and magnetic susceptibility are recorded. Skeena records geological and geotechnical information into a GeoSpark database.

Core is photographed wet.

10.4 Recovery

Skeena currently does not have access to the legacy RQD and recovery data.

Drilling undertaken by Skeena in 2018 and 2019 had excellent core recoveries, with core recovery averaging 92% over both programs.

10.5 Collar Surveys

10.5.1 Surface Drilling

10.5.1.1 Legacy

In September 2004, McElhanney was contracted to carry out a Real-Time Kinematic global positioning system (GPS) instrument survey to tie in 205 drill holes in addition to 12 mineral claim posts at the Eskay Creek Mine site. Collars from 205 surface core drill holes drilled during 2001–2004 were surveyed using a Lecia SR530 dual frequency GPS.

10.5.1.2 Skeena

During the Skeena programs, surface drill hole collars were initially located using hand-held GPS units and surveyed at the end of the drill program using a Trimble differential GPS (DGPS).

10.5.2 Underground Drilling

10.5.2.1 Legacy

Collar location surveys were performed by the mine surveyors. These provided accurate collar locations for the holes, and a check on the initial azimuth and dip was recorded for each hole.

10.5.2.2 Skeena

Skeena has not performed any underground drilling to date.

10.6 Downhole Surveys

10.6.1 Surface Drilling

10.6.1.1 Legacy

There is no documentation available to Skeena on the downhole survey procedures that may have been used during legacy drill programs.

10.6.1.2 Skeena

During the Skeena programs, down hole orientation surveys for surface drill holes were taken approximately every 30 m down the hole using a multi-shot Reflex orientation tool.

10.6.2 Underground Drilling

10.6.2.1 Legacy

Prior to 2004, most of the drill holes in the database were surveyed downhole using a Sperry Sun Single Shot instrument, with readings taken every 60 m, or by acid tubes, with readings every 30 m. In early 2004, downhole surveying used an Icefield Tools M13 instrument. This provided azimuths and dips for each hole every 3 m down the hole. Readings were reviewed by staff and inaccurate entries were removed from the database.

10.7 Sample Length/True Thickness

Drill hole spacing throughout the deposit varies from 5 m, where underground production drilling encountered complex areas, to 25 m at the surface. The average drill hole spacing is approximately 10–15 m throughout the deposit.

For surface drill holes, mineralisation true width approximates 80–100% of drilled width; for underground drill holes positioned on single platforms and drilled in radiating fans, true drilling widths are more variable.

10.8 Drilling Completed Since Database Close-out Date

Skeena has embarked on a major drilling program to provide additional information within the area of the proposed open pit, support potential upgrades of blocks currently classified as Inferred to higher-confidence categories in the 21A, 21E and HW Zones, and test for potential mineralization extents. During 2019 a total of 209 drill holes (14,267.27 m) had been completed. Assay results remain pending for a number of the drill holes.

Although a few of the newer drill holes are high-grade and may change the grades locally, those drill holes that are within the existing model should have no material effect on the overall tonnages and average grade of the current Mineral Resource. The drill spacing is sufficient to support some confidence category upgrades for blocks that have been classified as Inferred in the block model that supports the resource estimate in Section 14.

In addition, some of the recent holes have intersected high-grades outside of the existing mineralization model. Highlights of the 2019 drilling include:

- The western lobe of the 21A Zone was interpreted to be exclusively hosted in the tabular Contact Mudstone. The new drill campaign indicates the presence of a previously-unknown hydrothermal vent associated with gold–silver mineralization in this location;
- Gold–silver mineralization, hosted in the Lower Mudstone, was encountered 100 m stratigraphically below the Contact Mudstone. This horizon had not been previously expected to host higher-grade mineralized zones;
- The HW Zone occurs approximately 20 m stratigraphically above the Contact Mudstone which was host to the 21B Zone deposits at Eskay Creek. Drill data to date indicates the presence of gold–silver mineralization with higher grades and interval thicknesses than predicted from legacy data.

There is potential that drilling in these areas may support additional tonnage and grade estimates when incorporated into the block model.

10.9 QP Comments on “Item 10: Drilling”

The QP considers that the quantity and quality of the logging, geotechnical, collar and downhole survey data collected in the exploration and infill drill programs are sufficient to support Mineral Resource estimation. Drill orientations are generally appropriate for the mineralization style and have been drilled at orientations that are optimal for the orientation of mineralization for the bulk of the deposit areas. No factors were identified with the data collection from the drill programs that could significantly affect Mineral Resource or Mineral Reserve estimation.

11 SAMPLE PREPARATION, ANALYSES, AND SECURITY

Data collected prior to Skeena's project interest is referred to as legacy data.

11.1 Sampling Methods

11.1.1 Legacy

Historically, sampling at Eskay Creek was selective and primarily based on visual estimations of sulphide percent. All sample intervals sent to the laboratory were tested for gold and silver; however, lead, copper, zinc, mercury, antimony and arsenic were inconsistently sampled from one drilling campaign to the next. For underground drilling, lead, copper, zinc, mercury, antimony and arsenic were assayed when samples exceeded 8 g/t gold equivalent (AuEq, where AuEq equaled Au + (Ag/68)) (Barrick, 2005).

11.1.1.1 2003 and Prior

During the drill core logging process, portions of the core were selected for sampling based on lithology, mineralization, and alteration. Sample intervals varied from about 0.25 m up to 1.5 m though the optimum sample interval was 1.0 m. Sample intervals were always contained within one geological unit and did not straddle contacts. Assay tags were used for sample identification and were inserted at the end of each sample interval.

After the logging and photography had been completed, the core was sampled by means of splitting the core with a manual or pneumatic splitter or by cutting the core with a diamond bladed rock saw in the case of the massive sulphide zones. One half of the core was placed in plastic sample bags and sealed for shipment to the lab and the other half of the core was returned to the core box and then trucked to an unused gravel pit at km 45 for long-term storage; this storage area was turned into a logging facility for the 1998 drilling campaign.

Sample bags containing core for analysis were either carried to the mine assay laboratory that was located adjacent to the logging facilities or packed in rice bags/plastic pails for shipment via truck to Independent Plasma Laboratories (IPL) for analysis.

During 1996 and 1997, most of the drill core was processed at the core logging facilities located at the Eskay Creek mine site. However, during the 1998 drill campaign the drill core was processed at the core logging facilities located either at the Eskay Creek mine site or at Camp 45, an exploration site situated 45 km along the Eskay Creek Mine Road.

11.1.1.2 2004

Core was sampled at 1.0 m intervals, but smaller increments were applied where necessary to honour geological contacts.

Photographing of all diamond drill core using a digital camera was initiated in 2004. All core drilled for the mine geology department was either consumed during sampling or discarded once it had been logged. Skeena was unable to find photographic evidence of any of the core.

Holes drilled for the Regional Exploration group were shipped to the exploration camp. This camp has now been dismantled and all core was disposed of in Albino Lake, 9 km from the Eskay Creek mine site.

Production samples were also collected daily from each face. Representative geological contacts were identified, and these chip samples were analyzed for gold, silver, mercury and antimony.

11.1.2 Skeena

Skeena geologists mark the centre line of the core in red china marker in preparation for core cutting. All drill core is halved with a diamond core cutting saw.

One-metre assay intervals are established when visible mineralization is first observed, and then uniform intervals are continued down the drill length until there is no evidence of mineralization. Assay intervals honour geological contacts to a minimum of 0.5 m and a maximum of 1.5 m.

One assay sample ticket stub is placed into the sample bag with the half core and the other matching ticket stub is stapled into position onto the core box marking the appropriate assay interval.

Groups of samples are placed in a large rice bag and secured with tie wraps; high-grade samples are separated into batches to ensure that the appropriate method is applied at the laboratory. The sample number series within the sack are marked on the outside of the rice bag and a laboratory sample submission form is placed in the first rice bag in sequence. The laboratory is emailed in advance of the shipment, and when the laboratory receives the shipment a confirmation email is returned. Assay sample shipments are shipped to the assay facility in Kamloops twice weekly.

Samples are transported by truck from the Eskay mine site to the McLymont staging area by Skeena personnel and then loaded onto trucks driven by Rugged Edge Holdings (Skeena's expediter). The samples are delivered to Bandstra in Smithers and transported to the ALS preparation laboratory in Kamloops.

11.2 Density Determinations

11.2.1 Legacy

Specific gravity (SG) measurements were collected from diamond drill core in 1996 (250 measurements from 20 drill holes) and 1997 (84 measurements from seven drill holes). Sections of drill core up to 10 cm long of split or whole core were used to determine the SG. The core was first weighed in air on a beam balance, and then weighed in water. One or more measurements were taken from each sample interval.

SG models were subsequently created using a formula that was derived experimentally based on comparisons between actual measurements and analyses at Eskay Creek. This formula was utilized for all Mineral Reserves estimated on site in the mine’s history so that SG could be determined for mineralized intervals that did not have directly-measured values.

$$SG = (Pb + Zn + Cu) \times 0.03491 + 2.67$$

Where all metals are reported in percent.

A default value of 2.67 was applied to samples for which base metals were not reported. This is the average value of unmineralized rhyolite and mudstone host rocks combined.

The measured SG values from the early drill programs were primarily from relatively low base metal, 21B-style mineralization. The formula is therefore likely biased on the low side for rocks with higher base metal content.

11.2.2 Skeena

Specific gravity samples are collected one in every 20 m down the hole. A whole piece of NQ-sized competent core 10–15 cm in length is selected and measured using the water displacement method.

11.3 Analytical and Test Laboratories

11.3.1 Legacy

11.3.1.1 Surface

Surface holes drilled during the 1995 to 1998 drilling programs were sent to IPL in Vancouver for sample preparation and analysis. IPL was an independent laboratory; however, the accreditation of the facility at the time is unknown.

In 2002, surface drill holes were sent to Bondar Clegg in Vancouver for sample preparation and analysis. Bondar Clegg was an ISO 9002 accredited facility and an independent laboratory.

Sample preparation and analysis was undertaken at the Acme facility in Vancouver during 2002–2004. Acme held ISO accreditation IS0 9001:2000 for selected analytical methods and was an independent laboratory.

11.3.1.2 Underground

The Eskay Creek mine site laboratory was not independent, or an accredited facility. The majority of the underground drilling was prepared and analysed at the Eskay laboratory.

11.3.2 Skeena

Sample preparation was undertaken at the ALS sample preparation facility in Kamloops (ALS Kamloops) and sample analysis was completed at the ALS facility in Vancouver (ALS Vancouver), both of which are ISO 17025 accredited and independent of Skeena.

SGS Canada, located in Burnaby, BC, was used to independently test pulp duplicates and a select number of standards. SGS Canada holds ISO 17025 accreditation.

11.4 Sample Preparation and Analysis

11.4.1 Legacy Surface

11.4.1.1 2000 and Prior

At IPL, all drill core samples were crushed to -10 mesh, riffle split and 250 g pulverized to -15 mesh. Gold was assayed by fire assay (30 g) with an atomic absorption (AA) finish. All gold values >1.00 g/t were re-assayed by fire assay (30 g) and finished gravimetrically. Silver was assayed by fire assay (30 g) with an AA finish. Every batch of 24 assays consisted of 22 samples, one internal standard or blank and a random re-weigh of one of the samples.

Analysis for lead, zinc, copper, arsenic and antimony was done by an ore grade assay method using a 0.50 g sample digested in a dilute aqua regia solution. These elements were analyzed for using AA. Calibration was undertaken using three known standards and a blank. Internal quality control consisted of insertion of standards and blanks, and re-analysis.

Mercury analysis consisted of an aqua regia digestion and inductively-coupled plasma (ICP) finish.

At the Eskay Creek mine assay laboratory, the drill core was jaw-crushed to -⅛ inch, riffle split and 250–300 g was pulverized. Gold was assayed by fire assay (10 g) with an AA finish. Every batch of 24 samples included two duplicate assay checks.

For analysis for zinc, antimony, copper, and lead, a 0.20g sample was digested in a heated solution of tartaric, nitric, perchloric and hydrochloric acids, and finished by AA. For mercury and arsenic, a 1.00 g sample was digested in a heated solution of nitric, perchloric and hydrochloric acids and finished by AA.

11.4.1.2 2001

Surface drilling in 2001 was sent to Bondar Clegg in North Vancouver. All samples were crushed to -10 mesh, rifle split, and 250 g was pulversized to -150 mesh. Gold was analyzed by fire assay (30 g) with an AA finish.

Analysis for silver, lead, zinc, copper, arsenic and antimony was determined from a 0.5 g sample digested in aqua regia and analysed by ICP atomic emission spectroscopy (AES).

Mercury was digested in aqua regia and analysed by cold vapour AAS.

Internal quality control consisted of insertion of control standards, blanks and duplicate data.

11.4.1.3 2002 to 2004

Surface drilling during 2002 to 2004 was sent to Acme Laboratories is Vancouver. All samples were crushed to -10 mesh and a 250g riffle split is then pulverized to -150 mesh in a mild-steel ring-and-puck mill.

Analysis for gold was by fire assay (30 g) with an ICP mass spectrometry (MS) finish. Gold values in excess of 30g/t were reassayed by fire assay (30 g) with a gravimetric finish. Silver was digested in aqua regia and analysed by ICP-MS. Silver values in excess of 300 g/t was reported from the fire assay (30 g).

Analysis for lead, zinc, copper, arsenic, mercury and antimony was done using a 0.5g sample digested by a dilute aqua regia solution. The elements were then analysed by either ICP-MS or ICP emission spectroscopy (ES).

Every batch consisted of 34 samples and included a sample-prep blank, a pulp duplicate, a -10mesh reject duplicate, two blanks and one internal Standard Reference Material.

11.4.2 Legacy Underground

Underground samples were sent and processed at the Eskay Creek mine laboratory. Core was submitted whole to the Eskay mine assay laboratory for gold and silver determination by fire assay. Samples reporting >8 g/t AuEq, using the following formula:

- AuEq = Au + (Ag ÷ 68)

were also analyzed for lead, zinc, copper, mercury and arsenic.

At the Eskay Creek mine assay laboratory, the drill core was jaw-crushed to -⅛ inch, riffle split and 250–300 g was pulverized. Gold was assayed by fire assay (10 g) with an AA finish. Every batch of 24 samples included two duplicate assay checks.

For analysis for zinc, antimony, copper, and lead, a 0.20g sample was digested in a heated solution of tartaric, nitric, perchloric and hydrochloric acids, and finished by AA. For mercury and arsenic, a 1.00 g sample was digested in a heated solution of nitric, perchloric and hydrochloric acids and finished by AA.

11.4.3 Skeena

All samples were initially sent and prepared at ALS Kamloops after which the pulp samples were split and shipped for analysis to ALS Vancouver.

The entire sample was dried and crushed using a two-stage Terminator crusher. Crushing was done to better than 70% passing a 2 mm Tyler 10 mesh screen. Approximately 1000 g of the crushed material was taken and pulverized to better than 85% passing a 75 µm Tyler 200 mesh screen (PREP-31BN).

Gold assays were performed on 50 g samples by fire assay and atomic absorption (ALS code: Au-AA26) with a lower and upper detection limit of 0.01 g/t and 100 g/t, respectively. For assays above the upper detection limit then samples were analysed by fire assay with a gravimetric finish (ALS code: Au-GRA22) with lower and upper detection limits of 0.05 g/t and 10,000 g/t Au, respectively.

Silver assays were performed on 50 g samples by fire assay and gravimetric finish (ALS code: Ag-GRA22) with lower and upper detection limits of 5 g/t and 10,000 g/t, respectively. For assays above the upper detection limit, a concentrate and bullion grade fire assay and gravimetric finish were performed (ALS code: Ag-CON01) with lower and upper detection limits of 0.7 g/t Ag and 995,000 g/t Ag, respectively.

Multi-element assays were performed using a combination of digest and finish methods: a 0.25 g sample using a four-acid digest followed by an ICP atomic emission spectroscopy (AES) finish (ALS code: ME-ICP61), and a 0.1 g sample using lithium borate fusion followed by an ICP-MS finish (ALS code: ME-MS81). This combination in assay methods for the multi-elements ensured that the range of concentrations for all elements of interest, particularly for antimony, were covered. In the Skeena database, the ICP-AES finish method took precedence.

A limited number of samples exceeded the upper limits for silver, arsenic, copper, lead and zinc. For these samples, the laboratory was instructed to apply overlimit methods on a 0.4 g sample (ALS code: OG62) using a four-acid digest and ICP or AAS finish. Sulphur overlimits were re-analyzed using the total sulphur Leco furnace method using a 0.1 g sample (ALS code: S-IR08) with a lower detection limit of 0.01% and upper detection limit of 50%.

Mercury was separately analysed using low temperature aqua regia digestion followed by an ICP-AES finish (ALSO code: Hg-ICP42) with a lower detection limit of 1 ppm and an upper detection limit of 100,000 ppm.

11.5 Quality Assurance and Quality Control

11.5.1 Legacy

A summary of the known quality assurance or quality control (QA/QC) protocols is provided in Table 11-1. The table also notes where analytical certificates are available.

11.5.1.1 2002 and Prior

Eskay Creek mine initiated QA/QC measures into their sample stream in 1997. With progressive years the QA/QC protocol became more comprehensive and detailed.

Prior to 2002, there was no formal QA/QC program in place; however, the Eskay Creek mine laboratory and IPL were regularly monitored using pulp duplicates.

In 1998 a series of blanks were inserted into the Eskay mine laboratory assaying procedure. Some anomalous background values were observed; however, the source of the blank material has not been documented. Field duplicates initially tested at the Eskay mine laboratory were sent to IPL for independent checking. There was good agreement between the original sample and field duplicate for gold and silver, as well as the base and deleterious elements. Pulp duplicates were also assessed within the Eskay mine laboratory as well as sent to IPL for an independent check. The data mostly indicate a high correlation between the original and the duplicate assays.

No Eskay mine laboratory pulp repeats were documented in 2002. The surface drill hole samples were, however, being routinely sent to Acme for processing. Acme inserted three of their own in-house standards: DS3, DS4 and DS4 into the sample stream. Acme in-house pulp repeats were also routinely completed and monitored.

11.5.1.2 2003

In 2003, the Eskay mine laboratory started to implement QA/QC procedures into the sampling process. Control blanks and SRMs were added to the sample stream, but no record of the type, acceptable value and standard deviation of the control samples submitted have been found.

Acme inserted their own in-house SRMs, blanks and pulp repeats into the sample stream. Acme also routinely used preparation, pulp and reject duplicates.

11.5.1.3 2004

An official QA/QC program was undertaken in 2004 whereby the Eskay Creek exploration team added SRMs, blanks and field duplicates to the sample stream and submitted them to Acme for checking.

Five in-house assay SRMs were manufactured by ALS Chemex using material collected from the Eskay Creek Mine (Barrick, 2005). The acceptable values were certified through round-robin analyses at six different laboratories and statistically evaluated by the previous operator's Chief Geochemist. One in every 50 drill core samples was a QA/QC SRM.

Blanks were sourced from barren rocks found regionally around the mine. One in every 50 drill core samples was a QA/QC blank.

Table 11-1: Legacy QA/QC Protocols

Year	Laboratory	Type(s)	Certificate Availability
1997	Eskay mine laboratory	Repeat (pulp?)	No certificates found
1998	Eskay mine laboratory Bondar Clegg IPL MIN-EN (used only for round-robin) ALS Chemex (used only for round-robin)	Round robin standard reference materials, blanks, field and pulp duplicates	No certificates found
1999	Eskay mine laboratory	Pulp repeats	Certificates found
2001	Eskay mine laboratory	Pulp repeats	Certificates found
2002	Acme	In-house standard reference materials, in-house pulp repeats	Certificates found
2003	Eskay mine laboratory	Unknown standard reference materials and blanks	Certificates found
	Acme	In-house standard reference materials, in-house preparation, pulp and reject repeats	Certificates found
2004	Eskay mine laboratory	Standard reference materials, blanks, preparation, pulp and reject repeats	Certificates found
	Acme	In-house standard reference materials, in-house preparation, pulp and reject repeats	Certificates found

The previous operator generated control charts in Excel and included the results in the month-end drilling reports. These control charts showed that the QA/QC measures taken to ensure unbiased, accurate and precise sampling were effective.

Sample repeatability at Eskay Creek was closely monitored during the 2004 drilling campaign by the regular insertion of field duplicates into the sample stream. Field duplicates at the Eskay mine laboratory performed well with the duplicate sample set.

An audit was conducted on the 2004 QA/QC results and procedures by Dr. Barry Smee, of Smee & Associates Consulting Ltd. (Gale et al., 2004). The findings from the analysis identified a low bias in relation to Acme's internal SRMs for both aqua regia and fire assay methods. Acme corrected the inconsistencies with batch repeats. The sampling precision by means of using duplicate preparation and pulp samples was found to be within acceptable limits.

11.5.2 Skeena

Skeena implemented a formal QA/QC program from the inception of their 2018 Phase 1 drilling program, consisting of blanks, duplicates and SRMs.

The blank material used was a marble garden rock obtained from Canadian Tire in Smithers, BC. Approximately 1 kg of this material was used for each blank sample. Three blanks were inserted for every 100 samples.

Five SRMs were used during the 2018 Phase 1 drilling program. The SRMs were purchased from CDN Resource Laboratories Ltd. (CDN) of Delta, British Columbia to best match the rock matrix seen at Eskay Creek, as well as to match the analytical method used on the samples. One SRM was certified for gold only (CDN-GS-1T), two SRMs were certified for gold and silver only (CDN-GS-5T and CDN-GS-25), and two SRMs were polymetallic standards certified for gold, silver, copper, lead, and zinc (CDN-ME-1312 and CDN-ME-1601). Five SRMs were inserted for every 100 samples. SRMs were usually inserted in rotation, except where high-grade intervals above approximately 20 g/t Au were encountered; in this instance the high-grade SRM (CDN-GS-25) was inserted.

SRMs and blanks were monitored when batches of assay data were first received. SRM or blank control charts were routinely updated for the following elements: gold, silver, copper, lead, and zinc. Other elements were analyzed on an as needed basis. Control charts for SRM charts were prepared using the acceptable value plus or minus three standard deviations. If analyses were outside of the acceptable range after checking for data entry errors, then repeat assay were requested. Where two or more consecutive SRMs were both biased high or low (more than 105% of the expected value or less than 95% of the expected value) repeat assays were requested. The laboratory was instructed to retrieve five pulp samples before and after the QC failure.

Two kinds of duplicates were processed during the Phase 1 Eskay Creek drilling program: preparation and pulp duplicates. The preparation duplicate is a split that the laboratory takes from the reject material at a rate of one in every 50 samples. The pulp duplicate is an exact repeat of the primary pulp sample analysed immediately after the original sample. Pulp repeat insertion rates were at the discretion of the laboratory manager. Preparation and pulp duplicate data sets were routinely charted using X-Y scatterplots, relative percent difference versus average graphs and quartile-quartile plots.

Skeena monitored the laboratory performance and reported any concerns to the laboratory representative. If re-assays were deemed necessary, a series of five to nine surrounding samples on either side of the failure were requested to undergo re-assay.

11.6 Databases

11.6.1 Legacy

11.6.1.1 2004

Drill logs and sample data were compiled into an SQL server-based database where all geological, assay and survey information were entered. Once the drill hole data had been approved, the drill hole was locked from further editing and data were transferred to a Vulcan database to allow plotting and spatial interpretation. Hole locations were visually checked on import for collar and survey errors.

Information collected from each underground face was entered daily into an inhouse Access database and then transferred to a Vulcan database.

11.6.1.2 Skeena Legacy Database Review

In early 2018, Skeena obtained access to the legacy database. The database files, assay certificates, drill hole logs, and report files were stored in various locations and in various states of order. No single complete data set was located.

Between May and July 2018 Skeena personnel compiled and reviewed all available drilling and assay data to rebuild and produce a validated database in Microsoft Access format. The legacy database originated as a Vulcan file that was extracted and used as the building block for the final Skeena legacy database.

Digital certificates of original and rerun assays were located for the 1999–2004 period from the Eskay Creek mine laboratory as well as from three IPL, Bondar Clegg and Acme. Although only a partial set, the assays with certificates were imported into the Skeena database, and were flagged to take precedence over any other assay values within the legacy database. A total of 27,609 of the 426,367 assays in the Skeena legacy database were validated with original certificates. Gold and silver make up most of the assays in the Skeena legacy database, whereas base metals (lead, copper, zinc) and deleterious elements (arsenic, mercury, antimony) account for a lesser proportion in the Skeena legacy database because they were only selectively analysed during the 1999–2004 period.

Lower detection limit (LDL) inconsistencies were encountered in the legacy database. The Eskay mine laboratory did not consider values below 1 g/t Au and 10 g/t Ag as significant, therefore those grades were either set to a default of 0.5 g/t Au and 5 g/t Ag or left as <1 g/t Au and <10 g/t Ag. Base metal and deleterious elements below detection limits were set to 0.0%. Due to the high cut-off grades at the time that the mine was in production, the use of these default lower detection limits had little impact.

Skeena reviewed the methodology and assays certificates from the Eskay mine laboratory and determined reporting to 0.1 g/t for Au and Ag. For assays below this true detection limit, a value of half of this limit was applied in the Skeena legacy database (0.05 g/t for gold, 0.05 g/t for silver, and 0.005% for lead, copper, zinc, arsenic and antimony). In addition, all LDLs from the independent assay laboratories were originally set to 0.0 g/t in the legacy database for all elements analyzed. Skeena reset the LDLs to the actual limits used by the independent laboratories at the time.

The legacy database had a total of 41,624 duplicate primary sample numbers. These duplicates were a result of previous operators reusing a sample tag number that had already used by earlier drilling campaigns in different years. Skeena rectified the conflicts by creating a new column in the Skeena legacy database that uniquely identifies the sample by year of drilling first and then by sample number.

For data integrity purposes, the Skeena legacy database retains all the original sample numbers with unmodified assay values in separate, searchable columns. This applies to multiple element rerun samples as well. A priority system was set up so that a final "element_best" column gives precedence to assay values with validated assay certificates over unconfirmed samples.

Drill core at Eskay Creek was selectively sampled by the previous operators based on visual estimations of mineralization, which resulted in many unsampled intervals within the body of mineralization. Skeena tagged these unsampled intervals with an assigned value of -99 in the Skeena legacy database. In some cases, samples were not analyzed due to insufficient material provided to the laboratory or samples not received. The previous operator coded these samples with one of five default values. Skeena denoted these samples with a value of -66 in the Skeena legacy database.

Once the Skeena legacy database had been rebuilt, it was validated for gaps, overlapping intervals, duplicates, and lower detection limits. Surface drill hole collar locations were checked against the topographic surface for accuracy, and underground drill hole collar locations were checked against underground development wireframes. Where available, drill holes collar locations were confirmed from the original drill logs.

Following validation, 306 drill holes were flagged in the Skeena legacy database and were excluded from the data export used to create the Mineral Resource estimate. The excluded drill holes include:

- 31 holes where collar locations were reported as suspicious in 2004 and 2006 internal company reports;
- 4 surface holes where mineralized intervals do not correlate with underground development;
- 19 holes with duplicate sample numbers and/or overlapping assay intervals;
- 24 drain holes;
- 228 surface holes south of 8250N that were outside the extents of the Mineral Resource estimate.

Drill holes were imported using a mine grid that is rotated 23º to the east. The Skeena legacy database was updated with complete UTM and mine coordinates based on a formula provided by McElhanney (McElhanney, 2004). The mine grid coordinates were established by applying a rotation and scale factor as well as northing, easting and elevation shifts to the UTM values.

11.6.2 Skeena

Assay data are received via email from ALS Canada Ltd. Emails contain a PDF and XLS of the data.

Original copies are saved to the appropriate location on the internal file server. Altered data, formatted to how the Skeena database program handles the importing of data, is downloaded from the ALS Webtrieve website. These data are then saved on the internal file server, then imported directly into the commercially-available logging and assay software, GeoSpark. Data are processed in GeoSpark and managed in GeoSpark and Microsoft Access. GeoSpark uses MS Access as it's backend Database system.

GeoSpark backups are performed on a weekly basis to the internal file server, at minimum, or whenever there has been an update to the database. The Skeena file server is backed up (hourly backups) to a mirror server located at our IT Partner, Transparent Solutions, office in Burnaby. Another off-site backup is sent to a data vault in Kelowna in the event of a major catastrophic event within the greater Vancouver area.

11.7 Sample Security

The QP deems that the samples collected and prepared with the oversight of Skeena geologists in 2018 and 2019 were treated according to best industry standards in terms of sample preparation, security and analytical procedures.

The process by which legacy samples were treated, however, are not strictly known. All available evidence suggests that sample preparation, security and analytical procedures were conducted according to best industry standards.

11.8 Sample Storage

Core from the Skeena drill programs is stored at both the Eskay Creek Mine site carpentry shop (drill holes SK-18-001 to SK-18-020), and the McLymont Creek staging area (drill holes SK-18-021 to SK-18-040, SK-18-042, SK-18-043, SK-18-048 to -051, and SK-19-052 through SK-19-247).

11.9 QP Comments on "Item 11: Sample Preparation, Analyses, and Security"

Sample preparation, analysis and security are acceptable, meet industry-standard practice and are suitable for Mineral Resource estimation purposes. Drill sampling has been adequately spaced to first define, then infill, gold anomalies to produce prospect-scale and deposit-scale drill data. The assay data are adequately accurate, precise and contamination-free for use in Mineral Resource estimates. Verification is performed on all digitally-collected data on upload to the main database and includes checks on surveys, collar co-ordinates, lithology and assay data. The checks are appropriate and consistent with industry standards.

12 DATA VERIFICATION

12.1 Internal Data Verification

Data verification that was conducted by Skeena is described in Section 11.6.

12.2 External Data Verification

12.2.1 SRK (2018–2019)

SRK conducted an independent review of the Skeena database, which consisted of review of the available legacy data in 2018, and review of the data from the Skeena 2018 Phase 1 drilling program in 2019. In addition, SRK reviewed the QA/QC programs.

12.2.1.1 Skeena Legacy Database

SRK conducted routine verifications to ascertain the reliability of the electronic drill hole database provided by Skeena. All assays in the Skeena legacy database were verified against Eskay mine laboratory and IML assay certificates, where assay certificates were available. No significant errors or omissions were discovered; however, the large number of missing assay certificates was a limitation on the validation effort.

The Skeena legacy database was checked for missing values, duplicate records, overlapping intervals, sample intervals exceeding maximum collar depths, borehole deviations, drill holes collars versus topography, laboratory certificate vs database values and special values (i.e. non-numeric or less than zero). Minor errors were reviewed with Skeena personnel. All modifications to the Skeena legacy database were checked to ensure appropriate allocation. These included assay priorities ranking and accurate, consistent LDL updates.

SRK viewed the collar locations of underground drill holes by means of 50 m sections with drill hole volume projections of 25 m. There was no obvious discrepancy between collar location and underground workings. Viewed on 50 m sections, the drill holes collars originating from the surface appeared to correlate reasonably well with the topographic layer. There were, however, several drill holes that occur approximately 20 m above or below the surface layer. Given the fact that the collar locations have more accurate spatial resolutions than the topographic surface, this discrepancy was not thought to be a material concern. SRK cross-checked the UTM and mine grid coordinates from the 2004 McElhanney report with the Skeena legacy database. The checks confirmed that the imposed UTM-mine grid shift was acceptably accurate.

12.2.1.2 Skeena 2018 Data

SRK inspected the 2018 data for collar survey discrepancies, erroneous downhole deviation paths, and overlapping or missing assay and lithology intervals. All errors found were corrected and the dataset used for resource estimation included the correct values.

12.2.1.3 Site Visit

SRK performed a site visit in June 2018, and undertook the following:

- Located and resurveyed 50 drill hole collars, located on 22 drill pads;
- Visited the existing mine infrastructure;
- Visited the historical regional exploration camp at km 45; now owned by a third-party;
- Visited Albino Lake, the disposal site for drill core and low-grade waste.

12.2.1.4 Legacy QA/QC Review

SRK reviewed all available QA/QC data as follows:

- Compiled 190 samples out of a total of 17 drill holes from the 1997 data files. Scatter plots constructed from these data show high correlations between the original and the duplicate assays;
- Compiled all the mine assay certificates available and 126 pulp duplicates from the 1999 drilling campaign. A high correlation between the original and the duplicate assays were observed in scatter plots;
- Compiled all available mine assay certificates and retrieved 306 pulp duplicates from the 2001 drilling campaign. A high correlation between the original and the duplicate assays were observed in scatter plots;
- Located the SRM certificates for DS3, DS4 and DS5 used in the 2002 programs, and independently compiled quality control charts using the results from the original exploration certificates. Gold data, using scatter plots, indicated that the samples fell within the acceptable limit of three standard deviations;
- Evaluated two Acme in-house SRMs from 2003. The SRMs fell within three standard deviations of the acceptable values, although a slight low bias was noted;
- Reviewed the previous operator's QA/QC data from 2004. SRK re-created SRM and blank charts based on some of the data that that the previous operator used, and the results fell within an acceptable range of values for gold. Field duplicates at the Eskay mine laboratory performed well with the duplicate sample set.

12.2.1.5 Skeena 2018 QA/QC Review

Review of 106 blank samples indicated two instances of likely contamination. One sample occurred immediately following a high-grade sample. Since the elevated blank sample was <1% of the previous high-grade sample result, it was considered to be acceptable. No re-assays were requested for the blank results for the 2018 Phase 1 drill program.

Five commercially-produced SRMs were inserted into the sample stream during the 2018 Phase 1 drilling program. Some SRM mislabelling was identified and corrected during Skeena's QA/QC reviews. An analysis of standard charts for gold showed no obvious errors or biases. Review of data outside a three standard deviation limit typically showed the assay values to be acceptable as the anomalous values occurred within a sequence of low-grade assays.

Preparation (rejects) and pulp duplicates were routinely run at ALS as part of the laboratory's internal QA/QC procedures. Paired preparation and pulp data performed within acceptable tolerance criteria at both lower grade and higher-grade values.

SGS Canada was used as a check laboratory for the 2018 drill program, and re-assayed about 2.5% of all samples. A total of 45 pulps were checked against pulps originally processed at ALS Vancouver. Overall, the check assays performed within acceptable limits for both gold and silver.

12.3 QP Comments on "Item 12: Data Verification"

All available QAQC files were provided to the QP, who conducted the following verification steps:

- Rebuilt CRM, duplicate and blank charts from original data by year of analysis;
- Assessed both Eskay mine lab and independent laboratory QA/QC data;
- Assessed the results in relation to existing QA/QC documentation, and according to industry accepted standards.

The results of the QA/QC analysis indicate that the historical data are unbiased. A large number of assays in the database were validated against the original digital assay certificates. These assays ranged from the years 1999 to 2004, and less than 1% errors were found. In addition, the data analysed for the 2018 Phase 1 drilling program was collected and analysed in a systematic and unbiased manner. The data verification of this data did not identify any material issues and the QP is satisfied that the assay data is of suitable quality to be used as the basis for the resource estimate.

13 MINERAL PROCESSING AND METALLURGICAL TESTING

13.1 Background

In 1991 and 1992, metallurgical testwork for the feasibility study had defined a complex hydrometallurgical flowsheet for the recovery of gold and silver, as well as copper and zinc. This process required a large capital outlay with high unit operating costs. The original operating plan was to construct the mining infrastructure at the mine site and transport ore to a processing facility located close to Placer Dome's Equity Silver mine, near Houston, B.C.

In late 1994, mining operations commenced at Eskay Creek. In 1996, a testwork program was initiated at Process Research Associates with follow up locked-cycle testing at International Metallurgical and Environmental Inc. to evaluate the potential of a gravity/flotation process for upgrading ore from the NEX and 109 Zones into marketable concentrates.

The work indicated that the mineralized material could be economically upgraded to a saleable concentrate.

In 1997, Prime completed the engineering and construction of a 150 t/d mill to concentrate the gold and silver values for the NEX and 109 Zones. Over the next several years, the mill was steadily upgraded and expanded to its final production capacity of 350 t/d. Since 2008, the mine area has been under a state of reclamation, care and maintenance.

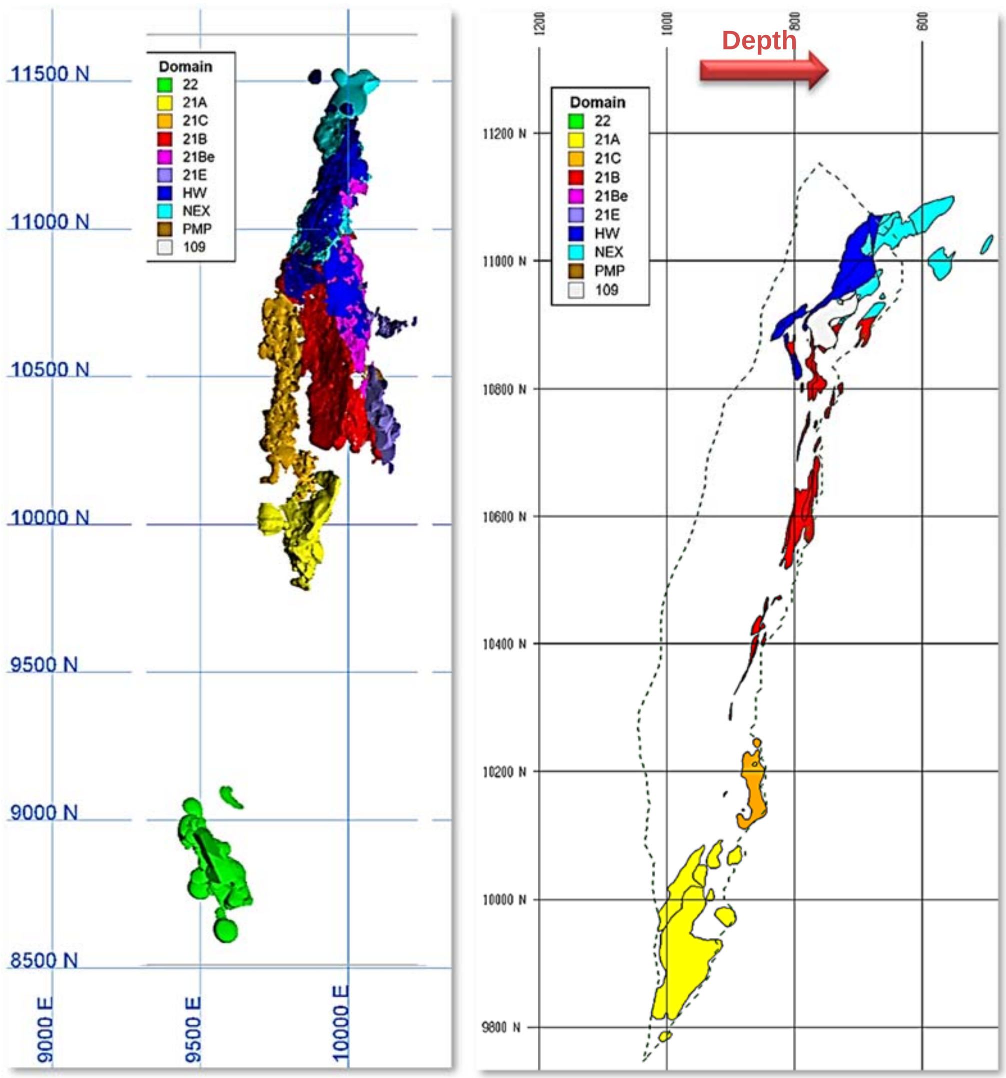
As part of this PEA update, recent testwork has been completed by BlueCoast Research (BlueCoast) in Parksville BC, including comminution, whole ore leaching, gravity and flotation recovery methods. The process plant flowsheet assumed for this PEA includes only flotation recovery of a precious metal concentrate, for transport and shipment overseas. To further investigate to generate doré as a saleable product, a number of concentrate treatment alternatives are being evaluated. Concentrate treatment is an opportunity to transform the deleterious minerals into a safe form rather than incur higher treatment charges and penalties by including them in the concentrate.

13.2 Mineralised Zones

The Eskay Creek mineralization is divided in a number of mineralised zones or domains as shown in Figure 13-1. Within each zone, the main rock types are mudstone, rhyolite and hanging wall andesite.

Extensive underground workings are present below 1,000m RL, mainly in the 21C, HW and NEX Zones. A significant part of the open pit mining area will be in the 21A, 21B and 22 Zones over the proposed nine-year mine life.

Figure 13-1: Mineralised Zones (left: plan view, right: rotated vertical section)



Note: Figure prepared by SRK, 2019.

Table 13-1 shows typical gold and silver grades within each mineralised zone, including arsenic, antimony and mercury as possible impurities to a bulk, precious metal concentrate. Zone 21A has elevated arsenic and mercury, while several zones have 0.16% Sb or higher.

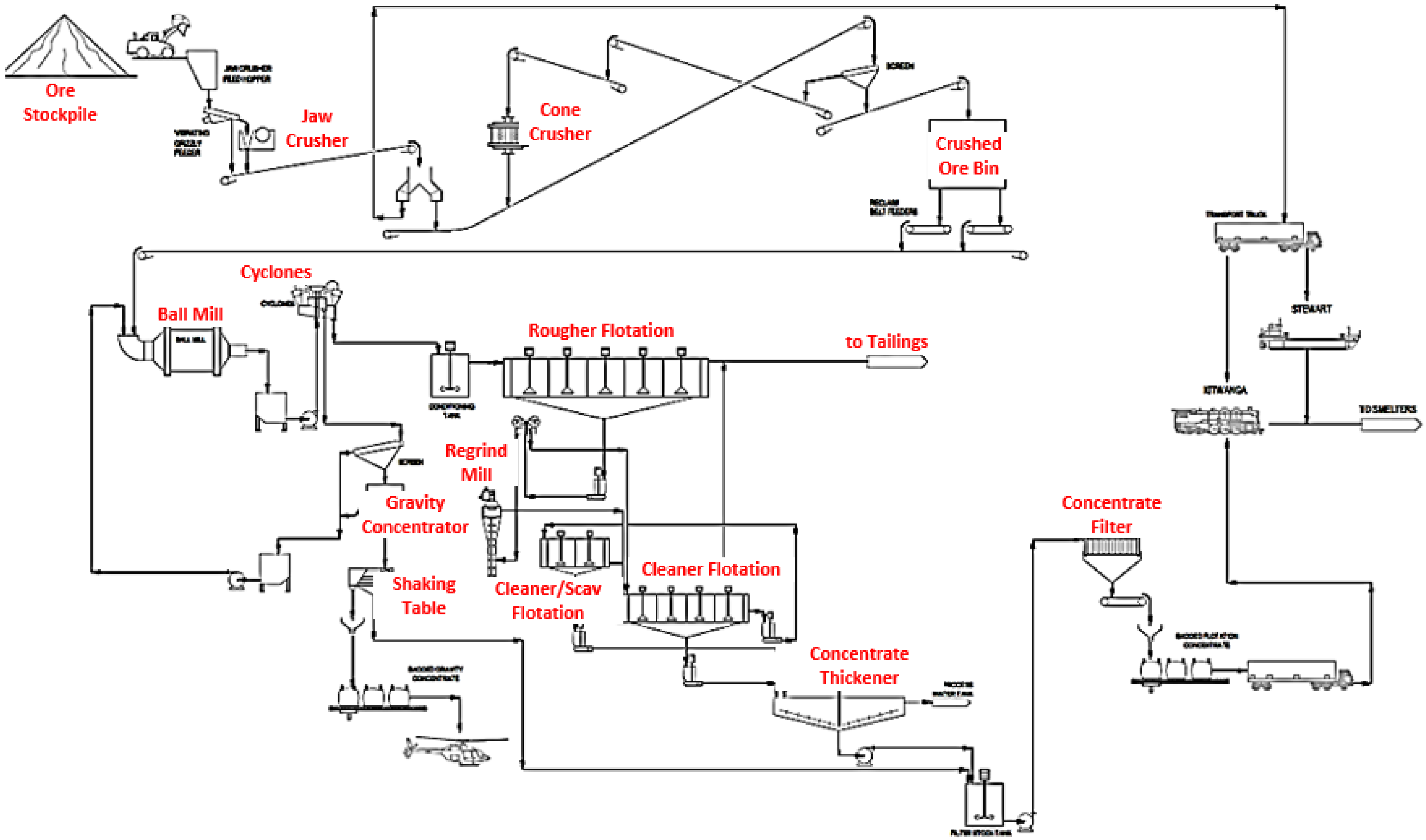
13.3 Original Process Flowsheet

The Eskay Creek plant flowsheet which operated until 2008 is shown in Figure 13-2. Following crushing and grinding, a gravity concentrate was cleaned on a shaking table before being transported off site via helicopter. Gravity tails reported to rougher, cleaner and cleaner/scavenger flotation to produce a bulk sulphide concentrate. The concentrate was thickened/filtered prior to bagging and truck transport to a rail load-out station at Kitwanga.

Table 13-1: Mineralised Zone Typical Grades

Zone	Au (g/t)	Ag (g/t)	As (ppm)	Sb (ppm)	Hg (ppm)
21A	5.3	85	3,000	2,418	231
21B	4.9	130	537	1,899	91
21Be	5.4	192	1,627	1,727	82
21C	3.5	61	295	584	15
21E_n	2.5	88	363	4,076	21
22	2.7	43	909	262	6
HW	3.6	164	480	1,645	31
NEX	4.3	155	446	768	20

Figure 13-2: Original Eskay Creek Flowsheet



Note: Figure prepared by SRK, 2019.

For some high-grade zones, direct shipping ore (DSO) was transported to the port of Stewart after being crushed down to a manageable size by the jaw crusher.

An estimate of plant performance from 2006 showed that throughput varied from 225 t/d to 325 t/d across the different mineralised zones. In addition, combined gravity + float concentrate recoveries ranged from 69–92% for gold and 87–96% for silver.

13.4 Historical Testwork

Testwork performed by SGS Lakefield for International Corona in 1991 and 1992 examined the amenability of whole ore samples to pressure oxidation (POX) and carbon-in-leach (CIL) and cyanide leaching for the recovery of copper, zinc, gold and silver. Results summarised in a series of reports submitted to International Corona in 1991 and 1992 covering POX operating conditions, cyanide detoxification and environmental impact assessment of the tailings products. The final flowsheet was trialled in a series of pilot plant runs at SGS Lakefield.

A paper was presented by SGS Lakefield at the Randol Gold Forum in 1995 on the testwork program and flowsheet development (Fleming, 1995).

13.4.1 Pressure Oxidation and Two-stage Leaching

The testwork focused on a multi-stage, novel flowsheet that included recovery of saleable cemented copper and zinc via selective precipitation prior to cyanidation (Figure 13-3). Following pressure oxidation and counter-current-decantation (CCD) of the residues, the liquor was sent for copper–zinc cementation. The autoclave residue was submitted to both CIL and direct cyanidation, with a lime boil stage inserted to minimise silver losses from jarosite generation.

A wide range of pressure oxidation, CIL and direct cyanidation tests were completed on four high-grade composite samples ranging from 14–109 g/t Au, and 254–6,651 g/t Ag. Copper, lead and zinc grades were quite high (up to 1.68% Cu, 15.2% Pb and 11.4% Zn). However, arsenic grades were under 800 ppm and only one sample was >1% Sb.

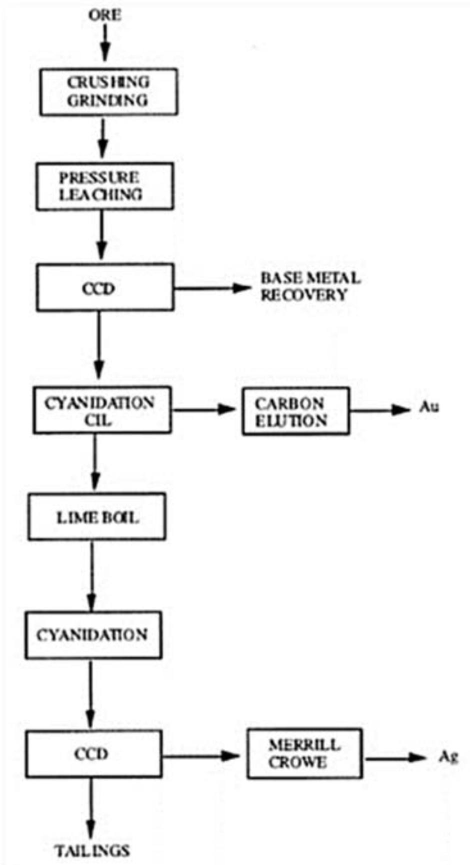
After POX treatment, leaching of autoclave residues resulted in gold extractions of 82–97%, with cyanide residues assaying 2–12 g/t Au. Silver extraction ranged from 60–95%, producing cyanide residues assaying 100–300 g/t Ag.

Leading up to October 1991, further optimisation leach tests were conducted on pressure oxidation residues from previous tests. Direct cyanidation tests on autoclave residues resulted in gold recoveries in the range of 35–68%, and silver recoveries in the range of 66–97%. These tests showed that higher recoveries of gold were obtained at lower percentage solids, higher temperature, higher NaCN concentration, and longer leach time.

Cyanidation followed by CIL testwork on an autoclave residue resulted in gold recoveries of 75–92%, and silver recoveries of 97–98%. These tests demonstrated gold and silver recovery increased with increasing CIL retention time. Maximum silver recovery was reached at 48 hours while gold recoveries continued to improve over the full 120-hour duration of the experiments.

Cyanide consumption, recovery and detoxification tests were conducted between October and November 1991. Cyanide consumption tests showed the CIL conditions favours cyanide destruction with long residence times, high carbon concentrations and high temperatures. Tests showed zinc and copper were the main cyanide consumers, accounting for 60–70% of total NaCN consumption.

Figure 13-3: SGS Lakefield Flowsheet Evaluated in 1991



Note: Figure prepared by SGS, 1991.

Seven continuous pilot plant tests were conducted at SGS Lakefield, examining process variables such as temperature, oxygen pressure and retention time. At 200°C and 60 minutes, increasing the over-pressure from 50 psi to 111 psi increased the gold extraction from 91% to 94%, and reduced the residual sulphide from 1% to 0.8%. Gold extractions were 91–93% and silver extractions were 85–88% for all tests after 60 minutes.

Five more pilot plant tests were performed to reduce the amount of zinc and copper reporting to cyanide leaching. The test results showed overall cyanidation and CIL recoveries of 94% for gold and 89% for silver, leaving a final residue of 4.2 g/t Au and 300 g/t Ag. NaCN consumption was high at 9 kg/t for the CIL and 14 kg/t overall.

The final section of the testwork conducted in 1992 focused on copper and zinc recovery as well as the environmental impact of tailings disposal. Approximately 92–98% of copper and zinc was extracted in the autoclave liquor. Selective precipitation of copper and zinc was demonstrated, although the zinc cementation product did contain some contaminants that might impact its saleability.

Copper recovery ranged from 87–99% to an 80% Cu cementation product. Zinc recovery was 99% and the zinc cementation product assayed 44% Zn.

Environmental tests on the tailings indicated high biological toxicity of combined discharge products. Tailings solids were also tested for leachability and acid generating potentials. While the results indicated the solids did not have acid-generating potential, the leachate exceeded the allowable levels for aluminium, antimony, lead and zinc.

13.4.2 Bulk and Selective Flotation

A series of flotation tests was also conducted on the main composite sample as an alternative to POX and two-stage leaching. Attempts to pre-float graphitic carbon showed limited reduction with a rougher concentrate at 16% mass pull and 10% Au and 13% Ag recoveries.

Selective flotation of a copper–lead concentrate recovered 40% of the gold, 63% of the silver and copper in 8% of the mass. An additional 37% of the gold was recovered from the copper–lead tailings to a zinc–pyrite concentrate, which was 29% of the sample mass.

Bulk flotation yielded the highest gold recovery, but >90% Au recovery was achieved at mass recoveries >46% and as much as 60%. No improvement in the mass pull-gold recovery relationship was realized during these tests.

Flotation concentrates typically contained 120–150 g/t Au and 7,000–8,000 g/t Ag with significant amounts of lead, zinc and antimony. Total sulphur content was approximately 20% and >20% silica as the main non-sulphide gangue mineral.

13.5 Current Testwork

As part of this PEA, additional metallurgical samples were obtained from the 2019 drilling program and submitted to BlueCoast for testing and evaluation.

13.5.1 Sample Details

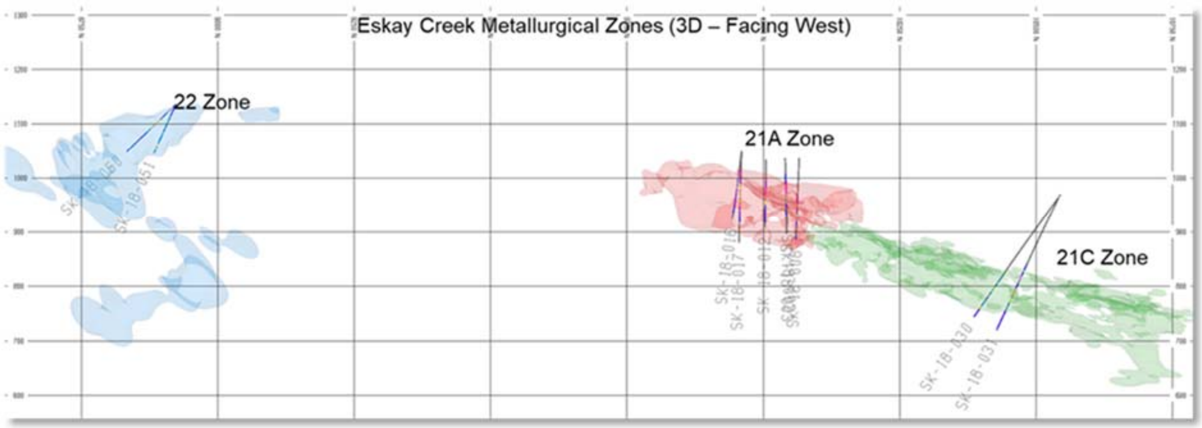
The drilling program in 2019 focused primarily in the 21A mineralised zone with auxiliary drill holes added in zones 21C and 22 (Figure 13-4).

From these drill holes, six metallurgical samples were collected including a “hot” sample to represent the mudstone unit in the 21A zone. This sample was elevated in silver, arsenic, antimony and mercury, significantly higher sulphur and sulphide content together with organic carbon (C_{org}). The grades of the 2019 metallurgical samples are provided in Table 13-2.

Zones 21A, 21C and 22 represent a significant portion of the life-of-mine (LOM) plant feed but Zone 21B was not sampled in the 2019 testwork program. Overall, the samples included a reasonable range in gold grade; however, they were lower in copper, lead, and zinc compared with the expected LOM average and future samples should be collected with higher base metal values.

The samples selected for metallurgical testing were representative of various mineralisation forms present within the different zones. Samples were selected from a range of locations within the zones and sufficient mass and testing was performed to support this level of study. Recommendations for additional testing as part of future studies are included in Section 13.8.

Figure 13-4: Location of 2019 Metallurgical Drill Holes



Note: Figure prepared by SRK, 2019.

Table 13-2: Metallurgical Sample Grades

Composite	Au (g/t)	Ag (g/t)	As (ppm)	Sb (ppm)	Hg (ppm)	S_{tot} (%)	S_{2-} (%)	C_{tot} (%)	C_{org} (%)
Hot	32.6	690	43,350	100,200	3,024	8.08	7.54	0.86	0.48
21A Low As	1.9	53	315	205	49	1.33	1.37	0.31	0.03
21A High As	8.3	54	4,005	4,240	127	2.59	2.25	0.62	0.43
21C	3.4	207	187	409	12	1.93	1.74	0.16	0.06
22 Low As	1.3	107	205	166	4	0.42	0.43	0.02	0.02
22 High As	2.8	10	1,180	330	9	0.77	0.77	0.02	0.03

The 21A and 22 zones were divided into High and Low arsenic samples, with the samples covering a range in grades from 1.3–32.6 g/t Au and 10–690 g/t Ag.

Two composite samples were generated to estimate the expected gold grade for the first three years and the life-of-mine (LOM). The composites were a blend of Hot (mudstone) and 21A Low As (rhyolite) samples:

- LOM sample: 91% 21A Low As + 9% Hot;
- Y1–3 sample: 83% 21A Low As + 17% Hot.

The consequence of blending with the Hot sample was elevated arsenic, antimony and mercury levels for the two composites which resulted in flotation concentrates being produced with artificially high impurity levels. Separate testing on lower grade samples produced concentrates with lower impurity levels so the performance estimates provided in Section 13.6 were not biased by the Hot sample blending.

Figure 13-5 shows the modal mineralogy of the mudstone and rhyolite material with the principal minerals labelled in the breakdown. The mudstone has appreciable amounts of sphalerite, realgar and stibnite while the rhyolite sample had almost none detected using automated mineralogy.

13.5.2 Comminution

Comminution or hardness testing on each sample consisted of semi-autogenous grind (SAG) mill comminution (DWi), Bond rod mill work index (RWi) and Bond ball mill work index (BWi) tests at a closing screen size of 150 µm.

The test results indicated a range of material hardness with mudstone being moderately soft (DWi of 3.2 kWh/m^3, BWi of 13.0 kWh/t), while the 22 Zone exhibited BWi values of as much as 26.4 kWh/t. The hardness values for each sample and test type are summarized in Table 13-3.

Previous testwork by SGS Lakefield on samples from the 21B, 21C, HW, 109 and NEX Zones reported SAG power index (SPI) values between 49 and 171 minutes with BWi results of 17.0–20.0 kWh/t, to an unreported closing screen size.

13.5.3 Whole Ore Leaching

Bottle roll cyanidation tests were performed on 21A Low As, 21C and Hot samples to evaluate potential for whole ore leaching compared with the historical testwork done by SGS on much higher-grade samples. Overall, gold and silver extractions after 48 hours were poor, with silver generally higher than gold. In particular, extractions from the Hot, mudstone sample >5% for gold and >3% for silver.

Leaching under a range of 80% passing (P_{80}) grind sizes (80 µm, 50 µm and 30 µm) did not show any significant effect. The 21C sample reported 31% Au extraction after 48 hours with 1 g/L NaCN and a P_{80} size of 33 µm. For most samples, the initial dissolved oxygen (DO) levels were low, and required air sparging throughout the bottle roll test. In addition, cyanide consumption was significant, ranging from 1.3–13 kg/t and increased at finer grind sizes.

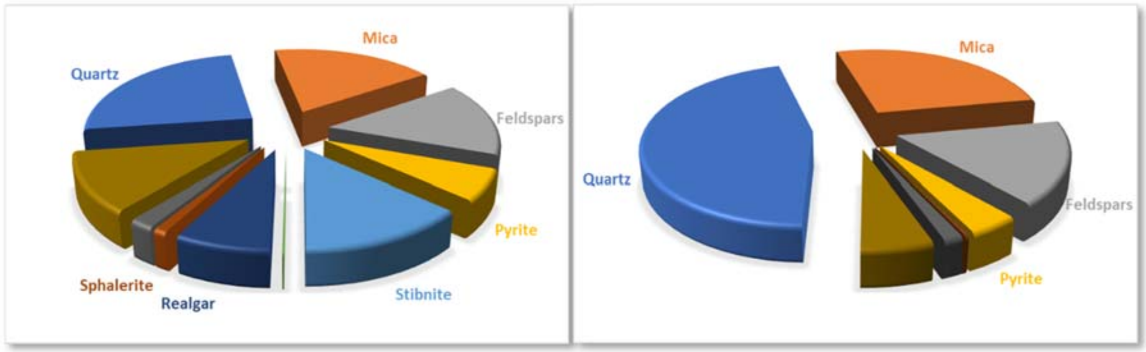
Low gold extractions were attributed to a number of possible factors: fine-grained gold particles, the presence of preg-robbing sulphides and/or organic carbon and possible passivation of gold surfaces by antimony. A CIL test on the 21A Low As sample was performed at a P_{80} size of 18 µm with 3 g/L NaCN and pre-oxidation, and reported only a 10% higher gold extraction compared with the base case leach conditions.

Samples of coarsely-ground, 21A Low As rougher flotation tailings were subjected to cyanide leaching but only reported 9–15% Au and 14–24% Ag extractions after 48 hours. These tests were done to estimate the sample amenability to cyanidation after the preg-robbing agents were removed to the flotation concentrate. Based on this sample, it does not appear that reasonable amount of precious metals can be recovered from the flotation tailings stream at grind P_{80} sizes down to 138 µm.

13.5.4 Gravity Recovery

The LOM composite sample (9% Hot + 91% 21A Low As) was tested using an extended gravity recoverable gold (E-GRG) procedure with a three-pass grind and recovery sequence. Only 45% of the gold was recovered to a grind P_{80} size of 82µm with only 13% of the silver recovered. More importantly, the combined gravity + flotation recovery was not higher than flotation alone. Based on this test result, gravity recovery was not recommended in the process flowsheet as part of this PEA.

Figure 13-5: Modal Mineralogy of Mudstone vs. Rhyolite (Hot & 21A Low As samples)



Note: Figures prepared by SRK, 2019.

Table 13-3: Summary of Comminution Testwork

Composite	Particle SG	DWi (kWh/m^3)	RWI (kWh/t)	BWI_{150} (kWh/t)
Hot	3.06	3.18	-	13
21A Low As	2.69	4.84	14	16.1
21A High As	2.69	4.73	15.2	16.2
21C	3.00	5.80	16.4	16.6
22 High As	2.62	7.34	21	23.5
22 Low As	2.59	5.91	21.8	26.4

Note: BWi measured at a closing screen size of 150 µm.

13.5.5 Bulk Flotation

A considerable number of open-circuit, rougher and rougher/cleaner float tests were conducted on all samples included in Table 13-2. The 21A Low As sample was initially tested under a wide range of conditions and later applied to the other samples, as part of variability testing.

The testwork objective was maintaining high precious metal recoveries at a lower mass pull to concentrate, which was evident in early testing as well the historical work done by SGS in 1991.

A range of primary P_{80} grind sizes were tested (from 338 µm down to 39 µm) with ~60 µm used as the target P_{80} grind size for further float work. Rougher concentrate was also reground prior to cleaning, with a target P_{80} size of ~25 µm used as the base case.

It was noted that the grind and regrind times were quite long (up to 40 minutes being required for the 25 µm regrind size); however, an investigation into possible overgrinding of phyllosilicate minerals did not reveal anything significant. Blue Coast noted that the flotation concentrate was very slow to pressure filter, and this remains a concern to be investigated and possibly addressed in solid/liquid separation testing in the future.

The use of dispersants (sodium silicate and carboxymethyl cellulose, or CMC) was investigated as well as collector dosage. Much lower mass pulls were obtained with affecting recovery using stainless steel grinding media (with the resulting positive oxidation reduction potential (ORP) values after grinding) as well as lower percentage solids from a larger volume float cell.

Samples exhibited relatively slow float kinetics with 80% Au recovery after 20 minutes of rougher flotation and 90% recovery after 40 minutes. An investigation into possible sliming did not reveal any explanation for the slow-floating nature of the samples.

Up to three stages of cleaning were done, with concentrates generated after 25 min, 15 min and 10 min of float time. Copper sulphate was added at 100 g/t to the primary grind as an activator with potassium amyl xanthate (PAX) used throughout as the collector, with a total of 200 g/t added.

The base case flowsheet used to evaluate the range of samples is shown in Figure 13-6.

Overall, the flotation testwork was able to produce a bulk concentrate with gold recoveries of 80–95% at grades of 40–50 g/t Au. Silver recoveries were in the range of 84–97% with grades from 1,000–1,300 g/t Ag.

The base case float conditions were tested on the lower-grade samples (21A Low As, 21C, 22 Low As, 22 High As) to generate a metal recovery versus head grade relationship. The results showed a consistent behaviour across all samples, reflecting the relatively low amount of sulphides being recovered from rougher and cleaner flotation.

The range of final concentrate precious metal grades and impurity levels are shown in Table 13-4, sorted in order of gold head grade. As will be discussed in Section 13.6, the sulphide minerals containing arsenic, antimony and mercury closely follow the gold–silver-bearing minerals to final concentrate.

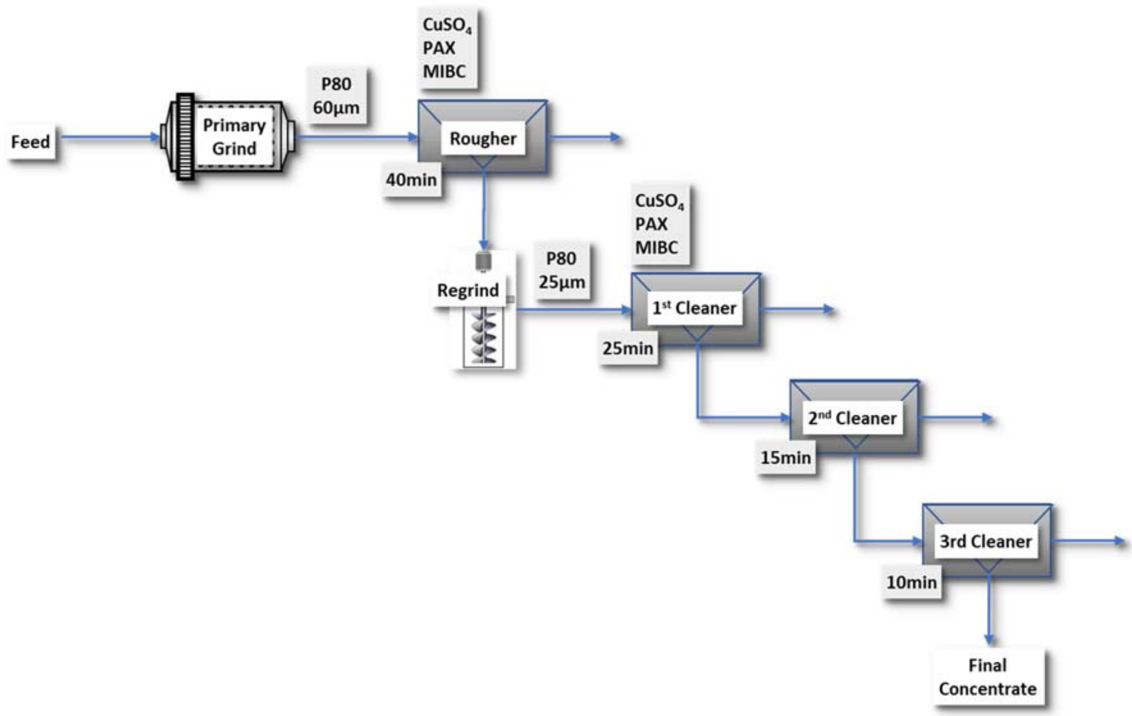
For the <3.5 g/t Au samples, the final concentrate contained around 1% As and Sb with ~200 ppm Hg. The LOM composite generated concentrates with much higher impurity levels due to the blend of Hot sample. As the expected mine plan calls for material at 4 g/t Au and below, the lower-grade sample results were used to generate the forecasted concentrate quality and quantity.

13.5.6 Flotation Product Mineralogy

Automated mineralogical analysis was performed on both the final concentrate and tailings from the LOM sample float testing. Table 13-5 summarises the main minerals in the two streams.

The LOM sample concentrate contained 19% pyrite with ~7% stibnite and realgar, together with 25% silica and 35% phyllosilicate minerals. In contrast, the tailings contained minimal sulphides (after an extended rougher flotation period) with 54% silica and 31% phyllosilicate minerals.

Figure 13-6: BlueCoast Laboratory Float Test Flowsheet



Note: Figure prepared by SRK, 2019.

Table 13-4: Concentrate Grades for Eskay Creek Samples (results sorted by gold head grade)

Sample	Head Au (g/t)	Head Ag (g/t)	Final Concentrate Au (g/t)	Final Concentrate Ag (g/t)	Final Concentrate As (%)	Final Concentrate Sb (%)	Final Concentrate Hg (ppm)
22 Low As	1.3	107	53.8	7,565	1.0	1.2	239
	1.3	107	49.1	5,395	0.7	0.7	138
21A Low As	1.9	53	40.2	1,208	0.6	0.7	1,091
22 High As	2.8	10	48.8	219	3.5	0.9	
	2.8	10	38.1	154	2.1	0.6	140
21C	3.4	207	52.5	4,150	0.3	1.2	230
	3.4	207	55.9	4,779	0.3	1.2	249
LOM Comp	3.9	96	40.6	1,036	3.5	9.2	
	3.9	96	52.3	1,115	4.8	9.8	3,817
	3.9	96	41.3	1,042	4.2	9.1	3,501
Yr 1-3 Comp	7.7	164	54.8	1,182	5.4	12.3	
	7.7	164	50.0	1,244	5.9	12.3	4,464
21A High As	8.3	54	51.3	382	2.9	3.5	

Table 13-5: Modal Mineralogy of Flotation Products (LOM sample)

	Concentrate	Tailings
Pyrite	19	0.5
Chalcopyrite	0.2	—
Sphalerite	1.1	—
Stibnite	7.5	0.1
Realgar	6.5	—
Quartz	25	54
Phyllosilicates	35	31
Calcite	2.1	2.6
Barite	0.1	0.26
Other	3.3	11.6

13.6 Expected Performance Estimates

Based on the 2019 testwork results on samples with a range of head grades, a flotation concentrate of saleable precious metal content can be produced at high recoveries of both gold and silver. This concentrate will contain impurities of arsenic, antimony and mercury that will be subject to penalties. Depending on the concentrate customer, the antimony content may be included as a payable metal, provided the level is above a threshold value (e.g. 3% Sb). This is discussed in Section 19.

The open-circuit rougher and cleaner float test results were used to generate relationships between the gold and silver recovery versus head grade as well as the expected mass pull to concentrate. The concentrate impurity levels were well established from the testwork results. These relationships were done for 50 g/t, 40 g/t and 25 g/t Au concentrate to assist the marketing review completed as part of this PEA. The lower-grade concentrate required few stages of cleaner flotation.

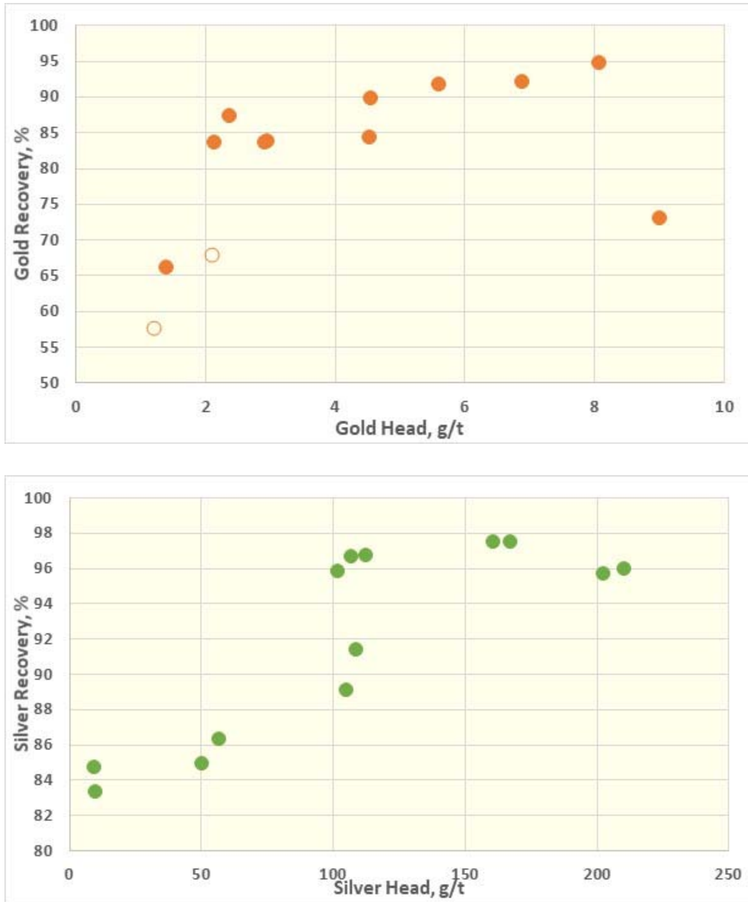
Figure 13-7 shows the forecast trend of gold–silver recovery to final concentrate versus head grade.

Figure 13-8 also shows estimated mass pull against the sample head grade, and shows the 25 g/t Au grade concentrate generating considerably higher tonnes of concentrate.

Figure 13-9 shows the consistent upgrade of sulphide minerals containing impurities to a 40 g/t Au concentrate. The very predictable behaviour of the samples tested added confidence that metallurgical performance could be reasonably well estimated using these simple relationships.

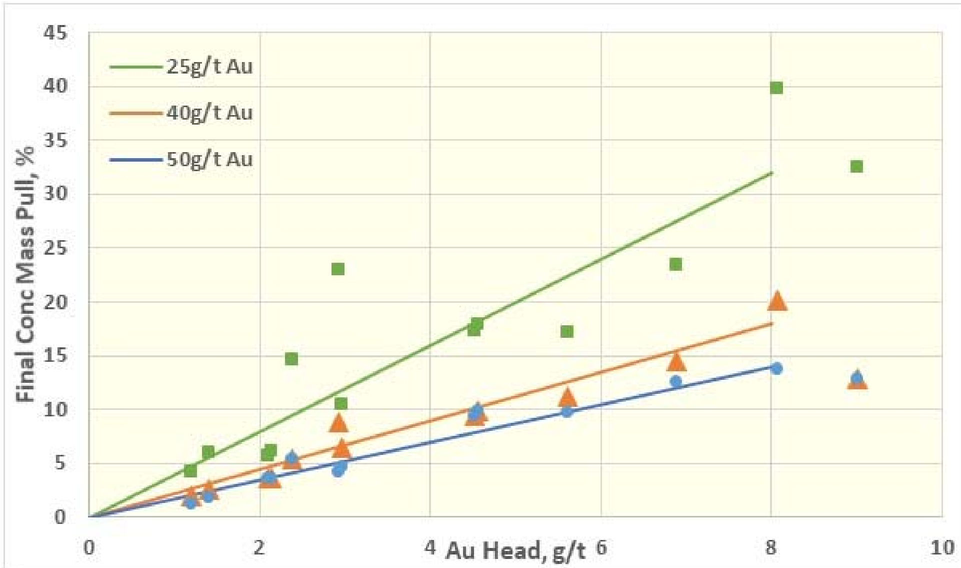
Across the nine-year mine life (Figure 13-10), 60% of the plant feed anticipated to be rhyolite with 20% mudstone and 20% hanging wall andesite material. In Year 1, almost 60% of plant feed will be from the 21A Zone with higher precious metal grades and impurity levels. As the percentage of the 21A material decreases over time, the gold head grade will fall from almost 5 g/t Au to around 3 g/t Au. Similarly, silver grade will be higher in years 1–6 at 100 g/t Ag, and will fall to around half this value in Year 7.

Figure 13-7: Gold and Silver Recovery vs. Head Grade (40 g/t Au concentrate)



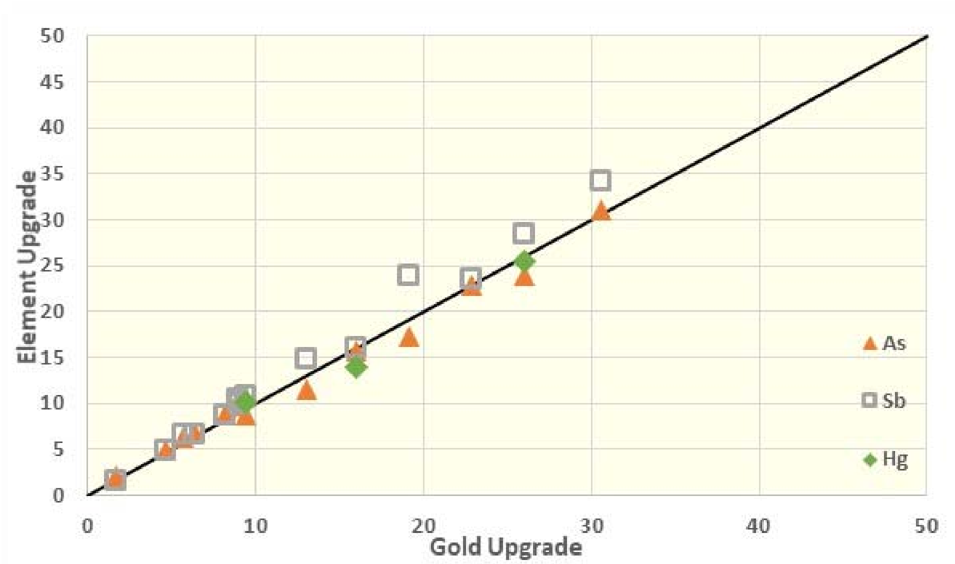
Note: Figures prepared by SRK, 2019.

Figure 13-8: Mass Pull vs. Head Grade



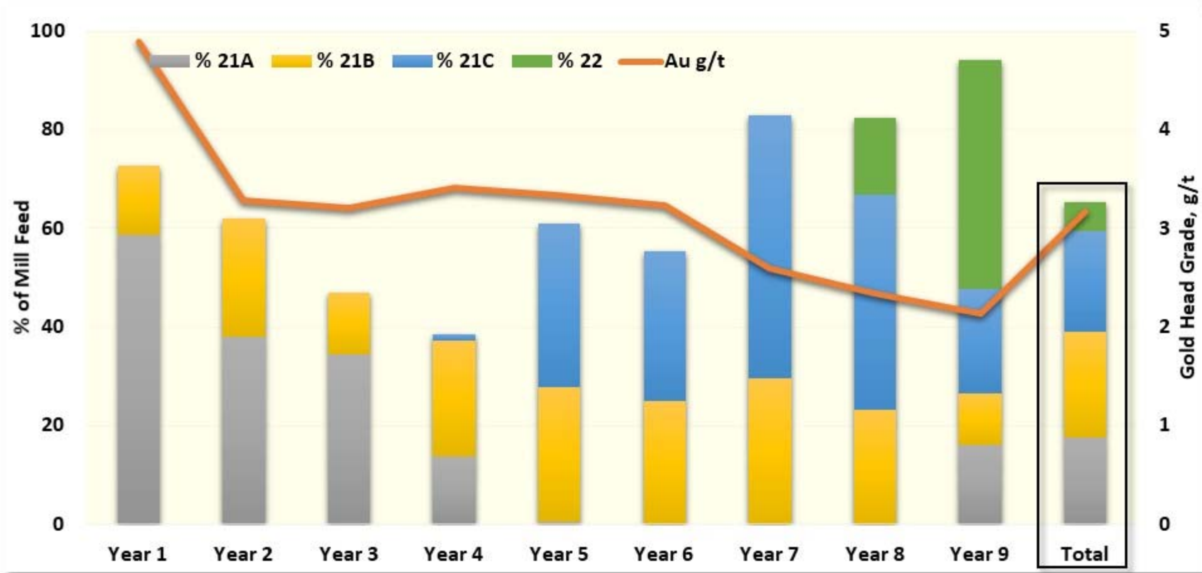
Note: Figure prepared by SRK, 2019.

Figure 13-9: Impurity vs. Gold Upgrade to 40 g/t Au Concentrate



Note: Figure prepared by SRK, 2019.

Figure 13-10: Mine Plan by Mineralised Zone and Gold Head Grade



Note: Figure prepared by SRK, 2019.

Table 13-6 summarises the gold and silver recoveries used in the mine plan optimisation for the three concentrate grades considered. The tonnes of concentrate were estimated from the mass pull relationships shown in Figure 13-8. Concentrate impurity levels were estimated from the gold concentrate upgrade as shown in Figure 13-9.

13.7 Concentrate Treatment Options

While the generation of a precious metal concentrate was demonstrated for all metallurgical samples tested in 2019, supplementary testwork is ongoing into options for concentrate treatment. These treatments involve hydrometallurgical or pyrometallurgical oxidation of the sulphide content prior to cyanide leaching with/without carbon to minimise the impact of preg-robbing agents.

For this PEA, concentrate treatment is considered an opportunity and the testwork is discussed in this section for background purposes only. The base case plant flowsheet and economics consider only a flotation concentrate being produced for the life of mine.

While the 1991–1992 SGS testwork investigated whole ore pressure oxidation as well as cementation of a saleable copper and zinc product, this work focussed on concentrate oxidation only, with the aim of generating a final doré product. The metallic impurities would either be encapsulated in a stable form (As, Sb) for disposal or recovered in the electrowinning stage (mercury retort). The concentrate treatment options being evaluated are:

- Pressure oxidation: continuation of historical work done by SGS Canada;
- Teck Metals CESL process: medium temperature, pressure acid leach;
- Glencore Technology's Albion process: medium temperature, atmospheric neutral leach.

Table 13-6: Estimated Precious Metal Recoveries

Au Recovery (%)			
Au Head (g/t)	**50 g/t Au Conc**	**40 g/t Au Conc**	**25 g/t Au Con**
<1.0	65	70	75
1.0 to 1.5	70	75	80
1.5 to 2.0	75	80	85
2.0 to 2.5	80	85	90
>2.5	90	90	92
Ag Recovery %			
Ag Head (g/t)	**50 g/t Au Conc**	**40 g/t Au Conc**	**25 g/t Au Con**
<100	86	88	90
>100	96	97	97

These options consider a range of temperature and pressure conditions as well as pulp acidity. Two other options that could be considered are partial roasting and bio-oxidation.

Preliminary testwork results show limited leaching of arsenic, with a stable iron arsenate (e.g. scorodite) remaining in the residue. Similarly, the antimony should remain unleached and present as a ferric antimonate in the residue. Any mercury present will go into solution and need to be recovered at the electrowinning stage via retort.

The historical SGS testwork showed that pressure and temperature can be successfully applied to the whole ore (or concentrate) to oxidise sulphide minerals and improve the downstream extraction in either a direct or CIL leach circuit. However, the flowsheet that was proposed was multi-stage and involved, with likely high capital and operating costs.

The investigation of alternate treatment options to POX can be evaluated during future technical studies. The advantage of concentrate treatment is the production of doré at >99% payable gold, opposed to a precious metal concentrate, subject to market conditions for treatment charges and penalties.

13.8 Recommended Future Testwork

Based on the flotation concentrate flowsheet being recommended for this PEA study, additional testwork is warranted to improve the confidence in the metallurgical performance estimates:

- Variability testing program for both comminution and flotation performance;
- Locked cycle flotation testing of year 1–3 and year 4–6 composite samples;
- Investigate options for blending high arsenic, antimony and mercury zones;
- Solid/liquid separation testing and investigation into source of filtering issues;
- Geochemical analysis and environmental testing of tailings products;
- Continued investigation into concentrate treatment options (including economic analysis).

14 MINERAL RESOURCE ESTIMATES

14.1 Introduction

The Mineral Resource model was prepared by Skeena and independently validated and signed off by SRK. The resource model considers 7,583 historical holes and 46 holes drilled by Skeena in 2018. The 2019 Mineral Resource estimate is primarily reported within a conceptual open pit shell, whereas the 2018 Mineral Resource estimate principally assumed underground mining methods. The estimation work was completed by Ms. K. Dilworth, a Skeena employee, and was reviewed and accepted by Ms. S. Ulansky, PGeo (EGBC#36085), Senior Resource Geologist with SRK, a Qualified Person as this term is defined in NI 43-101. The effective date of this Mineral Resource estimate is February 28, 2019.

This section describes the resource estimation methodology and summarizes the key assumptions considered. In the opinion of SRK, the Mineral Resource estimate is a reasonable representation of the global gold and silver Mineral Resources within the Eskay Creek deposit.

The database used to estimate the Mineral Resources was audited by SRK. The QP is of the opinion that the current drilling information is sufficiently reliable to interpret with confidence the boundaries for gold and silver mineralization and that the assay data are sufficiently reliable to support mineral resource estimation.

Leapfrog Geo™ (version 4.3.0) was used to construct the litho-structural model and mineralization domains that define the 2019 Eskay Creek model. Snowden Supervisor ™ (version 8.90) was used to conduct geostatistical analyses, variography and a portion of model validation. For block modelling, Maptek Vulcan™ (version 11.0.1) software was used to prepare assay data for geostatistical analysis, modify mineralization domains, construct the block model, estimate metal grades and to tabulate the Mineral Resources.

14.2 Resource Estimation Procedures

The Mineral Resource evaluation methodology involved the following procedures:

- Database compilation and verification;
- Construction of wireframe models for the litho-structural model;
- Construction of wireframe models for gold–silver mineralization;
- Definition of resource domains;
- Data conditioning (compositing and capping) for geostatistical analysis and variography;
- Block modelling and grade interpolation;
- Resource validation;
- Resource classification;
- Assessment of “reasonable prospects for economic extraction” and selection of appropriate cut-off grades; and
- Preparation of the Mineral Resource statement.

14.3 Resource Database

The Eskay Creek database used for the creation of the resource estimate contains 7,629 drill holes totalling 659,069 m. This includes 7,583 historical drill holes within the extents of the resource estimate, for a total of 6,061 underground drill holes and 1,522 surface drill holes (Table 14-1). An additional 46 surface diamond drill holes were completed by Skeena during the 2018 program totalling 7,737.45 m (Table 14-2).

Drill hole spacing throughout the deposit varies from 5 m, where underground production drilling encountered complex areas, to 25 m at the surface. The average drill hole spacing is approximately 10–15 m throughout the deposit. Historically, sampling at Eskay Creek was selective and primarily based on visual estimations of sulphide percent. All sample intervals sent to the laboratory were tested for gold and silver, however, lead, copper, zinc, mercury, antimony and arsenic were inconsistently sampled from one drilling campaign to the next. For underground drilling, lead, copper, zinc, mercury, antimony and arsenic were assayed when samples exceeded 8 g/t gold equivalent (AuEq; where AuEq is calculated using the formula Au+(Ag/68); Barrick, 2005).

Figure 14-1 shows the traces of all drill holes in the legacy database as well as the traces of surface drilling completed in 2018 (shown in blue).

14.4 Solid Body Modelling

14.4.1 3D Litho-Structural Model

In April 2018, Ms. Amelia Rainbow, PhD., PGeo, an independent consultant, was contracted to create the Eskay Creek litho-structural model, focusing on the area north of 8250N. The interpretation is based predominantly on historical surface and underground drill hole data. Orientated drill core, geological level plans, cross-sections and/or structural data were not available. Surface geological maps were found and made available to Ms. Rainbow part way through the modelling process. They were included into the structural interpretation where possible.

The historical database contained more than 200 individual lithology codes. Lithologies were grouped in Leapfrog Geo™ in accordance with known stratigraphy. Three main lithologies (rhyolite, contact mudstone and hanging-wall andesite) were recognized as being meaningful for resource modelling. Lithology units were further subdivided into lithology domains by one or more cross-cutting faults. Mineralization continuity was defined within these mutually-exclusive lithological domains.

Dr. Ron Uken, a Principal Structural Geologist with SRK, conducted a peer review of the 3D litho-structural model. He simplified the structural model, reducing the number of lithological domains from 25 to five, and the number of faults from 43 to five (Figure 14-2).

Table 14-1: Historical Drill Holes

Zone	No. of holes	Length (m)	Assays
ALL	7,583	651,332	427,200

Table 14-2: 2018 Drill Holes

Zone	No. of holes	Length (m)	Assays
22	5	531.2	368
21A	32	5,121.5	2,252
21C	9	2,084.75	695
Total	**46**	**7,737.45**	**3,315**

Figure 14-1: **Oblique View and Surface View of the 7,629 Diamond Drill Holes (location of the 2018 drill holes shown in blue)**

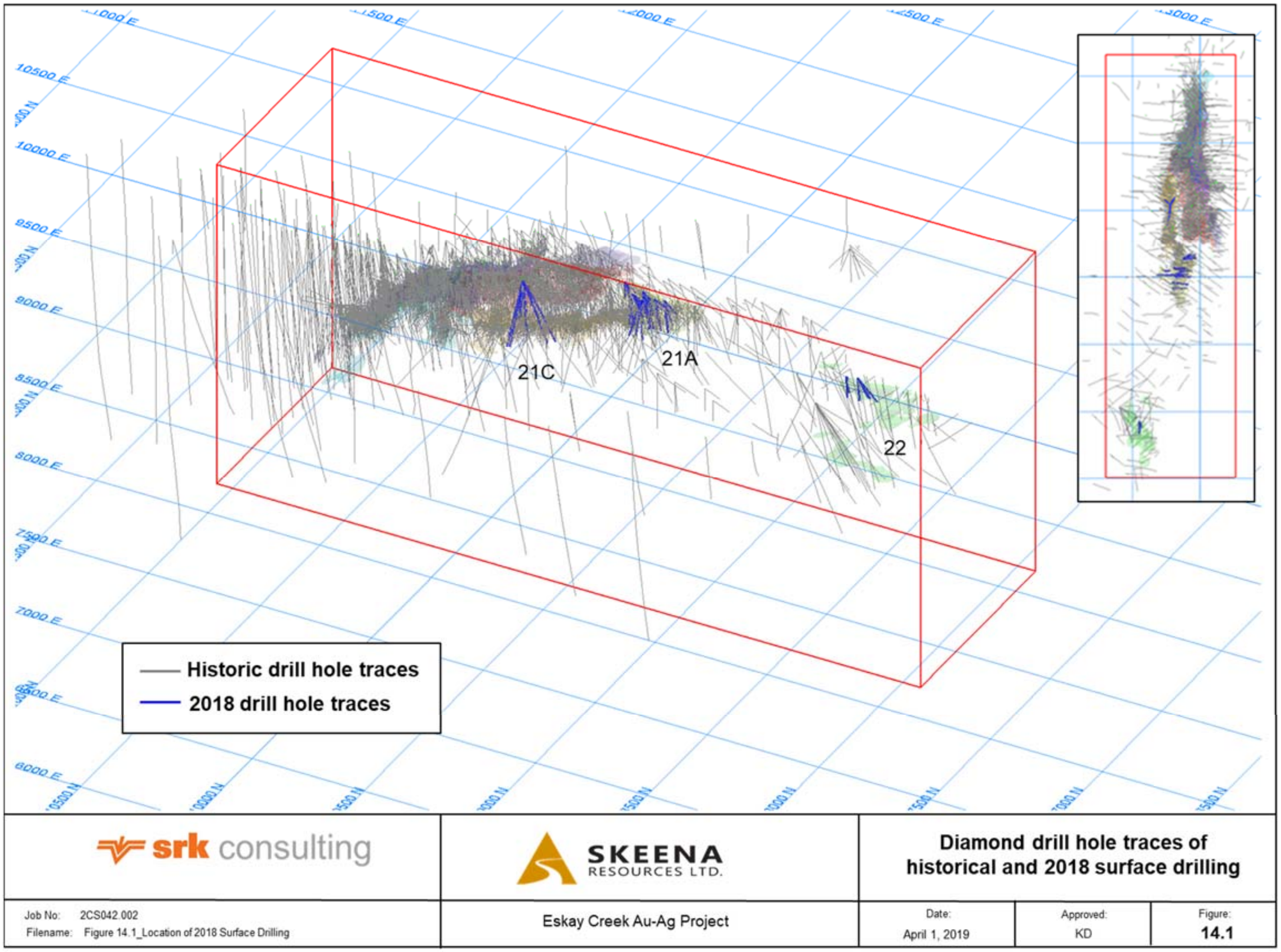
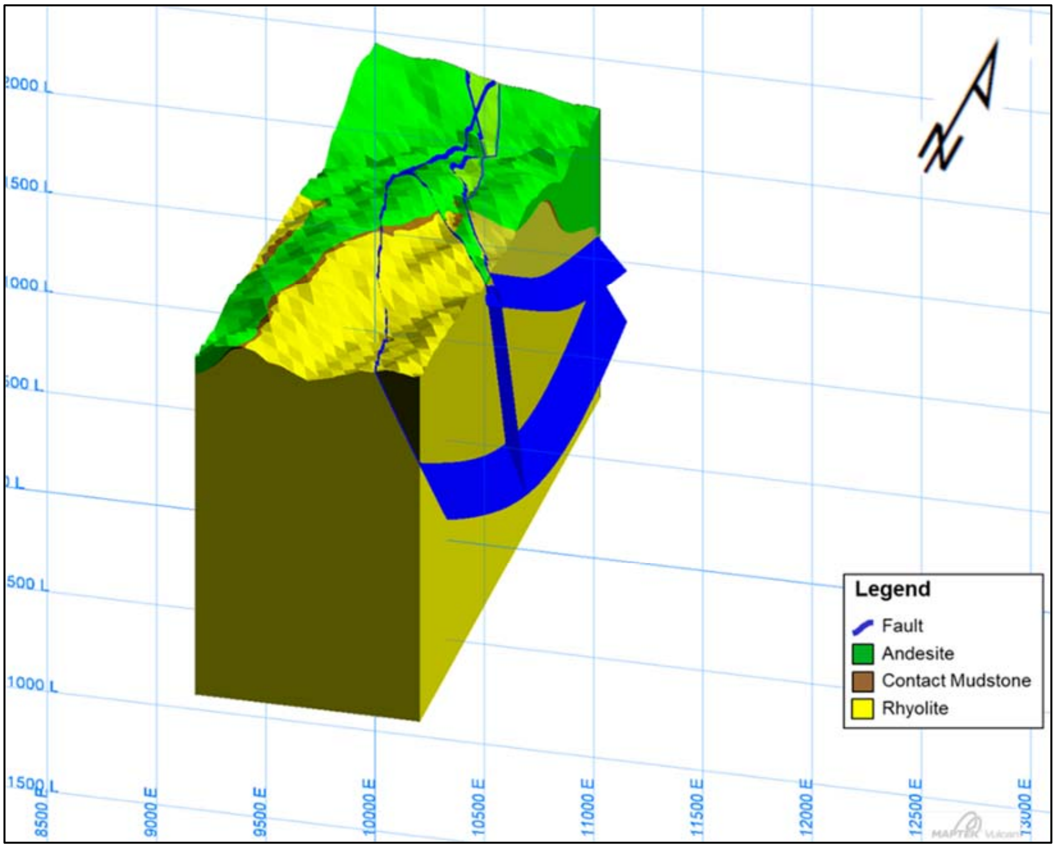


Figure 14-2: Simplified Litho-Structural Model Used to Create the 2019 Mineralization Domains



Note: Figure prepared by SRK, 2019.

14.4.2 Mineralization Domaining

The solid body modelling undertaken for the 2019 Mineral Resource estimate was updated and improved from the 2018 model. In total, 49 solids were created for the 2019 estimate; 41 mineralization solids, seven low-grade envelope solids and one solid used to limit the influence of high-grade, mined-out material.

14.4.2.1 Mineralization Domains

Forty-one mineralization solids were created to constrain the mineralization at Eskay in Leapfrog Geo™ based on the following assumptions and input parameters:

- The design was based on litho-structural domains that were subdivided using two main lithology groupings (1 – Rhyolite, and 2 - Contact Mudstone/Hangingwall Andesite);
- All holes were composited to 1 m, with left over samples at the end of the holes appended to the previous sample;
- A cut-off grade of 0.5 g/t AuEq was used to define mineralization domains, where $AuEQ = Au + (Ag/75)$;

- Wireframes were manually adjusted to include grade intervals greater than 0.5 g/t AuEq located immediately outside the margins of the mineralization solid. Five small additional wireframes were manually created;
- Wireframes were manually adjusted to remove volumes that excessively exceeded the size of the domains originally created for the 2018 estimate;
- The resultant wireframes were reviewed by SRK in section and level plan view and were deemed to be representative of the underlying geology.

The resulting mineralization solids were different from the 2018 MRE due to the following changes:

- The cut-off grade was reduced from 1.0 g/t to 0.5 g/t AuEq due to the change in mining method from an almost exclusive underground mining scenario in 2018 to a predominantly open pit mining setting in 2019;
- The gold equivalent calculation used to generate mineralization wireframes for the 2019 estimate included gold and silver only, whereas the AuEq calculation used for the 2018 mineralization wireframes included base metals as well;
- One meter down-the-hole composites were used in the 2019 estimate as opposed to 2 m composites in 2018.

For consistency, the forty-one mineralization domain solids were split and/or combined and named according to location within the previously established historical mining area zones: 22, 21A, 21C, 21B, 21Be, 21E, HW, NEX, 109 and PMP (as shown in Figure 14-3). For the purposes of this Technical Report, “domain(s)” refer to mineralization solid(s) within the historically defined mining area zones.

14.4.2.2 Low-grade Envelope Domain

In addition to the drill hole intervals contained within the mineralized domains, a significant number of drill hole intervals with grades greater than 0.5 g/t AuEq, were unaccounted for. A separate low-grade envelope was created around these intervals in the anticipated open pit area. The low-grade envelope was subdivided into seven domains based on litho-structural fault block groupings.

Figure 14-4 shows the low-grade envelope in relation to the composite assay grades higher than 0.5 g/t AuEq outside mineralization domain boundaries.

14.4.2.3 3 m Buffer Domain

Due to the high-grade nature of the mined-out areas at Eskay Creek, a 3 m buffer domain around the mined-out stopes and lifts was created. This was done to limit the smearing effect of the high-grade samples into the remaining resources areas.

Figure 14-5 is a representation of the 21B Domain showing the Contact Mudstone, Rhyolite and 3 m buffer domain used for estimation.

14.4.2.4 Solid Model Coding

Estimation domains were coded successively based on the following division scheme: (1) location within historical mining area, (2) dominant lithology type, (3) position within litho-structural domain, and (4) location within the 3 m high grade buffer zone. Table 14-3 summarizes the coding scheme used.

Figure 14-3: 2019 Mineral Resource Estimate Mineralization Domains at the Eskay Creek Project

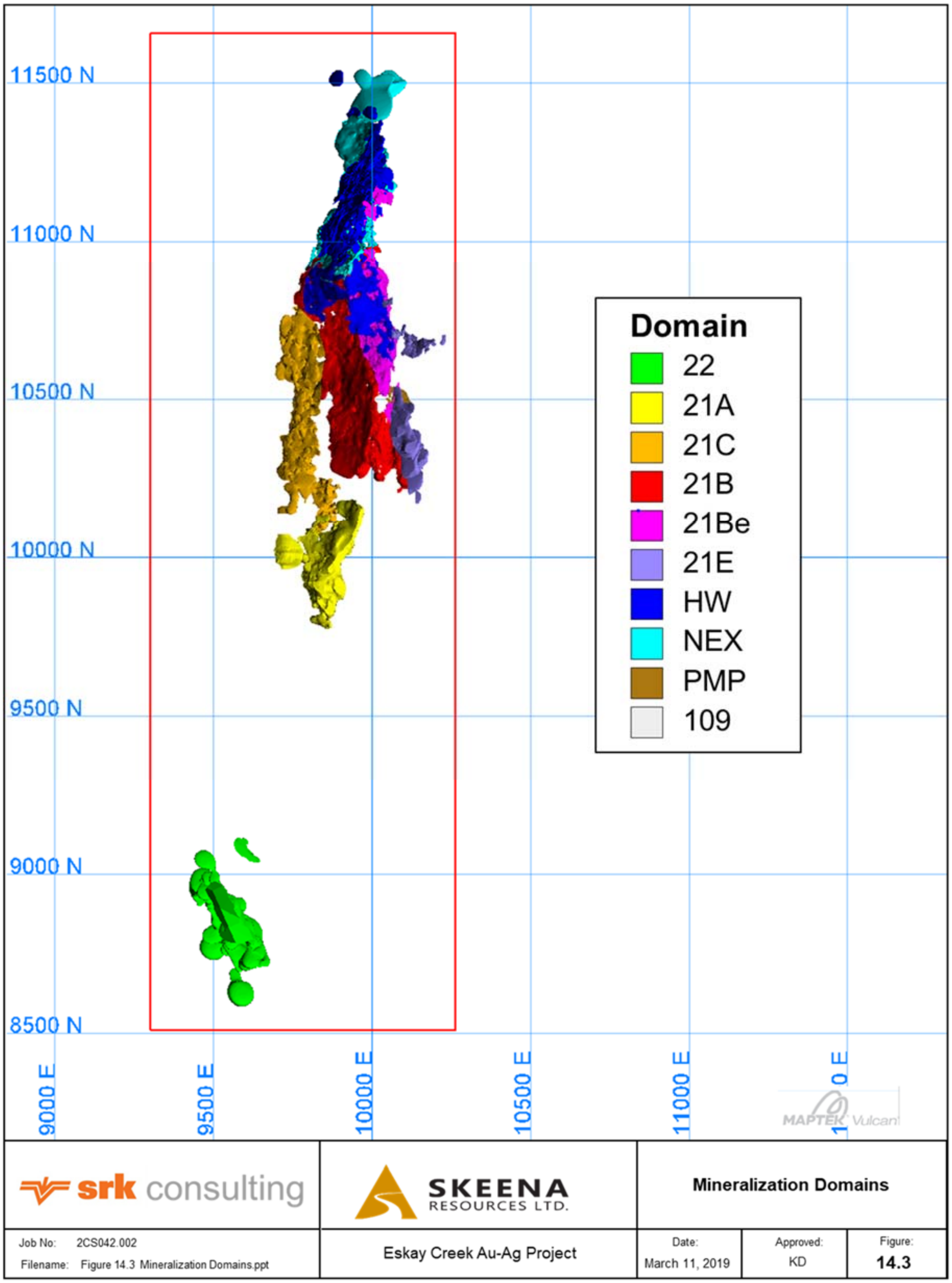


Figure 14-4: **Low-Grade Envelope Domain With 2 m Composites >2 g/t AuEq (located outside of the mineralized domains)**

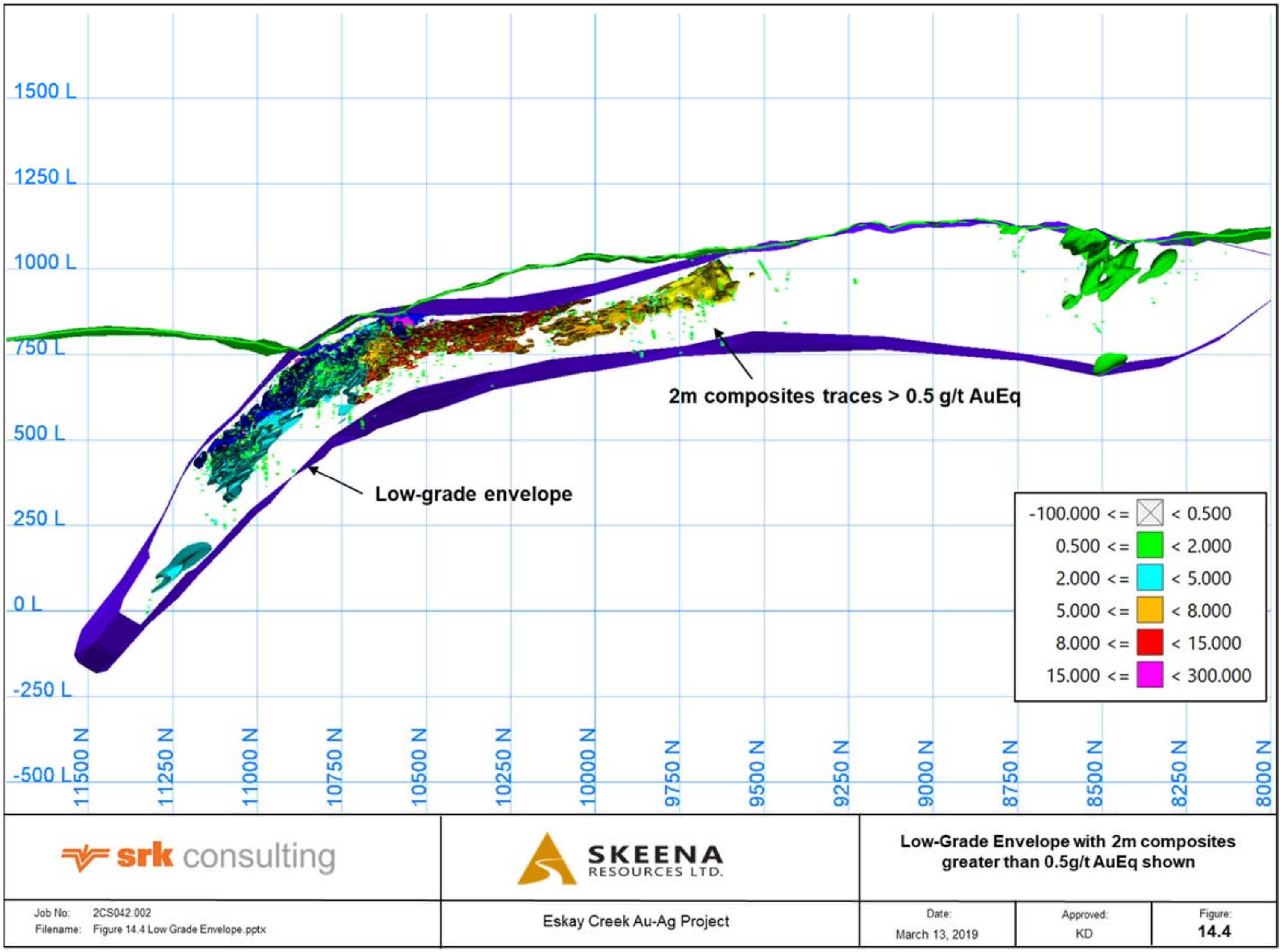


Figure 14-5: 3 m Buffer Domain Used to Constrain High-Grade (mined-out material in the 21B Domain)

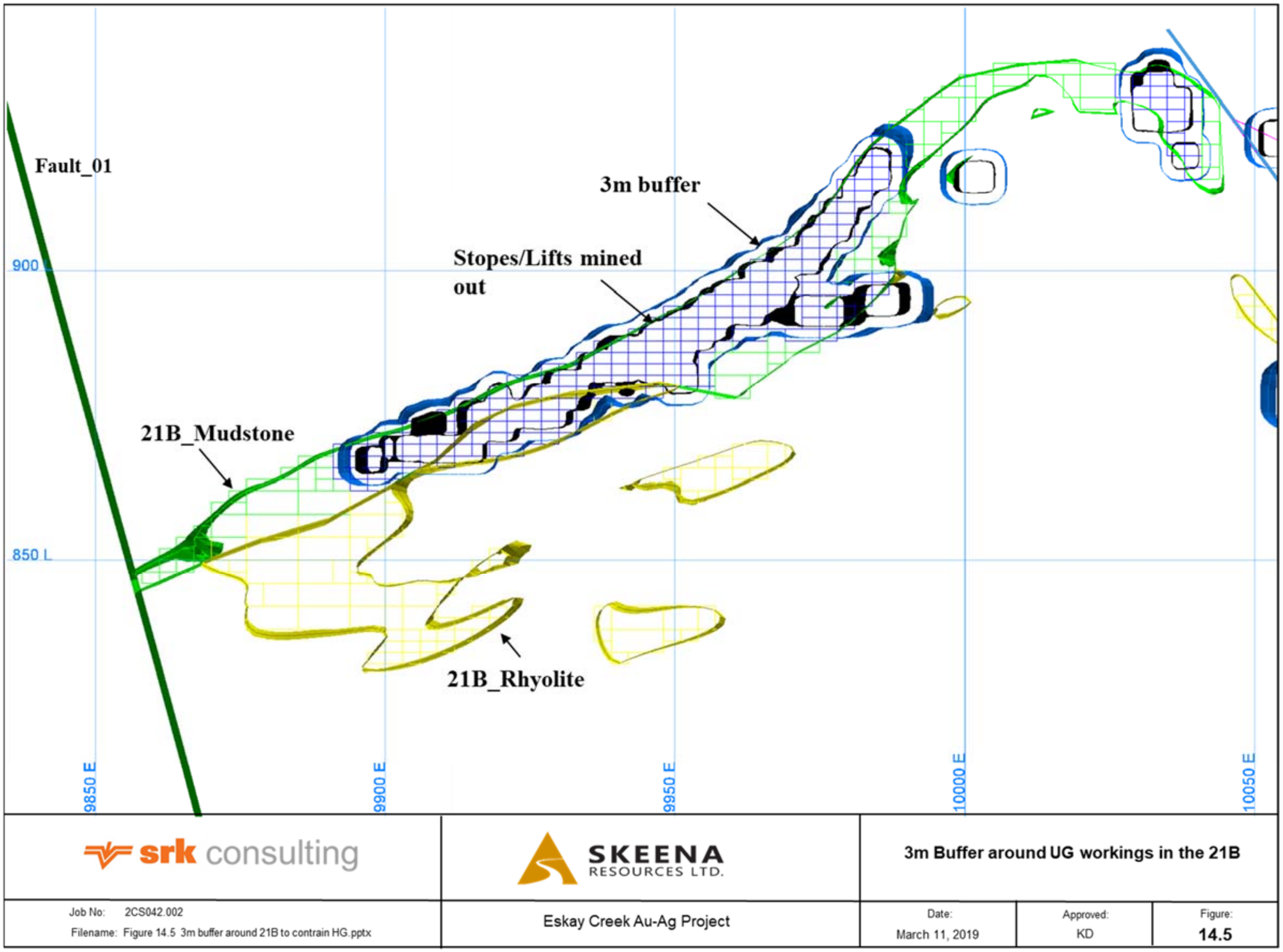


Table 14-3: Mineralization Coding Summary

Domain Name	Domain	Rock Type	Zone	Litho/Structural Domain	Est_Zone (outside buffer)	Est_Zone (inside buffer)
Low Grade Envelope	1		1	D1	1	-
			2	D2	2	-
			3	D3	3	-
			4	D3 -Rhyolite	4	-
			5	D245	5	-
			6	D6	6	-
			8	22 Zone	8	-
22	10	Rhyolite	10	D245	1000	91000
21A	20	Rhyolite	201	D245	2010	92010
		Mudstone	202	D245	2020	92020
21C	30	Rhyolite	301	D245 North	3011	93011
				D245 South	3012	93012
		Mudstone	302	D245	3020	93020
21B	40	Rhyolite	401	D245	4010	94010
		Mudstone	402		4020	94020
21Be	50	Rhyolite	501	D3	5010	95010
		Mudstone	502		5020	95020
21E	60	Rhyolite	601	D1	6010	96010
		Mudstone	602		6020	96020
HW	70	Mudstone/ Hangingwall Andeiste	702	D2	7022	97022
				D3	7023	97023
				D5	7026	97026
				D6	7025	97025
NEX	80	Rhyolite	801	wireframe	8010	98010
				D2	8012	98012
				D5	8015	98015
				D6	8016	98016
		Mudstone	802	D2	8022	98022

Domain Name	Domain	Rock Type	Zone	Litho/Structural Domain	Est_Zone (outside buffer)	Est_Zone (inside buffer)
				D5	8025	98025
PMP	95	Rhyolite	95	wireframe	9500	99500
109	99	Rhyolite	99	wireframe	9900	99900

14.4.3 Underground Workings

A complete dataset for all underground workings is available in 3D Vulcan-format. The historical underground workings are a combination of stopes, lifts and development drives. The previous operator reported that all the lifts in the stopes were backfilled with cobble, where cobble was made at the site in a batch cement plant that consisted of screened gravel from the Iskut River supplemented with 4–12% cement (Barrick, 2005).

Skeena checked the location of the underground drill holes in relation to the underground working solids and found no obvious spatial errors. Although the underground workings were routinely surveyed, there is a small measure of uncertainty in the location of the solids due to survey method limitations. As a measure of caution against possible location discrepancies and unknown ground conditions, a 1 m exclusion zone around the underground workings was employed in the open pit model to deplete the final resource estimate, and a 3 m exclusion zone around the underground workings was employed in the underground model to deplete the resources amenable to underground mining methods.

Figure 14-6 and Figure 14-7 show the underground workings used to deplete the current estimate in plan view and long section, respectively.

14.5 Data Analysis

The ZONE code item was used to code the assay file in the database for geostatistical analysis, as this split the domain into two main lithology groupings: Rhyolite and Contact Mudstone/Hangingwall Andesite (refer to Table 14-3). These coded intercepts were used to analyse sample length and generate statistics for assays and composites. Table 14-4 summarizes the statistical analysis of original assays for gold and silver.

14.6 Compositing

To minimize bias introduced by variable sample lengths, assays were composited honouring the relevant mineralization domain boundaries to 1 m lengths, for the underground model, and 2 m lengths, for the open pit model. Most samples inside the mineralization domains were collected at approximately 1 m and shorter intervals (Figure 14-8). One-meter composites were created and used for geostatistical analysis, top cutting and variography. Composite lengths that fell short of 1 m were merged into the previous sample. Summary statistics between the assays and 1 m composites are shown in Table 14-5.

A total of 161,760 one-meter composites was coded into mineralization domains, not including composites within the low-grade envelope. All gold and silver unsampled intervals were given a default value of 0.001 g/t during compositing. Missing samples due to lost core, voids or insufficient sample were ignored.

The composites were assigned codes on a majority basis corresponding to the mineralized domain, zone and estimation zone in which they occur. The compositing and coding processes were viewed in 3D to ensure that coding had been applied correctly as shown in Figure 14-9.

Figure 14-6: Plan View Of Historical Underground Mine Workings with the 1 m Buffer Applied (domain wireframes are shown for reference)

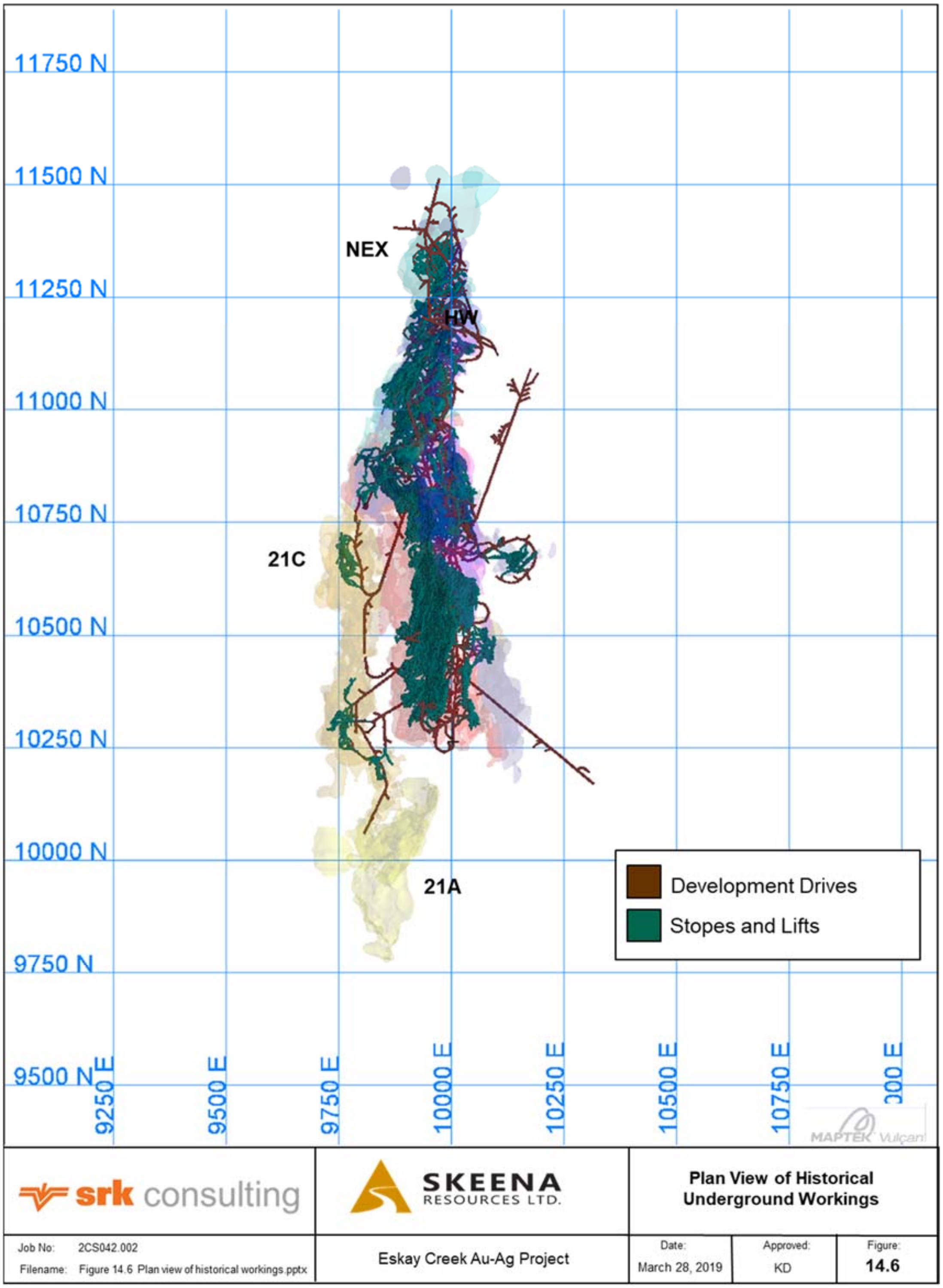


Figure 14-7: Long Section of the Historical Underground Mine Workings Looking East with the 1 m Buffer Applied (domain wireframes are shown for reference)

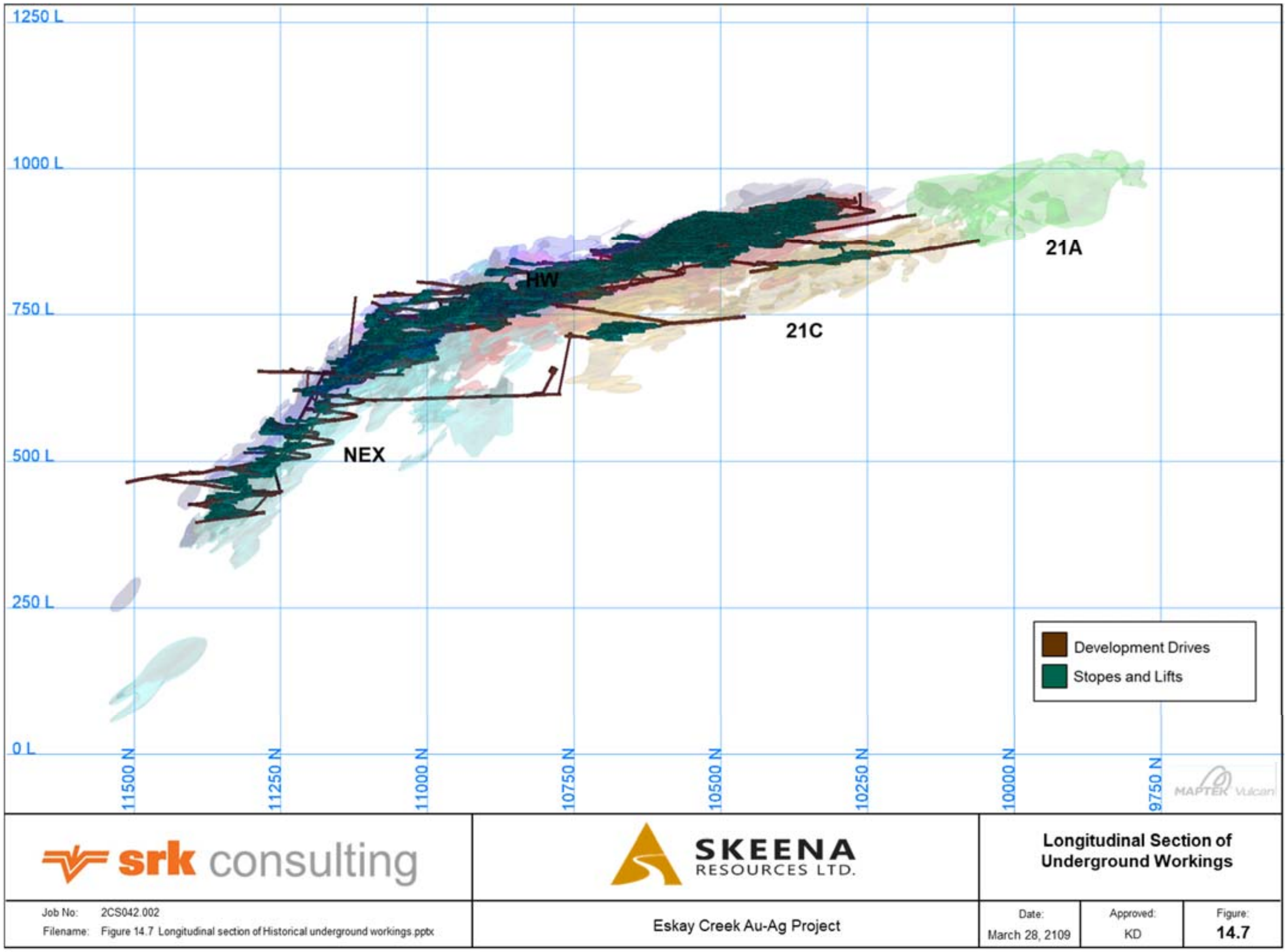
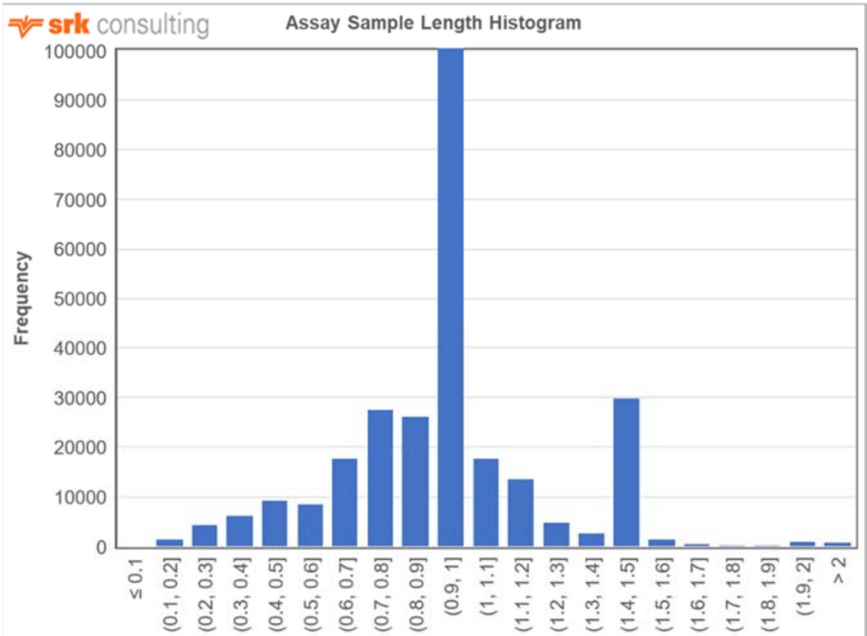


Table 14-4: Summary Statistics for Drill Hole Gold and Silver Assays by Zone

Domain	Zone	Rock Type	No. of Samples	Mean	CV	Min	Max
Gold (g/t)							
22	10	RHY	1,609	1.87	3.94	0.000	225.6
21A	201	RHY	6,086	2.91	2.91	0.001	238.0
	202	HWA/MS	1,277	16.44	2.23	0.005	677.8
21C	301	RHY	22,600	4.83	3.48	0.013	1,278.4
	302	HWA/MS	4,495	4.83	8.16	0.005	1,774.4
21B	401	RHY	19,902	6.29	12.09	0.017	9,659.0
	402	HWA/MS	16,845	29.49	3.18	0.017	6,437.9
21BE	501	RHY	13,465	10.77	4.74	0.017	1,352.7
	502	HWA/MS	8,679	20.39	4.01	0.017	2,072.7
21E	601	RHY	367	2.21	1.22	0.017	21.8
	602	HWA/MS	1,509	5.25	2.24	0.017	115.9
HW	702	HWA/MS	24,963	5.74	3.87	0.017	1,139.2
NEX	801	RHY	22,249	5.63	6.02	0.005	1,971.1
	802	HWA/MS	12,887	9.72	5.53	0.017	1,682.3
PMP	95	RHY	2,395	8.46	3.04	0.017	704.8
109	99	RHY	11,753	12.18	3.74	0.017	1,625.8
Silver (g/t)							
22	10	RHY	1,609	56.4	3.27	0.05	3,460.9
21A	201	RHY	6,086	53.3	3.92	0.05	5,628
	202	HWA/MS	1,277	199.7	5.44	0.05	22,353
21C	301	RHY	22,600	56.9	6.31	0.05	28,419
	302	HWA/MS	4,494	164.1	4.81	0.05	36,696
21B	401	RHY	19,902	277.4	5.56	0.05	44,767
	402	HWA/MS	16,845	1162.5	2.82	0.05	43,658
21BE	501	RHY	13,464	608.6	5.70	0.05	155,086
	502	HWA/MS	8,679	1063.0	3.68	0.05	54,899
21E	601	RHY	367	73.3	3.26	0.50	3,034
	602	HWA/MS	1,509	259.9	4.13	0.05	17,274
HW	702	HWA/MS	24,963	274.8	5.35	0.05	56,359
NEX	801	RHY	22,242	195.0	8.04	0.05	47,619
	802	HWA/MS	12,887	452.3	5.90	0.05	59,545
PMP	95	RHY	2,395	217.8	4.34	5.00	23,117
109	99	RHY	11,752	18.0	6.71	0.05	5,852

Figure 14-8: Histogram and Statistics of Assay Sample Lengths at Eskay Creek



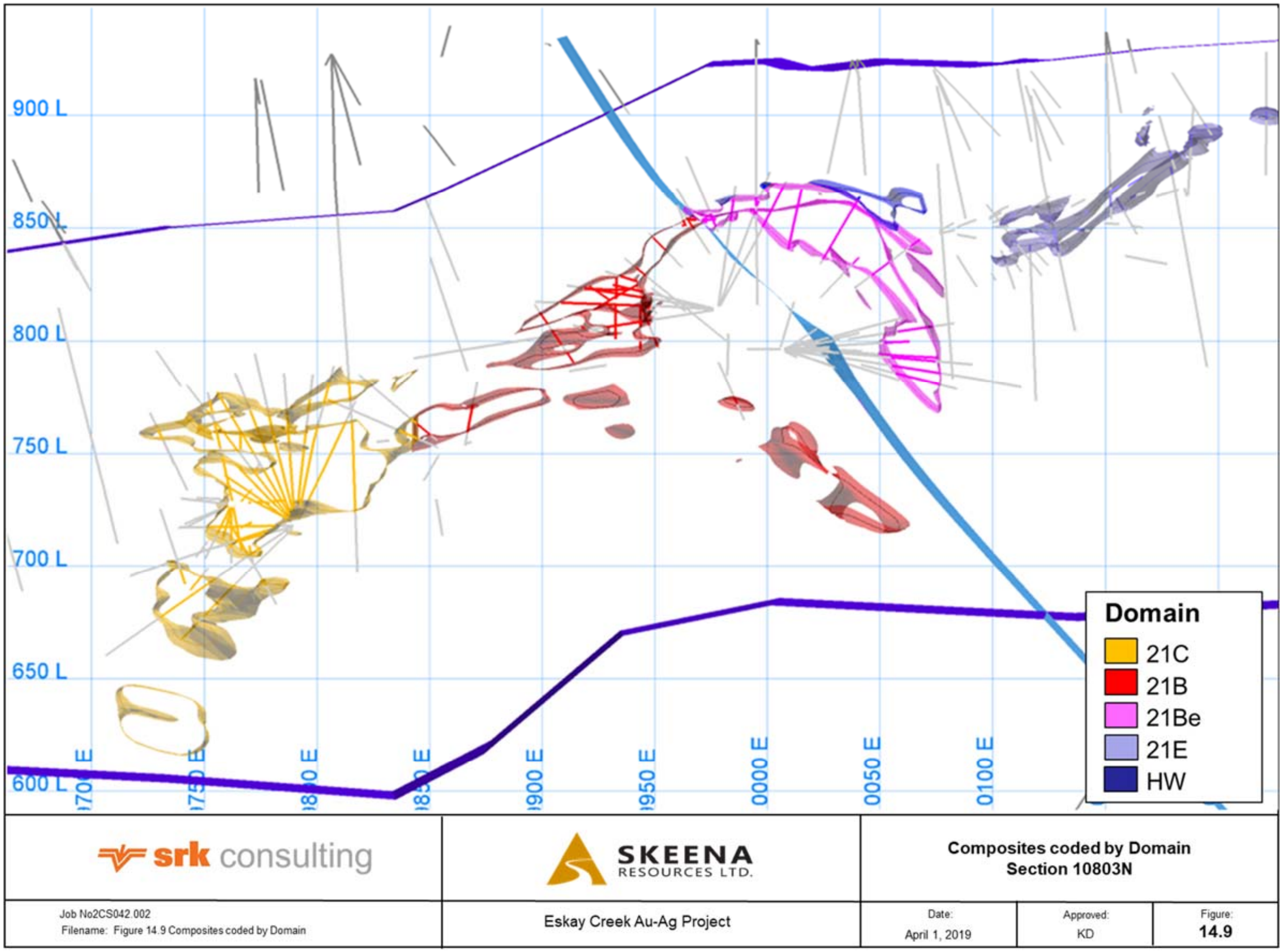
Note: Figure prepared by SRK, 2019.

Table 14-5: Comparison of Assay Data to 1 m Composites

Domain	Zone	Assays			1 m Composites		
		No. of Samples	Mean	CV	No of samples	Mean	CV
Gold							
22	10	1,609	1.87	3.94	2,071	1.7	2.93
21A	201	6,086	2.91	2.91	7,196	2.6	2.68
	202	1,277	16.44	2.23	1,419	14.0	1.99
21C	301	22,600	4.83	3.48	21,268	4.5	2.60
	302	4,495	4.83	8.16	4,056	4.3	6.46
21B	401	19,902	6.29	12.09	18,723	5.6	5.84
	402	16,845	29.49	3.18	15,815	27.6	3.12
21BE	501	13,465	10.77	4.74	12,516	10.0	4.43
	502	8,679	20.39	4.01	8,108	18.0	3.88
21E	601	367	2.21	1.22	347	2.2	1.19
	602	1,509	5.25	2.24	1,455	4.3	2.13
HW	702	24,963	5.74	3.87	22,798	5.2	3.50
NEX	801	22,249	5.63	6.02	20,678	5.0	4.94
	802	12,887	9.72	5.53	11,678	8.6	5.24
PMP	95	2,395	8.46	3.04	2,316	7.7	2.65
109	99	11,753	12.18	3.74	11,316	11.3	3.56
Sub-total		***171,081***			***161,760***		

Domain	Zone	Assays			1 m Composites		
		No. of Samples	Mean	CV	No of samples	Mean	CV
Silver							
22	10	1,609	56.39	3.27	2,071	52.3	2.71
21A	201	6,086	53.26	3.92	7,196	49.4	3.51
	202	1,277	199.69	5.44	1,419	172.0	5.05
21C	301	22,600	56.87	6.31	21,268	52.5	4.59
	302	4,494	164.13	4.81	4,056	142.9	3.51
21B	401	19,902	277.45	5.56	18,723	254.0	5.24
	402	16,845	1162.53	2.82	15,815	1101.3	2.76
21BE	501	13,464	608.59	5.70	12,516	546.7	5.34
	502	8,679	1062.97	3.68	8,108	929.9	3.57
21E	601	367	73.31	3.26	347	74.9	3.28
	602	1,509	259.89	4.13	1,455	201.0	3.68
HW	702	24,963	274.82	5.35	22,798	240.7	4.58
NEX	801	22,242	194.99	8.04	20,678	162.6	7.82
	802	12,887	452.33	5.90	11,678	402.7	5.90
PMP	95	2,395	217.76	4.34	2,316	199.4	3.91
109	99	11,752	17.96	6.71	11,316	16.0	5.86
Sub-total		***171,071***			***161,760***		

Figure 14-9: 1 m Composites Coded by Domain



14.7 Evaluation of Outliers

14.7.1 1 m Composites

Block grade estimates may be overly affected by very high-grade assays therefore capping was applied to all domains. An analysis of sample lengths versus gold grade shows that effort was taken to sample intervals based on visible mineralization, since gold grades are highest in the smallest assay lengths (Figure 14-10). For this reason, capping was applied after compositing. Capping values were selected on a zone by zone basis using the results from log probability plots, histograms, co-efficient of variation values and percent metal loss. Less than 1% of the entire assay data set was capped for high-grade outliers.

To assess the impact on capping and % metal lost, preliminary ordinary kriged (OK) block models were run using, (1) capped and, (2) uncapped 1 m composite data within each zone (Table 14-6). Percent metal loss was variable between zones, ranging from as little as 0.5% to as high as 14.6% for gold, and 0.4% to 13.3% for silver. For domains with percent metal loss of more than 5%, the uncapped mean values were sensitive to the extremely high-grade samples. On average, less than 3% gold and 5% silver were lost during the process of capping.

14.7.2 2 m Composites

For the open pit model, 2 m composites were used. The capping values established from the 1 m composites were used for the 2 m composites, except for Zones 401 and 402 where lower capping values of 300 g/t Au were applied. In addition, top cuts were reduced to 40 g/t outside the 3 m buffer in Zones 401 and 402. Statistics for the uncapped and capped 2 m composites are shown below in Table 14-7.

A low-grade envelope was created for the open pit model and capping values in the envelope were determined from 2 m composite statistics. Table 14-8 shows the capping values and statistics for the 2 m composites in the low-grade envelope.

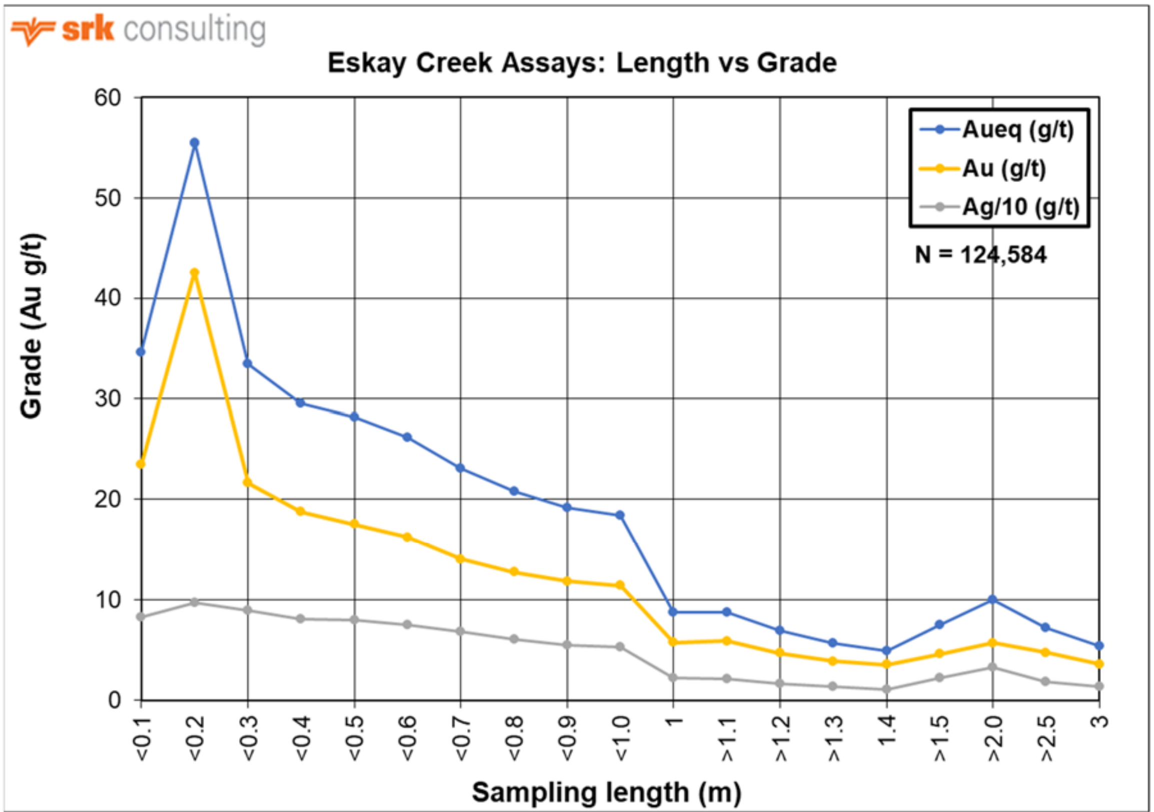
14.8 Variography

Variograms were used to assess for grade continuity, spatial variability in the estimation domains, sample search distances and kriging parameters.

All variograms were prepared using 1 m composites. Three zones encompass both limbs of the anticline: 21B, 21E and NEX. These domains were split into east and west limbs and sample pairs from the more continuous western limbs were selected for variography analysis.

Spatial continuity was assessed using variogram maps and 3D representations of grade continuity. The most suitable orientation was selected based on the general understanding of the attitude of each mineralized zone. Initially, the variograms were produced on normal scores of the composite assay grades. Downhole variograms were calculated to characterize the nugget effect. Final variogram models on original gold and silver assays were designed from the variograms on normal scores. Spherical variogram models used for determining grade continuity are summarized in Table 14-9 for gold and Table 14-10 for silver. Figure 14-11 illustrates gold search ellipsoids showing ranges used for dynamic anisotropy.

Figure 14-10: Gold Grade Versus Sample Length



Note: Figure prepared by SRK, 2019.

Table 14-6: Gold and Silver Assay Capped Grades Per Zone

Domain	Zone	Top Cut	No. Cut	% Cut	Preliminary Block Model Uncapped	Preliminary Block Model Capped	% Metal Lost
Gold							
22	10	45	5	0.24	1.69	1.67	1.4
21A	201	80	12	0.17	2.33	2.28	1.9
	202	150	8	0.56	11.94	11.66	2.4
21C	301	100	19	0.09	3.39	3.31	2.4
	302	150	9	0.22	3.52	3.01	14.6
21B	401	500	16	0.09	4.61	4.50	2.4
	402	700	13	0.08	25.60	25.47	0.5
21Be	501	600	15	0.12	7.05	6.93	1.7
	502	900	10	0.12	15.82	15.52	1.8
21E	601	10	7	2.02	1.94	1.91	1.7
	602	60	8	0.55	3.48	3.43	1.5
NEX	702	300	12	0.05	4.08	3.97	2.6
HW	801	400	21	0.10	4.02	3.86	3.9

Domain	Zone	Top Cut	No. Cut	% Cut	Preliminary Block Model		% Metal Lost
					Uncapped	Capped	
	802	600	15	0.13	6.53	6.26	4.2
PMP	95	100	11	0.47	6.14	5.76	6.1
109	99	500	14	0.12	9.48	9.10	4.0
Sub-total			***195***				***3.3***
Silver							
22	10	990	11	0.53	52.44	51.37	2.0
21A	201	1,950	12	0.17	50.78	49.62	2.3
	202	7,000	7	0.49	131.72	120.41	8.6
21C	301	2,100	36	0.17	37.20	35.90	3.5
	302	3,400	16	0.39	117.86	113.65	3.6
21B	401	15,000	34	0.18	227.45	219.63	3.4
	402	23,000	22	0.14	1,002.05	998.01	0.4
21Be	501	30,000	18	0.14	331.28	322.42	2.7
	502	20,000	58	0.72	691.58	658.59	4.8
21E	601	600	7	2.02	57.13	49.53	13.3
	602	5,000	12	0.82	145.63	139.67	4.1
NEX	702	16,000	27	0.12	183.27	177.49	3.2
HW	801	20,000	26	0.13	115.29	111.26	3.5
	802	30,000	17	0.15	276.58	268.52	2.9
PMP	95	4,600	11	0.47	152.12	137.98	9.3
109	99	1,500	11	0.10	19.12	17.24	9.8
Sub-total			***325***				***4.8***

Table 14-7: Summary Statistics for 2 m Capped and Uncapped Composites by Zone

Domain	Zone	# Samples	Top cut	Uncapped		Capped	
				Mean	CV	Mean	CV
Gold							
22	10	1,033	45	1.69	2.93	1.63	2.11
21A	201	3,599	80	2.61	2.68	2.57	2.20
	202	703	150	13.99	1.99	13.80	1.75
21C	301	10,659	100	4.53	2.60	4.43	1.66
	302	2,028	150	4.29	6.46	3.72	2.82
21B	401	9,370	300	5.56	5.84	5.01	3.70
	402	7,930	300	27.58	3.12	25.37	2.18
21Be	501	6,268	600	10.01	4.43	9.99	3.90
	502	4,059	900	17.97	3.88	17.77	3.35
21E	601	601	10	2.20	1.19	1.96	1.03
	602	378	60	4.33	2.13	6.45	1.63
NEX	702	11,367	300	5.23	3.50	5.17	2.62
HW	801	10,328	400	5.00	4.94	4.91	3.81

Domain	Zone	# Samples	Top cut	Uncapped		Capped	
				Mean	CV	Mean	CV
	802	5,860	600	8.55	5.24	8.45	4.13
PMP	95	1,152	100	7.73	2.65	7.27	1.61
109	99	5,667	500	11.30	3.56	11.00	2.63
Total		***81,002***					
Silver							
22	10	1,033	990	52.30	2.71	50.94	2.30
21A	201	3,599	1,950	49.36	3.51	49.11	2.88
	202	703	7,000	172.02	5.05	163.91	3.78
21C	301	10,659	2,100	52.48	4.59	50.51	3.33
	302	2,028	3,400	142.89	3.51	134.01	2.62
21B	401	9,370	15,000	254.03	5.24	244.43	4.70
	402	7,930	23,000	1101.33	2.76	1094.98	2.58
21Be	501	6,268	30,000	546.73	5.34	521.46	4.04
	502	4,059	20,000	929.90	3.57	890.13	3.02
21E	601	530	600	74.86	3.28	68.16	2.90
	602	378	5,000	201.00	3.68	303.35	2.63
NEX	702	11,367	16,000	240.65	4.58	240.01	3.87
HW	801	10,328	20,000	162.56	7.82	158.05	6.32
	802	5,860	30,000	402.73	5.90	375.92	5.03
PMP	95	1,152	4,600	199.45	3.91	178.31	2.49
109	99	5,667	1,500	15.95	5.86	15.02	3.98
Total		***80,931***					

Note: * based on composites that are not declustered

Table 14-8: Capping Values in the Low-Grade Envelope by Zone

Domain	Zone	# Samples	Au cap (top cut)	Metal lost by capping (%)	Max Value	CV		Mean		Samples Cut
						Uncapped	Capped	Uncapped	Capped	
Gold										
1	1	15,732	4.6	34.3	324.10	21.45	3.97	0.134	0.088	33
	2	18,516	7	13.1	81.96	7.80	4.33	0.145	0.126	38
	3/4	13,640	30	13.0	266.00	10.87	5.21	0.316	0.275	15
	5	99,139	10	15.7	371.49	9.21	3.32	0.242	0.204	184
	6	15,625	7	12.7	73.96	5.53	2.99	0.212	0.185	32
	8	11,977	3	3.3	13.85	4.13	3.31	0.061	0.059	12
Sub-total Capped										***314***
Silver										
1	1	15,732	300	42.7	10,305	21.24	5.72	4.944	2.835	28
	2	18,516	200	28.8	2,217	13.34	6.12	2.556	1.821	34

Domain	Zone	# Samples	Au cap (top cut)	Metal lost by capping (%)	Max Value	CV		Mean		Samples Cut
						Uncapped	Capped	Uncapped	Capped	
	3/4	13,640	2000	21.7	17,904	13.93	7.04	17.234	13.502	13
	5	99,139	900	19.7	10,482	15.82	7.36	5.258	4.221	58
	6	15,625	400	22.9	4,397	10.98	4.66	6.442	4.967	26
	8	11,977	60	7.5	378	5.70	3.69	1.055	0.976	14
Sub-total Capped										***173***

Note: * % metal loss equals (mean – meanCap)/mean*100 where mean is the average grade of the assays before capping and meanCap is the average grade of assays after capping.

Table 14-9: Variogram Parameters for Gold by Estimation Zone

Vario	Est_Zone	Structure	Nugget	Sill	Major (y)	Semi (x)	Minor (z)	Final Rotation (yxz)
10	1000	1	0.15	0.64	15	25	10	149/-14/33
		2		0.21	50	35	40	
201	2010	1	0.21	0.56	7	5	17	13/-15/27
		2		0.14	49	43	20	
		3		0.1	76	43	20	
202	2020	1	0.10	0.72	17	10	12	23/-7/45
		2		0.18	45	20	20	
3011	3011	1	0.20	0.37	4	4	8	352.5/-9.8/28.5
		2		0.26	18	6	10	
		3		0.17	35	20	15	
3012	3012	1	0.11	0.56	8	5	5	10/0/-60
		2		0.33	45	20	20	
302	3020	1	0.18	0.76	6	6	6	12/-10/23
		2		0.06	45	25	11	
4011*	4010	1	0.19	0.71	5	5	3	168.0/12.2/-27.6
		2		0.06	10	10	25	
		3		0.04	30	15	25	
4021*	4020	1	0.12	0.67	7	4	4	4.4/-12.7/38.3
		2		0.2	95	60	5	
501	5010	1	0.23	0.64	5	4	3	0/-22/45
		2		0.13	35	20	10	
502	5020	1	0.09	0.81	5	2	5	174.3/26.1/24.2
		2		0.1	25	10	6	
601	6010	1	0.09	0.58	24	9	3	30/0/30
		2		0.33	40	35	15	
6021*	6020	1	0.05	0.38	7	7	5	69.9/35.4/45.3
		2		0.57	25	15	5	
7023	7023	1	0.18	0.68	10	5	3	160/0/40

Vario	Est_Zone	Structure	Nugget	Sill	Major (y)	Semi (x)	Minor (z)	Final Rotation (yxz)
		2		0.15	35	30	15	
7025	7025	1	0.05	0.24	5	5	5	260/-35/0
		2		0.71	20	15	10	
70261*	7026	1	0.16	0.66	5	7	7	15.2/-45.2/35.5
		2		0.18	40	40	25	
8012	8012	1	0.19	0.69	12	8	6	10/-42/39
		2		0.12	60	40	30	
8016	8016	1	0.38	0.55	13	12	6	9.1/-48/32
		2		0.07	30	30	20	
8025	8025	1	0.15	0.81	4	4	3	41.9/-21.5/57.5
		2		0.04	35	22	10	
8022	8022	1	0.17	0.72	8	7	7	10/-42/39
		2		0.11	42	20	10	
95	9500	1	0.12	0.74	12	8	8	350/-26/-44
		2		0.14	40	20	10	
99	9000	1	0.37	0.51	6	3	7	296/-54/172
		2		0.12	45	20	20	

Note: * based on western limb

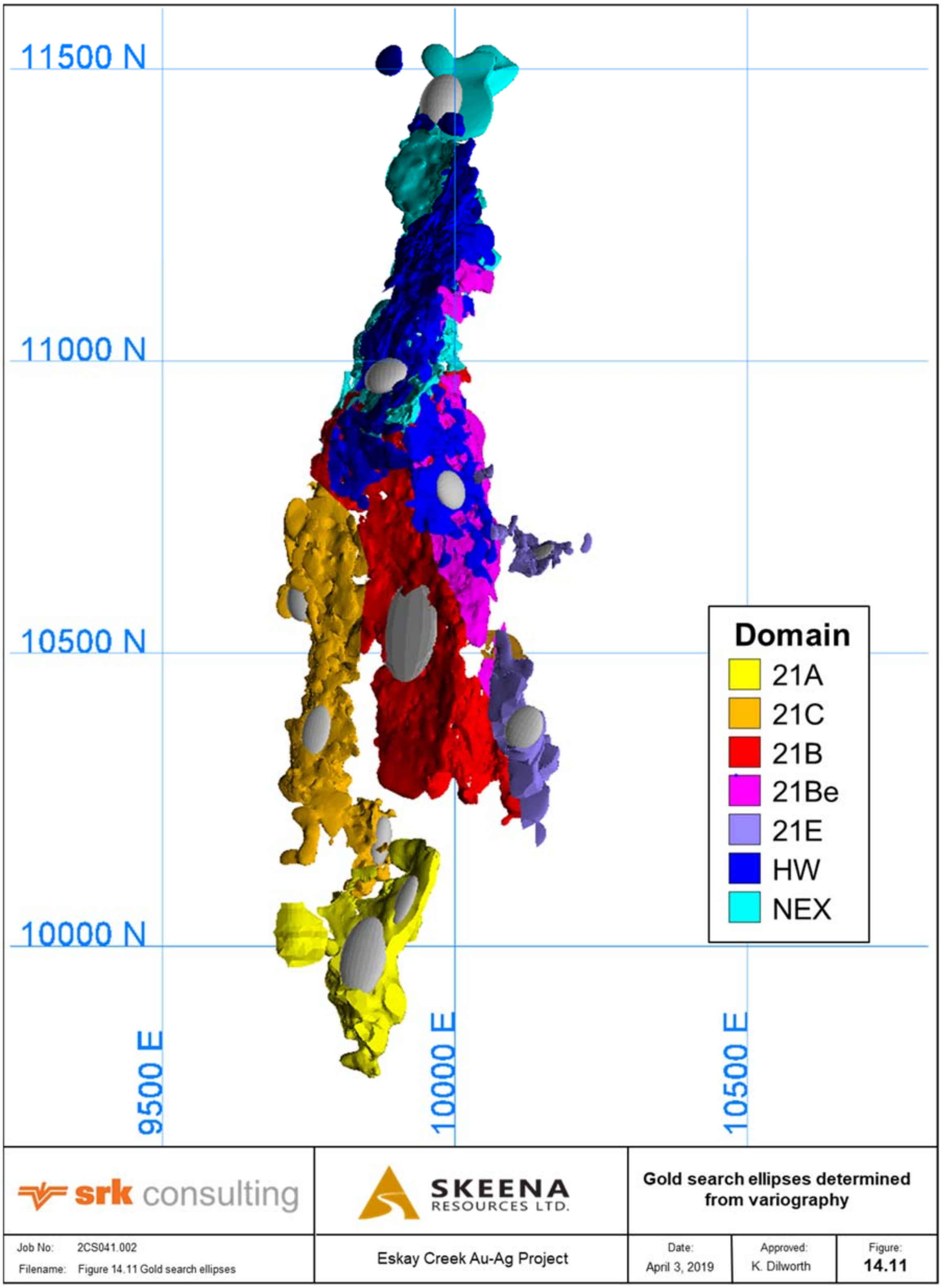
Table 14-10: Variogram Parameters for Silver by Estimation Zone

Vario	Zone	Structure	Nugget	Sill	Major (y)	Semi (x)	Minor (z)	Final Rotation (yxz)
10	1000	1	0.05	0.79	14	14	10	149/-14/33
		2		0.15	20	20	10	
201	2010	1	0.16	0.73	17	15	18	13/-15/27
		2		0.11	80	40	20	
202	2020	1	0.09	0.76	5	10	10	23/-7/45
		2		0.15	17	17	17	
3011	3011	1	0.24	0.68	8	7	10	352.5/-9.8/28.5
		2		0.08	40	30	30	
3012	3012	1	0.11	0.77	14	8	8	10/0/-60
		2		0.11	40	30	25	
302	3020	1	0.13	0.68	11	9	9	12/-10/23
		2		0.19	60	40	20	
4011*	4011	1	0.25	0.57	6	8	5	168.0/12.2/-27.6
		2		0.18	30	16	20	
4021*	4020	1	0.06	0.68	8	4	5	4.4/-12.7/38.3
		2		0.26	65	60	10	
501	5010	1	0.61	0.3	13	10	5	0/-22/45
		2		0.09	45	20	10	
502	5020	1	0.07	0.84	7	5	6	174.3/26.1/24.2

Vario	Zone	Structure	Nugget	Sill	Major (y)	Semi (x)	Minor (z)	Final Rotation (yxz)
		2		0.08	40	20	10	
601	6010	1	0.10	0.74	10	10	3	30/0/30
		2		0.16	25	30	10	
6021*	6020	1	0.04	0.73	20	20	20	69.9/35.4/45.3
		2		0.23	25	23	23	
7023	7023	1	0.10	0.84	4	4	2	160/0/40
		2		0.06	30	20	5	
7025	7025	1	0.13	0.69	10	5	4	260/-35/0
		2		0.18	35	20	4	
70261*	7026	1	0.19	0.64	8	5	5	15.2/-45.2/35.5
		2		0.09	28	45	22	
		3		0.07	120	50	25	
8012	8012	1	0.22	0.68	6	5	5	10/-42/39
		2		0.1	65	24	13	
8016	8016	1	0.37	0.56	6	7	4	9.1/-48/32
		2		0.07	55	23	12	
8025	8025	1	0.12	0.73	8	8	4	41.9/-21.5/57.5
		2		0.15	40	40	10	
8022	8022	1	0.11	0.8	8	6	6	10/-42/39
		2		0.09	50	20	10	
95	9500	1	0.16	0.72	6	5	4	350/-26/-44
		2		0.12	25	25	20	
99	9900	1	0.42	0.55	6	6	8	296/-54/172
		2		0.03	20	20	20	

Note: * based on western limb.

Figure 14-11: Gold Search Ellipses (in grey) Determined by Variography



14.9 Dynamic Anisotropy

Due to the folded nature of the deposit, search ellipsoid orientations were not considered suitable for effectively estimating all mineral domains. Dynamic anisotropy was selected as the preferred estimation method because adjustments in each block could be made in relation to the presiding mineralization trend. The anisotropy direction was defined from the base of the Contact Mudstone (see example in Figure 14-12).

14.10 Specific Gravity

Specific gravity was determined for 312 samples collected during 1996 and 1997 from the 21B, 21C, 21E, NEX and HW Zones. SG measurements were collected from 10 cm-long split or whole diamond drill core. The core was first weighed in air on a beam balance, then weighed in water. The volume of the core was calculated which was then used to calculate the SG.

During the 2018 drilling program, an additional 355 measurements were taken from the 22, 21A, and 21C Zones.

Due to the limited number of specific gravity measurements taken, an empirical bulk density formula was derived using lead, zinc, copper and antimony grades and verified against actual measurements. The empirical density equation determined from the previous operator is:

$SG = (Pb + Zn + Cu + Sb) * 0.03491 + 2.67$ (where all metals are reported in %).

A default of 2.67 was applied to missing SG in blocks; historically this value was used to represent the average SG value of unmineralized rhyolite and mudstone host rocks.

A comparison of the calculated specific gravity using the empirical formula versus actual specific gravity measurements is presented in (Figure 14-13).

Skeena appropriated the empirical density equation defined by the previous Operator and used it without modification for the 2019 resource estimate. Lead, zinc, copper and antimony were first interpolated into blocks using inverse distance to the second power methodology (ID^2), and then densities in the block model were assigned. The ID^2 estimates used the same estimation parameters as those applied for gold; however, the 3 m buffer was not used.

14.11 Block Model and Grade Estimation

The grade estimate for the 2019 estimate was constructed in two stages: (1) open pit modelling and, (2) underground modelling. For the open pit model, grades were estimated in 10 mineralization domains, and seven low-grade envelope domains. Three estimation domains below the bottom of the optimized resource pit were reported as resources potentially amenable to underground mining methods. Each of the models were optimized based on the defining mining scenario, and the separate methodologies and parameters are described below.

14.11.1 Open Pit Model

The block model geometry and extents used for grade estimation in the open pit model are summarized in Table 14-11.

OK was used to estimate gold and silver in all domains, except for the low-grade envelope domain where ID^2 interpolation was selected. Two-metre capped composites were used for the open pit model. Gold and silver grades within the mineralization domains were estimated in two successive passes with increasing search radii based on variogram ranges as outlined in Table 14-12 and Table 14-13.

Figure 14-12: Dynamic Anisotropy Vectors Used in the Folded 21B Domain (looking north)

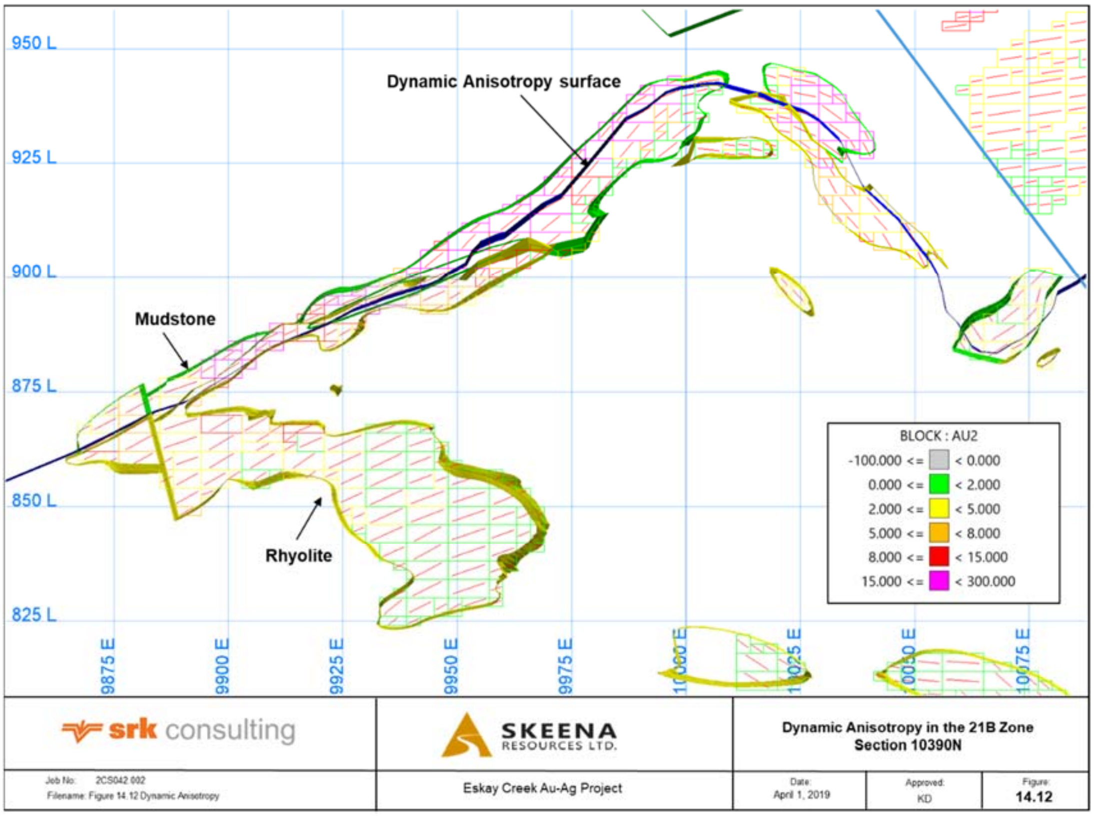


Figure 14-13: Measured Versus Calculated SG by Empirical Formula

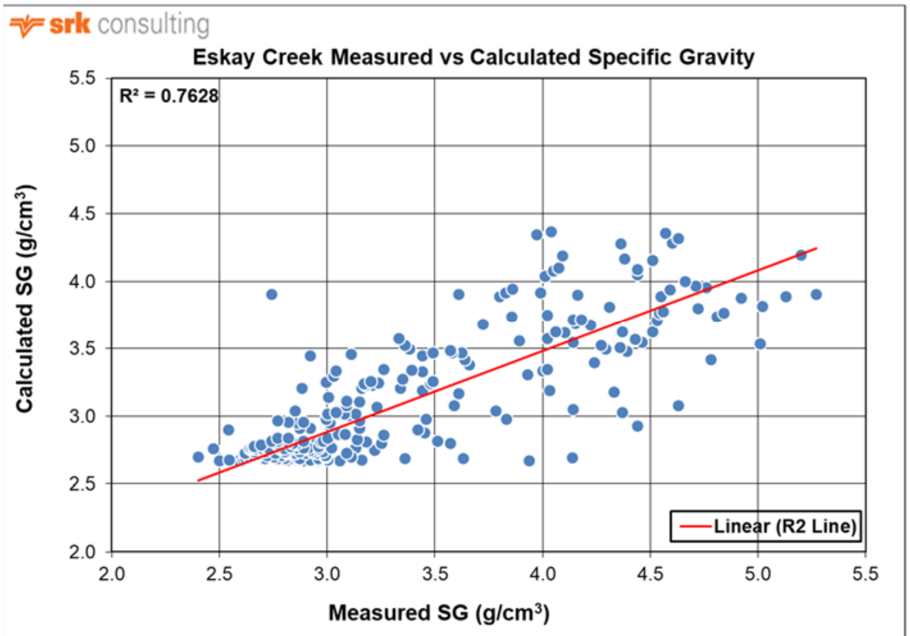


Table 14-11: Details of the Open Pit Block Model Dimensions and Block Size

	Bearing	Plunge	Dip	Start Offset			End Offset			Block Size		
				X	Y	Z	X	Y	Z	X	Y	Z
Parent	90	0	0	9300	8508	-50	963	3150	1500	9	9	4
Sub-block	90	0	0	9300	8508	-50	963	3150	1500	3	3	2

Table 14-12: Gold Grade Estimation Parameters by Estimation Zone

Est_Zone	Rock Type	Search Pass	Orientation	Gold Search Radii			Number of Composites		Max Composites per Drill Hole	Boundary
				X	Y	Z	Min	Max		
1	Predominantly Rhyolite	1	D1	25	10	5	3	10	2	HARD
2		1	D2	20	10	5	3	10	2	
3		1	D3	12.5	5	3	3	10	2	
4		1	D4	12.5	5	5	3	10	2	
5		1	D245	22.5	10	7.5	3	10	2	
6		1	D6	17.5	17.5	10	3	10	2	
8		1	Zone 22	25	17.5	15	3	10	2	
1000	Rhyolite	1	134/0/45	22.5	15.8	18	5	15	2	HARD
		2		50	35	40	3	15	2	
2010	Rhyolite	1	D245	34.2	19.4	9	5	15	2	SOFT
		2		76	43	20	3	15	2	2010
2020	Mudstone	1	D245	20.3	9	9	5	15	2	SOFT
		2		45	20	20	3	15	2	2020

Est_Zone	Rock Type	Search Pass	Orientation	Gold Search Radii			Number of Composites		Max Composites per Drill Hole	Boundary
				X	Y	Z	Min	Max		
3011	Rhyolite	1	D245 North	15.8	9	6.8	5	15	2	SOFT
		2		35	20	15	3	15	2	3020
3012	Rhyolite	1	D245 South	20.3	9	9	5	15	2	SOFT
		2		45	20	20	3	15	2	3020
3020	Mudstone	1	D245	20.3	11.3	5	5	15	2	SOFT
		2		45	25	11	3	15	2	3011, 3012
4010	Rhyolite	1	D245	13.5	6.8	11.3	5	15	2	SOFT
		2		30	15	25	3	15	2	4020
4020	Mudstone	1	D245	42.8	27	2.3	5	15	2	SOFT
		2		95	62	5	3	15	2	4010
5010	Rhyolite	1	D3	15.8	9	4.5	5	15	2	HARD
		2		35	20	10	3	15	2	
5020	Mudstone	1	D3	11.3	4.5	2.7	5	15	2	HARD
		2		25	10	6	3	15	2	
6010	Rhyolite	1	D1	18	15.8	6.8	5	15	2	HARD
		2		40	35	15	3	15	2	
6020	Mudstone	1	D1	11.3	6.8	2.3	5	15	2	HARD
		2		25	15	5	3	15	2	
7022	Mudstone	1	D2	9	6.8	4.5	5	15	2	HARD
		2		20	15	10	3	15	2	
7023		1	D3	15.8	13.5	6.8	5	15	2	HARD
		2		35	30	15	3	15	2	
7025		1	D5	9	6.8	4.5	5	15	2	HARD
		2		20	15	10	3	15	2	
7026		1	D6	18	18	11.3	5	15	2	HARD
		2		40	40	25	3	15	2	
8010	Rhyolite	1	020/0/80	27	18	13.5	5	15	2	HARD
		2		60	40	30	3	15	2	
8012		1	D2	27	18	13.5	5	15	2	SOFT 8022
		2		60	40	30	3	15	2	
8015		1	D5	27	18	13.5	5	15	2	SOFT
		2		60	40	30	3	15	2	8025
8016		1	D6	13.5	13.5	9	5	15	2	HARD
		2		30	30	20	3	15	2	
8022	Mudstone	1	D2	18.9	9	4.5	5	15	2	SOFT 8012
		2		42	20	10	3	15	2	
8025		1	D5	15.8	9.9	4.5	5	15	2	

Est_Zone	Rock Type	Search Pass	Orientation	Gold Search Radii X	Gold Search Radii Y	Gold Search Radii Z	Number of Composites Min	Number of Composites Max	Max Composites per Drill Hole	Boundary
		2		35	22	10	3	15	2	SOFT 8015
95	Rhyolite	1	350/-26/-44	18	9	4.5	5	15	2	HARD
		2		40	20	10	3	15	2	
99	Rhyolite	1	296/-54/172	20.3	9	9	5	15	2	HARD
		2		45	20	20	3	15	2	

Note: * Dynamic Anisotropy (DA) using a structural surface.

Table 14-13: Silver Grade Estimation Parameters by Estimation Zone

Rock Type	Est_Zone	Search Pass	Orientation	Silver X	Silver Y	Silver Z	Number of Composites Min	Number of Composites Max	Max Composites per Drill Hole	Boundary
Predominantly Rhyolite	1	1	D1	12.5	12.5	7.5	3	10	2	HARD
	2	1	D2	30	10	5	3	10	2	
	3	1	D3	20	10	5	3	10	2	
	4	1	D4	22.5	10	5	3	10	2	
	5	1	D245	22.5	15	5	3	10	2	
	6	1	D6	20	12.5	7.5	3	10	2	
	8	1	Zone 22	10	10	5	3	10	2	
Rhyolite	1000	1	135/0/45	9	9	4.5	5	15	2	HARD
		2		20	20	10	3	15	2	
Rhyolite	2010	1	D245	36	18	9	5	15	2	SOFT
		2		80	40	20	3	15	2	2010
Mudstone	2020	1	D245	7.7	7.7	7.7	5	15	2	SOFT
		2		17	17	17	3	15	2	2020
Rhyolite	3011	1	D245 North	18	13.5	13.5	5	15	2	SOFT
		2		40	30	30	3	15	2	3020
Rhyolite	3012	1	D245 South	18	13.5	11.3	5	15	2	SOFT
		2		40	30	25	3	15	2	3020
Mudstone	3020	1	D245	27	18	9	5	15	2	SOFT
		2		60	40	20	3	15	2	3011,3012
Rhyolite	4010	1	D245	29.3	27	4.5	5	15	2	SOFT
		2		65	60	10	3	15	2	4020
Mudstone	4020	1	D245	13.5	7.2	9	5	15	2	SOFT
		2		30	16	20	3	15	2	4010
Rhyolite	5010	1	D3	20.3	9	4.5	5	15	2	HARD
		2		45	20	10	3	15	2	
Mudstone	5020	1	D3	18	9	4.5	5	15	2	HARD
		2		40	20	10	3	15	2	

Rock Type	Est_Zone	Search Pass	Orientation	Silver			Number of Composites			Boundary
				X	Y	Z	Min	Max	Max Composites per Drill Hole	
Rhyolite	6010	1	D1	11.3	13.5	4.5	5	15	2	HARD
		2		25	30	10	3	15	2	
Mudstone	6020	1	D1	11.3	10.4	10.4	5	15	2	HARD
		2		25	23	23	3	15	2	
Hangingwall Andesite	7022	1	D2	15.8	9	2.3	5	15	2	HARD
		2		15.8	9	2.3	3	15	2	
	7023	1	D3	13.5	9	2.3	5	15	2	HARD
		2		30	20	5	3	15	2	
	7025	1	D5	15.8	9	1.8	5	15	2	HARD
		2		35	20	4	3	15	2	
	7026	1	D6	54	22.5	11.3	5	15	2	HARD
		2		120	50	25	3	15	2	
Rhyolite	8010	1	020/0/80	29.3	10.8	5.9	5	15	2	HARD
		2		65	24	13	3	15	2	
	8012	1	D2	29.3	10.8	5.9	5	15	2	SOFT 8022
		2		65	24	13	3	15	2	
	8015	1	D5	29.3	10.8	5.9	5	15	2	SOFT
		2		65	24	13	3	15	2	8025
	8016	1	D5	24.8	10.4	5.4	5	15	2	HARD
		2		55	23	12	3	15	2	
Mudstone	8022	1	D2	22.5	9	4.5	5	15	2	SOFT 8012
		2		50	20	10	3	15	2	
	8025	1	D5	18	18	4.5	5	15	2	SOFT 8015
		2		40	40	10	3	15	2	
Rhyolite	95	1	350/-26/-44	11.3	11.3	9	5	15	2	HARD
		2		25	25	20	3	15	2	
Rhyolite	99	1	296/-54/172	9	9	9	5	15	2	HARD
		2		20	20	20	3	15	2	

* D=Dynamic Anisotropy using a structural surface.

Pass 1 equalled 90% of the variogram range, and Pass 2 equalled two times the variogram range. The nugget and first sill were updated for the 2 m composite variograms; all other variogram parameters remained the same as those derived from the 1 m composites.

The low-grade envelope domain was estimated using ranges and orientations inherited from variograms in the nearest mineralization domain using one pass.

Hard boundary interpolation was honoured, except in domains split by lithology and having similar orientations and structures. Between these zones a soft boundary was used (refer to Table 14-12). A hard boundary was applied within the 3 m buffer domain to limit the spread of high-grade values from mined-out intervals into the remaining resources area.

14.11.2 Open Pit Model, Visual Validation

Estimated block grades were assessed in plan and sectional view along with composite assay intervals. This method provides a local visual assessment of interpolated blocks in relation to the nearest composite. Figure 14-14 and Figure 14-15 show estimated AuEq block grades in relation to 2 m AuEq composite intervals in the 21B and 21A domains, respectively. Overall, the data show good agreement and no obvious discrepancies between block grades and composites were observed.

14.11.3 Open Pit Model, Comparison of Interpolation Models

ID^2 and nearest neighbour (NN) models were produced to check for local biases. Although variable between zones, the overall bias was less than 1% for gold and 2% for silver in the open pit model. A summary of global bias between the ID^2, NN, and OK estimation methods for gold and silver by estimation zone are summarized in Table 14-14. The differences are within acceptable limits.

Composite statistics were derived using hard boundaries for all domains, however, OK interpolation methods utilized either hard or soft boundary conditions. For domains that were estimated using soft boundaries, composite statistics do not fully correspond with block estimated statistics. For example, estimation zones 4010 and 4020 in the 21B Domain used a soft boundary across the Rhyolite/Mudstone contract.

Figure 14-14: Visual Comparison of Block Model AuEq Grades vs 2 m Composite AuEq Grades in the 21B and 21E Domains

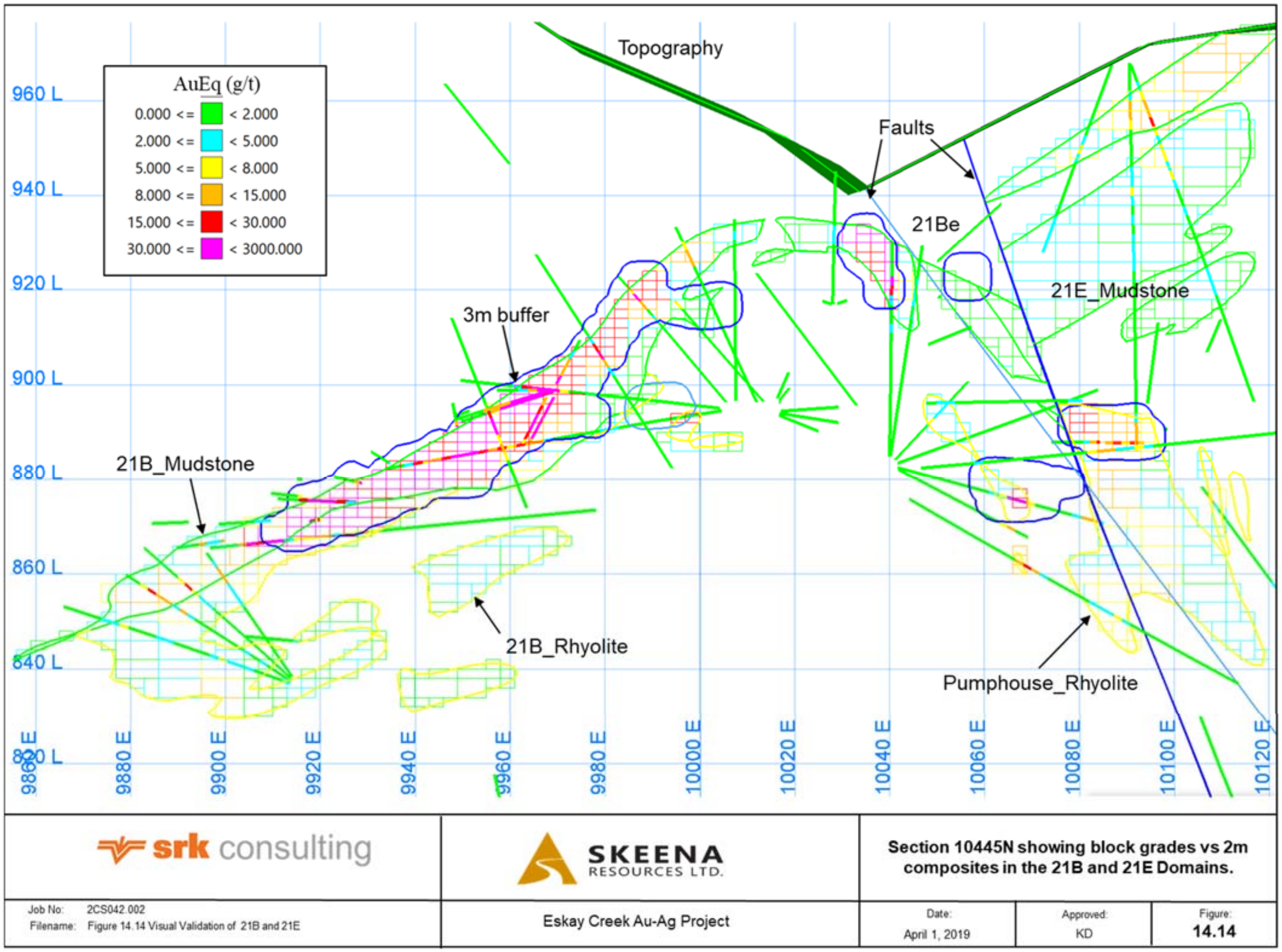


Figure 14-15: Visual Comparison of Block Model AuEq Grades and 2 m Composite AuEq Grades in the 21A Domain

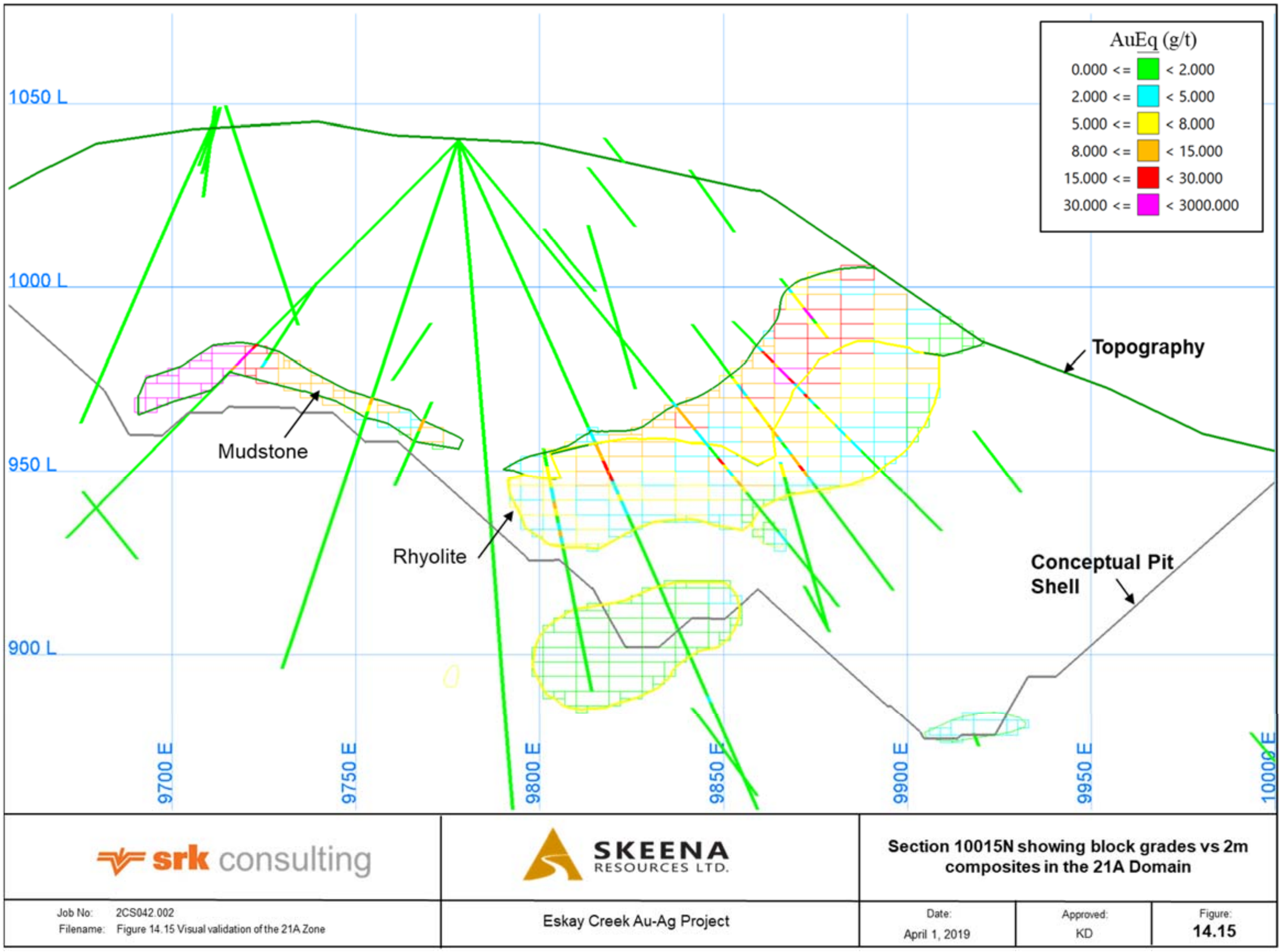


Table 14-14: Global Bias Check for Gold and Silver by Estimation Zone

Gold						Silver					
Est_Zone	AUNN	AUID	AUOK	OK vs ID (%)	OK vs NN (%)	Est_Zone	AUNN	AUID	AUOK	OK vs ID (%)	OK vs NN (%)
1000	1.34	1.32	1.27	-3	-5	1000	43.86	47.59	47.43	0	8
2010	2.11	2.11	2.10	-1	-1	2010	43.97	43.72	43.65	0	-1
2020	9.16	10.23	9.61	-6	5	2020	132.09	135.70	128.47	-5	-3
3011	2.62	2.66	2.67	0	2	3011	25.50	26.20	27.11	3	6
3012	4.10	4.36	4.31	-1	5	3012	81.54	83.13	80.37	-3	-1
3020	2.72	2.69	2.66	-1	-2	3020	93.63	101.13	97.72	-3	4
4010	1.73	1.81	1.84	2	7	4010	25.69	29.99	30.22	1	18
4020	3.04	3.12	3.15	1	4	4020	60.74	63.98	62.20	-3	2
5010	2.96	3.14	3.14	0	6	5010	92.39	95.64	95.98	0	4
5020	2.62	2.59	2.59	0	-1	5020	50.00	50.18	48.42	-4	-3
6010	2.04	1.98	1.94	-2	-5	6010	69.97	72.55	69.63	-4	0
6020	4.13	3.69	3.70	0	-10	6020	163.45	151.00	151.58	0	-7
7022	3.39	3.50	3.45	-1	2	7022	75.89	69.38	65.72	-5	-13
7023	2.41	2.42	2.36	-2	-2	7023	98.21	111.25	107.28	-4	9
7025	2.78	2.96	2.86	-4	3	7025	138.66	154.66	160.45	4	16
7026	2.61	2.57	2.54	-1	-3	7026	63.63	61.26	59.00	-4	-7
8012	2.49	2.59	2.57	-1	3	8012	48.98	48.35	49.19	2	0
8015	1.61	1.73	1.76	2	9	8015	36.75	37.18	37.26	0	1
8016	2.22	2.29	2.23	-2	1	8016	42.33	46.84	45.38	-3	7
8022	3.21	3.12	3.08	-1	-4	8022	62.12	54.35	54.96	1	-12
8025	2.08	2.22	2.19	-1	5	8025	43.92	40.74	38.35	-6	-13
9500	4.18	4.14	4.09	-1	-2	9500	79.88	81.80	79.55	-3	0
9900	4.01	3.88	3.85	-1	-4	9900	9.31	9.62	9.61	0	3
			Total	***-1***	***1***				***Total***	***-2***	***1***

14.11.4 Open Pit Model, Swath Plots

The model was checked for local trends in the grade estimate using swath plots within each domain. This was done by plotting the mean values from the ID^2, NN, and declustered composites against the OK estimate along north–south, east–west and horizontal swaths. The ID^2, NN and OK models show similar trends in grades with the expected smoothing for each method when compared to composite data. The observed trends show no significant metal bias in the estimate. Swath plots for gold and silver in the 21A Domain rhyolite and mudstones are illustrated in Figure 14-16 and Figure 14-17, respectively.

14.11.5 Underground Model

The block model geometry and extents used for grade estimation in the underground model are summarized in Table 14-15. Three domains were captured within the underground model: 22, NEX and HW Zones. OK was used to estimate gold and silver in all three domains. One-meter capped composites were used for the underground model. Gold and silver grades within mineralized domains were estimated in two successive passes with increasing search radii. Pass 1 approximated 90% of the variogram range and Pass 2 equalled two times the variogram range.

Hard boundary interpolation was honoured, except in domains having similar orientation and structure split only by lithology; between these zones a soft boundary was applied. Hard boundaries were used for composites within the 3 m buffer domain to limit the effect of high-grade smearing from mined out intervals. Hard and soft boundaries were used and are domain-specific.

Dynamic anisotropy was applied where domains were folded using search ellipses established from 1 m variograms. For Pass 1 a minimum of five and maximum of 10 composites were used per block. For Pass 2, a minimum of three and maximum of 10 composites were used per block. A maximum of two composites per drill hole was specified for both passes. The same 3 m buffer solid was used as the depletion zone for reporting remaining resources. All other parameters remained the same.

14.11.6 Underground Model, Visual Validation

A visual inspection of the block estimates with drill hole composites in plan and cross-section was performed as a first pass check on the estimates. Good agreement between the composite grades and block estimates was observed, as well as suitably oriented estimates relative to dynamic anisotropy surfaces (Figure 14-18).

14.11.7 Underground Model, Comparison Of Interpolation Models

To validate the OK estimates, gold and silver were estimated using ID^2 and NN models to assess for global bias. Although variable between zones, the overall bias was less than 3% for gold and 3% for silver in the Underground model. A difference of more than +/-10% was used as a guideline to indicate bias or significant over or underestimation. As seen in Table 14-16, the results are within acceptable limits.

14.11.8 Underground Model, Swath Plots

As part of the validation process, composite samples were compared with block model grades in three principal directions to assess for grade and local trend discrepancies. The observed block trends follow the overall composite trends as was expected. Figure 14-19 and Figure 14-20 show OK, ID^2, NN and declustered composites for the HW and NEX zones for gold and silver grades, respectively.

Figure 14-16: Swath Plot for Gold (left) and Silver (right) in Est_Zone 2010, 21A Rhyolite, (top) Northing, (middle) Easting, (bottom) Elevation

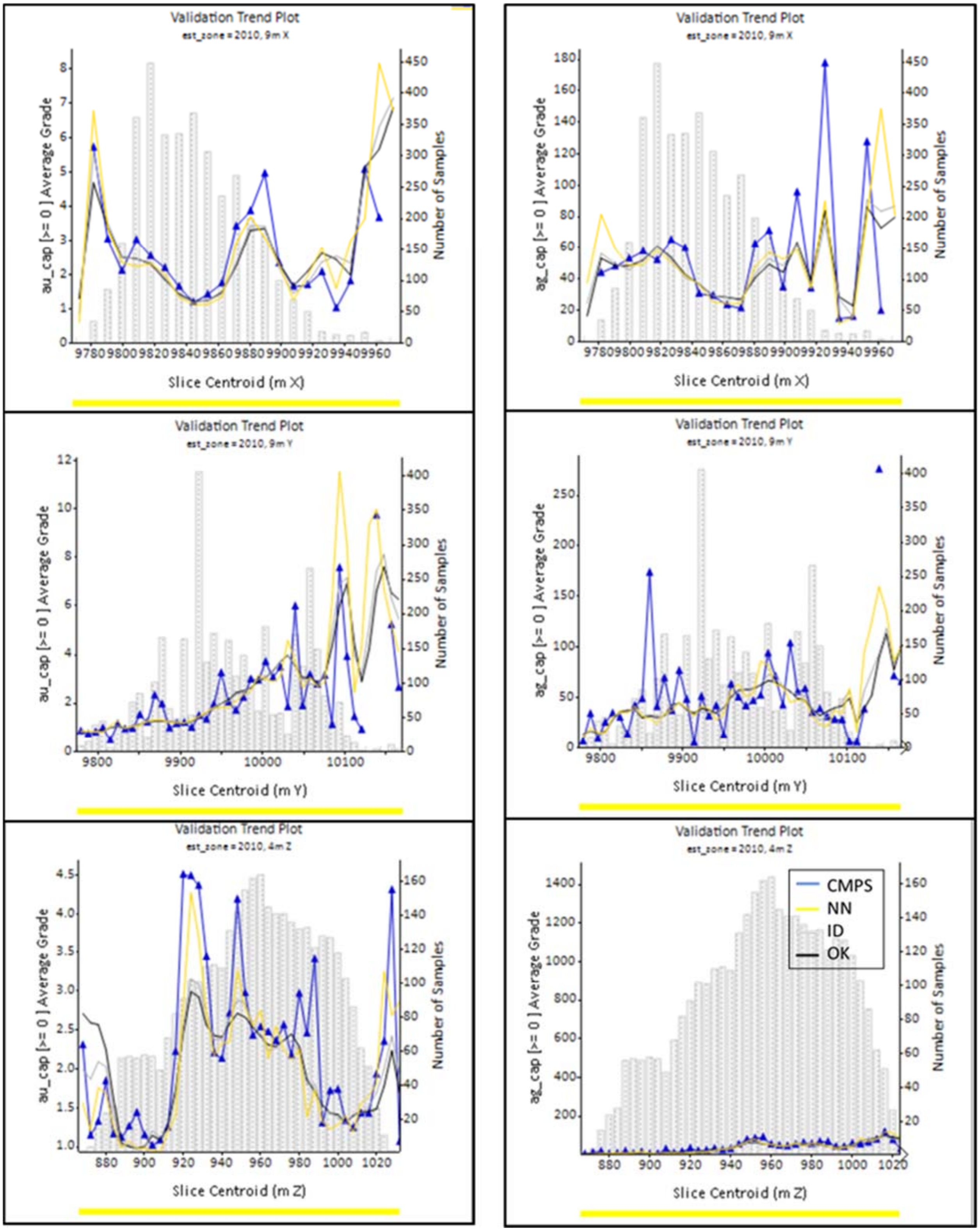


Figure 14-17: Swath Plot for Gold (left) and Silver (right) in Est_Zone 2010, 21A Mudstone, (top) Northing, (middle) Easting, (bottom) Elevation

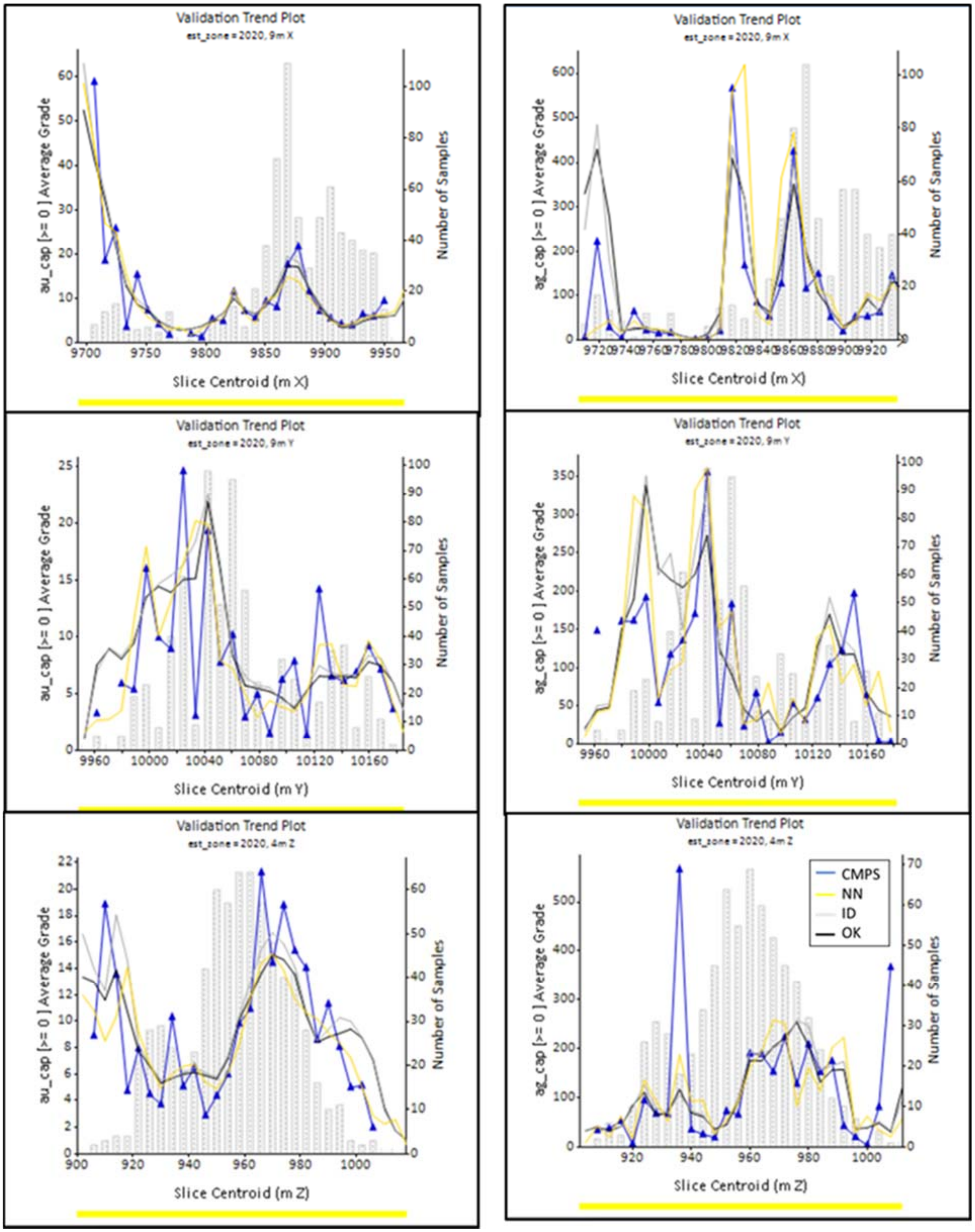


Table 14-15: Details of block model dimensions and block size for the Underground model

	Bearing	Plunge	Dip	Start Offset			End Offset			Block Size		
				X	Y	Z	X	Y	Z	X	Y	Z
Parent	90	0	0	9300	8508	-50	963	3150	1500	3	3	2
Sub-block	90	0	0	9300	8508	-50	963	3150	1500	1	1	1

Figure 14-18: Visual Check of the Underground Model Showing 1 m AueEq Composites and Estimated AuEq Block Grades

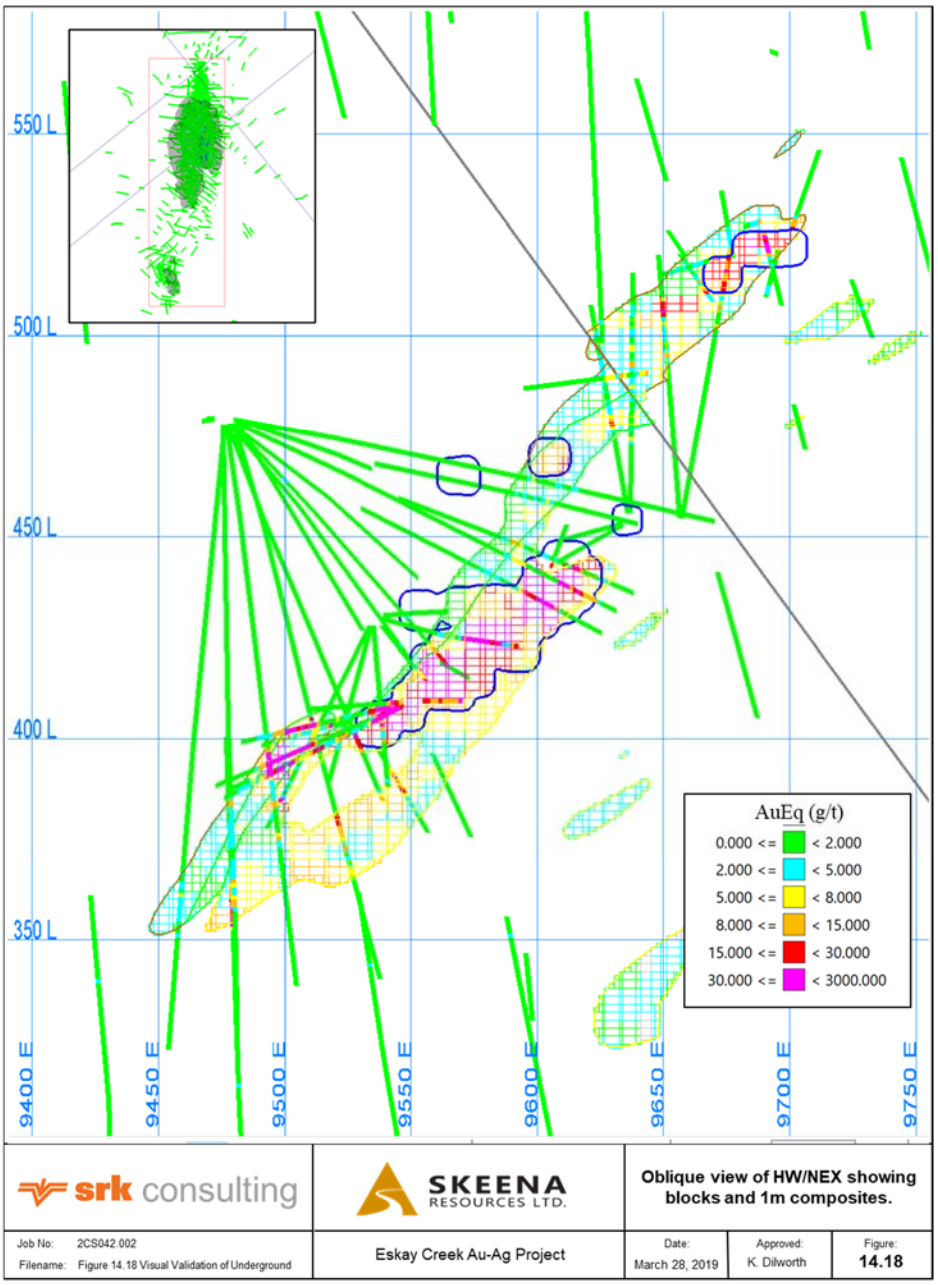


Table 14-16: Global Bias Gold and Silver by Zone

Est_Zone	AUNN	AUID	AUOK	OK vs ID (%)	OK vs NN (%)	Est_Zone	AUNN	AUID	AUOK	OK vs ID (%)	OK vs NN (%)
7022	2.70	2.63	2.63	0	-3	7022	54.79	54.31	52.47	-3	-4
7026	2.48	2.48	2.45	-1	-1	7026	62.45	58.30	56.08	-4	-10
8012	1.70	1.76	1.74	-1	2	8012	29.26	28.14	28.34	1	-3
8015	1.43	1.50	1.53	2	7	8015	34.14	30.93	32.69	6	-4
8016	2.08	2.23	2.20	-1	6	8016	38.70	41.69	41.67	0	8
8022	3.08	3.15	3.12	-1	1	8022	57.52	53.31	52.98	-1	-8
8025	2.11	2.22	2.24	1	6	8025	49.47	49.03	49.69	1	0
			Total	***0***	***3***				***Total***	***0***	***-3***

Figure 14-19: Swath Plot for Gold (left) and Silver (right) in Est_Zone 7026, HW – Mudstone/HW Andesite, (top) Northing, (middle) Easting, (bottom) Elevation

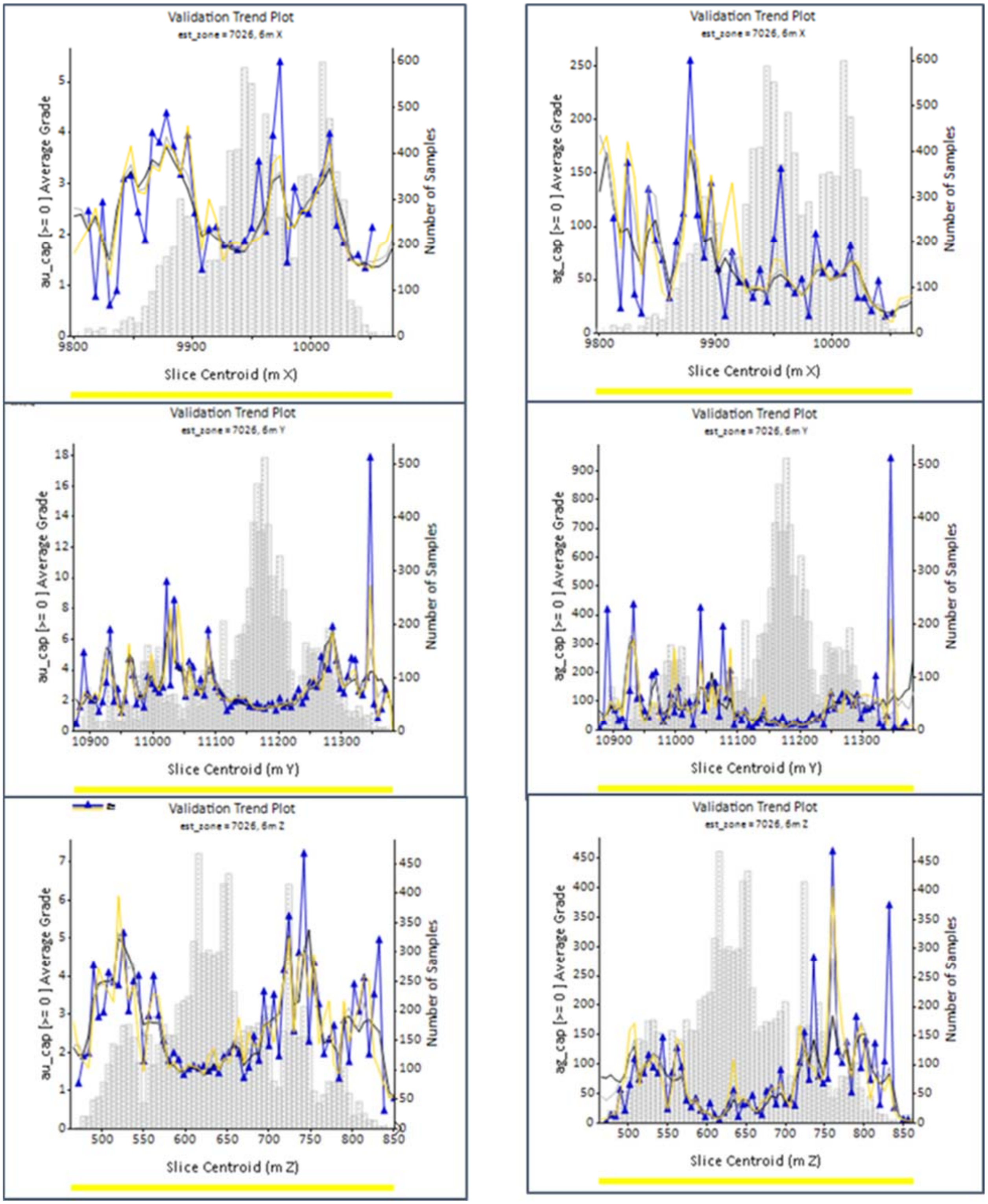
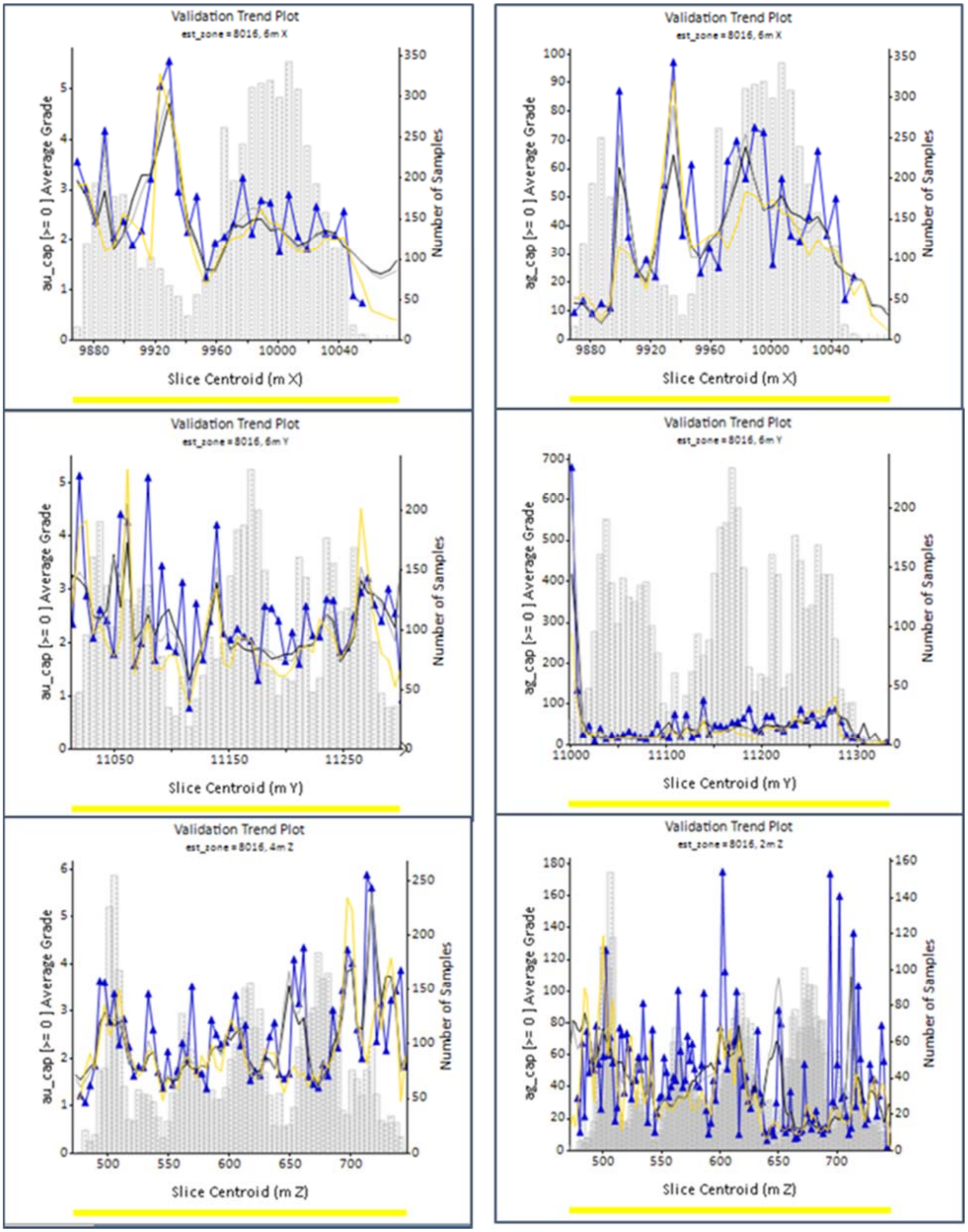


Figure 14-20: Swath Plot for Gold (left) and Silver (right) in Est_Zone 8016, NEX- Rhyolite, (top) Northing, (middle) Easting, (bottom) Elevation



14.12 Rhyolite versus Mudstone Estimates

The majority of remaining mineralization at Eskay Creek is hosted in the rhyolite facies feeder structures which are not enriched in the exhalative epithermal suite of elements (Hg–As–Sb). Preferential historical development and mining of the bonanza grade mineralization hosted in the mudstone has resulted in extensive depletion of resources in this rock type. The 2019 pit constrained Mineral Resource estimate indicates that on a tonnage weighted basis, 70% of the resource is hosted within the rhyolite facies with only 30% hosted in the remaining unmined mudstone/hanging all andesite (Figure 14-21). On an ounce weighted basis, 60% of the pit constrained resource is contained within the rhyolite with the remaining 40% hosted within the contact mudstone/hanging wall andesite.

14.13 Mineral Resource Classification

Block model quantities and grade estimates for the Eskay Creek Project were classified according to the 2014 CIM Definition Standards.

Mineral resource classification is typically a subjective concept. Industry best practices suggest that resource classification should consider the following: the confidence in the geological continuity of the mineralized structures, the quality and quantity of exploration data supporting the estimates, and the geostatistical confidence in the tonnage and grade estimates. Appropriate classification criteria should aim at integrating all above requirements to delineate regular areas at similar resource classification.

SRK is satisfied that the geological model honours the current geological interpretation and knowledge of the deposit. The location of the samples and the assay data are sufficiently reliable to support resource evaluation.

For mineralization exhibiting good geological continuity using adequate drill hole spacing, SRK considers that blocks estimated during the first estimation pass at 90% of the variogram range may be classified in the Indicated category. For those blocks, the level of confidence is adequate for evaluating the economic viability of the deposit, as well as suitable for assessing technical and economic parameters to support mine planning.

For blocks estimated during the second pass, which uses search distances twice the variogram range, the blocks may be classified in the Inferred category. For those blocks, the level of confidence is inadequate for evaluating the economic viability of the deposit, as well as unsuitable for assessing technical and economic parameters to support mine planning.

All interpolated blocks coded during Pass 1 and Pass 2 were assigned to the Inferred category during the first stage of classification. Blocks were reclassified in a second stage if they met the following conditions:

- Blocks interpolated during Pass 1 using a minimum of 3 holes and a maximum distance of 43 m to a drill hole showing reasonable grade and continuity were reclassified to Indicated;
- All blocks within the 3 m buffer domain around the high-grade, mined-out areas were reclassified to Indicated;
- In areas where blocks were interpolated during Pass 1, but continuity was insufficient or blocks were isolated, the blocks were reclassified to Inferred on a visual basis.

Figure 14-22 shows the distribution of the Indicated and Inferred Mineral Resources in the pit constrained Open Pit model.

Figure 14-21: Breakdown of Rhyolite and Mudstone Lithologies in the 21C and 21A Domains

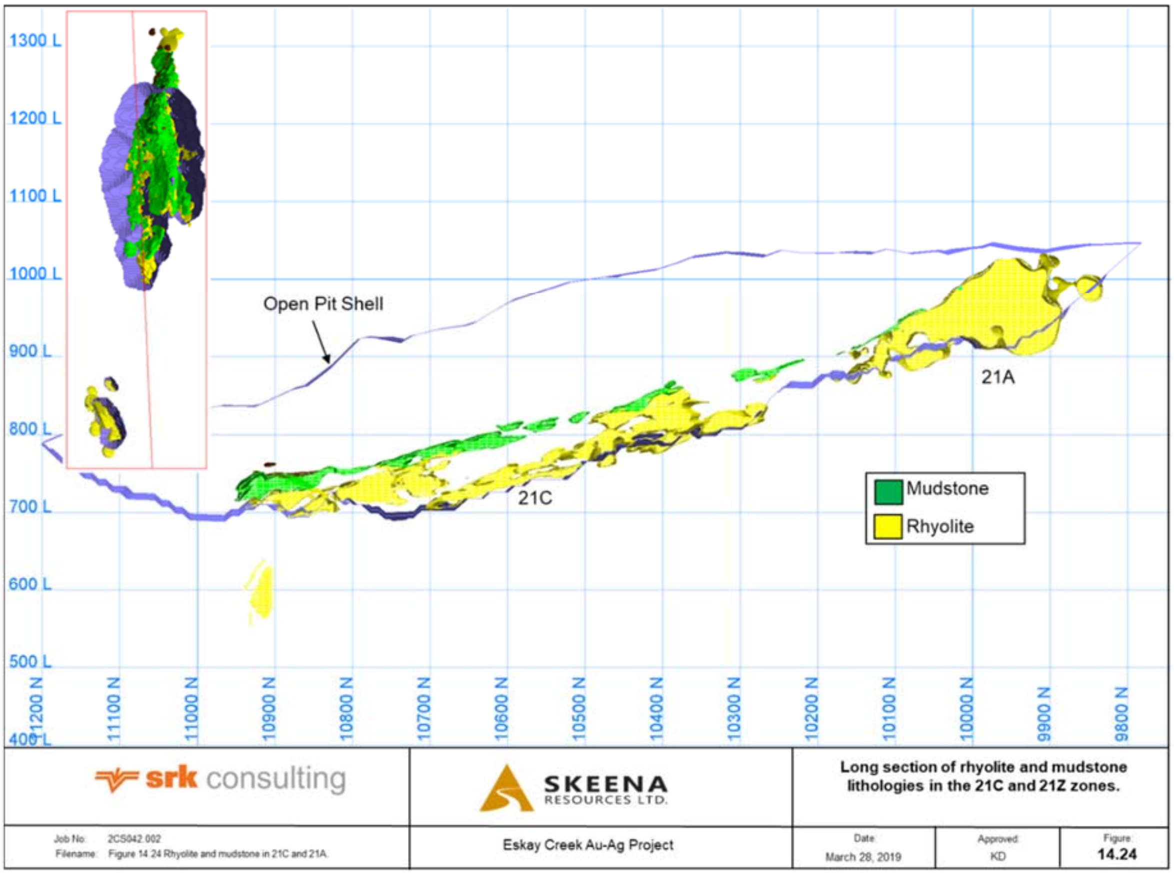
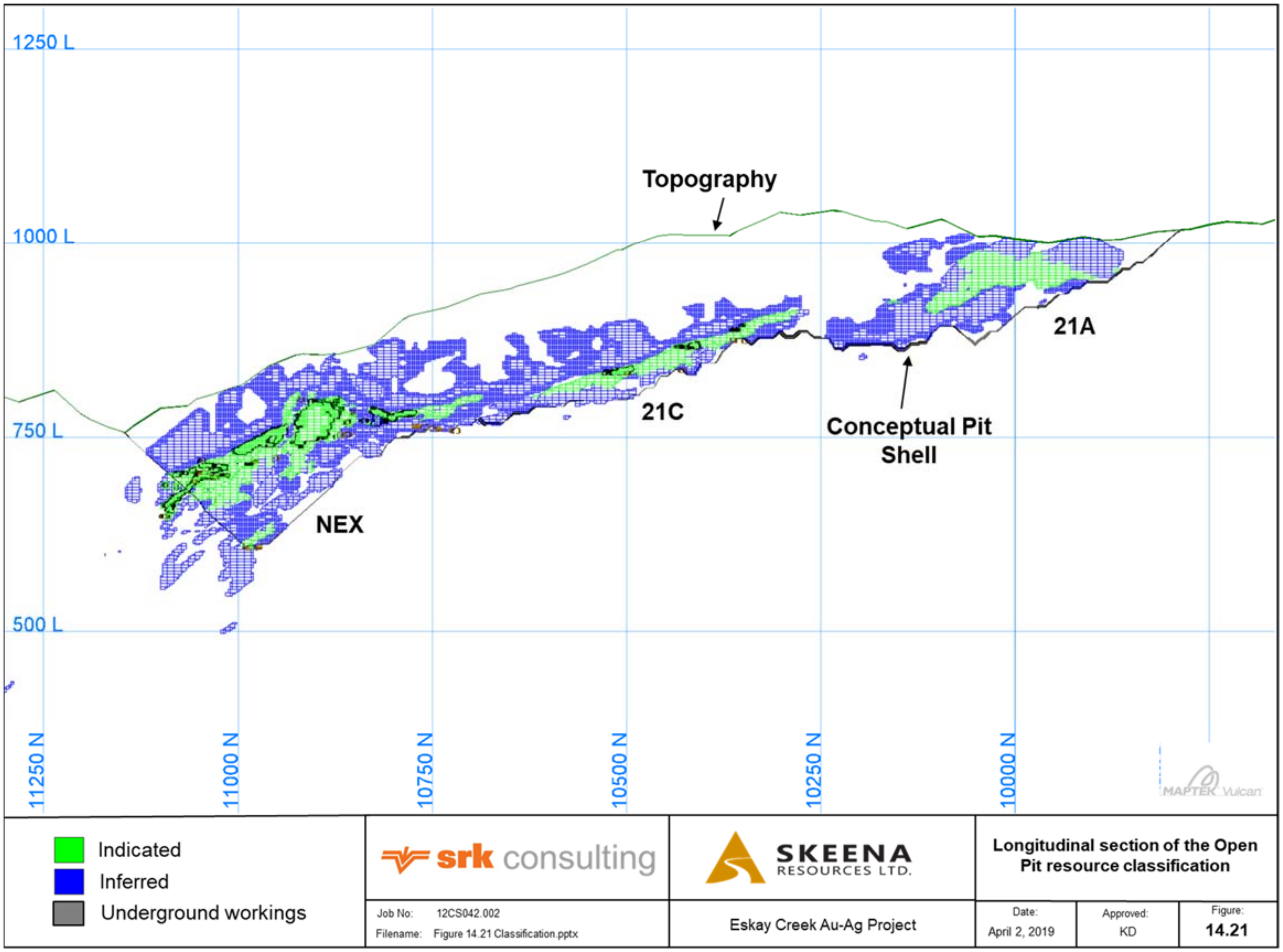


Figure 14-22: Long Section View of the Mineral Resource Classification In Blocks Looking East in the Open Pit Model



14.14 Mineral Resource Statement

The QP for the resource estimate is Ms. S. Ulansky, Senior Resource Geologist, PGeo (EGBC#36085), an employee of SRK.

The 2014 CIM Definition Standards define a mineral resource as:

"(A) concentration or occurrence of solid material of economic interest in or on the Earth's crust in such form, grade or quality and quantity that there are reasonable prospects for eventual economic extraction.

The location, quantity, grade or quality, continuity and other geological characteristics of a Mineral Resource are known, estimated or interpreted from specific geological evidence and knowledge, including sampling".

The "reasonable prospects for economic extraction" requirement generally imply that the quantity and grade estimates meet certain economic thresholds and that the mineral resources are reported at an appropriate cut-off grade considering extraction scenarios and processing recoveries. To meet this requirement, SRK considers that major portions of the Eskay Creek Project are amenable to open pit extraction, and minor areas are amenable to underground mining.

To determine the quantities of material offering "reasonable prospects for economic extraction" by open pit methods, SRK used a pit optimizer and reasonable mining assumptions to evaluate the proportion of the block model (Indicated and Inferred blocks) that could be "reasonably expected" to be mined from an open pit.

The optimization parameters were selected based on experience, and benchmarking against similar projects (Table 14-17). The reader is cautioned that the results from the pit optimization are used solely for testing "reasonable prospects for economic extraction" by open pit methods and do not represent an attempt to estimate Mineral Reserves. There are no Mineral Reserves on the Eskay Creek Project. The results are used as a guide to assist in the preparation of a Mineral Resource statement and to select an appropriate resource reporting cut-off grade.

The block model quantities and grade estimates were also reviewed to determine the portions of the Eskay Creek Project having "reasonable prospects for economic extraction" using an underground mining scenario. The parameters are summarized in Table 14-18.

The cut-off grade for the open pit model, using the parameters presented in Table 14-17, was determined to be 0.56 g/t AuEq. The underground cut-off grade, using the parameters presented in Table 14-18, was determined to be 4.2 g/t AuEq. At the request of Skeena, the cut-off grades applied for the resource statement were increased to 0.7 g/t AuEq for the open pit and 5.0 g/t AuEq for the underground resource.

The Mineral Resource statement for the Mineral Resources considered potentially amenable to open pit mining methods is presented in Table 14-19 and for the Mineral Resources considered potentially amenable to underground mining methods in Table 14-20. The Mineral Resources potentially amenable to underground mining methods are reported independently of the Mineral Resources considered potentially amenable to open pit mining methods. In addition, all potential resources that occur within 1 m of any historical workings in the open pit model were excluded from the reported resource. In the underground model, all potential resources that occur within 3 m of any historical working were excluded from the resource.

Table 14-17: Assumptions Considered for Conceptual Open Pit Optimization

Parameter	Value	Unit
Overall pit wall angles	45	degrees
Mining cost	2	US$ per tonne mined
Processing cost	15	US$ per tonne of feed
General and administrative	5.75	US$ per tonne of feed
Mining dilution	5	percent
Mining recovery	95	percent
Gold process recovery	80	percent
Silver process recovery	90	percent
Sell price gold	1275	US$ per ounce
Sell price silver	17	US$ per ounce
Sell cost	30	US$ per ounce
In situ cut-off-grade	0.56	grams per tonne
Combined strip ratio	7.5 : 1	unitless

Table 14-18: Assumptions Considered for Underground Resource Reporting

Parameter	Value	Unit
Mining costs	79.25	US$ per tonne mined
Process cost	15	US$ per tonne of feed
General and administrative	5.75	US$ per tonne of feed
Process recovery Au	80	percent
Process recovery Ag	90	tonne feed per year
Sell price gold	1275	US$ per ounce
Sell price silver	17	US$ per ounce
Sell cost	30	US$ per ounce

Table 14-19: Open Pit Mineral Resource Statement Reported at 0.7 g/t AuEq Cut-Off Grade

Classification	Domain	Tonnes (000)	Grade AuEq g/t	Au g/t	Ag g/t	Contained Ounces AuEq oz (000)	Au oz (000)	Ag oz (000)
Indicated	22	270	3.0	2.0	74	30	20	640
	21A	3,530	4.0	3.2	62	450	360	6,990
	21C	2,800	4.5	3.7	65	410	330	5,850
	21B	2,510	8.4	6.0	175	680	490	14,120
	21Be	860	9.7	6.5	241	270	180	6,660
	21E	200	4.1	2.6	112	30	20	720
	HW	880	6.0	3.8	170	170	110	4,820
	NEX	720	6.8	4.5	171	160	100	3,960
	PMP	180	5.9	4.5	106	30	30	620
	109	710	5.2	5.0	13	120	110	300
	Total	***12,650***	***5.8***	***4.3***	***110***	***2,340***	***1,740***	***44,660***
Inferred	ENV	3,110	2.2	1.4	57	220	140	5,740
	22	1,350	2.1	1.9	15	90	80	660
	21A	1,330	5.7	5.0	51	240	210	2,190
	21C	2,080	2.6	2.2	32	180	150	2,160
	21B	3,220	2.5	2.0	32	250	210	3,290
	21Be	720	4.0	2.9	85	90	70	1,960
	21E	900	2.9	2.0	61	80	60	1,750
	HW	740	3.8	2.4	105	90	60	2,500
	NEX	800	2.8	2.2	48	70	60	1,240
	PMP	100	4.9	3.9	70	20	10	220
	109	80	2.7	2.6	10	10	10	20
	Total	***14,420***	***2.9***	***2.3***	***47***	***1,340***	***1,050***	***21,720***

Table 14-20: Underground Mineral Resource Statement Reported at a 5.0 g/t AuEq Cut-Off Grade

Classification	Tonnes (000)	Grade AuEQ g/t	Au g/t	Ag g/t	Contained Ounces AuEQ oz (000)	Au oz (000)	Ag oz (000)
Indicated	819	8.2	6.4	139	218	169	3,657
Inferred	295	8.2	7.1	82	78	68	778

Notes to accompany the Mineral Resource estimate:

1. Mineral Resources are not Mineral Reserves as they do not have demonstrated economic viability. Results are reported in-situ and undiluted and are considered to have reasonable prospects for economic extraction.
2. As defined by NI 43-101, the Independent and Qualified Person is Ms. S Ulansky, PGeo of SRK Consulting (Canada) Inc., who has reviewed and validated the Mineral Resource Estimate.

3. The open pit block model was regularized to 9 m x 9 m x 4 m whole blocks using mineralization > 0.5 g/t gold equivalent (AuEq) within a single mineralisation percent field. AuEq is calculated using the formula The $AuEQ = Au\,(g/t) + (Ag\,(g/t)/75)$.
4. The effective date of the Mineral Resource estimate is February 28, 2019.
5. The number of metric tonnes and ounces were rounded to the nearest thousand. Any discrepancies in the totals are due to rounding.
6. Pit constrained Mineral Resources are reported in relation to a conceptual pit shell.
7. Block tonnage was estimated from volumes using a density formula that applied using interpolated Pb, Zn, Cu, and Sb whereby $SG = (Pb + Zn + Cu + Sb) * 0.03491 + 2.67$ (where all metals are reported in %).
8. All composites have been capped where appropriate.
9. Mineral Resources potentially amenable to open pit mining methods are reported at a cut-off grade of 0.7 g/t AuEq and Mineral Resources potentially amenable to underground mining methods are reported at a cut-off grade of 5.0 g/t AuEq.
10. Cut-off grades are based on a price of US$1,275 per ounce of gold, US$17 per ounce silver, and gold recoveries of 80%, silver recoveries of 90% and without considering revenues from other metals.
11. Estimates use metric units (metres, tonnes and g/t). Metals are reported in troy ounces (metric tonne * grade / 31.10348).
12. 2014 CIM definitions were followed for the classification of mineral resources.
13. Neither Skeena nor SRK is aware of any known environmental, permitted, legal, title-related, taxation, socio-political, marketing or other relevant issue that could materially affect this Mineral Resource estimate.

Figure 14-23: Oblique View of Open Pit Constrained Resources at a 0.7 g/t AuEq Cut-Off Grade

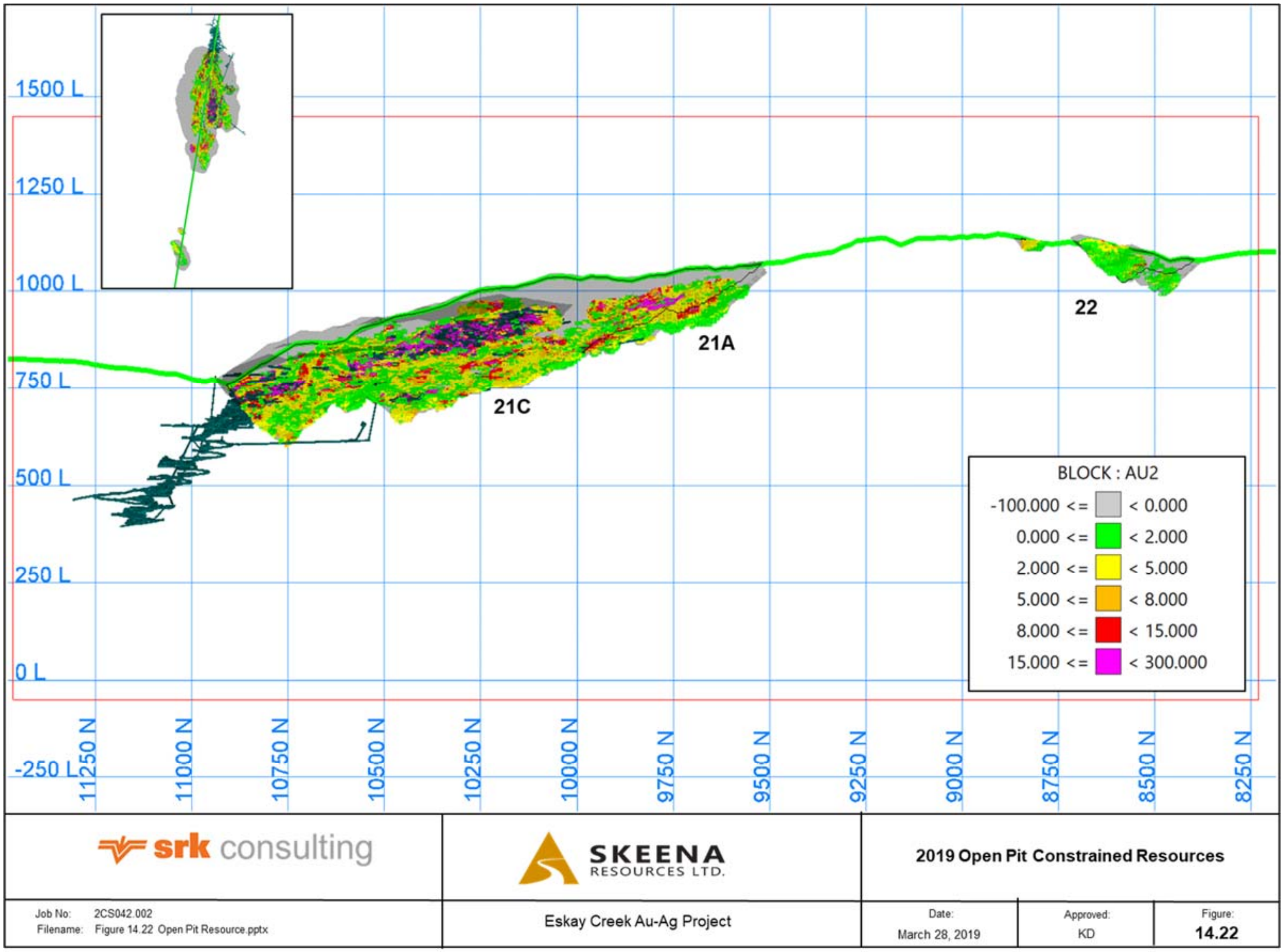


Figure 14-24: Oblique View of Underground Resources Remaining In The Eskay Creek Project at a 5.0 g/t AuEq Cut-Off Grade

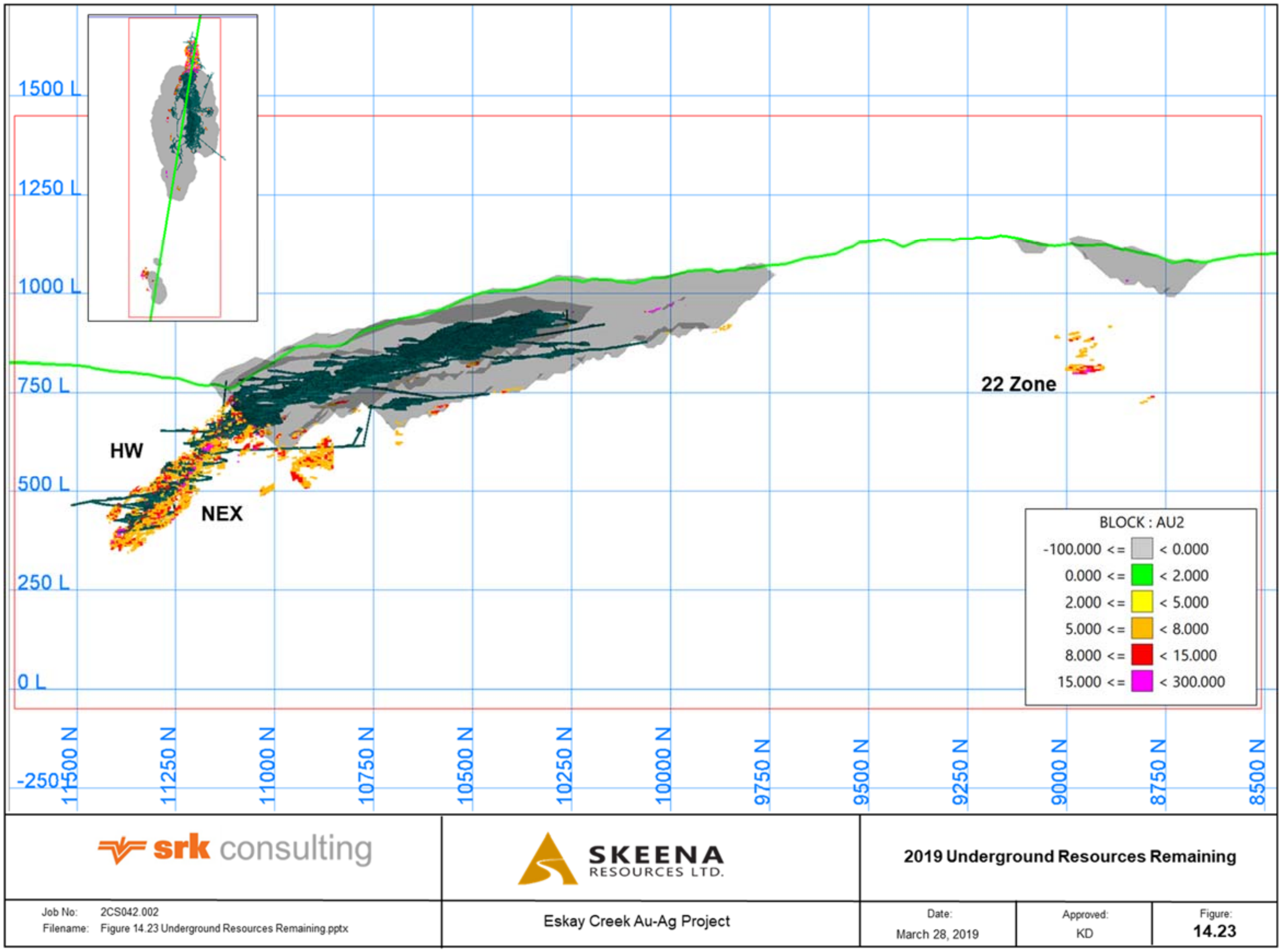


Table 14-19 shows the resources constrained within a conceptual open pit at a 0.7 g/t AuEq cut-off outside of the 1 m geotechnical exclusion zone. The model has been regularized to 9 m x 9 m x 4 m whole blocks using mineralization greater than 0.5 g/t AuEq within a single ore percent field; any differences from the sub-blocked model released on February 28, 2019, are due to inherent data transformations during the regularization process. Table 14-20 shows the remaining underground resources above the 5.0 g/t AuEq cut-off outside the 3 m buffered historical workings, and exclusive of the pit constrained resource.

14.15 Grade Sensitivity Analysis

The Eskay Creek Mineral Resources were assessed in terms of cut-off grade selection by means of sensitivity analyses. Global block model quantities and grade estimates within the conceptual open pit are presented in Table 14-21 at different cut-off grades. The resource is not sensitive to minor adjustments in cut-off grade selection; average ore zone grades are substantially higher than the selected cut-offs and a significant difference in tonnage and ounces is not demonstrated. The reader is cautioned that the figures presented in this table should not be misconstrued with a Mineral Resource Statement apart from the official scenario at 0.7 g/t AuEq.

Table 14-22 presents global block model quantities and grade estimates within the underground resource model at different cut-off grades. The underground scenario is sensitive to adjustments in cut-off grade selection due to the high average grades in this area. The reader is cautioned that the figures presented in this table should not be misconstrued with a Mineral Resource statement apart from the official scenario at 5.0 g/t AuEq.

14.16 Reconciliation to Previous Mineral Resource Model

A comparison between the 2019 and 2018 Mineral Resource Statements is shown in Table 14-23 and Table 14-24 for the open pit constrained and underground models, respectively. The large changes in the 2019 estimate is a direct function of the change in mining scenario. The 2019 mineral resources are principally Open Pit constrained resources, whereas the 2018 mineral resources were primarily reported as Underground resources. In additional, several changes were made to the 2019 estimation methodology, including:

- Updated geological and resource domain modelling;
- Inclusion of a 3 m buffer around the high-grade, mined-out stopes and lifts to limit the influence of the high-grade samples on the remaining resources;
- Revised and enhanced geostatistical inputs such as variograms, capping values and estimation parameters by principal rock type;
- 46 additional drill holes from the 22, 21A and 21C Zones;
- A change in classification strategy where the minimum number of holes required for the Indicated category was reduced from 5 to 3;
- A change in classification strategy where the minimum number of holes required for the Inferred category was reduced from 3 to 2;
- The geotechnical depletion zone around the mined-out stopes and/or lifts in the Open Pit model was reduced from 3 m to 1 m;
- The cut-off grades for the open pit resources was reduced from 1.0 g/t AuEq to 0.7 g/t AuEq;
- The cut-off grade for the Underground mineral resource was reduced from 5.5 g/t AuEq to 5.0 g/t AuEq.

Table 14-21: Block Model Quantities and Grade Estimates for the Open Pit Constrained Resource at the Eskay Creek Project Using Variable Cut-Off Grades (base case is highlighted)

AuEq Cut-off (g/t)	Tonnes (000)	Grade			Contained Ounces		
		AuEq (g/t)	Au (g/t)	Ag (g/t)	AuEq oz (000)	Au oz (000)	Ag oz (000)
Indicated							
>0.5	12,810	5.7	4.2	109	2,340	1,750	44,710
>0.7	**12,650**	**5.8**	**4.3**	**110**	**2,340**	**1,740**	**44,660**
>0.9	12,250	5.9	4.4	113	2,330	1,730	44,560
Inferred							
>0.5	15,950	2.7	2.1	43	1,370	1,080	22,090
>0.7	**14,420**	**2.9**	**2.3**	**47**	**1,340**	**1,050**	**21,720**
>0.9	12,930	3.1	2.5	51	1,310	1,020	21,260

* The reader is cautioned that the figures in this table should not be misconstrued with a Mineral Resource Statement. The figures are only presented to show the sensitivity of the block model estimates to the selection of cut-off grade.

Table 14-22: Block Model Quantities and Grade Estimates for the Underground Resources at the Eskay Creek Project Using Variable Cut-Off Grades (base case is highlighted)

AuEq Cut-off (g/t)	Tonnes (000)	Grade			Contained Ounces		
		AuEq (g/t)	Au (g/t)	Ag (g/t)	AuEq oz (000)	Au oz (000)	Ag oz (000)
Indicated							
>4.5	982	7.7	6.0	128	243	189	4,038
>5.0	**819**	**8.2**	**6.4**	**139**	**218**	**169**	**3,657**
>5.5	680	8.9	6.9	151	195	151	3,301
Inferred							
>4.5	343	7.7	6.6	81	85	73	894
>5.0	**295**	**8.2**	**7.1**	**82**	**78**	**68**	**778**
>5.5	233	9.0	7.8	91	67	58	686

* The reader is cautioned that the figures in this table should not be misconstrued with a Mineral Resource statement. The figures are only presented to show the sensitivity of the block model estimates to the selection of cut-off grade.

Table 14-23: 2019 vs 2018 Resource Comparison for the Open Pit Constrained Mining Scenario

Model Year	Tonnes (000)	Grade			Contained Ounces		
		AuEq (g/t)	Au (g/t)	Ag (g/t)	AuEq Ounces (000)	Au Ounces (000)	Ag Ounces (000)
Indicated							
2019	12,650	5.8	4.3	110	2,340	1,740	44,660
2018	1,088	5.9	4.9	72	207	173	2,533
Inferred							
2019	14,420	2.9	2.3	47	1,340	1050	21,720
2018	4,261	4.3	3.3	72	589	458	9,805

Table 14-24: 2019 vs 2018 Mineral Resource Statements for the Underground Mining Scenario

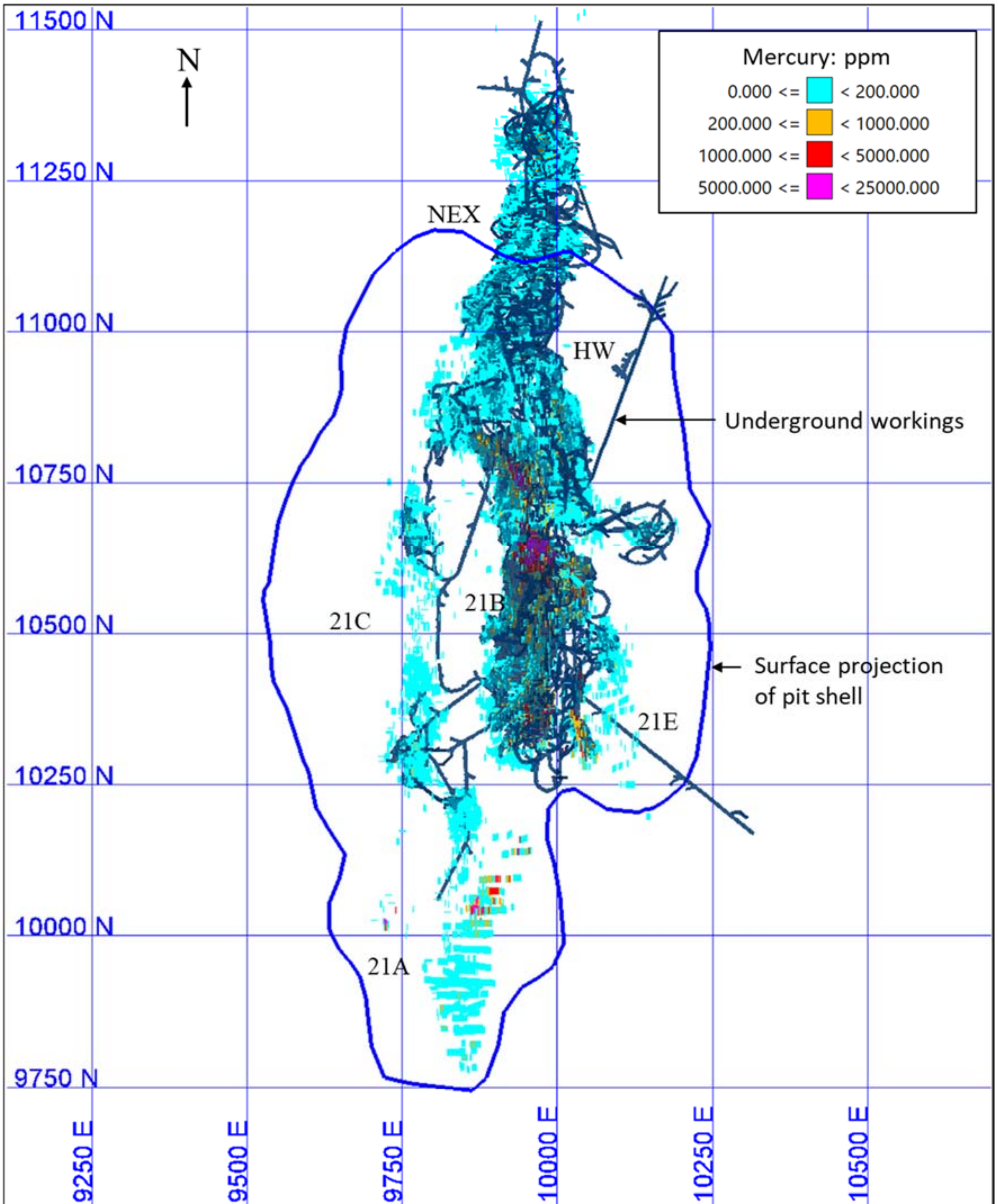
Model Year	Tonnes (000)	Grade			Contained Ounces		
		AuEq (g/t)	Au (g/t)	Ag (g/t)	AuEq Ounces (000)	Au Ounces (000)	Ag Ounces (000)
Indicated							
2019	819	8.2	6.4	139	218	169	3,657
2018	2513	10.1	7.2	215	814	582	17,340
Inferred							
2019	295	8.2	7.1	82	78	68	778
2018	812	10	7.2	214	261	187	5590

14.17 Epithermal and Base Metal Element Estimates in the Open Pit Model for Metallurgical Characterization

The epithermal suite of elements (antimony, mercury, and arsenic) and base metals (lead, copper and zinc) were estimated into the open pit block model to provide results for the metallurgical study conducted concurrently. Note that none of these additional elements add value to the economics of the project. A high degree of variability of the epithermal elements exists between the different zones and rock types, and elevated concentrations occur in localized zones and/ or pods. The Contact Mudstone lithology within the 21A and 21B Zones have elevated levels of arsenic, mercury and antimony. The 21A Zone is geologically and geochemically equivalent to the 21B Zone, an area which accounted for the bulk of mineralization historically mined at Eskay Creek. Smelter penalties for the elevated concentrations of arsenic, mercury and antimony in the 21B Zone were often prevented via blending with material from other zones while maintaining a profitable head grade (Barrick, 2004).

Figure 14-25 shows the uneven distribution of mercury within the outline of the proposed open pit.

Figure 14-25: Plan View of the Eskay Creek Deposit Showing Mercury Distribution in 2 m Composites Within The Mineralized Zones (elevated mercury within the 21B Zone has largely been mined out)



srk consulting	SKEENA RESOURCES LTD.	Plan view of mercury distribution in 2m composites		
Job No: 2CS042.002 Filename: Figure 14.25: Plan view of Hg distribution.pptx	Eskay Creek Au-Ag Project	Date: 10/15/2019	Approved: K. Dilworth	Figure: 14.25

14.17.1 Epithermal and Base Metal Elements Data Analysis

For all drilling campaigns prior to Skeena involvement, epithermal and base metal elements were selectively sampled. Historical documentation note that these elements were analysed when AuEq > 8 g/t; however, this was not always the case. This selective sampling process resulted in a dataset that is biased towards higher grade material because lower grade sample intervals were mostly excluded. The sampling inconsistencies are evident for all historical drilling campaigns, where the mineralization zones were either fully sampled, not sampled or intervals were selectively sampled. Table 14-25 and Table 14-26 show the percent of intervals assayed for the epithermal and base metal elements in relation to total gold assays, within each of the zones. The interval percentages ranges from 98% in the 22 Zone where antimony, arsenic and mercury were almost always assayed alongside gold and silver, to as low as 19% in the 21E Zone, where there was a major discrepancy between antimony, arsenic and mercury assays in relation to gold intervals. Figure 14-26 is a cross section of the 21A Zone showing sampling bias where drill holes are either fully sampled, non-sampled or selectively sampled.

Element correlations were generated for antimony, arsenic, mercury, lead, copper and zinc in relation to gold and silver assays (see Section 14.17). Relationships with gold and silver were moderate, at best, in only a select few zones. Without strong associations with either gold or silver it was not possible to generate regression relationships to populate the missing intervals. Seeing as this was the case, the gold-equivalent mineralization domains were utilized for estimating the spatial extent of the epithermal and base metal elements. This domaining approach was considered appropriate for metallurgical characterization studies. Sub-domaining the additional elements would have biased the outcome due to artefacts produced by the missing samples.

Table 14-27 and Table 14-28 summarize the statistical analysis of the epithermal and base metal elements within each of the zones.

14.17.2 Element Correlations

Correlations between the epithermal and base metal elements, in relation to gold and silver assays per zone, were generated with the purpose of using regression techniques for the missing intervals (Table 14-29 and Table 14-30). In most zones, there is a low to moderate coefficient of correlation of antimony and mercury with gold and silver, and negligible correlation of arsenic with gold and silver. Base metals show a moderate to high correlation with silver in some of the zones, most notably within the mudstone, however the scatterplots show a large degree of scatter (Figure 14-27). These correlations were, therefore, deemed unacceptable for regression implementation. In addition, elements were combined in a variety of suites to evaluate the degree of relationship with gold and/or silver, without success.

14.17.3 Evaluation of Outliers

Capping of high-grade assays was applied to the epithermal and base metal elements by zone using original assays. High-grade capping was examined using four tools: (1) histograms, (2) log probability plots, (3) cutting statistics, and (4) percent metal loss values. Visual inspections of the high-grade outliers in relation to the surrounding data was also undertaken to ensure that the locations were spatially disassociated. Less than 1% of the data was capped for high-grade outliers (Table 14-31 and Table 14-32). Overall, the epithermal elements lost 7%, 4% and 6% of the metal for antimony, mercury and arsenic, respectively. The base metals lost between 2–4% of the total metal due to capping. Several zones show percent metal loss values of >5%, which are the result of a limited number of extreme high-grade outlier samples.

Table 14-25: Percentage of Intervals Estimated for Epithermal and Base Metal Elements in Relation to Total Gold Assays According to Zone (Sb, Hg, As)

Domain	Zone	No. of Gold Assays	Antimony		Mercury		Arsenic	
			No. of Antimony Assays	%	No. of Mercury Assays	%	No. of Arsenic Assays	%
22	10	1,609	1,583	98	1,583	98	1,581	98
21A	201	6,086	3,191	52	2,871	47	3,210	53
	202	1,277	838	66	639	50	870	68
21C	301	22,600	6,760	30	6,637	29	6,761	30
	302	4,495	1,691	38	1,677	37	1,684	37
21B	401	19,902	5,883	30	5,598	28	4,501	23
	402	16,845	7,985	47	7,707	46	6,690	40
21Be	501	13,465	6,515	48	6,471	48	4,496	33
	502	8,679	3,497	40	3,452	40	2,574	30
21E	601	367	70	19	61	17	70	19
	602	1,509	707	47	695	46	707	47
NEX	702	24,963	8,734	35	8,598	34	7,350	29
HW	801	22,249	5,867	26	5,749	26	5,525	25
	802	12,887	4,282	33	4,253	33	3,669	28
PMP	95	2,395	1,072	45	1,079	45	1,079	45
109	99	11,753	4,871	41	4,782	41	3,799	32
	Sub-total	***171,081***	***63,543***	***37***	***61,852***	***36***	***54,566***	***32***

Table 14-26: Percentage of Intervals Estimated for Epithermal and Base Metal Elements in Relation to Total Gold Assays According to Zone (Pb, Cu, Zn)

Domain	Zone	No. of Gold Assays	Lead		Copper		Zinc	
			No. of Lead Assays	%	No. of Copper Assays	%	No. of Zinc Assays	%
22	10	1,609	1,608	100	1,513	94	1,609	100
21A	201	6,086	3,437	56	3,388	56	3,438	56
	202	1,277	851	67	844	66	852	67
21C	301	22,600	7,477	33	7,477	33	7,477	33
	302	4,495	1,869	42	1,872	42	1,872	42
21B	401	19,902	6,970	35	6,820	34	6,969	35
	402	16,845	8,015	48	8,000	47	8,023	48
21Be	501	13,465	5,058	38	5,031	37	5,061	38
	502	8,679	3,427	39	3,402	39	3,433	40
21E	601	367	104	28	104	28	104	28
	602	1,509	861	57	860	57	861	57
NEX	702	24,963	9,208	37	9,191	37	9,212	37
HW	801	22,249	6,499	29	6,493	29	6,503	29
	802	12,887	4,191	33	4,180	32	4,191	33
PMP	95	2,395	1,188	50	1,188	50	1,188	50
109	99	11,753	4,725	40	4,314	37	4,720	40
	Sub-total	***171,081***	***65,488***	***38***	***64,677***	***38***	***65,513***	***38***

Figure 14-26: Cross Section 9870E Showing Selective Sampling of Mercury in the 21A Zone Within a 25 m Window (unsampled drill hole traces are shown in grey)

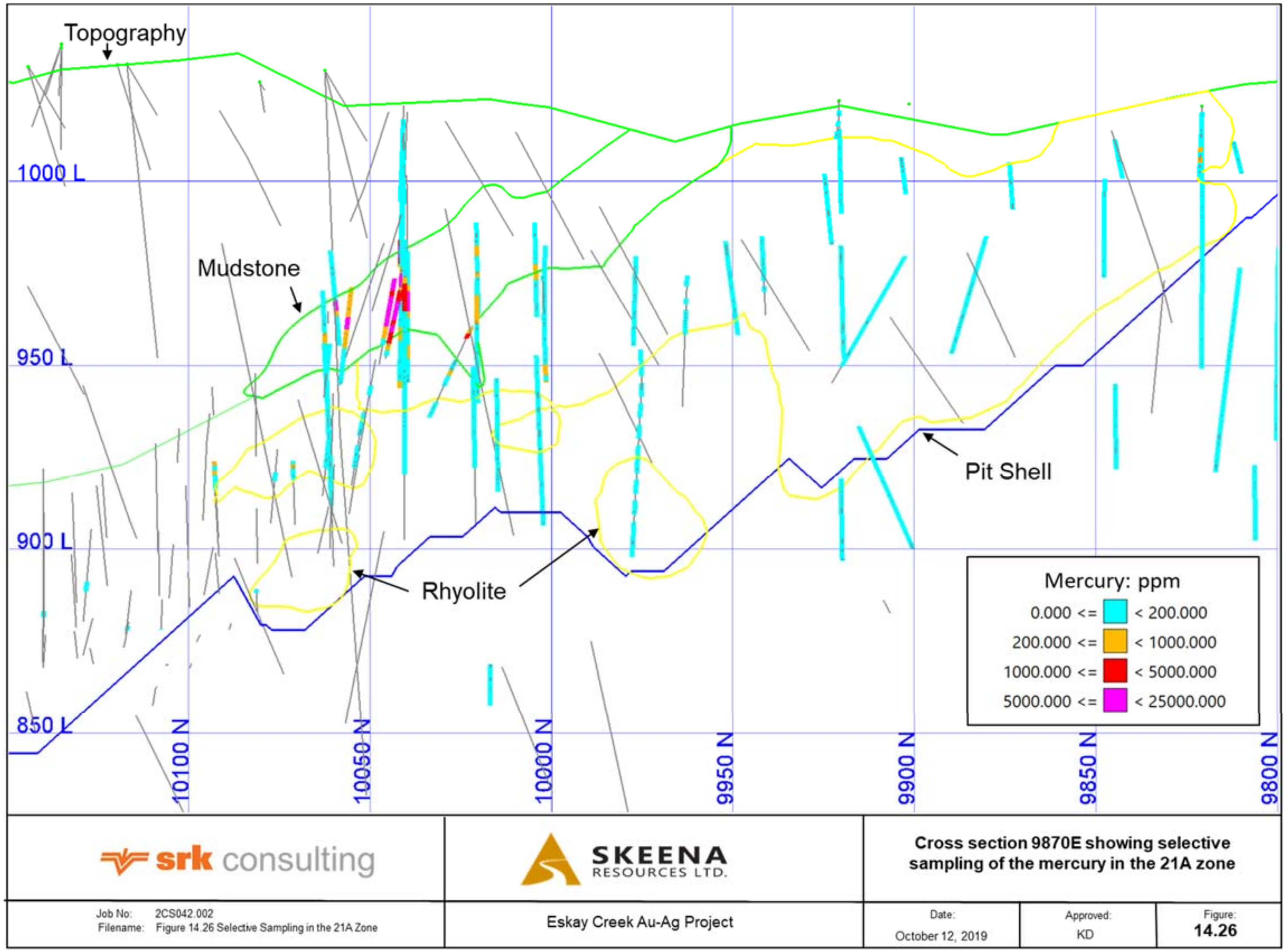


Table 14-27: Summary Statistics for Drill Hole Epithermal Element Assays by Zone (Sb, Hg, As)

Domain	Zone	Rock Type	No. of Samples	Mean	CV	Min	Max
Antimony (ppm)							
22	10	RHY	1,583	449	5.77	3	64,240
21A	201	RHY	3,191	933	8.96	0	328,000
	202	HWA/MS	838	20,241	3.66	10	591,000
21C	301	RHY	6,760	363	2.35	9	31,900
	302	HWA/MS	1,691	2,538	3.82	17	162,500
21B	401	RHY	5,883	4,101	4.83	0	483,500
	402	HWA/MS	7,985	18,038	2.55	50	545,000
21BE	501	RHY	6,515	2,760	3.71	50	197,000
	502	HWA/MS	3,497	7,282	2.71	50	516,400
21E	601	RHY	70	776	1.94	50	9,200
	602	HWA/MS	707	8,216	3.91	41	388,000
HW	702	HWA/MS	8,734	1,978	3.56	46	334,000
NEX	801	RHY	5,863	2,082	5.26	5	342,000
	802	HWA/MS	4,284	2,801	4.01	50	340,000
PMP	95	RHY	1,072	2,749	5.48	50	382,000
109	99	RHY	4,871	270	3.57	50	50,800
Mercury (ppm)							
22	10	RHY	1,583	7	1.77	0.0	140
21A	201	RHY	2,871	65	4.61	0.0	7,530
	202	HWA/MS	639	1,444	2.69	0.0	29,000
21C	301	RHY	6,637	14	1.97	0.1	887
	302	HWA/MS	1,677	42	1.85	0.5	723
21B	401	RHY	5,598	169	4.46	0.5	20,020
	402	HWA/MS	7,707	965	2.86	0.5	44,775
21BE	501	RHY	6,471	93	4.50	0.5	11,930
	502	HWA/MS	3,452	298	2.90	0.5	17,590
21E	601	RHY	61	26	2.14	1.0	382
	602	HWA/MS	695	30	2.85	0.5	1,898
HW	702	HWA/MS	8,600	35	1.36	0.5	600
NEX	801	RHY	5,745	30	2.85	0.0	2,488
	802	HWA/MS	4,255	41	1.97	0.5	1,378
PMP	95	RHY	1,079	39	4.43	0.5	4,160
109	99	RHY	4,782	14	1.28	0.5	387
Arsenic (ppm)							
22	10	RHY	1,581	1,260	2.17	10	39,200
21A	201	RHY	3,210	499	5.00	10	82,400
	202	HWA/MS	870	22,613	4.49	10	540,000
21C	301	RHY	6,761	248	0.99	12	5,300
	302	HWA/MS	1,684	543	4.14	12	47,600
21B	401	RHY	4,497	823	3.13	0	120,000

Domain	Zone	Rock Type	No. of Samples	Mean	CV	Min	Max
	402	HWA/MS	6,694	1,804	5.60	50	530,000
21BE	501	RHY	4,498	1,879	1.53	50	19,500
	502	HWA/MS	2,574	1,320	1.66	50	22,000
21E	601	RHY	70	387	1.62	50	3,700
	602	HWA/MS	707	331	1.40	50	5,000
HW	702	HWA/MS	7,352	767	1.99	50	100,000
NEX	801	RHY	5,521	577	2.16	25	27,000
	802	HWA/MS	3,671	658	1.22	50	11,800
PMP	95	RHY	1,079	958	5.12	50	110,000
109	99	RHY	3,799	611	1.50	50	10,800

Table 14-28: Summary Statistics for Drill Hole Base Metal Assays by Zone (Pb, Cu, Zn)

Domain	Zone	Rock Type	No. of Samples	Mean	CV	Min	Max
Lead (%)							
22	10	RHY	1,609	0.11	4.75	0.00	15.09
21A	201	RHY	3,439	0.13	3.34	0.000	11.92
	202	HWA/MS	852	0.10	4.57	0.000	7.15
21C	301	RHY	7,478	0.13	3.30	0.000	14.80
	302	HWA/MS	1,872	0.82	5.01	0.000	20.20
21B	401	RHY	6,965	0.85	2.48	0.005	24.21
	402	HWA/MS	8,020	2.05	1.76	0.005	53.15
21BE	501	RHY	5,060	1.16	2.47	0.005	24.40
	502	HWA/MS	3,427	2.36	1.69	0.005	20.90
21E	601	RHY	104	0.11	2.39	0.005	1.90
	602	HWA/MS	861	0.37	3.27	0.005	10.75
HW	702	HWA/MS	9,213	2.53	1.61	0.000	52.00
NEX	801	RHY	6,496	0.83	2.65	0.000	29.49
	802	HWA/MS	4,193	1.81	2.06	0.001	25.83
PMP	95	RHY	1,188	0.17	2.57	0.005	5.30
109	99	RHY	4,725	1.57	1.77	0.005	65.36
Copper (%)							
22	10	RHY	1,583	0.014	4.96	0.000	1.44
21A	201	RHY	3,428	0.022	3.37	0.000	1.34
	202	HWA/MS	854	0.024	3.41	0.000	1.51
21C	301	RHY	7,477	0.043	3.09	0.001	5.44
	302	HWA/MS	1,872	0.182	2.63	0.001	5.24
21B	401	RHY	6,815	0.161	3.29	0.001	5.66
	402	HWA/MS	8,005	0.519	2.26	0.005	26.40
21BE	501	RHY	5,033	0.306	3.03	0.005	10.14
	502	HWA/MS	3,402	0.584	2.07	0.005	10.70
21E	601	RHY	104	0.041	2.23	0.005	0.80

Domain	Zone	Rock Type	No. of Samples	Mean	CV	Min	Max
	602	HWA/MS	860	0.110	3.18	0.005	3.95
HW	702	HWA/MS	9,192	0.408	2.00	0.003	35.00
NEX	801	RHY	6,494	0.149	3.66	0.000	8.58
	802	HWA/MS	4,182	0.331	2.45	0.001	7.30
PMP	95	RHY	1,188	0.670	3.10	0.005	4.22
109	99	RHY	4,314	0.039	5.90	0.005	5.70
Zinc (%)							
22	10	RHY	1,609	0.146	4.04	0.001	15.36
21A	201	RHY	3,439	0.210	2.87	0.000	13.52
	202	HWA/MS	852	0.220	3.64	0.004	12.50
21C	301	RHY	7,477	0.241	3.00	0.001	22.58
	302	HWA/MS	1,872	1.459	2.47	0.002	33.10
21B	401	RHY	6,964	1.442	2.67	0.002	44.40
	402	HWA/MS	8,028	3.623	1.78	0.005	33.95
21BE	501	RHY	5,063	1.966	2.49	0.005	36.90
	502	HWA/MS	3,433	4.049	1.70	0.005	39.44
21E	601	RHY	104	0.199	2.41	0.005	3.73
	602	HWA/MS	861	0.691	3.11	0.010	19.08
HW	702	HWA/MS	9,213	3.867	1.59	0.005	48.88
NEX	801	RHY	6,499	1.378	2.66	0.001	35.00
	802	HWA/MS	4,193	2.714	2.02	0.005	33.90
PMP	95	RHY	1,188	0.325	3.05	0.005	21.00
109	99	RHY	4,720	2.400	1.59	0.010	31.80

Table 14-29: Correlation Coefficient (R value) Between the Epithermal and Base Metal Elements in Relation to Gold and Silver by Zone (Sb, Hg, As)

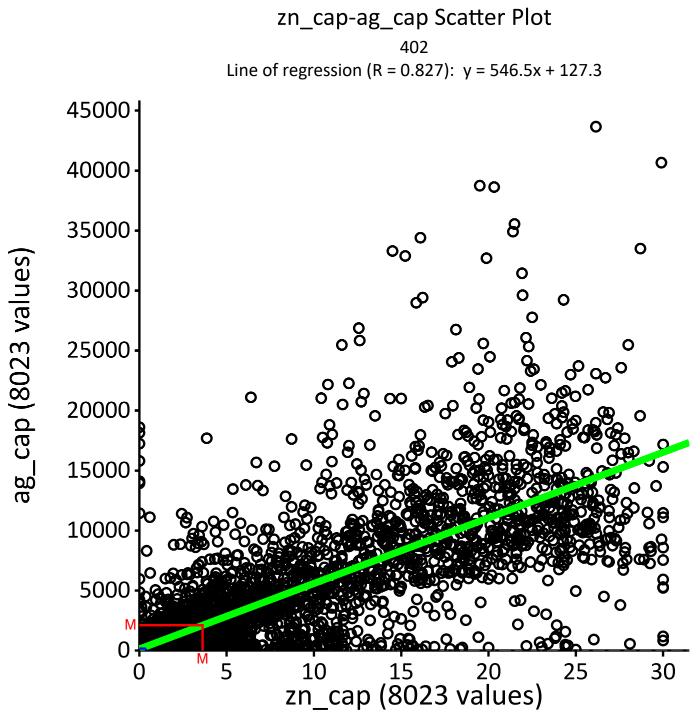
Zone		Gold	Silver		Gold	Silver		Gold	Silver
10	Antimony	0.35	0.34	Mercury	0.24	0.16	Arsenic	0.20	-0.13
201		0.33	0.19		0.23	0.29		0.44	0.04
202		0.39	0.43		0.57	0.40		0.53	-0.02
301		0.09	0.55		0.26	0.56		0.11	0.11
302		0.22	0.28		0.32	0.44		0.02	0.01
401		0.40	0.55		0.44	0.57		0.12	0.06
402		0.40	0.53		0.59	0.61		0.12	0.03
501		0.59	0.53		0.37	0.32		0.07	-0.01
502		0.45	0.67		0.46	0.39		0.17	0.24
601		0.33	0.82		0.21	0.86		0.23	-0.05
602		0.17	0.26		0.40	0.56		-0.05	-0.01
702		0.40	0.56		0.47	0.47		0.26	0.19
801		0.46	0.57		0.42	0.46		0.17	0.08
802		0.30	0.61		0.35	0.45		0.10	0.15
95		0.44	0.64		0.38	0.75		0.14	-0.06
99		0.06	0.17		0.19	0.48		-0.02	0.07

Table 14-30: Correlation Coefficient (R value) Between the Epithermal and Base Metal Elements in Relation to Gold and Silver by Zone (Pb, Cu, Zn)

Zone		Gold	Silver		Gold	Silver		Gold	Silver
10	Lead	0.17	0.17	Copper	0.22	0.51	Zinc	0.19	0.25
201		-0.04	0.31		0.02	0.56		-0.03	0.36
202		0.24	0.86		0.20	0.53		0.34	0.81
301		0.07	0.44		0.09	0.56		0.11	0.43
302		0.24	0.32		0.23	0.36		0.27	0.34
401		0.56	0.69		0.61	0.85		0.57	0.71
402		0.67	0.80		0.66	0.90		0.70	0.83
501		0.62	0.61		0.63	0.72		0.62	0.61
502		0.52	0.70		0.52	0.87		0.53	0.70
601		0.12	0.77		0.13	0.82		0.14	0.81
602		0.75	0.90		0.77	0.93		0.74	0.90
702		0.32	0.30		0.44	0.51		0.34	0.32
801		0.43	0.55		0.60	0.76		0.48	0.58
802		0.34	0.45		0.51	0.69		0.39	0.49
95		0.30	0.79		0.21	0.87		0.30	0.83
99		0.31	0.60		0.09	0.59		0.33	0.58

Note: blue text = low coefficient of correlation (<0.30); red = high coefficient of correlation (>0.70).

Figure 14-27: Scatterplot of Silver Versus Zinc in Zone 402, Showing a Strong Coefficient of Correlation but a High Degree of Scatter (21B - Mudstone)



Note: Figure prepared by SRK, 2019.

Table 14-31: High-Grade Capping Statistics for the Epithermal Elements by Zone (Sb, Hg, As)

Domain	Zone	Number of Assay Samples	Uncapped		Capped		Maximum	Topcut	Number Cut	% Samples	% Metal Lost
			Mean	CV	Mean	CV					
Antimony (ppm)											
22	10	1,583	449	5.77	390	3.99	64,240	20,000	4	0.3	13.0
21A	201	3,191	933	8.96	690	4.55	328,000	40,000	8	0.3	26.1
	202	838	20,241	3.66	19,147	3.52	591,000	400,000	14	1.7	5.4
21C	301	6,760	363	2.35	358	2.05	31,900	9,500	6	0.1	1.5
	302	1,691	2,538	3.82	2,440	3.41	162,500	70,000	4	0.2	3.9
21B	401	5,882	4,101	4.83	3,870	4.21	483,500	200,000	13	0.2	5.6
	402	7,986	18,037	2.55	17,925	2.50	545,000	375,000	16	0.2	0.6
21Be	501	6,514	2,756	3.71	2,719	3.58	197,000	110,000	7	0.1	1.3
	502	3,497	7,282	2.71	7,094	2.35	516,400	150,000	7	0.2	2.6
21E	601	70	776	1.94	669	1.55	9,200	4,200	3	4.3	13.8
	602	707	8,216	3.91	7,497	3.36	388,000	210,000	4	0.6	8.7
NEX	702	8,732	1,978	3.56	1,897	2.63	334,000	65,000	8	0.1	4.1
HW	801	5,867	2,082	5.26	2,040	4.89	342,000	90,000	6	0.1	2.0

Domain	Zone	Number of Assay Samples	Uncapped Mean	Uncapped CV	Capped Mean	Capped CV	Maximum	Topcut	Number Cut	% Samples	% Metal Lost
	802	4,282	2,802	4.01	2,644	3.08	340,000	110,000	6	0.1	5.6
PMP	95	1,072	2,749	5.48	2,323	3.47	382,000	90,000	6	0.6	15.5
109	99	4,871	270	3.57	250	1.60	50,800	6,000	7	0.1	7.3
Mercury (ppm)											
22	10	1,583	7	1.77	7	1.62	140	75	12	0.8	2.5
21A	201	2,871	65	4.61	61	3.71	7,530	3,000	7	0.2	6.7
	202	639	1,444	2.69	1,383	2.54	29,000	20,000	6	0.9	4.2
21C	301	6,637	14	1.97	14	1.74	887	300	7	0.1	1.4
	302	1,677	42	1.85	42	1.76	723	500	10	0.6	1.6
21B	401	5,598	169	4.46	163	3.92	20,020	10,000	6	0.1	3.3
	402	7,707	965	2.86	958	2.80	44,775	25,900	9	0.1	0.7
21Be	501	6,471	93	4.50	89	3.81	11,930	6,000	10	0.2	4.1
	502	3,452	298	2.90	290	2.61	17,590	9,000	9	0.3	2.6
21E	601	61	26	2.14	22	1.58	382	150	1	1.6	14.7
	602	695	30	2.85	28	1.71	1,898	300	4	0.6	9.4
NEX	702	8,598	35	1.36	35	1.33	600	400	9	0.1	0.4
HW	801	5,749	30	2.85	29	2.47	2,488	1,000	7	0.1	2.4
	802	4,253	41	1.97	41	1.83	1,378	700	8	0.2	1.4
PMP	95	1,079	39	4.43	33	2.48	4,160	900	6	0.6	14.1
109	99	4,782	14	1.28	14	1.23	387	170	5	0.1	0.6
Arsenic (ppm)											
22	10	1,581	1,260	2.17	1,218	1.97	39,200	15,000	10	0.6	3.3
21A	201	3,210	499	5.00	456	3.44	82,400	20,000	5	0.2	8.6
	202	870	22,613	3.02	20,960	2.81	540,000	300,000	18	2.1	7.3
21C	301	6,761	248	0.99	245	0.87	53,000	2,400	11	0.2	0.9
	302	1,684	543	4.14	423	1.31	47,600	5,500	9	0.5	22.0
21B	401	4,501	823	3.12	782	2.07	120,000	20,000	10	0.2	5.0
	402	6,690	1,804	5.60	1,615	2.42	530,000	55,000	10	0.1	10.5
21Be	501	4,496	1,880	1.53	1,879	4.36	19,500	16,000	3	0.1	0.1
	502	2,574	1,320	1.66	1,316	1.64	22,000	19,000	7	0.3	0.3
21E	601	70	387	1.62	377	1.53	3,700	3,000	1	1.4	2.7
	602	707	331	1.40	324	1.28	5,000	3,000	4	0.6	1.9
NEX	702	7,350	767	1.99	750	1.26	100,000	10,000	13	0.2	2.2
HW	801	5,525	577	2.16	466	1.89	27,000	14,000	9	0.2	19.2
	802	3,669	658	1.22	654	1.18	11,800	6,000	9	0.2	0.5
PMP	95	1,079	958	5.12	770	1.84	110,000	9,900	12	1.1	19.6
109	99	3,799	611	1.50	608	1.46	10,800	8,000	6	0.2	0.4

Table 14-32: High-grade Capping Statistics for the Base Metals by Zone (Pb, Cu, Zn)

Domain	Zone	Number of Assays	Uncapped		Capped		Maximum	Topcut	Number Cut	% Samples	% Metal lost
			Mean	CV	Mean	CV					
Lead (%)											
22	10	1,608	0.109	4.75	0.101	3.58	15.09	4.0	3	0.2	7.3
21A	201	3,437	0.127	3.34	0.122	2.67	11.90	4.0	4	0.1	3.9
	202	851	0.102	4.57	0.093	3.88	7.15	2.3	4	0.5	8.8
21C	301	7,477	0.129		0.125	2.67	14.80	5.0	8	0.1	3.1
	302	1,869	0.820	2.50	0.817	2.48	20.20	17.5	5	0.3	0.4
21B	401	6,970	0.851	2.48	0.848	2.45	24.20	17.5	11	0.2	0.4
	402	8,015	2.049	1.76	2.044	1.74	53.15	20.0	2	0.0	0.2
21Be	501	5,058	1.159	2.47	1.157	2.47	24.40	20.0	2	0.0	0.2
	502	3,427	2.360	1.69	2.360	1.69	20.90	-	0	0.0	0.0
21E	601	104	0.105	2.39	0.075	1.43	1.90	-	0	0.0	0.0
	602	861	0.369	3.27	0.357	1.25	10.75	7.5	5	0.6	3.3
NEX	702	9,208	2.532	1.61	2.528	1.60	52.00	26.0	3	0.0	0.2
HW	801	6,499	0.829	2.65	0.825	2.62	29.40	20.0	6	0.1	0.5
	802	4,191	1.806	2.06	1.804	2.06	25.83	22.0	2	0.0	0.1
PMP	95	1,188	0.168	2.57	0.159	2.18	5.30	3.0	9	0.8	5.4
109	99	4,725	1.570	5.97	1.560	1.68	65.36	24.0	6	0.1	0.6
					Copper (%)						
22	10	1,513	0.015	4.84	0.013	2.84	1.44	0.5	5	0.3	13.3
21A	201	3,388	0.022	3.35	0.021	2.90	1.34	0.7	8	0.2	4.5
	202	844	0.024	3.39	0.022	2.33	1.51	0.5	4	0.5	8.3
21C	301	7,477	0.043	3.09	0.042	2.39	5.44	1.2	9	0.1	2.3
	302	1,872	0.182	2.63	0.176	2.47	5.24	3.3	13	0.7	3.3
21B	401	6,820	0.161	3.29	0.160	3.25	5.66	4.0	20	0.3	0.6
	402	8,000	0.520	2.26	0.502	1.91	26.40	7.0	12	0.2	3.5
21Be	501	5,031	0.305	3.03	0.295	2.80	10.10	6.0	34	0.7	3.3
	502	3,402	0.584	2.07	0.582	2.05	10.70	8.0	5	0.1	0.3
21E	601	104	0.041	2.23	0.041	2.23	0.80	-	0	0.0	0.0
	602	860	0.110	3.18	0.106	2.98	3.98	2.0	5	0.6	3.6
HW	702	9,191	0.408	2.00	0.403	1.79	35.00	5.0	11	0.1	1.2
NEX	801	6,493	0.149	3.65	0.147	3.58	8.58	5.0	17	0.3	1.3
	802	4,180	0.331	2.45	0.329	2.42	7.30	5.0	10	0.2	0.6
PMP	95	1,188	0.067	3.10	0.059	2.06	4.22	1.0	6	0.5	11.9
109	99	4,314	0.039	5.90	0.037	5.10	5.70	3.5	5	0.1	5.1
Zinc (%)											
22	10	1,609	0.146	4.04	0.136	2.92	15.36	4.5	6	0.4	6.8
21A	201	3,438	0.210	2.87	0.200	2.40	15.52	4.0	18	0.5	4.8
	202	852	0.220	3.64	0.191	2.15	12.50	4.0	7	0.8	13.2
21C	301	7,477	0.241	3.00	0.227	2.20	22.58	5.3	17	0.2	5.8
	302	1,872	1.459	2.47	1.441	2.41	33.10	25.0	12	0.6	1.2

Domain	Zone	Number of Assays	Uncapped		Capped		Maximum	Topcut	Number Cut	% Samples	% Metal lost
			Mean	CV	Mean	CV					
21B	401	6,969	1.441	2.67	1.431	2.62	44.40	30.0	18	0.3	0.7
	402	8,023	3.625	1.78	3.623	1.77	33.95	30.0	11	0.1	0.1
21Be	501	5,061	1.965	2.49	1.956	2.47	36.90	30.0	17	0.3	0.5
	502	3,433	4.049	1.70	4.035	1.69	39.44	30.0	20	0.6	0.3
21E	601	104	0.199	2.41	0.199	2.41	3.73	-	0	0.0	0.0
	602	861	0.691	3.11	0.666	2.96	19.08	13.0	6	0.7	3.6
NEX	702	9,212	3.867	1.59	3.860	1.58	48.80	30.0	17	0.2	0.2
HW	801	6,503	1.379	2.65	1.376	2.63	35.00	30.0	10	0.2	0.2
	802	4,191	2.715	2.02	2.711	2.02	33.90	30.0	10	0.2	0.1
PMP	95	1,188	0.325	3.05	0.282	1.76	21.00	6.0	9	0.8	13.2
109	99	4,720	2.410	1.59	2.401	1.57	31.80	25.0	16	0.3	0.4

14.17.4 Compositing

Epithermal and base metal elements were composited to 2 m, using the same intervals determined for gold and silver composites. Since the epithermal and base metal elements are all considered penalty elements, a conservative approach was undertaken for compositing. To assure that the estimate wasn't unduly affected by the missing or unsampled intervals, the unsampled intervals were allocated a default value of -66 prior to compositing and ignored during estimation thereby removing the risk of underestimating the values of the penalty elements. Table 14-33 shows the mean and CV of the non-declustered, capped 2 m composites.

Selecting an appropriate cell declustering size was problematic as selective sampling of primarily higher-grade samples was previously done. The drill density was generally closer in high-grade pockets, as opposed to samples within lower-grade areas, where intervals were more likely to be missing.

14.17.5 Block Model

The epithermal and base metal elements used the same block model geometry and extents as the gold and silver block model with 9 m x 9 m x 4 m parent blocks, and 3 m x 3 m x 2 m subblocks, where subblocks occur around the zone boundaries.

14.17.6 Estimation Parameters

Due to selective sampling, insufficient data were available to produce reliable variograms necessary for a kriged estimate. Therefore, the block model grades were estimated using ID^2. A NN model was also estimated for model validation purposes. The low-grade envelope used in the gold–silver model was not populated for the epithermal and base metal elements.

Sensitivity estimates were conducted for each element using variable estimation parameters to determine the least biased grade estimate. Estimation parameters such as the number of samples, proportion of blocks estimated in each zone, boundary analysis, search distances, number of passes, and declustering were successively revised until the best estimate was achieved.

Table 14-33: 2 m Composite Statistics Showing Number of Composites, Mean and CV of the Epithermal and Base Metals by Zone

Domain	Zone	# of Comps	Antimony		Mercury		Arsenic	
			Mean	CV	Mean	CV	Mean	CV
22	10	1,028	344	3.47	6	1.40	1,148	1.84
21A	201	2,290	580	3.90	52	2.99	426	5.11
	202	506	14,210	3.83	1,160	2.43	17,830	2.76
21C	301	4,228	326	1.55	13	1.39	243	0.80
	302	1,007	1,946	2.72	35	1.43	409	1.26
21B	401	3,817	3,444	3.78	152	3.67	757	1.88
	402	4,253	15,616	2.36	856	2.80	1,558	2.23
21Be	501	2,811	2,470	3.11	82	3.56	1,878	1.47
	502	1,884	6,155	2.10	253	2.48	1,305	1.58
21E	601	63	644	1.33	22	1.36	412	1.39
	602	456	5,428	3.21	23	1.56	322	1.10
NEX	702	4,973	1,619	2.17	31	1.17	711	1.09
HW	801	3,631	1,444	3.50	24	1.88	530	1.69
	802	2,278	2,133	2.65	35	1.57	624	0.98
PMP	95	654	1,958	2.85	29	1.99	716	1.63
109	99	2,677	251	1.23	14	1.02	603	1.25
	Sub-total	***36,556***						
Domain	**Zone**	**# of comps**	**Lead**		**Copper**		**Zinc**	
			Mean	**CV**	**Mean**	**CV**	**Mean**	**CV**
22	10	1,028	0.092	3.14	0.011	2.99	0.122	2.72
21A	201	2,290	0.116	2.16	0.020	2.31	0.189	1.94
	202	506	0.072	2.76	0.020	1.82	0.164	2.20
21C	301	4,228	0.112	2.14	0.037	1.91	0.207	1.79
	302	1,007	0.668	1.96	0.144	2.05	1.169	1.95
21B	401	3,817	0.819	2.21	0.144	2.98	1.349	2.33
	402	4,253	1.839	1.67	0.451	1.84	3.253	1.71
21Be	501	2,811	1.037	2.30	0.261	2.68	1.752	2.30
	502	1,884	2.171	1.58	0.526	1.87	3.678	1.58
21E	601	63	0.098	1.92	0.040	1.69	0.185	1.88
	602	456	0.260	2.53	0.077	2.46	0.496	2.37
NEX	702	4,973	2.305	1.46	0.356	1.63	3.516	1.46
HW	801	3,631	0.742	2.32	0.121	3.23	1.215	2.31
	802	2,278	1.615	1.96	0.283	2.31	2.408	1.89
PMP	95	654	0.141	1.65	0.055	1.63	0.263	1.59
109	99	2,677	1.490	1.34	0.034	4.41	2.288	1.33
	Sub-total	***36,556***						

* based on composites that were not declustered

The final parameters selected for the epithermal and base metals estimates are presented in Table 14-34. A discretization grid of 3 x 3 x 2 was used during all estimation runs, and the anisotropic search distance determined from gold variography was used. The estimate was generated in one pass using dynamic anisotropy and 2.5 x the variogram range (see Section 14.7 and Section 14.8). A minimum of three and maximum of 16 composites were used per block to ensure that at least two drill holes were used for the estimate. An octant search was used to aid in declustering, and hard boundaries were honoured in all zones.

14.17.7 Block Model Validation

The block model estimates were validated for the elements using several methods to ensure an unbiased estimate; these include: a thorough visual review of the block model grades in relation to the informing drill hole samples, comparisons with NN estimates and, grade distribution evaluations using swath plots.

14.17.8 Open Pit Model, Visual Validation

Detailed section and plan view visual inspections of the block model were conducted for each element to evaluate final estimated grades with the neighbouring informing composites. In addition, domain coding accuracies were checked during this stage. Figure 14-28 shows estimated mercury block grades in relation to 2 m mercury composite intervals in the 21A Zone. Overall, the data show good agreement, and no major discrepancies between block grades and composites were observed.

14.17.9 Comparison of Interpolation Models

The ID^2 model was compared against the NN model to check for the occurrence of global bias. Before the final ID^2 model was confirmed, sensitivity interpolation estimates, using variable modelling parameters, were conducted successively to minimize global bias. Although variability exists between the different zones for both the ID^2 and NN estimates, there is an average difference of less than 5% for all elements (Table 14-35), confirming that global bias is not a concern for the epithermal and base metal estimates.

14.17.10 Swath Plots

Swath plots were generated in three orthogonal directions to graphically display grade distribution in each of the zones in north–south, east–west and horizontal directions throughout the deposit. Grade variations from the ID^2 model were compared to the NN grade distribution, along with clustered and declustered composite data.

Declustering weights were applied to the composite data for model validation. Appropriate declustering cell sizes were not established due to artefacts produced by numerous missing samples. All zones and all elements (Sb, As, Hg, Pb, Cu, Zn) were visually assessed using swath plots in three directions. For all zones of economic interest, the swath plots showed acceptable correspondence between grade distributions, although the ID^2 model inherently smoothed the results.

An example of mercury swath plots in the 21A domain is shown in Figure 14-29, which depicts the block model grade (black line), NN grade (yellow), and declustered composite grade (blue line).

14.17.11 Epithermal Element and Base Metal Concentrations

The average estimated epithermal and base metal element concentrations remaining in each domain with the pit shell at the resource cut off grade of AuEq > 0.7 g/t is shown in Table 14-36.

Table 14-34: Interpolation Parameters for the Epithermal and Base Metal Elements by Zone

BMZONE	Search Pass	Orientation	Search X	Search Y	Search Z	Number of Composites Minimum	Number of Composites Maximum	Max Composites per Drill Hole	Max Composites per Octant	Boundary
10	1	135/0/45	75	52.5	60	3	16	2	2	Hard
201	1	D245	114	64.5	30	3	16	2	2	Hard
202	1	D245	67.5	30	30	3	16	2	2	Hard
3011	1	D245	52.5	30	22.5	3	16	2	2	Hard
3012	1	D245	67.5	30	30	3	16	2	2	Hard
302	1	D245	67.5	37.5	16.5	3	16	2	2	Hard
401	1	D245	45	37.5	22.5	3	16	2	2	Hard
402	1	D245	142.5	90	7.5	3	16	2	2	Hard
501	1	D4	52.5	30	15	3	16	2	2	Hard
502	1	D3	37.5	15	9	3	16	2	2	Hard
601	1	D1	60	52.5	22.5	3	16	2	2	Hard
602	1	D1	60	52.5	22.5	3	16	2	2	Hard
6021	1	D1	37.5	22.5	7.5	3	16	2	2	Hard
7022	1	D2	30	22.5	15	3	16	2	2	Hard
7023	1	D3	52.5	45	22.5	3	16	2	2	Hard
7025	1	D245	30	22.5	15	3	16	2	2	Hard
7026	1	D6	60	60	37.5	3	16	2	2	Hard
8010	1	020/0/85	90	60	45	3	16	2	2	Hard
8012	1	D2	90	60	45	3	16	2	2	Hard
8015	1	D245	45	45	30	3	16	2	2	Hard
8016	1	D6	90	60	45	3	16	2	2	Hard
8022	1	D2	63	30	15	3	16	2	2	Hard
8025	1	D245	52.5	33	15	3	16	2	2	Hard
95	1	350/-26/-44	60	30	15	3	16	2	2	Hard
99	1	296/-54/172	67.5	30	30	3	16	2	2	Hard

*BMZONE = Zone split by lithology and spatial uniqueness

Figure 14-28: **Example of Visual Validation of Mercury Distribution in the 21A Zone**

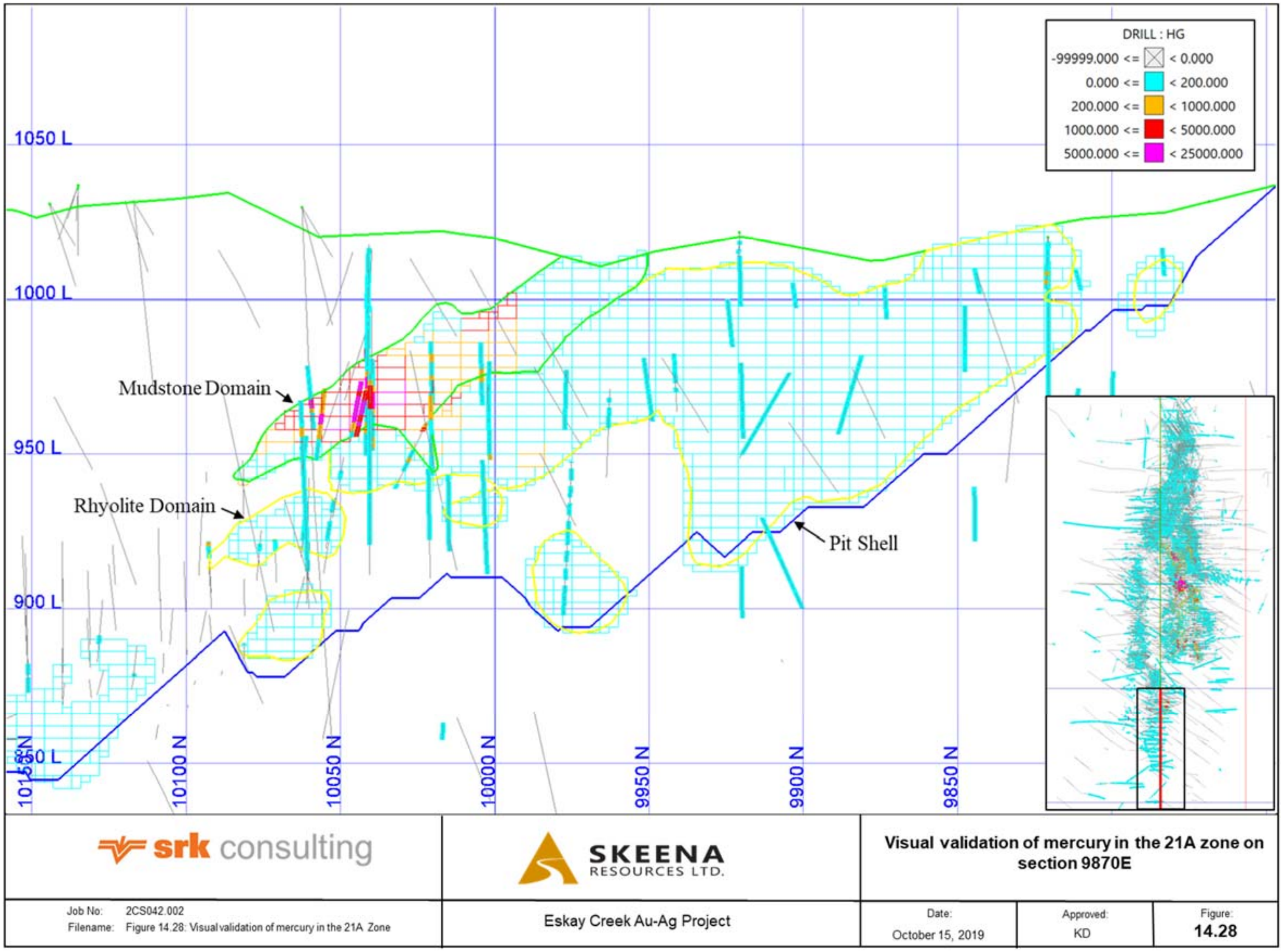


Table 14-35: Comparison ID² vs NN Estimates Within Each Zone for the Epithermal and Base Metal Elements

Zone	Antimony			Zone	Mercury			Zone	Arsenic		
	ID	NN	ID vs NN %		ID	NN	ID vs NN %		ID	NN	ID vs NN %
10	299	286	5	10	6	6	-3.4	10	1,076	992	9
201	401	400	0	201	46	45	0.2	201	310	297	4
202	6,809	6,021	13	202	675	549	23.0	202	9,194	8,737	5
301	266	262	2	301	9	9	4.7	301	251	238	6
302	1,435	1,426	1	302	30	29	4.5	302	471	488	-4
401	1,415	1,330	6	401	74	66	11.5	401	605	614	-2
402	9,342	9,116	2	402	513	501	2.3	402	1,204	1,195	1
501	1,616	1,583	2	501	78	78	0.7	501	1,713	1,699	1
502	4,828	4,403	10	502	242	231	4.5	502	1,193	1,250	-5
601	640	616	4	601	25	22	10.4	601	519	529	-2
602	5,124	5,384	-5	602	22	22	-0.5	602	380	369	3
702	1,376	1,251	10	702	29	27	6.7	702	601	596	1
801	721	710	2	801	16	15	6.6	801	422	418	1
802	1,472	1,485	-1	802	27	26	5.6	802	553	558	-1
95	1,458	1,276	14	95	27	26	2.9	95	653	596	9
99	257	262	-2	99	14	14	2.2	99	664	674	-2
Sub-totals			***4***				***5***				***2***
Zone	**Lead**			**Zone**	**Copper**			**Zone**	**Zinc**		
	ID	**NN**	**ID vs NN %**		**ID**	**NN**	**ID vs NN %**		**ID**	**NN**	**ID vs NN %**
10	0.095	0.093	2	10	0.012	0.013	-8	10	0.134	0.139	-4
201	0.132	0.127	4	201	0.021	0.020	5	201	0.215	0.210	2
202	0.058	0.061	-5	202	0.019	0.020	-5	202	0.137	0.141	-3
301	0.102	0.104	-2	301	0.029	0.029	0	301	0.175	0.178	-2
302	0.536	0.514	4	302	0.109	0.102	7	302	0.921	0.875	5
401	0.491	0.463	6	401	0.060	0.053	13	401	0.809	0.773	5
402	1.325	1.320	0	402	0.303	0.301	1	402	2.280	2.273	0
501	0.668	0.655	2	501	0.155	0.150	3	501	1.133	1.103	3
502	1.557	1.509	3	502	0.337	0.330	2	502	2.608	2.525	3
601	0.088	0.090	-2	601	0.036	0.037	-3	601	0.180	0.190	-5
602	0.094	0.092	2	602	0.031	0.030	3	602	0.212	0.210	1
702	1.697	1.638	4	702	0.251	0.237	6	702	2.572	2.471	4
801	0.497	0.460	8	801	0.066	0.058	14	801	0.799	0.732	9
802	1.212	1.168	4	802	0.188	0.172	9	802	1.792	1.742	3
95	0.116	0.116	0	95	0.044	0.042	5	95	0.213	0.219	-3
99	1.555	1.520	2	99	0.037	0.033	12	99	2.499	2.444	2
Sub-totals			***2***				***4***				***1***

Figure 14-29: Swath Plots of Mercury in the 21A Zone Rhyolite (left) and Mudstone (right)

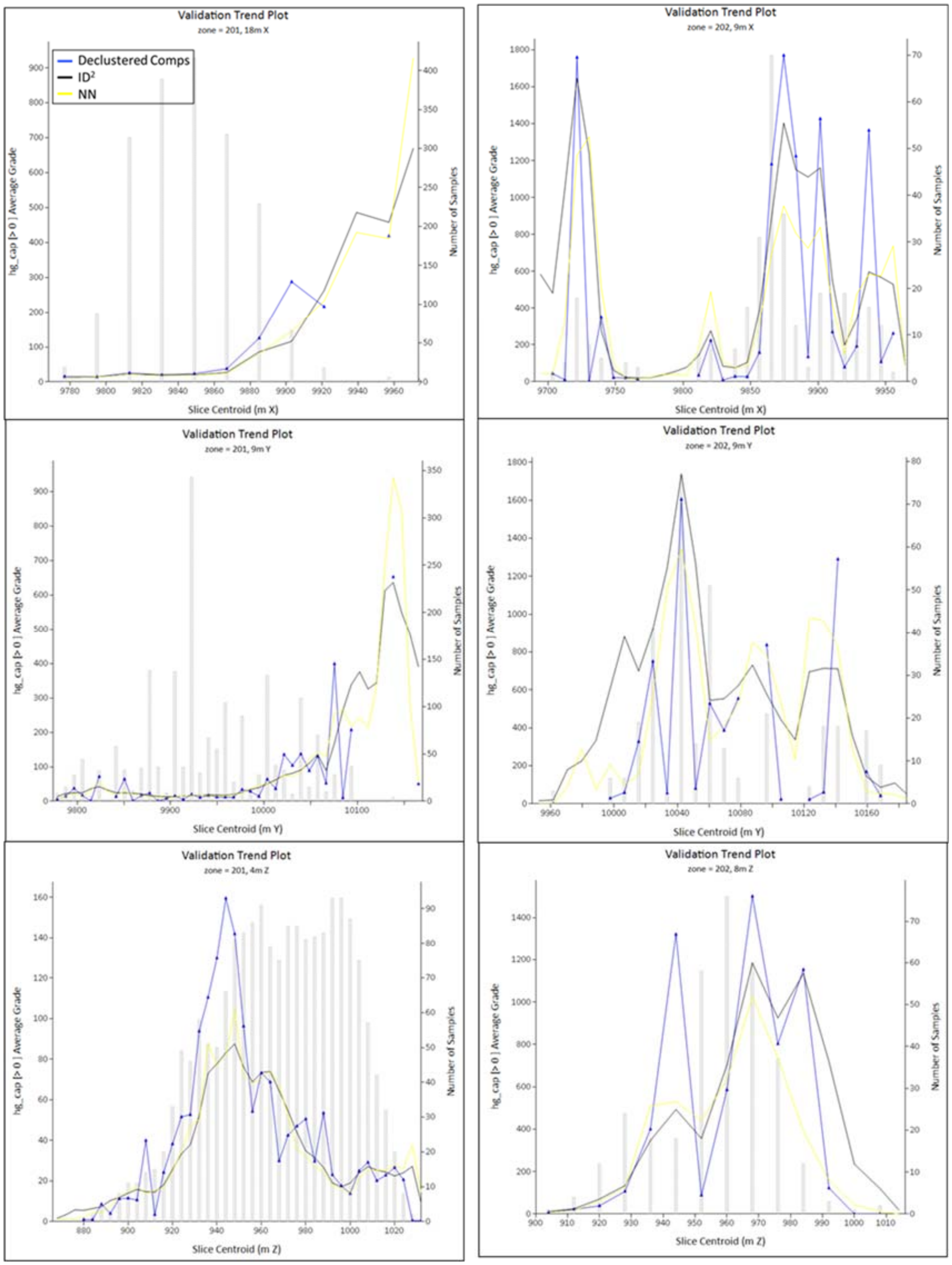


Table 14-36: Average Estimated Epithermal and Base Metal Concentrations Remaining in Each of the Domains Within the Pit Shell at an AuEq Cut-Off Grade of 0.7 g/t

Domain	Antimony (ppm)	Mercury (ppm)	Arsenic (ppm)	Lead (%)	Copper (%)	Zinc (%)
22	227	6	696	0.139	0.015	0.166
21A	1,606	159	2,004	0.122	0.021	0.204
21C	528	14	287	0.208	0.049	0.356
21B	1,520	73	517	0.480	0.064	0.787
21Be	1,677	79	1,522	0.566	0.119	0.952
21E_n	3,339	18	339	0.068	0.025	0.159
HW	1,605	29	458	1.441	0.219	2.202
NEX	683	19	485	1.458	0.157	2.078
PMP	1,313	19	662	0.111	0.041	0.201
109	255	15	694	1.551	0.038	2.521
Total	***1,140***	***57***	***767***	***0.433***	***0.060***	***0.684***

14.18 Factors That May Affect the Mineral Resource Estimate

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

- Changes to long-term metal price assumptions;
- Changes in local interpretations of mineralization geometry and continuity of mineralized zones;
- Changes to the density values applied to the mineralized zones;
- Changes to geological shape and continuity assumptions;
- Potential for unrecognized bias in the assay results from legacy drilling where there was limited documentation of the QA/QC procedures;
- Changes to metallurgical recovery assumptions;
- Changes in assumptions of marketability of final product;
- Changes to the conceptual input assumptions for assumed open pit operation;
- Changes to the input values for the AuEq grade used to constrain the estimate;
- Variations in geotechnical, hydrogeological and mining assumptions;
- Changes to environmental, permitting and social license assumptions.

14.19 QP Comments on “Item 14: Mineral Resource Estimates”

The Mineral Resources have been classified using the 2014 CIM Definition Standards.

The QP is not aware of any environmental, legal, title, taxation, socioeconomic, marketing, political or other relevant factors that would materially affect the estimation of Mineral Resources that are not discussed in this Report.

15 MINERAL RESERVE ESTIMATES

This section is not relevant to this Report.

16 MINING METHODS

16.1 Overview

Open pit mining was selected for PEA purposes, based on the size of the resource, grade tenor, grade distribution and proximity to topography. AGP's opinion is that with current metal pricing levels and knowledge of the mineralization and previous mining activities, open pit mining offers the most reasonable approach for development.

The Project is located to the south of Tom Mackay Creek. A 100 m buffer zone was kept with the river for all infrastructure, pits and waste rock storage facilities (WRSFs). Underground mining has previously been conducted in the northern portion of the project, so additional details have been incorporated for mining near old workings. The potential for underground development beneath the open pit was examined in preliminary evaluations but has not been included as part of this PEA. There is still potential for the inclusion of underground mining in future mining studies.

The mine plan is partly based on Inferred Mineral Resources that are considered too speculative geologically to have the economic considerations applied to them that would enable them to be categorized as Mineral Reserves, and there is no certainty that the PEA based on these Mineral Resources will be realized.

16.2 Geological Model Importation

The 2019 resource estimates were created using Leapfrog software for mineralization domains and Vulcan software for block modelling. SRK provided Skeena with support and review of the updated resource model, together with a resource estimate completed in compliance with NI 43-101 and a technical report prepared using the requirements of Form 43-101F1. Skeena provided AGP with resource models in MineSight block model format for open pit and underground mine planning. The original Vulcan resource models were sub-blocked models. The final resource models provided to AGP for mine design were single mineralization percentage models.

Framework details of the different open pit block models are provided in Table 16-1. Resource model item descriptions are shown in Table 16-2 while the final open pit mine planning model items are displayed in Table 16-3. The mining model created by AGP in MineSight includes additional items for mine planning purposes. MineSight was used for the mining portion of the PEA, using their Lerchs Grossmann (L–G) shell generation, pit and WRSF design and mine scheduling tools.

Only Indicated and Inferred Mineral Resources were used for the PEA. No Measured Mineral Resources were reported in the model provided. The block SG values provided in the resource model were estimated based on estimated lead, zinc, copper and antimony grades, with blocks without values receiving a default value of 2.67 t/m^3.

Table 16-1: Open Pit Model Framework

Framework Description	Skeena Resource Open Pit Model (Value)	Final PEA Open Pit Model (Value)
MineSight file 10 (control file)	REG10.dat	REG10.dat
MineSight file 15 (model file)	REG15.AGP	reg15.eng
X origin (m)	9,300	9,300
Y origin (m)	8,508	8,508
Z origin (m) (max)	1450	1450
Rotation (degrees clockwise)	0	0
Number of blocks in X direction	132	132
Number of blocks in Y direction	350	350
Number of blocks in Z direction	375	375
X block size (m)	9	9
Y block size (m)	9	9
Z block size (m)	4	4

Table 16-2: Resource Model Item Descriptions

Field Name	Min	Max	Precision	Units	Comments
SG	2	5	0.0001	t/m^3	specific gravity
DOMIN	1	99	1	—	Mineralized domains: 1 = low grade envelope; 10, 20, 30, 40, 50, 60, 70, 80, 95, 99
ROCPE	0	2	1	—	Rocktype (1 = Rhyolite; 2 = Mudstone)
RESAT	0	3	1	—	Classification categories (2 = Indicated; 3 = Inferred)
AUEAL	0	460	0.001	g/t	Gold equivalent grade
AUFAL	0	290	0.001	g/t	Gold
AGFAL	0	21,000	0.1	g/t	Silver
SBIRE	0	220,000	1	ppm	Antimony (note: assuming antimony as a penalty element)
HGIRE	0	13,000	0.1	ppm	Mercury (note: assuming mercury as a penalty element)
ASIRE	0	240,000	1	ppm	Arsenic (note: assuming arsenic as a penalty element)
PBIRE	0	17	0.0001	%	Lead (note: assuming lead as a penalty element)
CUIRE	0	5	0.0001	%	Copper (note: assuming copper as a penalty element)
ZNIRE	0	23	0.0001	%	Zinc (note: assuming zinc as a penalty element)
SBNLL	0	380,000	1	ppm	Antimony (note: assuming antimony as a credit)
ORE%	0	100	0.001	—	Percent of mineralized material left remaining (with mined percent accounted for in this item)
MINE%	0	100	0.001	—	Percent of block that has been mined out using 1 m buffer; 100% = unmined

Table 16-3: Open Pit Model Item Descriptions

Field Name	Min	Max	Precision	Units	Comments
SG	2	5	0.0001	t/m^3	specific gravity
DOMIN	1	99	1	—	Mineralized domains: 1 = low grade envelope; 10, 20, 30, 40, 50, 60, 70, 80, 95, 99
ROCPE	0	2	1	—	Rocktype (1 = Rhyolite; 2 = Mudstone)
RESAT	0	3	1	—	Classification categories (0 = unclassified, 2 = Indicated; 3 = Inferred)
AUEQ	0	460	0.001	g/t	Gold equivalent grade
AU	0	290	0.001	g/t	Gold
AG	0	21000	0.1	g/t	Silver
SB	0	220000	1	ppm	Antimony (note: assuming antimony as a penalty element)
HG	0	13000	0.1	ppm	Mercury (note: assuming mercury as a penalty element)
AS	0	240000	1	ppm	Arsenic (note: assuming arsenic as a penalty element)
PB	0	17	0.0001	%	Lead (note: assuming lead as a penalty element)
CU	0	5	0.0001	%	Copper (note: assuming copper as a penalty element)
ZN	0	23	0.0001	%	Zinc (note: assuming zinc as a penalty element)
SBNLL	0	380000	1	ppm	Antimony (note: assuming antimony as a credit)
ORE%	0	100	0.001	%	Percent of mineralization left remaining (with mined percent accounted for in this item)
TOPO%	0	100	0.01	%	Topography percent (AGP updated with 2018 LiDAR)
LITH	0	100	1	—	Lithology coded from solids (1 = HWA, 2 = Contact Mudstone, 3 = Rhyolite)
VALB	-100,000	20000000	1	$	Value per block
VALT	-1,000	20000	0.01	$/t	Value per tonne for 1275/oz Au, 15/oz Ag (Aug POX base case)
MINE	0	1	1	—	Entire model coded as 1 for pit optimization
FLAG	0	5	1	—	Dilution flag: 1 = mineralized material, 2 = mineralized waste, 3 = waste
DILBK	0	4	1	—	waste blocks touching and ore block
DILBO	0	4	1	—	Mineralized blocks touching a waste block
DORE%	0	100	0.01	%	Diluted mineralization %
DWAS%	0	100	0.01	%	Diluted waste %
DAU	0	290	0.001	g/t	Diluted gold
DAG	0	21000	0.1	g/t	Diluted silver
BERM	0	99	0.01	m	Berm width for pit design

Field Name	Min	Max	Precision	Units	Comments
VALT1	-1000	20000	0.01	$/t	Value per tonne for 50 g/t Au con, variable penalty, Oct 10 terms
VALB1	-100000	1000000	1	$	Value per block for 50 g/t Au con, variable penalty, Oct 10 terms
NSR2	0	100000	0.01	$/t	NSR per tonne for 50 g/t Au con, variable penalty, Oct 10 terms
VALT2	-1000	20000	0.01	$/t	Value per tonne for 40 g/t Au con,variable penalty, Oct 10 terms
VALB2	-100000	1000000	1	$	Value per block for 40 g/t Au con, variable penalty, Oct 10 terms
VALT3	-1000	20000	0.01	$/t	NSR per tonne for 40 g/t Au con, variable penalty, Oct 10 terms
VALB3	-100000	1000000	1	$/t	Value per tonne for 25 g/t Au con, variable penalty, Oct 10 terms
NSR3	0	100000	0.01	$	Value per block for 25 g/t Au con, variable penalty, Oct 10 terms
VALT4	-1000	20000	0.01	$/t	NSR per tonne for 25 g/t Au con, variable penalty, Oct 10 terms
VALB4	-100000	1000000	1	$/t	Value per block for 50 g/t Au con, fixed penalty, Oct 11 terms
NSR4	0	100000	0.01	$	NSR per tonne for 50 g/t Au con, fixed penalty, Oct 11 terms
VALT5	-1000	20000	0.01	$/t	Value per tonne for 40 g/t Au con, fixed penalty, Oct 11 terms
VALB5	-100000	1000000	1	$/t	Value per block for 40 g/t Au con, fixed penalty, Oct 11 terms
NSR5	0	100000	0.01	$	NSR per tonne for 40 g/t Au con, fixed penalty, Oct 11 terms
VALT6	-1000	20000	0.01	$/t	Value per tonne for 25 g/t Au con, fixed penalty, Oct 11 terms
VALB6	-100000	1000000	1	$/t	Value per block for 25 g/t Au con, fixed penalty, Oct 11 terms
NSR6	0	100000	0.01	$	NSR per tonne for 25 g/t Au con, fixed penalty, Oct 11 terms
VALT7	-1000	20000	0.01	$/t	Value per tonne for 25 g/t Au con, fixed penalty
VALB7	-100000	1000000	1	$/t	Value per block for 25 g/t Au con, fixed penalty
NSR7	0	100000	0.01	$	NSR per tonne for 25 g/t Au con, fixed penalty
CHECK	0	1	1	—	Check flag
VALT8	-1000	20000	0.01	$/t	Value per tonne for 25 g/t Au con, variable penalty (PEA Final base case)
VALB8	-100000	1000000	1	$/t	Value per block for 25 g/t Au con, variable penalty (PEA Final base case)
NSR8	0	100000	0.01	$	NSR per tonne for 25 g/t Au con, variable penalty (PEA Final base case)

16.3 Open Pit Geotechnical Analysis

16.3.1 Site Visit

AGP completed a site inspection, and a compilation, review, and preliminary assessment of available geotechnical data and information for the project. AGP's initial scope also included a mining geotechnical assessment of the underground mining option. While not pursued over the duration of the study, this work contributed significantly to AGP's conceptualization of ground conditions likely to be encountered during open pit mining. Information and data reviewed for the underground option included information such as previous ground control management plans, ground support recommendations, underground inspection reports, mine plans and previous stope designs.

During the AGP site visit the following tasks were completed:

- Met with Skeena geology and exploration staff to discuss and review the current exploration drill plan and status
- Reviewed local and regional geology reports, plans and sections
- Collected and compiled available site geological and geotechnical data
- Completed domain-scale geotechnical logging for select intervals of drill core available at the time of inspection
- Completed vehicle and on-foot traversing of rock slopes, drill access roads, historic portals and plant site, and conducted geotechnical mapping and rock mass characterization of constituent rock masses with data collection tasks focused on verifying and supplementing existing information, including lithology, rock mass strength, and discontinuity characteristics

Figure 16-1 is a photograph taken during the site visit of a 2019 drill pad. Figure 16-2 shows an outcrop of the rhyolite unit. Figure 16-3 provides an example of the hanging wall andesite contact. Figure 16-4 is a core photograph showing the typical Contact Mudstone lithology.

16.3.2 Rock Mass Considerations

Initial estimates of suitable pit slope angles for PEA-level mine planning have been determined. The assessment is based primarily on resource drilling data and core photographs, simple RQD data, economic pit shells, geologic models, and relevant background reports. No material geotechnical drilling, logging, mapping, sampling, or laboratory testing was completed for the current study.

Overall, the data indicates generally 'fair' to 'good' rock mass conditions throughout the mining zone (i.e. the 'general/mean' geotechnical unit, consisting of hanging wall andesites (HWAs) and overlying rhyolites). Poorer-quality rock masses and local bench-scale slope instability are likely to be encountered in zones proximal to Contact Mudstone intercepts, and adjacent to fault zones.

Limited RQD data of uncertain quality typically ranges from zero, in upper hole intervals and fault zones, to 20% to 50%+ in most drill runs. Joint spacing typically varies from 0.2 to 0.6 m, and significantly less in many cases. Intact rock strength varies from R1 to R5, with most rock reporting strengths in the R3 or 25 to 50 MPa range. Typical joint characteristics include slightly rough to (mostly) smooth to slicken-sided surfaces, with soft clayey infill >5 mm thick.

Figure 16-5 and Figure 16-6 show the RQD data locations available from the 2018 drill program. Figure 16-7 is an example logging sheet from the legacy drilling, showing RQD data collected.

Figure 16-1: 2019 Drill Pad, North Pit Slope

Note: Photograph taken by AGP, 2019.

Figure 16-2: Rhyolite Outcrop, North Pit Centre

Note: Photograph taken by AGP, 2019.

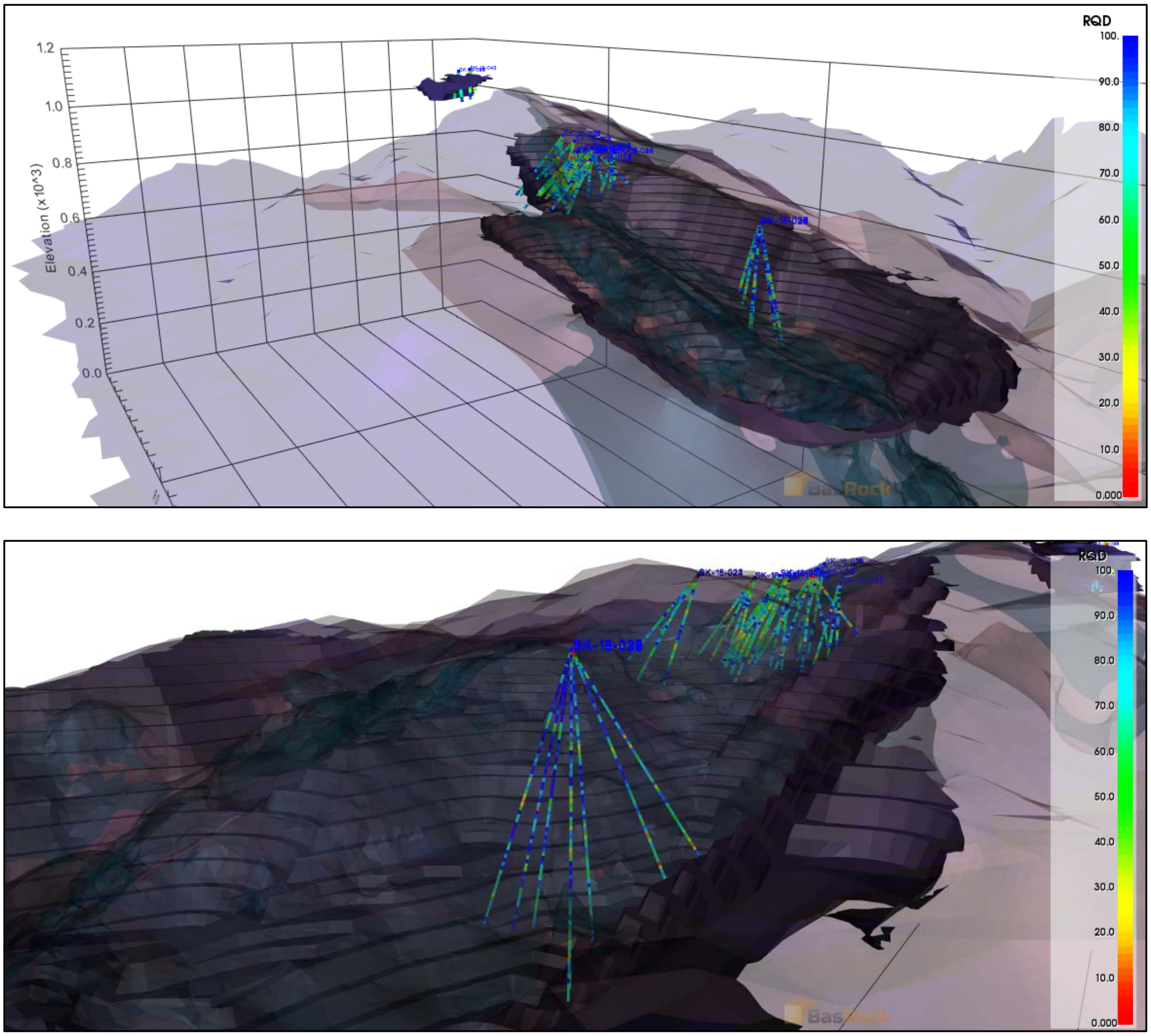
Figure 16-3: Hanging Wall Andesite Outcrop, North Pit

Note: Photograph taken by AGP, 2019.

Figure 16-4: Contact Mudstone Typical Drill Core Intercept

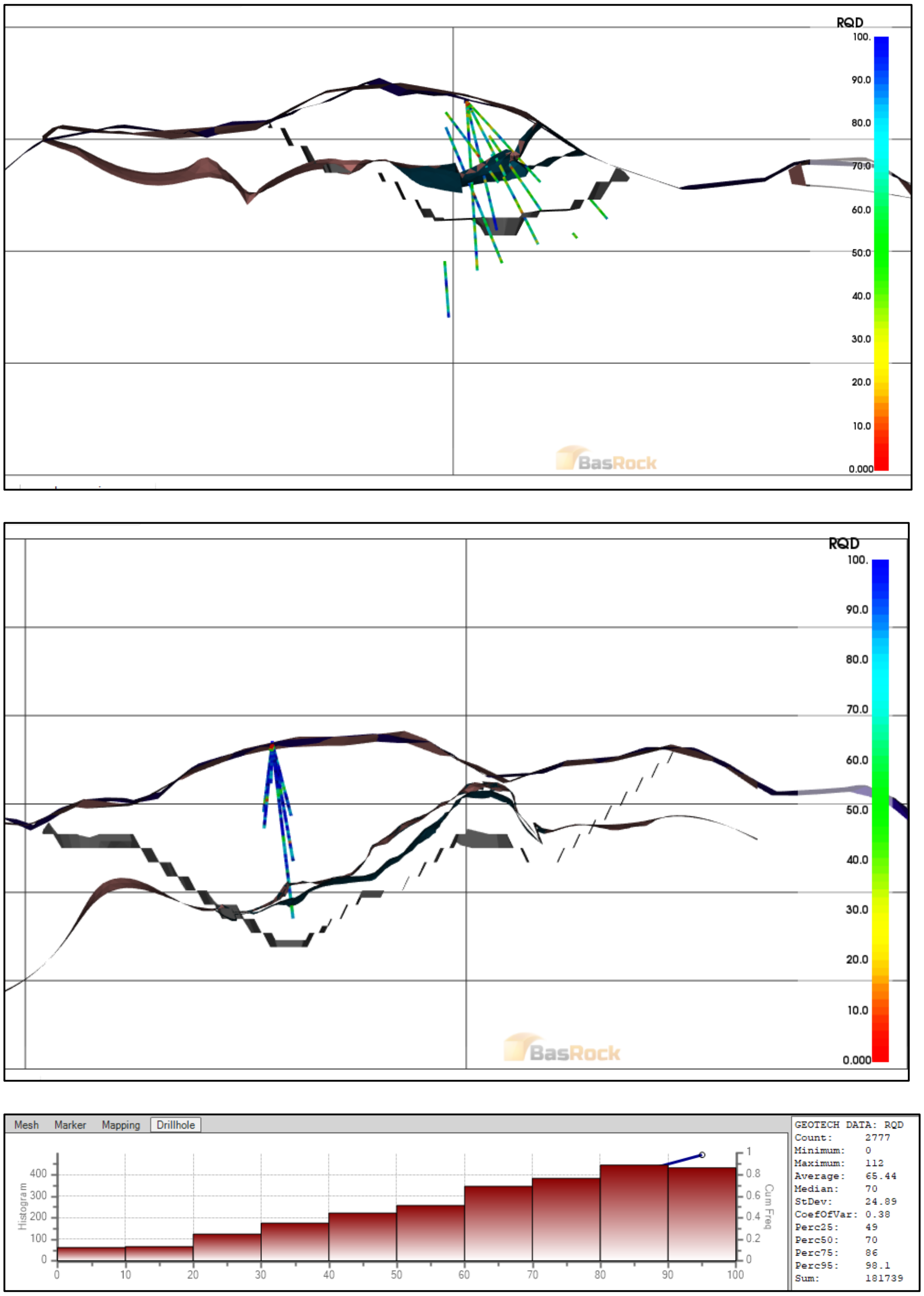
Note: Photograph taken by AGP, 2019.

Figure 16-5: 2018 Drill Program RQD Data



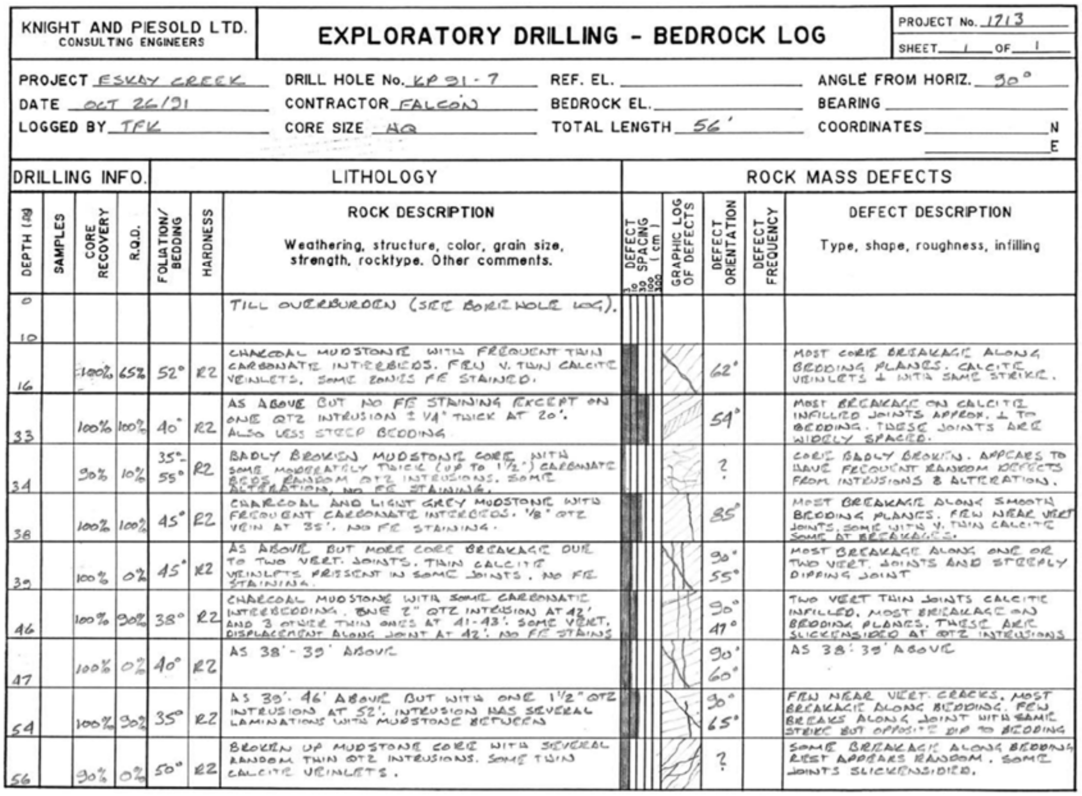
Note: Figure prepared by AGP, 2019.

Figure 16-6: 2018 Drill Program RQD Data



Note: Figure prepared by AGP, 2019.

Figure 16-7: Legacy Exploratory Drill Hole Log with RQD data



KNIGHT AND PIESOLD LTD.
CONSULTING ENGINEERS

EXPLORATORY DRILLING - BEDROCK LOG

PROJECT No. 1713
SHEET 1 OF 1

PROJECT ESKAY CREEK | DRILL HOLE No. KP 91-7 | REF. EL. | ANGLE FROM HORIZ. 90°
DATE OCT 26/91 | CONTRACTOR FALCON | BEDROCK EL. | BEARING
LOGGED BY TFK | CORE SIZE HQ | TOTAL LENGTH 56' | COORDINATES N E

DRILLING INFO.				LITHOLOGY			ROCK MASS DEFECTS				
DEPTH (ft)	SAMPLES	CORE RECOVERY	R.Q.D.	FOLIATION/ BEDDING	HARDNESS	ROCK DESCRIPTION Weathering, structure, color, grain size, strength, rocktype. Other comments.	DEFECT SPACING (cm) 10 30 100 300	GRAPHIC LOG OF DEFECTS	DEFECT ORIENTATION	DEFECT FREQUENCY	DEFECT DESCRIPTION Type, shape, roughness, infilling
0 – 10						TILL OVERBURDEN (SEE BOREHOLE LOG).					
16		100%	65%	52°	R2	CHARCOAL MUDSTONE WITH FREQUENT THIN CARBONATE INTERBEDS. FEW V. THIN CALCITE VEINLETS. SOME ZONES FE STAINED.			62°		MOST CORE BREAKAGE ALONG BEDDING PLANES. CALCITE VEINLETS ⊥ WITH SAME STRIKE.
33		100%	100%	40°	R2	AS ABOVE BUT NO FE STAINING EXCEPT ON ONE QTZ INTRUSION 1 1/4" THICK AT 20'. ALSO LESS STEEP BEDDING			54°		MOST BREAKAGE ON CALCITE INFILLED JOINTS APPROX. ⊥ TO BEDDING. THESE JOINTS ARE WIDELY SPACED.
34		90%	10%	35°–55°	R2	BADLY BROKEN MUDSTONE CORE WITH SOME MODERATELY THICK (UP TO 1½") CARBONATE BEDS, RANDOM QTZ INTRUSIONS. SOME ALTERATION, NO FE STAINING.			?		CORE BADLY BROKEN. APPEARS TO HAVE FREQUENT RANDOM DEFECTS FROM INTRUSIONS & ALTERATION.
38		100%	100%	45°	R2	CHARCOAL AND LIGHT GREY MUDSTONE WITH FREQUENT CARBONATE INTERBEDS. ⅛" QTZ VEIN AT 35'. NO FE STAINING.			85°		MOST BREAKAGE ALONG SMOOTH BEDDING PLANES. FEW NEAR VERT JOINTS, SOME WITH V. THIN CALCITE SOME AT BREAKAGES.
39		100%	0%	45°	R2	AS ABOVE BUT MORE CORE BREAKAGE DUE TO TWO VERT. JOINTS. THIN CALCITE VEINLETS PRESENT IN SOME JOINTS. NO FE STAINING.			90° 55°		MOST BREAKAGE ALONG ONE OR TWO VERT. JOINTS AND STEEPLY DIPPING JOINT
46		100%	90%	38°	R2	CHARCOAL MUDSTONE WITH SOME CARBONATE INTERBEDDING. ONE 2" QTZ INTRUSION AT 42' AND 3 OTHER THIN ONES AT 41-43'. SOME VERT. DISPLACEMENT ALONG JOINT AT 42'. NO FE STAINS			90° 47°		TWO VERT THIN JOINTS CALCITE INFILLED. MOST BREAKAGE ON BEDDING PLANES. THESE ARE SLICKENSIDED AT QTZ INTRUSIONS
47		100%	0%	40°	R2	AS 38'-39' ABOVE			90° 60°		AS 38'-39' ABOVE
54		100%	90%	35°	R2	AS 39'-46' ABOVE BUT WITH ONE 1½" QTZ INTRUSION AT 52'. INTRUSION HAS SEVERAL LAMINATIONS WITH MUDSTONE BETWEEN			90° 65°		FEW NEAR VERT. CRACKS. MOST BREAKAGE ALONG BEDDING. FEW BREAKS ALONG JOINT WITH SAME STRIKE BUT OPPOSITE DIP TO BEDDING
56		90%	0%	50°	R2	BROKEN UP MUDSTONE CORE WITH SEVERAL RANDOM THIN QTZ INTRUSIONS. SOME THIN CALCITE VEINLETS.			?		SOME BREAKAGE ALONG BEDDING REST APPEARS RANDOM. SOME JOINTS SLICKENSIDED.

Note: Photograph by AGP, 2019.

Estimates of rock mass rating (RMR) RMR89 values typically range from 40–55 for the hanging wall andesite and rhyolite geotechnical units. RMR89 values for the mudstones are significantly lower, ranging from 20 to 40. These RMR ranges have been used to estimate rock mass strength and deformation parameters. Related Mohr-Coulomb and Hoek-Brown strength envelopes have been estimated over stresses that are a function of the proposed slope heights. A conservative "disturbance factor" (D) of 0.75–1.0 has been assumed in deriving the various rock mass strength parameters, indicative of significant disturbance to the rock mass due to production blasting and local stress redistributions resulting from mining activities.

No laboratory test data were available for review. Consequently, conservative estimates have been made for ranges of intact rock and discontinuity strengths, based on a review of the qualitative data and relevant experience in similar rock masses. Intact rock strengths for HWAs and rhyolites are estimated in the 25–50 MPa range, and 1–5 MPa (or less) for the contact mudstones. Joint and discontinuity strengths are estimated between 25–35º friction, with faults likely between 15–25º, both with zero to nominal cohesion.

16.3.3 Lithology and Structure Considerations

The pit slopes are expected to consist primarily of HWA along the upper pit walls with the rhyolite being more prevalent at lower pit elevations. The contact mudstone is expected to only affect narrow zones between the HWA and rhyolite. The exposure of the various lithologies are displayed on the northern ultimate pit surface in Figure 16-8. The parameters developed for the north pit were also applied to the South Pit due to limited information available and the small size of the south pit.

Beyond the trends and specific faults noted in Figure 16-9 to Figure 16-11, the orientations, persistence, and conditions of regional and local geologic structures are largely unknown at this time. However, several features interpreted as shears, faults, and fault zones/systems have been mapped in outcrops and intersected by drill holes within the proposed pit extents.

AGP reviewed the data associated with these features to better understand the frequency and ranges in material conditions. The data suggested that local faults typically occur as discrete to coalescing features with apparent moderate-to-high persistence (i.e. greater than 5–10 m continuity), with variable thickness ranging from tens of centimeters to >10 m.

Based on current information, several general observations are made regarding mining geotechnics:

- The mineralized material occurs in rhyolite volcanic rocks and bedded mudstones;
- There are areas of intense faulting within the mineralized zones;
- Rock quality varies from good to extremely poor, and this is exacerbated by water during the spring freshet;
- Rock quality can change significantly over a very short distance;
- There is extensive faulting and contact zones;
- In many parts of the deposit the rock is extremely weak and more soil-like than rock;
- The rock mass rating and the ground conditions have a wide range of values that are specifically related to the lithology and to the different zones of the deposit body;
- Overall the rock mass rating can vary from very poor to good.

16.3.4 Pit Slope Design

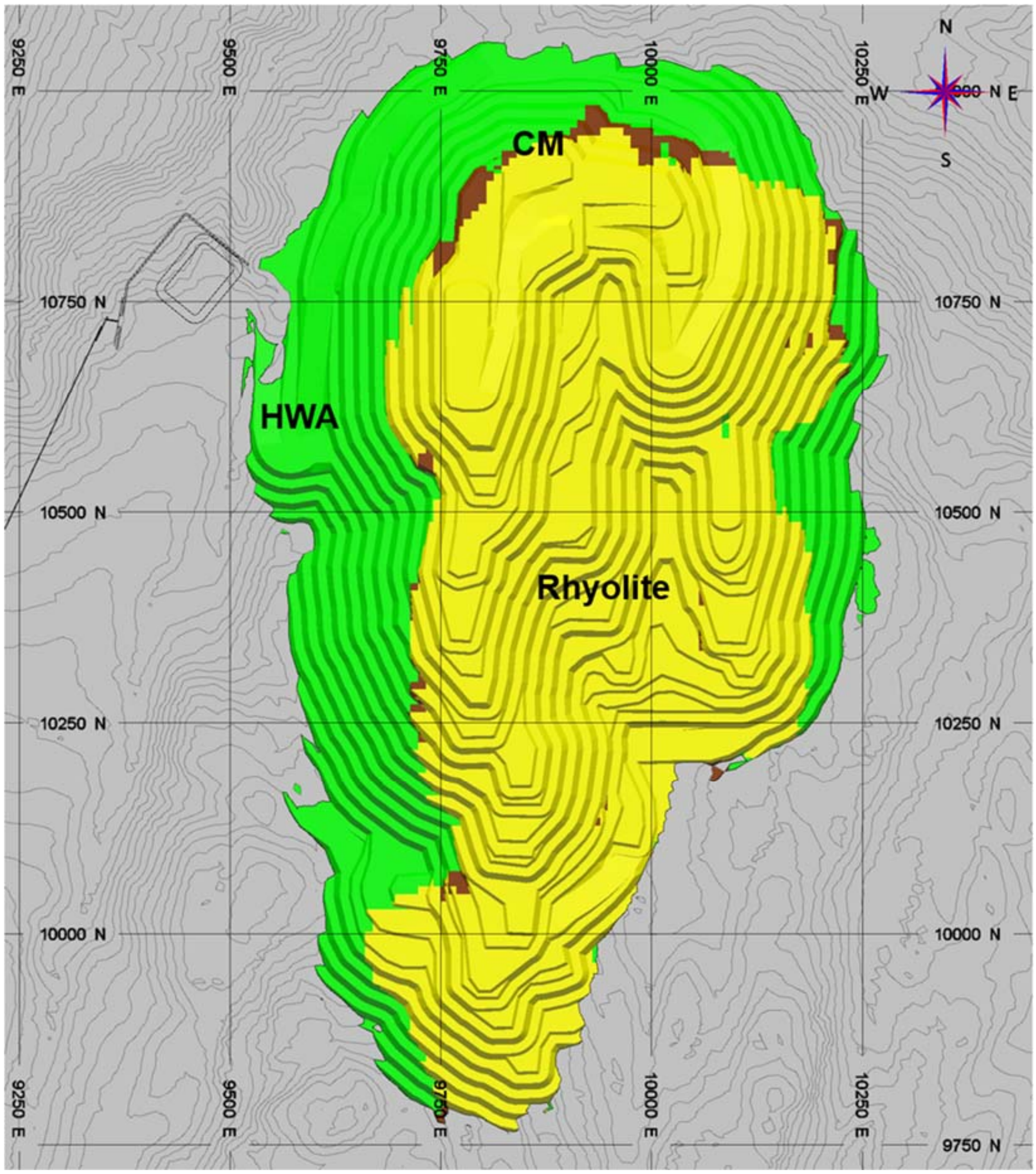
16.3.4.1 Recommended Slope Angles

Efforts to ‘maximize’ slope angles based on limited data can often lead to overly optimistic designs and related project economics. These can be difficult to ‘walk back’ if, or when, contrarian data are recorded during subsequent investigation work. AGP recognizes the potential to improve upon/optimize relatively conservative initial guidance, if/when additional confirmatory data become available.

Based on the limited geotechnical and hydrogeological data, the simplified slope design criteria in Table 16-4 are recommended for current PEA-level studies. These may be updated and refined once reasonable levels of confidence in geotechnical conditions have been established.

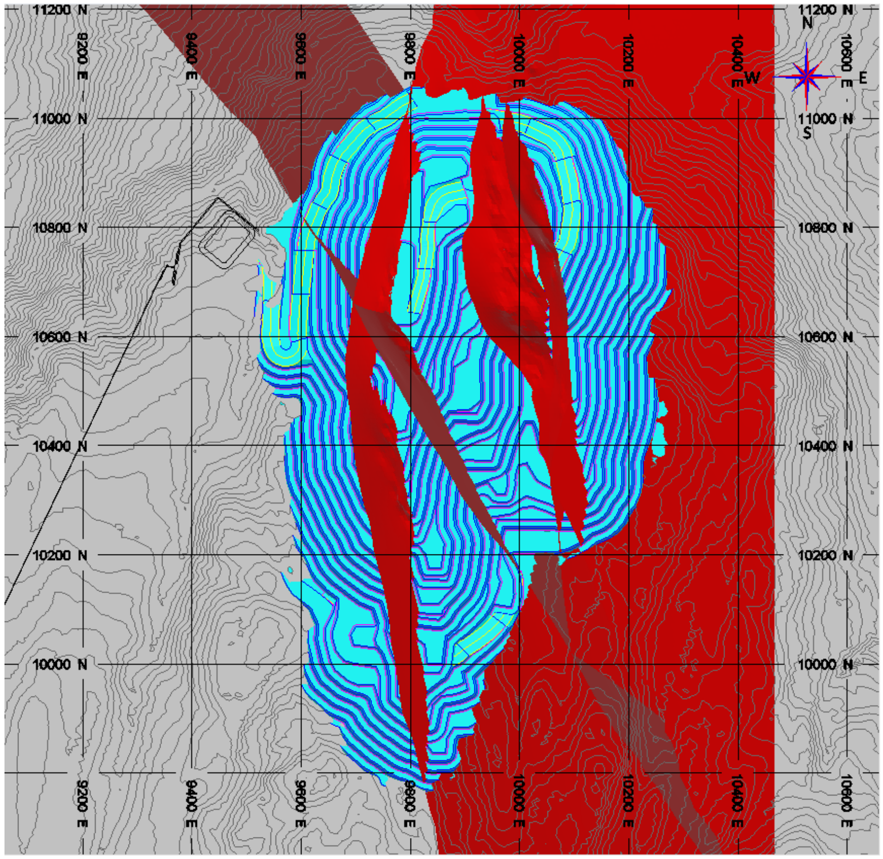
In AGP’s experience the noted criteria are practical estimates of achievable slope configurations; this is by design. If the project is feasible at these inter-ramp slope angles, any improvements or further optimization that can be achieved as a result of additional geotechnical study for given scenarios will be value additive. If, on the other hand, project economics are marginal when these criteria are incorporated, this simply highlights the necessity of determining the project’s geotechnical conditions as soon as practicable.

Figure 16-8: Lithologies Displayed on North Ultimate Pit



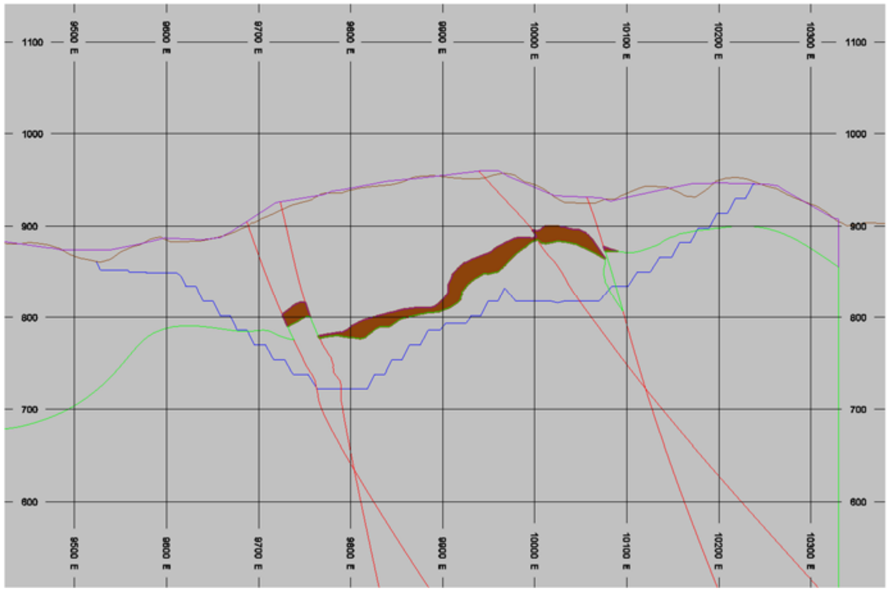
Note: Figure prepared by AGP, 2019.

Figure 16-9: North Ultimate Pit with Interpreted Major Fault Structures



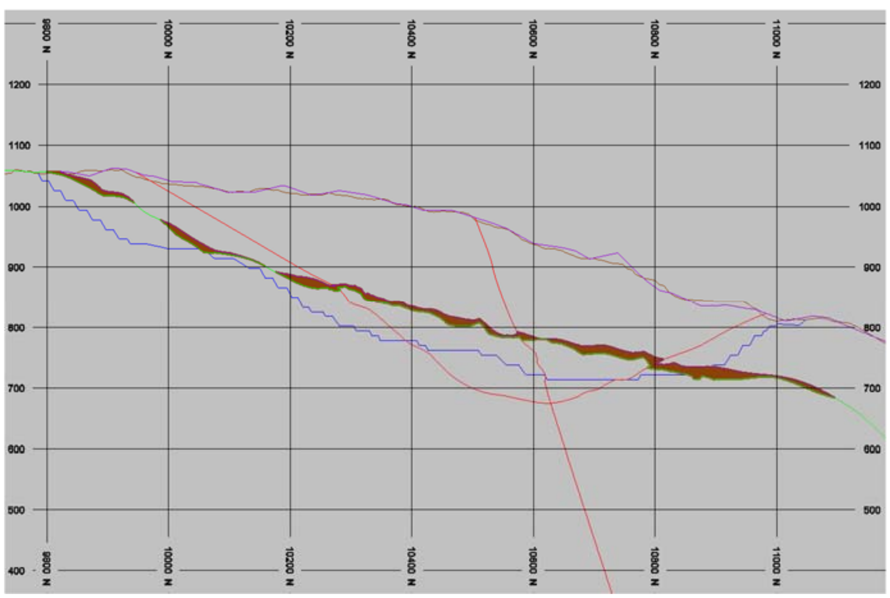
Note: Figure prepared by AGP, 2019.

Figure 16-10: 2019 PEA Ultimate North Pit - Section 10605 N



Note: Figure prepared by AGP, 2019.

Figure 16-11: 2019 PEA Ultimate North Pit - Long Section



Note: Figure prepared by AGP, 2019.

Table 16-4: 2019 PEA Open Pit Slope Design Criteria

Lithology	Inter-Ramp Angle (°)	Bench Face Angle (°)	Height Between Berms (m)	Catch Bench Width (m)
Default (Rhyolite, HWA)	42	65	16	10.31
Mudstone	32	65	16	18.14

Note: 8 m benches during mining.

Due to the current limited quantity of geotechnical data and relatively low data density in relation to a pit of the proposed size, geotechnical units for the project have been defined by AGP based on observed ranges in rock mass classification values, rather than (for example), lithological or structural domains that are more typical at advanced levels of study. This will need to be re-assessed for more detailed studies, once additional geotechnical data are available.

To allow steeper slope angles in areas with better quality rock and to minimize stripping to the greatest extent possible, AGP divided the pit into individual slope design sectors, based on slope height and dominant geology. Estimates of suitable overall slope angles were then developed for each of the individual sectors. As indicated, the inter-ramp slope recommendations ranged between 32º and 42º.

16.3.4.2 Hydrogeology

Hydrogeological conditions are not well known for the site; however, it is understood that the historic underground mine has been split into two regimes with respect to ‘flood’ water:

- Upper Mine Water discharge (UMW): water being made above the main portal;
- Lower Mine Water discharge (LMW): water being made below the main portal.

The UMW is a gravity system that constantly flows via pipelines from a bulkhead in the main portal to the surface pond system located on the historic processing/accommodation pad. This surface pond system ultimately discharges to Ketchum Creek. The LMW is allowed to fluctuate in accordance with natural recharge coming from surrounding groundwater sources. Occasionally the level in the lower mine approaches an elevation (768 masl) where it will discharge to Tom Mackay Creek through the D-raise. To prevent this, water is pumped to the upper mine where it is directed to the surface pond system used to manage discharge from the upper mine. The historic decline located within the upper central portion of the proposed pit is flooded below the portal elevation (approximate elevation 1,280 m), suggesting the pit slopes will be at least partially saturated and subject to inflows.

Hydrogeological conditions will need to be investigated and more broadly understood during more detailed studies.

16.3.4.3 Geotechnical Model Limitations

The preceding section summarizes information and knowledge gathered to date, primarily by others, along with information collected by AGP during a recent four-day site visit. This information provides the basis for preliminary pit slope design and guidelines to assist with mine design, planning, and cost estimating for the project.

The current geotechnical dataset is considered adequate for conceptual level designs. Where data gaps exist, the engineering geology of the area has been inferred from available data. When quantifying material properties of the rock, ranges of values have been estimated.

Engineering geology interpretations presented in this Report should be considered preliminary. Data collected to date may not accurately reflect the rock mass comprising the final open pit walls. Where appropriate, geological features identified should be verified and validated with additional field work and interpretation.

16.3.4.4 Preliminary Open Pit Slope Stability Assessment

Empirical slope stability charts demonstrate typical safety factors for a variety of slope configurations and rock types. AGP's preliminary guidance is illustrated in Figure 16-12 and Figure 16-13; the recommended slope angles are consistent with the data and stability guidelines presented.

Preliminary 2D limit equilibrium analyses were completed for the project by AGP using the 2019 Mineral Resource pit shell and the initial pit design guidance described above, to gather initial insight into inter-ramp and overall slope geotechnics (Figure 16-14). While currently generic in nature, these models have been used to assess and interpret a wide variety of geotechnical and slope stability issues that may arise as a result of future investigations, including changes to the geological and structural models, variable non-linear and anisotropic rock mass strength criteria, and ground water conditions, as well as the effects of excavation rates and sequencing.

AGP commonly uses the following approach for target factor of safety (FOS) values at the PEA level of study:

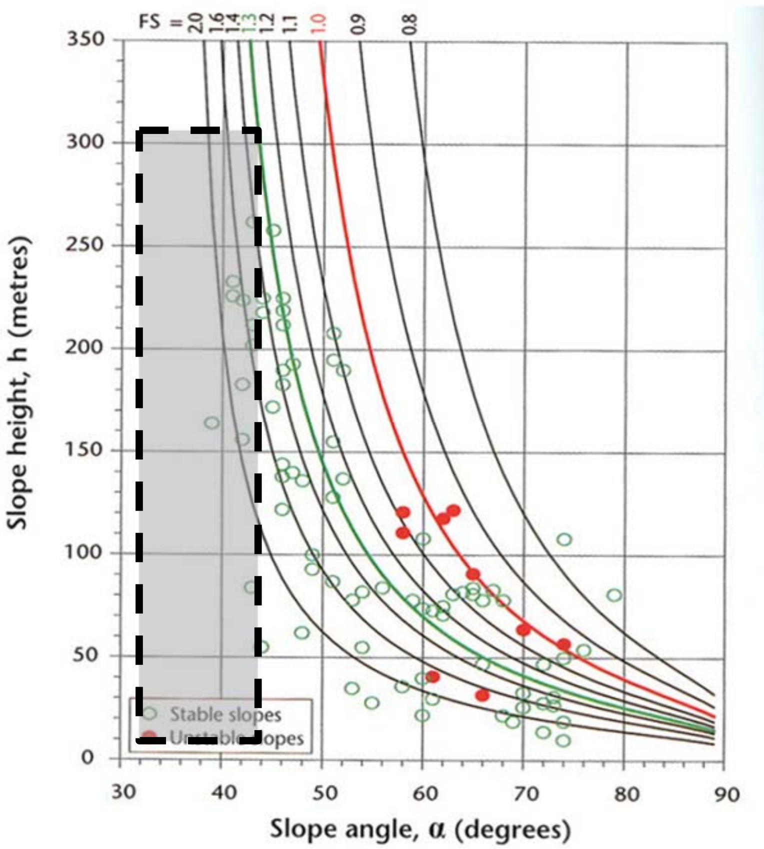
- Multi-bench or inter-ramp slopes controlled by discontinuities should achieve a minimum FOS of 1.2;
- Inter-ramp or overall slopes involving shearing through the rock mass and with a low or medium consequence of instability should meet a minimum FOS of 1.3;
- Overall slopes with a high consequence of instability should meet a minimum FOS of 1.5.

Slope heights ranging from 100–300 m with inter-ramp and global slope angles varying from 30–45° were analyzed under fully- to partially-saturated conditions.

The preliminary analyses indicate the as-designed slopes are predicted to exhibit generally 'stable' conditions for a variety of scenarios, with typical 'minimum' FOSs ranging from ~1.1–>2.0 for inter-ramp and global slopes. Bench scale slope instabilities have not been assessed for the current study, due to insufficient discontinuity and orientation data. Bench configurations have included an allowance for reasonable catchment widths to help manage operational challenges that may arise from local bench-scale stabilities.

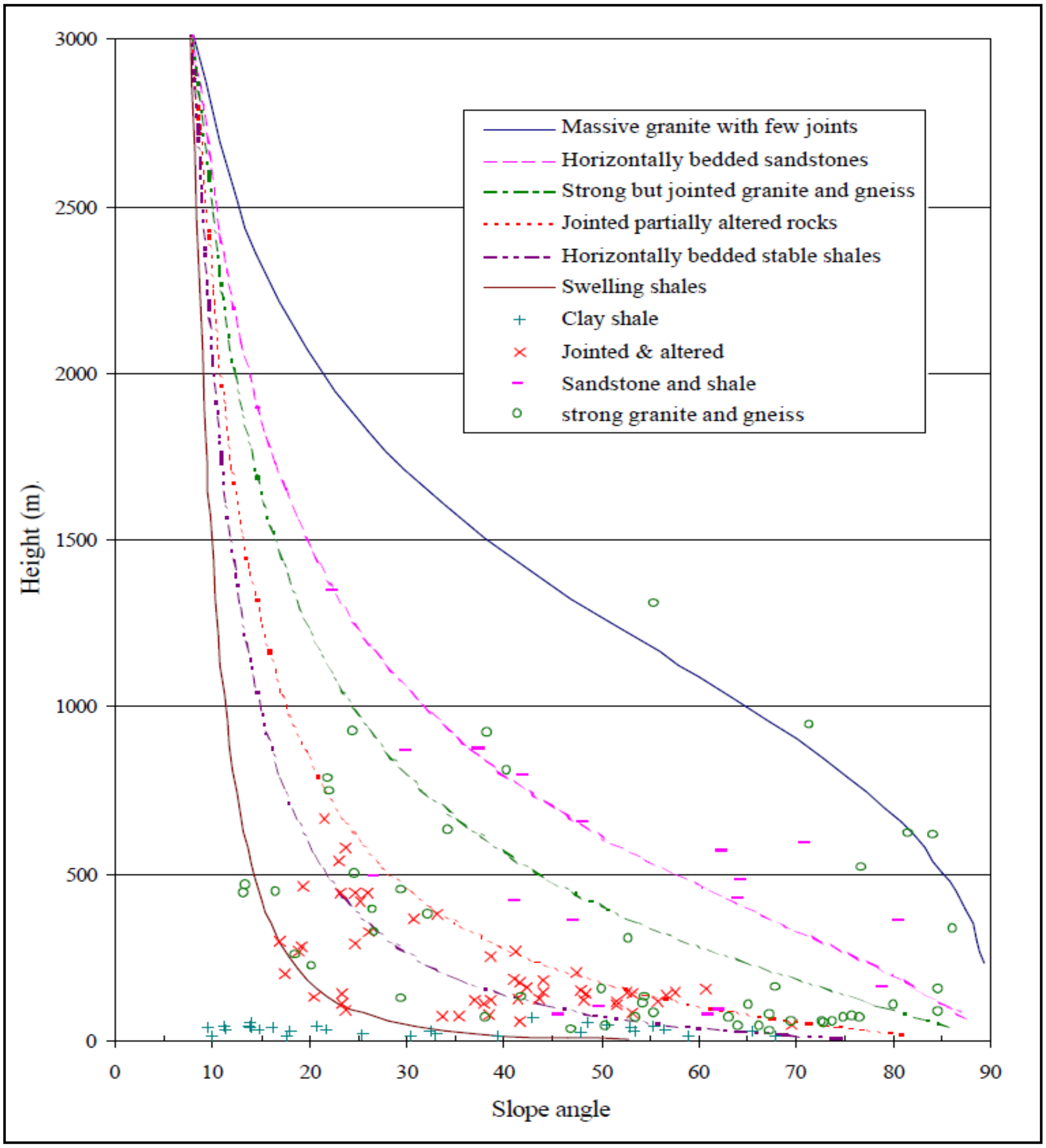
It is probable that unfavorably oriented geological structures are present locally within various slope pit sectors, particularly given the size and extents of the pit and the observed variability in discontinuity orientations. It is assumed at present that small bench-scale failures developed along these features can be managed with careful blasting techniques and regular berm maintenance/clearing, wherever access is possible.

Figure 16-12: Slope Stability – Height vs Slope Angle vs Factor of Safety



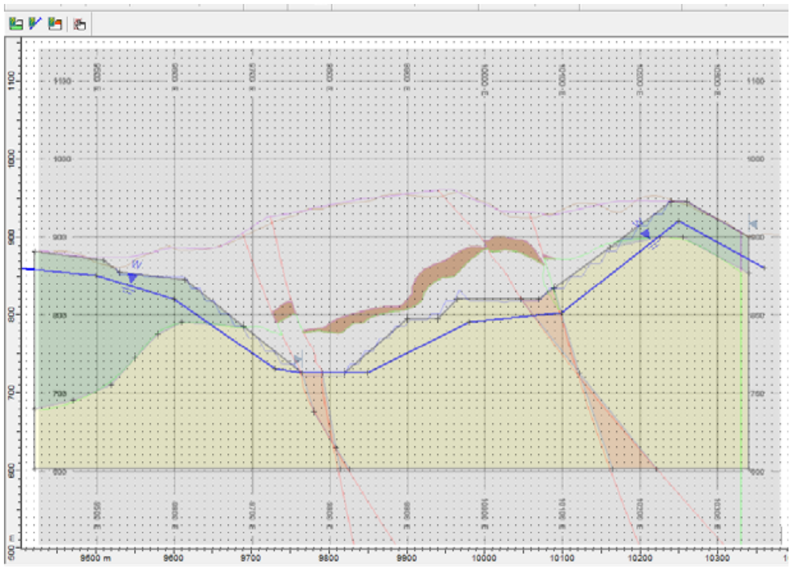
Note: Figure prepared by AGP, 2019.

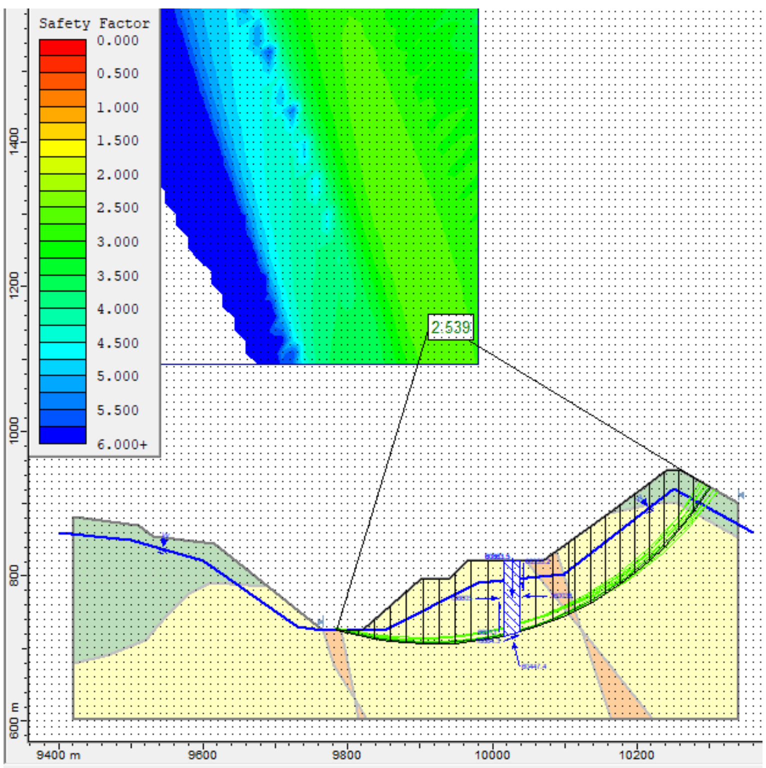
Figure 16-13: Slope Stability – Height vs Slope Angle for Various Rock Types



Note: Figure prepared by AGP, 2019.

Figure 16-14: Typical Input and Results from Preliminary 2D Limit Equilibrium Modelling





Note: Figure prepared by AGP, 2019.

Both seismic loading and multi-bench-scale to pit-scale structures have the potential to significantly affect overall pit slope stability. The current status and impact of these are both largely unknown. Preliminary seismic analysis completed for the project indicates a peak ground acceleration of 0.085 *g*, with a probability of exceedance of 10% in 50 years.

The inclusion of hypothetical adversely oriented faults and bedding planes in the stability analyses indicates potential FOSs <1.0, particularly with seismic loading applied. Further geotechnical investigations are warranted to determine the location and character of inter-ramp to global-scale features that may impact stability and mining outcomes.

Sufficient data has been compiled regarding geotechnical strengths and characteristics of the primary rock types to provide a range of potential pit wall design guidelines. However, numerous assumptions had to be made about the primary controls on rock mass stability, geology, rock mass strength, groundwater pressures, and potential failure mechanisms. As such, the stability models should be considered conceptual in nature. Updated models should be generated and analyzed when updates to the mine plan and/or geotechnical domain model become available.

16.3.4.5 Existing Underground Workings

As illustrated in Figure 16-15, the proposed open pit will intersect and mine into the historical underground workings at approximately mid-slope height on the mid to north side of the pit. This will result in increased risks for safely mining in this area and proscriptive plans will need to be developed to adequately mitigate these risks to acceptable levels.

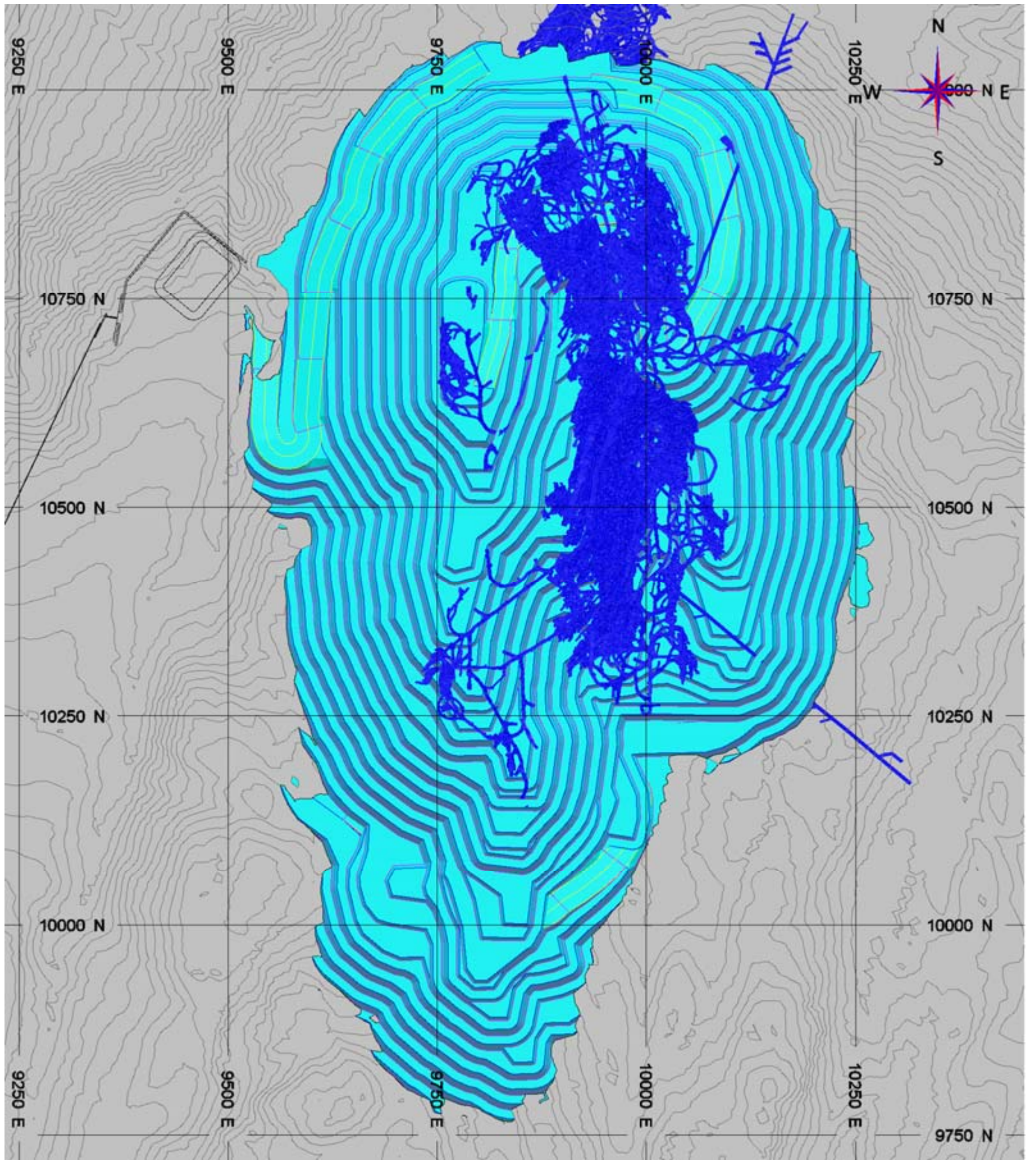
Current best practice for advancing open pit mining operations through existing underground voids is to fill them with either waste or low-grade mineralized material, which removes the void and partially supports the wall rock around the void. If a source of waste rock is available and that will be visibly distinguishable from the mill feed after blasting, dilution can be kept to a minimum, while not tying up feed that could be processed sooner. Failing this, using lower-grade feed material to fill the voids is a practical approach.

Although working around known voids will present safety and productivity challenges, a larger concern is the potential for unknown voids. Even with historical mining records and as-built level maps, one must assume that unidentified voids exist. Mining should therefore advance from lower-risk areas toward higher-risk areas, with probe drilling and perhaps geophysical detection methods.

16.3.4.6 Data Gap Analysis

A geotechnical data gap analysis was completed by AGP to determine data requirements to support more detailed mine designs for the proposed open pit (Table 16-5).

Figure 16-15: North Ultimate Pit and Existing Underground Workings



Note: Figure prepared by AGP, 2019.

Table 16-5: 2019 PEA Mining Geotechnical Data Gap Analysis

Gap Analysis Criteria	Status	Gaps
Spatial coverage	Poor	Very limited geotechnical data has been collected to date within the pit extents; drill holes do not intersect large portions of the proposed pit walls Detailed geotechnical data, and structural data, required for all slope sectors
Geological coverage	Fair	Moderate geological data and limited geotechnical data (mainly from past underground mining) currently exists for the project area Limited spatial and geotechnical knowledge exists regarding location and intensity of fault impacted zones, Mmudstone intercepts and local characteristics
Coverage of Major Features	Poor	Preliminary outcrop mapping has been completed; more is required to confirm trends, assess for current unknowns. Initial fault characterization work initiated based on core data, more work required Limited orientation and persistence data available
Orientation data bias	No data	Orientation data and analysis required (evaluated using coring data and/or ATV/OTV surveys)
Orientation data quality	No data	Orientation data and analysis required (evaluated using coring data and/or ATV/OTV surveys)
Field and Laboratory Strength Testing	Limited	Limited rock strength estimates, mainly from past underground mining UCS, tri-axial, tensile, direct shear and other standard laboratory tests are required to determine/confirm rock strength and deformation parameters, discontinuity strength criteria.

The available data were evaluated relative to the following considerations:

- Spatial coverage: ensuring sufficient coverage of rock mass quality and discontinuity orientations of the rock masses in the walls of each major sector of the open pit mine;
- Geological coverage: ensuring sufficient characterization of the different geological units (lithologies) expected to be in and around the open pits;
- Coverage of major features: ensuring known faults and other features have been intersected and characterized;
- Orientation data bias: ensuring the discontinuity orientation data is sufficiently free of directional bias;
- Orientation data quality: ensuring the discontinuity orientation data is of suitable quality;
- Laboratory strength testing: ensuring sufficient laboratory strength testing has been completed to characterize the intact rock properties of the different geological units expected at each deposit.

The results of the geotechnical gap analysis indicate several important factors that require additional investigation. For more detailed designs, a higher level of confidence is required, and the preliminary geotechnical model presented will need to be updated with additional higher-quality data.

16.4 Hydrogeological Considerations

16.4.1 Eskay Creek Mine and Local Geology

The old Eskay Creek Mine workings are located beneath an anticline ridge which is cut by Tom Mackay Creek to the north; the mine workings pass underneath the Tom Mackay Creek (approximate crown pillar) and continue at depth to the northeast. The elevation of the anticline ridge hosting the mine workings varies from about 800–1,000 masl, rising to the south, along strike to about 1,300 masl. Rivers and creeks flow to the north and on both sides of the ridge that encompass the old workings; the Tom Mackay Creek flows into Ketchum Creek which drains to the Unuk River, south of the former mine site.

The rocks around the Eskay Creek Mine comprise folded and faulted volcanic and sedimentary sequences of the Hazelton Group. The stratigraphic sequence (refer to Figure 7-3) is subdivided into the Lower Footwall Units; Rhyolite; Contact Mudstone and Hanging Wall Andesites and Mudstones. Most of the ore zones are contained in the west flank of an anticline structure at the contact between the Rhyolite and overlying Contact Mudstone.

16.4.2 Physical Hydrogeology

The local hydrogeology is determined by bedrock storage and transmissivity; there is no overburden aquifer on the anticline ridge overlying the old mine workings. The transmissivity is thought to be strongly influenced by geological structures that can locally increase hydraulic conductivity or behave as aquitards that compartmentalize groundwater in the bedrock aquifer. The bedrock around the mine is reportedly fractured and permeable as seen from local seepage zones.

The regional groundwater regime is most likely controlled by the regional groundwater flow system, and from seasonal snow melt. The regional faults likely provide high permeability recharge pathways and groundwater storage areas; however, the rock units themselves are highly fractured and even away from major faults constitute fractured aquifers. Faulted andesite most likely provides the highest permeability and highest storage capacity of all the rock units.

The mean annual precipitation (MAP) between 2010 and 2014 was 2,078 mm with from 50% to 70% occurring as snow characteristic of a freshet-driven hydrograph. The joints in the rock mass allow rapid infiltration of precipitation as seen by increased mine water level in the fall and spring. Previous studies have estimated approximately recharge of 50% of MAP or approximately 1 m/a.

Major structures (faults) in the mine area sub-parallel the north–south-trending features that include the Andesite Creek Fault Zone; the Argillite Creek Fault; the Pumphouse Creek Fault; the Pumphouse Creek Splay (or Pathfinder Fault) and the Portal Faults (Figure 16-16). The Andesite Creek Fault Zone is a water-bearing structure, and was intercepted by Ramp 5 at 640 masl, whereas the Pumphouse Creek and Pathfinder Faults are gouge–filled and are not water-bearing. Reidel shears act as aquitards and prevent water flow across them indicated by different groundwater levels on either side of the shears.

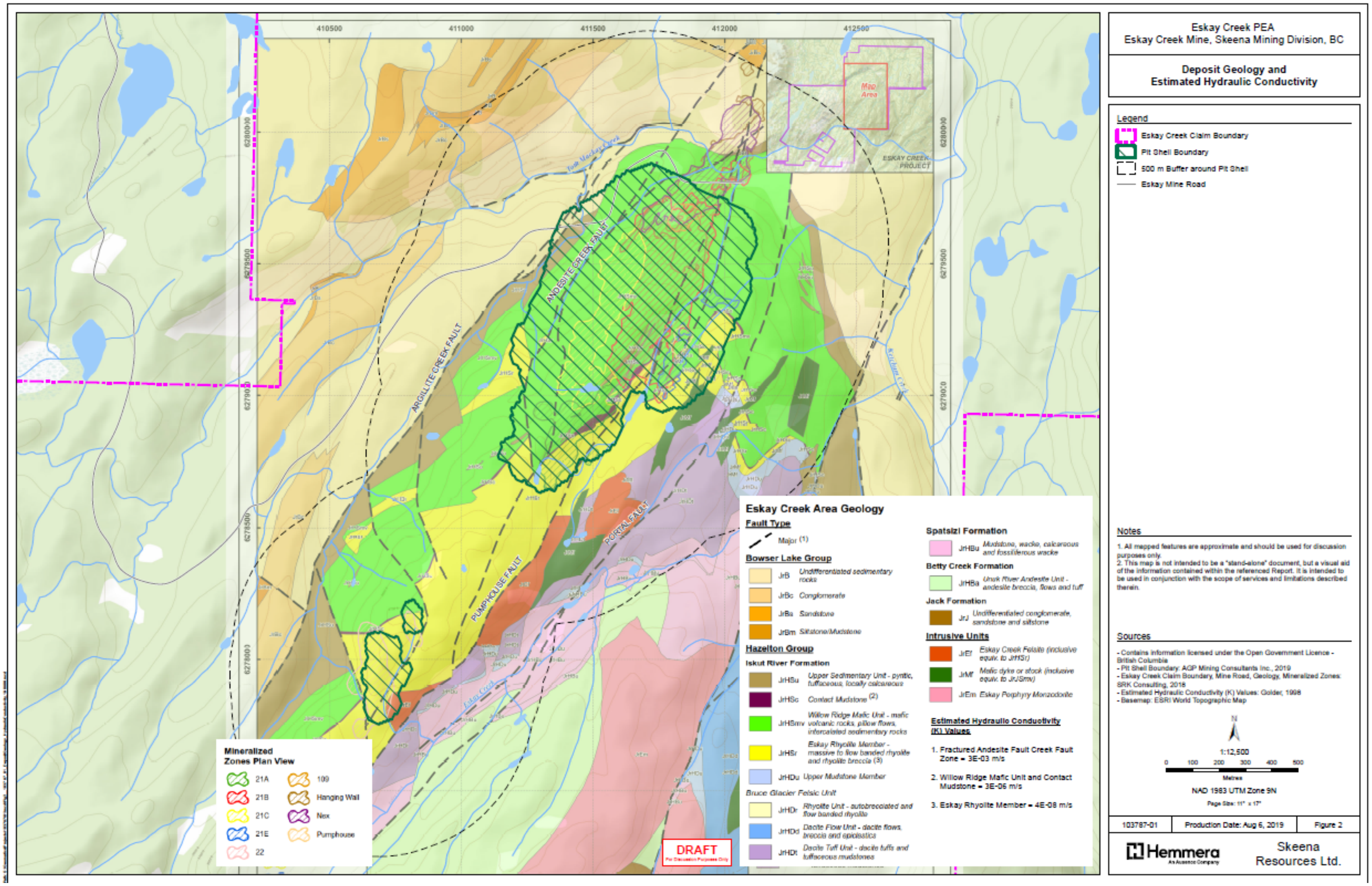
Three high-permeability zones with large areal extent were identified by Golder (1998) in the mine area:

- The Andesite Creek Zone which mostly crosscuts andesite units (the Willow Ridge Mafic Unit in Figure 16-16);

Ausenco

SKEENA

Figure 16-16: Deposit Geology and Estimated Hydraulic Conductivity



- Competent andesite and mudstone units;
- Fractured mudstones associated with Reidel shears.

Six hydrostratigraphic units were identified at the mine (Golder, 1998):

- Andesite Creek Fault Zone: K 3E-03 m/s (high permeability aquifer). Faulted andesite is most often characterized by rubble zones that are like sharp angular gravel when recovered in drill core. There are often lost zones associated with andesite. The non-faulted andesite is good quality core, with few fractures, but the faulted portions appear as rubble. The faulted andesites are high permeability pathways and storage zones for groundwater;
- Competent andesite and mudstone: K 3E-06 m/s (aquifer). The andesite, unlike the rhyolite is relatively unweathered and contains little clay alteration in its matrix. There is some surface weathering present, but for the most part, the andesites exposed underground and in exploration drill core are quite fresh;
- Fractured mudstones associated with Reidel shears: K 3E-05 m/s (aquifer). The contact mudstones seem to appear intermediate in permeability to the rhyolites and andesites. Observations underground show that the mudstones often tend to exhibit flow parallel to bedding surfaces; therefore, is more likely to have an anisotropic flow regime (more permeable in the plane of bedding than across it). Faulted mudstone often appears as mixed gouge and brittle zones;
- Mineralized zone comprising vuggy sulphides: K 2E-06 m/s (aquifer);
- Competent rhyolite: K 4E-08 m/s (aquitard). Most of the rhyolite in the general area of the mine has a high percentage of clay alteration product in its matrix. Faulted portions of the rock are characterized by gouge zones rather than brittle fault zones; however, there are areas where the rhyolite is much more competent and faulted portions may comprise brittle zones with higher permeability.
- Reidel shears: K 1E-08 m/s (aquitard). Groundwater levels were different on either side of the Reidel shears, indicating that the shears act as aquitards (Golder, 1998).

16.4.3 Groundwater Quality

Groundwater quality information was sourced from the following:

- Groundwater quality baseline: Hatfield (1993);
- Current groundwater quality: annual reclamation reports from 2014–2016;
- Tom Mackay Creek water quality investigation: Barrick (2017).

The Eskay Creek Mine baseline study, initiated in 1990, was based on monthly groundwater samples collected from two producing drill holes in the 21A and 21B zones, as well as additional shallow piezometers (8.5–30.5 m). Groundwater was generally circum-neutral but acidic (pH 3.7) in drill holes in the 21A and 21B mineralized zones (Table 16-6). The total dissolved solid (TDS) and sulphate concentrations were typical of a mineralized zone; whereas the trace metals antimony and arsenic concentrations were generally low (in the parts to tens of parts per billion) whereas iron, aluminium and other metals were elevated.

Table 16-6: Pre-Mining Groundwater Quality (1990–1991)

Samples	pH	TDS (mg/L)	Sulphate (mg/L)	Metals (µg/L)
Drill Holes - 21A&B Zones	pH 3.7 – 7.9	160 – 270	30 – 88	Sb 2 – 2.9; As 2 – 34. Other metals low except for Fe and Zn
Piezometers KP90-13-15 and -16	pH 7.4 – 8.8	40 – 237	<18	Sb <2; As <3. Elevated concentrations noted for Al, Fe and occasionally for Cd; Cr; Cu; Pb; Mn; Ni; Se; Ag and Zn
Piezometers KP91-5 to KP91-9	pH 6.8 – 12	128 – 2030	n/a	Sb <25 and As <40. Elevated concentrations noted for Al, Fe and occasionally for Cd; Cr; Cu; Pb; Mn; Ni; Se; Ag and Zn
Emma Adit discharge: low volume	pH 6.2 – 7.5	n/a	n/a	Elevated Fe (up to 23,400) and Sb; As; Cd; Pb; Mn and Zn.

During Eskay Creek Mine operations, treated mine water from underground mining operations, treated mill effluent, and surface run-off from the mine site, were collected in settling ponds located near the lower portal and the discharge released to Ketchum Creek (D7 pond out monitoring station). Since production ceased in 2008, mine water discharge has occasionally been treated with lime for low pH and elevated dissolved zinc; in the event of elevated pH (underground mine water pH in the D-raise is generally above pH 10 due to cement backfill but decreases to circum-neutral during freshet), a sulfuric drip may be used.

Turning off the dewatering pumps after completion of a mine flooding study (AMEC, 2016), resulted in rebound of groundwater that flushed accumulated oxidation products from the mineralized bedrock to Tom Mackay Creek, as seen by elevated zinc and antimony concentrations in the creek at the beginning of the study that declined in time. Sulphate concentrations in the D-raise are generally <250 mg/L, increasing up to 350 mg/L during freshet. The pH drops during freshet from around pH 10 to pH 7.5, but zinc concentrations can spike from <0.01 mg/L to around 1 mg/L.

Although variations in water quality in Tom Mackay Creek do not appear to be directly related to the water quality in the Eskay Creek Mine workings, 'mine pool' water likely seeps to the creek through faults and fractures and as seepage from old exploration drill holes. Iron precipitates observed in the creek bed are thought to originate from unsealed boreholes connected to underground workings.

During mining operations, high levels of suspended solids were reportedly problematic for treatment plant and ponds. Water treatment will be required for the project to treat water from the old workings as well as surplus pit dewatering and other mine contact water. Using this water in the milling process and reclaiming tailings supernatant water could significantly reduce the treatment volumes, providing there are no metallurgical constraint to using these waters.

16.4.4 Gap Analysis

The following were noted during a gap analysis review:

- Aquifer hydraulic properties (obtained from packer, pumping and slug testing). Although a reasonable level of information is available from the Golder (1998) study, additional hydraulic data should be collected to refine hydrogeologic characterization of the proposed mining and waste rock storage areas. These data can be collected during further geological or geotechnical drilling at modest expense and will facilitate three-dimensional numerical groundwater modelling to support water balance, sizing of the water treatment plant, site-wide water quality modelling and environmental impact assessment/permitting studies;

- The Andesite Creek Fault is considered a major conduit of water in the mine area. The hydraulic conductivity and storage potential of this fault should be investigated so that efficient dewatering can be achieved on the west wall of the pit. Similarly, the Pumphouse Fault on the east side of the pit should be characterized from pumping tests of sufficient duration to stress the local aquifer.

16.4.5 Potential Groundwater Risks Based On Current Mine Plan

The andesite and mudstone lithologies will likely dewater easily compared to the rhyolite, which reportedly has high fines content and drains poorly (significantly lower hydraulic conductivity than the andesite). The rhyolite will generally occupy lower elevations in the final pit extent; however, rhyolite would be present on the south and east pit highwall and may be susceptible to failure if pore-water pressure builds up on fault planes. Horizontal boreholes drilled from pit benches may be a more efficient and effective means of depressurizing this material than vertical dewatering wells.

16.4.6 Potential Effects On Groundwater Quantity And Quality

16.4.6.1 Groundwater Quantity

The project will divert groundwater that would otherwise flow to Tom MacKay Creek and relocate it to the TMSF in the form of tailings slurry, or discharge treated groundwater to Ketchum Creek (downgradient of Tom Mackay Creek). The net effect would be reduction of flow in Tom Mackay Creek from lock-up in tailings.

During operations of the former Eskay Creek Mine, mine dewatering rates from 2000–2004 varied seasonally from a June peak of 82–107 L/s to an April low of 13–38 L/s). Average flow measured at the settlement pond discharge weir (D7) was about 45 L/s. Similar peak and average dewatering rates for the proposed pit can be expected. A pit dewatering model is recommended for the next study phase.

Since mine closure, water flowing through the mine was estimated at 450,000 m^3/year (14 L/s) assuming recharge of 1 m/a over an area of 3 km x 500 m. Decant through the upper portal is approximately 350,000 m^3/a (11 L/s), which is treated before release. The balance seeps to local streams and creeks. The lower workings ‘mine-pool’ water is likely of poorer quality than the recharge decanting through the upper portal.

The total flooded volume of mine workings (below approximately 768 masl) was estimated at 312,000 m^3 (BGC, 2014) and would require dewatering should underground mining be carried out. However, the planned ultimate pit bottom will be at 714 masl and therefore only about 50 m of flooded working is likely to require dewatering.

Pit stability can be managed by progressive dewatering of the ground behind the pit slope with vertical or horizontal boreholes. The hanging wall (andesite and mudstone) rocks are rated as highly conductive (K = 3E-06 m/s) compared to the footwall (rhyolite) rock (K = 4E-08 m/s). The mudstones may require special attention as matrix pore pressures could remain elevated despite successful dewatering.

16.4.6.2 Groundwater Quality

Baseline studies showed elevated metal concentrations in groundwater related to the Eskay Creek mineralized zones. Groundwater in the area around, and stored in, the underground workings, has been impacted by mining. The future mine will potentially impact groundwater in proximity of mining infrastructure. Groundwater affected by acid rock drainage can have low pH, elevated metals and total dissolved solids (sulphate and hardness). Contaminated groundwater could impact aquatic habitat in the Tom Mackay and Ketchum Creeks downstream of the mine.

A water quality investigation of Tom Mackay Creek conducted in 2017 in response to a regulator inspection, attributed iron staining in the Tom McKay Creek to open drill holes that are discharging groundwater from the flooded mine and surrounding area. Plugging the drill holes to prevent potential surface groundwater interaction is recommended.

16.4.7 Conclusions and Potential Mitigation Measures

Flow reductions in Tom Mackay Creek will occur because of pit and underground mine dewatering. These effects are difficult to mitigate and will occur throughout the life or mine, and for several years after closure. Their significance may be limited given the large size of the upstream catchment relative to the potentially impacted area. The system will return to equilibrium once the groundwater rebounds and a pit lake has formed.

Groundwater quality will be impacted beneath and downgradient of the waste rock dump and open pit. These impacts will likely only occur after mine closure. Mitigation measures could include:

- Use waste rock placement methods that limit air ingress, and progressive WRSF reclamation where feasible;
- Divert WRSF run-off and seepage to treatment/settling ponds or institute passive wetland treatment. The topography is generally beneficial for seepage capture and diversion;
- Maximize final pit lake extent to cover as large an area of the pit wall as possible.

16.5 Pit Shell Development

The open pit ultimate size and phasing requirements were determined with various input parameters including estimates of the expected mining, processing and general and administrative (G&A) costs, as well as metallurgical recoveries, pit slopes and reasonable long-term metal price assumptions. AGP worked together with Skeena personnel to select appropriate operating cost parameters for the proposed Eskay Creek open pit. The mining costs are estimates based on cost estimates for equipment from vendors and previous studies completed by AGP. The costs represent what is expected as a blended cost over the life of the mine for all material types to the various dump locations. Process costs and a portion of the G&A costs were provided by Skeena based on results of their other ongoing studies and test results.

The parameters used are shown in Table 16-7. The net value calculations are in United States dollars (US$) unless otherwise noted. Costs and revenues are converted to Canadian dollars for use in pit shell determination. The mining cost estimates are based on the use of 142 t trucks using an approximate WRSF configuration to determine incremental hauls for mineralized material and waste. The smelting terms and recovery assumptions are based on creating a 25 g/t gold bulk concentrate.

Table 16-7: Pit Shell Parameter Assumptions

Description	Units	Value	Gold Value	Silver Value
Exchange rates				
CAD	US$ =	1.2975		
Resource Model				
Block classification used		M+I+I		
Block Model height	M	4		
Mining Bench height	m	8		
Metal Prices				
Gold Price	$/oz		1275.00	
Silver Price	$/oz			15.00
Royalty	%		2%	2%
Smelting, Refining, Transportation Terms				
Payable	%		95.0	80.0
Minimum deduction	unit, g/dmt		0	0
Participation (on profits)	%		100	100
Bulk concentrate treatment charge	$/dmt	180.00		
Refining	$/oz		15.00	1.00
Concentrate moisture	%	12		
Transit losses	%	0.5		
Concentrate transportation cost	C$/wmt	118.00		
As Penalty (free up to 0.5%)	$/1%	7.50		
Hg Penalty (free up to 500 ppm)	$/100 ppm	4.00 for 500–700 ppm		
		7.00 for >700 ppm		
Metallurgical Information				
Recovery	%		92% for >2.5 g/t	97% for >100 g/t
	%		90% for <2.5 g/t	90% for <100 g/t
	%		85% for <2.0 g/t	
	%		80% for <1.5 g/t	
	%		75% for <1.0 g/t	
Power Cost				
Cost of power	C$/Kwhr	$0.04		
		Fuel Cost		
Diesel fuel cost to site	C$/ l	$1.26		
Mining Cost *				
Waste base rate - 880 elevation	C$/t	$2.87		
Incremental rate - above	C$/t/4m bench	-$0.0073		
Incremental rate - below	C$/t/4m bench	$0.0161		
Mill feed base rate - 880 elevation	C$/t	$2.58		
Incremental rate - above	C$/t/4m bench	$0.0081		
Incremental rate - below	C$/t/4m bench	$0.0131		

Description	Units	Value	Gold Value	Silver Value
Processing **				
Processing cost	C$/t mill feed	$24.64		
Conveyor cost	C$/t mill feed	$0.50		
Tailings cost	C$/t mill feed	$0.41		
Water treatment	C$/t mill feed	$1.33		
Total processing cost	C$/t mill feed	**$26.88**		
General and Administrative Cost				
G&A cost	C$/t mill feed	$7.16		
Total Process and G&A				
Process + G&A	C$/t mill feed	**$34.04**		

Note: * mining costs based on using 142 t haul trucks. ** process costs based on 2.5 Mt/a dry throughput

Wall slopes for pit optimization were based on review of available historical underground data and observations as outlined in Section 16.3. Allowances were made for ramps in the slopes to determine an overall angle for use in the L–G routine. The overall slope angle calculations are shown in Table 16-8. Slopes were flattened as required due to inclusion of haulage ramps.

Nested L–G pit shells were generated to examine sensitivity to the gold and silver prices with a target of US$1,275/oz Au and US$15.00/oz Ag. This was to gain an understanding of the deposit and highlight potential opportunities in the design process to follow. Undiluted Indicated and Inferred material was used in the analysis. The net smelter return (NSR) was varied by applying revenue factors of 0.10 to 1.20 at 0.05 increments, to generate a set of nested L–G shells. The chosen set of revenue factors result in an equivalent gold price varying from US$128/oz up to US$1,530/oz. All other parameters were fixed. The resulting nested pit shells assist in visualizing natural breakpoints in the deposit and selecting shells to act as design guidance for phase design. The net profit before capital for each pit was calculated on an undiscounted basis for each pit shell using US$1,275/oz Au and US$15.00/oz Ag. A 100 m river offset was used to restrict the pit shells from the Tom Mackay Creek and Ketchum Creek. Mill feed material/waste tonnages and potential net profit were plotted against gold price and are displayed in Figure 16-17.

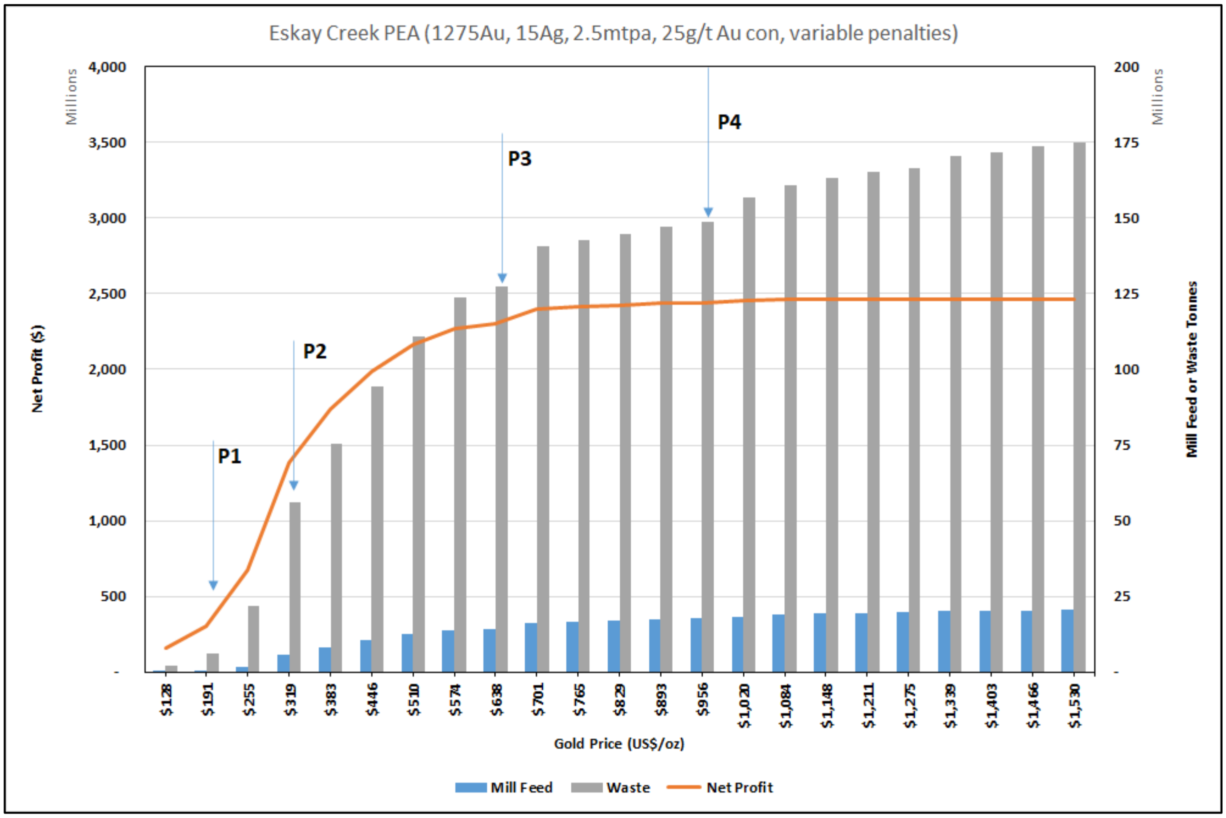
Figure 16-17 illustrates various break points in the pit shells. With each incremental the increase in the waste tonnage, and to a lesser degree the mill tonnage, the undiscounted net profit also increased. In the case of the first break point shown at US$191/oz Au, the cumulative waste tonnage is 6.3 Mt, with a corresponding mill feed tonnage of 539 kt or a strip ratio of 11.7:1. The net profit also increased beyond this point. showing that there was still value to be obtained by going with a higher metal price or an additional phase. This break point represented 12% of the net value of a $1,275/oz pit but with only 4% of the waste of the larger pit shell.

This pit shell was used to guide the splitting of the next significant pit shell into two phases which would include a starter phase at the south end of the north pit.

Table 16-8: Pit Shell Slopes

Lithology	LITH Code	Overall Slope (°)	Description
HWA	1	33.9	One 30.2 m-wide ramp added in HWA, assuming a wall height of 80 m
CM	2	32	
Rhyolite	3	42	

Figure 16-17: Eskay Creek Potential Profit vs. Price by Pit Shell



Note: Figure prepared by AGP, 2019.

The second break point was at US$319/oz Au. The incremental waste tonnage from the first break point is 49.9 Mt, with a corresponding increase in mill feed tonnage of 5.0 Mt or a strip ratio of 9.9:1. The net profit also increased beyond this point showing that there was still value to be obtained by going with a higher metal price. This pit shell was used for the pit design of Phases 1 and 2. There is a significant waste tonnage in the next higher pit price to achieve the next increase in profit. The cumulative value of the first two break points was 56% of the US$1,275/oz Au pit shell but with only 34% of the waste movement of the larger pit required. This pit shell ran the entire length of the deposit and therefore effectively split the remaining pits into east and west pushbacks.

The third and fourth major break points were at US$638/oz Au and US$956/oz Au respectively. This resulted in a substantial jump in the waste tonnage from the second break point to the third break point by 71.1 Mt with a gain of 8.6 Mt of feed material for an incremental strip ratio of 8.3:1. The net profit continues to increase beyond the third break point, although at a flatter rate than in earlier breakpoints. The cumulative value of the first three break points was 93% of the US$1,275/oz Au pit shell but with only 77% of the waste movement of the larger pit required.

This resulted in a substantial jump in the waste tonnage from the third break point to the fourth break point by 21.4 Mt with a gain of 3.6 Mt of feed for an incremental strip ratio of 5.9:1. The net profit continues to increase beyond the fourth break point, although at a flatter rate than in earlier breakpoints. The cumulative value of the first four break points was 99% of the US$1,275/oz Au pit shell but with only 89% of the waste movement of the larger pit required. The additional potential pit value was considered insufficient to cover schedule discounting.

The US$638/oz Au and US$956/oz Au pit shells were very similar on the east side of the pit, so the east side pit was designed to the ultimate wall to facilitate haul road access. The Phase 4 pit design then mined the west wall to ultimate using the US$956/oz Au pit shell as a guide. Particular attention would need to be taken to ensure haul road access was available to each of the phases at various elevations.

16.6 Dilution

The open pit resource model was provided as an undiluted percentage type model. This means the grades from the wireframes were reported into separate percentage parcels of mill feed and waste in each block. The provided feed percentage values exclude underground workings and all material within 1 m of their original solids. These underground solids were viewed on several plan views with ORE% values and the workings appear to have been properly adjusted in a consistent manner. As the mine workings were mostly backfilled, they were included in the waste percentage.

To account for mining dilution, AGP modeled contact dilution into the in-situ resource blocks. To determine the amount of dilution, and the grade of the dilution, the size of the block in the model was examined. The block size within the model was 9 x 9 m in plan view, and 4 m high. Mining would be completed on 8 m lifts for waste and 4 m lifts for mill feed, if required, and the equipment selected is capable of mining in that manner.

The percentage of dilution is calculated for each contact side using an assumed 1.25 m contact dilution distance. This dilution skin thickness was selected by considering the spatial nature of the mineralization, proposed grade control methods, GPS-assisted digging accuracy, and blast heave.

If one side of a mineralized block above cut-off is in contact with a waste block, then it is estimated that dilution of 13.9% (1.25 m/9 m) would result. If two sides are contacting, it would rise to 27.8%. Three sides would be 41.7%, and four sides 55.6%. Four sides represent an isolated block of mill feed.

All mineralized blocks in the resource model contain grade values; however, the material outside the mineralized shapes have no grade estimates and have been treated as though the gold and silver grades are zero for dilution purposes. The net value per tonne that was stored to the block model during the L–G runs was used as the grade for cut-off application. As that net value per tonne is inclusive of all on-site operation costs except for mining, applying a $0.01/t cut-off represents the marginal cut-off grade to flag initial feed and waste blocks. Using this marginal cut-off grade, all model blocks were flagged as either:

- Feed blocks;
- Waste blocks within mineralized material;
- Default waste blocks outside of mineralization.

MineSight has a routine called gndiln.dat that enables the user to query surrounding blocks against a set of conditions. AGP applied a two-pass approach to all model block dilution calculations in order to define new items called DORE%, DWAS%, DAU, and DAG. Note that the other non-revenue items were not included in dilution grade calculations. The default waste blocks would receive DORE%=0 and diluted gold and silver grades of 0 g/t. The two-pass dilution calculations are summarized as follows:

- For the first pass of dilution calculations, the procedure was run to determine how many waste blocks contacted a feed block. A dilution percentage with no grade was then added to the original ore percentage to determine a new diluted mineralization percentage up to a maximum of either 100% or the TOPO item. The assumption was that the feed block was at the edge of the mineralization when 0% < ORE% < 100%, so it could be diluted with barren waste material;
- For the second pass of dilution calculations, the procedure was run to determine how many feed blocks contacted a waste block from within the mineralization area. A dilution percentage at the waste block grade was then added as the new diluted mineralization percentage. The assumption was that only the feed contact sides of the mineralized waste blocks would be added as dilution.

In this manner, the contact diluted blocks were included in the tonnage and grade calculation of mill feed tonnes. The mill feed tonnage report was then run with the block model DORE% item to report out the diluted tonnes and grade.

Comparing the in-situ to the diluted values for the designed final pits, the diluted feed contained 20.8% more tonnes and 16.6% lower gold grade than the in-situ feed summary. The grade dilution percentage was lower than the feed tonnage percentage since the mineralized waste blocks included some grade. AGP considers these dilution percentages to be reasonable considering the expected seasonal working conditions as well as mining through underground workings.

16.7 Pit Designs

Pit designs were developed for the north and south pit areas. The north pit consists of four main phases, while the south pit only contained a single small phase. The pit optimization shells used to determine the ultimate pits were also used to outline areas of higher value for targeted early mining and phase development.

Geotechnical parameters outlined in Table 16-8 were applied to pit designs. It is likely that the mudstone will be reduced to single benching as more geotechnical information is obtained in future. This is not considered significant for the PEA pit designs as the inter-ramp angle would likely use similar values.

Equipment sizing for ramps and working benches is based on the use of 142 t rigid-frame haul trucks. The operating width used for the truck is 6.9 m. This means that single lane access is 23.3 m (twice the operating width plus berm and ditch) and double lane widths are 30.2 m (three times the operating width plus berm and ditch). Ramp gradients are 10% in the pit and WRSF for uphill gradients. Working benches were designed for 35–40 m minimum mining width on pushbacks, although some pushbacks in the north pit did work in a retreat manner to facilitate access resulting in a ramp width pushback at times. As the haul road grades exceed 5%, runaway lanes or retardation barriers will need to be incorporated into designs as the project progresses to more detailed studies.

The north and south pits are displayed in Figure 16-18. The south pit is significantly smaller than the north pit and is likely to be mined near the end of the mine schedule.

Tonnes and grade for the final pit designs are reported in Table 16-9 using the diluted tonnes and grade from the model and a mining recovery of 98% to account for additional mineralized material losses. Positive marginal block values from the pit optimization run were used to determine mill feed material blocks.

The north pit will contain four main phases, with phase 3 split into three smaller sub-phases to show access at various elevations. The phase designs are described in further detail in the following sub-sections.

16.7.1 North Phase 1

Phase 1 will be the first phase mined. This phase begins mining at the upper elevations of the north pit and targets a shallow high-grade zone for mill feed. Phase bench elevations will range from 1066 masl down to 922 masl. All waste and mineralized material accesses will be on the northwest side of the phase, where the WRSFs and mill feed crusher will be located. The north phase 1 design is shown in Figure 16-19.

16.7.2 North Phase 2

Phase 2 will also be accessed from the west side of the pit. As the phase advances down benches to the north of phase 1, haul road accesses will be left in place along the west side so that they may be use by later phases. An access point will also be left at 802 m elevation near the historic underground access road. Phase bench elevations will range from 1002 masl down to 754 masl. The north phase 2 design is shown in Figure 16-20.

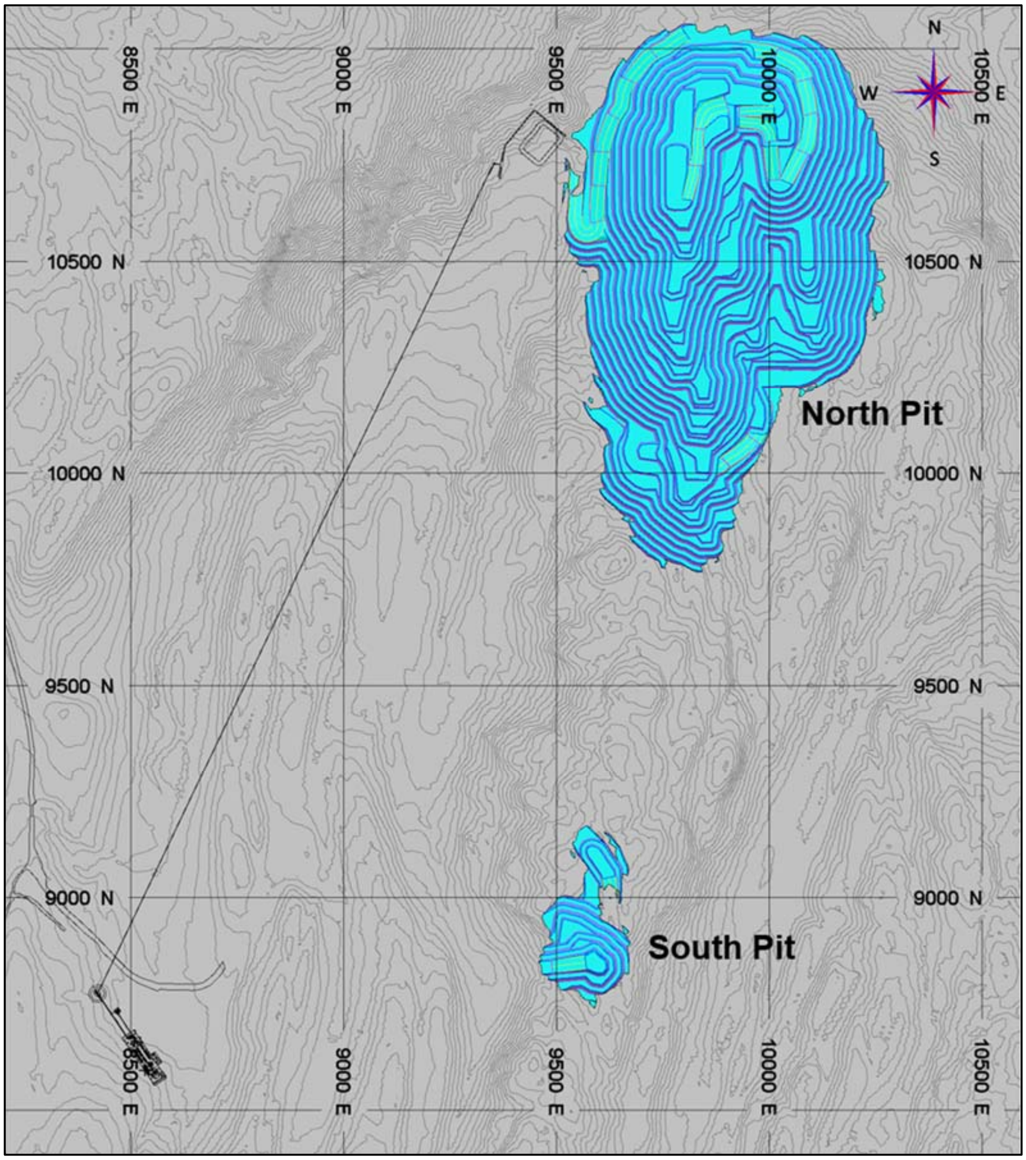
16.7.3 North Phase 3a

Phase 3a will make use of haul roads left in place by phase 2 to retreat mine its benches down to the 882 m elevation. The pit exit at 858 m elevation pit is where the mined material will leave the pit. Phase bench elevations will range from 994 masl down to 882 masl. The phase 3a design is shown in Figure 16-21.

16.7.4 North Phase 3b

Phase 3b will be developed to make a haul road access that ties into the phase 2 ramp at 826 m elevation. The pit exit at 858 m elevation pit will be where the mined material leaves the pit. Phase bench elevations will range from 874 masl down to 826 masl. The phase 3b design is shown in Figure 16-22.

Figure 16-18: Proposed Eskay Creek North and South Pits

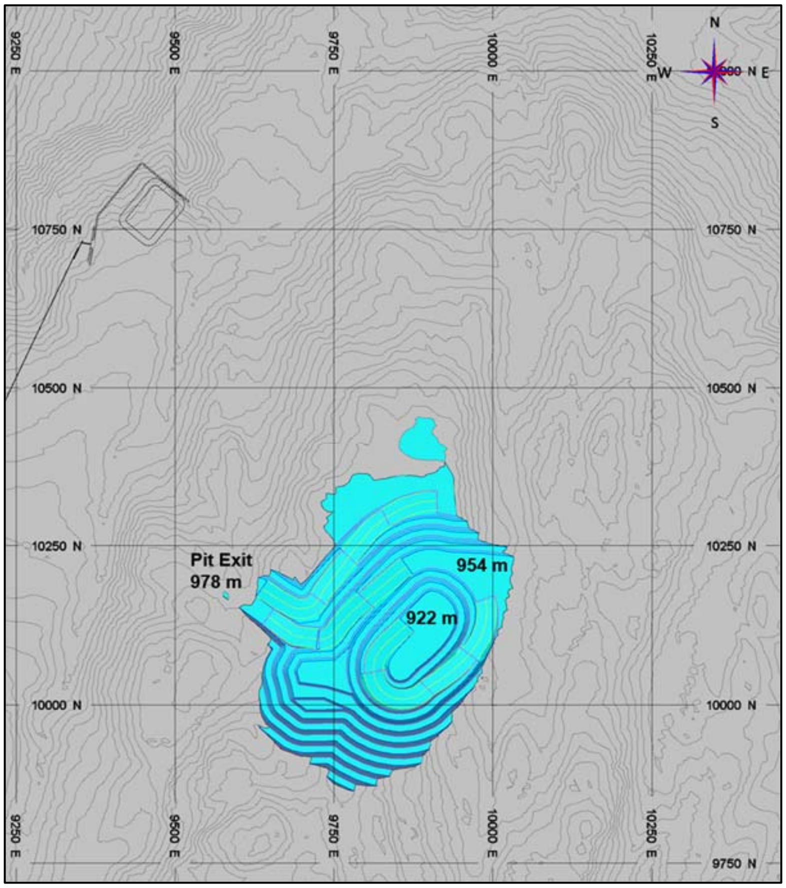


Note: Figure prepared by AGP, 2019.

Table 16-9: Final Design – Phases, Tonnages, and Grades

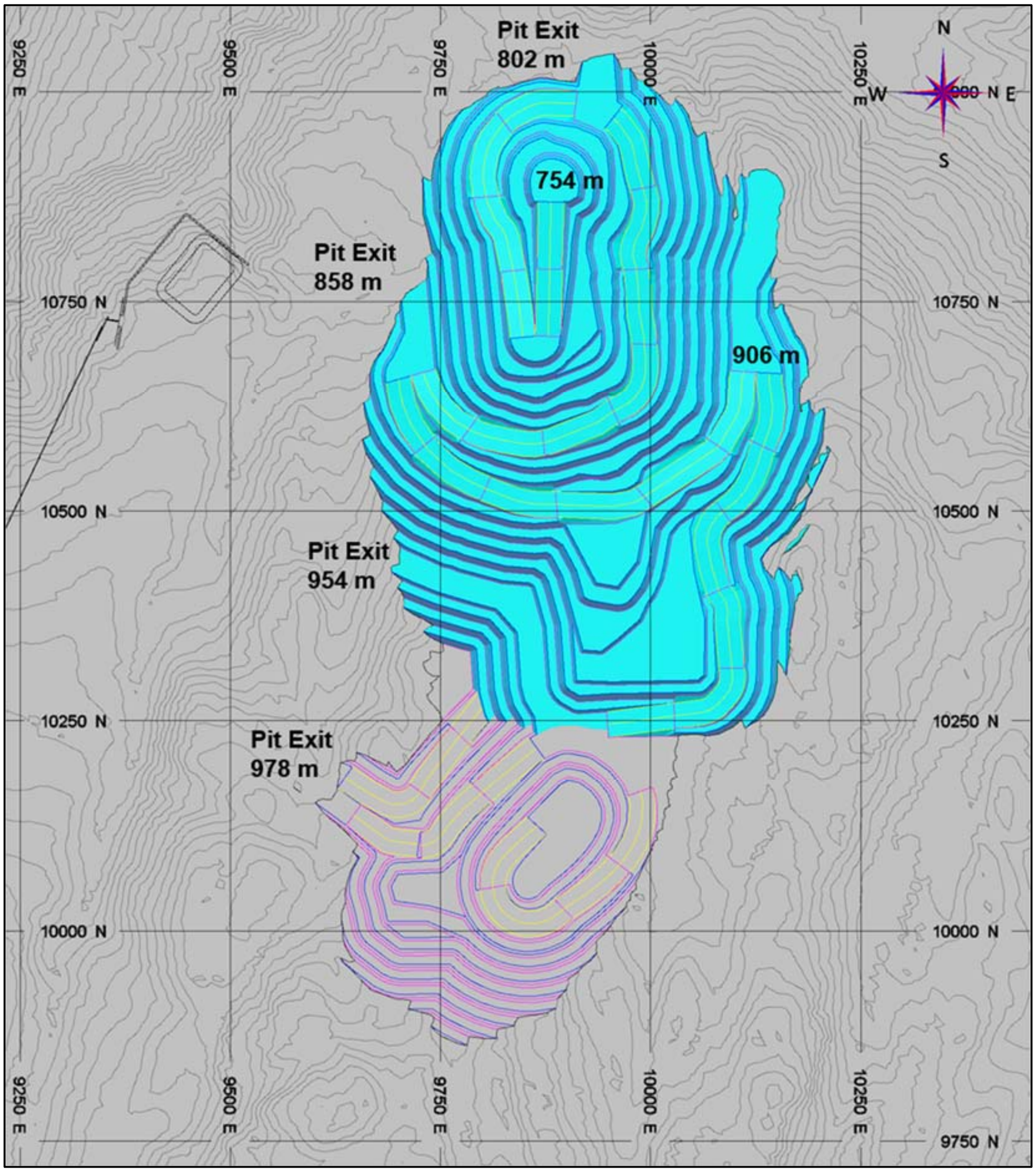
Phase	Mill Feed (Mt)	Au (g/t)	Ag (g/t)	Pb (%)	Zn (%)	Cu (%)	As (ppm)	Hg (ppm)	Sb (ppm)	Waste (Mt)	Total (Mt)	Strip Ratio
North Phase 1	2.21	5.75	84	0.07	0.13	0.021	4,499	347	3,515	15.3	17.5	6.9
North Phase 2	4.33	3.92	135	0.73	1.17	0.121	576	135	3,568	48.6	52.9	11.2
North Phase 3a	1.36	2.10	52	0.10	0.22	0.028	375	36	985	10.9	12.3	8.1
North Phase 3b	0.18	2.58	70	0.12	0.23	0.036	1,753	43	1,471	2.0	2.2	11.5
North Phase 3c	2.61	2.95	107	0.58	0.91	0.090	896	34	1,058	14.9	17.5	5.7
North Phase 4	9.48	2.72	52	0.26	0.43	0.048	321	22	682	59.6	69.0	6.3
South Phase 1	1.14	2.05	31	0.08	0.07	0.015	790	5	247	2.5	3.7	2.2
Total	***21.31***	***3.23***	***78***	***0.35***	***0.57***	***0.062***	***917***	***80***	***1,611***	***154.0***	***175.3***	***7.2***

Figure 16-19: Proposed North Phase 1



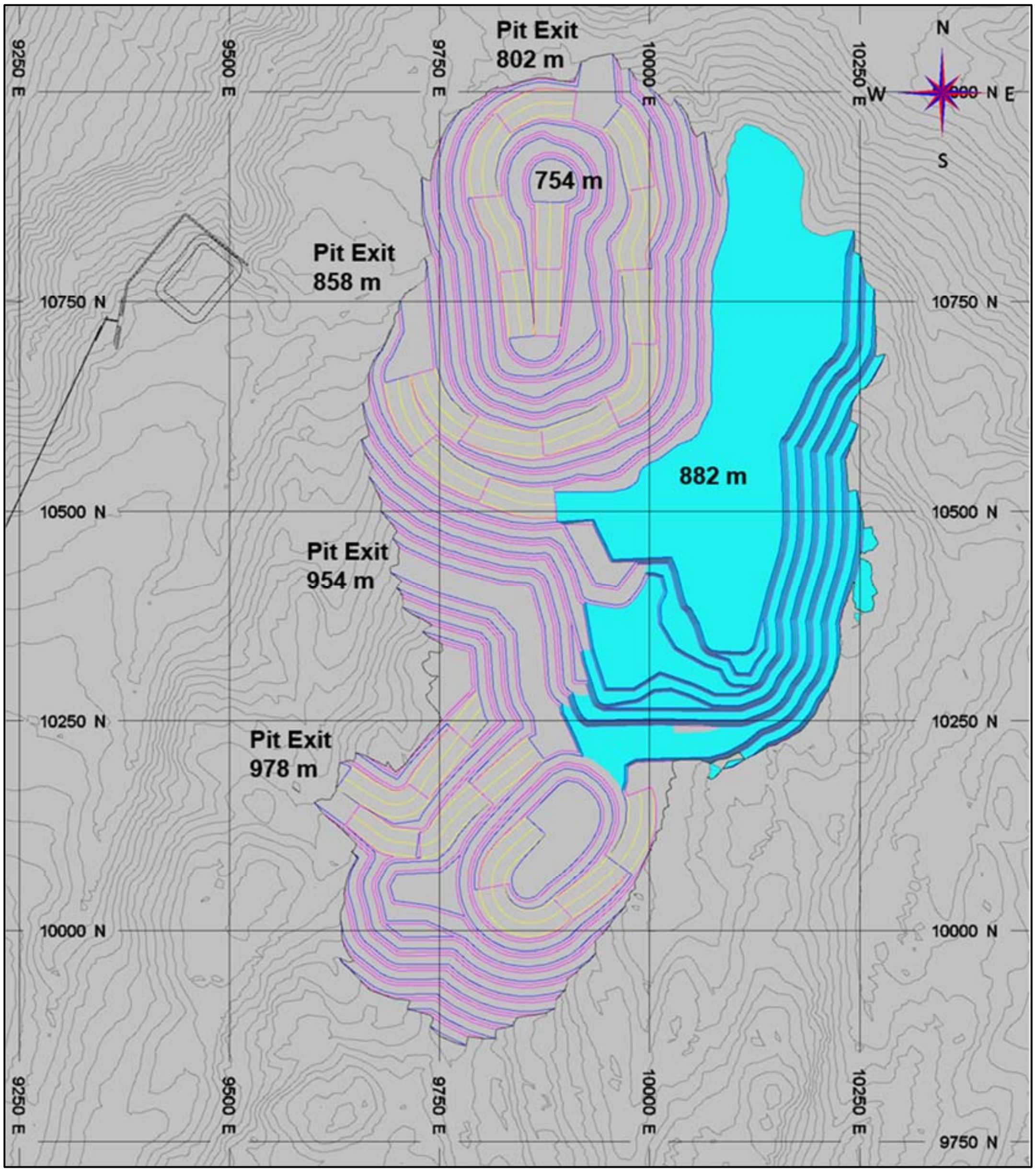
Note: Figure prepared by AGP, 2019.

Figure 16-20: Proposed North Phase 2



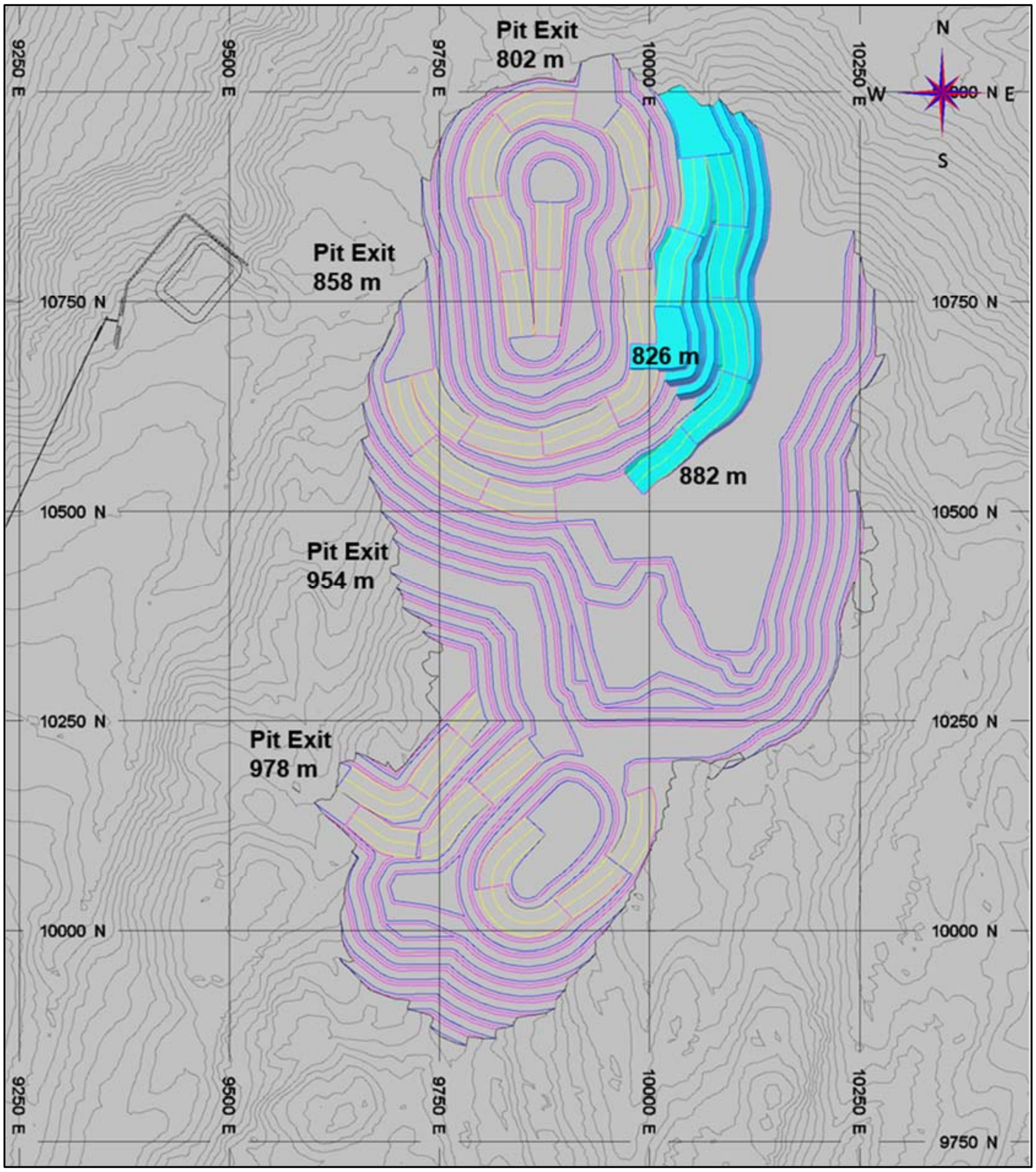
Note: Figure prepared by AGP, 2019.

Figure 16-21: Proposed North Phase 3a



Note: Figure prepared by AGP, 2019.

Figure 16-22: Proposed North Phase 3b



Note: Figure prepared by AGP, 2019.

16.7.5 North Phase 3c

Phase 3c will use the ramp developed in phase 3b to retreat mine down to the 826 m elevation with material being directed out via the pit exit at 858 m elevation. Below 826 m elevation, material will be directed out using a pit exit at 802 m elevation. Phase bench elevations will range from 874 masl down to 738 masl. At this stage, the entire east side of the pit will be completed to the ultimate pit limits. The phase 3c design is shown in Figure 16-23.

16.7.6 North Phase 4

Phase 4 will be the final phase to be mined and it will mine the west wall to its limits and extend the pit down to its pit bottom. The various pit exit points along the west wall will be used as the pit is advanced downward. Several of the bottom benches will be accessed by using the phase 3c haulage ramp from 802 m elevation downward. A small amount of waste will be required to connect to the bottom three benches. A single-lane ramp is designed for the last two benches. Phase bench elevations will range from 1066 masl down to 714 masl. The phase 4 design is shown in Figure 16-24.

16.7.7 South Phase 1

There will only be a single small phase in the south pit. Phase bench elevations will range from 1134 masl down to 1038 masl. This phase will be mined at the highest elevations of any other pit phase and it is likely to be accessed from the top of the WRSFs near the end of mining. The south pit design is shown in Figure 16-25.

16.8 Working Around Underground Voids

Best practice for advancing open pit mining operations through existing underground voids is to fill them with either waste or mill feed, which removes the void and supports the wall rock around the void (refer to discussion in Section 16.3.4.5).

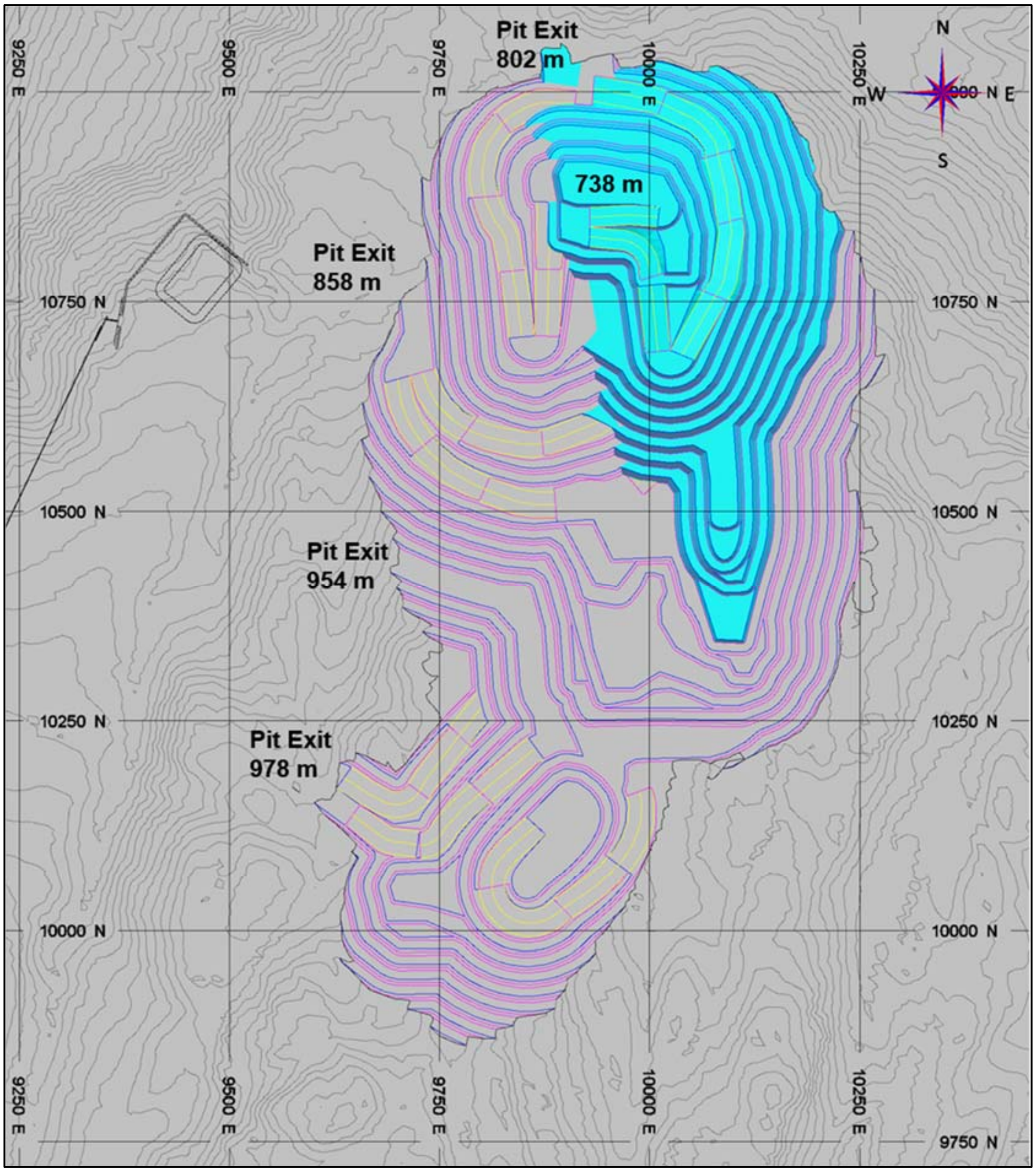
Although working around known voids will present some safety and productivity challenges, the bigger concern is the unknown voids. It is anticipated that the RC grade control drilling program will provide additional information regarding the location of the voids in advance of mining equipment being present. Additional support hours have been included in the cost estimate to compensate for the extra time required working around and preparing the old mine workings. The expected issue will be drifts as opposed to stopes as the stopes were backfilled with cemented material for stability. The location of the old workings is noted in the north pit and shown in dark blue in Figure 16-26.

16.9 Waste Rock Storage Facility Design

Various rock types are present in the material mined within the final pits. The waste rock includes the lithological types of hanging wall andesite, contact mudstone and rhyolite. Hanging wall andesite is the most prevalent waste rock as it makes up 75% of the total waste, while contact mudstone and rhyolite account for 5% and 20% respectively. All material types will be co-mingled in the WRSFs. The total amount of waste within the mine plan is 154 Mt.

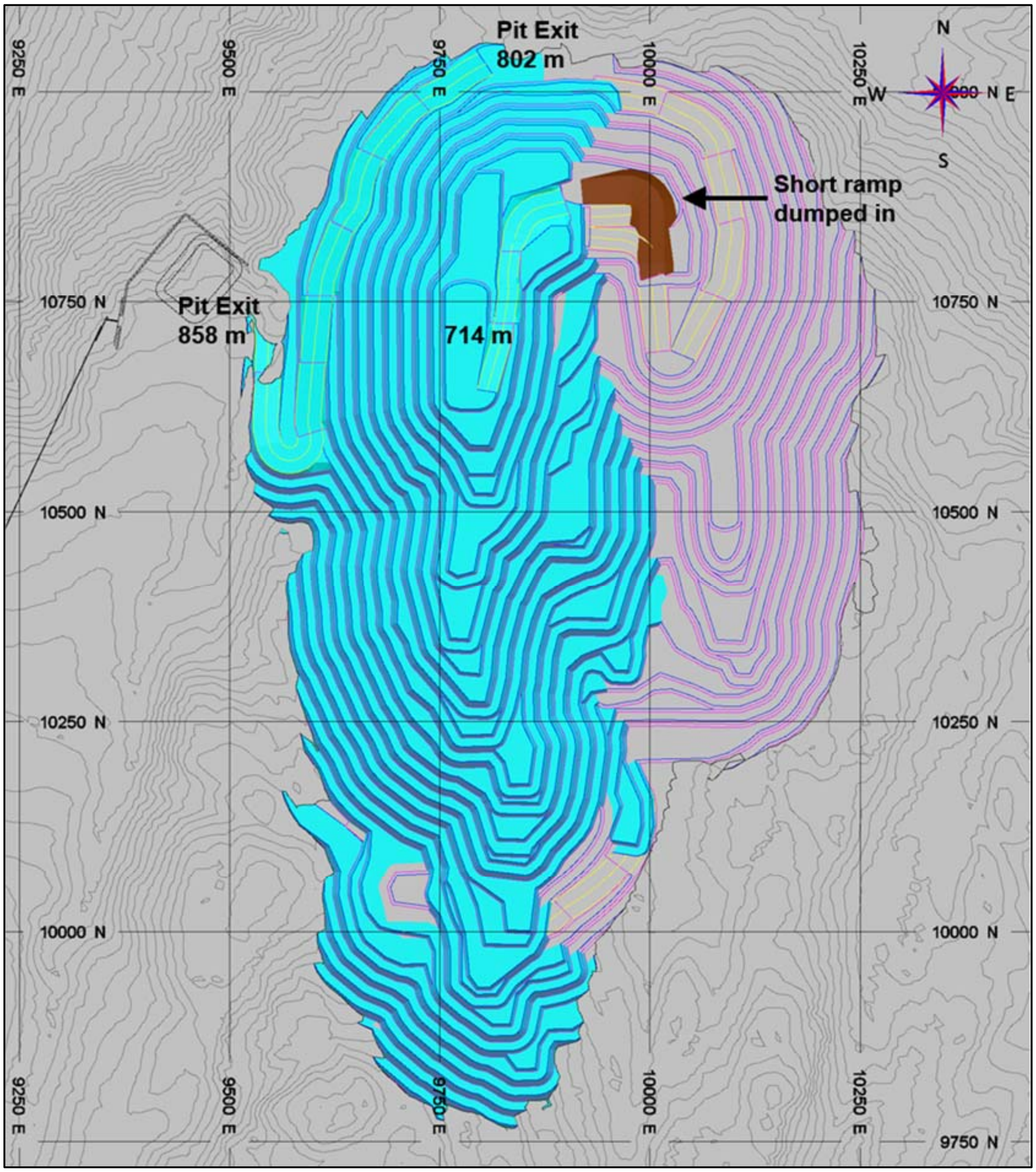
There will be two main WRSFs. The largest storage area will be WRSF WD01, which will be located along the west side of the pit. The remainder of the waste will be placed into the mined-out north pit as backfill. These two waste storage area locations are displayed in Figure 16-27. The projected storage capacities are shown in Table 16-10.

Figure 16-23: Proposed North Phase 3c



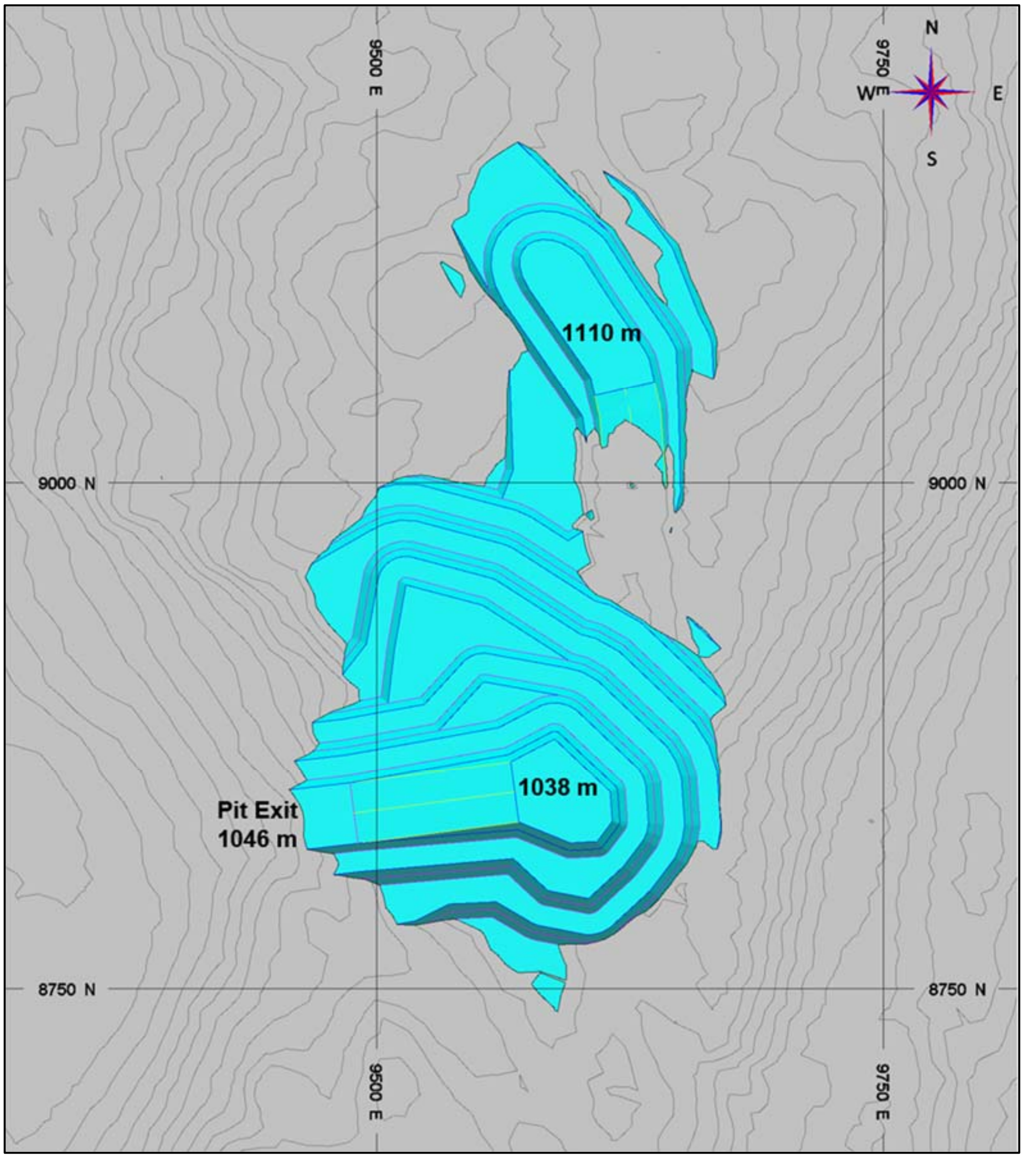
Note: Figure prepared by AGP, 2019.

Figure 16-24: Proposed North Phase 4



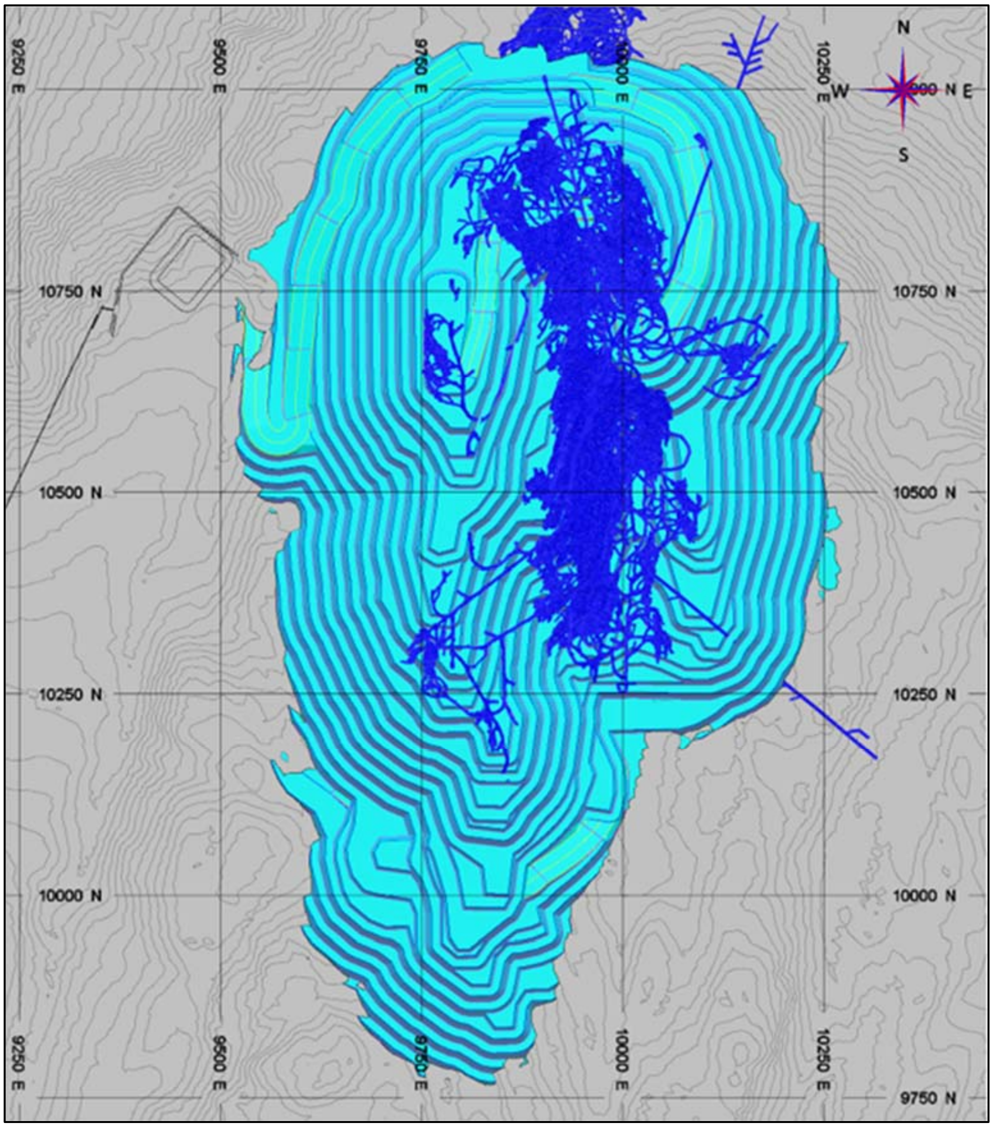
Note: Figure prepared by AGP, 2019.

Figure 16-25: Proposed South Phase 1



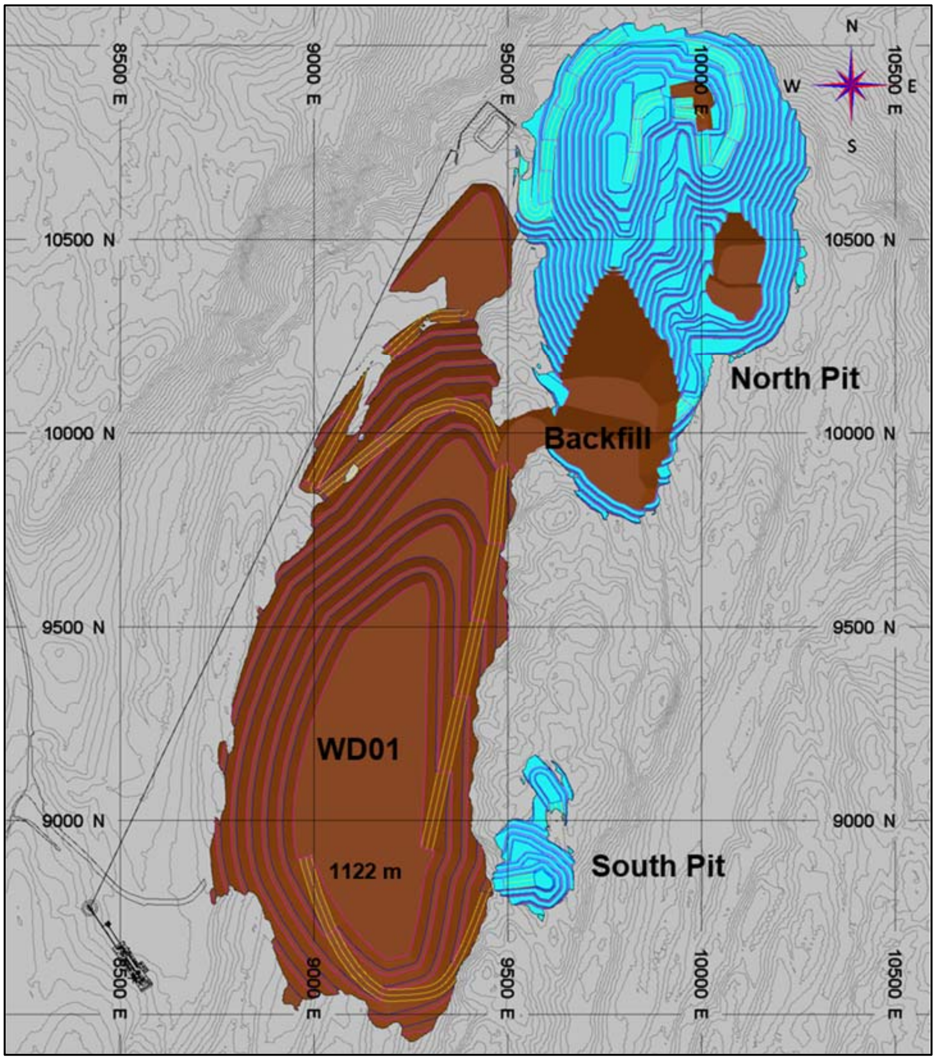
Note: Figure prepared by AGP, 2019.

Figure 16-26: Location of Historic Underground Workings



Note: Figure prepared by AGP, 2019.

Figure 16-27: Planned Waste Storage Areas



Note: Figure prepared by AGP, 2019.

Table 16-10: WRSF Parameters

Parameter	Units	WD01	North Pit Backfill
Waste storage capacity	Mm^3	70.4	6.8
Maximum elevation	masl	1122	1022

The WRSF design used a swell factor of 1.30. For the WD01 facility, the lift height will be 20 m. Assuming a 37° face slope, the overall slope will be 26.5° with 13.6 m berm widths. A 37° face slope was also used for the in-pit backfill WRSFs.

The WRSFs will be actively reclaimed as they are developed. Dozers will re-slope as the facilities are advanced to allow revegetation to occur as soon as possible. Drainage ditches will need to be in place along the west side of the WD01 facility, so water does not flow directly into Tom Mackay Creek.

For the PEA, the assumption was made that all waste material will be PAG. This will need to be confirmed in during more detailed studies. Drainage from the WRSF will be pumped to the settling pond to the west of the pit and treated as required.

16.10 Mine Schedule

The mine schedule was prepared on an annualised basis and is planned to deliver 21.3 Mt of mill feed grading 3.23 g/t gold and 78.0 g/t silver over a mine life of nine years. Waste tonnage totalling 154 Mt will be placed into either a WRSF or as pit backfill. The overall strip ratio is 7.2:1. The detailed planned mine schedule is shown in Table 16-11 and by phase in Table 16-12 and Figure 16-28. Figure 16-29 and Figure 16-30 show the variation of the proposed mill feed over the life of mine by mill feed type, grade, and contained ounces.

The mine schedule assumes a maximum of 2.5 Mt/a of feed will be sent to the process facility. Mill feed will be a mixture of hanging wall andesite, contact mudstone, and rhyolite. A maximum descent rate of eight benches per year per phase was applied.

The current mine life includes two years of pre-stripping and nine years of mining. Mill feed is stockpiled during the pre-production years. A maximum combined stockpile capacity of 800 kt was used due to limited storage space, with separate low-grade and high-grade piles. Mineralized material above a diluted gold grade of 3 g/t was considered high-grade material. The stockpiled mill feed, together with pit phasing, will be used to ensure mill feed is available during periods of poor weather. High precipitation will also necessitate in-pit sumps and surface ditches around the pits.

When mining starts, various infrastructure items will require development and construction activities. Significant activities near the pit will include construction of the mill feed overland conveyor and establishing proper roads to the mill feed crusher and to the WRSFs (ex-pit and in-pit). Operationally, ditching around the pits to intercept surface run-off will help to minimize reductions in mine production.

Year -2 has mining being initiated in Phase 1 of the north pit. In this time period, a total of 346 kt of waste material will be moved as the project ramps up. Year -1 mining brings an additional total material movement of 11.7 Mt moved. This includes the addition of 382 kt of mill feed to the mill stockpile grading 3.82 g/t Au and 64.4 g/t Ag in anticipation of plant commissioning and operation. Phases 1 and 2 will be the only active phases in the preproduction periods.

Project activities in the pre-production period include:

- Haul road construction;
- Overland conveyor construction;
- Initiation of mining in Phase 1 and 2;
- Development of a mill feed stockpile at the crusher for commissioning and operations.

Table 16-11: PEA Mine Schedule

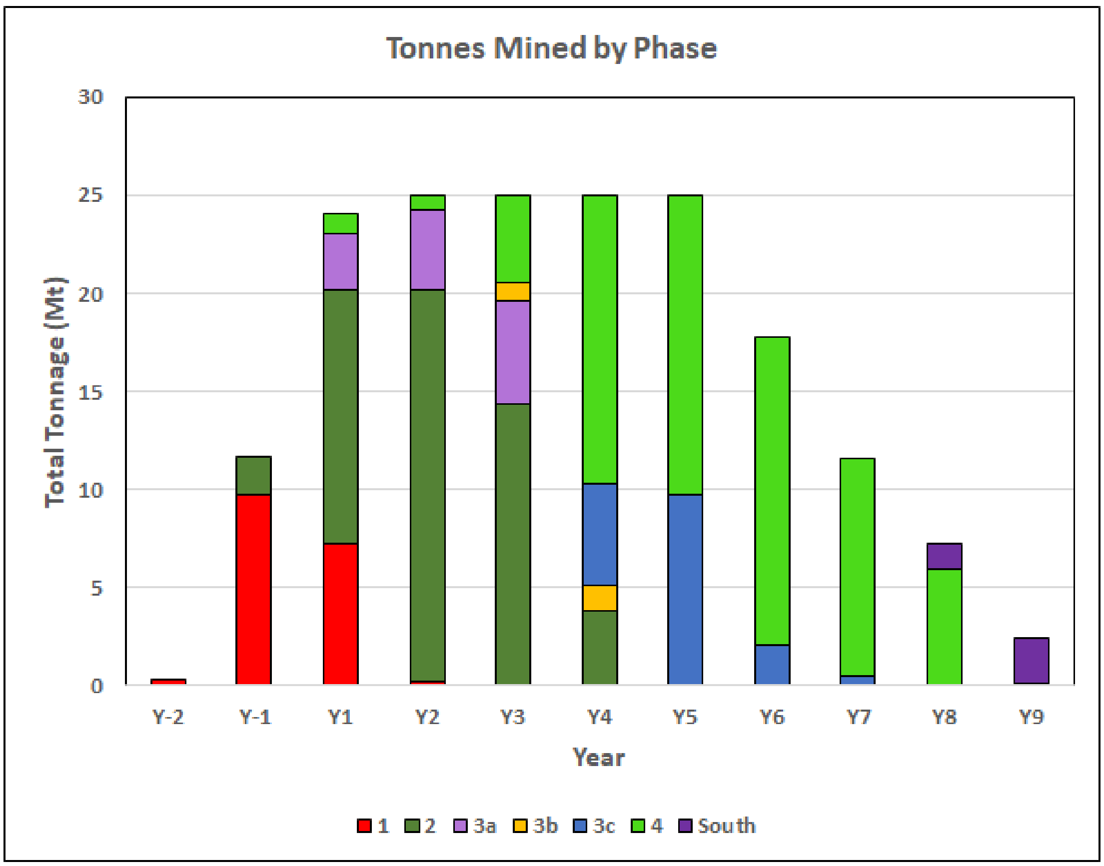
		Y-2	Y-1	Y1	Y2	Y3	Y4	Y5	Y6	Y7	Y8	Y9	Total
Mining Summary	***Waste (Mt)***	***0.3***	***11.3***	***21.3***	***23.2***	***22.4***	***22.5***	***22.3***	***15.1***	***9.1***	***4.8***	***1.6***	***154***
	Mill Feed (Mt)	***0.00***	***0.38***	***2.77***	***1.77***	***2.61***	***2.51***	***2.68***	***2.73***	***2.50***	***2.51***	***0.83***	***21.3***
	Au (g/t)	0.00	3.82	4.75	3.53	3.19	3.47	3.12	3.19	2.40	2.43	2.06	***3.23***
	Ag (g/t)	0.0	64.4	93.8	105.1	110.8	86.5	104.9	67.1	52.8	25.1	29.0	***78.0***
	Sb (ppm)	0	674	4,399	2,282	2,418	1,290	1,387	857	507	294	260	***1,611***
	Hg (ppm)	0	149	290	108	116	41	39	27	18	8	5	***80***
	As (ppm)	0	2,407	3,454	493	502	569	743	480	286	411	455	***917***
	Pb (%)	0.000	0.075	0.101	0.361	0.508	0.575	0.416	0.292	0.372	0.363	0.068	***0.353***
	Zn (%)	0.000	0.126	0.200	0.646	0.817	0.900	0.681	0.481	0.599	0.533	0.090	***0.572***
	Cu (%)	0.000	0.019	0.030	0.077	0.090	0.083	0.083	0.056	0.062	0.042	0.019	***0.062***
	Total (Mt)	***0.3***	***11.7***	***24.0***	***25.0***	***25.0***	***25.0***	***25.0***	***17.8***	***11.6***	***7.3***	***2.5***	***175***
Processed Material	***Mill Feed (Mt)***	***0.00***	***0.00***	***2.35***	***2.50***	***2.50***	***2.50***	***2.50***	***2.50***	***2.50***	***2.50***	***1.45***	***21.3***
	Au (g/t)	0.00	0.00	4.90	3.71	3.29	3.49	3.27	3.38	2.39	2.42	1.64	***3.23***
	Ag (g/t)	0.0	0.0	102.3	91.4	114.2	86.8	110.9	71.3	53.3	25.6	25.2	***78.0***
	Sb (ppm)	0	0	4,068	2,755	2,507	1,297	1,457	915	516	302	264	***1,611***
	Hg (ppm)	0	0	243	190	120	42	41	29	18	9	8	***80***
	As (ppm)	0	0	3,353	1,386	516	571	778	500	281	416	358	***917***
	Pb (%)	0.000	0.000	0.110	0.272	0.517	0.578	0.430	0.296	0.363	0.367	0.146	***0.353***
	Zn (%)	0.000	0.000	0.215	0.491	0.834	0.904	0.705	0.495	0.586	0.539	0.205	***0.572***
	Cu (%)	0.000	0.000	0.032	0.060	0.092	0.083	0.087	0.059	0.062	0.042	0.020	***0.062***
Stockpile Balance	***Low Grade (Mt)***	***0.00***	***0.27***	***0.60***	***0.07***	***0.18***	***0.19***	***0.37***	***0.60***	***0.60***	***0.60***	***0.00***	
	Au (g/t)	0.00	1.47	1.47	0.83	0.85	0.84	0.91	0.98	0.98	0.99	0.00	
	Ag (g/t)	0.0	47.9	38.8	25.4	27.0	25.9	23.5	21.7	20.2	18.5	0.0	
	Sb (ppm)	0	323	810	479	410	377	393	328	293	264	0	
	Hg (ppm)	0	47	150	58	30	27	21	17	15	13	0	
	As (ppm)	0	689	962	140	157	152	202	223	246	227	0	
	Pb (%)	0.000	0.081	0.060	0.102	0.218	0.202	0.209	0.221	0.261	0.246	0.000	
	Zn (%)	0.000	0.125	0.108	0.183	0.315	0.293	0.318	0.317	0.377	0.350	0.000	
	Cu (%)	0.000	0.017	0.017	0.016	0.019	0.018	0.024	0.022	0.022	0.021	0.000	

		Y-2	Y-1	Y1	Y2	Y3	Y4	Y5	Y6	Y7	Y8	Y9	***Total***
	High Grade (Mt)	***0.00***	***0.11***	***0.20***	***0.00***	***0.00***	***0.00***	***0.00***	***0.00***	***0.01***	***0.02***	***0.00***	
	Au (g/t)	0.00	9.29	10.98	0.00	4.74	4.74	4.74	6.53	4.59	3.81	0.00	
	Ag (g/t)	0.0	102.9	103.8	0.0	615.2	615.2	615.2	405.4	172.6	64.6	0.0	
	Sb (ppm)	0	1,489	11,950	0	3,374	3,374	3,374	1,838	827	386	0	
	Hg (ppm)	0	388	992	0	128	128	128	48	23	13	0	
	As (ppm)	0	6,400	10,118	0	398	398	398	293	207	278	0	
	Pb (%)	0.000	0.060	0.061	0.000	1.944	1.944	1.944	1.021	0.508	0.401	0.000	
	Zn (%)	0.000	0.129	0.154	0.000	2.868	2.868	2.868	1.553	0.775	0.592	0.000	
	Cu (%)	0.000	0.022	0.022	0.000	0.363	0.363	0.363	0.197	0.088	0.047	0.000	
Total Stockpile Additions	***(Mt)***	***0.00***	***0.38***	***0.53***	***0.07***	***0.11***	***0.02***	***0.18***	***0.23***	***0.13***	***0.11***	***0.00***	***1.77***
Total Stockpile Reclaim	***(Mt)***	***0.00***	***0.00***	***0.11***	***0.80***	***0.00***	***0.01***	***0.00***	***0.00***	***0.12***	***0.10***	***0.62***	***1.77***
Total Material Movement	***(Mt)***	***0.3***	***11.7***	***24.2***	***25.8***	***25.0***	***25.0***	***25.0***	***17.8***	***11.7***	***7.4***	***3.1***	***177***
Concentrate	***Tonnages (kt)***	***0***	***0***	***425***	***342***	***302***	***321***	***301***	***311***	***215***	***218***	***81***	***2,516***
	Au (g/t)	***0***	***0***	***25***	***25***	***25***	***25***	***25***	***25***	***25***	***25***	***25***	***25***
	Ag (g/t)	***0***	***0***	***208***	***167***	***217***	***182***	***189***	***157***	***211***	***191***	***0***	***184***
	Sb (ppm)	***0***	***0***	***20,756***	***18,547***	***19,061***	***9,300***	***11,143***	***6,762***	***5,393***	***3,113***	***4,009***	***12,527***
	Hg (ppm)	***0***	***0***	***1,239***	***1,281***	***915***	***299***	***311***	***213***	***191***	***89***	***124***	***623***
	As (ppm)	***0***	***0***	***17,109***	***9,330***	***3,927***	***4,096***	***5,949***	***3,700***	***2,933***	***4,293***	***5,448***	***7,116***
	Pb (%)	***0.00***	***0.00***	***0.56***	***1.83***	***3.93***	***4.14***	***3.29***	***2.19***	***3.79***	***3.79***	***2.22***	***2.73***
	Zn (%)	***0.00***	***0.00***	***1.10***	***3.31***	***6.34***	***6.48***	***5.39***	***3.66***	***6.12***	***5.57***	***3.11***	***4.43***
	Cu (%)	***0.00***	***0.00***	***0.17***	***0.40***	***0.70***	***0.60***	***0.66***	***0.44***	***0.65***	***0.44***	***0.31***	***0.48***

Table 16-12: Tonnes Mined by Phase

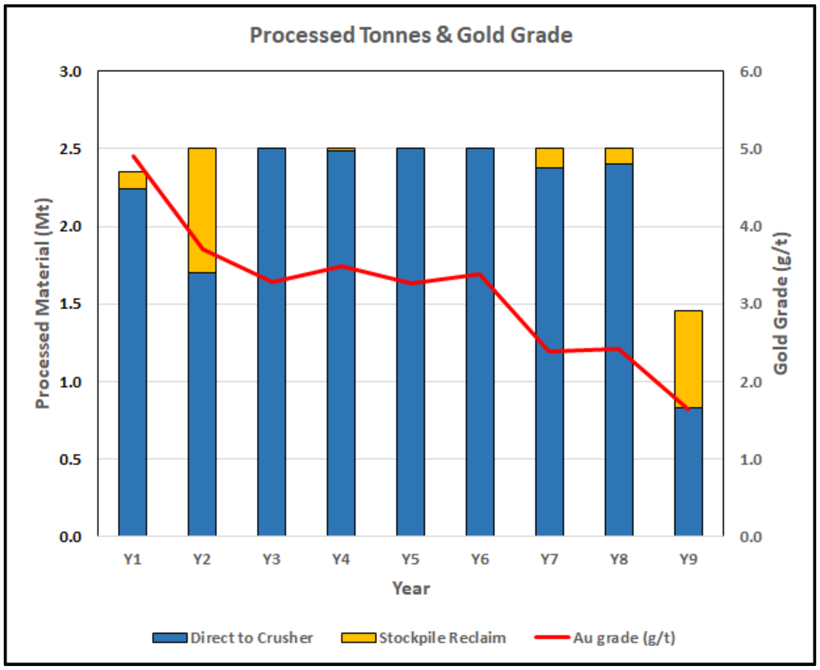
Phase	Total Tonnage (Mt)											Total (Mt)
	Y-2	Y-1	Y1	Y2	Y3	Y4	Y5	Y6	Y7	Y8	Y9	
1	0.3	9.7	7.2	0.2								17.5
2		2.0	12.9	19.9	14.3	3.8						52.9
3a			2.9	4.1	5.3							12.3
3b					0.9	1.3						2.2
3c						5.2	9.8	2.0	0.5			17.5
4			1.0	0.8	4.4	14.7	15.2	15.8	11.1	6.0	0.1	69.0
South										1.3	2.4	3.7
Total	***0.3***	***11.7***	***24.0***	***25.0***	***25.0***	***25.0***	***25.0***	***17.8***	***11.6***	***7.3***	***2.5***	***175.3***

Figure 16-28: Tonnes Mined by Phase



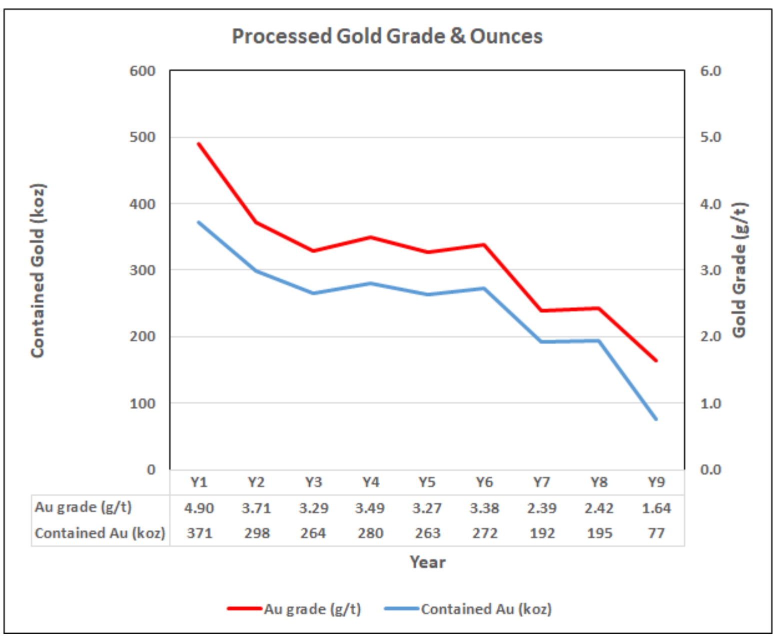
Note: Figure prepared by AGP, 2019.

Figure 16-29: Planned Life-of-Mine Mill Feed Tonnes and Ounces



Note: Figure prepared by AGP, 2019.

Figure 16-30: Process Grade and Contained Ounces of Gold



Note: Figure prepared by AGP, 2019.

Year 1 production assumes the plant will require three months to achieve full production levels. The first month the plant will be capable of 60% of capacity, the second month 80%, and the third month 90%. Subsequent months will be at 100% of nameplate capacity in the mill. This plant ramp-up schedule requires the Year 1 production to be 2.35 Mt. Mill feed will be from stockpile, Phase 1, Phase 2 and Phase 3a. This period has the initial mining in Phase 3a, with Phases 1 and 2 continuing to be active.

Year 2 production will be at the full 2.5 Mt of mill feed. Phase 1 mining will be completed with its final level being 922 masl. Phases 2 and 3a will continue to be mined down to the levels of 866 masl, and 906 masl respectively. All waste will be directed to the WD01 facility.

Table 16-13 displays a summary of the resource classifications for the mill feed.

Year 3 production will be the final year for Phase 3a with its final level being 882 masl. Phase 2 will be the dominant phase of mining in this period, driving to a depth of 802 masl. Phases 3b and 4 will be initiated and advance down to levels 866masl and 970 masl respectively. All waste will be directed to the WD01 facility.

Year 4 production will be the final year of mining for Phases 2 and 3b, with their final levels being 754 masl and 826 masl respectively. Mining will be initiated in Phase 3c and will continue down to 842 masl. Phase 4 will continue to advance down to 906 masl. From this period forward, a small portion of the waste material will be directed back into the pit as backfill as space allows. A small amount of backfill space will be used on the east side of the pit in this period.

Year 5 will have Phase 3c and 4 as the only active mining phases. Phases 3c and 4 will be advanced down to 778 masl and 858 masl respectively. Most of the waste will be directed to the WD01 facility, but approximately 15% of the waste will be directed to the east and west sides of the pit as more of the ultimate pit is exposed.

Year 6 will have Phase 3c and Phase 4 continuing to be mined. Phase 3c will be mined down to 754 masl while Phase 4 will be advanced down to 810 masl. All waste will be sent to the WD01 facility.

Year 7 will have Phase 3c continuing and mining to completion at 738 masl. Phase 4 will continue to be mined and will be advanced down to 762 masl. Approximately 40% of the waste material will be directed to a Phase 4 backfill elevation at 1022 masl, with the remainder to the WD01 facility.

Year 8 will have mining in Phase 4 while the south pit mining will be initiated. Phase 4 will be advanced down to 714 masl while the south pit will be advanced down to 1094 masl. Phase 4 waste material will be directed to a Phase 4 backfill elevation at 978 masl, while the south pit waste material will be directed to the WD01 facility.

Year 9 will be the final mining period with mining being completed in both Phase 4 and the south pit. Phase 4 will have mining completed on the 714 masl level while the south pit will be advanced down to 1038 masl.

The mine schedule was completed on an annual basis for the entire schedule. The mine is scheduled to deliver 21.3 Mt of mill feed grading 3.23 g/t Au and 78.0 g/t Ag. Waste totalling 154 Mt will be stored in the WD01 facility that will be located along the west side of the pit, as well into the north pit as backfill. The overall strip ratio is 7.2:1.

Table 16-13: Resource Summary of Scheduled Material

Resource Class	Mill Feed (Mt)	Grade		Contained Ounces	
		Au (g/t)	Ag (g/t)	Au (Moz)	Ag (Moz)
Indicated	10.7	4.22	107	1.45	36.8
Inferred	10.6	2.24	48	0.76	16.5
Total	***21.3***	***3.23***	***78***	***2.21***	***53.3***

16.11 Mine Plan Sequence

Anticipated end-of-year positions for the open pits are shown in Figure 16-31 to Figure 16-41.

Mining will be initiated in the north pit and will continue throughout the schedule, while the south pit will only be active in years 8 and 9.

16.12 Mining Equipment Selection

The mining equipment selected to meet the required production schedule is conventional mining equipment, with additional support equipment for snow removal and surface ditching maintenance.

Drilling will be completed with down the hole hammer (DTH) drills with a 140 mm bit. This will provide the capability to drill patterns for either 4 m or 8 m bench heights.

The primary loading units will be 22 m^3 hydraulic shovels. Additional loading will be completed by 13 m^3 loaders. It is expected that one of the loaders will be at the primary crusher for the majority of its operating time. The haulage trucks will be conventional 142 t rigid body trucks.

The support equipment fleet will be responsible for the usual road, pit and dump maintenance requirements, but due to the climate conditions expected, will have a larger role in snow removal and water management. Snowplows and additional graders were included in the fleet. In addition, smaller road maintenance equipment is included to keep drainage ditches open and sedimentation ponds functional.

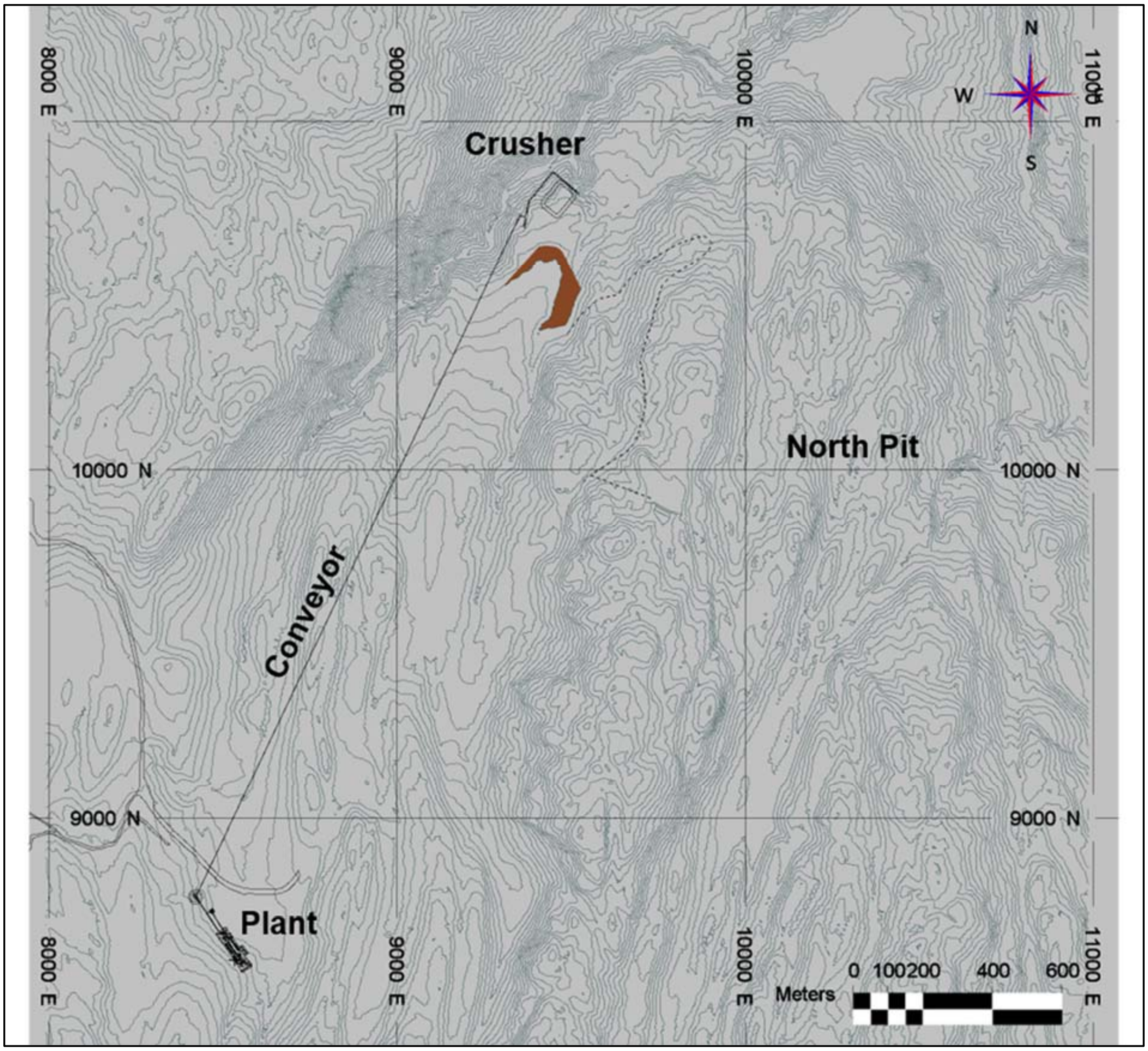
Within the planned pit, an additional large backhoe will assist the mill feed preparation. It will be responsible for cleaning hanging wall and footwall material around the old cemented stopes from the underground mining. While capable of loading the 142 t trucks if required, it is not scheduled to do so because of the extended loading time necessary. The backhoe/truck combination is not as efficient as the proposed primary loading units.

The proposed equipment requirements for the LOMP are included in Section 21.2.4.

16.13 Grade Control

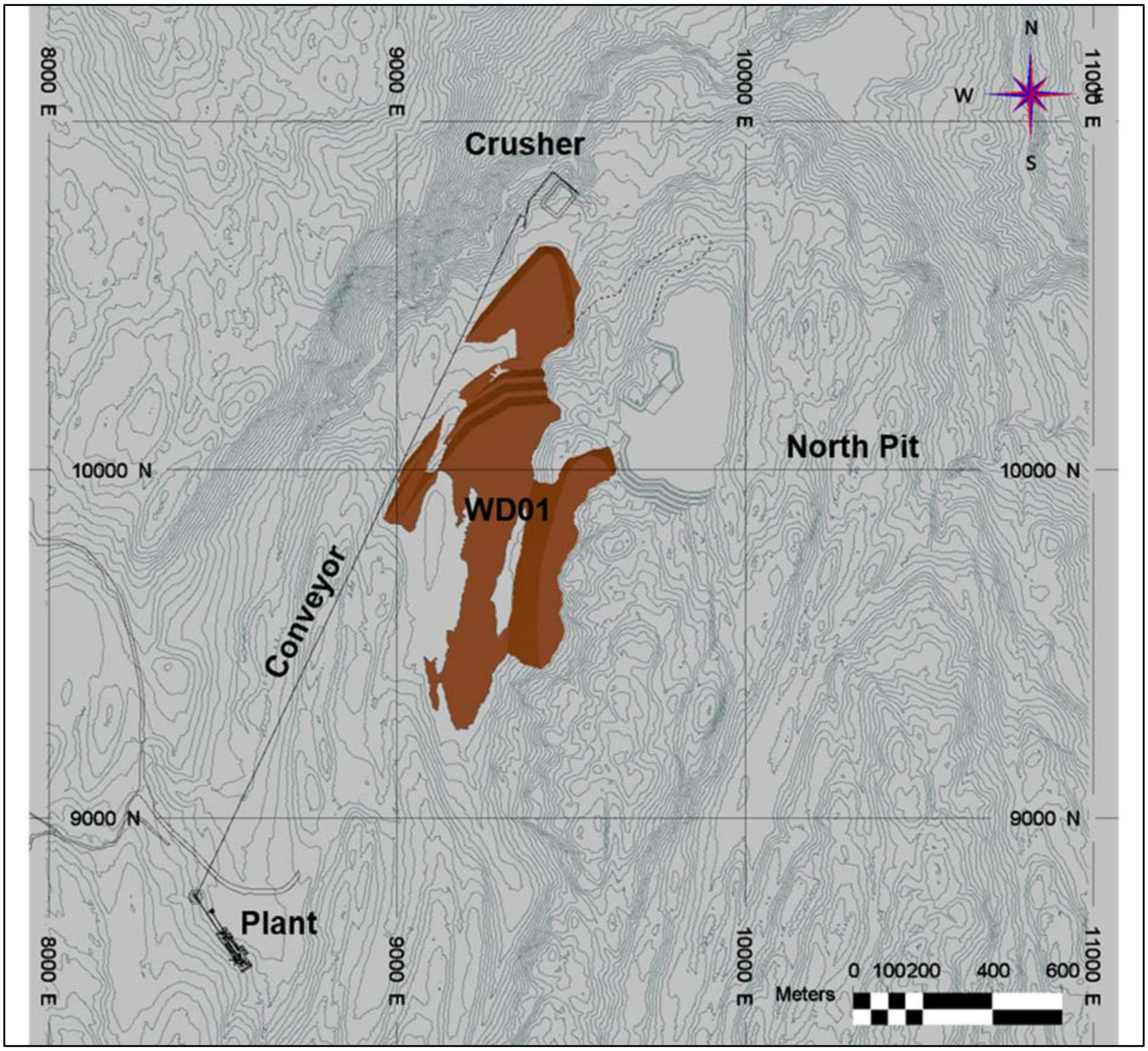
Grade control will be completed with a separate fleet of RC drill rigs. They will drill the deposit off on a 10 m x 5 m pattern in areas of known mineralization taking samples each metre. The holes will be inclined at 60º.

Figure 16-31: End of Preproduction Period – Year -2



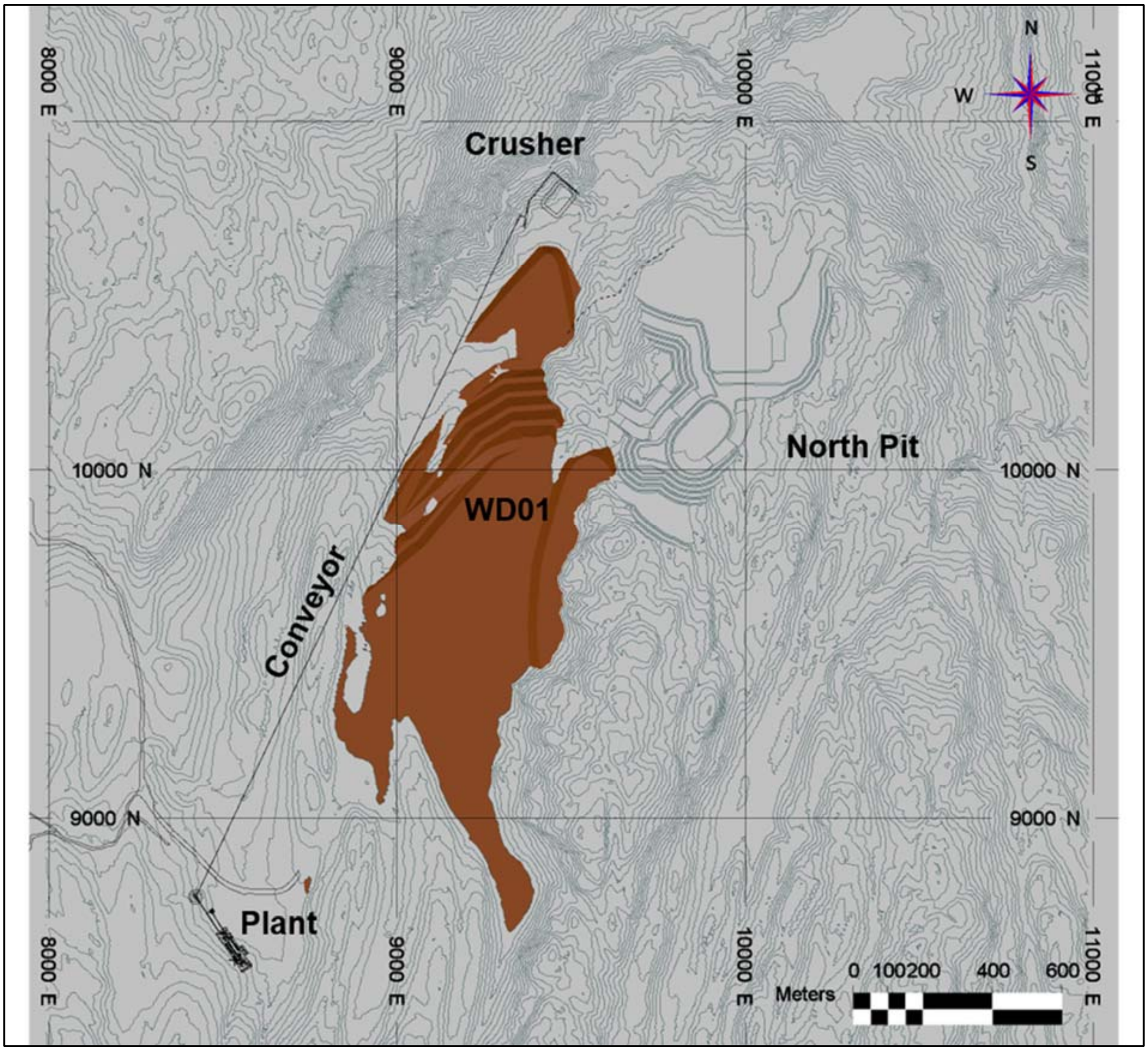
Note: Figure prepared by AGP, 2019.

Figure 16-32: End of Preproduction period – Year -1



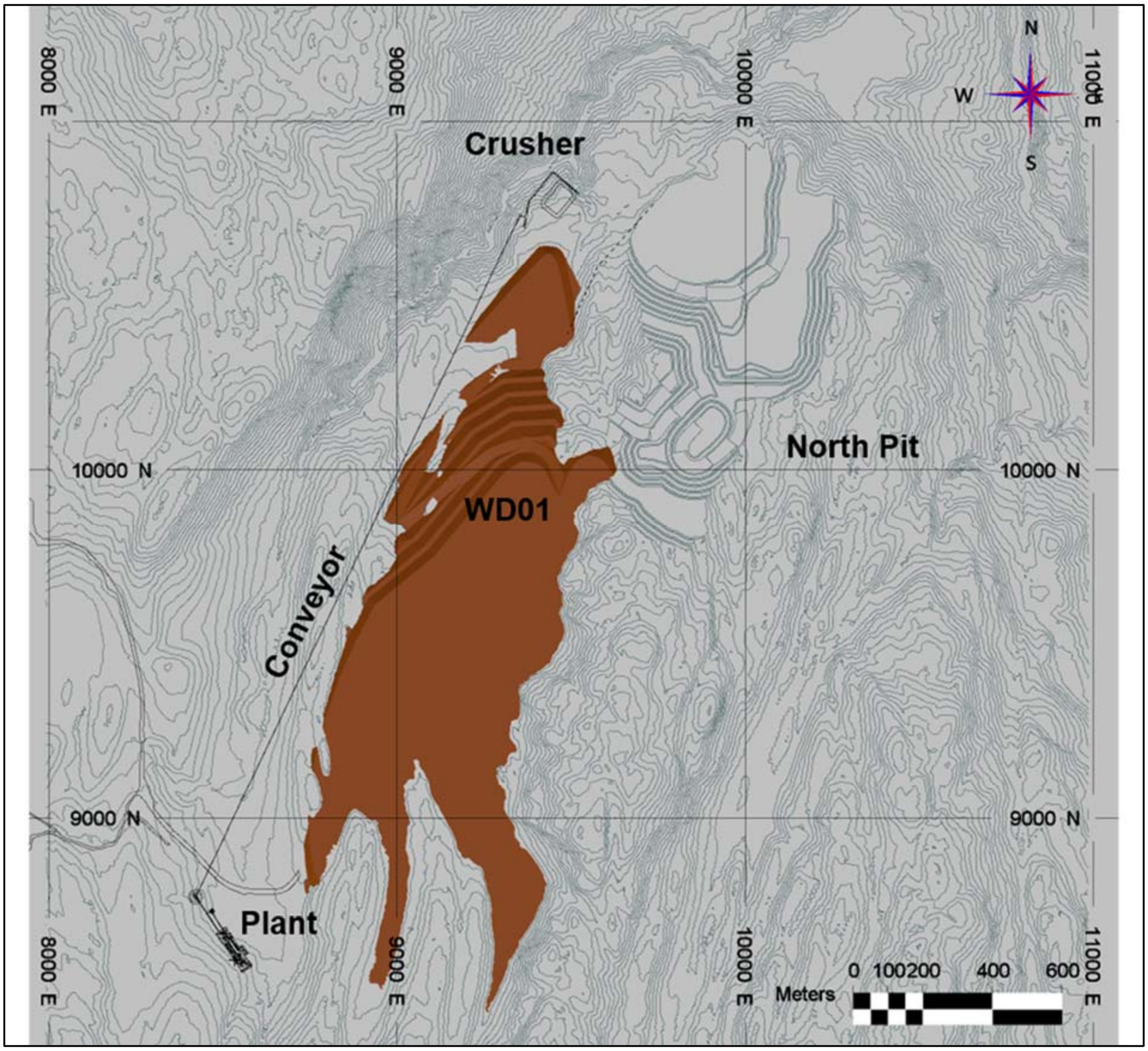
Note: Figure prepared by AGP, 2019.

Figure 16-33: End of Year 1



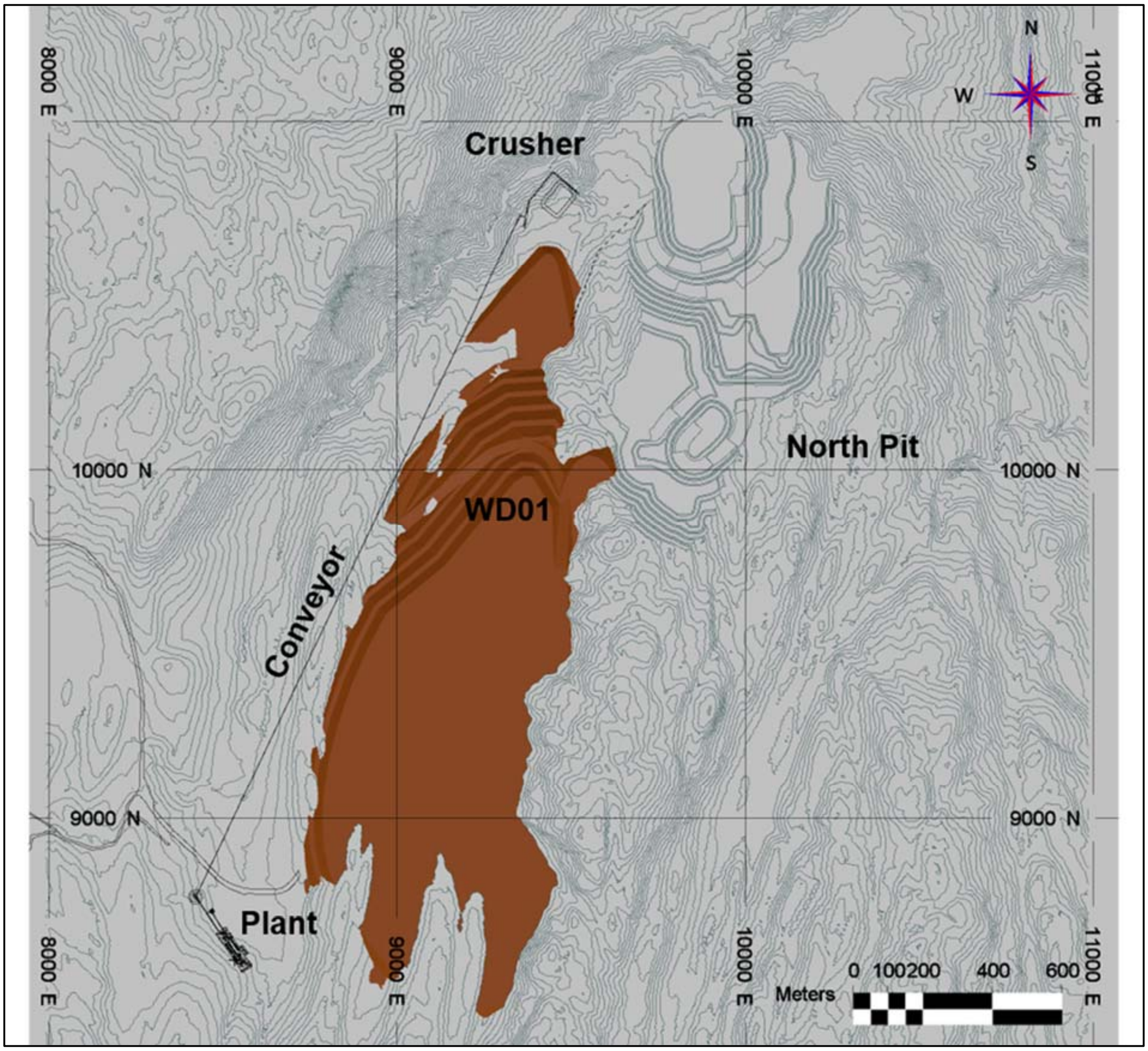
Note: Figure prepared by AGP, 2019.

Figure 16-34: End of Year 2



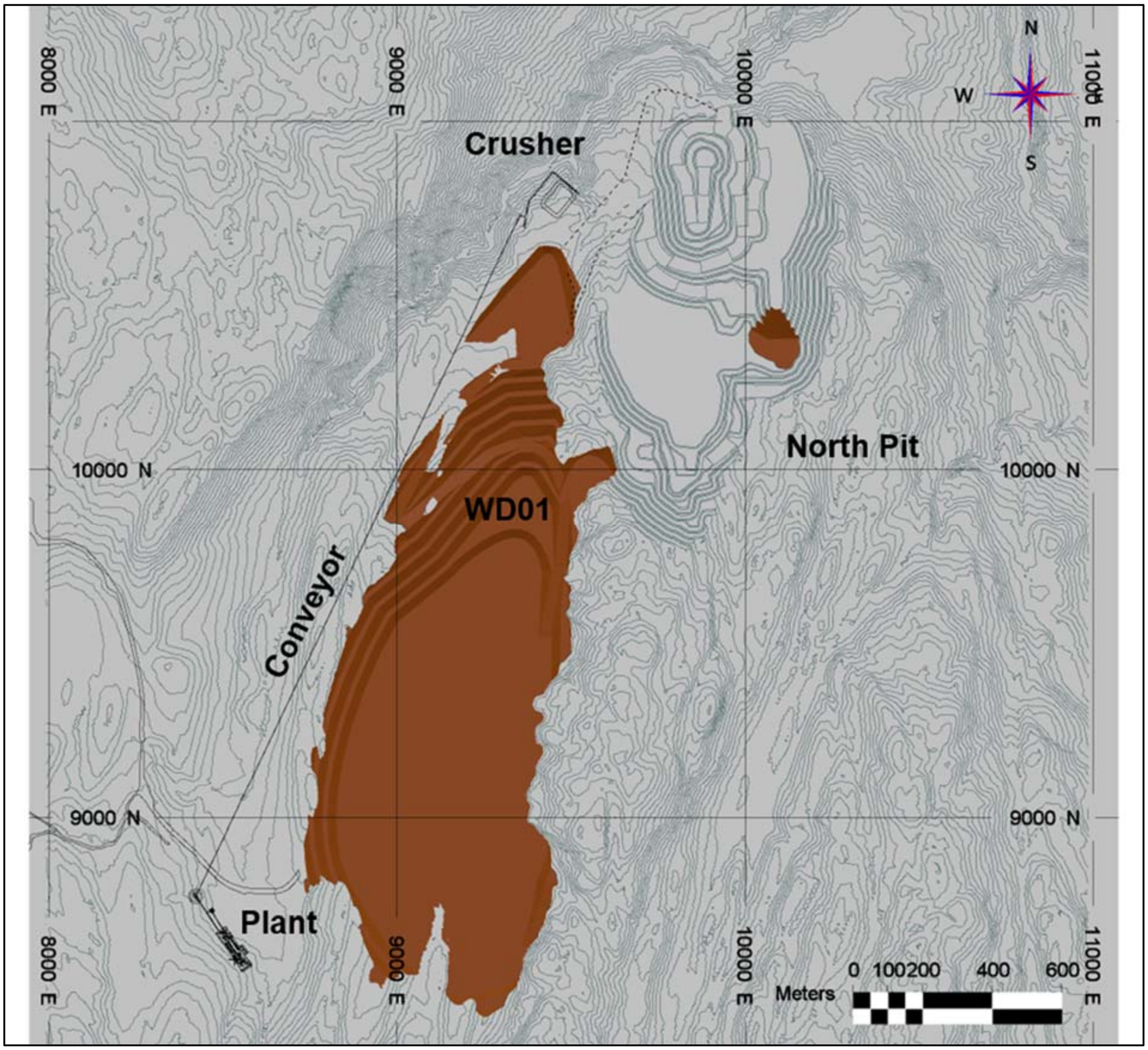
Note: Figure prepared by AGP, 2019.

Figure 16-35: End of Year 3



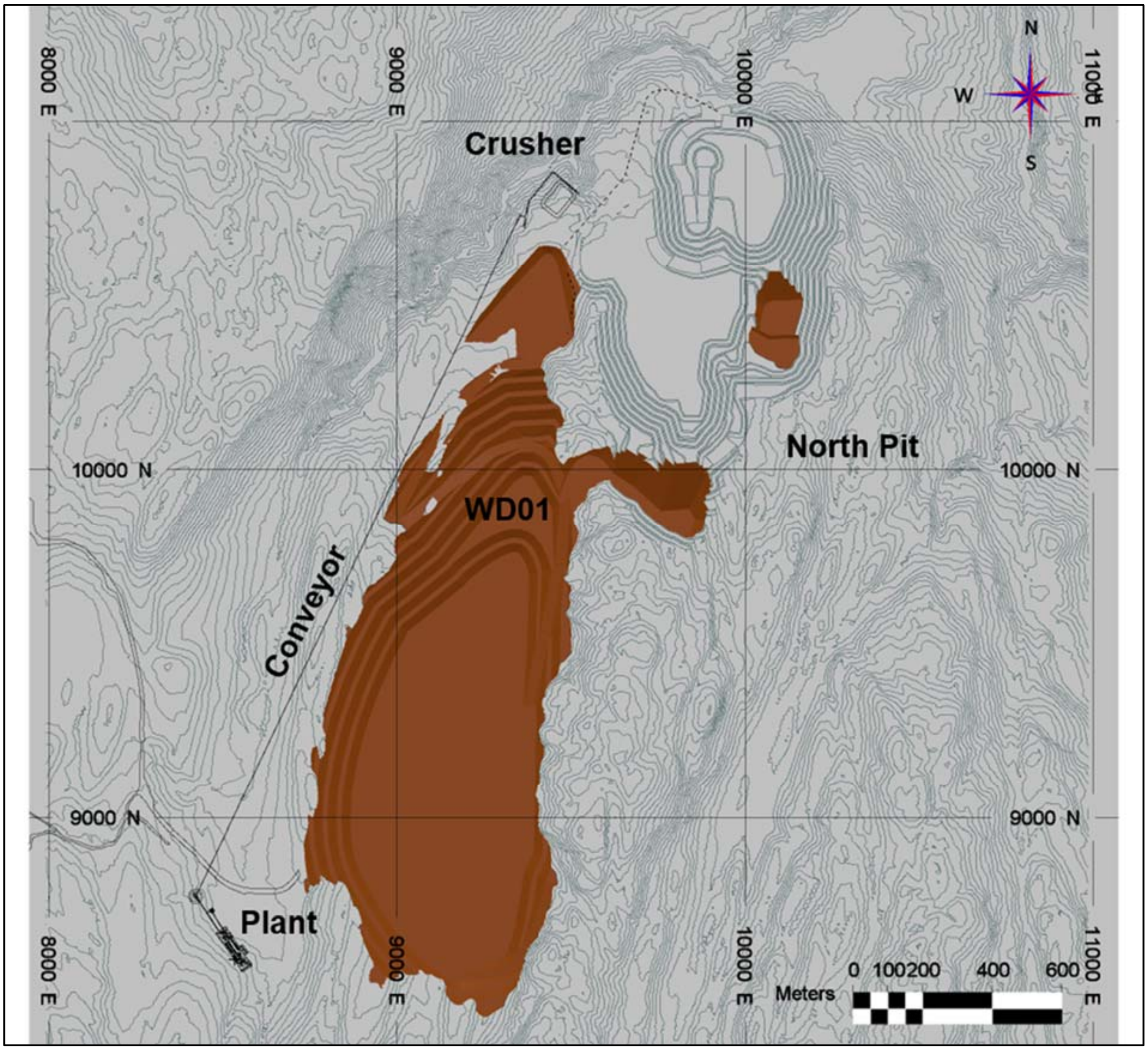
Note: Figure prepared by AGP, 2019.

Figure 16-36: End of Year 4



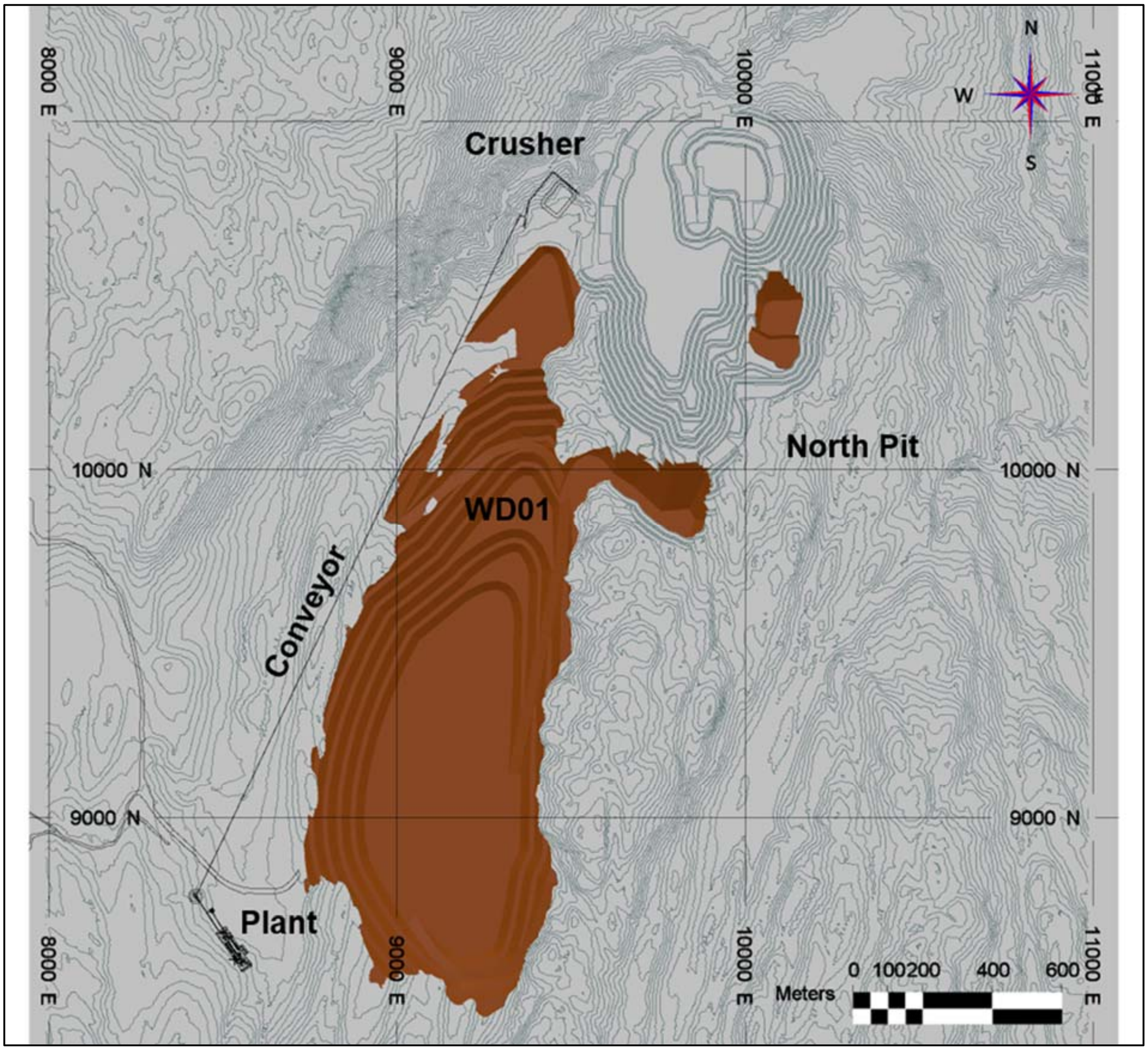
Note: Figure prepared by AGP, 2019.

Figure 16-37: End of Year 5



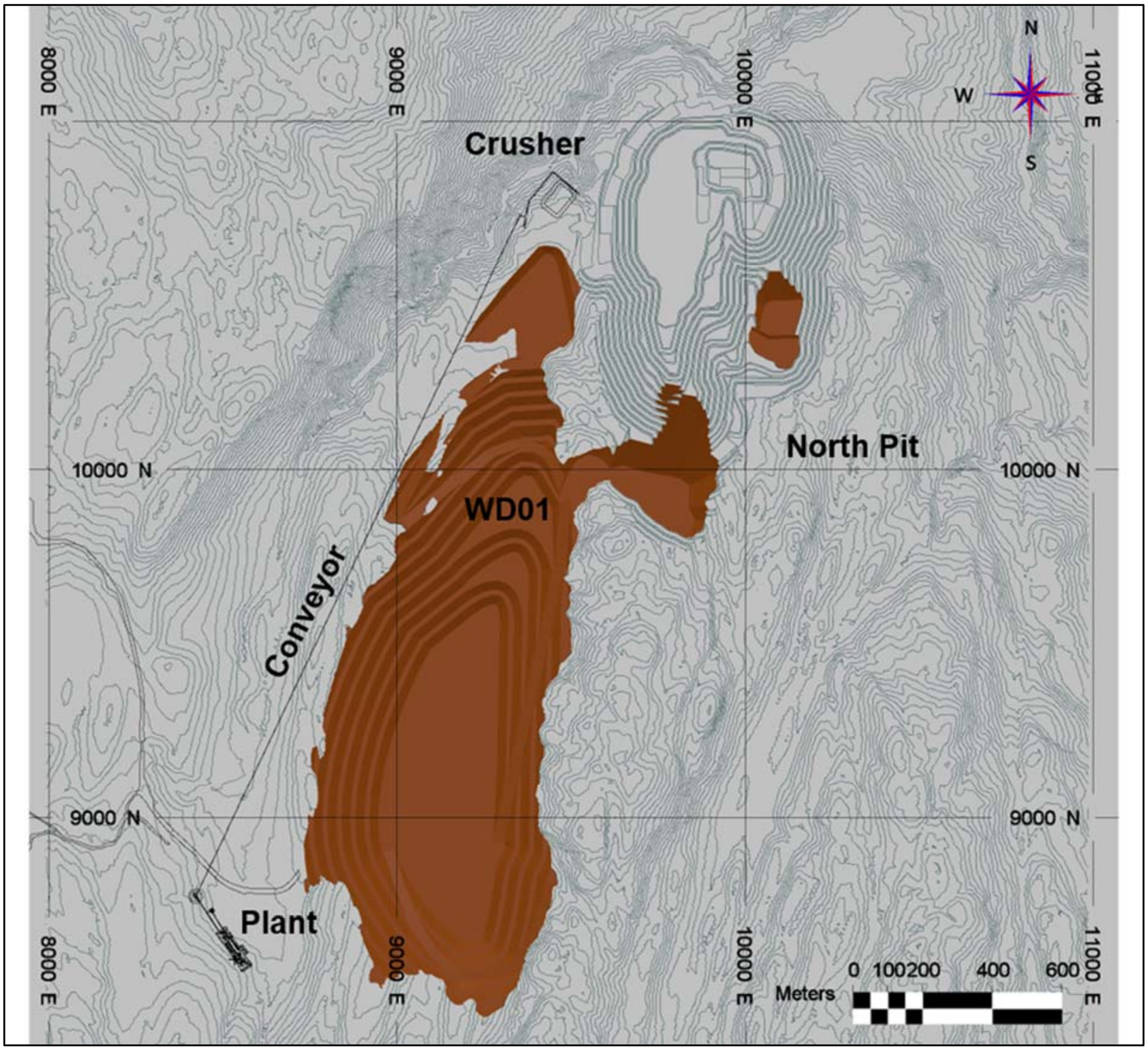
Note: Figure prepared by AGP, 2019.

Figure 16-38: End of Year 6



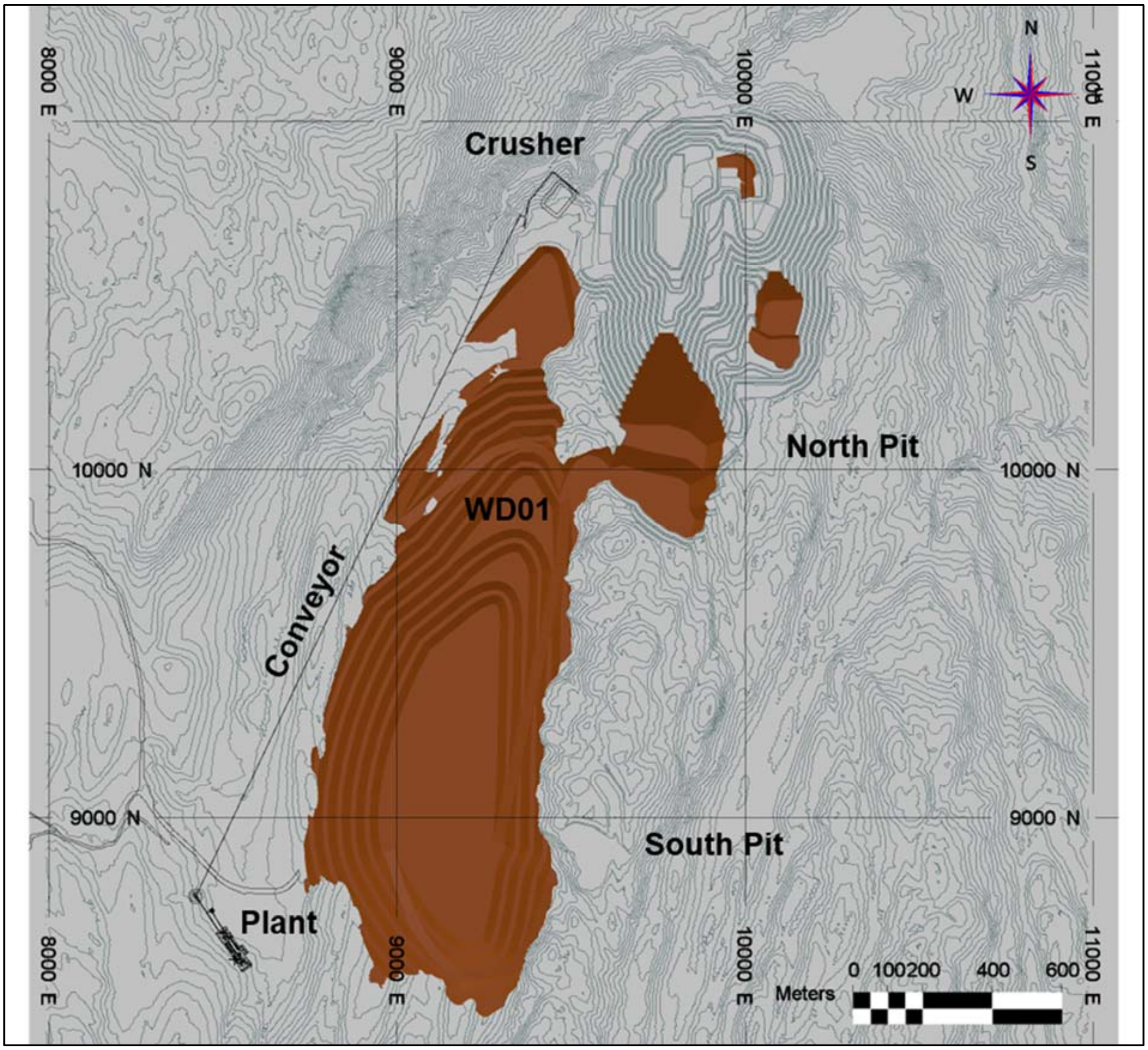
Note: Figure prepared by AGP, 2019.

Figure 16-39: End of Year 7



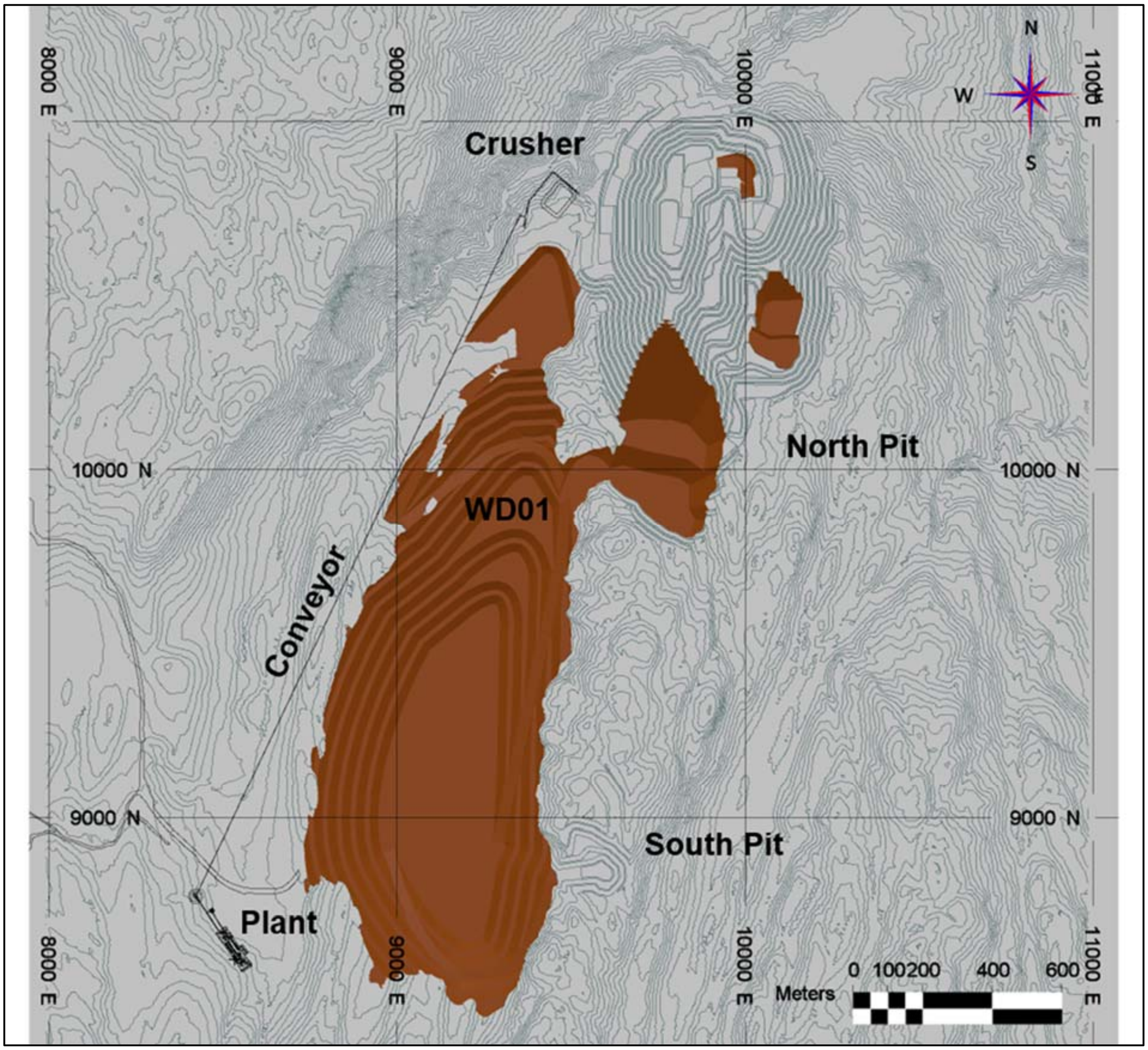
Note: Figure prepared by AGP, 2019.

Figure 16-40: End of Year 8



Note: Figure prepared by AGP, 2019.

Figure 16-41: End of Year 9 (End of Mining)



Note: Figure prepared by AGP, 2019.

In areas of low-grade mineralization or waste the pattern spacing will be 20 m x 10 m, with sampling over 6 m. These drill holes will be used to find undiscovered veinlets or pockets of mineralization.

The grade control holes will serve two purposes:

- Definition of the mill feed grade and contacts;
- Location of previous underground infrastructure prior to blasthole rigs drilling.

Samples collected will be sent to the assay laboratory and assayed for use in the short-range mining model. Blasthole sampling will also be part of the grade control program initially to determine the best method for Eskay Creek.

17 RECOVERY METHODS

17.1 Introduction

The plant will process material at a rate of 2.5 Mt/a with an average head grade of 3.2 g/t Au and 78 g/t Ag to produce a flotation concentrate.

The results of preliminary metallurgical test work were used to select the recovery method for the project (refer to Section 13). The resulting design criteria were used to design the process facility described in this section.

The processing plant will consist of the following areas:

- Primary crushing: a vibrating grizzly feeder and jaw crusher;
- Crushed material storage and reclaim: stockpile with two reclaimers;
- Grinding: SAG/ball mill circuit;
- Rougher flotation: rougher flotation cells;
- Regrind and cleaner flotation: fine grinding and final cleaner flotation cells;
- Concentrate dewatering and filtration: concentrate thickener and filtration;
- Concentrate load-out: storage shed to allow front-end loader filling of concentrate transportation;
- Final tailings disposal: tailings slurry pumped to TMSF.

17.2 Plant Design

The process plant design is based on a robust metallurgical flowsheet developed for optimum recovery while minimizing capital expenditure and life of mine operating costs. Conceptual design criteria for the plant are listed in Table 17-1. Comminution parameters are provided in Table 17-2. From the comminution design criteria in Table 17-2, the sizes of the mills were calculated using in-house methods. Key comminution equipment includes:

- SAG mill: 7.9 m diameter x 3.7 m effective grinding length, 3.3 MW mill motor;
- Ball mill: 6.1 m diameter x 8.8 m length, 6.0 MW mill motor;
- Pebble crusher: 132 kW installed power (HP200 or equivalent).

The majority of the flotation testwork was conducted at P_{80} of 60 µm. Grind sensitivity tests conducted at 39, 57 and 83 µm demonstrated low impact on recovery, as shown in Figure 17-1. Given the low grind sensitivity, 75 µm was selected for design.

Design criteria for the flotation plant were determined from the test work described in Section 13 of this report, and is summarised in Table 17-3.

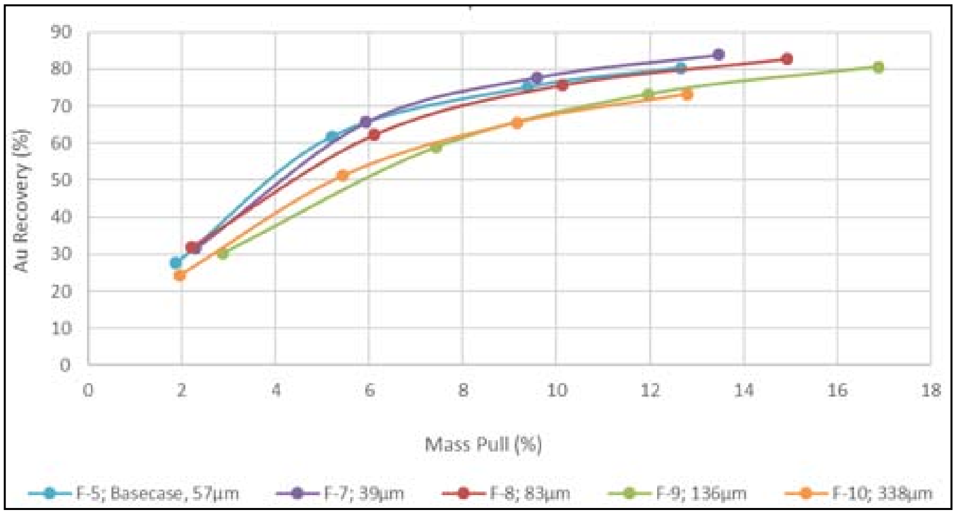
Table 17-1: Process Design Criteria – Overview

Description	Units	Value
Annual throughput	Mt/y	2.5
Crusher availability	%	70
Grinding and flotation availability	%	92
Concentrate filter availability	%	80
Crushing feed rate	t/h	408
Flotation feed rate	t/h	310
Average head grade, Au	g/t	3.2
Average head grade, Ag	g/t	78
Recovery to concentrate, mass	%	20
Recovery to concentrate, Au	%	75-92
Recovery to concentrate, Ag	%	75-97
Concentrate grade, Au	g/t	25
Concentrate grade, Ag	g/t	650

Table 17-2: Comminution Design Criteria

Description	Units	Value
JK parameters (Axb)		31.7
Specific gravity	t/m^3	2.9
Bond rod mill work index	kWh/t	21.0
Bond ball mill work index (150 mm closing screen)	kWh/t	19.4
Crushing circuit feed F_{100}	mm	800
Cyclone overflow final product P_{80}	µm	75
SAG mill discharge slurry solids	%w/w	70
SAG mill ball volume (nominal)	%	18
Ball mill discharge slurry solids	%w/w	72
Ball mill ball volume (nominal)	%	26
Pebble crusher		
Crushing rate	% new mill feed	25
Crusher P_{80}	mm	12

Figure 17-1: Grind Sensitivity – Rougher Flotation



Note: Figure prepared by Ausenco, 2019.

Table 17-3: Flotation Plant Design Criteria

Description	Units	Value
Feed rate	t/h	310
Rougher flotation		
Flotation time lab testing	min	40
Scale up factor		1
No of cells		7
Cell type		Tank cells (100 m^3)
Regrind circuit		
Specific energy	kWh/t	20
Feed size	µm	75
Product size	µm	20
Cleaner circuit		
No of stages		3
Cell type		Tank cells (20–100 m^3)
1st Cleaner flotation time	min	15
2nd Cleaner flotation time	min	15
3rd cleaner flotation time	min	10
Concentrate dewatering		
Settling rate	$t/m^2.h$	0.5
Thickener underflow density	%w/w	55
Filtration rate	$t/m^2.h$	0.3
Filter cake moisture	%w/w	Approximately 12

According to the flotation kinetic test curves, as shown in Figure 17-2, the recovery benefit diminishes beyond 15 minutes residence time in the laboratory scale unit. Subsequent tests were floated in the laboratory for 45 minutes. This is considered excessive. However, to remain consistent with the testwork an overall residence time of 40 minutes was selected without a scale-up factor. This compares well with 20 minutes residence time and a scale-up factor of 2, resulting in 40 minutes residence time. There is opportunity to reduce the residence time in the next phase of testwork.

Key flotation equipment will include:

- Rougher flotation cells – 6 x 100 m^3 tank cells;
- Concentrate regrind mill – HIG Mill 1600;
- First cleaner cells – 4 x 100 m^3 cells;
- Cleaner/scavenger cells – 4 x 70 m^3;
- Second cleaner cells – 3 x 50 m^3;
- Third cleaner cells - 3 x 20 m^3;
- Concentrate thickener – 13 m diameter;
- Concentrate filter – VPA-2040-50 or equivalent

17.1 Process Description

17.1.1 Crushing

Run-of-mine (ROM) material will be delivered to the surface primary crusher by haul trucks from the open pit mine. The ROM material will feed a dump pocket via a stationary grizzly. Grizzly oversize will be broken by a rock breaker. Grizzly undersize will be discharged to a dump hopper and fed via feeder to the primary jaw crusher and crushed to a target P80 of 107 mm.

Crusher product will be transported on a belt conveyor to a stockpile located approximately 1.5 km distant at the processing plant.

17.1.2 Grinding

The crushed mill feed will be reclaimed from the stockpile by two belt feeders at a controlled rate, and fed via a conveyor to the SAG mill.

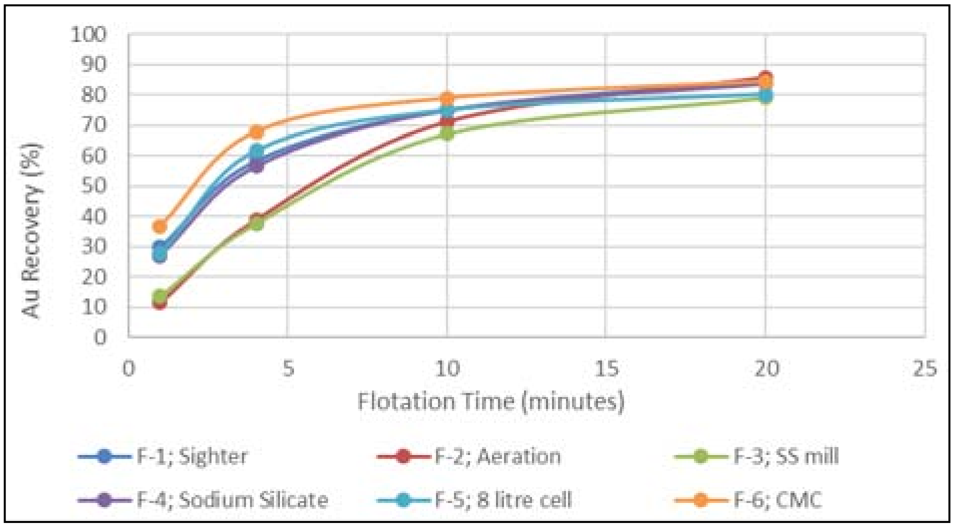
The SAG mill discharge will feed onto a vibrating screen. Any screen oversize will be returned to the circuit.

Secondary grinding will take place in a ball mill. SAG mill screen undersize will discharge into a cyclone feed pump box together with the ball mill discharge.

The combined slurry will be pumped to cyclones. The cyclone underflow will report back to the ball mill grinding circuit. The cyclone overflow will be directed to the flotation circuit.

Steel balls will be used as the grinding media.

Figure 17-2: Gold Flotation Kinetic Curves



Note: Figure prepared by Ausenco, 2019.

17.1.3 Flotation

The cyclone overflow will feed a bank of rougher flotation cells.

Rougher flotation will produce a concentrate which will be advanced to the regrind mill, complete with hydrocyclone classification.

The reground rougher concentrate will be cleaned in a bank of flotation cells. The first cleaner tailings will be scavenged before being sent to tailings.

The cleaner scavenger concentrate will be recycled to the regrind circuit.

The first cleaner concentrate will be cleaned again in a bank of flotation cells. The second cleaner tailings will return to the first cleaner flotation.

The second cleaner concentrate will be further cleaned in the third cleaner circuit. The third cleaner concentrate will be pumped to the concentrate thickener. The third cleaner tailings will return to the second cleaner.

Reagents used in the circuit will include PAX (collector), MIBC (frother) and copper sulphate (modifier), plus flocculant for thickening.

17.1.4 Concentrate Dewatering

Flotation concentrate will be thickened and further dewatered in pressure filters to a moisture content of 12%. The dewatered concentrate will discharge to a covered storage area.

Dedicated front-end loaders will load the concentrate into road tractor trailers which will be weighed prior to transportation.

17.1.5 Tailings Disposal

Rougher tailings will be pumped to the TMSF.

17.1.6 Reagents

Package plants will be provided to supply the following reagents required for the process:

- PAX;
- MIBC;
- Copper sulphate;
- Flocculant.

17.1.7 Services

Compressed air will be generated for filter, instrument and maintenance purposes. Blowers will produce low pressure air for the flotation process.

To the greatest extent possible the process plant will re-use process water recovered from the TSF to meet process plant requirements. Raw water will only be used where water quality with low dissolved solids is required and as make-up in the process water circuit.

Power will be provided from the recently-commissioned 195 MW hydroelectric facilities and linked power grid via a newly-installed high voltage overhead power line and transformer. Additional information on the proposed power supply is included in Section 18.9.

17.2 Plant Design

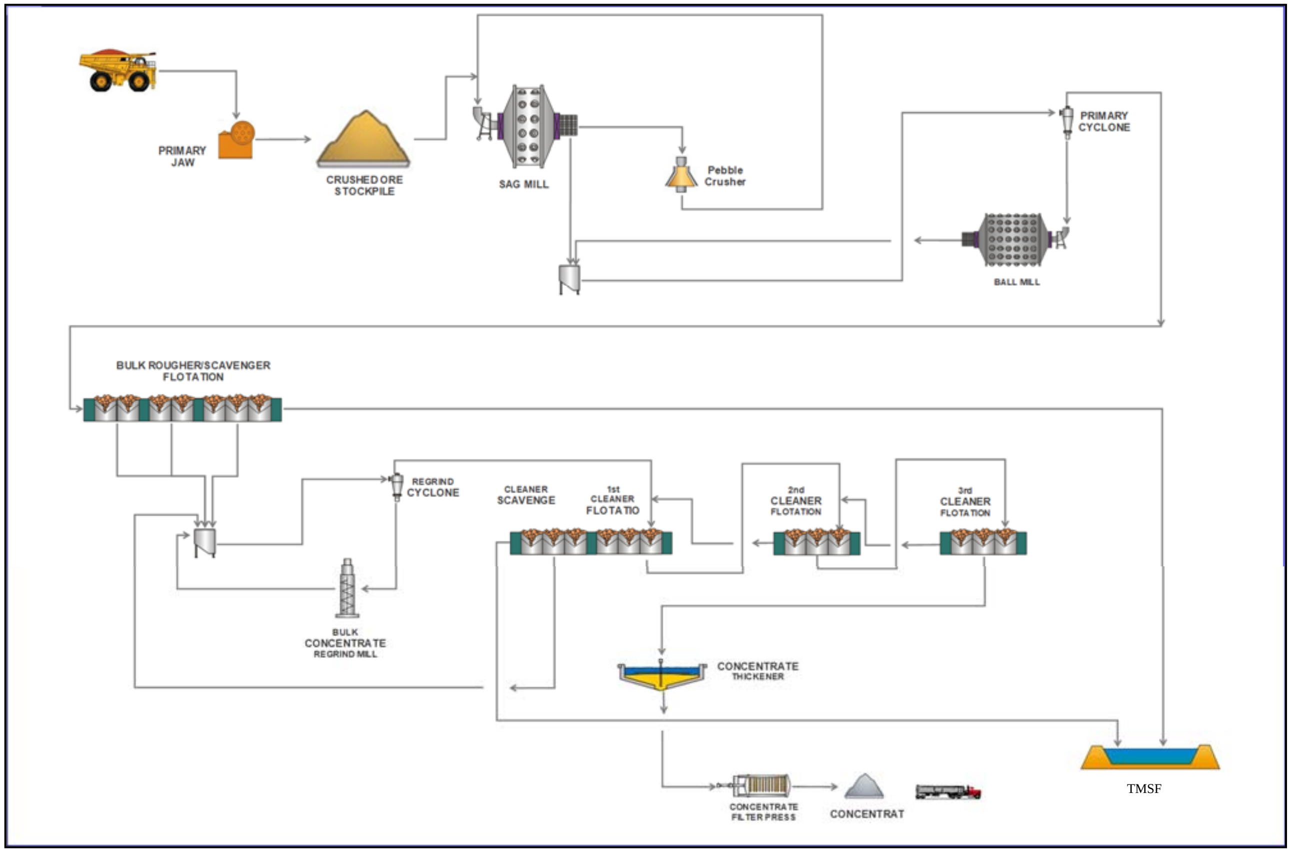
The flowsheet for the process is shown in Figure 17-3.

17.2.1 Process Control Strategy

The process control system will be a programmable logic controller (PLC) based system. The PLCs will be used to control and monitor operation of the plant and will be broken into different process areas. Each process area will be controlled by a single PLC system. The PLCs will be tied together to form a plant wide control system by the use of an ethernet communication system.

Process control and monitoring for the facility will be performed in two centralized control rooms located in the main process plant and in the primary crusher area. Human machine interface (HMI) operator stations will be located in the control rooms. These HMIs will contain graphical representation of the process equipment. The PLC in conjunction with the HMI will perform all equipment and process interlocks, level control, alarms, trends and report generation.

Figure 17-3: **Process Flowsheet**



Note: Figure prepared by Ausenco, 2019.

18 PROJECT INFRASTRUCTURE

18.1 Introduction

Infrastructure to support the Eskay Creek project will consist of site civil work, site facilities/building, a water system, and site electrical. These are indicated in Figure 18-1.

Site civil work includes designs for the following infrastructure:

- Light vehicle and heavy equipment roads;
- Conveyor corridor, 2 km long;
- Growth media stripping and stockpiling;
- Mine facility platforms and process facility platforms;
- Contact water pond;
- Tailings storage facility (TSF);
- Waste storage facilities;
- High voltage substation platform.

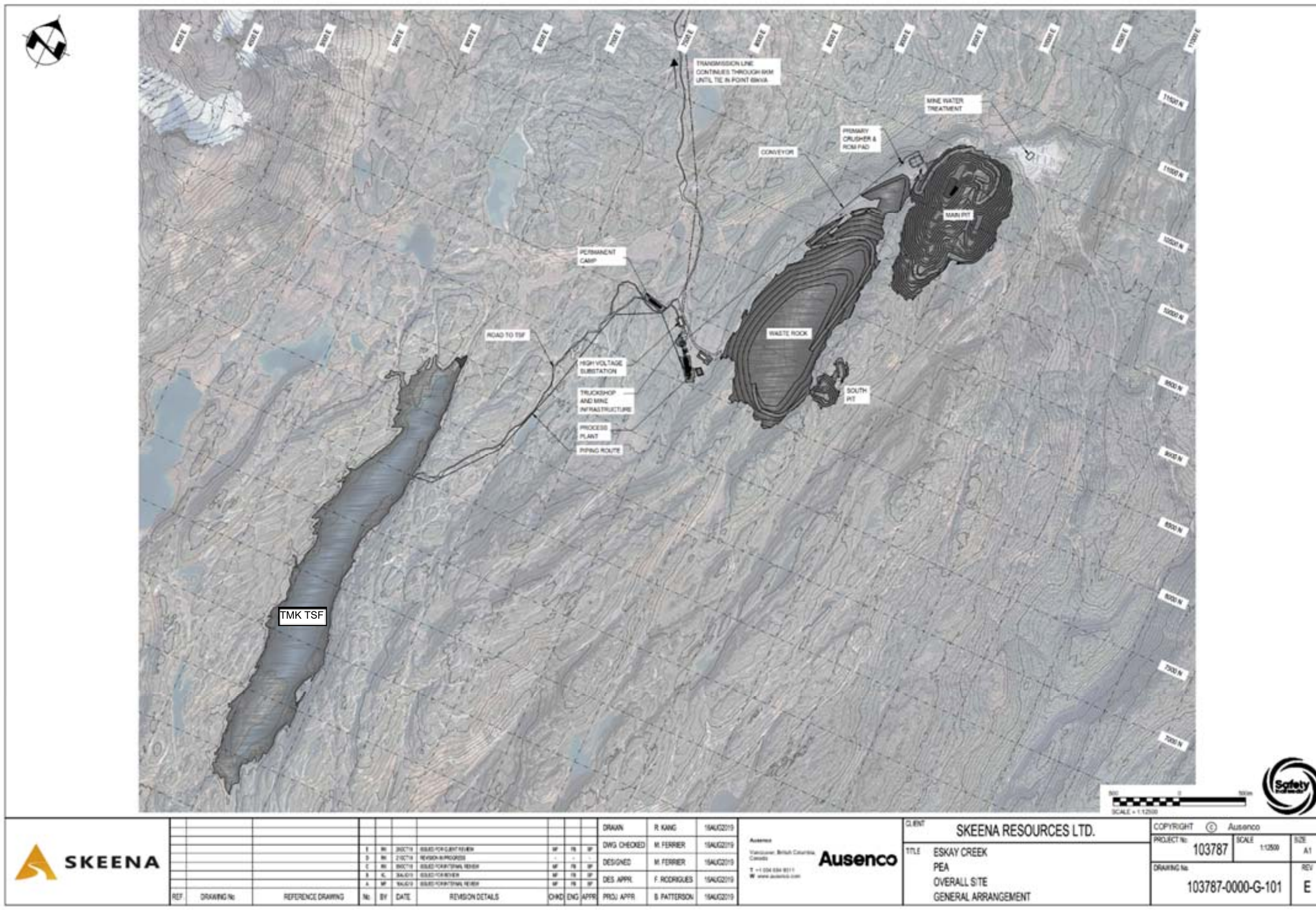
Site facilities will include both mine facilities and process facilities:

- The mine facilities will include the administration offices, truckshop and warehouse, tire repair shop, mine workshop, mine dry, fuel storage and distribution, permanent camp facility and miscellaneous facilities;
- The process facilities will include the process plant, crusher facility, process plant workshop and assay laboratory;
- Both the mine facilities and process facilities will be serviced with potable water, fire water, compressed air, power, diesel, communication, and sanitary systems.

18.2 Road and Logistics

Access to the Eskay Creek Project is via Highway 37 (Stewart Cassiar Highway). The Eskay Mine Road is an all-season gravel road that connects to Highway 37 approximately 135 km north of Meziadin Junction. Within the site, heavy equipment roads will connect the main pit and waste rock pit to the main facilities and processing areas. Secondary roads will connect the plant to the crusher, crusher to top of main pit, and around the north end of main pit for light duty traffic.

Figure 18-1: Project Proposed Layout Plan



Note: Figure prepared by Ausenco, 2019.

18.3 Stockpiles

Stockpiles are discussed in Section 16 of this Report.

18.4 Waste Storage Facilities

The planned waste storage is discussed in Section 16.9.

18.5 Tailings Storage Facilities

The TSF and associated surface water management design features were undertaken by Ausenco.

18.5.1 Historical Tailings Deposition

The Tom MacKay Lake was used by Barrick for subaqueous tailings disposal due to the potentially acid-generating (PAG) nature of the tailings produced from the Eskay Creek Mine. In 2002, the BC Government, in accordance with Schedule 2 of Section 36 of the Fisheries Act, classified the lake as a tailings impoundment area. Over 585,000 dry tonnes were deposited in the facility during the period 2001–2008.

18.5.2 Site Selection

A desktop TSF siting study was undertaken for PEA purposes. Ausenco reviewed satellite imagery and BC Government topographic maps to identify potential TSF sites within a 5 km range of the proposed plant site. The project physiography consists of a plateau bisected by two deep valleys, Harrymel Creek and Unuk Creek. The general area includes a number of small catchment basins, with the TMSF being the largest. Figure 18-2 shows the general physiographic and hydrogeological setting.

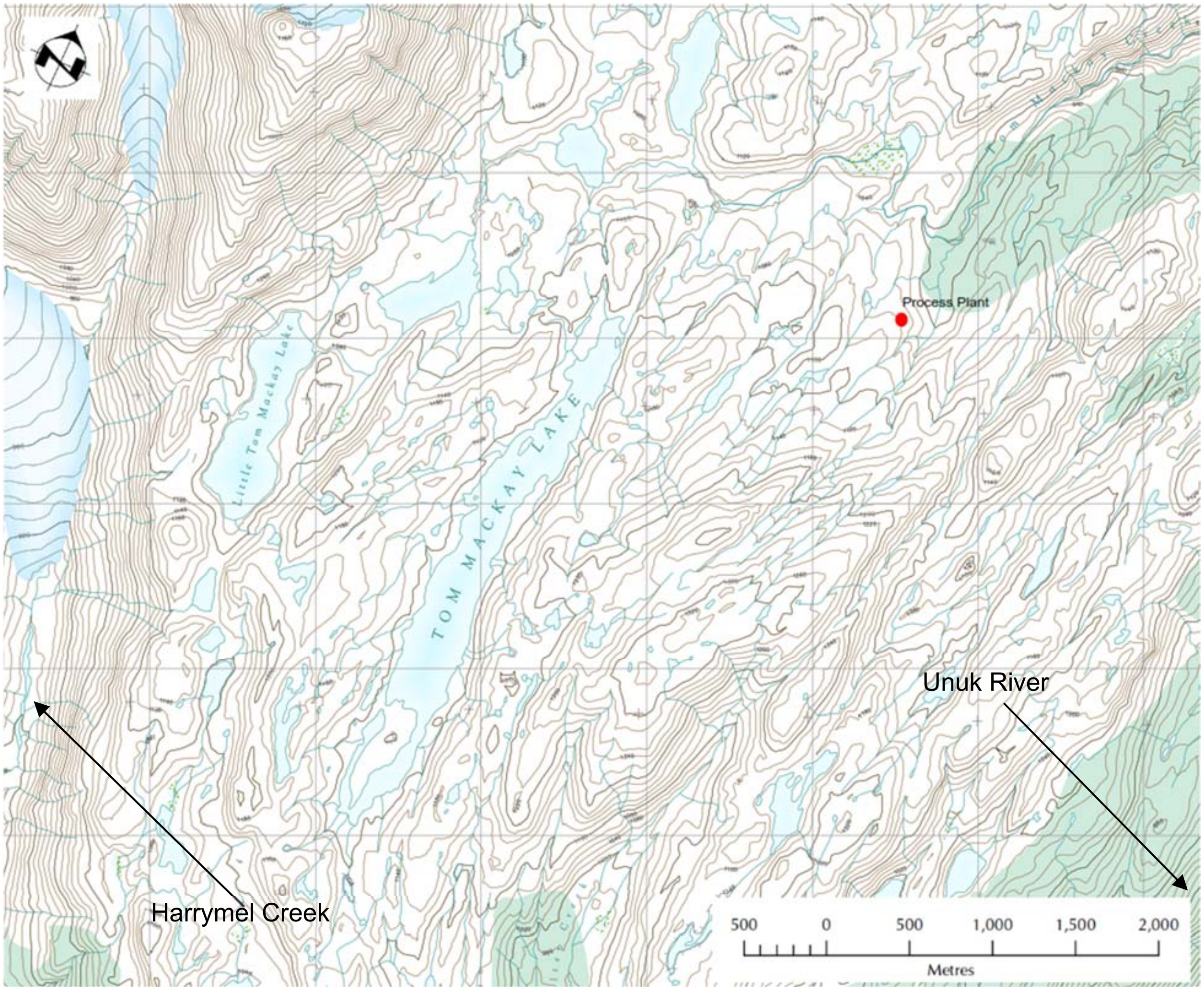
The general design criteria for the siting study considered a tailings storage requirement of 19.5 Mt deposited subaqueously while maintaining a minimum of 7 m water cover over the tailings to prevent acidification, together with ensuring the consolidated bed of tailings is not remobilized due to ice and wind/wave action.

The TMSF was selected as the preferred tailings storage option since it is permitted as a TSF and has sufficient capacity to contain 19.5 Mt of tailings. The TMSF only requires a small embankment to contain the required volume of tailings with the majority of the tailings located below the existing outlet of the TMSF.

18.5.3 Tailings Storage Facility Design Assumptions/Criteria

The proposed process plant site location in relation to the TMSF was shown in Figure 18-1. The flotation process will produce a PAG tailing stream. Sub-aqueous disposal requirement was conservatively assumed as a TSF design requirement.

Figure 18-2: Topographic and Hydrological Setting, Eskay Mine Area



Note: Topographic background sourced from GeoBC of the Ministry of Forests, Lands, and Natural Resource Operations, Map 104B068. Original topographic map at 1:20,000 scale. Process plant location shown is proposed.

Design criteria included:

- Required storage of 19.5 Mt PAG tailings;
- Dry tailings density of 1.40 t/m^3;
- Subaqueous deposition;
- Minimizing disturbance footprint through use of existing mine infrastructure;
- Limiting watershed disturbance to a single catchment basin;
- Limiting impacts to wildlife and fisheries resources;

- Designing for closure;
- Meeting or exceeding applicable regulatory requirements and industry guidelines for stability and design flood events.

The proposed TMSF design assumptions include:

- Single embankment raise;
- Rockfill dam with filter along the upstream embankment face;
- Embankment with 2:1 (H:V) upstream slope and 2.5:1 (H:V) downstream slope;
- Embankment with an upstream reinforced concrete face;
- Reinforced concrete spillway;
- Minimum freeboard of 1 m;
- Max embankment height of 27.5 m from crest to downstream toe;
- A permanent minimum water cover of 7 m over the LOM tailings mass;
- Tailing deposition system that maximizes settling time in the water column and ensures a continuous sub-aqueous environment for tailing material.

18.5.4 Tailings Storage Facility Design

The project is located in a moderate seismic zone and the embankment was designed to meet BC and Canadian Dam Association guidelines.

The TMSF is approximately 3.4 km long and 0.3 km wide, and its long axis orientation is southwest–northwest. The facility ranges in depth from 10 m at the south end to 42 m in the north–central section of the lake. The existing volume of the TMSF is around 12.9 Mm^3 at elevation 1082 masl, which is the current spill elevation of the basin.

Tailings would be slurried from the process plant to the TMSF by way of a pipeline, which would extend onto the TMSF to a floating barge. The end of the pipeline would be positioned close to the base of the TMSF to maximise settling, and minimize entrainment of fine particles to the surface of the TMSF. The minimum water depth would be 7 m to prevent both wind and ice remobilization of the tailings. The barge would move around the TMSF to develop an even tailings distribution across the TMSF floor.

Treated wastewater from the camp will be discharged into the TMSF via the tailings transportation pipeline. Pit water will be sent directly to a water treatment plant (WTP), then to D7 polishing ponds and finally to Ketchum Creek. The water treatment plant's maximum capacity is approximately 150 L/s. The overflow, i.e. the portion of flow greater than 150 L/s will be sent to the TMSF. Water for processing will come from the TMSF. The site wide water balance discusses any potential impacts on the discharge from the TMSF (refer to Section 18.6).

The tailings deposition rate is provided in Table 18-1 and the projected TMSF storage capacities are outlined in Table 18-2. Tailings are planned to be discharged at 35% solids and will have an overall dry bulk density of 1.4 t/m^3. The TMSF has sufficient capacity to store tailings without an embankment during the initial years of operations while maintaining 7 m (6–8 Mm^3) of water cover over the tailings bed. In year 4 of operations, a single embankment will be required to be constructed, so as to store the balance of the LOM tailings while maintaining 7 m of water cover.

Table 18-1: Planned Tailings Deposition Schedule

Year	Annual Tailings Production (t)	Annual Tailings Production (m^3)	Accumulated Tailings Deposition (m^3)
-2	54,000	38,000	38,000
-1	697,000	498,000	536,000
1	2,015,000	1,439,000	1,975,000
2	1,934,000	1,382,000	3,357,000
3	1,926,000	1,376,000	4,733,000
4	2,243,000	1,602,000	6,335,000
5	2,250,000	1,607,000	7,942,000
6	2,250,000	1,607,000	9,549,000
7	2,250,000	1,607,000	11,156,000
8	2,250,000	1,607,000	12,763,000
9	1,698,000	1,213,000	13,976,000

Table 18-2: TSF Projected Tailings Storage Capacity

Relative Elevation to Lake Surface (m)	Incremental Storage Volume (m^3)	Accumulated Storage Value (m^3)	Accumulated Storage Capacity (t)
0.7	1,062,000	14,006,000	19,609,000
0	3,034,000	12,944,000	18,122,000
-3	2,023,000	9,910,000	18,874,000
-6	1,744,000	7,888,000	11,043,000
-9	1,551,000	6,144,000	8,602,000
-12	1,329,000	4,594,000	6,431,000
-15	1,131,000	3,264,000	4,570,000
-18	888,000	2,133,000	2,987,000
-21	797,000	1,246,000	1,744,000
-25	220,000	448,000	627,000
-28	131,000	28,000	320,000
-31	98,000	98,000	137,000

The TMSF will also provide the water for the process plant.

18.5.5 Embankment Design

The tailings embankment will be constructed with local quarried compacted rockfill and an upstream reinforced concrete facing. The embankment will be 6 m high at the northeast end, 27 m high at the southwest end and will be 78 m in length. It will have overall slopes of 2:1 (H:V) on the upstream face and 2.5:1 (H:)V) on the downstream face. The spillway will be the new permanent discharge point from the TMSF and is designed to pass the probable maximum flood (PMF) of a 1:10,000 year event. A layout plan is included as Figure 18-3.

To maintain a baseline flow during construction and prior to the TSF filling and discharging over the spillway, a syphon or pipeline through the embankment will be installed. It is estimated that it could take 5–8 months, based on average annual discharges rate, to raise the water level to its ultimate elevation while maintaining a minimum discharge rate of 200 L/s. Therefore, a flow of 70 L/s will be discharged from the TMSF to maintain a base flow in the creek. This flow is based on historical low outflows from the TMSF.

18.5.6 Tailings Storage Facility Stability

A section through the height portion of the embankment was selected as the critical section as shown in Figure 18-4. Stability of the embankment was assessed using the limit-equilibrium modelling software Slope/W, (Geostudio, 2018) for the following scenarios:

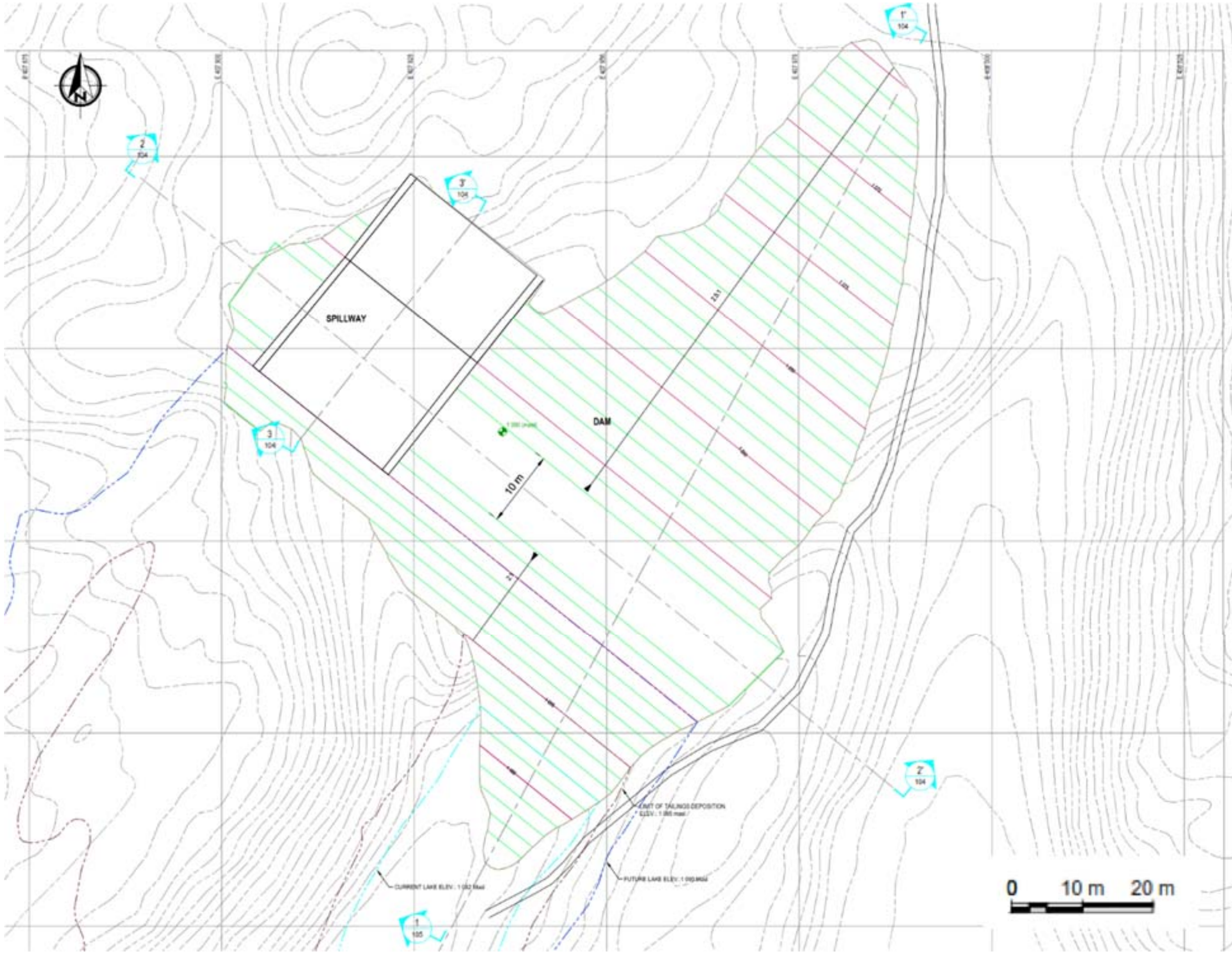
- Static: effective friction angles applied to tailings embankment; no seismic loading;
- Pseudo-static: rockfill effective friction angle with the design horizontal seismic coefficient equal to 50% of the peak ground acceleration.

Stability analysis was undertaken for both static and pseudo-static conditions with the calculated factors of safety (FOS) higher than the minimum required values in accordance with CDA guidelines. The tailings embankment is designed to withstand potential dynamic displacement without release of tailings during the maximum design earthquake event. The embankment stability analysis exceeds both static and pseudo-static CDA guidelines.

18.5.7 Tailings Storage Facility Closure

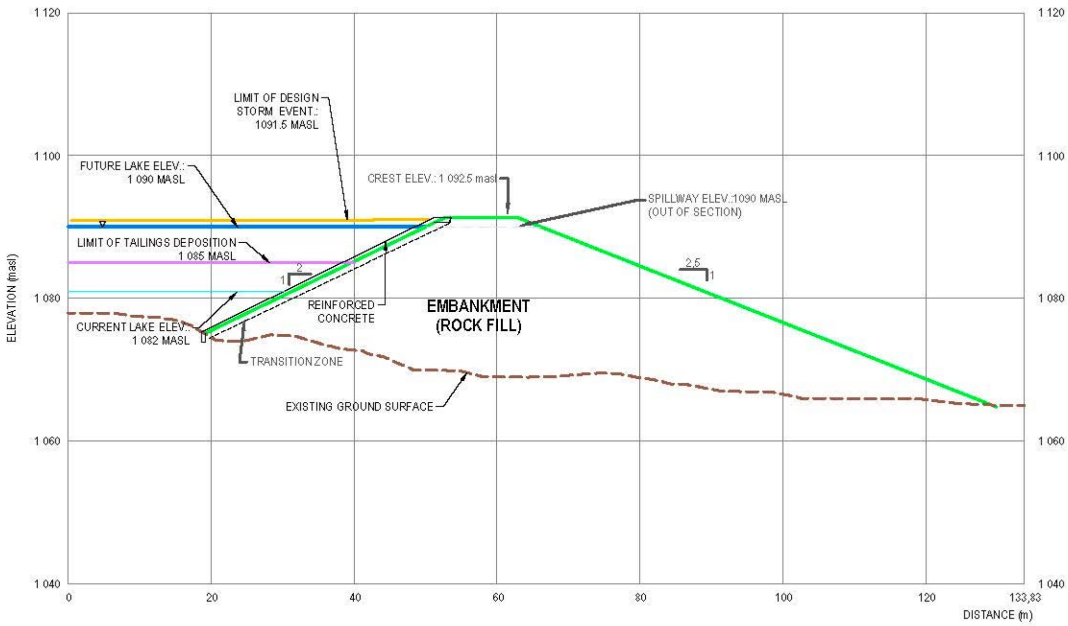
TMSF closure will consist of removing the tailings discharge line and barge, process water pipeline, the pit dewatering pipeline and the reclaim of any road not required for post closure monitoring. Since the tailings will remain subaqueous, there is no cover system planned.

Figure 18-3: Tailings Embankment Design



Note: Figure prepared by Ausenco, 2019.

Figure 18-4: Planned TMSF Cross Section with Critical Stability Section



Note: Figure prepared by Ausenco, 2019.

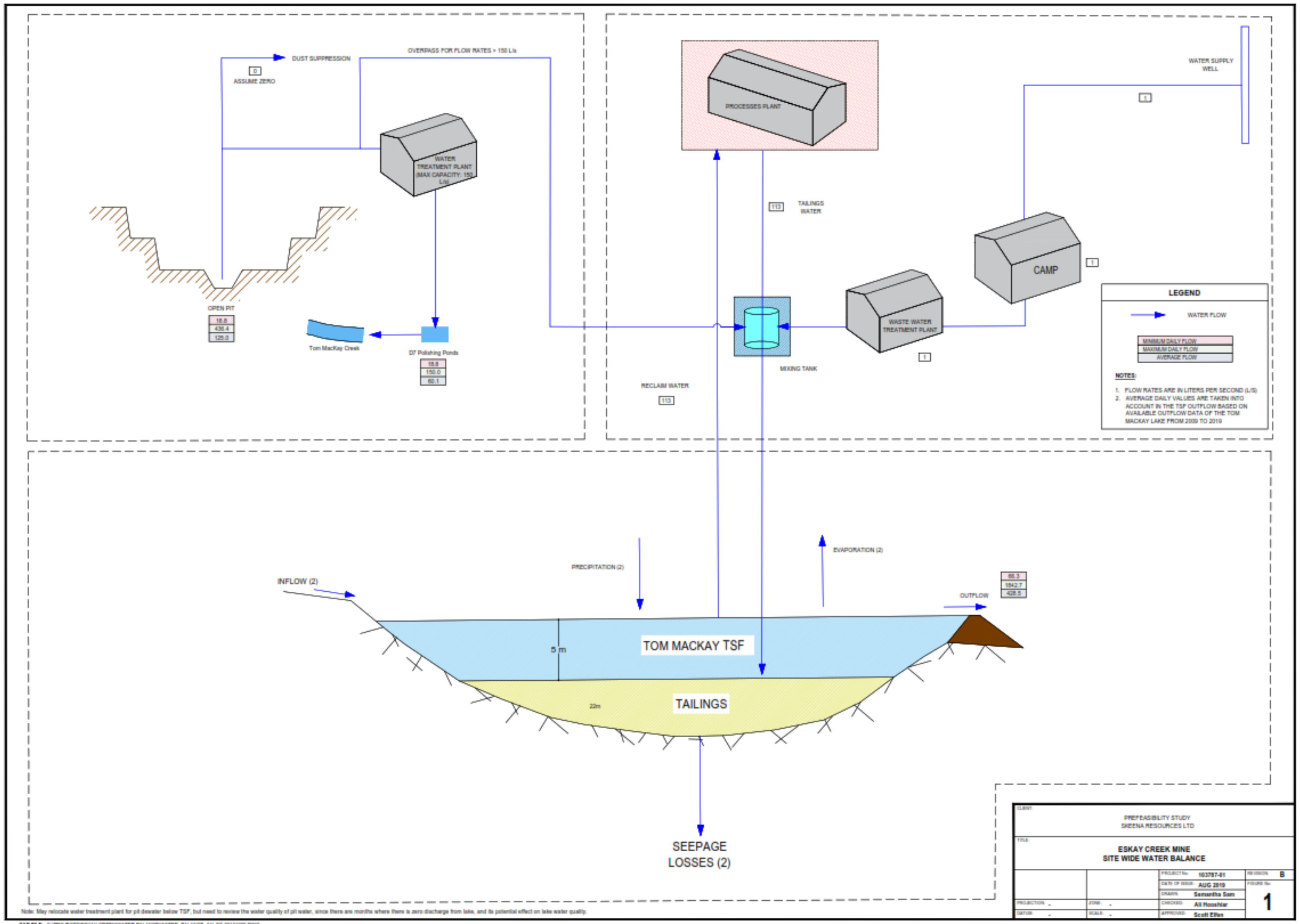
18.6 Water Balance

A projected site-wide water balance for PEA purposes is shown in Figure 18-5. The following were included:

- Tailings slurry will be discharged subaqueously into the TMSF;
- Treated camp wastewater will be discharged into the TMSF with the tailings;
- No diversion works are anticipated. There will be inflow of water into the TMSF from direct rainfall and snow and runoff from the surrounding catchment into the TMSF;
- Overflow (untreated pit water), i.e. all pit water greater than the 150 L/s processed by the water treatment plant will be discharged into the TMSF;
- Evaporation from the TMSF;
- Seepage from the TMSF;
- Discharge from the TMSF;

Water will be reclaimed from the TMSF for mineral processing.

Figure 18-5: Projected Site Water Balance



The site meteorological data compiled at the time of PEA analysis was insufficient to support a site-specific hydrometeorological model of the TMSF. However, there were several years of discharge data from the TMSF that were used to develop the site wide water balance for the project, i.e. the outflow incorporates all the TMSF inflows and losses. The makeup water requirement is a direct reflection of the impact on the outflow from the TMSF since process water requirements come from the TSF.

Pit dewater will be sent directly to the water treatment plant (WTP), then to D7 polishing ponds, and finally to Ketchum Creek. Since the water treatment plant's maximum capacity will be approximately 150 L/s, the overflow, i.e. portion of the flow greater than 150 L/s will be sent to the TMSF as illustrated in Figure 18-5. Estimated pit dewatering flow rates are all less than 150 L/s during the initial years of operations, therefore no water will be sent to the TMSF. As the open pit becomes larger toward the end of the project, pit dewatering flow rates are estimated to surpass 150 L/s between late spring and fall. During this period, the overflow portion sent to the TMSF will range from 4.5–286.4 L/s. The overflow will be pumped to the tailings mixing tank and sent with the tailings in the tailings transportation pipeline to the TMSF.

The industrial water requirements will come from the TMSF, which are estimated to be 113 L/s to be used in mineral processing. The process plant will produce a concentrate with approximately 12% moisture content. The balance of the waste (tailings) and process water will be pumped to the TMSF and discharged subaqueously. The approximate discharge of water, along with the tailings, is 114 L/s (refer to Figure 18-5.). In typical TSFs, there is a loss of water due to filling the interstitial spaces between the tailings particles. However, since the tailings are being discharged into a body of water subaqueously there is no loss of water. In addition, the project does not need to take into account evaporation or seepage from the TMSF, since it is a natural, water-retaining catchment. The tailings deposition does not significantly affect the net evaporation or seepage losses from the TMSF. For the planned operations there is almost no net loss of water from mineral processing, i.e. <1 L/s.

The camp will be supplied for all its water needs from a local well. It is estimated that the average consumption of water, based on the size of the camp, is 1 L/s. Any effluent coming from the camp will be treated and discharged into the TMSF.

The impact of the Eskay Creek Project on TMSF outflows was reviewed. The deposition of tailing displaces water in the TMSF, together with the seasonal overflow from the pit dewatering program, increases the average discharge from the TMSF by 120–240% (Figure 18-6). Therefore, the process plant and camp make-up water required based on the current mine production schedule and general operations is approximately 2 L/s.

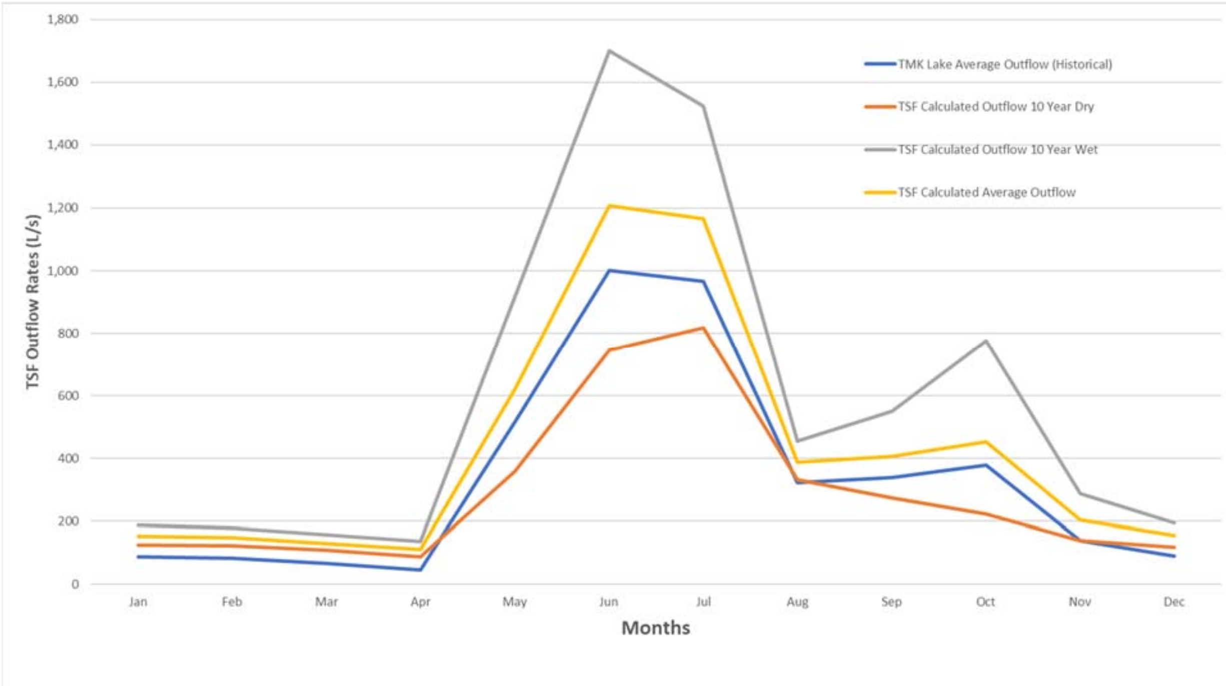
18.7 Site Infrastructure

18.7.1 Mine Facilities

The key facilities required in support of the mining operation include:

- Administration offices for the G&A staff and the Owner's mining staff;
- Truckshop, warehouse and truck wash sized for 142 t haul trucks;
- High voltage substation;
- Fuel storage and distribution;
- Miscellaneous facilities: gatehouse, ready line, hazardous waste storage pad, change pad, class III landfill, and explosives storage facility.

Figure 18-6: Monthly Discharge Scenarios, TMSF



Note: Figure prepared by Ausenco, 2019.

18.7.2 Process Facilities

The key facilities required in support of the process operation include:

- Process plant and crushing facility;
- Assay laboratory;
- Process plant workshop.

18.8 Camps and Accommodation

The permanent camp will be housed in portable modular units comprising of 200 jack-and-jill-type dormitories. In addition to the dormitories area, the camp will have a kitchen/dining area and recreation room. The camp facility will also house a portable generator, water treatment plant, sanitary septic system, propane storage, fire-water storage tank and a pump house. Parking will include an electrical bull-rail for vehicle block heater hook up.

18.9 Power and Electrical

The project power will come from the local and recently-commissioned 195 MW hydroelectric facilities and leverage on the existing power grid. The assumed required supply is 18.8 MW. A new 14 km power transmission line will tie-in to the existing transmission line and feed a high voltage substation at the project site. The tie-in point will be close to the hydroelectric facilities.

18.10 Water Supply

The proposed water supply is included in the discussion on the site water balance in Section 18.6.

18.11 QP Comments on "Item 18: Project Infrastructure"

The project as envisaged in the PEA will use conventional infrastructure and construction methods. Electrical requirements will be sourced from existing hydroelectric facilities.

19 MARKET STUDIES AND CONTRACTS

19.1 Markets

The concentrate as proposed is a complex gold concentrate with relatively low gold content and elevated levels of arsenic, mercury and antimony. Deleterious element assays are notably elevated in the first few years of mine life (arsenic in Years 1 and 2 and mercury in Years 1 to 3) before dropping to values which fall within typical industry expectations. Given the complexity of the Eskay Creek concentrate, in combination with the historical production of relatively difficult to market concentrates from the mine during its previous operational period, two independent, preliminary market studies were completed to support the payabilities used in the PEA.

Assumptions relating to concentrate quality are based on the results of ICP analysis of gold–silver concentrates produced during the batch flotation testwork at Blue Coast Metallurgy, and SRK's projections therefrom to a commercially produced concentrate. On the basis of the assumed concentrate analysis, this PEA considers that the concentrates will be sent to an Asian port for smelting and refining.

The most likely market for the concentrate is China, where the material will be imported as a gold concentrate (exceeding the minimum gold content criterion) and would therefore not be subject to arsenic import limits that would be imposed on base metal concentrate imports. The Chinese market offers the best payable terms and does not penalize mercury at the expected amounts in the Eskay Creek concentrate. The Chinese market is more than capable of absorbing the volumes under consideration and, Chinese smelters are expected to actively be looking for feedstock in future to keep their utilisation high. This, allied with an expected shortage of concentrates in the future, should provide a ready market for Eskay Creek material.

It is notable that Asian smelters were not the only options for treatment of the Eskay Creek concentrate. Other smelters around the world such as the Horne smelter in Canada may also be interested in purchasing some of the concentrate, although the mercury levels could be a challenge.

Based on the predicted analysis, the Eskay Creek concentrates will be readily saleable. The relatively high levels of deleterious elements, particularly mercury in the initial years of operation, may require that concentrate sales be spread across a number of buyers as individual smelters are likely to need to blend small volumes of concentrate with cleaner concentrates to remain within acceptable effluent limits. An alternative option could be to sell the concentrate to traders who may be able to buy it all and spread distribution across a range of end customers. Expectations of payability vary but if the concentrate can be spread across enough buyers then favourable payabilities may be achieved and penalties for deleterious elements may be minimised.

19.2 Contracts

No contracts have been entered into at the Report effective date for mining, concentrating, smelting, refining, transportation, handling, sales and hedging, and forward sales contracts or arrangements. It is expected that the sale of concentrate will include a mixture of long-term and spot contracts.

Most concentrate is traded on the basis of term contracts. These frequently run for terms of one to 10 years, although many long-term contracts are treated as evergreen arrangements which continue indefinitely with periodic renegotiation of key terms and conditions. Generally, a term contract is a frame agreement under which a specified tonnage of material is shipped from mine to smelter, with charges renegotiated at regular intervals (typically annually).

Spot contracts are normally a one-off sale of a specific quantity of concentrate with a merchant or smelter. The material is paid for in much the same way as a concentrate shipped under a term contract. Merchant business is a mixture of one-off contracts with smelters and long-term contracts with both miners and smelters.

Often terms of sale for a term contract between miners and smelters are at "benchmark terms", which is the consensus annual terms for the sale of concentrate, and negotiated annually. Spot sales are made at spot terms, negotiated on a contract by contract basis.

19.3 Smelter Term Assumptions

The contract terms for the gold–silver concentrate are generally anticipated to include the following payment terms:

- Gold: payable value of gold is at 95% with a threshold of 1 g/t Au;
- Silver: payable value of silver is at 80% with a threshold of 50 g/t Ag;
- Antimony: no payable value has been attributed to the antimony present in the concentrate;
- Penalty elements:
  - A penalty of US$15 per 1% As is applied at As contents in excess of 0.5%;
  - A penalty of US$8 per 100 ppm Hg is applied at Hg contents in excess of 500 ppm;
- Treatment charge: a flat treatment charge of US$180/dmt is applied;
- Refining charge: a flat refining charge of US$15/oz Au and US$1.0/oz Ag is applied.

19.4 Transportation and Logistics

Transportation cost assumptions for the concentrate are summarized as follows:

- Eskay Creek mine to Port of Stewart: C$38/wmt;
- Port terminal and handling: C$25/wmt;
- Ocean freight to Asian port: C$55/wmt;
- Total transport costs: C$118/wmt.

19.5 Insurance, Representation and Marketing

No allowance in has been made for insurance, marketing or representation

19.6 Metal Prices

Ausenco and Skeena established metal price projections for use in the PEA, on which this Report is based. The projections incorporate consideration of recent metal market information, in combination with two-year trailing actual metal prices, and bank analyst forward price projections.

The resulting long-term consensus metal price assumptions used in the PEA are:

- Gold: US$1,325/oz Au;
- Silver: US$16.00/oz Ag.

An exchange rate of 0.77:1 US$:C$ was used.

19.7 QP Comments on "Item 19: Market Studies and Contracts"

The QP is of the opinion that the marketing and commodity price information is suitable to be used in cashflow analyses to support the PEA.

20 ENVIRONMENTAL STUDIES, PERMITTING, AND SOCIAL OR COMMUNITY IMPACT

20.1 Environmental Setting

Information on the project climate and physiographic setting is included in Section 5.

20.1.1 Vegetation

The Project area is represented by five biogeoclimatic zones:

- Alpine tundra: occurs throughout the TMSF area. Essentially treeless conditions. Vegetation is dominated by heather, lichens, mosses, sedges, and hardy alpine flowers. Much of this area is interspersed with rock and standing water;
- Engelmann spruce–subalpine fir: occurs in the planned mine site area and mid-Tom MacKay Creek, lower Argillite Creek, and mid-upper Eskay Creek. Includes continuous forest cover at its lower and middle elevations and subalpine parkland near its upper limits. Engelmann spruce dominates the canopy of mature stands, while subalpine fir is most abundant in the understorey (Meidinger and Pojar, 1991);
- Mountain hemlock: occurs in subalpine areas west and southwest of the mine site area below the alpine tundra zone. The major tree species include mountain hemlock, subalpine fir with Sitka spruce, and western hemlock occurring at lower elevations (Hallam Knight Piésold Ltd, 1993);
- Coastal western hemlock: low-elevation landscapes along Unuk River near the outlets of both Eskay Creek and Ketchum Creek. Tree species include western hemlock, Sitka spruce, black cottonwood, subalpine fir, and a hybrid of white spruce and Sitka spruce known as Roche spruce (Hallam Knight Piésold Ltd, 1993);
- Interior cedar hemlock: valley bottoms and low-elevation uplands along Iskut River and Forest Kerr Creek. Vegetation is dominated by black cottonwood with Sitka spruce and birch present in lesser numbers (Hallam Knight Piésold Ltd, 1993).

20.1.2 Wildlife

Large wildlife species recorded within the project area include black bear and mountain goat. Small mammals present in the project vicinity include American marten, wolverine, voles, and the hoary marmot.

Furbearing mammals with suitable habitat in the project area include grizzly bear, wolf, lynx, ermine, mink, fisher, least weasel, and snowshoe (Hallam Knight Piésold Ltd, 1993).

Biophysical inventory mapping identified the project area as potentially suitable to support woodland caribou and moose (MOE, 1982). However, the project area is not overlapped by any caribou herd ranges shown on provincial range mapping (BC, 2019).

Mid and lower elevations provide habitat for porcupine, northern flying squirrel and red squirrel. Plovers, Canada goose, harlequin duck, and numerous passerine species have been recorded in the area. Raptors recorded in the area include bald eagle, sharp-shinned hawk and owls.

Four species of amphibian and one reptile species are known to inhabit the project area. They include the common garter snake, long-toed salamander, western toad, wood frog and rough-skinned newt. Wood frogs are the only amphibian recorded near the project area (Hallam Knight Piésold Ltd, 1993).

20.1.3 Fisheries and Aquatic Resources

Fisheries resources of rivers, lakes and their tributaries potentially affected by the Eskay Creek mine, were assessed from 1982 to 1993 (Hemmera, 2000). No fish have been observed or captured in the upper tributaries of the Unuk River in the vicinity of the Project, in headwater lakes including Albino Lake, Little Tom Mackay Lake, and the TMSF, nor in the Eskay Creek and Tom Mackay Creek downstream of the former mine. The high-alpine, natural lakes and streams in the Tom MacKay watershed, including Albino Lake and TMSF, are naturally low in plant nutrients and barren of fish due to impassible waterfalls as well as gradient/velocity barriers to approximately 10 km downstream of the former Eskay Creek mine site (McGurk et al., 2006; Hallam Knight Piesold Ltd, 1993).

A series of obstacles to fish passage are also located immediately upstream of the confluence of Tom MacKay Creek with Ketchum Creek. Pink, chum, chinook, and sockeye salmon, Dolly Varden, and cutthroat trout, were observed 7–8 km downstream of the planned mine site in the Unuk River (Hemmera, 1997).

20.1.4 Environmental Studies

Several environmental studies were completed at the Eskay Creek mine under various owners. A limited number of reports were available for review, the key reports reviewed are discussed in this sub-section. The environmental baseline data were mostly collected between 1990 and 1993 by Hallam Knight and Piésold for Prime Resources Ltd to support their application for a Mine Development Certificate. Updates were made in 1997 to support the proposed mill expansion (Hemmera, 1997), and again in 2000 to amend the environmental assessment (EA) to deposit tailings and waste rock in the TMSF (Hemmera, 2000).

Due to the age of the baseline assessment, additional environmental, social, economic, heritage, and health studies are expected be required to update the baseline to current environmental conditions to address refinement of the project design and reflect current regulatory requirements in support of provincial and federal EA submissions.

20.2 Environmental Management

20.2.1 Historical Waste Disposal Activities

Waste was stored underwater at the permitted Albino SF. In late 1997, the processing plant was permitted and began operations. The filtered tailings generated from the mill were initially trucked to the Albino SF along with the waste rock (Barrick, 2014a).

From September 2001 to the end of operations, slurry tailings were discharged into TMSF via a dedicated pipeline while waste rock continued to be stored in the Albino SF (Barrick, 2014b). A small percentage of slurry tailings were trucked to the Albino SF during maintenance or other events that restricted normal pipeline discharge to the TMSF.

Significant reclamation activities started in 2007; activities included removal of surface buildings including the mill, removal of concrete pads and decommissioning of the tailings pipeline. Details of the reclamation activities undertaken to date are included in annual reclamation reporting

(Barrick, 2017). The mine became a recognized closed mine by the provincial regulators in 2011 (Barrick, 2017).

20.2.2 Waste Management – Waste Rock and Tailings Disposal

The main waste management issue for the project is the prevention and control of metal leaching/acid rock drainage (ML/ARD) from the tailings, and any acid generating or PAG waste rock that is produced during mine development or operations.

The project will create waste rock from mine development and tailings as a byproduct of mineral processing. The waste streams will be managed on site as follows:

- Non acid-generating (NAG) waste rock will be deposited in two locations: approximately 90% will be stored in the WD-01 facility that will be located to the west of the main pit. The remaining 10% of the total waste rock will be backfilled in the north pit;
- PAG waste rock, if encountered, will be deposited in the WD-01 facility and effluent managed to reduce any environmental impacts;
- Tailings will be deposited sub-aqueously in the TMSF with a minimum of 7 m water cover (refer to discussion in Section 18.5). The TMSF is already permitted for tailings disposal.

To quantify the ML/ARD potential a kinetic testing program for mine waste rock was initiated in August 2006. Additional kinetic testing was undertaken in January 2008 for waste rock samples from the Albino SF which accepted both waste rock and tailings. The objectives of this program were to determine the subaerial and subaqueous weathering characteristics of rocks present at the site, including rates of sulphide oxidation, neutralization potential consumption, and metal release to help estimate the time to sulphide and neutralization potential exhaustion, future geochemical conditions of the rock, and prediction of mine water quality. Selection of mine rock samples for testing was based upon a review of acid base accounting and metal analyses for more than 1,000 samples from Eskay Creek mine that were collected during operations.

Mine rock samples collected from the underground workings and waste rock from the Albino SF were subjected to kinetic testing that consisted of:

- Humidity cell testing;
- Subaqueous column testing;
- Column testing for two backfill samples.

The drainage from the Albino SF and the TMSF has not shown any significant ML/ARD-related changes in water quality as a result of the historic tailings and waste rock deposition. The drainage from these facilities have water quality within the applicable permit limits.

To manage the potential for ML/ARD, Skeena has incorporated appropriate design features and mitigation measures in the project that are consistent with best practices for waste and water management to address these issues, including:

- WRSF seepage collection systems;
- Water treatment plant;
- Subaqueous disposal of all tailings.

20.2.2.1 Non-Hazardous Waste

Non-hazardous waste management will involve the segregation of industrial and domestic waste into appropriate management streams. Project-related waste collection and disposal facilities will include one or more incinerators for domestic/putrescible waste, separate waste collection areas for recyclable and industrial waste streams for off-site disposal, and sewage effluent and sludge disposal for onsite disposal. Waste collection areas will be managed following regulatory requirements and best management practices for the safety of workers and environment, including standard operating procedures for spill management, fire safety and wildlife attractant.

20.2.2.2 Hazardous Waste

Hazardous waste materials such as spoiled reagents, waste petroleum products and used batteries will be generated throughout the life of the project, from construction to decommissioning. Storage facilities will facilitate the segregation and inventory of the various hazardous waste streams generated during the project. A separate secure storage area will be established with appropriate controls and best management practices to maintain the safety of workers and the environment. Hazardous materials will be labelled and stored in appropriate containers for shipment to approved off-site disposal facilities. Waste streams will be tracked in accordance with federal and provincial regulations, such as the federal *Transportation of Dangerous Goods Act, 1992* (SC 1992, c 34).

20.2.3 Water Management

Mine water can be divided into two categories depending on the potential for contamination:

- Non-contact water from upstream catchments that has not been in contact with mine workings will be kept separate from water that has been in contact with mine workings and discharged to the environment with no treatment;
- Contact water that has been in contact with potential sources of contamination; includes seepage from the WRSF, process water, and pit dewatering. Contact water from the WRSF will be collected and sent to a water treatment plant for treatment prior to discharge if testing shows any onset of ML/ARD. If contact water quality from the WRSF is within permitted parameter limits, and is confirmed with regular testing, this water will be discharged without treatment. Water from pit dewatering will be pumped to a water treatment plant for treatment prior to discharge to the existing mine water polishing ponds and ultimate discharge through permitted effluent discharge point D7 (identification number E219595) to Ketchum Creek. Process water will be discharged to the TMSF.

A site-wide water balance is included in Section 18.6. Strategies for water management include collecting surface water from disturbed areas (mine-contact) to:

- Manage surface water erosion;
- Recycle mine-contact water whenever possible;
- Treat mine-contact water as required, and;
- Monitor water quality to meet discharge standards prior to discharge.

20.3 Site Management and Monitoring

The project will be designed, constructed, operated, and decommissioned to meet all applicable BC environmental and safety standards and practices. Some of the provincial legislation that establishes or enables these standards are:

- *Mines Act*;
- *Land Act*;
- *Environmental Management Act*;
- *Forest Act*;
- *Forest and Range Practices Act*;
- *Water Sustainability Act*.

Skeena will develop and implement an Environmental Management System (EMS) that defines the processes by which compliance will be met and demonstrated. The EMS will include ongoing monitoring and reporting to relevant parties at the various project stages.

Site water management will be a critical component of project design. The most likely avenue for transport of contaminants into the natural environment will be through surface, groundwater, and dust. Skeena will develop a Water Management Plan and Dust Control Management Plan that applies to all activities undertaken during all project phases.

20.4 Closure Plan

A Closure and Reclamation Plan will be developed as part of the EA and refined for the permitting process. In summary, the mine closure concept is to meet water quality objectives without ongoing treatment for ARD. This will be achieved by placing all of the PAG waste rock into designated WRSF with appropriate controls in place. A graded earthfill/rockfill cover will be constructed on top of the WRSF, and revegetated to facilitate runoff and minimize infiltration. The surface workings will be progressively backfilled with waste rock throughout mine operations as far as is practicable, without hindering mine operations.

The structures on the site will be decommissioned and removed from the site upon completion of mining. Explosives, explosive magazines, fuel, and storage facilities will be removed from the site at closure. Concrete slabs, footings, and broken and placed appropriately to meet project closure and reclamation objectives. Process buildings, camp facilities, pipelines, conveyor systems, and equipment, will be removed from site or appropriately landfilled in an approved facility. In addition to removing bridges from the mine roads, Skeena will remove all culverts and install cross-ditches for drainage. The mine site access road will not be deactivated, as it will be required to allow access for continued reclamation activities and monitoring. Similarly, the transmission line to the site will be maintained to continue power supply required for reclamation, closure and monitoring activities. Compacted surfaces including laydowns and roads will be decompacted and revegetated.

Closure planning will include dialogue with Indigenous groups and stakeholders to determine post mining land use objectives and necessary investigations required to achieve and monitor those objectives.

The estimated closure and reclamation costs are included in the economic analysis in Section 22.

20.5 Permitting

20.5.1 Environmental Approvals

Major mining projects in BC are subject to EA and review prior to certification and issuance of permits to authorize construction and operations. The EA process is a means of addressing the potential for adverse environmental, social, economic, health, and heritage effects or the potential adverse effects on Indigenous interests or rights prior to project approval.

At a provincial level, proposed mining developments that exceed any of the thresholds specified in the Reviewable Projects Regulation (BC Reg. 370/2002), are required under the BCEAA to obtain an EA Certificate (EAC) before the issuance of any permits to construct or operate. The project will require a provincial EAC.

At a federal level, proposed mining developments that exceed any of the thresholds specified in the Regulations Designating Physical Activities (SOR/2012-147), are required under the Impact Assessment Agency of Canada's (IAAC) *Impact Assessment Act* (IAA) to obtain a federal decision statement before the issuance of any permits to construct or operate. The project will require a federal decision statement.

Skeena has not filed a federal or provincial EA application. Once an application is filed, the BC Environmental Assessment Office (EAO) and IAAC will issue their decision for the project. Once the project has a provincial EAC and a federal decision statement, Skeena can apply for the necessary statutory permits and authorizations to commence project construction.

No technical or policy issues are anticipated for obtaining the required project permits and approvals, given its long mining history.

20.5.2 Anticipated Provincial Permits and Authorizations

No permits for project commercial development will be issued before an EAC is obtained. Consequently, Skeena will apply for synchronous permitting within the environmental review process for all permits. Synchronous permitting will expedite the permitting process and reduce the time to start construction.

Table 20-1 presents a preliminary list of the key provincial authorizations, licences, and permits that will be required to develop the project. The list includes only the major permits and is not intended to be comprehensive.

20.5.3 Anticipated Federal Approvals and Authorizations

Table 20-2 presents a preliminary list of the key federal authorizations, licences, and permits required for project development. The list includes only the major permits and is not intended to be comprehensive.

Table 20-1: Preliminary List of Provincial Authorizations Likely Required for the Project

Authorization	Responsible Agency	Legislation	Purpose
EA Certificate	Environmental Assessment Agency	Environmental Assessment Act	Minimize or avoid adverse environmental, heritage, health, social, and economic effects and incorporate environmental factors and Indigenous and stakeholder consultation into decision-making.
Mines Act Permit	BC Ministry of Energy, Mines and Petroleum Resources	*Mines Act* and Health, Safety and Reclamation Code for Mines in BC	Authorizes development including fuel storage, operations, closure, and reclamation and abandonment.
Camp operating permit	Northern Health	*Drinking Water Protection Act* *Public Health Act* Municipal Wastewater Regulation	Issues a camp operations permit for sewage disposal, drinking water supply, and food handling.
Environmental Management Act (Effluent) Permit	BC Ministry of Environment and Climate Change Strategy	*Environmental Management Act*	Authorizes discharge of liquid effluent to the environment.
Environmental Management Act (Air) Permit	BC Ministry of Environment and Climate Change Strategy	*Environmental Management Act*	Authorizes discharge of airborne emissions to the environment.
Hazardous Waste Registration	BC Ministry of Environment and Climate Change Strategy	*Environmental Management Act* Hazardous Waste Regulation	Authorizes temporary storage of hazardous waste.
Water License	BC Ministry of Environment and Climate Change Strategy	*Water Sustainability Act*	Authorizes storage, use or diversion of surface water, including installation of works.
Approval for Works in and about a Stream (Section 11)	BC Ministry of Environment and Climate Change Strategy	*Water Sustainability Act*	Provides approval to work in and about a stream (i.e., stream crossings).
Investigation or Inspection Permit	BC Ministry of Forests, Lands and Natural Resource Operations and Rural Development	*Heritage Conservation Act*	Authorizes investigation through an archaeological impact assessment or mitigation of impacts to sites (should any be identified) through systematic data recovery after an impact assessment has been completed.
Site Alteration Permit	BC Ministry of Forests, Lands and Natural Resource Operations and Rural Development	*Heritage Conservation Act*	Authorizes alteration or removal of site (should any be identified and impacted by the Project.)
Occupant License to Cut	BC Ministry of Forests, Lands and Natural Resource Operations and Rural Development	*Forest and Range Practices Act*	Authorizes cutting and removal of trees of merchantable size.
Road Use Permit	BC Ministry of Forests, Lands, and Natural Resource Operations and Rural Development	*Forest and Range Practices Act*	Authorizes use of a Forest Service Road.

Authorization	Responsible Agency	Legislation	Purpose
Special Use Permit	BC Ministry of Forests, Lands, and Natural Resource Operations and Rural Development	*Forest and Range Practices Act*	Authorizes the construction of and use of a new road. Authorizes occupation of Crown land.
License of Occupation	BC Ministry of Forests, Lands, and Natural Resource Operations and Rural Development	*Land Act*	Authorizes occupation of Crown land including temporary borrow and gravel pits, construction staging areas, and for remote areas where precise tenure boundaries are not required.

Table 20-2: Preliminary List of Federal Authorizations Likely Required for the Project

Authorization	Responsible Agency	Legislation	Purpose
Federal Decision Statement	Impact Assessment Agency of Canada	*Impact Assessment Act*	Minimize or avoid adverse environmental, heritage, health, social, and economic effects and incorporate environmental factors and Indigenous and stakeholder consultation into decision-making.
Permit	Natural Resources Canada	*Explosives Act*	Explosives authorizations are required during construction and operations. Authorization is required to manufacture and operate an explosives storage facility. Licenses are required by either the company or a blasting contractor.
Authorization	Fisheries and Oceans Canada	*Fisheries Act*	May require authorization(s) if the Project causes serious harm to fish or fish habitat (e.g., watercourse crossings and clearing riparian vegetation).
Permit	Environment Canada	*Migratory Birds Convention Act*	May require a permit if the Project is shown to affect nesting habitats used by migratory birds or if activities occur during the nesting season (e.g., clearing of vegetation, disturbance to nests).
Permit	Environment Canada	*Species at Risk Act*	Permits may be required if the Project has the potential to affect a species listed on Schedule 1 of the Act, including any part of its critical habitat, or the residences of its individuals.
Permit	Transport Canada	*Navigation Protection Act*	May require authorization(s) if the Project activities includes works built in, on, over, under, through, or across any navigable water that may interfere with navigation.
Authorization	Environment Canada	*Nuclear Safety and Control Act*	A license is required for possession of instruments containing radioactive material, such as nuclear density gauges (portable and fixed).
Authorization	Environment Canada	*Fisheries Act*	Metal and Diamond Mining Effluent Regulations are intended to reduce threats to fish and their habitat by improving the management of harmful substances in metal and diamond mining effluent.
License	Environment Canada	*Radio Communication Act*	A license is required for use of radio equipment on site.
Permits	Environment Canada	*Transportation of Dangerous Goods Act*	Transportation and handling of dangerous goods as described by the regulation.

20.6 Considerations of Social and Community Impacts

20.6.1 Social Setting

Northwestern BC is a sparsely populated and relatively undeveloped region of the province. Many of the smaller communities have predominantly Indigenous populations that are isolated from one another as well as from the main regional centres of Smithers and Terrace. Approximately one-third of the 40,000 to 45,000 people in the region are Indigenous, which is higher than the provincial average (MSBED, 2005).

Mining and forestry are the main sources of income. Forestry is in decline. Since the mid-1990s, the regional population has dropped by almost 15% although in recent years, the rate of decline has begun to slow (MSBED, 2005).

Community and socio-economic impacts of the project can potentially be very favourable for the region, as new long-term opportunities are created for local and regional workers. Such opportunities could reduce and possibly reverse the out-migration to larger centres.

Estimates made in 1993 expected the historic Eskay Creek mine to generate approximate 2,250 person years of direct and indirect employment for BC residents; approximately 50% of these would be for residents of northern BC (Hallam Knight Piésold Ltd, 1993). These estimates are an indicator of the potential employment benefits to local communities in the project area.

The region is intersected north to south by Highway 37 (refer to Figure 2-1). Highway 37 communities include Iskut, Dease Lake, and Good Hope Lake. With the exception of Stewart, the majority of the population are of First Nation descent. These communities rely heavily on public sector and mining employment. Since 1996, Highway 37 communities have experienced an overall decline in population (MSBED, 2005).

20.6.2 Engagement and Consultation

20.6.2.1 Consultation Policy Requirements

Both the BC *Environmental Assessment Act* (BCEAA) and the federal *Impact Assessment Act* (IAA) 2019 contain provisions for consultation with First Nations, and the public as a component of the EA process. Future engagement and consultation measures will comply with federal and provincial regulations, best practices, and Skeena's internal company policies.

20.6.2.2 First Nations

Skeena will be required to consult with local First Nations as part of the EA process, as identified by the provincial government's Section 11 Order, and as indicated in the federal government's EA guidelines when they are issued for the project. The project is located in the southernmost portion of the Tahltan Nation territory; the Tahltan Tribal Council (now the Tahltan Central Government) has formerly laid a comprehensive claim to the area containing the Eskay Creek mine (Rousseau, 1990). Previous developments for the expansion of the mill in 1997 included consultation with the Iskut Band, Telegraph Creek Band, and the Tahltan Central Government and Band Council. The relationship between the previous owners and the Tahltan have been favourable: the mine provided employment and business opportunities, and in return the local community provided a stable and capable local work force. In 1997, 57 of the 150 employees were Tahltan (Hemmera, 1997). Ongoing consultation efforts will aim to engage both community leaders and members, and attempt to resolve potential issues and concerns as they arise.

20.6.2.3 Government

Skeena will engage and collaborate with federal, provincial, regional, and municipal government agencies and representatives as required with respect to topics such as land and resource management, protected areas, official community plans, environmental and social baseline studies, and effects assessments. Skeena will be required to form a project specific working group at the early stages of the EA process, which will include representatives from many government groups. Skeena will be required to consult with the working group on project-related developments during the EA process.

20.6.2.4 Public and Stakeholders

Skeena will consult with the public and relevant stakeholder groups, including land tenure holders, businesses, economic development organizations, businesses and contractors (e.g., suppliers and service providers), and special interest groups (e.g. environmental, labour, social, health, and recreation groups), as appropriate.

21 CAPITAL AND OPERATING COSTS

21.1 Introduction

The following basic information pertains to the estimate:

- Base date is Q3, 2019;
- Expressed in Canadian dollars (C$);
- Currency exchange rate US$0.77:C$1.00;
- Accuracy is ±50%.

The purpose of the capital estimate is to provide substantiated costs which can be used to assess the preliminary economics of the project. The cost estimate is based on an engineering, procurement and construction management (EPCM) implementation approach.

21.2 Capital Cost Estimates

21.2.1 Summary

Table 21-1 summarizes the capital cost estimate.

21.2.2 Mine Capital Costs

The mining capital cost estimate is grouped into three main categories:

- Pre-production stripping costs;
- Mining equipment capital;
- Miscellaneous mine capital;

The cost breakdown has been shown in Table 21-2.

21.2.3 Pre-Production Stripping

Mining activity commences in advance of the process plant achieving commercial production. This includes the movement of 11.7 Mt of waste and placement of 0.4 Mt of mill feed in a stockpile adjacent to the primary crusher. The mine operating costs associated with this time period are included in the capital cost estimate and expected to cost $61.5 M. This cost covers all associated management, dewatering, drilling, blasting, loading, hauling, support, engineering and geology departments labour, grade control costs and financing costs.

Table 21-1: Capital Cost Estimate Summary (C$)

	Initial ($ M)	Sustaining ($ M)	LOM Total ($ M)
Mine			
Pre-stripping	62	—	62
Mining equipment	14	6	20
Mine capital	7	3	9
Sub-total mine	*83*	*9*	*91*
Processing			
Bulk earthworks	7	—	7
Processing	74	7	81
Reagents & plant services	7	1	8
Tailings & water treatment	19	2	21
Onsite infrastructure	22	2	23
Sub-total processing	129	12	141
Infrastructure			
Power	13	—	13
TSF, water supply & treatment	2	4	6
Sub-total infrastructure	*15*	*4*	*19*
Total directs	***226***	***24***	***250***
Indirects	27		27
Total directs + indirects	***253***	***24***	***277***
Owner's costs	10		10
Total excluding contingency	***263***	***24***	***287***
Project contingency	40	3	43
Sub-total	*303*	*27*	*330*
Closure	–	52	52
Total	***303***	***79***	***382***

Table 21-2: Mining Capital Cost Estimate

Mining Capital Category	Initial Cost ($M)	Sustaining Cost ($M)	Total Capital Cost ($M)
Pre-production stripping	61.5	-	61.5
Mine equipment capital	14.3	5.9	20.2
Miscellaneous mine capital	6.7	2.7	9.4
Total	***82.5***	***8.6***	***91.1***

21.2.4 Mining Equipment Capital

The mining equipment capital costs reflect the use of financing of the major equipment and some support equipment. Equipment prices used current quotations from local vendors. A 20% down payment is included in the capital cost for those units financed. The remaining cost was included in operating costs (refer to Section 21.3).

The base costs provided by the vendors are included in a calculation for each unit cost calculation and options added to that. The capital cost, the cost of financing, and down payment, are shown in Table 21-3.

The cost of spare truck boxes, loader buckets and is included in the capital cost for the major equipment cost estimate, due to the remote nature of the mine.

The distribution of capital costs is completed using the number of units required within a period. If new or replacement units are needed, that number of units, by the unit cost (20% of that for major equipment) is applied to the capital cost in that period. There is no allowance for escalation in any of these costs

The balancing of equipment units based on operating hours is completed for each major piece of mine equipment. The smaller equipment was based on number of units required, based on operational experience. This includes such things as pickup trucks (dependent on the field crews), lighting plants, mechanics trucks, etc. For Eskay Creek, additional support equipment for snow removal and site water control was included to accommodate the expected climatic conditions which includes on average 13 m of snow.

The most significant piece of major mine equipment is the haulage trucks. At the peak of mining, nine units are necessary to maintain mine production. This happens from Year 4 onwards. The maximum hours per truck/per year are set at 6,000. There are periods where the maximum hours per unit are below what the maximum possible can be. In those situations, increasing the maximum on the number of trucks still leaves residual hours required to complete the material movement, therefore, the number of total trucks is unchanged. In these cases, the hours required are distributed evenly across the number of trucks on site and available.

The other major mine equipment is determined in the same manner. Therefore, in some instances the loaders have a longer period of life (same number of hours between replacements) due to the sharing of hours with the other units in the fleet.

Table 21-3: Major Mine Equipment – Capital Cost, Full Finance Cost and Down Payment (C$)

Equipment	Unit	Capacity	Capital Cost ($)	Full Finance Cost ($)	Down Payment ($)
Production drill	mm	140	1,113,000	1,172,000	229,600
Production/crusher loader	m^3	13	2,450,000	2,581,000	490,000
Hydraulic shovel	m^3	22	9,322,000	9,822,000	1,864,000
Haulage truck	t	142	3,194,000	3,365,000	639,000
Track dozer	kW	474	1,708,000	1,800,000	342,000
Grader	kW	163	357,000	376,000	71,000
Support excavator	m^3	6.7	2,072,000	2,183,000	414,000

The support equipment is usually replaced on a number of years of usage basis. For example, pickup trucks are replaced every three years, with the older units possibly being passed down to other departments on the mine site. However, for the purpose of the capital cost estimate, new units are considered for mine operations, engineering, and geology.

The number of pieces of major equipment required by year are shown in Table 21-4.

In the case of the production loader, there is one full time at the primary crusher when the plant commences operation. Its role is to tram material from stockpile and manage the blending of various mill feed types.

The support excavator is a larger unit meant to clean mill feed from previously-mined stopes and windrow the material for loading by either the hydraulic shovel or production loader. It is capable of loading the haul trucks, but is not expected to, as a result of the significant loading time that would result.

The expected equipment life is:

- Production drill: 25,000 hrs;
- Production loader: 35,000 hrs;
- Hydraulic shovel: 72,000 hrs;
- Haulage truck: 50,000 hrs;
- Track dozer: 35,000 hrs;
- Grader: 25,000 hrs;
- Support excavator: 30,000 hrs.

Other support equipment is normally determined in number of years and varies by its duty in the mine. Light plants for example are replaced each four years. The integrated tool carrier for site support is purchased once at the project start and is not replaced over the mine life.

Table 21-4: Mine Equipment on Site

Equipment	Yr-2	Yr-1	Yr 1	Yr 2	Yr 3	Yr 4	Yr 5	Yr 6	Yr 7	Yr 8	Yr 9
Production drill	2	4	5	6	6	6	6	4	3	3	3
Production loader	1	1	2	2	2	2	2	2	2	2	2
Hydraulic shovel	1	2	2	2	2	2	2	2	2	2	2
Haulage truck	3	4	5	8	8	9	9	9	9	9	9
Track dozer	3	4	4	4	4	4	4	4	4	4	4
Grader	2	3	3	3	3	3	3	3	3	3	3
Support excavator	1	1	1	1	1	1	1	1	1	1	1
Snowplow	2	3	3	3	3	3	3	3	3	3	3

21.2.5 Miscellaneous Mine Capital

The miscellaneous mine capital includes various separate line items in the costing. These are shown in Table 21-5.

The engineering office equipment includes such items as desktop computers, plotter, copies of the mining and geology software, and survey equipment with associated peripherals. This cost is estimated at $1.2 M, with the majority being the mining/geology software.

The dispatch system will use an iPad-based system with a Wi-Fi system in the pit area. This provides checklists and truck routing in addition to data collection.

The communication system is the establishment of radio/cell coverage in the pit area for use by mine engineering and operations complete with lightning protection.

The dewatering system includes pumps and piping required to draw the existing underground water level down below the active pit level and handle expected annual rainfall. The pumps will be electric and will lift the water to the pit rim then pump horizontally to the settling ponds on the west side of the pit for treatment (if required), and discharge to the environment.

Waste rock storage facility preparation will include the removal of merchantable timber, grubbing, and any topsoil removed and stockpiled.

The pit road from the plant to the crusher will initially be sufficient for single-lane mine truck travel from the pit to the mine workshop, and will be widened with mine waste. This will include access to the proposed conveyor system as well as passing lanes for light vehicles. This road is expected to be 2.1 km in length.

The road from the crusher to the top of the pit (Phase 4 and South Pit) will be 1.7 km long, and will be developed as a single lane for access, and widened with pit material as development starts at the top.

The north end access road is planned to be 750 m long, and provide access around the north end of the pit to the settling ponds and water treatment facility on the west side of the pit. It will also be used by the pit for material movement and is constructed as double lane in size.

Table 21-5: Miscellaneous Mine Capital (C$)

Miscellaneous Mining Capital	Initial Cost ($)	Sustaining Cost ($)	Total Capital Cost ($)
Engineering office equipment	1,200,000	—	1,200,000
Dispatch system	800,000	—	800,000
Communications	400,000	—	400,000
Dewatering system – pumps/pipe	1,031,000	2,650,000	3,681,000
WRSF preparation (clear/grub)	200,000	—	200,000
Pit area (clear/grub)	200,000	—	200,000
Pit access road – plant to crusher	1,050,000	—	1,050,000
Pit access road – crusher to top of pit	1,275,000	—	1,275,000
Pit access road – around pit north end	563,000	—	563,000
Total	***6,719,000***	***2,650,000***	***9,369,000***

21.2.6 Process Plant Capital Cost Estimate

21.2.6.1 Source Documentation

Data for this estimate has been obtained from numerous sources including:

- Conceptual engineering design by Ausenco;
- Topographical information;
- Budgetary quotation;
- Historical database pricing less than six months old;
- Historical database pricing older than six months old, escalated to the current base date;
- Factored from costs with a basis.

21.2.6.2 Estimate Basis

The capital cost estimate has been developed to a conceptual level of accuracy based on Ausenco's in-house database of projects and studies and experience from similar operations. Contingency was estimated, per area, based on the confidence level of source information.

A summary of the project capital cost estimate is presented in Table 21-6.

Table 21-6: Capital Cost Estimate Summary

Area	Initial Capital Cost (C$ M)	Sustaining Cost (C$ M)	Contingency (%)	Contingency (C$ M)
Plant Site Facility				
Process plant	67.8	6.1	13	8.8
Concentrate handling and loadout	13.2	1.3	20	2.6
Reagents, services and tailings discharge	11.4	1.1	20	2.3
Water treatment plant	15.0	1.5	20	3.0
On-site infrastructure	21.6	1.8	20	4.3
Sub-total Plant & On-site Infrastructure	*128.9*	*11.9*	*16*	*21.1*
Off-Site Infrastructure				
Mining	83.0	8.5	5	4.2
Off-site Infrastructure	14.1	3.6	31	4.4
Sub-Total Off-Site Infrastructure	*97.1*	*12.1*	*9*	*8.6*
Total Direct Costs	***226.0***	***24.0***	***13***	***29.6***
Indirect Costs				
Field indirect costs	8.6	—	35	3.0
EPCM	18.1	—	20	3.6
Total Indirect Costs	***26.7***	***—***	***25***	***6.6***
Total Directs & Indirect Costs	***252.7***	***24.0***	***—***	***—***
Owners costs	10.1	—	35	3.5
Sub-total Project	*262.8*	*—*	*15*	*39.8*
Project contingency	39.8	3.2	—	—
Closure		52.0	—	—
Grand Total Project (C$)	***303***	***79***	***—***	***—***
Grand Total Project (US$)	***233***	***61***	***—***	***—***

21.2.6.3 Direct Costs

The definition of process equipment requirements was based on conceptual process flowsheets and process design criteria as defined in Section 17.

Process equipment costs were derived using recent similar projects, recent and historical budget quotes on file from vendors. Delivery and installation of process equipment is a factored cost relative to the total purchase price of equipment. A detailed estimate of man-hours was not completed for this study. The factors developed were based on installation by local contractors.

Bulk earthworks for the plant site, camp and mine ancillary buildings were developed based on semi-detailed cut and fill volumes based on process plant layout and site topographical information. Unit rates were benchmarked based on recent projects within the region. An allowance for plant site perimeter fencing was included.

On-site infrastructure costs were developed based on in-house database of costs and include:

- Camp for 200 workers complete with kitchen/dining, recreational and laundry facilities;
- Mine buildings including truckshop, mine workshop, tire repair shop and mine dry building;
- Process plant buildings including workshop and laboratory;
- Ancillary buildings including warehousing, administration and gatehouse.

Off-site infrastructure includes:

- High voltage (HV) overhead power line 14 km x 69 kV: benchmarked cost per distance;
- HV substation with transformer 69 kV to 33 kV: in-house database;
- An allowance for HV powerline tie-in;
- Widening of the access road: semi-detailed quantities;
- Water supply from wells including cut-off drains for diversion: allowance.

21.2.6.4 Indirect Costs

Indirect costs are those that are required during the project delivery period to enable and support the construction activities. Indirect costs include:

- Field indirect costs: construction distributable costs (tools, supplies, consumables) and temporary facilities;
- Field indirect costs: vendor representatives, first fill and spares;
- EPCM: home office engineering; site and home office expenses.

The indirect cost estimate was factored based on previous Ausenco experience with similar-sized projects. EPCM was estimated at 13% of total direct costs, and field indirect costs at 6% of total direct costs.

21.2.6.5 Owners Costs

Owners costs were estimated at 4% of total direct and indirect costs. These costs include an allowance for:

- General and administrative costs for the Owner’s project team on and off-site;
- Consultants and contractors;
- Mobile equipment and fixed plant;
- Pre-production operations.

21.3 Operating Cost Estimates

21.3.1 Summary

The operating cost estimate provided in Table 21-7 is based on a combination of first-principal calculations, experience, reference projects and factors as appropriate for a PEA.

21.3.2 Mining

The Eskay Creek mine operating costs have been estimated from base principals with vendor quotations for repair and maintenance costs and other suppliers for consumables. Key inputs to the mine cost are fuel and labour. The price provided for the project was $1.04/L delivered to the site. The mine fleet will be entirely diesel powered. The dewatering pumps will be electric powered and a price of $0.06 per kilowatt hour was used.

21.3.2.1 Labour

Labour costs for the various job classifications were obtained from salary surveys in British Columbia and other operations. A burden rate between 39% and 44% was applied to the various rates. Labour was estimated for both staff and hourly on a 12-hour shift basis using a rotation of either two weeks on/two weeks off or 4 days x 3 days. Mine positions and salaries are shown in Table 21-8.

The mine staff labour remains constant from Year 2 until Year 7, when positions are removed as the mine winds down. During the pre-production period and Year 1 there will be two trainer positions in mine operations.

Hourly employee labour force levels in mine operations and maintenance fluctuate with production requirements. The Year 5 hourly labour requirements are shown in Table 21-9.

Labour costs are based on an Owner-operated scenario, with Skeena responsible for the maintenance of the equipment with its own employees.

Overseeing all the mine operations, maintenance, engineering, and geology functions will be a Technical Superintendent. This person would have the Mine General Foreman and Maintenance Superintendent reporting to them, as well as the Chief Engineer and Chief Geologist.

The Mine General Foreman would have the Shift Foremen report directly to them.

The mine will have four mine operations crews, each with a Senior Shift Foremen who will have one Junior Shift Foreman reporting to them. Over the mine life, there will also be a Road Crew/Services Foreman responsible for roads, drainage, and pumping around the mine. This person would also be a backup Senior Mine Shift Foreman. The Training Foreman role is only required on site until the end of Year 1, at which time the positions are eliminated. The Mine Operations department will have its own Clerk/Secretary.

The Chief Engineer will have one Senior Engineer and two Open Pit Engineers reporting to them. The Blasting Engineer would be included in the Short-Range Planning Group and would double as Drill-And-Blast Foreman as required. The Geotechnical Engineer would cover all aspects of the wall slopes and WRSFs, together with shared technicians in blasting.

Table 21-7: Operating Cost Estimate Summary (C$)

Operating Cost	Annual Cost ($M)	Annual Cost ($/t Processed)
Mining	65.8	26.32
Processing	43.3	17.31
Contingency on process	6.5	2.60
Water treatment	4.4	1.74
Site G&A	15.15	6.06
Total	***135.1***	***54.02***

Table 21-8: Mine Staffing Requirements and Annual Employee Salaries (Year 5)

Position	Employees	Annual Salary (C$/a)
Mine Maintenance		
Maintenance Superintendent	1	207,000
Maintenance General Foreman	1	178,100
Maintenance Shift Foremen	4	144,900
Maintenance Planner/Contract Administration	2	132,100
Clerk	1	85,800
Subtotal	*9*	
Mine Operations		
Mine Operations/Technical Superintendent	1	220,800
Mine General Foreman	1	191,800
Senior Shift Foreman	4	144,900
Junior Shift Foreman	4	132,100
Road Crew/Services Foreman	1	144,900
Clerk	1	85,800
Subtotal	*12*	
Mine Engineering		
Chief Engineer	1	194,600
Senior Engineer	1	164,400
Open Pit Planning Engineer	2	144,900
Geotechnical Engineer	1	144,900
Blasting Engineer	1	144,900
Blasting/Geotechnical Technician	2	98,700
Dispatch Technician	1	98,700
Surveyor/Mining Technician	2	98,700

Position	Employees	Annual Salary (C$/a)
Surveyor/Mining Technician Helper	2	92,300
Clerk	1	85,800
Subtotal	*14*	
Geology		
Chief Geologist	1	180,700
Senior Geologist	1	151,800
Grade Control Geologist/Modeller	4	125,100
Sampling/Geology Technician	6	98,700
Clerk	1	85,800
Subtotal	*13*	
Total	***48***	

Table 21-9: Hourly Manpower Requirements and Annual Salaries (Year 5)

Position	Employees	Annual Salary (C$/a)
Mine General		
General Equipment Operator	12	103,100
Road/Pump Crew	8	100,000
General Mine Labourer	8	80,100
Trainee	4	76,100
Light Duty Mechanic	4	133,000
Tire Technician	4	107,900
Lube Truck Driver	4	92,600
Subtotal	*44*	
Mine Operations		
Driller	28	107,900
Blaster	2	107,900
Blast Helper	4	80,100
Loader Operator	4	119,300
Hydraulic Shovel Operator	8	119,300
Haul Truck Driver	36	103,000
Dozer Operator	10	107,900
Grader Operator	5	107,900
Crusher Loader Operator	3	119,300
Snowplow/Water Truck	7	101,000

Position	Employees	Annual Salary (C$/a)
Subtotal	*107*	
Mine Maintenance		
Heavy Duty Mechanic	25	133,000
Welder	19	133,000
Electrician	2	133,000
Apprentice	6	93,300
Subtotal	*52*	
Total Hourly	***203***	

The Short-Range Planning Group in Engineering will have two Surveyor/Mine Technicians and two Surveyors/Mine Helpers. These employees will assist in the field with staking, surveying, and sample collection with the geology group; they will have a Clerk/Secretary to assist the team.

In the Geology Department, there will be one Senior Geologist reporting to the Chief Geologist. There will also be four Grade Control Geologists/Modellers; two will be in short range and grade control drilling, and the others will be in long range/reserves. There will also be six Grade Control/Sampling Technicians and one Clerk/Secretary.

Four Mine Maintenance Shift Foremen will report to the Maintenance General Foreman who in turn will report to the Maintenance Superintendent. There will be two Maintenance Planners/Contract Administrators and a Clerk.

The hourly labour force includes positions for the Light Duty Mechanic, Tire Men, and Lube Truck Drivers. These positions will all report to Maintenance. There will generally be one of each position per crew. Other general labour includes General Mine Labourers (two per crew) and Trainees (one per crew) plus two Road/Pump Crew personnel per crew for water management/snow removal.

The drilling labour force is based on one operator per drill, per crew while operating. This peaks at 28 Drillers in Year 5 and then drops down over time as the drilling hours are diminished.

Shovel and Loader Operators peak at 12 in Year 1 and hold at that level until Year 6. Haulage Truck Drivers peak at 36 in Year 4 and 5 and then taper off to the end of the mine life.

Maintenance factors are used to determine the number of Heavy-Duty Mechanics, Welders and Electricians are required and are based on the number of equipment operators. Heavy Duty Mechanic requirements work out to 0.25 mechanics required for each Drill Operator for example. Welders are 0.25 per operator and Electricians are 0.05 per operator.

The number of Loader, Truck and Support Equipment Operators is estimated using the projected equipment operating hours. The maximum number of employees is four per unit, to match the mine crews.

21.3.2.2 Equipment Operating Costs

Vendors provided repair and maintenance (R&M) costs for each piece of equipment selected for the Eskay Creek PEA. Fuel consumption rates were estimated from the supplied information and knowledge of the working conditions. The costs for the R&M are expressed in $/h form.

Tire costs were also collected from various vendors for the sizes expected to be used. Estimates of tire life are based on AGP's experience. The operating cost of the tires is expressed in a $/hr form also. The life of the haulage truck tires is estimated at 5,000 hours per tire with proper rotation from front to back. Each truck tire costs $21,500 so the cost per hour for tires is $25.80 /hr for the truck using six tires in the calculation.

Ground engaging tools (GET) costing is estimated from other projects and is an area that would be fine-tuned once the project was operational.

Drill consumables are estimated as a complete drill string using the parts list and component lives provided by the vendor. Drill productivity is estimated at 24.6 m/hr for mill feed and waste. The equipment costs used in the estimate are shown in Table 21-10.

21.3.2.3 Drilling

Drilling in the open pit will use down the hole hammers drill rigs with 140 mm bits. The pattern size varies between mill feed and waste and is blasted in recognition of the equipment being used. The material will be smaller and finer to improve productivity and reduce maintenance costs as well as improve plant performance. The drilling pattern parameters are shown in Table 21-11.

The sub-drill is included to allow for caving of the holes in weaker zones, reducing re-drill requirements or short holes that would affect bench floor conditions. The extra sub-drill is above what is normally required.

The parameters used to estimate drill productivity are shown in Table 21-12.

21.3.2.4 Blasting

An emulsion product will be used for blasting to provide water protection. With the high rainfall known to occur in the area and large snow melt, it is expected that a water-resistant explosive will be required. The powder factors used in the explosives calculation are shown in Table 21-13.

The blasting cost is estimated using quotations from a local explosives vendor. The emulsion price is $83.60/100 kg. The operations will be responsible for guiding the loading process, including placement of boosters/Nonels, and stemming and firing the shot.

The explosives vendor will lease the explosives and accessories for a monthly cost. A service charge for the vendors pickup trucks, pumps, labour and cost of the explosives plant are included. The total monthly cost was $170,000 per month.

Table 21-10: Major Equipment Operating Costs – No labour ($/hr)

Equipment	Fuel	Lube/Oil	Tires/ Undercarriage	Repair & Maintenance	GET/ Consumables	Total
Production drill	93.60	9.36	—	98.96	46.99	248.91
Production/crusher loader	88.40	8.84	19.20	74.48	10.00	200.92
Hydraulic shovel	270.40	27.04	—	242.43	12.00	551.87
Haulage truck	91.52	9.15	25.80	89.53	4.00	220.00
Track dozer	83.20	8.32	10.00	72.84	5.00	179.36
Grader	22.88	2.29	3.73	18.68	5.00	52.58
Support excavator	62.40	12.48	—	67.31	8.00	150.19

Table 21-11: Drill Pattern Specifications

Specification	Unit	Mill Feed	Waste
Bench height	m	8	8
Sub-drill	m	0.8	0.8
Blasthole diameter	mm	140	140
Pattern spacing - staggered	m	4.8	4.6
Pattern burden – staggered	m	4.2	4.0
Hole depth	m	8.8	8.8

Table 21-12: Drill Productivity Criteria

Drill Activity	Unit	Mill Feed	Waste
Pure penetration rate	m/min	0.55	0.55
Hole depth	m	8.8	8.8
Drill time	min	16.00	16.00
Move, spot and collar hole	min	3.00	3.00
Level drill	min	0.50	0.50
Add steel	min	0.50	0.50
Pull drill rods	min	1.50	1.5
Total setup/breakdown time	min	5.50	5.50
Total drill time per hole	min	21.5	21.5
Drill productivity	m/hr	24.6	24.6

Table 21-13: Design Powder Factors

	Unit	Mill Feed	Waste
Powder Factor	kg/m^3	0.70	0.78
Powder Factor	kg/t	0.26	0.29

21.3.2.5 Loading

Loading costs for both mill feed and waste are based on the use of hydraulic shovels and front-end loaders. The shovels will be the primary diggers with the front-end loaders as backup/support units. The average percentage of each material type that the various loading units are responsible for is shown in Table 21-14, as at Year 5. This highlights the focus of the shovels over the loaders.

The trucks present at the loading unit refers to the percentage of time a truck is available to be loaded. To maximize truck productivity and reduce operating costs, it is more efficient to slightly under-truck the loading unit. One of the largest operating cost items is haulage and minimizing this cost by maximizing the truck productivity is crucial to lower operating costs. The value of 80% comes from the standby time shovels typically encounter due to a lack of trucks.

21.3.2.6 Hauling

Haulage profiles were determined for each pit phase for the primary crusher or the waste rock facility destinations. Cycle times were generated for the appropriate period tonnage by destination and phase to estimate the haulage costs. Maximum speed on the trucks is limited to 50 km/hr for tire life and safety reasons although few locations in the mine plan appeared to offer the truck the opportunity to accelerate to that velocity. Calculation speeds for various segments are shown in Table 21-15.

21.3.2.7 Support Equipment

Support equipment hours and costs are determined on factors applied to various major pieces of equipment. For the PEA, some of the factors used are shown in Table 21-16.

These factors resulted in the need for four track dozers, three graders, and one support backhoe. Their tasks will include clean-up of the loader faces, roads, WRSFs, and blast patterns. The graders will maintain the crusher and waste haul routes. In addition, snowplow/water trucks will have the responsibility for patrolling the haul roads for snow removal and controlling fugitive dust for safety and environmental reasons. The small backhoe and road crew dump trucks will be responsible for cleaning out sedimentation ponds and water ditch repairs.

The hours generated in this manner were applied to the individual operating costs for each piece of equipment. Many of these units will be support equipment, so no direct labour is allocated to them due to their variable function. The operators will come from the General Equipment operator pool.

Table 21-14: Loading Parameters – Year 5

	Unit	Hydraulic Shovel	Front End Loader
Bucket capacity	m^3	22	13
Truck capacity loaded	t	144	144
Waste tonnage loaded	%	90	10
Mill feed tonnage loaded	%	80	20
Bucket fill factor	%	95	95
Cycle time	sec	38	40
Trucks present at loading unit	%	80	80
Loading time	min	2.60	4.03

Table 21-15: Haulage Cycle Speeds

	Flat (0%) On Surface	Flat (0%) Inpit, Crusher, Dump	Slope Up (8%)	Slope Up (10%)	Slope Down (8%)	Slope Down (10%)
Loaded (km/hr)	50	40	16	12.1	30	30
Empty (km/hr)	50	40	35	25	35	35

Table 21-16: Support Equipment Operating Factors

Mine Equipment	Factor	Factor Units
Track dozer	30%	Of haulage hours to maximum of 4 dozers
Grader	15%	Of haulage hours to maximum of 3 graders
Crusher loader	45%	Of loading hours to maximum of 1 loader
Snowplow/water truck	12%	Of haulage hours to maximum of 3 trucks
Pit support backhoe	35%	Of loading hours to maximum of 1 backhoe
Road crew backhoe	6	hours/day/unit
Road crew dump truck	6	hours/day/unit
Road crew loader	6	hours/day/unit
Lube/fuel truck	6	hours/day/unit
Mechanics truck	14	hours/day/unit
Blasting loader	8	hours/day/unit
Blaster’s truck	8	hours/day/unit
Integrated tool carrier	4	hours/day/unit
Light plants	12	hours/day/unit
Pickup trucks	8	hours/day/unit

21.3.2.8 Grade Control

Grade control will be completed with a separate fleet of RC drill rigs. These rigs will drill the deposit off on a 10 x 5 m pattern in areas of known mineralization taking samples each metre. The holes will be inclined at 60º.

In areas of low-grade mineralization or waste, the pattern spacing will be 20 x 10 m, with sampling over 6 m. These drill holes will be used to find undiscovered veinlets or pockets of mineralization. Over the life of the mine, a total of 169,000 m of drilling are expected to be completed for grade control work. A total of 186,000 samples is anticipated to be assayed from that drilling.

The grade control holes will serve two purposes:

- Definition of the mill feed grade and contacts;
- Location of previous underground infrastructure prior to blasthole rigs drilling.

Samples collected will be sent to the assay laboratory and assayed for use in the short-range mining model.

Additional costing for blasthole sampling has been included. This may not be necessary once a gold deportment study is completed to best determine the sampling protocol.

Costs associated with this separate drill program will be tracked as a distinct line item for the mining cost. The drill crew will be one driller and two helpers with oversight by the Mine Geology Department. The cost of this drilling is expected to be over $2 M/a.

21.3.2.9 Dewatering

Pit and underground workings dewatering will be an important part of mining at Eskay Creek. Significant volumes will need to be pumped initially to allow the open pit to advance, in addition to the normally elevated rain/snow amounts.

For the purposes of the PEA, AGP reviewed historical dewatering data and compared this to the proposed mining area to estimate the water volume that will be required to be pumped. Initial pumping in Year -2 is expected to be just over 1 million cubic metres. That climbs rapidly to 3.5 Mm^3 in Years -1 to Year 2 then levels at around 4.5 Mm^3 for the remainder of the mine life. This volume also includes the WRSF areas as water from these areas is expected to be controlled, sampled and treated if required.

The dewatering is planned to be completed with a set of four pumps in the pit and two pumps on the surface to push the water to the settling ponds. These pumps will be electric to reduce the cost of this operation.

Additional dewatering in the form of horizontal drill holes is included as part of the dewatering costs. These holes will be campaigned and will be part of the sustaining mine capital.

Dewatering is expected to cost $5.3 M over the proposed mine life.

21.3.2.10 Leasing

Leasing of the mine fleet is considered a viable option to reduce initial capital. Various vendors offer this as an option to help select their equipment. Both Caterpillar and Komatsu have the ability, and desire, to allow leasing of their product lines.

Indicative terms for leasing provided by the vendors are:

- Down payment = 20% of equipment cost;
- Term Length = 3-5 years (depending on equipment);
- Interest Rate = LIBOR plus a percentage;
- Residual = $0.

The proposed interest rate is used to calculate a multiplier on the amount being leased. The multiplier is 1.067 to equate to the rate. It does not consider a declining balance on the interest, but rather the full amount of interest paid over the term, equally distributed over those years. The calculation is as follows:

- Annual Lease Cost = {[(Initial Capital Cost) x 80%] x 1.067} / term in years

The initial capital, down payments, and annual leasing costs were included in Section 21.2.

The support equipment fleet is calculated in the same manner as the major mining equipment.

All of the major mine equipment, and the majority of the support equipment, where it was considered reasonable, was assumed to be leased. If the equipment had a life greater than the lease term length, then the following years onward of the lease did not have a lease payment applied. In the case of the mine trucks, with an approximate 10-year working life, the lease would be complete, and the trucks would simply incur operating costs after that time. For this reason, the operating cost would vary annually depending on the equipment replacement schedule and timing of the leases.

Using the leasing option adds $0.32/t to the mine operating cost over the life of the mine. On a cost per tonne of feed basis, it was $2.47/t mill feed.

21.3.2.11 Total Mine Costs

The total life of mine operating costs per tonne of material moved and per tonne of mill feed processed are shown in Table 21-17 and Table 21-18.

The General Mine Engineering includes the cost associated with a contract crushing plant to make stemming material and road crush. That cost is approximately $1.5 M/a.

Table 21-17: Open Pit Mine Operating Costs – with Leasing ($/t Total Mined)

Open Pit Category	Unit	Year 1	Year 3	Year 5	LOM Average
General Mine and Engineering	$/t mined	0.53	0.50	0.50	0.65
Drilling	$/t mined	0.41	0.41	0.46	0.44
Blasting	$/t mined	0.43	0.42	0.42	0.45
Loading	$/t mined	0.33	0.33	0.33	0.35
Hauling	$/t mined	0.38	0.57	0.63	0.59
Support	$/t mined	0.38	0.47	0.49	0.50
Grade control	$/t mined	0.09	0.09	0.09	0.11
Leasing costs	$/t mined	0.45	0.45	0.17	0.32
Dewatering	$/t mined	0.02	0.03	0.03	0.04
Total	***$/t mined***	***3.02***	***3.25***	***3.13***	***3.44***

Table 21-18: Open Pit Mine Operating Costs – with Leasing ($/t Mill Feed)

Open Pit Category	Unit	Year 1	Year 3	Year 5	LOM Average
General Mine and Engineering	$/t mill feed	5.38	5.00	5.02	4.94
Drilling	$/t mill feed	4.20	4.06	4.64	3.34
Blasting	$/t mill feed	4.34	4.22	4.21	3.47
Loading	$/t mill feed	3.40	3.29	3.29	2.66
Hauling	$/t mill feed	3.91	5.67	6.35	4.50
Support	$/t mill feed	3.91	4.67	4.91	3.81
Grade control	$/t mill feed	0.95	0.87	0.88	0.85
Leasing costs	$/t mill feed	4.56	4.46	1.72	2.47
Dewatering	$/t mill feed	0.17	0.28	0.28	0.28
Total	***$/t mill feed***	***30.84***	***32.51***	***31.30***	***26.32***

21.3.3 Processing

Processing costs for power, consumables, maintenance consumables and labour are summarised in Table 21-19.

21.3.3.1 Power

Power costs were calculated from an estimate of annual power consumption and using a unit cost of $0.06/kWh.

Power consumption was derived from calculated power draw of the ball and SAG mills, plus an allowance for the remainder of the plant, based on typical flotation plants. The average on-line power draw is estimated at 19 MW.

Annual energy consumption is estimated at 127,564 MWh, or about $7.65 M.

21.3.3.2 Consumables

Processing reagent and consumable costs were estimated based on the throughput. Costs are summarised in Table 21-20.

Costs for liners were estimated based on vendor information and benchmarking similar plants. Costs for mill balls were estimated for expected consumption based on an assumed abrasion index ($A_{i)}$ of 0.2 and used a unit cost of $1.95/kg. These costs are summarized in Table 21-21.

Reagent costs were based on:

- Consumption rates determined in test work;
- Data base unit costs for the reagents;
- An allowance of 9% of reagent costs for freight.

Reagent costs are summarised in Table 21-22.

21.3.3.3 Maintenance Consumables

Annual maintenance spares and consumable costs were estimated at 3% of total installed costs for mechanical equipment, plate work, support steel and electrics ($81.4 M).

This results in an annual maintenance consumables cost estimate of $2.44 M.

21.3.3.4 Labour

Labour costs include all processing and maintenance costs (Table 21-23).

Costs were estimated from a breakdown of staffing positions, estimated at 120 in total, excluding G&A manpower.

Costs are average pays inclusive of all loadings applicable to the site.

Table 21-19: Processing Costs (C$)

Processing Cost item	Annual Cost ($M)	Annual Cost ($/t Processed)
Power	7.65	3.06
Operating consumables	15.53	6.21
Maintenance consumables	2.44	0.98
Labour	17.64	7.06
Other (15% contingency)	6.49	2.60
Total	***49.76***	***19.90***

Table 21-20: Processing Reagent and Consumable Costs (C$)

Consumable Item	Annual Costs ($M)
Liners and media	9.81
Reagents	5.72
Total	***15.53***

Table 21-21: Costs for Liners and Media

Consumable Item	Annual Consumption	Annual Cost (C$000)
Crusher liners	5 sets	871
SAG mill liners	2 sets	653
Ball mill liners	1 set	425
SAG mill balls	0.40 kg/t	2,097
Ball mill balls	1.10 kg/t	5,767
Total		***9,812***

Table 21-22: Reagents Costs

Reagent	Addition Rate (kg/t)	Annual Cost (C$000)
PAX	0.265	2,455
MIBC	0.134	2,775
Copper sulphate	0.11	423
Flocculant	0.006	67
Total		***5,720***

Table 21-23: Labour Costs

Cost Centre	Number	Annual Cost (C$M)
Plant management	7	1.42
Foremen and working staff	31	5.23
Mill operators and sample preparation	44	5.35
Plant maintenance	30	3.64
Contract labour	8	2.00
Total	***120***	***17.64***

21.3.3.5 Other (Contingency)

An allowance of 15% of all other operating costs was made, to include fuel costs, laboratory chemicals and similar sundry items.

21.3.4 General and Administration

The G&A operating costs were estimated based on benchmarked data from similar projects in BC Canada. Costs include camp operations, G&A personnel, off-site offices, contracts, and vehicle maintenance, as well as miscellaneous project costs.

The annual G&A cost is estimated at $15.2 M/a.

21.4 QP Comments on "Item 21: Capital and Operating Costs"

The QP notes:

- Capital costs are estimated at $303 M of initial capital, $79 M of sustaining capital, for an overall capital cost estimate of $382 M;
- Operating costs of $135.1 M/a, or $54.02/t processed.

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 estimates;
- Assumed commodity prices and exchange rates;
- The proposed mine production plan;
- Projected mining and process recovery rates;
- Assumptions as to mining dilution and ability to mine in areas previously exploited using underground mining methods as envisaged;
- Sustaining costs and proposed operating costs;
- Interpretations and assumptions as to joint venture and agreement terms;
- 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 or hydrogeological considerations during mining being different from what was assumed;
- Failure of mining methods to operate as anticipated;
- Failure of plant, equipment or processes to operate as anticipated;
- Changes to assumptions as to the availability of electrical power, and the power rates used in the operating cost estimates and financial analysis;
- Ability to maintain the social licence to operate;
- Accidents, labour disputes and other risks of the mining industry;
- Changes to interest rates;
- Changes to tax rates.

The mine plan is partly based on Inferred Mineral Resources that are considered too speculative geologically to have the economic considerations applied to them that would enable them to be categorized as Mineral Reserves, and there is no certainty that the PEA based on these Mineral Resources will be realized. Mineral Resources that are not Mineral Reserves do not have demonstrated economic viability.

Calendar years used in the financial analysis are provided for conceptual purposes only. Permits still have to be obtained in support of operations, and approval for development to be provided by Skeena's Board.

22.2 Methodology Used

An engineering economic model was developed to estimate annual pre-tax and post-tax cash flows and sensitivities of the Project based on a 5% discount rate. It must be noted, however, that tax estimates involve many complex variables that can only be accurately calculated during operations and, as such, the after-tax results are only approximations. Sensitivity analysis was performed to assess impact of variations in metal prices, head grades, operating costs and capital costs. The capital and operating cost estimates were developed specifically for this Project and are summarized in Section 21 of this Report (presented in 2019 dollars). The economic analysis has been run with no inflation (constant dollar basis).

22.3 Financial Model Parameters

The economic analysis was performed using the following assumptions:

- Commercial production start-up in 2023;
- Construction period of two years;
- Mine life of 8.6 years;
- Base case gold price of US$1,325/oz and silver price of US$16/oz was based on consensus analyst estimates and recently published economic studies. The forecasts used are meant to reflect the average metal price expectation over the life of the Project. No price inflation or escalation factors were taken into account. Commodity prices can be volatile, and there is the potential for deviation from the forecast;
- United States to Canadian dollar exchange rate assumption of 0.77 (US$/C$)
- Cost estimates in constant Q3 2019 C$ with no inflation or escalation factors considered;
- Results are based on 100% ownership with 1% NSR;
- Capital costs funded with 100% equity (i.e. no financing costs assumed);
- All cash flows discounted to December 31, 2019;
- All metal products are assumed sold in the same year they are produced;
- Project revenue is derived from the sale of gold concentrate into the international marketplace;
- No contractual arrangements for smelting or refining currently exist.

22.3.1 Taxes

The Project has been evaluated on an after-tax basis to provide approximate value of the potential economics. The tax model was prepared by MNP LLP, an independent tax consultant. The calculations are based on the tax regime as of the date of the PEA.

At the effective date of the cashflow, the Project was assumed to be subject to the following tax regime:

- The Canadian Corporate Income Tax system consists of the federal income tax (15%) and the provincial income tax (12%);
- The BC Minerals Tax was modelled using a net current proceeds rate of 2% and a net revenue tax rate of 13%.

Total tax payments are estimated to be C$514 M over the LOM.

22.3.2 Working Capital

A high-level estimation of working capital has been incorporated into the cash flow based on Accounts Receivable (30 days), Inventories (60 days) and Accounts Payable (60 days).

22.3.3 Closure Costs and Salvage Value

A 1% NSR royalty has been assumed for the Project, resulting in approximately C$30 M in royalty payments over life of mine.

22.4 Economic Analysis

The economic analysis was performed assuming a 5% discount rate. The pre-tax net present value discounted at 5% (NPV5%) is C$993 M, the internal rate of return IRR is 63.3%, and payback is 1.1 years. On an after-tax basis, the NPV5% is C$638 M, the IRR is 50.5%, and the payback period is 1.2 years.

A summary of the Project economics is included in Table 22-1 and shown graphically in Figure 22-1. The cashflow on an annualized basis is provided in Table 22-2.

22.5 Sensitivity Analysis

A sensitivity analysis was conducted on the base case pre-tax and after-tax NPV and IRR of the Project, using the following variables: metal price, discount rate, grade, capital costs, and operating costs. Table 22-3 summarizes the sensitivity analysis results. Table 22-4 shows the pre-tax sensitivity analysis findings, and Table 22-5 shows the results post-tax.

Analysis revealed that the Project is most sensitive to changes in metal prices and head grade, then, to a lesser extent, to operating costs and capital costs.

22.6 QP Comments on “Item 22: Economic Analysis”

Based on the assumptions and parameters presented in this Report, the PEA shows positive economics.

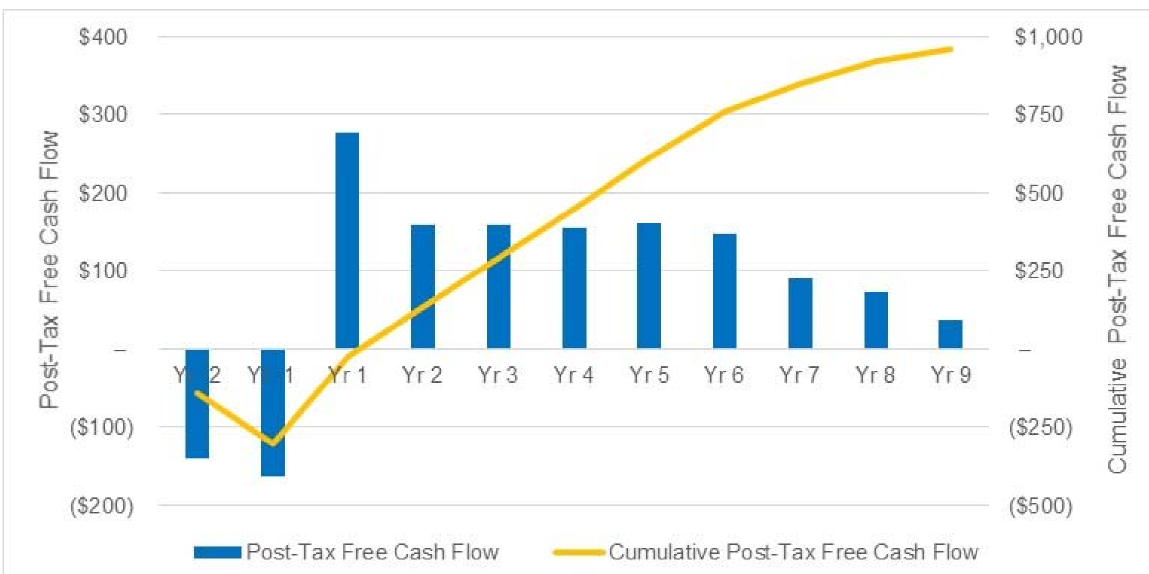
Table 22-1: Summary, Projected LOM Cashflow Assumptions and Results

	Units	Values
General Assumptions		
Gold price	(US$)	1,325
Silver price	(US$)	16
Exchange rate	(US$/C$)	0.77
Fuel cost	(C$/litre)	1.04
Power cost	(C$/kwh)	0.06
Discount rate	(%)	5
Net smelter royalty	(%)	1%
Contained Metals		
Contained gold ounces	(koz)	2,212
Contained silver ounces	(koz)	53,404
Contained gold equivalent ounces	(koz)	2,857
Production		
Gold recovery	(%)	91.1
Silver recovery	(%)	92.4
LOM gold production	(koz)	2,022
LOM silver production	(koz)	49,872
LOM gold equiv. production	(koz)	2,624
LOM avg. annual gold production	(koz)	236
LOM avg. annual silver production	(koz)	5,812
LOM avg. annual gold equiv. production	(koz)	306
Operating Costs Per Tonne		
Mining cost	(C$/t mined)	3.44
Mining cost	(C$/t milled)	26.32
Processing cost	(C$/t milled)	21.64
G&A cost	(C$/t milled)	6.06
Total operating costs	(C$/t milled)	54.03
NSR Parameters		
Gold payability	(%)	95%
Silver payability	(%)	80%
Treatment charges	(US$/dmt)	$180
Gold refining charges	(US$/oz)	$15
Silver refining charges	(US$/oz)	$1
Transport to smelter	(C$/wmt)	$118
Cash Costs and All-in Sustaining Costs		
LOM cash cost net of silver by-product	(US$/oz Au)	$582

	Units	Values
LOM cash cost co-product	(US$/oz AuEq)	$731
LOM AISC net of silver by-product	(US$/oz Au)	$615
LOM AISC co-product	(US$/oz AuEq)	$757
Capital Expenditures		
Pre-production capital expenditures	(C$M)	$303
Sustaining capital expenditures	(C$M)	$27
Reclamation cost	(C$M)	$52
Economics		
Pre-tax NPV (5%)	(C$M)	$993
Pre-tax IRR	(%)	63.3%
Pre-tax payback period	(years)	1.1
After-tax NPV (5%)	(C$M)	$638
After-tax IRR	(%)	50.5%
After-tax payback period	(years)	1.2
Average annual after-tax free cash flow (Year 1–9)	(C$M)	$147
LOM after-tax free cash flow	(C$M)	$959

Notes: Cash costs are inclusive of mining costs, processing costs, site G&A, treatment and refining charges and royalties. AISC includes cash costs plus corporate G&A, sustaining capital and closure costs. Gold equivalent (AuEq) calculated using the formula: Au (g/t) + [Ag (g/t) / 82.8].

Figure 22-1: Projected LOM Cashflow



Note: Figure prepared by Ausenco, 2019.

Table 22-2: Projected Cashflow on an Annualized Basis

				-3	-2	-1	1	2	3	4	5	6	7	8	9
Dollar figures in real C$ M unless otherwise noted	Inputs	Units	Total / Avg.	2020	2021	2022	2023	2024	2025	2026	2027	2028	2029	2030	2031
Macro Assumptions															
Gold Price		US$/oz	$1,325												
Silver Price		US$/oz	$16.00												
FX		C$:US$	0.77												
Fuel Cost		C$/litre	$1.04												
Power Cost		C$/kwhr	$0.06												
Free Cash Flow Valuation															
Gross Revenue		**C$ M**	**$4,138.8**		**–**	**–**	**$683.3**	**$559.3**	**$546.0**	**$526.3**	**$539.6**	**$495.0**	**$347.4**	**$317.5**	**$124.5**
Smelter Costs		C$ M	($765.0)		–	–	($156.1)	($122.5)	($97.6)	($86.3)	($84.4)	($82.8)	($57.6)	($56.2)	($21.4)
Transport		C$ M	($337.3)		–	–	($56.9)	($45.8)	($40.6)	($43.0)	($40.3)	($41.7)	($28.9)	($29.2)	($10.9)
Net Smelter Return		**C$ M**	**$3,036.4**		**–**	**–**	**$470.2**	**$390.9**	**$407.9**	**$396.9**	**$414.8**	**$370.5**	**$260.9**	**$232.1**	**$92.2**
Operating Expenses		C$ M	($1,151.1)		–	–	($137.8)	($149.6)	($150.5)	($148.0)	($147.5)	($132.9)	($120.3)	($109.0)	($55.6)
Royalties		C$ M	($30.4)		–	–	($4.7)	($3.9)	($4.1)	($4.0)	($4.1)	($3.7)	($2.6)	($2.3)	($0.9)
EBITDA		**C$ M**	**$1,854.9**		**–**	**–**	**$327.7**	**$237.5**	**$253.2**	**$245.0**	**$263.2**	**$233.9**	**$138.0**	**$120.8**	**$35.7**
Initial Capex		C$ M	($303.1)		($139.6)	($163.5)	–	–	–	–	–	–	–	–	–
Sustaining Capex		C$ M	($27.3)		–	–	($2.6)	($7.9)	($2.9)	($2.8)	($3.0)	($2.1)	($2.6)	($1.9)	($1.7)
Closure Capex		C$ M	($51.6)		–	–	($2.4)	($2.4)	($4.9)	($5.0)	($7.5)	($7.7)	($7.9)	($8.0)	($5.8)
Change in Working Capital		C$ M	–		–	–	($38.6)	$6.5	($1.4)	$0.9	($1.5)	$3.6	$9.0	$2.4	$19.1
Pre-Tax Unlevered Free Cash Flow		**C$ M**	**$1,472.9**	**–**	**($139.6)**	**($163.5)**	**$284.1**	**$233.7**	**$244.1**	**$238.1**	**$251.2**	**$227.8**	**$136.6**	**$113.2**	**$47.2**
Pre-Tax Cumulative Unlevered Free Cash Flow		C$ M	*$1,472.9*		*($139.6)*	*($303.1)*	*($19.0)*	*$214.7*	*$458.8*	*$696.8*	*$948.1*	*$1,175.9*	*$1,312.5*	*$1,425.7*	*$1,472.9*
Unlevered Cash Taxes		C$ M	($514.1)	–	–	–	($6.7)	($74.4)	($85.1)	($83.2)	($89.8)	($80.1)	($45.5)	($39.6)	($9.6)
Post-Tax Unlevered Free Cash Flow		**C$ M**	**$958.9**	**–**	**($139.6)**	**($163.5)**	**$277.4**	**$159.3**	**$159.0**	**$154.9**	**$161.4**	**$147.7**	**$91.1**	**$73.7**	**$37.6**
Post-Tax Cumulative Unlevered Free Cash Flow		*C$ M*	*$958.9*		*($139.6)*	*($303.1)*	*($25.7)*	*$133.6*	*$292.6*	*$447.5*	*$608.8*	*$756.5*	*$847.6*	*$921.2*	*$958.9*
		Pre-Tax	**Post-Tax**												
NPV (5%)		**C$993.2 (C$ M)**	**$638.0 (C$ M)**												
IRR		**63.3%**	**50.5%**												
Payback		**1.1 yrs**	**1.2 yrs**												
Production															
Mine Life		yrs	8.6												
Total Mill Feed		'000t	21,307		–	382	2,772	1,773	2,610	2,510	2,680	2,730	2,504	2,513	833
Stockpile Rehandle		'000t	1,771		–	382	533	73	110	23	180	230	128	112	–
Waste Mined		'000t	153,963		346	11,317	21,271	23,227	22,390	22,490	22,320	15,072	9,119	4,772	1,637
Total Material Mined (Includes Rehandle)		'000t	177,041		346	12,081	24,576	25,073	25,110	25,023	25,180	18,033	11,752	7,397	2,470
Total Material Mined (Excl. Rehandle)		'000t	175,270		346	11,699	24,043	25,000	25,000	25,000	25,000	17,803	11,624	7,285	2,470
Strip Ratio		waste:mineralization	7.23				9.04	9.29	8.96	9.00	8.93	6.03	3.65	1.91	1.13
Total Mill Feed		**'000t**	**21,307**		**–**	**–**	**2,354**	**2,500**	**2,500**	**2,500**	**2,500**	**2,500**	**2,500**	**2,500**	**1,453**
Beginning Stockpile Inventory		'000t			–	–	382	800	73	182	193	373	603	607	620
Add: Mine to Stockpile		'000t	1,771		–	382	533	73	110	23	180	230	128	112	–
Less: Stockpile to Mill		'000t	(1,771)		–	–	(115)	(800)	–	(12)	–	–	(124)	(99)	(620)
Ending Stockpile Inventory		'000t			–	382	800	73	182	193	373	603	607	620	–

				-3	-2	-1	1	2	3	4	5	6	7	8	9
Dollar figures in real C$ M unless otherwise noted	**Inputs**	**Units**	**Total / Avg.**	**2020**	**2021**	**2022**	**2023**	**2024**	**2025**	**2026**	**2027**	**2028**	**2029**	**2030**	**2031**
Au Head Grade		g/t	3.2		–	–	4.9	3.7	3.3	3.5	3.3	3.4	2.4	2.4	1.6
Ag Head Grade		g/t	78.0		–	–	102.3	91.4	114.2	86.8	110.9	71.3	53.3	25.6	25.2
Hg Head Grade		ppm	80.0		–	–	243	190	120	42	41	29	18	9	8
As Head Grade		ppm	916.9		–	–	3,353	1,386	516	571	778	500	281	416	358
Sb Head Grade		ppm	1,611.2		–	–	4,068	2,755	2,507	1,297	1,457	915	516	302	264
Contained Gold		**kozs**	**2,212.2**		**–**	**–**	**370.9**	**298.4**	**264.3**	**280.2**	**262.8**	**271.8**	**192.3**	**194.7**	**76.8**
Contained Silver		**kozs**	**53,404.2**		**–**	**–**	**7,739.2**	**7,343.9**	**9,181.3**	**6,977.0**	**8,916.9**	**5,731.4**	**4,282.3**	**2,056.8**	**1,175.5**
Contained Gold Equivalent		**kozs**	**2,857.1**		**–**	**–**	**464.3**	**387.1**	**375.1**	**364.4**	**370.5**	**341.0**	**244.1**	**219.5**	**91.0**
Au Recovery	–		91.1%		–	–	92.0%	92.0%	92.0%	92.0%	92.0%	92.0%	90.0%	90.0%	85.0%
Ag Recovery			92.4%		–	–	97.0%	90.0%	97.0%	90.0%	97.0%	90.0%	90.0%	90.0%	90.0%
	Per Annum		**Total LOM**												
Recovered Gold in Concentrate	**235.6**	**kozs**	**2,022.1**		**–**	**–**	**341.2**	**274.6**	**243.1**	**257.8**	**241.8**	**250.1**	**173.1**	**175.2**	**65.3**
Recovered Silver in Concentrate	**5,811.8**	**kozs**	**49,872.4**		**–**	**–**	**7,507.0**	**6,609.5**	**8,905.9**	**6,279.3**	**8,649.4**	**5,158.2**	**3,854.0**	**1,851.1**	**1,058.0**
Recovered Gold Equivalent in Concentrate	**305.8**	**kozs**	**2,624.3**		**–**	**–**	**431.9**	**354.4**	**350.7**	**333.6**	**346.2**	**312.3**	**219.6**	**197.6**	**78.1**
Au Payablity	95.0%		95.0%		–	–	95.0%	95.0%	95.0%	95.0%	95.0%	95.0%	95.0%	95.0%	95.0%
Ag Payablity	80.0%		80.0%		–	–	80.0%	80.0%	80.0%	80.0%	80.0%	80.0%	80.0%	80.0%	80.0%
	Per Annum		**Total LOM**												
Total Payable Gold	**223.9**	**kozs**	**1,921.0**		**–**	**–**	**324.1**	**260.8**	**231.0**	**244.9**	**229.7**	**237.6**	**164.5**	**166.4**	**62.0**
Total Payable Silver	**4,649.5**	**kozs**	**39,897.9**		**–**	**–**	**6,005.6**	**5,287.6**	**7,124.7**	**5,023.4**	**6,919.5**	**4,126.6**	**3,083.2**	**1,480.9**	**846.4**
Total Payable Gold Equivalent	**280.0**	**kozs Au Eq**	**2,402.8**		**–**	**–**	**396.7**	**324.7**	**317.0**	**305.5**	**313.2**	**287.4**	**201.7**	**184.3**	**72.3**
Revenue	Revenue Split														
Gold Revenue	80%	C$ M	$3,308.9		–	–	$558.3	$449.3	$397.8	$421.8	$395.6	$409.2	$283.3	$286.7	$106.8
Silver Revenue	20%	C$ M	$829.9		–	–	$124.9	$110.0	$148.2	$104.5	$143.9	$85.8	$64.1	$30.8	$17.6
Total Revenue	**100%**	**C$ M**	**$4,138.8**		**–**	**–**	**$683.3**	**$559.3**	**$546.0**	**$526.3**	**$539.6**	**$495.0**	**$347.4**	**$317.5**	**$124.5**
Total Mill Feed		**'000t**	21,307		–	–	2,354	2,500	2,500	2,500	2,500	2,500	2,500	2,500	1,453
Concentrate Produced		**000t (dmt)**	2,516		–	–	425	342	302	321	301	311	215	218	81
TC, RC & Penalties															
Treatment Charges	$180 US$/t dmt	C$ M	$588.7		–	–	$99.3	$79.9	$70.8	$75.0	$70.4	$72.8	$50.4	$51.0	$19.0
Gold Refining Charges	$15.00 US$/oz	C$ M	$37.5		–	–	$6.3	$5.1	$4.5	$4.8	$4.5	$4.6	$3.2	$3.2	$1.2
Silver Refining Charges	$1.00 US$/oz	C$ M	$51.9		–	–	$7.8	$6.9	$9.3	$6.5	$9.0	$5.4	$4.0	$1.9	$1.1
Penalties		C$ M	$87.0		–	–	$42.7	$30.6	$13.1	–	$0.6	–	–	–	$0.1
Total TC, RC & Penalties		**C$ M**	**$765.0**		**–**	**–**	**$156.1**	**$122.5**	**$97.6**	**$86.3**	**$84.4**	**$82.8**	**$57.6**	**$56.2**	**$21.4**
Concentrate Moisture	12.0%														
Transport to Smelter	$118 C$/t wmt	C$ M	$337.3		–	–	$56.92	$45.80	$40.56	$43.00	$40.34	$41.72	$28.88	$29.23	$10.89
Net Smelter Return			**$3,036.4**		**–**	**–**	**$470.2**	**$390.9**	**$407.9**	**$396.9**	**$414.8**	**$370.5**	**$260.9**	**$232.1**	**$92.2**
Royalty	1.0%	%	–		–	–	1.00%	1.00%	1.00%	1.00%	1.00%	1.00%	1.00%	1.00%	1.00%
Total Royalties		**C$ M**	**$30.4**		**–**	**–**	**$4.7**	**$3.9**	**$4.1**	**$4.0**	**$4.1**	**$3.7**	**$2.6**	**$2.3**	**$0.9**
Operating Costs															
Per Tonne Basis															
Mining Cost		C$/t mined	$3.44		–	–	$3.02	$3.21	$3.25	$3.15	$3.13	$3.58	$4.39	$5.45	$6.21
Processing Cost	$21.64	C$/t milled	$21.64		–	–	$21.64	$21.64	$21.64	$21.64	$21.64	$21.64	$21.64	$21.64	$21.64
G&A Cost	$6.06	C$/t milled	$6.06		–	–	$6.06	$6.06	$6.06	$6.06	$6.06	$6.06	$6.06	$6.06	$6.06

					-3	-2	-1	1	2	3	4	5	6	7	8	9
Dollar figures in real C$ M unless otherwise noted		**Inputs**	**Units**	**Total / Avg.**	**2020**	**2021**	**2022**	**2023**	**2024**	**2025**	**2026**	**2027**	**2028**	**2029**	**2030**	**2031**
Annual C$M Basis																
Mining Cost			C$ M	$560.8		–	–	$72.6	$80.3	$81.3	$78.7	$78.2	$63.6	$51.0	$39.7	$15.3
Processing Cost			C$ M	$461.2		–	–	$51.0	$54.1	$54.1	$54.1	$54.1	$54.1	$54.1	$54.1	$31.4
G&A Cost			C$ M	$129.1		–	–	$14.3	$15.2	$15.2	$15.2	$15.2	$15.2	$15.2	$15.2	$8.8
Total Operating Costs			**C$ M**	**$1,151.1**		**–**	**–**	**$137.8**	**$149.6**	**$150.5**	**$148.0**	**$147.5**	**$132.9**	**$120.3**	**$109.0**	**$55.6**
Operating Costs per Tonne Milled - excl. smelter costs & royalties			*C$/t milled*	*$54.0*		–	–	*$58.5*	*$59.8*	*$60.2*	*$59.2*	*$59.0*	*$53.2*	*$48.1*	*$43.6*	*$38.3*
By-Product Basis																
Cash Cost *			US$/oz Au	$582		–	–	$547	$625	$482	$555	$444	$568	$679	$767	$883
All-in Sustaining Cost (AISC) **			US$/oz Au	$615		–	–	$566	$664	$517	$589	$489	$609	$742	$826	$1,013
Co-Product Basis																
Cash Cost *			US$/oz AuEq	$731		–	–	$690	$762	$711	$708	$679	$699	$799	$821	$945
All-in Sustaining Cost (AISC) **			US$/oz AuEq	$757		–	–	$705	$794	$737	$735	$712	$733	$850	$875	$1,057
Capital Expenditures																
Initial Capital																
Mining Equipment			C$ M	$14.3		$9.0	$5.2									
Mining Other			C$ M	$6.7		$6.1	$0.6									
Pre-Production Stripping			C$ M	$61.5		$14.1	$47.4									
Processing			C$ M	$107.3		$53.7	$53.7									
Onsite Infrastructure			C$ M	$21.6		$10.8	$10.8									
Offsite Infrastructure (Access Road, Water, Power)			C$ M	$14.9		$7.4	$7.4									
Processing Indirects (Incl. EPCM)			C$ M	$26.7		$13.4	$13.4									
Owners Cost			C$ M	$10.1		$5.1	$5.1									
Contingency			C$ M	$40.0		$20.0	$20.0									
Sub-Total Initial Capital				**$303.1**		**$139.6**	**$163.5**									
Sustaining Capital																
Mining		**LOM Total (C$mm)**	C$ M	$8.6				$0.9	$2.7	$1.2	$1.1	$1.3	$0.4	$0.9	$0.2	–
Processing		$10.02	C$ M	$10.0				$1.1	$1.1	$1.1	$1.1	$1.1	$1.1	$1.1	$1.1	$1.1
Onsite Infrastructure		$1.84	C$ M	$1.8				$0.2	$0.2	$0.2	$0.2	$0.2	$0.2	$0.2	$0.2	$0.2
		Tailings in Yr 2	Water (LOM Total)													
Onsite Infrastructure (Tailings + Water)	Spent in Yr 2	$3.50	$0.18	$3.68				$0.02	$3.52	$0.02	$0.02	$0.02	$0.02	$0.02	$0.02	$0.02
Contingency		$3.18	C$ M	$3.2				$0.4	$0.4	$0.4	$0.4	$0.4	$0.4	$0.4	$0.4	$0.4
Sub-Total Sustaining Capital			**C$ M**	**$27.3**				**$2.6**	**$7.9**	**$2.9**	**$2.8**	**$3.0**	**$2.1**	**$2.6**	**$1.9**	**$1.7**
Closure Cost			C$ M	$51.6				$2.4	$2.4	$4.9	$5.0	$7.5	$7.7	$7.9	$8.0	$5.8
Total Capital Expenditures			**C$ M**	**$382.0**		**$139.6**	**$163.5**	**$4.9**	**$10.3**	**$7.7**	**$7.8**	**$10.5**	**$9.7**	**$10.4**	**$9.9**	**$7.5**

Note: * Cash costs are inclusive of mining costs, processing costs, site G&A, treatment and refining charges and royalties. ** AISC includes cash costs plus corporate G&A, sustaining capital and closure costs

Table 22-3: Sensitivity Summary Table

Sensitivity Summary	Units	Lower Case	Base Case	Higher Case
Gold price	US$/oz	1200	1325	1500
Silver price	US$/oz	14	16	18
After-tax NPV	C$ M	453	638	878
After-tax IRR	%	39.7	50.5	62.5
After-tax payback	years	1.6	1.2	0.9
Average annual after-tax free cashflow, Year 1–9	C$ M	117	147	187

Table 22-4: Pre-Tax Sensitivity Analysis

Pre-Tax NPV 5% Sensitivity To Metal Prices

Silver Price (US$/oz)	Gold Price (US$/oz) $1,100	$1,200	$1,325	$1,400	$1,500
$12.00	$455	$629	$847	$978	$1,152
$14.00	$528	$702	$920	$1,051	$1,225
$16.00	$601	$776	$993	$1,124	$1,298
$18.00	$675	$849	$1,066	$1,197	$1,371
$20.00	$748	$922	$1,140	$1,270	$1,444

Pre-Tax IRR Sensitivity To Metal Prices

Silver Price (US$/oz)	Gold Price (US$/oz) $1,100	$1,200	$1,325	$1,400	$1,500
$12.00	36.7%	46.0%	56.6%	62.6%	70.3%
$14.00	40.8%	49.7%	60.0%	65.9%	73.4%
$16.00	44.7%	53.3%	63.3%	69.1%	76.4%
$18.00	48.4%	56.8%	66.6%	72.2%	79.4%
$20.00	52.0%	60.1%	69.7%	75.2%	82.3%

Pre-Tax NPV 5% Sensitivity To Head Grade

Head Grade	Gold Price (US$/oz) $1,100	$1,200	$1,325	$1,400	$1,500
(20.0%)	$206	$344	$517	$620	$758
(10.0%)	$403	$560	$755	$873	$1,029
--	$601	$776	$993	$1,124	$1,298
10.0%	$797	$989	$1,229	$1,374	$1,566
20.0%	$989	$1,198	$1,461	$1,618	$1,827

Pre-Tax IRR Sensitivity To Head Grade

Head Grade	Gold Price (US$/oz) $1,100	$1,200	$1,325	$1,400	$1,500
(20.0%)	21.5%	30.3%	40.1%	45.6%	52.5%
(10.0%)	33.7%	42.2%	52.0%	57.6%	64.7%
--	44.7%	53.3%	63.3%	69.1%	76.4%
10.0%	54.5%	63.3%	73.6%	79.5%	87.1%
20.0%	63.4%	72.4%	83.1%	89.2%	97.0%

Pre-Tax NPV 5% Sensitivity To Opex

Opex	Gold Price (US$/oz) $1,100	$1,200	$1,325	$1,400	$1,500
(20.0%)	$762	$936	$1,154	$1,284	$1,458
(10.0%)	$682	$856	$1,073	$1,204	$1,378
--	$601	$776	$993	$1,124	$1,298
10.0%	$521	$695	$913	$1,044	$1,218
20.0%	$441	$615	$833	$963	$1,138

Pre-Tax IRR Sensitivity To Opex

Opex	Gold Price (US$/oz) $1,100	$1,200	$1,325	$1,400	$1,500
(20.0%)	52.0%	60.1%	69.7%	75.2%	82.2%
(10.0%)	48.4%	56.8%	66.6%	72.1%	79.3%
--	44.7%	53.3%	63.3%	69.1%	76.4%
10.0%	40.7%	49.7%	60.0%	65.9%	73.4%
20.0%	36.6%	45.9%	56.6%	62.6%	70.3%

Pre-Tax NPV 5% Sensitivity To Total Capex

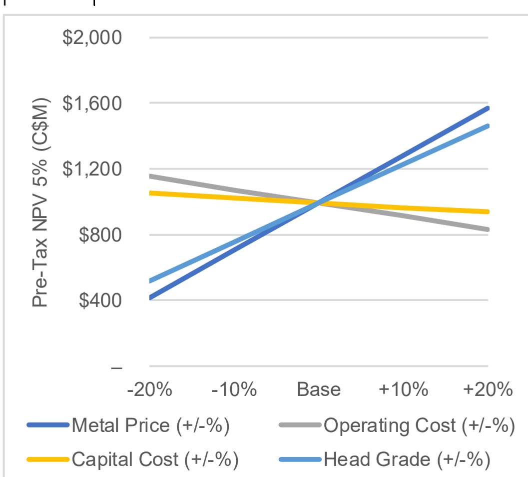
Total Capex	Gold Price (US$/oz) $1,100	$1,200	$1,325	$1,400	$1,500
(20.0%)	$659	$833	$1,051	$1,181	$1,355
(10.0%)	$630	$804	$1,022	$1,153	$1,327
--	$601	$776	$993	$1,124	$1,298
10.0%	$573	$747	$965	$1,095	$1,269
20.0%	$544	$718	$936	$1,066	$1,241

Pre-Tax IRR Sensitivity To Total Capex

Total Capex	Gold Price (US$/oz) $1,100	$1,200	$1,325	$1,400	$1,500
(20.0%)	55.9%	65.8%	77.3%	83.9%	92.3%
(10.0%)	49.8%	59.0%	69.7%	75.8%	83.6%
--	44.7%	53.3%	63.3%	69.1%	76.4%
10.0%	40.3%	48.4%	57.9%	63.3%	70.2%
20.0%	36.5%	44.3%	53.3%	58.4%	64.9%

Pre-Tax NPV 5% Sensitivity To Discount Rate

Discount Rate	Gold Price (US$/oz) $1,100	$1,200	$1,325	$1,400	$1,500
0.0%	$917	$1,164	$1,473	$1,658	$1,906
3.0%	$711	$911	$1,160	$1,310	$1,509
5.0%	$601	$776	$993	$1,124	$1,298
8.0%	$469	$612	$791	$898	$1,041
10.0%	$398	$524	$682	$776	$903



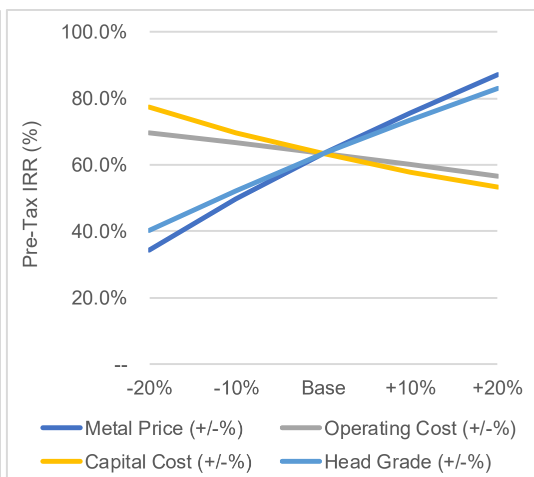


Table 22-5: Post-Tax Sensitivity Analysis

Post-Tax NPV 5% Sensitivity To Metal Prices

Silver Price (US$/oz)	Gold Price (US$/oz)				
	$1,100	$1,200	$1,325	$1,400	$1,500
$12.00	$295	$406	$545	$628	$739
$14.00	$342	$453	$591	$674	$785
$16.00	$388	$499	$638	$721	$831
$18.00	$435	$546	$684	$767	$878
$20.00	$482	$593	$731	$814	$924

Post-Tax IRR Sensitivity To Metal Prices

Silver Price (US$/oz)	Gold Price (US$/oz)				
	$1,100	$1,200	$1,325	$1,400	$1,500
$12.00	29.4%	36.8%	45.2%	50.0%	55.7%
$14.00	32.6%	39.7%	47.9%	52.4%	58.1%
$16.00	35.7%	42.5%	50.5%	54.9%	60.3%
$18.00	38.6%	45.3%	53.0%	57.2%	62.5%
$20.00	41.5%	48.0%	55.4%	59.5%	64.7%

Post-Tax NPV 5% Sensitivity To Head Grade

Head Grade	Gold Price (US$/oz)				
	$1,100	$1,200	$1,325	$1,400	$1,500
(20.0%)	$133	$223	$335	$400	$488
(10.0%)	$261	$362	$486	$561	$661
--	$388	$499	$638	$721	$831
10.0%	$513	$635	$788	$879	$1,001
20.0%	$635	$768	$934	$1,034	$1,167

Post-Tax IRR Sensitivity To Head Grade

Head Grade	Gold Price (US$/oz)				
	$1,100	$1,200	$1,325	$1,400	$1,500
(20.0%)	17.1%	24.2%	32.2%	36.4%	41.9%
(10.0%)	26.8%	33.8%	41.5%	45.9%	51.6%
--	35.7%	42.5%	50.5%	54.9%	60.3%
10.0%	43.4%	50.5%	58.3%	62.7%	68.3%
20.0%	50.5%	57.4%	65.3%	69.8%	75.5%

Post-Tax NPV 5% Sensitivity To Opex

Opex	Gold Price (US$/oz)				
	$1,100	$1,200	$1,325	$1,400	$1,500
(20.0%)	$491	$602	$740	$823	$933
(10.0%)	$440	$550	$689	$772	$882
--	$388	$499	$638	$721	$831
10.0%	$337	$448	$587	$670	$780
20.0%	$285	$397	$536	$619	$729

Post-Tax IRR Sensitivity To Opex

Opex	Gold Price (US$/oz)				
	$1,100	$1,200	$1,325	$1,400	$1,500
(20.0%)	41.6%	48.0%	55.5%	59.6%	64.8%
(10.0%)	38.7%	45.3%	53.0%	57.3%	62.6%
--	35.7%	42.5%	50.5%	54.9%	60.3%
10.0%	32.6%	39.6%	47.8%	52.4%	58.0%
20.0%	29.2%	36.6%	45.1%	49.9%	55.7%

Post-Tax NPV 5% Sensitivity To Total Capex

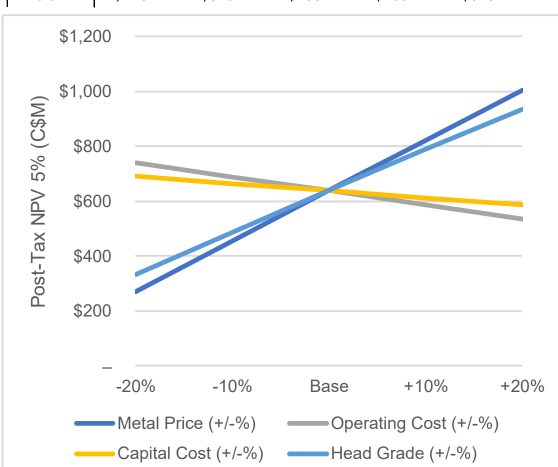
Total Capex	Gold Price (US$/oz)				
	$1,100	$1,200	$1,325	$1,400	$1,500
(20.0%)	$441	$552	$690	$773	$884
(10.0%)	$415	$526	$664	$747	$857
--	$388	$499	$638	$721	$831
10.0%	$362	$473	$612	$695	$805
20.0%	$336	$447	$585	$668	$779

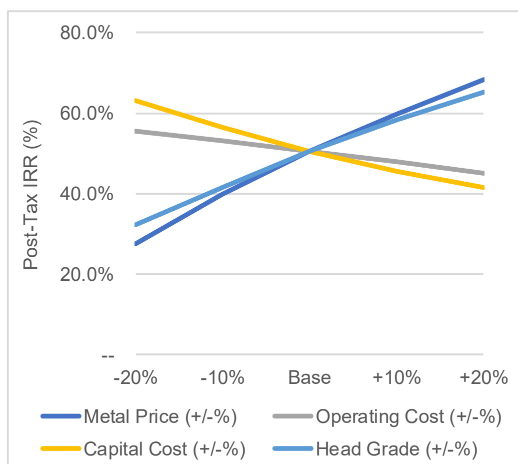
Post-Tax IRR Sensitivity To Total Capex

Total Capex	Gold Price (US$/oz)				
	$1,100	$1,200	$1,325	$1,400	$1,500
(20.0%)	46.2%	54.0%	63.1%	68.0%	74.2%
(10.0%)	40.5%	47.7%	56.3%	60.8%	66.6%
--	35.7%	42.5%	50.5%	54.9%	60.3%
10.0%	31.6%	38.1%	45.6%	49.7%	55.0%
20.0%	28.1%	34.3%	41.4%	45.3%	50.3%

Post-Tax NPV 5% Sensitivity To Discount Rate

Discount Rate	Gold Price (US$/oz)				
	$1,100	$1,200	$1,325	$1,400	$1,500
0.0%	$606	$763	$959	$1,077	$1,234
3.0%	$464	$591	$750	$845	$971
5.0%	$388	$499	$638	$721	$831
8.0%	$297	$389	$503	$571	$662
10.0%	$248	$329	$430	$490	$570





23 ADJACENT PROPERTIES

This section is not relevant to this Report.

24 OTHER RELEVANT DATA AND INFORMATION

This section is not relevant to this Report.

25 INTERPRETATION AND CONCLUSIONS

25.1 Introduction

The QPs note the following interpretations and conclusions in their respective areas of expertise, based on the review of data available for this Report.

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

Information from legal experts and Skeena's in-house experts support that the tenure held is valid and sufficient to support a declaration of Mineral Resources.

Under the terms of an option agreement with Barrick, Skeena may earn a100% interest in the Project.

Where on-ground work commitments have not been met, Skeena has made cash-in-lieu payments as stipulated under BC regulations. All statutory annual reporting obligations have been met.

Royalties are payable on a number of the claims.

Skeena holds an interest in two surface leases and the Eskay Road access. Skeena will need to acquire surface rights in support of any future mining operations.

No water rights are currently held.

Skeena's current environmental liabilities are related to activities undertaken by Skeena, and activities arising from permitting. The key liabilities would be remediation of drill pads and drill access roads. Skeena has posted an environmental bond with the relevant BC authorities in relation to the work programs that have been conducted.

To the extent known to the QP, there are no other significant factors and risks that may affect access, title, or the right or ability to perform work on the Project that are not discussed in this Report.

25.3 Geology and Mineralization

The Eskay Creek deposit is generally classified as an example of a high-grade, precious metals-rich epithermal volcanogenic massive sulphide (VMS) deposit; however, it has also been suggested to be an example of a subaqueous hot spring gold–silver deposit.

The understanding of the Eskay Creek deposit settings, lithologies, mineralisation, and the geological, structural, and alteration controls on mineralisation is sufficient to support estimation of Mineral Resources.

There is significant remaining exploration potential in the Eskay Creek deposit and environs. Exploration targets include syn-volcanic feeder structures at depth and along strike; mineralization hosted within the largely unexplored Lower Mudstone horizon; and the in the vicinity of the 22 Zone, which remains open along strike and at depth. Due to limited legacy exploratory drilling in the area between the 21A and 22 Zones, additional opportunities exist to discover and delineate near-surface, rhyolite-hosted feeder mineralization.

25.4 Exploration, Drilling and Analytical Data Collection in Support of Mineral Resource Estimation

The exploration programs completed to date are appropriate for the style of the deposits in the Project area.

Sampling methods are acceptable for Mineral Resource estimation.

Sample preparation, analysis and security are generally performed in accordance with exploration best practices and industry standards at the time the information was collected.

The quantity and quality of the logged geological data, collar, and downhole survey data collected in the exploration and infill drill programs are sufficient to support Mineral Resource estimation.

No material factors were identified with the data collection from the drill programs that could significantly affect Mineral Resource estimation.

The sample preparation, analysis, and security practices and are acceptable, meet industry-standard practices at the time they were undertaken, and are sufficient to support Mineral Resource estimation.

Eskay Creek mine initiated QA/QC measures into their sample stream in 1997. With progressive years the QAQC protocol became more comprehensive and detailed. QA/QC submission rates meet industry-accepted standards at the time of the campaign. The QA/QC programs did not detect any material sample biases in the data reviewed that supports Mineral Resource estimation.

The data verification programs concluded that the data collected from the Project adequately support the geological interpretations and constitute a database of sufficient quality to support the use of the data in Mineral Resource estimation.

25.5 Metallurgical Testwork

Metallurgical testwork and associated analytical procedures were appropriate to the mineralisation type, appropriate to establish the optimal processing routes, and were performed using samples that are typical of the mineralisation styles found within the various mineralized zones.

Samples selected for testing were representative of the various types and styles of mineralisation. Samples were selected from a range of depths within the deposits. Sufficient samples were taken so that tests were performed on sufficient sample mass.

Supplementary testwork is ongoing into options for concentrate treatment. These treatments involve hydrometallurgical or pyrometallurgical oxidation of the sulphide content prior to cyanide leaching with/without carbon to minimise the impact of preg-robbing agents.

Recovery factors estimated are based on appropriate metallurgical testwork, and are appropriate to the mineralisation types and the selected process route. Based on the 2019 testwork results on samples with a range of head grades, a flotation concentrate of saleable precious metal content can be produced at high recoveries of both gold and silver. This concentrate will contain impurities of arsenic, antimony and mercury that will be subject to penalties. Depending on the concentrate customer, the antimony content may be included as a payable metal, provided the level is above a threshold value (e.g. 3% Sb).

Across the nine-year mine life, 60% of the plant feed anticipated to be rhyolite with 20% mudstone and 20% hanging wall andesite material. In Year 1, almost 60% of plant feed will be from the 21A

Zone with higher precious metal grades and impurity levels. As the percentage of the 21A material decreases over time, the gold head grade will fall from almost 5 g/t Au to around 3 g/t Au. Similarly, silver grade will be higher in years 1–6 at 100 g/t Ag, and will fall to around half this value in Year 7.

25.6 Mineral Resource Estimates

The Mineral Resource estimation for the Project conforms to industry-accepted practices, and is reported using the 2014 CIM Definition Standards.

Factors that may affect the estimate include: changes to long-term metal price assumptions; changes in local interpretations of mineralization geometry and continuity of mineralized zones; changes to the density values applied to the mineralized zones; changes to geological shape and continuity assumptions; potential for unrecognized bias in the assay results from legacy drilling where there was limited documentation of the QA/QC procedures; changes to the input values used to generate the AuEq cut-off grade; changes to metallurgical recovery assumptions; changes in assumptions of marketability of final product; changes to the conceptual input assumptions for assumed open pit operation; variations in geotechnical, hydrogeological and mining assumptions; changes to environmental, permitting and social license assumptions.

25.7 Mine Plan

25.7.1 Geotechnical Considerations

Pit slope angle assessments were primarily based on resource drilling data and core photographs, simple RQD data, economic pit shells, geologic models, and relevant background reports. No material geotechnical drilling, logging, mapping, sampling, or laboratory testing was completed for the PEA. Overall, the data indicate generally ‘fair’ to ’good’ rock mass conditions throughout the planned mining zone.

The pit slopes are expected to consist primarily of hanging wall andesite along the upper pit walls with rhyolite being more prevalent at lower pit elevations. The contact mudstone is expected to only affect narrow zones between the hanging wall andesite and rhyolite. The parameters developed for the north pit were also applied to the south pit due to limited information available and the small size of the south pit.

To allow steeper slope angles in areas with better quality rock and to minimize stripping to the greatest extent possible, AGP divided the pit into individual slope design sectors, based on slope height and dominant geology. Estimates of suitable overall slope angles were then developed for each of the individual sectors. The inter-ramp slope recommendations ranged between 32º and 42º.

25.7.2 Hydrological Considerations

The regional groundwater regime is most likely controlled by the regional groundwater flow system, and from seasonal snow melt. The regional faults likely provide high permeability recharge pathways and groundwater storage areas; however, the rock units themselves are highly fractured and even away from major faults constitute fractured aquifers. Faulted andesite most likely provides the highest permeability and highest storage capacity of all the rock units. Historically, three high-permeability zones with large areal extents, and six hydrostratigraphic units were identified.

The planned ultimate pit bottom will be at 714 masl, and therefore only about 50 m of flooded working is likely to require dewatering. The andesite and mudstone lithologies will likely dewater easily compared to the rhyolite, which reportedly has high fines content and drains poorly (significantly lower hydraulic conductivity than the andesite). The rhyolite will generally occupy lower elevations in the final pit extent; however, rhyolite would be present on the south and east pit highwall and may be susceptible to failure if pore-water pressure builds up on fault planes. Horizontal boreholes drilled from pit benches may be a more efficient and effective means of depressurizing this material than vertical dewatering wells.

Groundwater interaction with surface water may be exacerbated by dewatering of the underground workings; however, historic mine inflow records do not suggest a significant flow path for creek water to enter the mine. Pit stability can be managed by progressive dewatering of the ground behind the pit slope with vertical or horizontal boreholes. The mudstones may require special attention as matrix pore pressures could remain elevated despite successful dewatering.

25.7.3 Mine Plan

The mine plan is partly based on Inferred Mineral Resources that are considered too speculative geologically to have the economic considerations applied to them that would enable them to be categorized as Mineral Reserves, and there is no certainty that the PEA based on these Mineral Resources will be realized.

The PEA is based on open pit only mining of the Eskay Creek deposit. AGP's opinion is that with current metal pricing levels and knowledge of the mineralization and previous mining activities, open pit mining offers the most reasonable approach for development.

The Eskay Creek PEA has two pit designs: the north pit and the south pit. The north pit will have four phases with Phase 3 split into three parts for access. The south pit will be a small single-phase pit that will be mined at the end of the mine life. These pits will provide a total of 21.3 Mt of mill feed grading 3.23 g/t Au and 78 g/t Ag. Waste movement from these phases will amount to 154.0 Mt giving a strip ratio of 7.2:1 (waste:mill feed).

The mill feed cut-off is based on a value per tonne which is often referred to as the milling cut-off. This was determined to be 1 g/t Au, and considers all the penalties, transportation costs and smelting charges for the bulk concentrate.

The feed to the plant was diluted. The calculation is based on a 1.25 m dilution skin on contacting blocks. This higher level of dilution skin was assumed considering the requirement to work around old underground mine workings which could result in mixing of waste and feed material. The result of the dilution calculation was a 20.8% increase in feed tonnage and a 16.6% lower feed grade. A mining recovery of 98% was also applied.

The phases are scheduled to provide 2.5 Mt/a of feed to the mill over a nine-year operating mine life, after two years of pre-production stripping. The pits are sequenced to minimize initial stripping and provide higher feed grades in the early years of the mine life. This is accomplished with stockpiling of lower-grade material.

The pits will be built on 8 m benches with safety berm placement each 16 m. Minimum mining widths of 35–40 m were maintained in the design. Ramps will be at 10% gradient and will vary in width from 23.3 m (single lane width) to 30.2 m (double lane width). They have been designed for 142 t haulage trucks.

The mine equipment fleet is anticipated to be leased to lower capital requirements. The fleet will consist of six 140 mm rotary drills, two 22 m^3 hydraulic shovels and one 13 m^3 front-end loader. The truck fleet will peak at nine trucks in Year 4. This is due to the long hauls anticipated from the pit bottom to the higher waste rock storage facility (WRSF) elevations. Dozers, graders, small backhoes and other support equipment are considered in the equipment costing. Additional support equipment in the form of snowplows and small excavators will be part of the fleet to maintain operations year-round with the expected amount of annual snowfall. An additional front-end loader (13 m^3) will be at the primary crusher full time and tramming material from the stockpile as required. The pit front end loader will be the backup for crusher loader.

The WRSF will fill the valley from the primary crusher towards the plant on the western side of the pits. The WRSF will have a top elevation of 1122 masl and the toe will be near the primary crusher at 902 masl for a total height of 220 m. A total volume of 70.4 Mm^3 has been designed, which is sufficient for the mine needs with a total of 6.8 Mm^3 of in-pit backfill.

Material from the mine has been assumed to be potentially acid-generating (PAG). All drainage from the WRSFs will be collected in ditches, pumped to the settling ponds and treated as required. Additional work on the exact nature of the material from a PAG perspective should be defined during more detailed studies.

25.8 Recovery Plan

The plant will process material at a rate of 2.5 Mt/a with an average head grade of 3.2 g/t Au and 78 g/t Ag to produce a flotation concentrate.

The process plant flowsheet designs were based on testwork results and industry-standard practices. The flowsheet was developed for optimum recovery while minimizing capital expenditure and life of mine operating costs. The process methods are conventional to the industry. The comminution and recovery processes are widely used with no significant elements of technological innovation.

25.9 Infrastructure

Access to the Eskay Creek Project is via Highway 37 (Stewart Cassiar Highway). The Eskay Mine Road is an all-season gravel road that connects to Highway 37 approximately 135 km north of Meziadin Junction. Within the site, heavy equipment roads will connect the main pit and waste rock pit to the main facilities and processing areas. Secondary roads will connect the plant to the crusher, crusher to top of main pit, and around the north end of main pit for light duty traffic.

Infrastructure to support the Eskay Creek project will consist of site civil work, site facilities/building, a water system, and site electrical. Site civil work includes designs for the following infrastructure: light vehicle and heavy equipment roads; conveyor corridor, 2 km long; growth media stripping and stockpiling; mine facility platforms and process facility platforms; contact water pond; TSF; waste storage facilities; and a high voltage substation platform.

The mine facilities will include the administration offices, truckshop and warehouse, tire repair shop, mine workshop, mine dry, fuel storage and distribution, permanent camp facility and miscellaneous facilities. The process facilities will include the process plant, crusher facility, process plant workshop and assay laboratory. Both the mine facilities and process facilities will be serviced with potable water, fire water, compressed air, power, diesel, communication, and sanitary systems.

The permanent camp will be housed in portable modular units comprising of 200 jack-and-jill-type dormitories. Water will be supplied by a well.

The project power will come from the local and recently-commissioned 195 MW hydroelectric facilities and leverage on the existing power grid.

There will be two main waste storage areas. The largest waste rock storage area will be WRSF WD01 along the west side of the pit. The remainder of the waste will be placed into the mined-out north pit as backfill.

The permitted TMSF will be used to subaqueously store 19.5 Mt of tailings from the proposed operation. The TMSF only requires a small embankment to contain the required volume of tailings with the majority of the tailings located below the existing outlet of the TSF. The TSF has sufficient capacity to store tailings without an embankment during the initial years of operations while maintaining 7 m (6–8 Mm^3) of water cover over the tailings bed. In year 4 of operations, a single embankment will be required to be constructed, so as to store the balance of the LOM tailings while maintaining 7 m of water cover. The TSF would be the preferred site for disposal of partially-treated and fully-treated water from the pit dewatering program, and treated wastewater from the camp. The TSF will also provide the water for the process plant.

Pit dewater will be sent directly to the WTP, then to D7 polishing ponds. and finally to Ketchum Creek. Estimated pit dewatering flow rates are all less than 150 L/s during the initial years of operations, therefore no water will be sent to the TSF. As the open pit becomes larger toward the end of the project, pit dewatering flow rates are estimated to surpass 150 L/s between late spring and fall. During this period, the overflow portion sent to the TSF will range from 4.5–286.4 L/s. The overflow will be pumped to the tailings mixing tank and sent with the tailings in the tailings transportation pipeline to the TSF. The industrial water requirements will come from the TSF, which are estimated to be 113 L/s to be used in mineral processing. The balance of the waste (tailings) and process water will be pumped to the TSF and discharged subaqueously. The approximate discharge of water, along with the tailings, is projected to be 114 L/s.

25.10 Environmental, Permitting and Social Considerations

The project will be designed, constructed, operated, and decommissioned to meet all applicable BC environmental and safety standards and practices. Skeena will develop and implement an Environmental Management System (EMS) that defines the processes by which compliance will be met and demonstrated. The EMS will include ongoing monitoring and reporting to relevant parties at the various project stages.

The main waste management issue for the project is the prevention and control of ML/ARD from the tailings, and any acid generating or PAG waste rock that is produced during mine development or operations. NAG waste rock will be deposited either in the WD-01 facility or as backfill in the north pit. PAG waste, if encountered, will be stored in the TMSF. To manage the potential for ML/ARD, Skeena has incorporated appropriate design features and mitigation measures in the project that are consistent with best practices for waste and water management.

Site water management will be a critical component of project design. The most likely avenue for transport of contaminants into the natural environment will be through surface, groundwater, and dust. Skeena will develop a Water Management Plan and Dust Control Management Plan that applies to all activities undertaken during all project phases.

Non-contact water from upstream catchments that has not been in contact with mine workings will be kept separate from water that has been in contact with mine workings and discharged to the environment with no treatment. Contact water that has been in contact with potential sources of contamination, includes seepage from the WRSF, process water, and pit dewatering. Contact water from the WRSF will be collected and sent to a water treatment plant for treatment prior to discharge if testing shows any onset of ML/ARD. If contact water quality from the WRSF is within permitted parameter limits, and is confirmed with regular testing, this water will be discharged without treatment.

Water from pit dewatering will be pumped to a water treatment plant for treatment prior to discharge to the existing mine water polishing ponds and ultimate discharge through permitted effluent discharge point D7 (identification number E219595) to Ketchum Creek. Process water will be discharged to the TMSF.

A Closure and Reclamation Plan will be developed as part of the EA and refined for the permitting process. Closure planning will include dialogue with First Nations and stakeholders to determine post mining land use objectives and necessary investigations required to achieve and monitor those objectives. The financial analysis includes a conceptual closure cost provision.

Major mining projects in BC are subject to EA and review prior to certification and issuance of permits to authorize construction and operations. The project will require provincial and federal approvals before the issuance of any permits to construct or operate.

Skeena has not filed a federal or provincial EA application. Once an application is filed, the BC EAO and IAAC will issue their decision for the project. Once the project has a provincial EAC and a federal decision statement, Skeena can apply for the necessary statutory permits and authorizations to commence project construction. No technical or policy issues are anticipated for obtaining the required project permits and approvals, given its long mining history.

Skeena has compiled a list of the key provincial and federal authorizations, licences, and permits that will be required in support of development and operations. No permits for project commercial development will be issued before an EAC is obtained. Consequently, Skeena will apply for synchronous permitting within the environmental review process for all permits. Synchronous permitting will expedite the permitting process and reduce the time to start construction.

Skeena will be required to consult with local First Nations as part of the EA process. Future First Nation engagement and consultation measures will comply with federal and provincial regulations, best practices, and Skeena's internal company policies.

The relationship between the previous owners and the Tahltan Nation have been favourable: the mine provided employment and business opportunities, and in return the local community provided a stable and capable local work force. Ongoing consultation efforts will aim to engage both community leaders and members, and attempt to resolve potential issues and concerns as they arise.

Skeena will engage and collaborate with federal, provincial, regional, and municipal government agencies and representatives as required with respect to topics such as land and resource management, protected areas, official community plans, environmental and social baseline studies, and effects assessments. Skeena will consult with the public and relevant stakeholder groups, including land tenure holders, businesses, economic development organizations, businesses and contractors (e.g., suppliers and service providers), and special interest groups (e.g. environmental, labour, social, health, and recreation groups), as appropriate.

25.11 Markets and Contracts

The concentrate as proposed is a complex gold concentrate with relatively low gold content and elevated levels of arsenic, mercury and antimony. Deleterious element assays are notably elevated in the first few years of mine life.

The PEA assumes that concentrates will be sent to an Asian port for smelting and refining. The most likely market for the concentrate is China, which offers the best payable terms and does not penalize mercury at the expected amounts in the Eskay Creek concentrate. Other smelters around the world such as the Horne smelter in Canada may also be interested in purchasing some of the concentrate, although the mercury levels could be a challenge.

The relatively high levels of deleterious elements, particularly mercury in the initial years of operation, may require that concentrate sales be spread across a number of buyers as individual smelters are likely to need to blend small volumes of concentrate with cleaner concentrates to remain within acceptable effluent limits. An alternative option could be to sell the concentrate to traders who may be able to buy it all and spread distribution across a range of end customers.

No contracts have been entered into at the Report effective date for mining, concentrating, smelting, refining, transportation, handling, sales and hedging, and forward sales contracts or arrangements. It is expected that the sale of concentrate will include a mixture of long-term and spot contracts.

Ausenco and Skeena established metal price projections for use in the PEA, on which this Report is based. The projections incorporate consideration of recent metal market information, in combination with two-year trailing actual metal prices, and bank analyst forward price projections.

25.12 Capital Cost Estimates

The capital cost estimate is presented at a ±50% accuracy, using a base date of Q3, 2019, and an exchange rate assumption of US$0.77:C$1.00.

Capital costs are estimated at $303 M of initial capital, $79 M of sustaining capital, for an overall capital cost estimate of $382 M

25.13 Operating Cost Estimates

The operating cost estimate is presented at a ±50% accuracy, using a base date of Q3, 2019, and an exchange rate assumption of US$0.77:C$1.00.

Operating costs are estimated at $135.1 M/a, or $54.02/t processed.

25.14 Economic Analysis

An engineering economic model was developed to estimate annual pre-tax and post-tax cash flows and sensitivities of the Project based on a 5% discount rate.

The analysis uses the following key inputs:

- Commercial production start-up in 2023;
- Construction period of two years;
- Mine life of 8.6 years;
- Base case gold price of US$1,325/oz and silver price of US$16/oz
- 100% ownership with 1% NSR
- The mine is scheduled to deliver 21.7 Mt of mill feed grading 3.17 g/t Au and 72.6 g/t Ag.

The pre-tax net present value discounted at 5% (NPV5%) is C$993 M, the internal rate of return IRR is 63.3%, and payback is 1.1 years. On an after-tax basis, the NPV5% is C$638 M, the IRR is 50.5%, and the payback period is 1.2 years.

A sensitivity analysis was conducted on the base case pre-tax and after-tax NPV and IRR of the Project, using the following variables: metal price, discount rate, grade, capital costs, and operating costs. Analysis revealed that the Project is most sensitive to changes in metal prices and head grade, then, to a lesser extent, to operating costs and capital costs.

25.15 Risks and Opportunities

25.15.1 Risks

25.15.1.1 Geology and Resource Modelling

The current understanding of the distribution variability of elements that can be deleterious in concentrates is based on incomplete data, as epithermal and base metal elements were only selectively sampled in the legacy drill programs. It is expected that information obtained from the planned drill programs will provide more complete data on elemental distributions within key lithologies and domains, which in turn is likely to affect the domain and grade-shell outlines as interpreted in the current Mineral Resource estimate. The risk is that the variability is much higher than currently estimated, and that the model underestimates the deleterious elemental tonnages and grades that the PEA mine plan and concentrate marketability assumptions are based on.

25.15.1.2 Mining

Mining through voids during open pit operations is a generally manageable risk where such voids are known to exist. However, unidentified voids may exist, and present a risk to mine and production plans if alternate schedules have to be derived, or new safety measures implemented.

25.15.1.3 Process

Solid/liquid separation issues could increase process costs due to larger thickeners and filters and use of flocculant.

Higher mass pull to final concentrate might result without careful control on grinding pulp chemistry (e.g. stainless-steel media).

25.15.1.4 Infrastructure

A portion of the access road passes through topography which is known to have an elevated geohazard (e.g. avalanche) risk. There is potential for geohazard events to temporarily halt movement along the access corridor.

25.15.1.5 Environmental, Permitting and Social

The current permits for the Eskay Mine do not consider operations at the scale contemplated in this PEA. Additional work will be required to support permit updates and amendment applications, which will include environmental baseline data collection and environmental assessment.

The project is within the territories of Indigenous groups. Agreements with such groups that may be affected by the envisaged project remain to be negotiated.

25.15.2 Opportunities

25.15.2.1 Exploration

Exploration activities are likely to identify additional mineralization, and these efforts could result in changes to the style of mineralization to that currently identified, the scale of the Project, and the deleterious elemental issues identified.

25.15.2.2 Resource Modelling

There is upside Project potential if mineralisation currently classified as Inferred can be upgraded to higher confidence categories.

25.15.2.3 Mining

Material within the 1 m buffer around old stopes is currently classified and modelled as waste in the open pit model, and in the underground model, a 3 m buffer is assumed. With additional sampling, some or all of the buffer zone materials may be able to be brought into the mill feed, and may contain grade.

With detailed metallurgical testwork information on lithologies and zones, the mining sequence may be altered to provide higher value initially

There is potential for improved slope design, when additional geotechnical data such as waste rock strength and joint orientations, are available from drill testing.

25.15.2.4 Process

Higher gold and silver recoveries may be obtained from lower head grade samples with optimised flotation conditions.

Pre-concentration by screening and/or bulk sorting might reject waste material and increase plant feed grade.

25.15.2.5 Marketability

There is upside potential for the project if the planned drill programs more comprehensively document deleterious elemental distributions such that the levels of these elements, in particular arsenic and mercury, can be minimised in the concentrate to below smelter penalty thresholds.

25.16 Conclusions

The mine plan is partly based on Inferred Mineral Resources that are considered too speculative geologically to have the economic considerations applied to them that would enable them to be categorized as Mineral Reserves, and there is no certainty that the PEA based on these Mineral Resources will be realized. Mineral Resources that are not Mineral Reserves do not have demonstrated economic viability.

Based on the assumptions and parameters presented in this Report, the PEA shows positive economics. The PEA supports that additional more detailed studies are warranted.

26 RECOMMENDATIONS

26.1 Introduction

The recommended work program is divided into two phases. The phases can be conducted concurrently, but some portions of the phase 1 work plan would be incorporated into the phase 2 recommendations.

The first recommendations phase totals approximately $11.49 M, and would be completed in support of more detailed studies. The program will consist of drilling; determination of whether bulk ore-sorting could potentially be implemented at the pre-mining stage; a study to determine if a relationship between rock mass structure and head grade exists; additional metallurgical testwork; materials handling tests; mine geotechnical data collection, data reviews in support of geotechnical and hydrological assumptions; additional hydrological data gathering; water treatment testwork; review of cost assumptions for grade control; additional mine studies, reviews of available climate data; collection of additional climate-related information, and geotechnical data collection in support of infrastructure locations and designs, and data collection on potential borrow pit sources.

The second phase is estimated at about $4.6 M, and will consist of project environmental, permitting, and social de-risking activities.

26.2 Phase 1

The planned Phase 1 work program is set out in the following subsection, by major discipline area.

26.2.1 Drilling

A total of 209 drill holes for a total of 14,267.27 m have been drilled at Eskay Creek since the database supporting the Mineral Resource estimate was closed out.

Skeena plans to drill a further 98 drill holes (approximately 16,500 m), using skid-mounted drill rigs and helicopter support. This program is estimated with all-in drilling costs of $475/m, to be approximately $8.5 M. At program completion, the intent is to update the block model and resource estimate.

26.2.2 Sampling and QA/QC

The QA/QC measures implemented in the 2018–2019 drill programs should be retained for future drill campaigns.

Lithological, alteration, mineralization and structural data captured during these programs should continue to be used to refine geological understanding and interpretations and inform the resource modelling process.

The current SG sampling process at Eskay Creek is to conduct on-site density determinations using the water displacement method. Future drill programs should adopt a method of independently analysing a percentage of the SG samples.

With the recently-completed LiDAR survey, there is the opportunity of incorporating the results into future structural modelling interpretations. The recent LiDAR results will also be used as the final topographic surface in future model.

Geotechnical inspections of the underground workings will need to be completed to determine rock conditions immediately adjacent to, and within, the mined-out solids; measurements that are needed for adjusting the depletion buffer zone appropriately.

26.2.3 Metallurgy

Sample selection for future mining studies should reflect mineralization that would be treated in the first five years of the mine life. Variability samples are required to understand the responses of the various mineralized zones to flotation kinetics and contaminant correlations.

Additional comminution tests (e.g. crushing work index, rod work index, SMC and abrasion index)) are recommended on material representative of the first 3–5 years of the planned operation, to provide more confidence in equipment selection, and to ensure that there is sufficient comminution information that is spatially representative of the variability within the various mineralized zones.

An extended gravity-recoverable gold test should be conducted on a master composite sample to confirm the PEA flowsheet.

A gold deportment analysis and trace mineral search should be undertaken on the master composite flotation rougher concentrate.

Flotation tests, including optimisation, locked cycle, QEMSCAN and contaminant removal tests, should be undertaken on variability samples. Performance optimization, contaminant liberation and/or confirmation should be tested on individual variability samples as per zone-optimized conditions determined from flotation kinetic tests.

A tailings dewatering test and a concentrate dewatering test are recommended. Rheology of the tailings slurry stream should be conducted on the combined composite sample, without any prior treatment that would affect the rheology.

The combined budget estimate for this work is $365,000.

26.2.4 Materials Handling

Material handling test work is recommended for design of bins, chutes, conveyors and stockpile drawdown. This program is estimated at $53,000.

26.2.5 Mine Geotechnical

A program to map and characterize discontinuity sets at exposed outcrops and at road and drill pad cuts is strongly recommended. It will be important to characterize the persistence and geotechnical properties of the discontinuity surfaces. This information is required to determine the shear strength of the discontinuities and assess whether they are likely to be significant with respect to bench, inter-ramp, and/or global stability. These programs are estimated at $10,000.

A nominal six to eight hole geotechnical drilling and rock mass characterization program is proposed to support more detailed studies, including targeted drilling of current data voids, particularly the portions of the higher wall sectors, to include discontinuity orientation measurements (where possible), sampling for additional laboratory strength testing, and televiewer surveys. These dedicated geotechnical core holes should target the proposed pit boundaries and rock masses containing the interim and final pit slopes. The holes should mainly target waste rock zones outside of the mineralized zone to determine the geotechnical properties of the units forming the pit walls. The core holes should be drilled using a triple tube core barrel to preserve the integrity of the core while drilling and retrieving. The program, assuming eight holes are completed, is estimated at $1.2 M, based on 2,000 m of drilling at $600/m.

Core orientation (using the ACT, EZ-Mark, or equivalent systems) and/or optical or acoustic televiewing of select holes will be needed to determine discontinuity data. Point load tests should be completed at regular intervals of drill core (~once per run to domain intercept scale). Additional laboratory testing is recommended, including uniaxial compressive strength testing (with strain measurements), tri-axial strength testing, direct shear testing of discontinuities, and index testing of discontinuity infill materials. Samples should be collected from dedicated (or first-priority) geotechnical drill holes to ensure the appropriate materials are sampled, and to avoid conflicts with exploration sampling and assaying requirements. UCS and triaxial testing should be completed for each of the significant lithological units. The triaxial testing should focus on characterizing the intact rock strength both across and parallel to foliation. Samples of fault or dike contact gouge should also be collected and tested to help characterize the strength of these materials. The combined orientation and testwork program is estimated at $50,000.

The following office-based data evaluation tasks are recommended:

- Updating the existing 3D lithological and/or structural models to incorporate the results of any additional exploration drilling and/or an improved understanding of the deposit geology;
- Interpretation of structural and geotechnical mapping and development of a site geologic structural model incorporating major fault and shear structures;
- Anisotropic/heterogeneous rock mass strengths should be investigated, defined, and utilized as appropriate to capture the conditions in directions parallel to structural fabric and orientations, and with respect to pit slope sector orientations;
- Rock mass disturbance due to blasting and stress effects should be modeled as a series of zones with decreasing values away from the excavation face;
- Pore pressure conditions should be based on transient numerical analyses, which consider the actual mine sequencing.

These studies are estimated at $50,000.

26.2.6 Mine Studies

The following should be addressed during more detailed studies. These studies are estimated at $400,000.

26.2.6.1 Grade Control

The PEA assumed that RC and blasthole sampling would be the preferred grade control methods. Sample sizes, methodology of sample selection and assaying procedures need to be defined to properly assess the cost of grade control.

26.2.6.2 Geology Model Improvement

The current PEA model has a 1 m buffer around the old stopes which is modelled as waste. This needs to be examined further to confirm whether this assumption is valid and assess the impact on the overall mine plan.

Currently the mine plan assumes that all waste material is PAG. A study needs to be completed to categorize the waste material by lithology type to determine if waste encapsulation is possible, with a resulting potential reduction in water treatment costs.

26.2.6.3 Dewatering Requirements

A proper understanding of pumping requirements and the hydrogeology is critical. Further work assessing this is recommended.

26.2.6.4 Pit Slope Sensitivity

A detailed examination of the slopes to reduce stripping while still providing a safe work environment is required. Detailed mapping of the slopes and recommendations and further analysis is required.

26.2.6.5 Mining Schedule Optimization

A review of the mining schedule and design should be completed with updated metallurgical inputs resulting from ongoing and planned testwork.

26.2.7 Hydrological

Hydrogeological testing (packer testing, profile tracer testing) and instrumentation (i.e. piezometers) should be installed in select holes to provide basic data for groundwater modelling and excavation dewatering/depressurization simulations. This program is budgeted at $75,000.

26.2.8 Water Treatment

After the site-wide water balanced has been further evaluated and ARD parameters are better understood, water treatment testwork should be conducted to confirm that impurities can be removed from water prior to discharge to the environment. This work is estimated at $50,000.

26.2.9 Infrastructure Geotechnical, Construction Borrow Materials, and Hydrological

Regional and local metrological data should be collected to support development of site climate data and hydrological parameters. Such data should be reviewed to ensure that it is statistically reliable for use by the Project, including effects of location and elevation. This should include:

- Examination of data from Seabridge Gold's weather station for the KSM project;
- Data sets from long-term public regional weather stations.

A weather station should be installed at the project to provide a correlation between the Eskay Creek and Seabridge Gold's KSM project data sets.

Field mapping, geotechnical sample collection (boreholes, tests pits) and laboratory studies should be conducted to identify borrow material sources for construction activities, and provide information for support of the WRSF, plant site, ancillary facilities locations and the TSF design.

This program is estimated at $1,265,000.

26.3 Phase 2

The second work phase will focus on project environmental, permitting, and social de-risking activities, which will include:

- Baseline and targeted environmental studies. As the majority of baseline information was collected prior to the 1994–2008 mining operation, Skeena's focus will be on re-establishing the baseline data for the project area. This would include studies such as habitat assessment, endangered or threatened species, and cross-checking sites selected to host infrastructure to ensure selected sites will have the minimal disturbance possible. Many of the studies are likely to have requirements for seasonal data collection;
- Environmental assessment;
- Documenting the required data to support applications for operating permits and completion of such applications;
- Consultations and negotiations with Indigenous groups;
- Other stakeholder engagement and consultation;
- Update water balance to better understand makeup requirements, distribution of site flows, site water quality and water treatment requirements.

A budget of approximately $4,595,000 is recommended.

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